CV5 Pegmatite Lithium-Only Feasibility Study
NI 43-101 Technical Report
Shaakichiuwaanaan Project

Prepared for:

PMET Resources Inc.
1801, McGill College, Suite 900,
Montréal, Québec, Canada H3A 1Z4

Prepared by:

G MINING SERVICES INC.
1010-5025, Lapinière Blvd.
Brossard, Québec, J4Z 0N5

Carl Michaud, P.Eng., G Mining Services.
Pascal Droz, P Eng., G Mining Services.
Todd McCracken, P.Geo., BBA Inc.
Ryan Cunningham, P Eng., M. Eng., Primero.
Sebastien Guido, P.Eng., M.Sc., Alius Mine Consulting.
Hugo Latulippe, P.Eng., BBA Inc.
Ryan Smilovici, P.Eng., Paterson & Cooke.
Philip Addis, P.Eng., AtkinsRéalis.
Antoine Cogulet, P.Eng., AtkinsRéalis.
Holman Tellez, P.Eng., AtkinsRéalis.
Nathalie Fortin, Eng., M. Env., WSP.
Geneviève Marchand, P.Eng., M.Sc., AtkinsRéalis.
Neal Sullivan, Ph.D., P.Geo., Vision Geochemistry.
Sandra Pouliot, P.Eng., M.Sc. A, AtkinsRéalis.
Michel Mailloux, P.Eng., P.Geo., M.Sc., Mailloux Hydrogéologie.
Charles Gagnon, P.Eng., M.Sc., CGM Expert.

Effective Date: October 20, 2025
Issue Date: November 14, 2025

IMPORTANT NOTE

General Conditions and limitations

Use of the report and its contents

This report has been prepared for the exclusive use of the Client or its agents. The factual information, descriptions, interpretations, comments, recommendations and electronic files contained herein are specific to the projects described in this report and do not apply to any other project or site. Under no circumstances may this information be used for any other purposes than those specified in the scope of work unless explicitly stipulated in the text of this report of formally interpreted when taken individually or out-of-context. As well, the final version of this report and its content supersedes any other text, opinion or preliminary version produced by G Mining Services Inc.

CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project

Final Revision

James Bay, QC, Canada

PMET Resources Inc.

11801, McGill College, Suite 900,
Montréal, Québec,
Canada H3A1Z4
Tel.: +1 (604) 279-8709
E-mail: info@pmet.ca
Web Address: https://www.pmet.ca/

G Mining Services Inc.

1010-5025, Lapinière Blvd.
Brossard, Québec
Canada J4Z 0N5
Tel: (450) 465-1950 • Fax: (450) 465-6344
E-mail: m.gignac@gmining.com
Web Address: www.gmining.com

November 14, 2025

Qualified Persons

Prepared by:

(signed and sealed) “Carl Michaud” Date: November 14, 2025
Name

(signed and sealed) “Pascal Droz” Date: November 14, 2025
Name

(signed and sealed) “Todd McCracken” Date: November 14, 2025
Name

(signed and sealed) “Ryan Cunningham” Date: November 14, 2025
Name

(signed and sealed) “Sebastien Guido” Date: November 14, 2025
Name

(signed and sealed) “Hugo Latulippe” Date: November 14, 2025
Name

(signed and sealed) “Ryan Smilovici” Date: November 14, 2025
Name.

(signed and sealed) “Philip Addis” Date: November 14, 2025
Name.

(signed and sealed) “Antoine Cogulet” Date: November 14, 2025
Name.

(signed and sealed) “Holman Tellez” Date: November 14, 2025
Name.

(signed and sealed) “Nathalie Fortin” Date: November 14, 2025
Name.

(signed and sealed) “Neal Sullivan” Date: November 14, 2025
Name.

(signed and sealed) “Sandra Pouliot” Date: November 14, 2025
Name.

(signed and sealed) “Michel Mailloux” Date: November 14, 2025
Name.

(signed and sealed) “Charles Gagnon” Date: November 14, 2025
Name.

(signed and sealed) “Geneviève Marchand” Date: November 14, 2025
Name.

CERTIFICATE OF QUALIFIED PERSON

Carl Michaud, P.Eng., MBA

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, Carl Michaud, P.Eng., MBA., do hereby certify that: I, Carl Michaud, P.Eng., MBA., do hereby certify that:

1) I am currently a Vice President of Mining Engineering, at G Mining Services with an office at 5025, Lapinière Blvd, Suite 1010, Brossard, Québec , Canada, J4Z 0N5.
2) I graduated from l'Université Laval with a B.Sc. (Mine Engineering) in 1996. In addition, I obtained an M.B.A. from the Université du Québec à Chicoutimi, in 2012.
3) I am a Professional Engineer registered with the "Ordre des Ingénieurs du Québec" (OIQLicence: No. 117090).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation. . I have occupied different positions, both technical and operational, related to mining engineering, in Underground and Open Pit operation. This experience includes Kiena and Sigma Gold mine (Placer Dome), Éléonore Mine (Goldcorp) and Mont Wright Mine (Arcelor Mittal).
5) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6) I am a contributing author for the preparation of the Technical Report and am responsible for the following sections and subsections 1.1, 1.2, 1.31.14, 1.15, 1.20, 1.21, 1.26, 2, 3.2, 15, 16(except 16.2,16.3, 16.5.6, 16.5.7.4), 21.3 (except 21.3.3), 22, 24.3, 25.1, 25.3, 25.4, 25.10, 25.11, 25.14, 26.2, 26.6).
7) I have visited the site property that is the subject of this report from June 10, 2025 to June 12, 2025. The purpose of the visit was to integrate the various topographic elements into the mine design and to understand the design limitations and the complexity of the area.
8) As of the effective date of the Technical Report, and to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
10) I have not had prior involvement with the property that is the subject of the Technical Report.
11) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 14th day of November 2025,

/signed and sealed/

_____

Carl Michaud P.Eng., MBA.
Vice President of Mining Engineering
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Pascal Droz**, do hereby certify that:

1) I am currently employed as E&I Engineering Director and Project Manager with G Mining Services Inc. in an office located at 5025, Lapinière Blvd., Suite 1010, Brossard, Québec, J4Z 0N5.
2) I have graduated from Sherbrooke University, Canada with a B.Sc. in Electrical Engineering in 2006.
3) I am a Professional Engineer registered with the Ordre des Ingénieurs du Québec, (OIQ Licence: 143145), and Professional Engineers of Ontario, (PEO licence: 100570076).
4) I have practiced my profession continuously since my graduation from university, and more precisely in the mining industry for the last eight years and an additional three years, earlier in my career. I have been involved in mining construction, engineering and financial evaluations for a total of ten years, including process plant and infrastructure cost design and cost evaluation and underground mine electrical design.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.18, 1.24, 1.26, 18.1, 18.2, 18.3, 18.4, 18.5.1, 18.8, 18.9, 21.1, 24.1, 24.2, 25.1, 25.9, 25.13, 25.14, 26.4.1, 26 (introduction only), 26.4.8, 26.4.9, 26.4.10 and 26.6.
7) I have visited the site property that is the subject of this report from June 10, 2025 to June 12, 2025. The purpose of the visit was to the integration of the projected surface infrastructure and roads according to current topography and water bodies.
8) As of the effective date of the Technical Report, and to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Pascal Droz, P.Eng.,
E&I Engineering Director and Project Manager
G Mining Services

144 Pine Street, Unit 501
Sudbury, ON P3C 1X3
T +1 705.265.1119

BBAconsultants.com

CERTIFICATE OF QUALIFIED PERSON

Todd McCracken, P.Geo.

This certificate applies to the NI 43-101 Technical Report titled "*CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report Shaakichiuwaanaan Project*" (the "Technical Report"), prepared for PMET Resources, with an effective date of October 20, 2025.

I, Todd McCracken, P.Geo., as a co-author of the Technical Report, do hereby certify that:

1. I am Senior Geologist and Director of Mining and Geology at BBA Inc., located at 144 Pine St., Unit 501, Sudbury, ON, P3C 1X3.
2. I am a graduate from University of Waterloo, Ontario, in 1992, with a bachelor's degree in Honors Applied Earth Sciences. I have practised my profession continuously since my graduation.
3. I am a member in good standing of Association of Professional Geoscientists of Ontario and License (PGO No. 0631) and *Ordre des Géologues du Québec* (OGQ No. 02371).
4. My relevant experience includes over 30 years in exploration, operations and consulting, including resource estimation on LCT pegmatite deposits since 2011.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of 1.4 to 1.11, 1.13, 1.22, 1.23, 3.1, 3,3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24.4, 25.2, 26. of the Technical Report.
8. I have visited the Shaakichiuwaanaan Property (previously known as Corvette) from June 4 to 7, 2024 as part of this current mandate, and previously from April 7 to 11, 2023.
9. I have prior involvement with the Property that is the subject of the Technical Report as I have participated as QP on the previous repors titled "Mineral Resource Estimate for the CV5 Pegmatite, Corvette Property" dated September 8, 2023, the "Preliminary Economic Assessment for the Shaakichiuwaanaan Project" dated September 12, 2024 and "Mineral Resource Estimate for the Shaakichiuwaanaan Project, Jame Bay Region, Quebec, Canada" dated August 28, 2025.
10. I have read NI 43-101, and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 14th day of November 2025.

/signed and sealed/

Todd McCracken, P.Geo.
BBA Inc.

PRIMERO

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Ryan Cunningham**, do hereby certify that:

1) I am an employee with the consulting firm Primero Group Americas Inc., located at 1801, McGill College, #1450, Montréal, Québec, H3A 2N4.
2) I am a graduate from McGill University in Montréal in 2006 with a B.Eng. in Metals and Materials Engineering, and in 2009 with a M.Eng. in Mineral Processing.
3) I am a member in good standing of the "Ordre des Ingénieurs du Québec" (OIQ No. 145792). I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum.
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have worked as a professional for a total of nineteen (19) years since graduating from university. My expertise was acquired while working as a process engineer in engineering consulting firms.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I am author and responsible for the preparation of Sections 13 and 17. I am also co-author for the following subsections 1.13, 1.17, 13, 17, 18.5 (except 18.5.1, 18.5.6), 21.3.4, 25.5, 26.3 of the Technical Report.
7) I have not visited the Shaakichiuwaanaan Property that is the subject of the Technical Report. I have visited SGS's Lakefield Ontario facility and while there, I have witnessed testwork being performed on material from the Shaakichiuwaanaan Project.
8) I have prior involvement with the Property that is the subject of the Technical Report as I have participated as QP on the previous report titled "Mineral Resource Estimate for the CV5 Pegmatite, Corvette Property" dated September 8, 2023, the "Preliminary Economic Assessment for the Shaakichiuwaanaan Project" dated September 12, 2024, and the "Mineral Resource Estimate for the Shaakichiuwaanaan Project, James Bay Region, Québec, Canada" dated August 28, 2025.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
10) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
11) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Ryan Cunningham, P.Eng., M.Eng.
Process Engineering Consultant
Primero Americas Inc.

CERTIFICATE OF QUALIFIED PERSON
SEBASTIEN GUIDO

This certificate applies to the technical report entitled, "CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, Sebastien Guido, P.Eng., do hereby certify that:

1) I am currently a Senior Engineer, Rock Mechanics for Alius Mine Consulting Inc., located at 985, rue du Mont-Saint-Denis, Quebec, Quebec, G1S 1B4.

2) I am a graduate of Université Laval, in Québec City, Québec, Canada in Mining Engineering (Bachelor of Engineering). I also obtained a Master of Science from the same university.

3) I am a professional engineer in good standing with the *Ordre des ingénieurs du Québec* (OIQ no. 5067847).

4) My relevant experience includes eleven years in various roles, including a site geotechnical engineer, a consulting engineer and in academia. I have actively worked in the mining industry since 2014 and have experience in geotechnical studies for various commodities in Canada, United States, Guyana and Mexico. Prior relevant experience involves supporting many active mines, and participating in geotechnical studies for the following mines and projects:

    - Cariboo Gold Project in British Columbia
    - Eleonore Mine in Quebec
    - Mountain View and Wildcat Projects in Nevada
    - Westwood Mine in Quebec

5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections related of the Technical Report: 16.2 and 26.2.

7) I visited the Shaakichiuwaanaan Project between October 22, 2024, and October 25, 2024; for a visit duration of 4 days. The purpose of the visit was to review core conditions and conduct geomechanical core logging.

8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.

9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Sebastien Guido, P.Eng., M.Sc.
Senior Engineer, Rock Mechanics
Alius Mine Consulting

990 de l'Église Road, Office 590
Québec, QC G1V 3V7
T +1 418.657.2110
F +1 450.464.0901
BBA.CA

CERTIFICATE OF QUALIFIED PERSON

Hugo Latulippe, P.Eng.

This certificate applies to the NI 43-101 Technical Report titled "CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Hugo Latulippe**, P.Eng., B.Sc.A., as a co-author of the Technical Report, do hereby certify that:

1. I am a Senior Engineer at BBA Inc., located at 990 de l'Église Road, Office 590, Québec, QC, G1V 3V7.
2. I am a graduate of Mining and Mineralogy Engineering at Laval University, Québec, Québec, Canada, 2001.
3. I am a member of the *Ordre des ingénieurs du Québec* (OIQ 126558), Professional Engineers Ontario (PEO No. 100520994) and Engineers and Geoscientists British Columbia (No 209460).
4. I have been working in the mining industry since 2001. I began as a mining engineer in underground mines in Abitibi and then worked in open pit operations in James Bay and New-Caledonia. I acquired solid experience in mining operations before working on the development of three projects. I have been involved in mining studies since 2012, including five lithium deposits.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapter 1.19, 19 and 25.12.
8. I have visited the Shaakichiuwaanaan Property (previously known as Corvette) from September 3 to 10, 2024.
9. I have prior involvement with the Property that is the subject of the Technical Report as I have participated as QP on the previous report titled "Preliminary Economic Assessment for the Shaakichiuwaanaan Project" dated September 12, 2024.
10. I have read NI 43-101, and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 14th day of November 2025

/signed and sealed/

_____

Hugo Latulippe, P.Eng., B.Sc.A.
BBA Inc.

Paterson & Cooke

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Ryan Smilovici**, do hereby certify that:

1) I am currently employed as a Process Engineer with Paterson & Cooke Canada Inc. in an office located at 3161-B Herold Drive, Greater Sudbury, ON, P3E 6K9, Canada.
2) I have graduated from the University of Waterloo, Canada with a B.A.Sc. in Chemical Engineering in 2020.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 6070315), Professional Engineers Ontario (PEO Licence: 100557999), and the Association of Professional Engineers & Geoscientists of Saskatchewan (APEGS Licence: 78821).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 7 years, including plant design, slurry pump and piping system design, and backfill distribution system design.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 16.5.7.4 and 18.5.6.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Ryan Smilovici, P.Eng.,
Process Engineer
Paterson & Cooke Canada Inc.

AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Philip Addis**, do hereby certify that:

1) I am currently employed as a Principal Tailings Engineer at AtkinsRéalis Canada Inc., in offices located at 191,The West Mall, Etobicoke, Ontario, M9C 5K8.
2) I have graduated from the University of the Witwatersrand, South Africa with a Bachelor of Science in Engineering, Civil in 1992 and a Master of Science in Engineering, Civil in 1997.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 6076892), and in the province of Ontario, Canada (PEO: Licence: 100193342).
4) I have practiced my profession continuously in the mining industry since 2003. I have been involved in mining operations, engineering and financial evaluations for 22 years, including various aspects of tailings and mine waste management.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections 18.6, 18.7.2.1.4 to 18.7.2.1.6, 20.4, and 25.7.1.
7) I visited the site property that is the subject of this report from October 16th to 18th, 2024. The purpose of the visit was to review the site conditions and the footprints of the proposed waste stockpiles.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Philip Addis, P.Eng.,
Principal Tailings Engineer
AtkinsRéalis

AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Antoine Cogulet**, do hereby certify that:

1) I am currently employed as Water Management Team Lead at AtkinsRéalis, in offices located at 200-5500, Boulevard Des Galeries, Quebec, QC, Canada, G2K 2E2.
2) I have graduated from Université de Tours, France, with a Bachelor of Applied Science in biology in 2012 and from Université de Limoges, France with a master's degree in chemical engineering in 2014 and from Université Laval, Canada with a Ph.D in wood sciences in 2018.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 6028243).
4) I have practiced my profession in the mining industry since my graduation from university. I have been involved in mining projects since 2019. Over these years, I have been actively involved in mining operations, including water treatment and water management design as well as water balance using Goldsim. I have relevant experience with various technologies of water treatment including biological water treatment, SBR and physico-chemical water treatment, coagulation, flocculation, decantation.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 18.7 (excluding: hydrology and hydraulics; and including: pumping, piping, treatment plants and side wide water balance), 20.7.2, 26.4.3 and 26.4.4.
7) I have visited that is the subject of this report in June 2025. The purpose of the visit was to conduct site reconnaissance and to familiarize myself with the site's hydrography and topography.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Antoine Cogulet, Ing. Ph.D.,
Team Lead, Mine Water Management
AtkinsRéalis

AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Holman Tellez**, do hereby certify that:

1) I am currently employed as Sr. Expert Hydraulics/Hydrology Engineer at AtkinsRéalis, in offices located at 191, The West Mall, Toronto, ON, M9C 5K8, Canada.
2) I have graduated from La Salle University, Bogota, Colombia, with a B.Sc. in Civil Engineering in 1998, and from the University of Los Andes, Bogota, Colombia, with a M.Sc. in Civil Engineering in 2002.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 6083507), and Professional Engineers of Ontario (PEO Licence: 100166945).
4) I have continuously practiced my profession in the mining industry from 2011 to 2013 and since 2018. Over these years, I have been actively involved in mining operations and water management engineering design, from conceptual to final stages.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections 18.7 (excluding: pipe and pump system design and treatment plants; and including: site water balance), 20.5, and 26.4.3.
7) I visited the property that is the subject of this report from October 16 to 18, 2024. The primary objective of this visit was to carry out a thorough reconnaissance of the areas designated for the water management infrastructure.
8) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101, and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of Nov. 2025

/signed and sealed/

_____

Homan Tellez MSc, P.Eng.,
Sr Expert Hydraulics/Hydrology
AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Nathalie Fortin**, do hereby certify that:

1) I am currently employed as Vice President, Environmental Management with WSP Canada Inc. in an office located at 1600, Rene Levesque West, Montreal, Quebec, H3P 1P9.
2) I have graduated from University of Sherbrooke, Quebec, Canada with a B.Sc.A. in Chemical Engineering in 1993, and from University of Sherbrooke, Quebec, with a M.Env. in Environment in 1999.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 112062).
4) I have practiced my profession continuously in different industrial sectors, mining industry and engineering and environmental consultation since my graduation from university. I have been involved in environmental management for 32 years, including environmental and social impact assessment of major industrial and mining projects and completion of the environmental and social aspects of feasibility studies.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision of the following sections and sub-sections: 20 (excluding 20.1.1.7, 20.4, 20.5, 20.6 and 20.7) and 26.5.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

Nathalie Fortin, Eng.,M. Env.
Vice President, Environmental Management
WSP Canada

AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Geneviève Marchand**, do hereby certify that:

1) I am currently employed as Mining Hydrogeology Specialist at AtkinsRéalis, in offices located at 455, René-Lévesque Boulevard West, Montreal, Quebec, H2Z 1Z3.
2) I have graduated from Université de Sherbrooke, Sherbrooke, Canada with a B.Sc. in Civil Engineering in 1998, and from University of Calgary, Calgary, Canada with a M.Sc. in Civil Engineering (Engineering for the Environment) in 2002.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 127590).
4) I have practiced my profession in the mining industry from 2005 to 2017 and since 2023 and in hydrogeology since 2001. Over these years, I have been involved in mining hydrogeology and environmental projects including hydrogeology baseline, mine dewatering and depressurization, development of hydrogeological conceptual model, seepage assessment from tailings storage facility and supervision of numerical model for tailings management and mine dewatering.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 20.1.1.7, 20.6, 25.6 and 26.4.2.
7) I have visited the site property that is the subject of this report from June 10 to June 12, 2025. The purpose of the visit was to conduct site reconnaissance and to familiarize myself with the site's hydrography, topography and geology.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Geneviève Marchand, P.Eng., M.Sc., PMP
Mining Hydrogeology Specialist
AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Neal Sullivan**, do hereby certify that:

1) I am currently employed as President & Principal Geochemist at Vision Geochemistry Ltd, located at 44, Patricia Street, Sudbury, Ontario, P3Y 1B1.
2) I have graduated from Laurentian University, Ontario, Canada with a B.Sc. in Geology in 2014, from University of Toronto, Ontario, Canada with a M.Sc. in Earth Sciences (Geochemistry) in 2015; and from University of Toronto, Ontario, Canada, with a Ph.D. in Earth Sciences (Geochemistry) in 2020.
3) I am a Professional Geoscientist registered with the Ordre des géologues du Québec, (OGQ Licence: 02399), and with the Professional Geoscientists Ontario (PGO License: 3552). I am a member of the Association of Applied Geochemists.
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have 11 years of combined experience in research and applied geochemistry, including five years specializing in environmental geochemistry for mining projects. My experience includes the characterization of waste rock and tailings materials, acid-base accounting, static and kinetic leach testing programs, and the modelling and prediction of drainage chemistry.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 20.7 and 26.4.5.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Neal Sullivan, Ph.D., P.Geo.
President & Principal Geochemist
Vision Geochemistry Ltd

AtkinsRéalis

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Sandra Pouliot**, do hereby certify that:

1) I am currently employed as Senior Mining Environment Engineer at AtkinsRéalis Canada Inc., in offices located at 455, René-Lévesque Boulevard West, Montreal, Quebec, H2Z 1Z3.
2) I have graduated from Université Laval (Québec, Canada) with a B.Sc. in Geological Engineering in 1998, and from Polytechnique -Université de Montréal (Montréal, Canada) with a M.Sc.A. in 2019.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 121003), and a member of the Engineers and Geoscientists of British Columbia (No. 59969).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 27 years, including environmental coordination on mine site, project management, engineering studies (conceptual to detailed engineering) for mine closure work, tailings and water management studies.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 20.8 (excluding 20.8.5), 21.2, 25.7.4, and 26.4.6.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43--101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Sandra Pouliot, P.Eng., M.A.Sc.,PMP
Title : Senior Mining Environment Engineer
AtkinsRéalis

MAILLOUX
Hydrogéologie inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Michel Mailloux**, do hereby certify that:

1) I am currently employed as Hydrogoelogist with Mailloux Hydrogéologie inc. in an office located at 7924, rue Saint-Gérard, Montréal, Québec, H2R 2K7. I am also the owner of this company.
2) I have graduated from Laval University, Québec, Canada with a B.Sc.A in Geological Engineering in 1998, and from INRS-Géoressources, Québec, Canada, with a M.Sc. in Earth Sciences in 2002.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 126 263) and Professional Geologist with the Ordre des géologues du Québec (OGQ license: 2323).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in consulting in hydrogeology for 25 years.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following section: 16.3.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Michel Mailloux, Eng., Geo., MSc.
Hydrogeologist
Mailloux Hyxrogéologie inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"CV5 Pegmatite Lithium-Only Feasibility Study NI 43-101 Technical Report – Shaakichiuwaanaan Project", prepared for PMET Resources Inc. with an effective date of October 20, 2025 and a report date of November 14, 2025 (the "Technical Report").

I, **Charles Gagnon**, do hereby certify that:

1) I am currently employed as Principal Mine Ventilation Engineer with CGM Expert Inc. in an office located at 1155, avenue des Érables, Quebec, G1R 2N4.
2) I have graduated from Laval University with a B.Sc. in Mining Engineering in 2002, and with a M.Sc.. in Mining Ventilation in 2005.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec (OIQ Licence 130730) and Professional Engineers of Ontario (PEO licence 40041012).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 20 years, including mineral reserve estimations, mine scheduling, cost evaluation and underground mine ventilation design and modeling.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 16.5.6.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 14th day of November 2025

/signed and sealed/

_____

Charles Gagnon, P.Eng.,
Principal Mine Ventilation Engineer
CGM Expert Inc.

Table of Contents

1. SUMMARY 1-1
1.1 Introduction 1-1
1.2 Terms of Reference 1-3
1.3 Reliance on Other Experts 1-3
1.4 Property Description and Location 1-4
1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography 1-5
1.6 History 1-6
1.7 Geological Setting and Mineralization 1-7
1.8 Deposit Types 1-9
1.9 Exploration 1-9
1.10 Drilling 1-11
1.11 Sample Preparation, Analyses and Security 1-12
1.12 Data Verification 1-13
1.13 Mineral Processing and Metallurgical Testing 1-13
1.13.1 Metallurgical Test Work 1-13
1.14 Mineral Resource Estimate 1-18
1.15 Mineral Reserve Estimate 1-19
1.16 Mining Methods 1-20
1.16.1 Open Pit 1-24
1.16.1.1 West Pit 1-24
1.16.1.2 East Pit 1-25
1.16.2 Underground 1-25
1.17 Recovery Methods 1-26
1.18 Project Infrastructure 1-27
1.19 The Project's Market Studies and Contracts 1-28
1.20 Environmental Studies, Permitting and Social or Community Impact 1-29
1.20.1 Introduction 1-29
1.20.2 Geochemistry 1-30
1.20.3 Reclamation and Closure 1-31
1.20.4 Monitoring and Maintenance Programs 1-32
1.21 Capital and Operating Costs 1-33
1.21.1 Capital Cost 1-33
1.21.2 Operating Cost 1-35
1.22 Economic Analysis 1-36

1.23 Adjacent Properties 1-37
1.24 Organization and Project Schedule 1-38
1.24.1 Organization 1-38
1.24.2 Project Schedule 1-38
1.25 Interpretations and Conclusions 1-39
1.25.1 Project Risks and Opportunities 1-39
1.26 Recommendations 1-40
2. INTRODUCTION 2-1
2.1 QP Responsibility 2-3
2.2 Effective Date 2-5
2.3 Previous Technical Reports 2-5
2.4 Use of Non-GAAP Financial Measures 2-5
2.5 Sources of Information 2-6
2.6 Agreements, Mineral Tenure, Surface Rights and Royalties 2-6
2.7 Site Visit 2-7
2.8 List of Abbreviations and Units of Measurement 2-7
3. RELIANCE ON OTHER EXPERTS 3-1
3.1 Introduction 3-1
3.2 Taxation 3-1
3.3 Mineral Tenure and Surface Rights 3-2
4. PROPERTY DESCRIPTION AND LOCATION 4-1
4.1 Location 4-1
4.2 Mineral Disposition 4-2
4.3 Tenure Rights 4-3
4.4 Royalties and Related Information 4-10
4.5 Permits 4-12
4.6 Environmental Liabilities 4-13
4.7 Other Relevant Factors 4-13
5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 5-1
5.1 Access 5-1
5.2 Climate 5-2
5.3 Local Resources 5-2
5.4 Infrastructure 5-3
5.5 Physiography 5-4

6. HISTORY 6-1
6.1 Prior Ownership 6-1
6.2 Previous Exploration and Development 6-2
6.3 Historical Mineral Resources 6-6
6.4 Production 6-6
7. GEOLOGICAL SETTING AND MINERALIZATION 7-1
7.1 Regional Geology 7-1
7.2 Property Geology 7-3
7.3 Structural Geology 7-6
7.4 Mineralization 7-7
7.4.1 CV Trend (LCT Pegmatite) 7-7
7.4.1.1 CV5 Pegmatite 7-10
7.4.1.2 CV13 Pegmatite 7-20
7.4.1.3 Other LCT Pegmatites 7-29
7.4.2 Maven Trend (Copper, Gold, Silver) 7-29
7.4.3 Golden Trend (Gold) 7-29
8. DEPOSIT TYPES 8-1
9. EXPLORATION 9-1
9.1 2017 Through 2020 9-1
9.2 2021 9-2
9.3 2022 9-3
9.4 2023 9-4
9.5 2024 9-5
9.6 2025 9-5
9.7 Lithium Pegmatite Surface Sampling Summary 9-6
9.7.1 CV5 Pegmatite 9-6
9.7.2 CV13 Pegmatite 9-6
9.7.3 CV4 Pegmatite 9-6
9.7.4 CV8 Pegmatite 9-6
9.7.5 CV9 Pegmatite 9-7
9.7.6 CV10 Pegmatite 9-7
9.7.7 CV12 Pegmatite 9-7
9.7.8 CV14 Pegmatite 9-7
9.7.9 CV15 Pegmatite 9-7
10. DRILLING 10-1
10.1 Drilling Campaigns 10-1
10.1.1 2021 Drill Program 10-6
10.1.2 2022 Drill Program 10-6

10.1.3 2023 Drill Program 10-8
10.1.3.1 CV5 Pegmatite 10-8
10.1.3.2 CV13 Pegmatite (Lithium) 10-9
10.1.3.3 CV13 Pegmatite (Caesium) 10-10
10.1.3.4 CV9 Pegmatite 10-11
10.1.4 2024 Drill Program 10-11
10.1.4.1 CV5 Pegmatite 10-12
10.1.4.2 CV13 Pegmatite (Lithium) 10-13
10.1.4.3 CV13 Pegmatite (Caesium) 10-14
10.1.5 2025 Drilling Program 10-15
10.2 Structure 10-16
10.3 Collar Survey 10-16
10.4 Downhole Deviation Survey 10-16
10.5 Core Logging and Sampling Procedures 10-16
10.5.1 2021 Drill Program 10-17
10.5.2 2022, 2023, 2024, & 2025 Drill Programs 10-18
10.6 Qualified Person's Opinion 10-20
11. SAMPLE PREPARATION, ANALYSES AND SECURITY 11-1
11.1 Sample Preparation 11-1
11.1.1 Rock and Channel Sampling Programs 11-1
11.1.2 2024 Drilling Program (April 2024 to December 2024, Holes CV24-527 to CV24-787) 11-2
11.2 Analytical Procedure 11-2
11.2.1 2024 Drill Program 11-2
11.2.2 Previous Years 11-2
11.3 Quality Assurance / Quality Control 11-3
11.3.1 Channel Sample Program 11-3
11.3.2 2021 to 2024 Drilling Programs 11-3
11.3.3 2024 Drilling Program (Holes CV24-527 to CV24-787) 11-4
11.3.3.1 Blanks 11-4
11.3.3.2 Certified Reference Materials 11-7
11.3.3.3 Pulp Duplicates 11-11
11.3.3.4 External Pulp Duplicates (Secondary Lab Check) 11-15
11.4 Sample Security 11-18
11.5 Qualified Person's Opinion 11-19
12. DATA VERIFICATION 12-1
12.1 Geology 12-1
12.1.1 Site Investigation 12-1
12.1.2 Drill Collar Validation 12-2
12.1.3 Database Validation 12-3
12.1.4 Mining Validation 12-3
12.1.5 Qualified Person's Opinion 12-3

13. MINERAL PROCESSING AND METALLURGICAL TESTING 13-1
13.1 Introduction 13-1
13.1.1 Testwork Overview 13-1
13.1.2 Test Material 13-3
13.1.2.1 Head Sample Characterization 13-10
13.1.2.2 Mineralogy 13-1
13.1.2.3 Particle Size Distribution 13-2
13.1.2.4 Comminution Tests 13-3
13.1.2.5 Heavy Liquid Separation 13-5
13.1.2.6 Magnetic Separation 13-9
13.1.2.7 Dense Media Separation 13-11
13.1.2.8 Recrush Heavy Liquid Separation 13-16
13.1.2.9 Flotation 13-17
13.1.2.10 Filtration 13-19
13.1.2.11 Gravity Separation 13-20
13.1.2.12 Lithium Hydroxide Conversion Test 13-22
13.2 Basis for Recovery and Throughput Estimates 13-24
13.3 Processing Factors and Deleterious Elements 13-26
13.4 Conclusions 13-28
13.5 Recommendations 13-29
14. MINERAL RESOURCE ESTIMATES 14-1
14.1 Resource Estimate 14-1
14.1.1 Database 14-1
14.1.2 Specific Gravity 14-1
14.1.3 Topography Data 14-4
14.1.4 Geological Interpretation 14-4
14.1.4.1 CV5 14-5
14.1.4.2 CV13 14-6
14.1.5 Exploratory Data Analysis 14-6
14.1.5.1 Assays 14-6
14.1.5.2 Compositing 14-12
14.1.5.3 Grade Capping 14-17
14.1.5.4 Spatial Analysis 14-21
14.1.5.5 Resource Block Model 14-24
14.1.5.6 Estimate Parameters for CV5 14-25
14.1.5.7 Estimate Parameters for CV13 14-27
14.1.6 Resource Classification 14-30
14.1.7 Mineral Resource Tabulation 14-31
14.1.7.1 CV5 – MRE Details 14-32
14.1.7.2 CV13 – MRE Details 14-35
14.1.8 Model Validation 14-40
14.1.8.1 Visual Validation 14-40
14.1.8.2 Statistics Comparison 14-43
14.1.8.3 Swath Plots 14-44

14.1.9 Sensitivity Analysis 14-47
14.1.10 Previous Estimates 14-50
15. MINERAL RESERVE ESTIMATES 15-1
15.1 Introduction 15-1
15.2 Open Pit 15-2
15.2.1 Mineral Resource Block Model 15-2
15.2.2 Open Pit Optimization 15-2
15.2.3 Slope Recommendations 15-3
15.2.4 Mining Dilution and Mining Recovery 15-4
15.2.5 Pit Optimization Parameters and Cut-Off-Grade 15-5
15.2.6 Pit Optimization Results 15-6
15.2.6.1 West Pit 15-7
15.2.6.2 East Pit 15-12
15.2.7 Mine Design 15-17
15.2.7.1 Ramp Design Criteria 15-17
15.2.8 Open Pit Mine Reserves 15-18
15.3 Underground Mining 15-19
15.3.1 Mineral Reserve Calculation Methodology 15-19
15.3.2 Dilution and Mining Losses 15-20
15.3.3 Underground Cut-Off Grades and Optimization Parameters 15-22
15.3.4 Underground Mineral Reserve 15-24
15.3.5 Factors Which May Affect Mineral Reserves 15-25
16. MINING METHODS 16-1
16.1 Summary 16-1
16.2 Geotechnical Considerations 16-2
16.2.1 Geotechnical Field Campaigns 16-2
16.2.2 Soil and Rock Mass Characterization 16-4
16.2.3 Stress State 16-6
16.2.4 Open Pit Geotechnical Considerations 16-6
16.2.4.1 Open Pit Geotechnical Domains 16-7
16.2.4.2 Slope Stability Analyses 16-8
16.2.4.3 Slope Design Guidelines 16-9
16.2.5 Underground Geotechnical Considerations 16-12
16.2.5.1 Underground Geotechnical Domains 16-12
16.2.5.2 Stope Sizing and Dilution Estimates 16-12
16.2.5.3 Backfill 16-13
16.2.5.4 Ground Support 16-14
16.2.5.5 Mine Design Considerations 16-17
16.3 Hydrology and Hydrogeology 16-19
16.3.1 Hydrology 16-19
16.3.2 Hydrogeology 16-20
16.3.2.1 Crown Pillar 16-21
16.4 Open Pit Mine 16-22

16.4.1 Mine Phases 16-22
16.4.2 Waste Rock Storage Facility 16-27
16.4.3 Ore Stockpiles 16-28
16.4.4 Mine Haul Roads 16-29
16.4.5 Open Pit Production Schedule 16-29
16.4.6 Mine Operations and Equipment Selection 16-37
16.4.6.1 Drilling and Blasting 16-37
16.4.6.2 Loading 16-38
16.4.6.3 Hauling 16-40
16.4.6.4 Support Operations 16-42
16.4.6.5 Automation 16-42
16.4.7 Open Pit Mine Dewatering 16-43
16.4.8 Controlled Blasting Techniques and Mitigation of Environment Disturbance 16-44
16.4.8.1 Fish Habitat Impact 16-45
16.4.8.2 Vibration and Air Blasts 16-47
16.4.8.3 Fly Rock 16-47
16.4.9 Mining Fleet Requirements 16-47
16.4.10 Mobile Crushing Plant 16-52
16.4.11 Mine Maintenance 16-52
16.4.12 Pit Slope Monitoring 16-52
16.5 Underground 16-52
16.5.1 Underground Mining Method 16-52
16.5.1.1 Mining Recovery 16-57
16.5.2 Underground Mine Design 16-58
16.5.2.1 Development Design 16-58
16.5.2.2 Stope Design 16-68
16.5.2.3 Physicals Summary 16-78
16.5.2.4 Development and Production Rates 16-79
16.5.3 Development and Production Sequencing 16-80
16.5.4 Material Handling 16-85
16.5.5 Underground Mine Equipment 16-86
16.5.5.1 Battery Electric Vehicles (BEVs) Infrastructure and Operating Conditions 16-91
16.5.5.2 Automation 16-93
16.5.6 Underground Mine Ventilation 16-94
16.5.6.1 Underground Mine Heating 16-100
16.5.7 Underground Mine Services 16-101
16.5.7.1 Underground Electrical Distribution 16-101
16.5.7.2 Industrial Water 16-103
16.5.7.3 Dewatering Underground 16-103
16.5.7.4 Cemented Paste Backfill 16-107
16.5.7.5 Compressed Air 16-108
16.5.7.6 Communication 16-109
16.5.7.7 Storage and Warehouse 16-110
16.5.7.8 Fuel Storage and Distribution 16-111
16.5.7.9 Explosives Storage and Handling 16-111

16.5.7.10 Personnel and Underground Material Transportation 16-112
16.5.7.11 Equipment Maintenance 16-113
16.5.7.12 Mobile Crushing Plant 16-114
16.5.7.13 Surface Portal 16-114
16.5.8 Underground Mine Safety Measures 16-116
16.5.8.1 Tag Boards 16-116
16.5.8.2 Emergency Exits 16-116
16.5.8.3 Refuge Stations 16-117
16.5.8.4 Mobile Equipment Safety 16-118
16.5.8.5 Mine Rescue 16-118
16.5.8.6 Emergency Stench System 16-119
16.6 Mine Workforce 16-119
16.6.1 Open Pit Mine Workforce Requirements 16-120
16.6.2 Underground Mine Workforce Requirements 16-125
16.7 Combined Production 16-131
17. RECOVERY METHODS 17-1
17.1 Mineral Processing Facility Design 17-1
17.2 Design Criteria 17-1
17.2.1 Recovery 17-3
17.2.2 Mass Balance 17-4
17.2.3 Water Balance 17-4
17.2.4 Reagents 17-7
17.2.5 Utilities 17-7
17.2.5.1 Raw Water 17-7
17.2.5.2 Process Water 17-7
17.2.5.3 Water Treatment 17-7
17.3 Mineral Processing Facility Description 17-7
17.3.1 Run-of-Mine Stockpiles 17-9
17.3.2 Crushing Circuit 17-9
17.3.3 Fine Ore Stockpile 17-10
17.3.4 Dense Media Separation Preparation Circuit 17-10
17.3.5 Coarse Dense Media Separation Circuit 17-11
17.3.6 Fine Dense Media Separation Circuit 17-11
17.3.7 Ultrafine Dense Media Separation Circuit 17-11
17.3.8 Recrush Dense Media Separation Circuit 17-11
17.3.9 Tailings (Bypass) Dewatering 17-12
17.3.10 Tailings Handling 17-12
17.3.11 Magnetic Separation and Final Product Handling 17-12
17.4 Recommendations 17-12
18. PROJECT INFRASTRUCTURE 18-1
18.1 Project Overview 18-1
18.2 Site Infrastructure 18-1
18.2.1 Layout 18-3

18.2.2 Access Roads and On-Site Roads 18-4
18.2.3 Haul Roads 18-5
18.3 Camp Infrastructure 18-5
18.3.1 Dormitories 18-5
18.3.2 Kitchen and Lunchroom 18-6
18.3.3 Camp Office and Welcome Centre 18-7
18.3.4 Fire Hall and Clinic 18-9
18.3.5 First Nations Cultural Centre 18-10
18.4 Mine Infrastructure 18-10
18.4.1 Mine Portal Area 18-10
18.4.2 Ventilation Air Raises 18-11
18.4.3 Underground Services 18-13
18.4.4 Mine Garage and Wash Bay 18-14
18.4.4.1 Temporary Truck Shop 18-16
18.4.5 Fuel Storage and Distribution 18-17
18.4.6 Compressed Natural Gas (CNG) 18-18
18.4.7 Mine Dry and Mine Office 18-18
18.4.8 Warehouse 18-19
18.4.9 Explosive Storage 18-22
18.5 Process Infrastructure 18-24
18.5.1 Mill Office and Laboratory 18-24
18.5.2 Process Buildings 18-25
18.5.3 Concentrate Load-Out 18-28
18.5.4 Tailings Load-Out 18-29
18.5.5 Paste Preparation Plant 18-29
18.5.6 Paste Backfill Plant 18-30
18.6 Tailing Storage Facility 18-31
18.6.1 Tailings Management 18-31
18.6.2 Material Production and Characteristics 18-33
18.6.3 Site Description of Tailings Storage 18-34
18.6.4 Tailings Storage Description and Related Infrastructure 18-36
18.6.5 Stability Analysis for Tailings Storage 18-38
18.6.6 Mine Waste Rock and Overburden Management 18-41
18.6.7 General Design Considerations 18-42
18.6.8 Material Production and Characteristics 18-43
18.6.9 Site Description for Waste Rock and Overburden Storage 18-47
18.6.10 Waste Rock Stockpiles and Related Infrastructure 18-48
18.6.11 Stability Analysis 18-51
18.7 Water Management Infrastructure 18-54
18.7.1 Contact Water Management 18-54
18.7.1.1 General 18-54
18.7.1.2 Contact Water Collection Ditches 18-56
18.7.1.3 Contact Water Collection Ponds 18-57
18.7.1.4 Industrial Area Water Treatment Infrastructure 18-60
18.7.1.5 Pumping Stations 18-62

18.7.1.6 Annual Contact Water Balance 18-62
18.7.2 Non-Contact Water Management 18-64
18.7.2.1 Lake 001 Water Management 18-64
18.7.2.2 Non-Contact Water Diversion Ditches for Stockpile 002 18-72
18.7.3 Fresh Water Intake and Potable Water Production 18-73
18.7.4 Sewage Water Treatment 18-73
18.7.5 Water Treatment 18-74
18.7.5.1 General 18-74
18.7.5.2 Water Treatment Plant Process – Contact Water with Trace Metals 18-74
18.7.5.3 Contingency TSS Treatment 18-76
18.7.5.4 Water Treatment Plant – Opportunity 18-76
18.8 Electrical Distribution and Telecom 18-77
18.8.1 Off-Site Transmission System 18-77
18.8.2 On-Site Distribution 18-80
18.8.2.1 Site Main Substation 18-80
18.8.2.2 Site Power Demand and Load Profile 18-82
18.8.2.3 13.8 kV Site Power Distribution and Electrical Rooms 18-84
18.8.2.4 Underground Power Distribution 18-85
18.8.3 Telecommunication 18-86
18.9 Spodumene Concentrate Transportation 18-86
18.9.1 Road Transportation 18-86
18.9.2 Matagami Transshipment Centre (MTC) 18-88
18.9.3 Railroad and Railcars 18-89
18.9.4 Train Unloading Station – Grande-Anse Maritime Terminal, La Baie 18-89
19. MARKET STUDIES AND CONTRACTS 19-1
19.1 Lithium Market Overview 19-1
19.2 Lithium Supply / Demand 19-2
19.2.1 Supply 19-2
19.2.2 Demand 19-4
19.3 Market Balance 19-6
19.4 Lithium Demand Outlook 19-7
19.5 Lithium Spodumene Concentrate Price Assumptions 19-8
19.5.1 Recent Spot Prices 19-8
19.5.2 Benchmark Intelligence 19-8
19.5.3 Recent NI 43-101 Reports & Press Releases 19-9
19.5.4 Commodities Consensus 19-11
19.6 Conclusion 19-11
19.7 Contracts 19-11
20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 20-1
20.1 Environmental Setting 20-1

20.1.1 Physical Environment 20-1
20.1.1.1 Physiography 20-1
20.1.1.2 Climate 20-1
20.1.1.3 Noise 20-1
20.1.1.4 Air Quality 20-2
20.1.1.5 Hydrology 20-3
20.1.1.6 Surface Water and Sediment Quality 20-4
20.1.1.7 Hydrogeology 20-5
20.1.2 Biological Components 20-6
20.1.2.1 Flora 20-6
20.1.2.2 Fish and Fish Habitat 20-7
20.1.2.3 Herpetofauna 20-7
20.1.2.4 Avian Fauna 20-8
20.1.2.5 Bats 20-9
20.1.2.6 Small Mammals 20-9
20.1.2.7 Small Fauna and Fur-bearing Animals 20-10
20.1.2.8 Large Fauna 20-11
20.1.2.9 Species at Risk 20-11
20.1.3 Social Components 20-13
20.1.3.1 Administrative Context 20-13
20.1.3.2 Local Communities 20-15
20.1.3.3 Local Land Use and Dwellings 20-20
20.1.3.4 Economic Activities 20-20
20.1.3.5 Regional Infrastructure 20-21
20.1.3.6 Traditional Land Use 20-23
20.1.3.7 Heritage and Archeology 20-26
20.2 Jurisdiction, Applicable Laws and Regulations 20-27
20.2.1 Provincial Laws and Regulations 20-27
20.2.2 Federal Laws and Regulations 20-28
20.3 Environmental Permitting 20-28
20.4 Ore, Waste Rock and Tailings Management 20-32
20.5 Site-Wide Water Management 20-32
20.6 Baseline Hydrogeology 20-33
20.7 Geochemistry 20-35
20.7.1 Static Testing 20-35
20.7.2 Kinetic Testing 20-36
20.7.3 COPC Screening and Stockpile Modelling 20-37
20.8 Rehabilitation and Closure Planning 20-40
20.8.1 Closure Sequence 20-40
20.8.2 Industrial Zone Closure and Linear Infrastructure 20-41
20.8.3 Mine Infrastructure: Pit, Waste Rock, Tailings and Overburden Storage Facilities 20-41
20.8.3.1 Stockpile 001 20-42
20.8.3.2 Stockpile 002 20-42
20.8.3.3 Organics and Overburden Piles 001, 004 and 005 20-44

20.8.3.4 Open Pit 20-44
20.8.4 Monitoring and Maintenance Programs 20-45
20.8.4.1 Post-Operation 20-45
20.8.5 Cost Estimation and Financial Guarantee 20-47
20.9 GHG Emissions 20-47
20.10 Community Relations 20-47
20.10.1 Consultation Activities 20-47
20.10.1.1 Consultation Process 20-48
20.10.1.2 Main Concerns 20-51
20.10.1.3 Future Engagement Plan 20-54
20.10.2 Stakeholder Engagement Approach 20-56
20.10.3 Agreements 20-57
20.11 Preliminary Environment Monitoring Plan 20-57
20.12 Preliminary Habitat Compensation Plan 20-58
21. CAPITAL AND OPERATING COSTS 21-1
21.1 Capital Expenditures 21-6
21.1.1 Basis of Estimate 21-6
21.1.2 General 21-9
21.1.3 Exclusions 21-9
21.1.4 Initial Capital 21-10
21.1.4.1 Phase 1 Capital Costs 21-10
21.1.4.2 Phase 2 Capital Costs 21-21
21.1.5 LOM Sustaining Capital Expenditures 21-29
21.2 Mine Closure Expenditures 21-32
21.3 Operating Cost 21-35
21.3.1 Summary 21-35
21.3.2 Open Pit Mining 21-37
21.3.2.1 Consumable Costs 21-37
21.3.2.2 Open Pit Operating Costs 21-37
21.3.3 Underground Mine Operating Cost 21-42
21.3.3.1 Summary 21-42
21.3.4 Processing Facility Operating Costs 21-42
21.3.4.1 Basis 21-43
21.3.4.2 Summary 21-43
21.3.4.3 Methodology 21-46
21.3.5 General and Administration Costs 21-48
21.3.6 Energy Costs 21-51
21.3.6.1 Electricity 21-51
21.3.6.2 Compressed Natural Gas 21-53
21.3.7 Concentrate Transportation Costs 21-53
22. ECONOMIC ANALYSES 22-1
22.1 Overview 22-1
22.2 Cautionary Statement 22-2

22.3 Key Assumptions 22-3
22.3.1 Mineral Reserve 22-3
22.3.2 Spodumene Price 22-4
22.3.3 Fuel Price 22-5
22.3.4 Exchange Rate 22-7
22.3.5 Metal Recovery 22-7
22.3.6 Other Assumptions 22-8
22.4 Spodumene Concentrate Production and Revenues 22-8
22.5 Royalties 22-12
22.6 Transportation Loss 22-12
22.7 Capital Expenditures 22-12
22.7.1 Initial Capital 22-14
22.7.2 Expansion and Sustaining Capital 22-16
22.7.3 Working Capital 22-17
22.8 Reclamation & Closure Cost 22-17
22.9 Salvage Value 22-18
22.10 Operating Cost Summary 22-18
22.11 Taxation 22-20
22.12 Economic Results 22-21
22.13 Sensitivity Analysis 22-27
23. ADJACENT PROPERTIES 23-1
24. OTHER RELEVANT DATA AND INFORMATION 24-1
24.1 Project Execution Plan 24-1
24.2 Project Execution Schedule 24-2
24.3 High-Grade Nova Zone 24-5
24.4 Shaakichiuwaanaan Exploration Target 24-5
25. INTERPRETATION AND CONCLUSIONS 25-1
25.1 Summary 25-1
25.2 Geology and Mineral Resources 25-3
25.3 Mineral Reserve Estimates 25-4
25.4 Mining Method 25-5
25.5 Processing 25-7
25.6 Hydrogeology 25-10
25.7 Tailings, Water, Waste Rock and Overburden Management 25-11
25.7.1 Mine Waste Management 25-11
25.7.2 Water Management 25-12
25.7.3 Water Diversion Infrastructure 25-13

25.7.4 Closure....25-13
25.8 Environmental Permitting....25-14
25.9 Capital Cost....25-15
25.10 Operating Cost....25-17
25.11 Economic Analysis....25-17
25.12 Market....25-18
25.13 Risks and Opportunities....25-19
25.14 Conclusions....25-24
26. RECOMMENDATIONS....26-1
26.1 Geology / Exploration....26-1
26.2 Mining....26-2
26.2.1 Open Pit....26-3
26.2.1.1 Blasting Mitigation Near Lake 001 and Permanent Infrastructure....26-3
26.2.1.2 Other Nearby Mineralized Deposits....26-3
26.2.1.3 Include Other Elements in the Blending Schedule....26-3
26.2.1.4 Early Work Mass Balance....26-3
26.2.2 Underground Infrastructure....26-4
26.2.2.1 Bulk Sample Program....26-4
26.2.2.2 Battery Electric Vehicles (BEVs) and Optimization of the Contract Development Phase....26-4
26.2.2.3 Mine Schedule and Design Optimization....26-4
26.2.2.4 Mineralized Areas Under the Open Pit....26-5
26.2.2.5 Portal and Ventilation Raise Location....26-5
26.2.2.6 Mine of the Future....26-5
26.3 Processing....26-6
26.3.1 Metallurgical Testing....26-6
26.3.2 Processing Plant....26-6
26.4 Infrastructure....26-8
26.4.1 Main Site Layout – General....26-8
26.4.2 Hydrogeology....26-8
26.4.3 Water Management....26-8
26.4.4 Water Treatment....26-10
26.4.4.1 Potable Water....26-10
26.4.4.2 Sewage System....26-11
26.4.5 Geochemical Testing and Modelling....26-11
26.4.6 Mine Closure....26-12
26.4.7 Geotechnical....26-13
26.4.8 Main Site Access Road....26-14
26.4.9 Power....26-14
26.4.10 Concentrate Transportation....26-14
26.4.10.1 Road....26-14
26.4.10.2 Matagami Concentrate Transloading Site....26-14

26.4.10.3 Concentrate Transportation Optimization 26-14
26.4.10.4 Decarbonation 26-15
26.5 Environmental Fieldwork and Studies 26-15
26.6 Detail Engineering 26-15
27. REFERENCES 27-1

List of Figures

Figure 1.1: Metallurgical Drill Core Map for CV5 1-14
Figure 1.2: Metallurgical Testwork Recovery Results & Industry-Based Recovery Estimates for 3x Size Range DMS Process Plant 1-16
Figure 1.3: Final Concentrate Product Generated from the CV5 Pegmatite via DMS 1-17
Figure 1.4: Open Pit Phase Limits – Plan View 1-21
Figure 1.5: Underground Mine and Surfaces Constraints – Plan View 1-22
Figure 1.6: CV5 Open Pit and Underground – Total Material Mined 1-23
Figure 1.7: Concentrate Production Schedule 1-23
Figure 1.8: Tornado Graph for the Sensitivity Analysis (+/-20%) 1-37
Figure 1.9: Timeline for Construction Phases 1 and 2 1-39
Figure 4.1: Property Location 4-2
Figure 4.2: Property Claim Blocks 4-5
Figure 4.3: Property Claims (West) 4-6
Figure 4.4: Property Claims (Central) 4-7
Figure 4.5: Property Claims (East) 4-8
Figure 4.6: Property Claims (North) 4-9
Figure 4.7: Net Smelter Royalty (NSR) per Claim Block 4-11
Figure 5.1: Local Infrastructure 5-2
Figure 6.1: Historical Work Summary (West) 6-7
Figure 6.2: Historical Work Summary (East) 6-8
Figure 6.3: Historical Drill Hole Summary 6-9
Figure 7.1: Regional Geology 7-2
Figure 7.2: Property Geology and Mineral Exploration Trends 7-4
Figure 7.3: ‘Whale-Back’ Spodumene Pegmatite Landform at CV13 7-5
Figure 7.4: ‘Whale-Back’ Spodumene Pegmatite Landform at CV5 7-6
Figure 7.5: LCT Pegmatite Clusters at the Property 7-7
Figure 7.6: Main Outcrop at CV5 Pegmatite (Looking Westerly) 7-10
Figure 7.7: Main Outcrop at the CV5 Pegmatite (Looking Northerly) 7-11
Figure 7.8: Spodumene Crystal at the CV5 Pegmatite 7-11
Figure 7.9: Strongly Fractured, Pinkish Weathered Spodumene Crystals in Matrix of White Feldspar and Grey Quartz at the CV5 Pegmatite 7-12
Figure 7.10: Coarse-grained Spodumene Mineralization in Drill Holes CV22-035 and 036 7-13
Figure 7.11: Coarse-grained Spodumene Mineralization from Nova Zone in Drill Hole CV22-083 7-14
Figure 7.12: Plan View of CV5 Pegmatite Geological Model – All Lenses 7-16
Figure 7.13: Inclined View of CV5 Pegmatite Geological Model Looking Down Dip (70°) – All Lenses 7-16

Figure 7.14: Side View of CV5 Geological Model 7-17
Figure 7.15: Simplified Cross-Section of CV5 Pegmatite Geological Model 7-18
Figure 7.16: Simplified Cross-Section of CV5 Pegmatite Geological Model 7-19
Figure 7.17: Simplified Cross-Section of CV5 Pegmatite Geological Model 7-20
Figure 7.18: Aerial View of the Spodumene Pegmatite Outcrop at CV13 (Looking Northeasterly) 7-22
Figure 7.19: Coarse-grained Spodumene Crystals in Drill Core from CV13 7-23
Figure 7.20: Plan View of CV13 Pegmatite Geological – All Lenses (top); Oblique View of CV13 Pegmatite Geological Model – All Lenses (bottom) 7-24
Figure 7.21: Pollucite Mineralization in High-Grade Caesium Drill Intersection at ~64.5 m Depth in Drill Hole CV23-271 at the Rigel Zone, CV13 Pegmatite 7-25
Figure 7.22: Pollucite with Lepidolite Veining (purple) in Grey Quartz Matrix from 139.3 m to 139.5 m in Drill Hole CV24-520 (Vega Zone), Within a Wider Zone of Caesium Mineralization Grading 7.39% Cs_2O over 7.1 m 7-26
Figure 7.23: Simplified Cross-section of the Rigel Caesium Zone Geological Model at the CV13 Pegmatite 7-27
Figure 7.24: Simplified Cross-section of the Vega Caesium Zone Geological Model at the CV13 Pegmatite 7-28
Figure 8.1: LCT Pegmatite Deposit Model 8-3
Figure 9.1: Company Exploration Summary Through 2024 (West) 9-8
Figure 9.2: Company Exploration Summary Through 2024 (Central) 9-9
Figure 9.3: Company Exploration Summary Through 2024 (East) 9-10
Figure 10.1: Drill Holes Completed at Shaakichiuwaanaan Property's Principal Claim Grouping (through to CV24-787) 10-3
Figure 10.2: Drill Holes and Channels Completed at CV5 LCT Pegmatite Through 2024 10-4
Figure 10.3: Drill Holes and Channels Completed at CV13 LCT Pegmatite Through 2024 10-5
Figure 10.4: Drill Holes Completed Through 2024 at the CV5 and CV13 LCT Pegmatites 10-15
Figure 11.1: Blank Sample Results (Li) from the 2024 Drilling Campaign 11-5
Figure 11.2: Blank Sample Results (Ta) from the 2024 Drilling Campaign 11-6
Figure 11.3: Blank Sample Results (Ga) from 2022-2024 Drilling Campaigns 11-7
Figure 11.4: Pulp Duplicates (Li) for the 2024 Program 11-12
Figure 11.5: Pulp Duplicates (Ta) for the 2024 Program 11-13
Figure 11.6: Pulp Duplicates (Ga) for the 2022 to 2024 Programs 11-14
Figure 11.7: Pulp Duplicates (Cs) for the 2022 to 2024 Programs for Assays Less than 10,000 ppm 11-15
Figure 11.8: External Pulp Duplicates (Li) for the 2024 Program 11-16
Figure 11.9: External Pulp Duplicates (Ta) for the 2024 Program 11-17
Figure 11.10: External Pulp Duplicates (Cs) for the 2022 to 2024 Programs (2,325 Samples; < 20,000 ppm) 11-18

Figure 12.1: Channeled Outcrop Examined During the Site Visit 12-1
Figure 12.2: Drill Collar Validation 12-2
Figure 13.1: Location of CV5 Spodumene Pegmatite Drill Core Collected HLS and DMS Samples 13-7
Figure 13.2: Variability Composite Head Grades 13-10
Figure 13.3: Size by Assay - Lithium Distribution 13-11
Figure 13.4: XRD Results on Variability Composite Samples 13-2
Figure 13.5: Particle Size Distributions for Variability Composite Samples 13-3
Figure 13.6: Global HLS Results for Variability Composite Samples 13-6
Figure 13.7: Test Program 19005-02A HLS Testwork Results 13-7
Figure 13.8: HLS Testwork Results 13-8
Figure 13.9: Stage HLS Results 13-9
Figure 13.10: Spodumene Concentrate (DMS + non-magnetic fractions) 5.8% Li_2O and 0.60% Fe_2O_3 at 79% Recovery 13-12
Figure 13.11: Close-up of Spodumene Concentrate (DMS + non-magnetic fractions) 5.8% Li_2O and 0.60% Fe_2O_3 at 79% Recovery 13-12
Figure 13.12: DMS Testwork Flowsheet 13-13
Figure 13.13: DMS Testwork Flowsheet 13-15
Figure 13.14: Combined DMS and Flotation Testwork Flowsheet 13-18
Figure 13.15: 19005-10 Spodumene Flotation Testwork Results 13-19
Figure 13.16: Summary of Filtration Tests 13-20
Figure 13.17: SGS Gravity Separation Flowsheet 13-21
Figure 13.18: Lithium Conversion Testwork Flowsheet 13-23
Figure 13.19: On Spec Battery-Grade Lithium Hydroxide Monohydrate Product 13-24
Figure 13.20: Metallurgical Testwork Recovery Results and Industry-based Recovery Estimates for 3x Size Range DMS Process Plant 13-26
Figure 14.1: SG Measurement Setup Employed by PMET Resources 14-2
Figure 14.2: Regression Function SG Derivation 14-3
Figure 14.3: LiDAR Topographic Image with Spodumene Pegmatite Outcrops at CV5 14-4
Figure 14.4: CV5 Mineralized Zones 14-5
Figure 14.5: Mineralized Zones of CV13 14-6
Figure 14.6: CV5 Capping Justification on Li_2O for Spodumene-Rich Domain (Zone 100) 14-20
Figure 14.7: CV13 Capping Justification on Li_2O for CV13_100 Domain 14-21
Figure 14.8: Variogram Model for CV5 Zone 100 (Li_2O) 14-22
Figure 14.9: Oblique View of the CV5 Spodumene Pegmatite Block Model with Respect to Applied OP and UG Conceptual Mining Constraint Shapes 14-35
Figure 14.10: CV13 Open Pit and Underground Constraints to MRE Block Model 14-39
Figure 14.11: CV13 Open Pit Constraint to Rigel and Vega Caesium Zone MRE Block Model 14-39

Figure 14.12: CV5 Comparison (composites vs block model Li_2O; eastern portion) 14-41
Figure 14.13: CV13 Comparison (composites vs block model Li_2O; Vega portion) 14-42
Figure 14.14: CV13 Comparison (composites vs block model Cs_2O; CV13_101 with Vega Cs-enriched zones) 14-42
Figure 14.15: CV13 Comparison (composites vs block model Cs_2O; CV13_100 with Rigel Cs-enriched zone) 14-43
Figure 14.16: Li_2O Swath Plot in X Direction (CV5 Zone 100) 14-44
Figure 14.17: Li_2O Swath Plot in Elevation (Z) Direction (CV5 Zone 100) 14-45
Figure 14.18: Li_2O Swath Plot in Easting (X) Direction (CV13 Zone 101) 14-46
Figure 14.19: Li_2O Swath Plot in Elevation (Z) Direction (CV13 Zone 101) 14-47
Figure 14.20: Shaakichiuwaanaan Mineral Resource Grade-Tonnage Curves for CV5 14-49
Figure 14.21: Shaakichiuwaanaan Mineral Resource Grade-Tonnage Curves for CV13 14-50
Figure 15.1: Geotechnical Domains by Lithology and Orientation (from Alius, 2025) 15-3
Figure 15.2: West and East Pits Delimitation 15-7
Figure 15.3: M&I Pit by Pit Results @ USD 1,303/t Conc. 15-10
Figure 15.4: West Pit Phasing 15-11
Figure 15.5: M&I Pit by Pit Results @ USD 1,303/t Conc. 15-15
Figure 15.6: East Pit Phasing 15-16
Figure 15.7: Double-Lane Ramp Design Criteria 15-17
Figure 15.8: Single-Lane Ramp Design Criteria 15-18
Figure 15.9: Equivalent Linear Overbreak Slough (ELOS) Schematic 15-20
Figure 15.10: Underground Mine Transverse Stope Types 15-22
Figure 16.1: Plan View of Open Pit and Relevant Geotechnical (Soil) Drill Holes 16-3
Figure 16.2: Plan View of Open Pit & Underground Mine and Geomechanical Drill Holes 16-4
Figure 16.3: Open Pit Design Sectors 16-9
Figure 16.4: Static Factor of Final Pit Slopes' Safety from 3D Model 16-11
Figure 16.5. Damage Indicator (Sloss) on Final Pit Slopes from 3D Model 16-11
Figure 16.6: Major Principal Stress (σ_1) and Damage Indicator (Sloss) Surrounding the Mined Stopes at Year 2049 From 3D Model 16-18
Figure 16.7: Open Pit Phase Limits - Plan View 16-23
Figure 16.8: Phase 1A Open Pit Design – Plan View 16-24
Figure 16.9: Phase 1B Open Pit Design -Plan View 16-25
Figure 16.10: Phase 2 Open Pit Design – Plan View 16-25
Figure 16.11: Phase 3A Open Pit Design – Plan View 16-26
Figure 16.12: Phase 3B Open Pit Design – Plan View 16-26
Figure 16.13: Phase 4 (Final Open Pit Design) – Plan View 16-27
Figure 16.14: Waste Storage Facility – Plan View 16-28

Figure 16.15: Ore Stockpile Location – Plan View 16-29
Figure 16.16: Open Pit and Underground Mined Tonnes 16-31
Figure 16.17: Open Pit Mining Schedule by Phase (without stockpile reclaim) 16-32
Figure 16.18: Mine Development End of Dec. 2028– Plan View 16-34
Figure 16.19: Mine Development End of June 2030– Plan View 16-34
Figure 16.20: Mine Development End of Q1 2031– Plan View 16-35
Figure 16.21: Mine Development End of Q1 2032 – Plan View 16-35
Figure 16.22: Mine Development End of Q2 2033 – Plan View 16-36
Figure 16.23: Mine Development End of Q3 2038 – Plan View 16-36
Figure 16.24: Mine Development End of Q3 2047 (LOM) – Plan View 16-37
Figure 16.25: Open Pit Average Haulage Cycle Times by Phase 16-41
Figure 16.26: Open Pit Truck Requirements 16-42
Figure 16.27: Open Pit Dewatering Volumes and Quantity of Pumps Over Time 16-44
Figure 16.28: Points of Interest – Risk Model 16-45
Figure 16.29: Representation of Transverse LHOS Mining Method 16-53
Figure 16.30: Representation of Longitudinal Mining Method 16-53
Figure 16.31: Typical Longitudinal Stoping Sequence 16-55
Figure 16.32: Typical Transverse Stoping Sequence 16-56
Figure 16.33: Ore Losses During Drilling & Blasting Operations - Section View 16-57
Figure 16.34: Ore Losses During Mucking Operations - Plan View 16-58
Figure 16.35: Typical Development Section of the Decline 16-60
Figure 16.36: Typical Development Section of a Draw Point 16-61
Figure 16.37: Typical Larger Level - Plan View 16-63
Figure 16.38: Typical Smaller Level - Plan View 16-64
Figure 16.39: Mine Development Longitudinal View - Looking West 16-65
Figure 16.40: Underground Mine Development Longitudinal View - Looking North 16-66
Figure 16.41: Underground Mine and Surface Constraints – Plan View 16-67
Figure 16.42: Stoping and Pegmatites - Looking West - Section at Easting 571,647 16-69
Figure 16.43: Underground Mine Longitudinal View by Mining Method - Looking 289° W 16-70
Figure 16.44: Underground Mine Longitudinal View by Mining Method - Looking North 16-70
Figure 16.45: Underground Mine Longitudinal View by Mining Blocks - Looking West 16-71
Figure 16.46: Underground Mine Longitudinal View by Mining Blocks - Looking North 16-71
Figure 16.47: Stope Height Histogram 16-72
Figure 16.48: Underground Mine with Open Pit – Plan View 16-73
Figure 16.49: Open Pit and Underground Mine Interaction Close-Up – Looking North 16-73
Figure 16.50: Typical Drilling Section for Underground Stoping 16-75
Figure 16.51: Typical Production Hole Loading Specifications 16-76

Figure 16.52: Underground Mine Recovered Ore Tonnes from Stoping Split by Mining Block 16-81
Figure 16.53: Typical Loading Bay - Plan View 16-85
Figure 16.54: Charging Bays Layout 16-92
Figure 16.55: BEVs Haul Trucks and LHDs Modelled Battery Charging Cycles Through Shift 16-93
Figure 16.56: Underground Mine Ventilation Network – Stage 1 – Temporary Ventilation Set-up – Isometric View from North-West 16-96
Figure 16.57: Underground Mine Ventilation Network – Stage 2 – Temporary Ventilation Set-up – Isometric View from North-West 16-97
Figure 16.58: Underground Mine Ventilation Network – Permanent Set-up – Isometric View from North-West 16-98
Figure 16.59: Typical Ventilation Layout 16-100
Figure 16.60: Typical Plan View of Electric Bay 16-102
Figure 16.61: Underground Electrical Distribution Network – Looking North 16-103
Figure 16.62: Typical Sump Located in Level Access 16-104
Figure 16.63: Typical Underground Decantation and Pumping Bays 16-105
Figure 16.64: Underground Pumping Network – Looking North 16-106
Figure 16.65: Underground Backfill Distribution Network – Looking North 16-108
Figure 16.66: Typical Layout of Underground Storage (utilizing an existing excavation) 16-110
Figure 16.67: Typical Cross-section of Bulk Emulsion Truck Parking 16-111
Figure 16.68: Typical Explosives Storage and Handling Facilities 16-112
Figure 16.69: Typical Maintenance Bay 16-114
Figure 16.70: Schematic Section View of UG Mine Portal 16-115
Figure 16.71: Example of a Mine Portal Culvert 16-116
Figure 16.72: Underground Refuge Stations Location – Section Looking North 16-117
Figure 16.73: Typical Permanent Refuge Station I 16-118
Figure 16.74: OP and UG Mine Workforce 16-120
Figure 16.75: Ore Processing by Source Material 16-133
Figure 16.76: Spodumene (5.5%Li_2O) Concentrate Production 16-134
Figure 17.1: Mineral Processing Facility Simplified Mass Balance 17-5
Figure 17.2: Mineral Processing Facility Simplified Water Balance 17-6
Figure 17.3: Mineral Processing Facility Simplified Process Flow Diagram 17-9
Figure 18.1: Main Site Infrastructure – Plan View 18-2
Figure 18.2: Main Site Infrastructure – Perspective View 18-3
Figure 18.3: Typical Camp Dorm – Type B 18-6
Figure 18.4: Kitchen Plan View 18-7
Figure 18.5: Kitchen 3D View 18-7
Figure 18.6: Camp Office and Welcome Centre, Plan View 18-8

Figure 18.7: Camp Office and Welcome Centre, 3D View 18-8
Figure 18.8: Fire Hall & Clinic, Plan View 18-9
Figure 18.9: Fire Hall & Clinic, 3D View 18-10
Figure 18.10: Typical Deep-Corrugated Galvanized Steel Plates Portal 18-11
Figure 18.11: Underground Mine Main Ventilation Set-up 18-13
Figure 18.12: Compressor Building 18-14
Figure 18.13: Truck Shop Layout 18-15
Figure 18.14: Temporary Truck Shop – 3D View 18-16
Figure 18.15: Dry & Mine Administrative Building - Plan View 18-19
Figure 18.16: Dry & Mine Administrative Building - 3D View 18-19
Figure 18.17: Warehouse Plan View 18-21
Figure 18.18: Warehouse – 3D View 18-21
Figure 18.19: Explosive Storage Area 18-23
Figure 18.20: Mill Office Plan View 18-25
Figure 18.21: Mill Office 3D View 18-25
Figure 18.22: Primary Crushing, Secondary and Tertiary Crushing, Screening Buildings 18-26
Figure 18.23: Crushed Ore Storage Dome 18-27
Figure 18.24: Phase 1 (Green) and Phase 2 (Grey) Structures 18-28
Figure 18.25: Concentrate Load-Out 18-29
Figure 18.26: Paste Plant Area 18-31
Figure 18.27: Typical Lined Base Stockpile 002 18-37
Figure 18.28: Typical Section for Tailings in Stockpile 002 Showing Intermediate Benches for Both Operation and Closure Stages 18-37
Figure 18.29: Stockpile 002 (Tailings and Waste Rock) and Related Infrastructure 18-38
Figure 18.30: Stockpile 002 Critical Sections for Stability Analysis 18-39
Figure 18.31: Typical Section for Waste Rock in Stockpiles 001 and 002 Showing Intermediate Benches for Both Operation and Closure Stages 18-49
Figure 18.32: Stockpile 01 and Associated Water Management Infrastructure 18-50
Figure 18.33: Layout of Critical Sections for Slope Stability Analysis – Waste Rock Stockpile 001 Cross Section A1-A1 18-52
Figure 18.34: Site-Wide Projected Water Management Infrastructures 18-55
Figure 18.35: Typical Section for Stockpile 001 Ditch with Adjacent Road Inspection 18-57
Figure 18.36: Typical Section for Stockpile 002 Lined Ditch with Adjacent Road Inspection 18-57
Figure 18.37: Waste Rock Stockpile Water Retention Dam Section 18-60
Figure 18.38: Site Water Management Flow Diagram (including annual flow rates during average climate condition) 18-64
Figure 18.39: Lake 001 Diversion Channel and Cut-off Dams 18-66

Figure 18.40: Profile of Diversion Channel with Proposed Fish Habitat / Passage 18-67
Figure 18.41: Typical Cross-Section for Cut-Off Dam (critical section) 18-69
Figure 18.42: Open Pit (West Pit and East Pit) Sediment and Water Management Layouts 18-72
Figure 18.43: Water Treatment Pond and Plants 18-74
Figure 18.44: General Process Flow Diagram 18-75
Figure 18.45: Transmission Route 18-79
Figure 18.46: Site Main Substation 18-81
Figure 18.47: Main Electrical Room 18-82
Figure 18.48: Load Profile 18-83
Figure 18.49: Overhead Network 18-85
Figure 18.50: Road Transportation 18-87
Figure 18.51: MTC Conceptual Design 18-89
Figure 19.1: Lithium Total Non-Risked Supply and Demand Scenarios 19-3
Figure 19.2: Risk-Weighted Lithium Mine Supply by Operating Status 19-3
Figure 19.3: Lithium Demand by End-Use Applications 19-5
Figure 19.4: BESS Installed Capacity Outlook by Storage Type, New Additions 19-6
Figure 19.5: Global Market Balance 19-7
Figure 19.6: Global Demand for Refined Lithium 19-8
Figure 19.7: Lithium Price Forecasts, $/tonne, Real 2025 19-9
Figure 20.1: Stockpile 001 Closure Configuration 20-42
Figure 20.2: Stockpile 002 Closure Configuration 20-43
Figure 21.1: Initial and Expansion & Sustaining Capital Expenditures Timeline 21-4
Figure 21.2: Timeline for Construction Phases 1 and 2 21-5
Figure 22.1: Tornado Graph for the Sensitivity Analysis (+/-20%) 22-2
Figure 22.2: Price Evolution of Fuel Rack Price Over the Last Five Years (per month) 22-5
Figure 22.3: Five-Year Exchange Rate 22-7
Figure 22.4: Annual Spodumene Concentrate (5.5% Li2O) Production 22-8
Figure 22.5: Open Pit Mine Production Profile 22-9
Figure 22.6: Underground Mine Production Profile 22-9
Figure 22.7: Process Production Profile 22-10
Figure 22.8: Initial CAPEX by Month 22-16
Figure 22.9: Life of Mine After-Tax Cumulative Cash Flow Projection 22-27
Figure 22.10: Free Cash flow Sensitivity 22-30
Figure 22.11: NPV 8% Sensitivity 22-30
Figure 22.12: IRR Sensitivity 22-31
Figure 22.13: NPV 8% Sensitivity 22-31
Figure 23.1: Adjacent Properties to the Shaakichiuwaanaan Property (as of August 2025) 23-2

Figure 24.1: Project Schedule – Level 1 24-3
Figure 24.2: General View of Underground Nova Zone – Looking SSE 24-5
Figure 25.1: Metallurgical Test Work Recovery (Global) Results & Industry-Based Recovery Estimates for 3x Size Range DMS Process Plant 25-9

List of Tables

Table 1.1: Pilot DMS Results 1-15
Table 1.2: Shaakichiuwaanaan Mineral Resource Estimate (CV5 & CV13 Pegmatites) 1-18
Table 1.3: Mineral Resources at Rigel and Vega Caesium Zones Within the CV13 Pegmatite 1-19
Table 1.4: Shaakichiuwaanaan Project Mineral Reserve 1-20
Table 1.5: M&I Push Back Selection @ USD 1,303/t Conc. 1-24
Table 1.6: M&I Phasing Selection @ USD 1,303/t Conc. 1-25
Table 1.7: Spodumene Concentrate Price Used 1-28
Table 1.8: Project Total Capital Cost Summary 1-34
Table 1.9: Total LOM Operating Cost at Site 1-35
Table 1.10: Unit Operating Cost 1-36
Table 1.11: Financial Analysis Summary 1-36
Table 1.12: Cost Estimate Associated with Recommendations 1-41
Table 2.1: Summary of Qualified Persons 2-3
Table 2.2: Table of Abbreviations 2-8
Table 6.1: Surface Showing Highlights from Historical Work on the Property 6-4
Table 10.1: Company Drill Hole Summary Through 2024 at the Property 10-1
Table 11.1: QA/QC Insertions for Drilling Programs from 2021 to 2024 11-3
Table 11.2: Certified Reference Materials Used in the 2024 Drilling Program 11-8
Table 11.3: Certified Reference Materials Used in the 2022 to 2024 Drilling Programs for Ga and Cs 11-10
Table 13.1: Sources of Samples for Metallurgical Programs Completed on CV5 Pegmatites 13-3
Table 13.2: Metallurgical Programs Samples Completed on Host Rock Surrounding CV5 Sources 13-8
Table 13.3: Sources of Samples for Metallurgical Programs Completed on Pegmatite from CV13 13-9
Table 13.4: Lithium (Li_2O) and Iron (Fe_2O_3) Grade Distribution with Size in CV5 13-12
Table 13.5: Standard Bond Abrasion Test Results 13-4
Table 13.6: Standard Bond Rod Mill Grindability Test Results 13-4
Table 13.7: Standard Bond Ball Mill Grindability Test Results 13-4
Table 13.8: Magnetic Separations Results for Variability Composite Samples 1, 3, 5 & 8 13-10
Table 13.9: CRIMMs Magnetic Separations Results at 9,000 Gs 13-11
Table 13.10: Master Comp DMS Testwork Results 13-14
Table 13.11: MC001 DMS Testwork Results 13-16
Table 13.12: MC002 DMS Testwork Results 13-16
Table 13.13: MC001 HLS Recrush Results 13-17
Table 13.14: Stage Gravity Separation Results of MC001 DMS Magnetic Rejects 13-21
Table 13.15: Pilot DMS Results 13-28
Table 14.1: MRE Specific Gravity Summary 14-3

Table 14.2: Assays Summary by Domain (CV5; length-weighted) 14-7
Table 14.3: Assay Summary by Domain (CV13; Length-weighted) 14-8
Table 14.4: Cs_2O Assays Summary by Refined Domain (CV13; length-weighted) 14-12
Table 14.5: Compositing Summary by Domain for CV5 14-12
Table 14.6: Compositing Summary by Domain for CV13 14-14
Table 14.7: Cs_2O (%) Compositing Summary in Cs-Enriched Zones CV13 (composites of 0.5 m) 14-17
Table 14.8: Grade Capping Summary by Domain for CV5 14-18
Table 14.9: Grade Capping Summary by Domain for CV13 14-19
Table 14.10: Grade Capping Summary by Refined Domain for Cs_2O at CV13 14-19
Table 14.11: Variogram Summary for CV5 14-23
Table 14.12: CV5 Parent Block Model Summary 14-24
Table 14.13: CV13 Parent Block Model Summary 14-25
Table 14.14: CV5 Search Ellipse Summary 14-26
Table 14.15: CV5 Estimation Criteria Summary 14-27
Table 14.16: CV13 Search Ellipse Summary 14-28
Table 14.17: CV13 Estimation Criteria Summary 14-29
Table 14.18: CV13 Estimation Criteria Summary 14-30
Table 14.19: Shaakichiuwaanaan Mineral Resource Estimate 14-31
Table 14.20: Detailed CV5 Mineral Resource Estimate 14-33
Table 14.21: CV5 Parameters for Reasonable Prospects of Economic Extraction – OP & UG 14-34
Table 14.22: Detailed CV13 Mineral Resource Estimate 14-36
Table 14.23: Mineral Resources at Rigel and Vega Caesium Zones within the CV13 Pegmatite 14-37
Table 14.24: Parameters for Reasonable Prospect of Economic Extraction – OP & UG 14-38
Table 14.25: Comparison of Estimation Method Statistics Between Composites, NN, ID^2 and OK on Li_2O (%) for CV5 and CV13 14-43
Table 14.26: Sensitivity Analysis for Shaakichiuwaanaan MRE 14-48
Table 14.27: Shaakichiuwaanaan Mineral Resource Estimate (January 6, 2025) 14-50
Table 14.28: Shaakichiuwaanaan Mineral Resource Estimate (August 21, 2024) 14-51
Table 14.29: Mineral Resource Statement (June 25, 2023) 14-51
Table 15.1 Shaakichiuwaanaan Project Mineral Reserve 15-1
Table 15.2: Geotechnical Benches Design Criteria 15-4
Table 15.3: Detailed Slope Design Parameters 15-4
Table 15.4: Economic Optimization Parameters 15-6
Table 15.5: M&I Pit by Pit Results @ USD 1,303/t Conc. 15-8
Table 15.6: M&I Pit Shell Selection @ USD 1,303/t Conc. 15-10
Table 15.7: M&I Push Back Selection @ USD 1,303/t Conc. 15-12
Table 15.8: M&I Pit by Pit Results @ USD 1,303/t Conc. 15-13

Table 15.9: M&I Final Pit Shell Selection @ USD 1,303/t Conc. 15-15
Table 15.10: M&I Phasing Selection @ USD 1,303/t Conc. 15-16
Table 15.11: Open Pit Mineral Reserves Statement 15-19
Table 15.12: Shaakichiuwaanaan Open Pit Mineral Reserves and Quantities 15-19
Table 15.13: Underground Mine Backfill Dilution Parameters 15-21
Table 15.14: Underground Mine Cut-Off Calculation Parameters 15-23
Table 15.15: Underground Mine Stope Optimizer Parameters 15-24
Table 15.16: Underground Mineral Reserves Statement 15-24
Table 16.1: Interpreted Strength Properties of Geotechnical Units 16-4
Table 16.2: Summary of Main Lithologies Intact Rock Strength Properties 16-5
Table 16.3: Rock Mass Classifications 16-6
Table 16.4: In-Situ Stress State 16-6
Table 16.5: Open Pit Geotechnical Domains 16-7
Table 16.6: Major Discontinuity Sets Identified per Geotechnical Domain in Open Pit Mine 16-8
Table 16.7: Guidelines for Open Pit Design Sectors 16-10
Table 16.8: Stope Dimensions and Dilution Estimates 16-13
Table 16.9: Cemented Paste Backfill Summary of Strength Requirements 16-14
Table 16.10: Strength Requirement for Longitudinal & Transversal Stopes Over the Years 16-14
Table 16.11: Ground Support Guidelines 16-16
Table 16.12: Groundwater Inflow Calculation – Base Case Scenario 16-21
Table 16.13: Physical Quantities per Open Pit Phase 16-23
Table 16.14: Open Pit Mining Schedule Summary 16-33
Table 16.15: Drill and Blast Parameters 16-38
Table 16.16: Loading Productivity for Ore Material 16-39
Table 16.17: Points of Interest and Blasting Location Distances 16-46
Table 16.18: Equipment Usage Assumption 16-48
Table 16.19: Major Equipment Purchase Schedule 16-49
Table 16.20: Major Equipment Requirement 16-49
Table 16.21: Support Equipment Purchase Schedule 16-50
Table 16.22: Support Equipment Requirement 16-51
Table 16.23: Production drilling Parameters 16-76
Table 16.24: Underground Mine Development Metres Summary 16-78
Table 16.25: Underground Mine Physical Quantities Summary 16-78
Table 16.26: Underground Mine Scheduler Rates 16-79
Table 16.27: Ore Production Schedule 16-82
Table 16.28: Lateral Development Schedule 16-83
Table 16.29: Vertical Development Perform by Contractor Schedule 16-84

Table 16.30: Truck Parameters for Hauling Studies 16-86
Table 16.31: Underground Contractor Mine Mobile Equipment Fleet Requirements 16-87
Table 16.32: UG Mobile Equipment Purchase Schedule 16-88
Table 16.33: UG Mobile Equipment Requirement Schedule 16-89
Table 16.34: Underground Peak Production Mine Fresh Air Requirements per Equipment 16-95
Table 16.35: Underground Mine Ventilation System Details 16-99
Table 16.36: Temperature Numbers Recorded at La Grande 4 Station 16-101
Table 16.37: Underground Mine Dewatering Assumption 16-106
Table 16.38: Underground Mine Pumping Requirements Details 16-106
Table 16.39: Open Pit Mine Operations Workforce 16-121
Table 16.40: Open Pit Maintenance Workforce 16-122
Table 16.41: Open Pit Mine Geology Workforce 16-122
Table 16.42: Open Pit Mine Engineering Workforce 16-123
Table 16.43: Open Pit Mine Total Workforce 16-124
Table 16.44: UG Mine Operations Workforce 16-126
Table 16.45: UG Mine Maintenance Workforce 16-127
Table 16.46: UG Geology Workforce 16-128
Table 16.47: UG Engineering Workforce 16-128
Table 16.48: UG Electrical Workforce 16-129
Table 16.49: UG Summary Workforce 16-130
Table 16.50: Total Mining Workforce 16-130
Table 16.51: LOM Production Details per Production Year 16-132
Table 17.1: Mineral Processing Facility Design Criteria 17-2
Table 17.2: Mineral Processing Facility Reagent Requirements 17-7
Table 18.1: Tailings Production – Stockpiled and Backfill 18-34
Table 18.2: Material Geotechnical Parameters – Tailings Stockpile[1] 18-40
Table 18.3: Safety Factor of Slope Stability Analysis – Tailings Stockpiles 18-41
Table 18.4: Waste Rock and Overburden Production and Storage Areas 18-45
Table 18.5: Factor of Safety of the Slope Stability Analysis – Waste Rock Stockpiles 18-53
Table 18.6: Water Collection Ditches – Stockpiles 001 and 002 18-56
Table 18.7: Contact Water Collection Ponds 18-59
Table 18.8: Industrial Area Ponds 18-61
Table 18.9: Hydraulic and Mechanical Properties for the Seepage & Slope Stability Analyses 18-69
Table 18.10: Slope Stability Analysis Results, Diversion Dams 18-70
Table 18.11: WTP Design Capacity Through the Years 18-76
Table 18.12: Electrical Loads by Sector 18-83
Table 19.1: Lithium Supply Growth Requirements 19-2

Table 19.2: Current Market Spot Price 19-8
Table 19.3: Jurisdiction of Canada Spodumene Price 19-10
Table 19.4: Spodumene Concentrate Price Recommendation 19-11
Table 20.1: List of Herpetofauna Species Likely to Frequent the Study Area 20-8
Table 20.2: List of Small Mammals Likely to Frequent the Study Area 20-10
Table 20.3: Special Status Wildlife Species Potentially Present in the Project Area 20-12
Table 20.4: Special Status Plant Species Potentially Present in the Project Area 20-13
Table 20.5: Cree Communities and Non-Indigenous Towns Near the Shaakichiuwaanaan Project 20-16
Table 20.6: Provincial and Federal List of Permits 20-28
Table 20.7: Preliminary and Non-exhaustive List of Permitting Requirements 20-31
Table 20.8: Summary of COPC Peak Concentration Estimates in Stockpile-001, Stockpile-002, and Ore Stockpile 20-39
Table 20.9: Stakeholders Targeted as Part of the Prior Information Process 20-49
Table 20.10: Main Comments and Concerns Expressed by Indigenous Groups during Engagement and Consultation Activities 20-51
Table 20.11: Main Comments and Concerns Expressed during Consultation Activities with Non-indigenous Stakeholders 20-54
Table 21.1: Initial and Expansion & Sustaining Capital Expenditures Summary 21-3
Table 21.2: Operating Costs Summary 21-6
Table 21.3: Phase 1 Infrastructures Capital Expenditures by WBS 21-11
Table 21.4: Phase 1 Power Supply and Communications Capital Expenditures by WBS 21-12
Table 21.5: Phase 1 Water and Waste Rock Management Capital Expenditures by WBS 21-13
Table 21.6: Phase 1 Surface Equipment Capital Expenditures by WBS 21-14
Table 21.7: Phase 1 Mining Capital Expenditures by WBS 21-15
Table 21.8: Phase 1 Process Expenditures by WBS 21-16
Table 21.9: Phase 1 Indirect Costs by WBS 21-17
Table 21.10: Phase 1 General Services – Owner's Costs by WBS 21-19
Table 21.11: Phase 1 - Pre-Production Costs 21-20
Table 21.12: Project Phase 2 Capital Cost Summary 21-22
Table 21.13: Phase 2 Infrastructures Capital Expenditures by WBS 21-22
Table 21.14: Phase 2 Power Supply and Communications Capital Expenditures by WBS 21-23
Table 21.15: Phase 2 Water Management Capital Expenditures by WBS 21-24
Table 21.16: Phase 2 Mining CAPEX by WBS 21-25
Table 21.17: Phase 2 Process Expenditures by WBS 21-26
Table 21.18: Phase 2 Indirect Costs by WBS 21-28
Table 21.19: Phase 2 General Services – Owner's Costs by WBS 21-28
Table 21.20: Phase 2 Pre-Production Costs 21-29

Table 21.21: Sustaining Cost Summary 21-29
Table 21.22: Phase 1 Sustaining Capital Expenditure – in $M 21-30
Table 21.23: Phase 2 Sustaining Capital Expenditure – in $M 21-31
Table 21.24: Mine Closure Cost Summary 21-34
Table 21.25: Total LOM Operating Cost including Concentrate Transportation 21-35
Table 21.26: Total Operating Costs Summary 21-36
Table 21.27: Mining Unit Cost Summary 21-37
Table 21.28: Consumables Costs 21-37
Table 21.29: Open Pit Mine Operating Costs Summary 21-38
Table 21.30: Underground OPEX Summary 21-42
Table 21.31: Processing Plant Cost Summary 21-44
Table 21.32: LOM Processing Plant Cost Summary 21-45
Table 21.33: Plant Personnel Summary 21-46
Table 21.34: G&A Costs Summary 21-49
Table 21.35: General Services & Administration Cost Summary 21-50
Table 21.36: Electrical Costs 21-52
Table 21.37: Concentrate Transportation Costs 21-53
Table 22.1: Base Case Economic Result 22-2
Table 22.2: Shaakichiuwaanaan Project Mineral Reserve 22-4
Table 22.3: Li_2O Concentrate Price 22-5
Table 22.4: Monthly Average Wholesale (Rack) Prices (CA$/L) for Diesel in Val-d'Or (NRCan) 22-6
Table 22.5: Diesel Fuel Price Analysis 22-6
Table 22.6: Annual Mine and Mill Production Summary 22-11
Table 22.7: Initial and Expansion & Sustaining Capital Expenditures Summary 22-13
Table 22.8: Initial Capital 22-15
Table 22.9: Expansion and Sustaining Capital 22-17
Table 22.10: Operating Cost Summary 22-19
Table 22.11: Total Amount of Tax 22-21
Table 22.12: Project Economic Results Summary 22-22
Table 22.13: Project Cash Flow Summary 22-25
Table 22.14: OPEX Sensitivity 22-27
Table 22.15: CAPEX Sensitivity 22-28
Table 22.16: Spodumene Price Sensitivity 22-28
Table 22.17: Lithium Grade Price Sensitivity 22-28
Table 22.18: Exchange Rate Sensitivity 22-29
Table 22.19: Process Recovery Price Sensitivity 22-29
Table 25.1: Technical Report Feasibility Study Update Life-of-Mine Results 25-1

Table 25.2: Pit Phases Tonnage Summary 25-6
Table 25.3: Mine Reserve Material Over the Life of Mine (CV5 Pegmatite) 25-7
Table 25.4: Stockpiles Water Management Infrastructure 25-13
Table 25.5: Project Total Capital Cost Summary 25-16
Table 25.6: Financial Analysis Summary 25-18
Table 25.7: Project Risks 25-20
Table 25.8: Project Opportunities 25-22
Table 26.1: Cost Estimate Associated with Recommendations 26-1
Table 26.2: Phase 1 Estimated Budget 26-2

1. SUMMARY

1.1 Introduction

PMET Resources Inc. (PMET or the Company), formerly Patriot Battery Metals Inc., retained G Mining Services Inc. (GMS) as lead consultant, with contributions from Primero Group Americas Inc. (Primero), AtkinsRéalis, BBA Inc., Paterson & Cooke, WSP, Alius Mine Consulting, Mailloux Hydrogéologie, CGM Expert, and Vision Geochemistry to prepare an independent Feasibility Study (FS) and Technical Report for the CV5 Pegmatite at the Shaakichiuwaanaan Property (the "Property). The Property, formerly referred to as the "Corvette Project," is located in the Eeyou Istchee James Bay region of Québec, Canada.

PMET Resources is a leading hard-rock lithium exploration company advancing its 100%-owned Shaakichiuwaanaan Property in Quebec's Eeyou Istchee James Bay region. The Company's common shares trade on the Toronto Stock Exchange (TSX:PMET), Australian Stock Exchange (ASX:PMT), OTCQX market (OTCQBX: PMETF) and Frankfurt market (FSE: R9GA).

On October 5, 2023, PMET Resources Inc. incorporated a wholly owned Québec-based subsidiary, Lithium Innova Inc. (Innova), which is the sole registered holder of the Exclusive Exploration Rights comprising the Shaakichiuwaanaan Property. This Technical Report is submitted by PMET in its capacity as the owner of Innova.

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) and Form 43-101F1. The objective of this report and the FS is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit and underground mine, including processing facilities and related infrastructure. This Report provides operating and capital costs estimations and an economic analysis of the Project.

The Mineral Resource Estimate (MRE) was prepared following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and in accordance with CIM Guidelines (2019) for Estimation of Mineral Resources and Reserves. The effective date of the mineral resource estimation is June 20, 2025. It is based, for the most part, on a drilling database and supporting geological information.

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).

As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste at a grade of 0% Li_2O.

The intention of this Technical Report is to provide sufficient, clear, and unambiguous scientific and technical information relating to the Project as of the Effective Date. The Qualified Persons (QPs) understand that a copy of this Report will be filed with the Canadian Securities Commissions and made publicly available.

As of the Effective Date of this Report, the QPs are not aware of any litigation potentially affecting the Project. The QPs did not verify the legality or terms of underlying agreements related to ownership, permits, binding offtake term sheets, licences, royalties, or other contracts between PMET and third parties.

The qualified persons (QP) of this Technical Report are the following:

- Mr. Carl Michaud, P.Eng., MBA, Vice President of Mining Engineering at G Mining Services.
- Mr. Pascal Droz, P.Eng., Electrical Engineering Director / Project Manager at G Mining Services.
- Mr. Todd MCraken, P.Geo., Director Mining & Geology at BBA Inc.
- Mr. Ryan Cunningham, P.Eng., M. Eng., Process Engineer at Primero.
- Mr. Sebastien Guido, P.Eng., M. Sc. Senior Engineer, Rock Mechanics at Alius Mine Consulting.
- Mr. Hugo Latulippe, P.Eng., Principal Mining Engineer at BBA Inc.
- Mr. Ryan Smilovici, P.Eng., Process Engineer at Paterson & Cooke.
- Mr. Philip Addis, P.Eng., Principal Tailings Engineer at AtkinsRéalis.
- Mr. Antoine Cogulet, P.Eng., Team Lead, Mine Water Management at AtkinsRéalis.
- Mr. Holman Tellez, P.Eng., Sr. Expert Hydraulics / Hydrology at AtkinsRéalis.
- Ms. Nathalie Fortin, Eng., M. Env., Vice-President, Environmental Management, Earth Sciences and Environment at WSP.
- Ms. Genevieve Marchand, P.Eng., M.Sc., PMP, Project Director and mining hydrogeology specialist at AtkinsRéalis.
- Mr. Neal Sullivan, Ph.D., P.Geo., President & Principal Geochemist at Vision Geochemistry.
- Ms. Sandra Pouliot, P.Eng., M. Sc. A, PMP, Senior mining environment engineer and Sustainable mining at AtkinsRéalis.

- Mr. Michel Mailloux, P.Eng., P.Geo, M. Sc, Sr. Hydrogeologist and Owner at Mailloux Hydrogéologie.
- Mr. Charles Gagnon, P.Eng., M.Sc., Sr. Ventilation Engineer and Owner at CGM Expert.

1.2 Terms of Reference

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system. Currency is in Canadian dollars (“CAD”, “CA$” or “$”), unless otherwise stated. Unless otherwise indicated, all references to “$”, “CA$” or “CAD” in this Report are to Canadian dollars and references to “US$” or “USD” are to US dollars. A foreign exchange conversion rate of US dollar of USD 1/CAD 1.34 has been used over the Life of Mine (LOM).

1.3 Reliance on Other Experts

The Qualified Persons (QPs) have relied on information provided by independent experts with respect to mineral rights, surface rights, property agreements, royalties, taxation, commodity pricing, and the spodumene concentrate off-take agreement terms. The QPs consider this reliance reasonable, as the experts are assumed to hold the necessary education, professional designations, and relevant experience on matters addressed in this Technical Report.

The conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS and other consultants at the time of preparation of this Report.
- Assumptions, conditions, and qualifications set forth in this Report.
- Data, reports, and opinions supplied by PMET Resources and other third-party sources.

The QPs believe that the underlying assumptions in the information provided are factual and accurate and that the resulting interpretations are reasonable. To the extent applicable, the QPs have relied on such data and have no reason to believe that any material facts have been withheld. In their professional judgement, the QPs have taken appropriate steps to ensure that the information relied upon is sound and, accordingly, do not disclaim responsibility for the content of this Report.

The QPs have assumed that all information and technical documents referenced in Chapter 27 (References) are accurate and complete in all material respects. While the QPs have reviewed the available documentation, its accuracy and completeness cannot be guaranteed. The QPs reserve the right, but are not obligated, to revise this Report and its conclusions should additional information become available after the effective date.

1.4 Property Description and Location

The Shaakichiuwaanaan Property is located in the Eeyou Istchee James Bay region of Québec, Canada, centred on 53°32'00" N, 73°55'00" W, and is situated approximately 220 km east of Radisson, Québec, and 240 km north-northeast of Nemaska, Québec. The Property consists of 463 mineral claims that cover an area of approximately 23,710 ha over two (2) primary claim groups. The principal and largest claim grouping extends dominantly east-west for approximately 51 km as a nearly continuous, single claim block. The CV5 Pegmatite is located central to the Property, approximately 13 km south of KM-270 on the Trans-Taiga Road, 14 km south of the powerline, and 50 km southwest of the LG-4 dam complex. The CV13 Pegmatite is located ~3 km west-southwest of CV5.

The Property is situated on Category III Land within the Eeyou Istchee Cree Territory (Cree Nation of Chisasibi, and Cree Nation of Mistissini), as defined under the James Bay and Northern Québec Agreement (JBNQA). The Eeyou Istchee James Bay Regional Government (EIJBRG) is the designated municipality for the region, including the Property.

The Exclusive Exploration Rights (EER) that comprise the Property are registered under, and subject to, the *Mining Act* of the Province of Québec. Full claim details can be found on the *Ministère des Ressources naturelles et des Forêts* (MRNF)) mineral tenure system's online portal (GESTIM) website (https://gestim.mines.gouv.qc.ca/). All 463 EERs, formerly referred to as mineral claims, that comprise the Shaakichiuwaanaan Property are in good standing with term expiry dates ranging from January 22, 2026, to November 30, 2027. Through direct claim staking and various option agreements, which are all fully vested, the Company holds 100% interest in the Property and, through its subsidiary Lithium Innova Inc., is the sole registered title holder for all 463 EERs, subject to underlying royalties.

The CV5 Lithium-Caesium-Tantalum (LCT) Pegmatite MRE is subject to a 2% NSR. The CV13 LCT Pegmatite MRE, as is currently defined, is subject to a 2% royalty over the northern portions of its eastern and western limbs. The Rigel Caesium Zone, within CV13, is located on royalty-free ground staked directly by the Company. The Vega Caesium Zone, within CV13, is subject to a 2% NSR (half buyback for $2M). The CV4, CV8, CV9, CV10, CV12, CV14, and CV15 LCT pegmatites are subject to a 2% royalty.

The Company currently holds permits / authorizations from the *Ministère de l'Environnement, de la Lutte contre les changements climatiques de la Faune et des Parcs* (MELCCFP), MRNF, and Fisheries and Oceans Canada (DFO) to carry out surface and drill exploration on the Property. Additionally, the Company holds a lease from the MRNF on an area immediately south of KM-270 of the Trans-Taiga Road for an exploration camp including staging (i.e., laydown), core processing, and storage areas (collectively "Camp Shaakichiuwaanaan). The Company holds various permits from the MRNF, MELCCFP, and EIJBRG for

the construction and operation of its Camp Shaakichiuwaanaan. Several authorizations from the MELCCFP have been obtained for drinking water and wastewater treatment for the permanent camp, and future requests will be filed accordingly. The Company also holds various authorizations from the ministry for the construction and maintenance of an all-season road extending south from KM-270 of the Trans-Taiga Road to the southwest side of the CV5 Pegmatite.

The Company has submitted the required notifications to the applicable municipality and stakeholders outlining its mineral exploration plans for the Property through 2025. Additionally, the Company has an active engagement and consultation program for stakeholders.

Potential environmental liabilities at the Property include an exploration camp at KM-270 of the Trans-Taiga Road, an all-season road and associated borrow pits, and exploration access trails in certain drill areas. If the Project were not to move forward, this road and access trails may have to be reclaimed, and the exploration camp disassembled, and the area reclaimed. The QPs are not aware of any additional environmental liabilities beyond the normal disturbance related to surface exploration.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Property consists of two (2) primary claim groups – one straddling KM-270 of the Trans-Taiga Road, and the second with its northern border located directly south of KM-270, approximately 5.8 km from the Trans-Taiga Road and powerline infrastructure corridor. The La Grande 4 (LG-4) hydroelectric dam complex is located approximately 30 km north-northeast of the Property. The CV5 Spodumene Pegmatite is located central to the Property, approximately 13 km south of KM-270 on the Trans-Taiga Road, 14 km south of the powerline, and 50 km southwest of the LG-4 dam complex. The CV13 Spodumene Pegmatite is located ~3 km west-southwest of CV5.

The mineral exploration and development activities at the Property are supported by the Company's exploration camp (Camp Shaakichiuwaanaan) and Mirage Lodge located at KM-270 and KM-358 of the Trans-Taiga Road, respectively.

The Trans-Taiga all-season gravel road and Hydro-Québec's 735 kV power line trends east-west through the region, within approximately 6 km of the northern border of the Company's largest claim grouping. The Trans-Taiga Road connects approximately 210 km to the west of the Shaakichiuwaanaan Property to the Billy-Diamond Highway (Route 109) at KM-541, also known as the James Bay Road, which extends north to Radisson and south to Matagami, where it connects to Québec's regional road and railroad network.

The Property, and specifically the CV5 Pegmatite, may be accessed directly by an all-season road extending south from KM-270 of the all-season Trans-Taiga Road. Apart from the all-season road, Shaakichiuwaanaan Camp, and various exploration drill access trails, there is no infrastructure on the Property.

The Property is located in a sub-arctic climate region. Over the course of the year, the temperature typically varies from -27°C to 20°C, with rare extremes of -35°C and 26°C. Snow covers the ground from mid-October to late May, limiting field work in the winter period to drilling and geophysics. The Property topography consists of forested, gently rolling hills, drainages, and muskeg swamps between approximately 260 m and 350 m elevation, typical of the James Bay Region.

1.6 History

Historical exploration of the Property area was initially focused on base and precious metal mineralization, beginning in the late 1950s. This early work resulted In the discovery of several Cu-Au-Ag showings including Tyrone T-9 (3.36% Cu, 0.82 g/t Au, 38.4 g/t Ag in outcrop and 1.15% Cu over 2.1 m in channel), and Lac Smokycat-SO (1.75% Cu, 1.47 g/t Au, and 40.5 g/t Ag in outcrop) located on the present-day FCI West claim block (Ekstrom, 1960 - GM10515).

From the 1950s through to 1997, the Property area was subject to only limited exploration work, including various regional mapping surveys by the federal and provincial governments, as well as airborne magnetic and electromagnetic surveys.

In 1997, Virginia Gold Mines Inc. (Virginia) acquired an extensive land position in the area, which overlapped the present-day Property. Exploration between 1997 and 2000 included various geophysical surveys, surface mapping, and prospecting. Numerous base and precious metal showings were discovered during this period including Golden Gap (32.7 g/t Au in outcrop), Golden East (20.3 g/t Au), Deca-1 to Deca-4 (1.91 g/t Au over 5 m in channel, and 6.91 g/t Au in grab sample), Goose-1 (1.98 g/t Au), Goose-2 (3.74 g/t Au), and Sericite (1.89% Cu, 0.3 g/t Au, 150 g/t Ag, and 1.45% Zn). Continued surface exploration in subsequent years by Virginia (and various option partners) resulted in the discovery of several additional base and precious metal showings at the Property.

In 2001, the first diamond drill holes on the Property were completed, targeting the Golden Gap Showing, with drilling expanding in 2007 and 2013. Holes were completed at the Sericite Showing (302 m over two (2) holes in 2013), the Lac Bruno boulder field (391 m over three (3) holes in 2007), Golden Gap (combined total of 5,267 m in 24 holes; between 2001 and 2013) and the Deca-Goose area (325 m over

three (3) holes in 2001). The best historical precious metals drill intercept is from Golden Gap with 10.48 g/t Au over 7 m, obtained in 2007 (drill hole FCI-07-003).

In 2016, the Company (then under the name of 92 Resources Inc.) acquired an initial claim position in the area (part of the present-day Corvette Main claim block). The claims were acquired, in part, because of the words "*cristaux de spodumène*" in pegmatite that was noted in an outcrop description (RO-IL-06-023) from a 2006 exploration program carried out by Virginia (Archer & Oswald, 2008b - GM63695). The description of the mineral spodumene indicated lithium pegmatite. Prior to 2016 and the acquisition by the Company, all mineral exploration at the Property had been focused on base and precious metals.

1.7 Geological Setting and Mineralization

The Property overlies a large portion of the Lac Guyer Greenstone Belt, considered part of the larger La Grande River Greenstone Belt, and is dominated by volcanic and sedimentary rocks metamorphosed up to amphibolite facies. The Property's principal claim group is dominantly host to rocks of the Guyer Group (amphibolite, iron formation, intermediate to mafic volcanics, peridotite, pyroxenite, komatiite, as well as felsic volcanic tuffs). The amphibolite and metasedimentary rocks that trend east-west (generally moderately to steeply south dipping) through this region are bordered to the north by the Magin Formation (conglomerate and wacke) and to the south by an assemblage of tonalite, granodiorite and diorite, in addition to metasediments of the Marbot Group (conglomerate, wacke). Several regional-scale Proterozoic gabbroic dykes also cut through portions of the Property (Lac Spirt Dykes, Senneterre Dykes). The KCG claim block, located to the north of the principal claim group, is situated within the Bezier Suite (monzodiorite and granodiorite), and outside the Guyer Group.

The lithium-caesium-tantalum mineralization pegmatites on the Property, including those at CV5 and CV13, are hosted predominantly within amphibolites, metasediments, and ultramafic rocks of the Guyer Group within the principal claim group.

The geological setting is prospective for multiple commodities over several different deposit styles, including orogenic gold (Au), volcanogenic massive sulphide (Cu, Au, Ag), komatiite-ultramafic (Au, Ag, PGE, Ni, Cu, Co), and LCT pegmatite (Li, Cs, Ta, Ga, Rb).

Exploration of the Property has outlined three (3) primary mineral exploration trends, crossing dominantly east-west over large portions of the Property's principal claim group – Golden Trend (gold), Maven Trend (copper, gold, silver), and CV Trend (LCT Pegmatite). The Golden Trend is focused on the northern areas of the Property, the Maven Trend in the southern areas, and the CV Trend "sandwiched" between. Historically, the Golden Trend has received the exploration focus followed by the Maven Trend. However,

the identification of the CV Trend and the numerous lithium-tantalum pegmatites discovered to date represents a previously unknown LCT pegmatite district that was first recognized in 2016/2017 by Dahrouge Geological Consulting Ltd. and the Company.

The CV LCT Pegmatite Trend is currently recognized as an approximate 1-km-wide and 25+ km-long corridor, which is host to numerous distinct LCT pegmatite occurrences, and extends in a general east-west direction across the central portions of the Felix, FCI West, Deca-Goose, FCI East, and Corvette Main claim blocks. The trend is interpreted to extend across the majority of the principal claim group of the Property (~50 km); however, large areas remain to be explored for LCT pegmatite.

To date, nine (9) distinct lithium pegmatite clusters have been discovered along this trend at the Property – CV4, CV5, CV8, CV9, CV10, CV12, CV13, CV14 and CV15. Each of these clusters includes multiple lithium pegmatite outcrops in close proximity and oriented along the same local trend and has been grouped to simplify the exploration approach and discussion.

The lithium-caesium-tantalum mineralization at the Property is observed to occur within quartz-feldspar LCT pegmatites, which may outcrop as high-relief 'whale-back' landforms as well as low-relief landforms. The pegmatite is often very coarse-grained and off-white in appearance, with darker sections commonly composed of smoky quartz (impure SiO_2), and occasional muscovite and tourmaline, and lighter sections composed of dominantly feldspars (albite and microcline). Minor accessory and trace minerals may include beryl, chlorite, tantalum, lepidolite, and apatite.

Spodumene ($LiAlSi_2O_6$) is the dominant lithium mineral identified at all the lithium occurrences documented to date at the Property and occurs as centimetre to metre-scale crystals. The colour of the spodumene crystals ranges from cream to light grey-green over the CV5 and CV13 Pegmatite area, to a more whitish colour in the pegmatites to the west (CV8, CV9, CV10 and CV12).

Of the nine (9) LCT pegmatite clusters identified at the Property to date, three (3) (CV5, CV13 and CV12) have documented pollucite ($(Cs,Na)_2Al_2Si_4O_{12}\cdot 2H_2O$) mineralization in drill hole and/or channel sampling. The Rigel and Vega caesium zones – nested entirely within the CV13 Pegmatite – are marked by significant occurrences of pollucite-hosted caesium. In addition to lithium and caesium, the pegmatites at Shaakichiuwaanaan typically carry a significant tantalum component, present in the form of Tantalum.

At the CV5 Pegmatite, multiple individual spodumene pegmatite dykes have been geologically modelled. However, a vast majority of the CV5 Mineral Resource is hosted within a single, large, principal spodumene pegmatite dyke, which is flanked on both sides by multiple, subordinate, subparallel trending dykes. The CV5 Pegmatite, including the principal dyke, is modelled to extend continuously over a lateral distance of

at least 4.6 km and remains open along strike at both ends and to depth along a large portion of its length. The width of the currently known mineralized corridor at CV5 is approximately ~500 m, with spodumene pegmatite intersected at depths of more than 450 m in some locations (vertical depth from surface). The pegmatite dykes at CV5 trend west-southwest (approximately 250°/ 070 using the right-hand rule), and therefore dip northerly, unlike the host amphibolites, metasediments, and ultramafic rocks, which dip moderately in a southerly direction.

At the CV13 Pegmatite, surface mapping and drilling completed to date interprets a series of flat-lying to moderately dipping (northerly), sub-parallel trending LCT pegmatite bodies, of which three (3) appear to dominate. The pegmatite bodies are coincident with the apex of a regional structural flexure whereby the pegmatite manifests a west arm trending ~290° and an east arm trending ~230°. The Rigel and Vega caesium zones are hosted entirely within the CV13 Pegmatite.

1.8 Deposit Types

The primary target and deposit model for the Property are LCT pegmatites. Most LCT pegmatites are hosted by metamorphosed supracrustal rocks in the upper greenschist to lower amphibolite metamorphic grades. LCT pegmatite intrusions generally are emplaced late during orogeny, with emplacement being controlled by pre-existing structures. Typically, they are located near evolved, peraluminous granites (i.e., S-Type) and leucogranites from which they are inferred to be derived by fractional crystallization. In cases where a parental granite pluton is not exposed, one is inferred to lie at depth.

1.9 Exploration

The Company's non-drilling exploration activities (2017 through 2024) at the Property include surface mapping and rock sampling, prospecting, channel sampling, ground and airborne geophysics, and remote sensing surveys. The focus has been predominantly on LCT pegmatite, although significant base and precious metal exploration has also been completed.

In 2017, the Company completed a short reconnaissance program, which confirmed the presence of coarse-grained spodumene in two (2) sub-parallel trending pegmatite outcrops at the CV5 Pegmatite (3.48% Li_2O and 1.22% Li_2O) (Smith D. L., 2018 - GM70744). The Company expanded upon the work in 2018 with additional surface prospecting and rock sampling, which resulted in the discovery of two (2) new LCT pegmatite outcrops, including the CV4 Pegmatite (0.74% Li_2O) (Smith D. L., 2019). Channel sampling was also completed at the CV5 Pegmatite with results including 2.28% Li_2O and 208 ppm Ta_2O_5 over 6 m (CV1-CH03) and 1.54% Li_2O and 136 ppm Ta_2O_5 over 8 m (CV1-CH01).

In 2019, the Company expanded its scope of exploration with a stronger focus on base and precious metals due to market conditions at the time. The field work included prospecting, rock sampling, and soil sampling and resulted in the discovery of new occurrences of gold (West Golden Gap, New Lac Bruno), copper-gold-silver (Elsass, Lorraine, Black Forrest, Hund), and lithium-tantalum (CV8, CV9 and CV10 pegmatites) (Smith D. L., 2020 - GM71564). Rock sample results ranged from nil to 11.9 g/t Au, nil to 171 ppm Ag, nil to 8.15% Cu, nil to 4.72% Li_2O, and nil to 1,011 ppm Ta_2O_5.

No field work was completed in 2020; however, a re-interpretation of historical induced polarization and resistivity surveys and airborne magnetic survey data was completed. The work indicates that a significant potential for follow-up drilling at Golden Gap remains.

Exploration continued in 2021 and focused on the Maven and CV trends ahead of initial diamond drilling, which followed in the fall. Airborne and surface work included geological mapping and rock sampling, ground-based induced-polarization and resistivity survey, airborne magnetic survey, and a remote sensing survey (Smith, Mickelson, & Blu, 2023 - GM73402). The most significant result of the 2021 mapping and rock sampling program was the recognition of the CV12 Spodumene Pegmatite cluster, where numerous lithium pegmatite outcrops were discovered. Analytical results at CV12 ranged from nil to 5.98% Li_2O and 49 to 1,478 ppm Ta_2O_5, with an average of 2.83% Li_2O and 438 ppm Ta_2O_5.

In 2022, the exploration campaign reoriented firmly towards LCT pegmatite, with only minor base and precious metals work completed. Exploration included prospecting and rock sampling, surface outcrop mapping, channel sampling, and a LiDAR and orthophoto survey. The most significant result of the 2022 surface exploration was the discovery of the CV13 LCT Pegmatite cluster, situated between the CV8 and CV12, and the CV5 Spodumene Pegmatite clusters. Of the 38 pegmatite samples collected at CV13, a total of 14 assayed > 1% Li_2O to a peak of 3.73% Li_2O. Outcrop channel sampling followed with results including 14.2 m at 1.17% Li_2O and 13.1 m at 1.57% Li_2O. Outcrop channel sampling was also completed at other known lithium pegmatite clusters and returned 1.5 m at 1.12% Li_2O (CV4), 5.6 m at 1.93% Li_2O (CV8), 15.0 m at 0.46% Li_2O (CV9), and 21.9 m at 0.80% Li_2O; 7.7 m at 1.46% Li_2O; and 3.3 m at 1.58% Cs_2O (CV12).

Surface exploration in 2023 included an orientation IP-Resistivity geophysical survey over a large portion of the CV5 Spodumene Pegmatite, a ground magnetic survey over the CV5 to CV13 corridor, a ground gravity orientation survey, as well as geological mapping and rock sampling, prospecting, and channel sampling. Additionally, an airborne magnetic and radiometric survey was completed over the Corvette Main, FCI East, and Felix claim blocks. The most significant result of the 2023 surface exploration was the discovery of the CV14 LCT Pegmatite cluster (0.94% Li_2O and 0.86% Li_2O in outcrop), situated approximately 1.5 km along the geological trend of the CV10 LCT Pegmatite.

In 2024, non-drill-related exploration by the Company included a surface exploration program of detailed geological mapping at the CV5 and CV13 pegmatites, channel sampling at multiple LCT pegmatite clusters, and regional prospecting. A LiDAR and orthophoto survey, and a heliborne magnetic and radiometric survey were also completed in 2024 over the JBN-57 claim block. The most significant result of the 2024 surface exploration was the discovery of the CV15 LCT Pegmatite cluster (2.11% Li_2O, 1.55% Li_2O, and 1.02% Li_2O in outcrop), situated approximately 1.9 km southwest and along geological trend from CV14, and collectively outlines a larger ~5.5 km long prospective trend extending from the CV9 LCT pegmatite cluster to CV15, now referred to as the Mickel Trend.

Through 2025 to the Issue Date of this report, non-drill related exploration by the Company included a surface exploration program of detailed geological mapping at the CV5 and CV13 pegmatites, channel sampling at multiple LCT pegmatite clusters, and regional prospecting. Additionally, a seismic survey was completed over the CV5 area to further constrain overburden thickness and bedrock topography. The surface exploration program began in June 2025, and no results have been reported to date by the Company.

1.10 Drilling

The Company completed drilling at the Property in 2021 (Maven and CV trends), 2022 (CV Trend), 2023 (CV Trend, Camp), 2024 (CV Trend), and 2025 (CV Trend, and north CV5). The Shaakichiuwaanaan database includes 800 diamond drill holes (DDH) and three (3) rotary drill holes completed over the 2021, 2022, 2023 and 2024 programs (through hole CV24-787), for a collective total of 235,061 m, as well as outcrop channels totalling 800 m.

At the Maven Trend (2021), the program returned anomalous to moderate grades over several drill holes, including individual sample highs comparable to prior surface results – 3.1 m of 0.34% Cu, 0.21 g/t Au, and 6.7 g/t Ag within a larger interval of 28.4 m of 0.12% Cu, 0.06 g/t Au, and 2.3 g/t Ag (CF21 013, Lorraine), and 0.2 m of 2.12% Cu, 0.26 g/t Au, and 25.4 g/t Ag (CF21-008A, Tyrone-T9). Mineralization consists of visible chalcopyrite present as stringers and disseminations.

The drilling programs at the LCT Pegmatite Trend (from September 2021 through 2024) were very successful, returning wide and well-mineralized intervals of lithium (spodumene) and caesium (pollucite) pegmatite at the Property, for which Mineral Resources have been determined for the CV5 and CV13 LCT pegmatites.

Highlights for lithium at CV5 include:

- 148.7 m at 0.92% Li_2O, including 73.0 m at 1.09% Li_2O (CF21-001, the ‘discovery hole’).

- 152.8 m at 1.22% Li_2O, including 66.0 m at 1.51% Li_2O (CV22-030).
- 156.9 m at 2.12% Li_2O, including 25.0 m at 5.04% Li_2O or 5.0 m at 6.36% Li_2O (CV22-083).
- 131.2 m at 1.96% Li_2O, including 57.0 m at 2.97% Li_2O (CV22-100).
- 83.7 m at 3.13% Li_2O, including 19.8 m at 5.28% Li_2O and 5.1 m at 5.17% Li_2O (CV23-105).
- 172.4 m at 0.95% Li_2O, including 34.5 m at 1.85% Li_2O (CV23-199).
- 123.3 m at 1.66% Li_2O, including 54.9 m at 2.50% Li_2O (CV24-374).

Highlights for lithium at CV13 include:

- 22.6 m at 1.56% Li_2O, including 6.0 m at 3.19% Li_2O (CV22-092).
- 28.7 m at 1.49% Li_2O, including 20.4 m at 2.03% Li_2O (CV23-311).
- 51.7 m at 1.77% Li_2O, including 9.7 m at 5.16% Li_2O (CV24-525).
- 34.4 m at 2.90% Li_2O, including 21.9 m at 3.58% Li_2O (CV24-470).

Highlights for caesium at CV13 (Rigel Zone) include:

- 5.9 m at 11.19% Cs_2O, including 1.0 m at 22.69% Cs_2O (CV23-271).
- 5.0 m at 13.32% Cs_2O, including 2.0 m at 22.90% Cs_2O (CV23-255).
- 3.2 m at 10.24% Cs_2O, including 1.1 m at 26.61% Cs_2O (CV23-204).
- 4.5 m at 3.36% Cs_2O (CV23-198).

Highlights for caesium at CV13 (Vega Zone) include:

- 18.1 m at 2.71% Cs_2O, including 7.4 m at 5.45% Cs_2O (CV24-754).
- 11.1 m at 4.87% Cs_2O, including 7.1 m at 7.39% Cs_2O (CV24-520).
- 5.7 m at 4.97% Cs_2O, including 3.0 m at 8.20% Cs_2O (CV24-525).
- 9.6 m at 1.59% Cs_2O, including 4.4 m at 2.34% Cs_2O (CV24-579).

1.11 Sample Preparation, Analyses and Security

Sample preparation and analysis of the surface and diamond drilling program samples have been completed at various independent commercial laboratories in Canada. All laboratories used are properly certified and accredited.

It is the QP's opinion that the Company has utilized appropriate quality assurance / quality control (QA/QC) protocols for all its mineral exploration programs. This includes the use of certified reference materials, blanks, duplicates, and check analysis at a secondary laboratory.

1.12 Data Verification

Data validations supporting the Mineral Resource estimation for the Shaakichiuwaanaan Project, including both the CV5 and CV13 pegmatites, have been carried out by qualified persons. This includes site visits, check sampling of drill core, validation of multiple collar locations, and validation of the drill hole database.

It is the QP's opinion that the sampling practices of the Company meet current industry standards. The QP also believes that the sample database provided by the Company and validated by the QP is suitable to support the Mineral Resource estimation.

1.13 Mineral Processing and Metallurgical Testing

1.13.1 Metallurgical Test Work

The Company engaged Primero and SGS Canada in 2023 to assist with a metallurgical test work program for the CV5 Deposit. Test work was completed at the SGS Lakefield Ontario facility. The scope of the program included both mineralogical characterization and metallurgical test work. Both SGS and Primero are independent of the Company and are industry-recognized in lithium pegmatite processing. The objectives of the metallurgical test work program are to confirm the dominant lithium-bearing mineral species for CV5 and evaluate the beneficiation performance of the deposit using a conventional spodumene Dense Media Separation (DMS) flowsheet. Target concentrate specifications were set at > 5.5% Li_2O and < 2.0% Fe_2O_3.

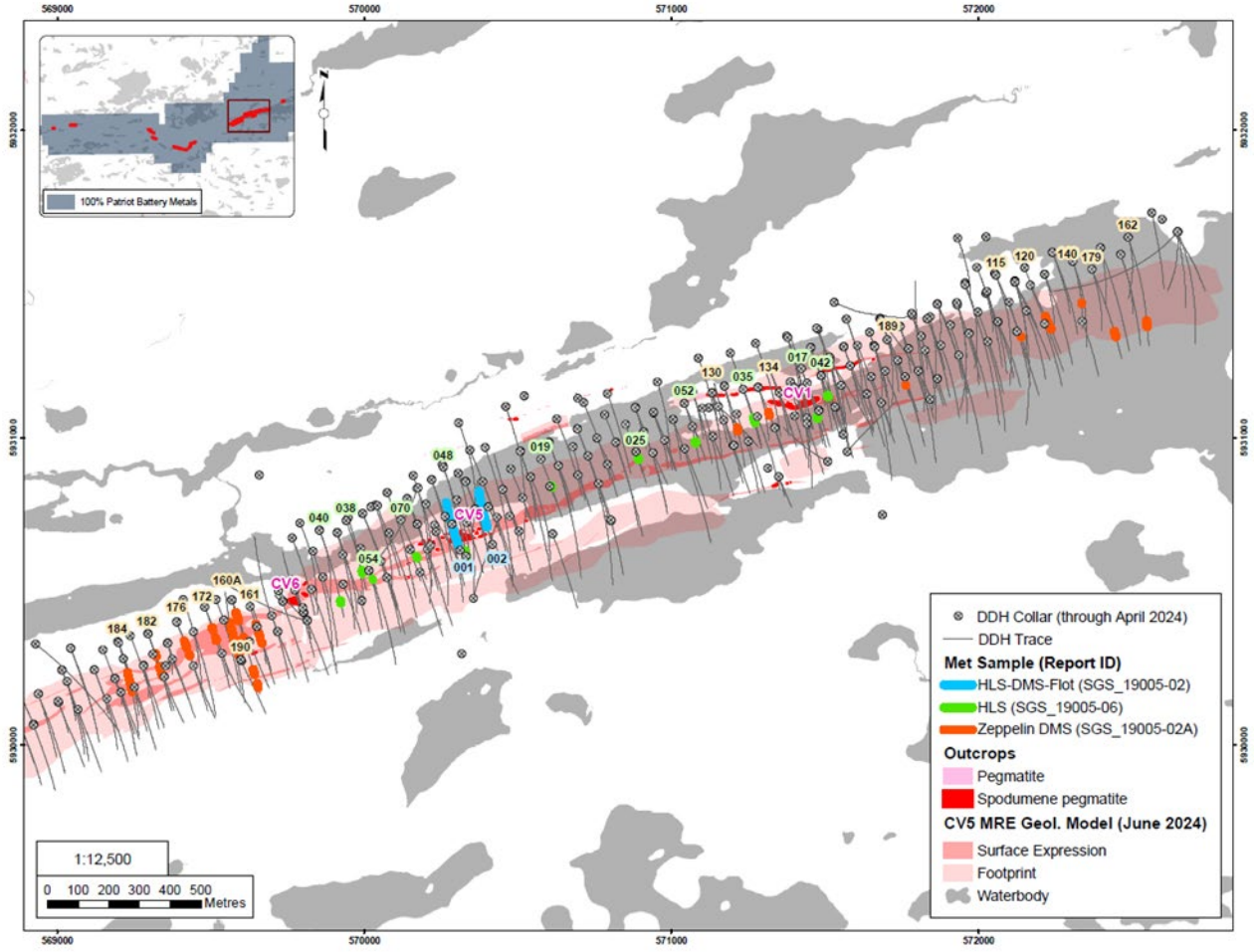
Mineralogical characterization consisted of TIMA-X (Quantitative SEM), Electron Probe Micro-Analysis (EPMA), Laser Ablation by Inductively Coupled Plasma Mass Spectrometry (LA by ICP-MS), X-ray diffraction (XRD) analysis, and chemical assays. Metallurgical test work included Heavy Liquid Separation (HLS) and DMS pilot scale test work. Preliminary flotation test work was completed on the DMS bypass fraction and DMS "middlings" (i.e., second stage DMS floats).

Test work completed to date indicates that the CV5 Pegmatite can be processed by DMS-only given the favourable metallurgical test work results. Test work revealed that a top size of 9.5 mm reporting to a gravity separation process provided relatively consistent results in terms of concentrate Li_2O grade and Li_2O recovery.

Testing of CV5 was done both in terms of variability across region, throughout the width of the dykes (i.e., at contact with host rock, in the centre of formation) as well as testing of the host rock separately (to properly gauge impacts of host rock dilution on metallurgical performance). The broad range of spatial locations with a range of gangue mineral assemblages, lithium and iron head grades provides thorough testing of the material. There is a strong indication that the positive HLS recoveries can be expected from other coarse spodumene samples taken from CV5.

Approximately 880 kg of quarter-core NQ and 1,826 kg half-core NQ of lithium-bearing pegmatite samples from CV5 has been used in the metallurgical test program. Additionally, 389 kg of half-core NQ samples, representing the host rock surrounding CV5, has been tested metallurgically. The length of drill core tested from CV5 Pegmatite sums up to approximately 1,136 m.

Figure 1.1: Metallurgical Drill Core Map for CV5



Source: PEA 2024, BBA.

Although not considered in the CV5 region, five (5) pegmatite composites were generated from drill core from the CV13 Pegmatite, representing a combined total of 42.7 kg of quarter-core NQ. Results were both

promising and consistent with CV5 results, with global lithium recoveries of 75% to 80%. These preliminary results offer future exploration potential.

Summary of the test work and findings:

- Test work supports a DMS-only process flowsheet to produce a spodumene concentrate grade of > 5.5% Li_2O and < 2.0% Fe_2O_3. Test work Li_2O recoveries of 70% to 85% were achieved for HLS test work (for feed grade in the range 1.0% to 1.5% Li_2O, respectively).
- Test work completed on CV5 includes five (5) DMS tests and 26 Heavy Liquid Separation (HLS) and magnetic separation tests. The HLS and magnetic separation tests were conducted using composites from across the CV5 Deposit.
- Coarse spodumene was found to be the dominant lithium mineral species across all samples, with minor quantities of lepidolite (values range between 0% to 4.3% with an average of 1.0%) and moderate quantities of mica (values range between 0% to 17.1% with an average of 6.5%) observed.
- Five (5) pilot DMS tests (cyclone diameter of 250 mm) were completed. Table 1.1 summarizes the global Li_2O feed grades (before fines screening), global lithium recoveries and the Li_2O and Fe_2O_3 grades of the concentrates achieved. These results strongly support adopting a DMS-only process flowsheet.

Table 1.1: Pilot DMS Results

DMS Feed Li_2O Grade (%)	Global DMS Lithium Recovery (%)	Concentrate Li_2O Grade (%)	Concentrate Fe_2O_3 Grade (%)
1.01	77.40	5.64	0.55
1.07	79.00	5.77	0.62
1.16	69.40	6.21	0.60
2.01	80.62	6.59	0.81
2.28	81.87	6.60	0.49

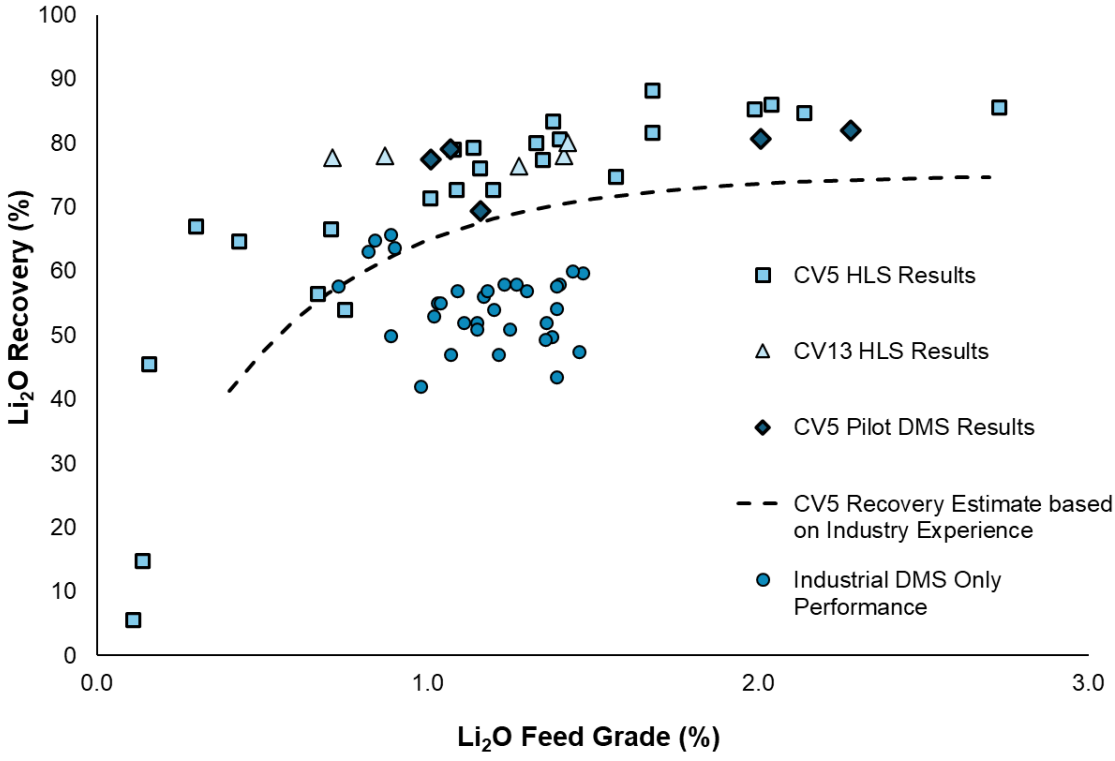
- Fe_2O_3 grades in HLS concentrates were in the range 0.52% – 1.79% and after magnetic separation was applied to 17 of the 26 composites, all concentrates were < 2.0% Fe_2O_3.
- The 26 CV5 HLS variability test results were adjusted to more appropriately represent the recoveries expected in an operating DMS plant. After fitting a trend to this data, it indicates:
  - Recoveries of 70% – 75% Li_2O expected at feed grades above 1.4% Li_2O.
  - Recoveries of 60% – 70% Li_2O expected at feed grades of 0.9% – 1.4% Li_2O.

- Recoveries of 50% – 60% Li_2O are possible at feed grades of 0.7% – 0.9% Li_2O.

- Some samples assayed contained elevated grades of Ta_2O_5 (with values as high as 300 ppm). Preliminary Tantalum suggests that tantalum can be recovered and concentrated from the CV5 deposit. There is further work warranted to assess the best flowsheet integration method to recover the tantalum from the non-product streams of the DMS plant.

The testwork results for both HLS (from CV5 and CV13) and DMS (from CV5) are shown in Figure 1.2. The concentrate grades achieved are all greater than 5.5% Li_2O and lower than 2.0% Fe_2O_3. All tests were performed on samples that had a 9.5-mm top size. Five (5) pilot scale DMS tests were conducted in 2023 to 2025, which resulted in lithium recoveries of 77.40%, 79.00%, 69.40%, 80.62% and 81.87%, and concentrate Li_2O grades of 5.64%, 5.77%, 6.21%, 6.59% and 6.60% respectively, for feed Li_2O grades of 1.01%, 1.07%, 1.16%, 2.01% and 2.28%, respectively. The diameter of the cyclone was 250 mm. The concentrate generated from one (1) of the DMS tests is shown in Figure 1.3.

Figure 1.2: Metallurgical Testwork Recovery Results & Industry-Based Recovery Estimates for 3x Size Range DMS Process Plant



The lithium recovery expected from a three-size range, DMS concentrator treating material 9.5 mm to 0.65 mm is shown in Figure 1.2. The recovery is deemed to be a relationship with the concentrators Li_2O feed grade. Expected concentrator recoveries are lower than testwork results based on scale-up

factors that are driven by the effects of both larger diameter cyclones and the crowding effect seen in the DMS sinks. This variation between laboratory testwork results and those achieved in operating plants has, to date, been observed within the industry with respect to operating spodumene DMS concentrators. Lithium recoveries achieved by other DMS-only concentrators are shown for reference ('Industrial DMS Only Performance' in Figure 1.2). The Project's higher expected recovery is due to both the quality of the material (large spodumene grains with a narrow grain size distribution) and the three-size range DMS plant (which lessens the impact of particle size effect in the DMS process).

Figure 1.3: Final Concentrate Product Generated from the CV5 Pegmatite via DMS



Recommendations for the next steps in the testwork program are:

- Further gravity and magnetic separation recovery testwork is recommended to improve the recovery of tantalum in a manner that is integrated in the spodumene flowsheet.
- Due to the width and orientation of the CV5 Pegmatite lenses, the expected dilution of the plant feed is expected to be relatively low. However, there may be opportunities to maximize the extraction of spodumene concentrate from the deposit if parts of the deposit with higher dilution are directed to an ore sorting processing solution. Ore sorting testwork is planned for the next phase of testwork.
- Given the heavy reliance of a DMS flowsheet on the particle size distribution, it is recommended to characterize spodumene samples representative of a Run-of-Mine material generated from blasting activities. Samples generated from crushed drill core (basis of most exploratory testwork) do not generate the same size distribution. These samples have fewer fines in them. Therefore, a blasted

bulk sample would validate the size distribution used in the design (which assumes more fines than those measured in the lab).

1.14 Mineral Resource Estimate

The MRE has been completed for the Shaakichiuwaanaan Project for the CV5 and CV13 pegmatites, including the Rigel and Vega caesium zones, and does not include any of the other known spodumene pegmatite clusters at the Property.

The Shaakichiuwaanaan Consolidated MRE (and host geological models), which includes the CV5 and CV13 pegmatites only, are supported by 720 DDH of NQ (predominant) or HQ size, completed over the 2021, 2022, 2023 and 2024 (through the end of 2024 – drill hole CV24-787) programs, for a collective total of 227,703 m, as well as 604 m of outcrop channels. This equates to 555 holes (188,695 m) and 179 m of outcrop channels at CV5, and 165 holes (39,008 m) and 425 m of outcrop channels at CV13. Included within the CV13 dataset are 32 holes totalling 7,808 m, completed over the 2022, 2023 and 2024 programs, as well as 7 m of channels, which were used to support the Vega and Rigel Caesium Zone MRE and geological models.

The Mineral Resource (Table 1.2 and Table 1.3) reported is effective as of June 20, 2025, and has been tabulated in terms of pit and underground mining shapes. Both underground and open pit conceptual mining shapes were applied as constraints to demonstrate reasonable prospects for eventual economic extraction. Open pit cut-off is 0.40% L_2O, and underground cut-off is 0.60% and 0.70% for CV5 and CV13, respectively.

Table 1.2: Shaakichiuwaanaan Mineral Resource Estimate (CV5 & CV13 Pegmatites)

Pegmatite	Classification	Mass (t)	Li_2O (%)	Cs_2O (%)	Ta_2O_5 (ppm)	Ga (ppm)	Contained LCE (Mt)
CV5 & CV13	Indicated	107,991,000	1.40	0.11	166	66	3.75
	Inferred	33,380,000	1.33	0.21	155	65	1.09

The Caesium zones have a further metallurgical cut-off of 0.5% Cs_2O applied within the CV13 pit-constrained material.

Table 1.3: Mineral Resources at Rigel and Vega Caesium Zones Within the CV13 Pegmatite

Caesium Zone	Classification	Tonnes (t)	Li_2O (%)	Cs_2O (%)	Ta_2O_5 (ppm)	Contained Cs_2O (t)
Rigel	Indicated	163,000	1.78	10.25	646	16,708
	Inferred	–	–	–	–	–
Vega	Indicated	530,000	2.23	2.61	172	13,833
	Inferred	1,698,000	1.81	2.40	245	40,752
Rigel + Vega	Indicated	693,000	2.13	4.40	283	30,541
	Inferred	1,698,000	1.81	2.40	245	40,752

1.15 Mineral Reserve Estimate

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).

In line with these standards, the Mineral Reserves are based solely on Measured and Indicated Mineral Resources. Inferred mineral Resources have not been included in the Mineral Reserve. Any Inferred Resources that fall within the mine design envelope have been treated as waste and assigned a grade of 0% Li_2O for the purposes of mine planning and economic evaluation.

The Proven and Probable Reserve for the Shaakichiuwaanaan Project is estimated at 84.30 Mt at an average grade of 1.26% Li_2O for 492.4 kt of contained lithium oxide, as summarized in Table 1.4.

It should be noted that there are no Proven Reserves; only Probable Reserves are declared.

Table 1.4: Shaakichiuwaanaan Project Mineral Reserve

Area	Category	Tonne (Mt)	Grade (%Li_2O)	Contained Li_2O (kt)	Contained Lithium (Li) (kt)	Contained LCE (kt)
Open Pit	Proven	0	0	0	0	0
	Probable	49.2	1.12	551.9	256.4	1,364.7
	Proven and Probable	**49.2**	**1.12**	551.9	**256.4**	1,364.7
Underground	Proven	0	0	0	0	0
	Probable	35.1	1.45	508.0	236.0	1,256.0
	Proven and Probable	**35.1**	**1.45**	508.0	**236.0**	1,256.0
Total (Open Pit + Underground)	Proven	0	0	0	0	0
	Probable	84.3	1.26	1,059.9	492.4	2,620.7
	Proven and Probable	**84.3**	**1.26**	1,059.9	**492.4**	2,620.7

**Notes on Mineral Reserves:*

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).*
2. *The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. As such, the Mineral Reserves are based on the Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.*
3. *Mineral Reserves are estimated using a long-term lithium price of USD 1,303/t of spodumene concentrate @5.5% Li_2O and an exchange rate CAD/USD of 1.32.*
4. *The qualified person for the estimate is Carl Michaud, P.Eng., MBA. The estimate has an effective date of September 11, 2025.*
5. *The Mineral Reserves for open pit are estimated using a cut-off grade of 0.40% Li_2O. Open pit marginal material containing a grade above 0.37% Li_2O is also included within this statement. Mineral Reserves for underground stoping are estimated using a cut-off grade of 0.70%. Underground development tonnages containing material above 0.37% Li_2O are also included in the statement.*
6. *The following mill recovery equation was used in the cut-off grade recovery:*

$$Mill\ Recovery = \frac{75 * (1 - e^{-1.995 * Li2O\ feed\ Grade})}{100}$$

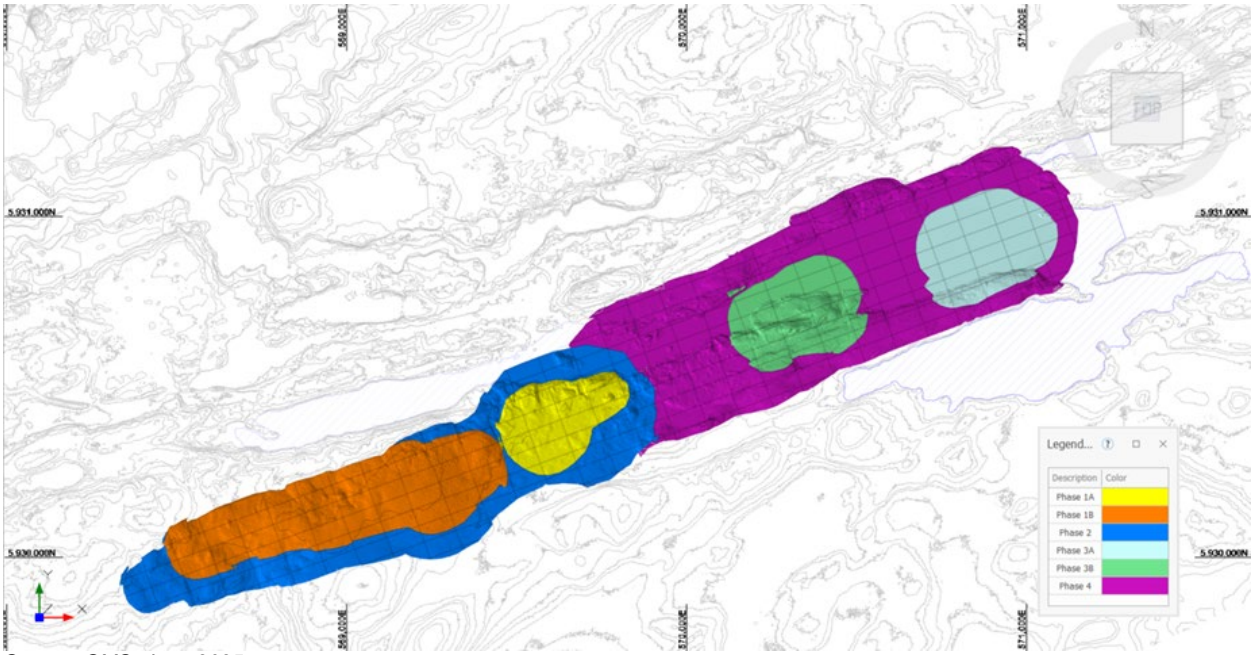
7. *The Open Pit Strip Ratio is 3.40, and the dilution factor is 2.0% based on the smallest mining unit (SMU). The OP mine mining recovery is 97%.*
8. *The underground mine average external dilution factor is 12.7% including 3.9% for backfill dilution and 8.8% for ELOS dilution.*
9. *For the underground Mineral Reserves, a minimum mining width of 5 m was applied with a mining recovery of 90% for all stopes, while 100% extraction was assumed for all development mining.*
10. *Contained Lithium Oxide (Li_2O), Lithium and LCE are reported without accounting for metallurgical recovery*
11. *Total may not sum due to rounding; rounding followed the recommendations of the NI 43-101.*

1.16 Mining Methods

The CV5 Shaakichiuwaanaan Deposit consists of a large main pegmatite dyke, accompanied by several smaller dykes striking approximately east-west. Planned mining operations will incorporate both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The pegmatite dykes dip at approximately 80° to the north. The deposit extends over a strike length of

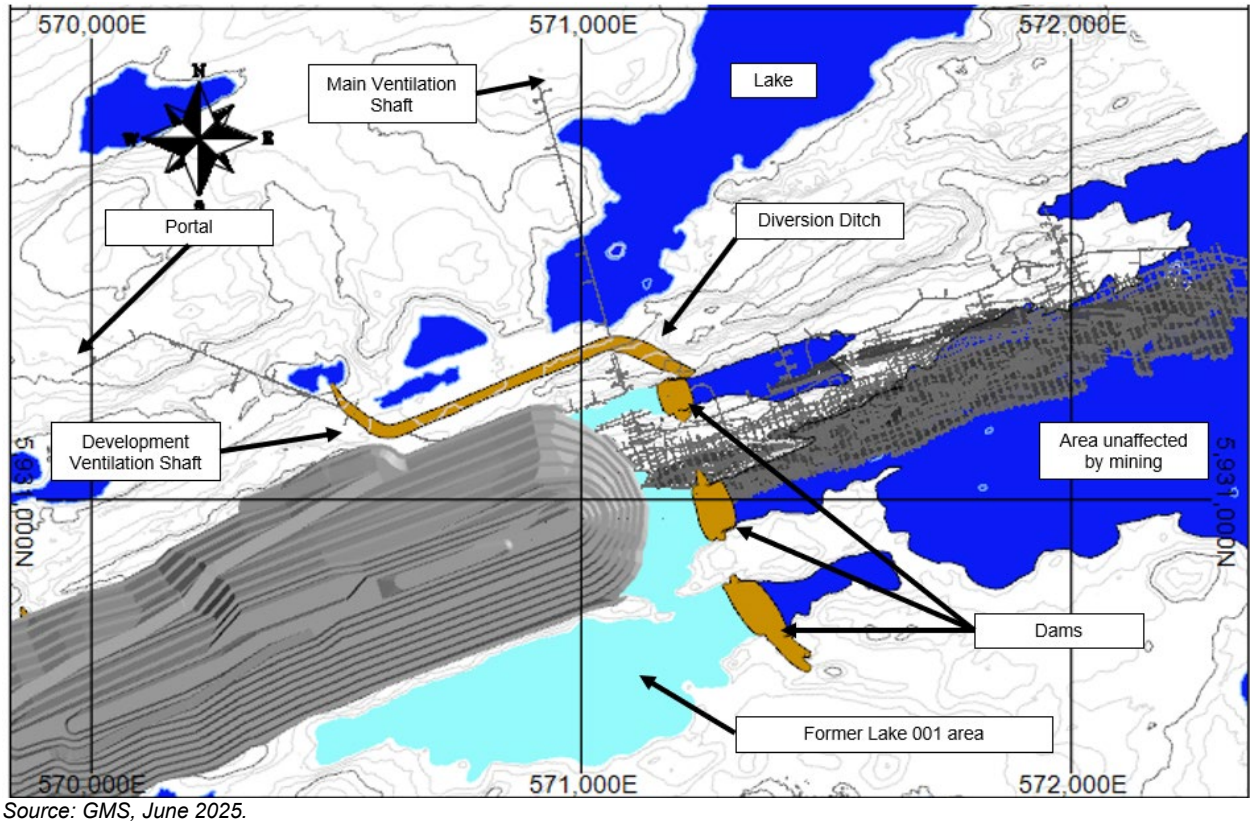
approximately 4.6 km with mineralization modelled from surface to a depth of 650 m. A significant portion of the orebody is located under Lake 001, which will require partial dewatering to enable open pit operation. Figure 1.4 presents the final configuration of the pits and Figure 1.5 presents the underground mine regarding the surface constraints.

Figure 1.4: Open Pit Phase Limits – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 1.5: Underground Mine and Surfaces Constraints – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

The average grade of the Mineral Reserve is approximately 1.26% Li_2O after dilution and mining recovery, where the mineralized material from the pit provides a grade of 1.12% Li_2O and 1.45% Li_2O from the underground. Figure 1.6 shows a summary of the mining schedules for both mines. The combination of the two (2) mines allows for a smooth lithium grade in the mill, which is key for dense media separation mills. Blending is key in this Project, allowing for smooth lithium grade and impurities as seen in Figure 1.7, demonstrating the smooth mill recovery mirrored by the constant output of concentrate.

Figure 1.6: CV5 Open Pit and Underground – Total Material Mined

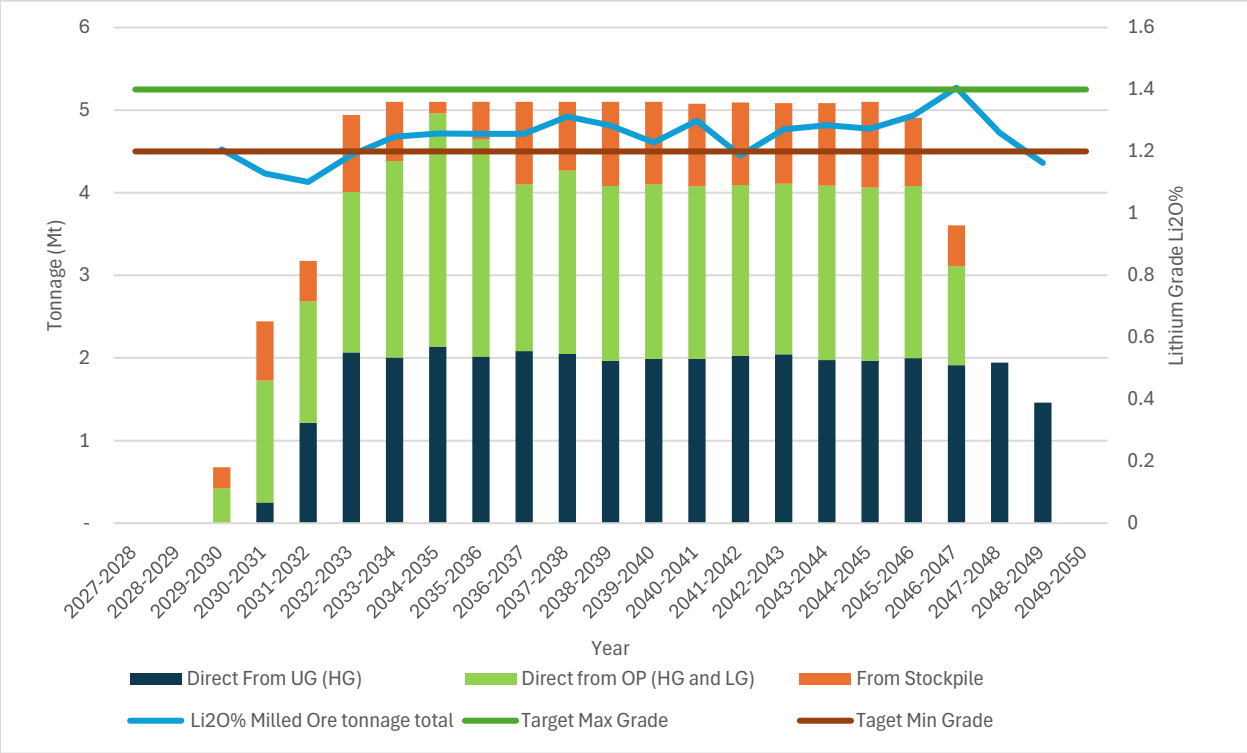
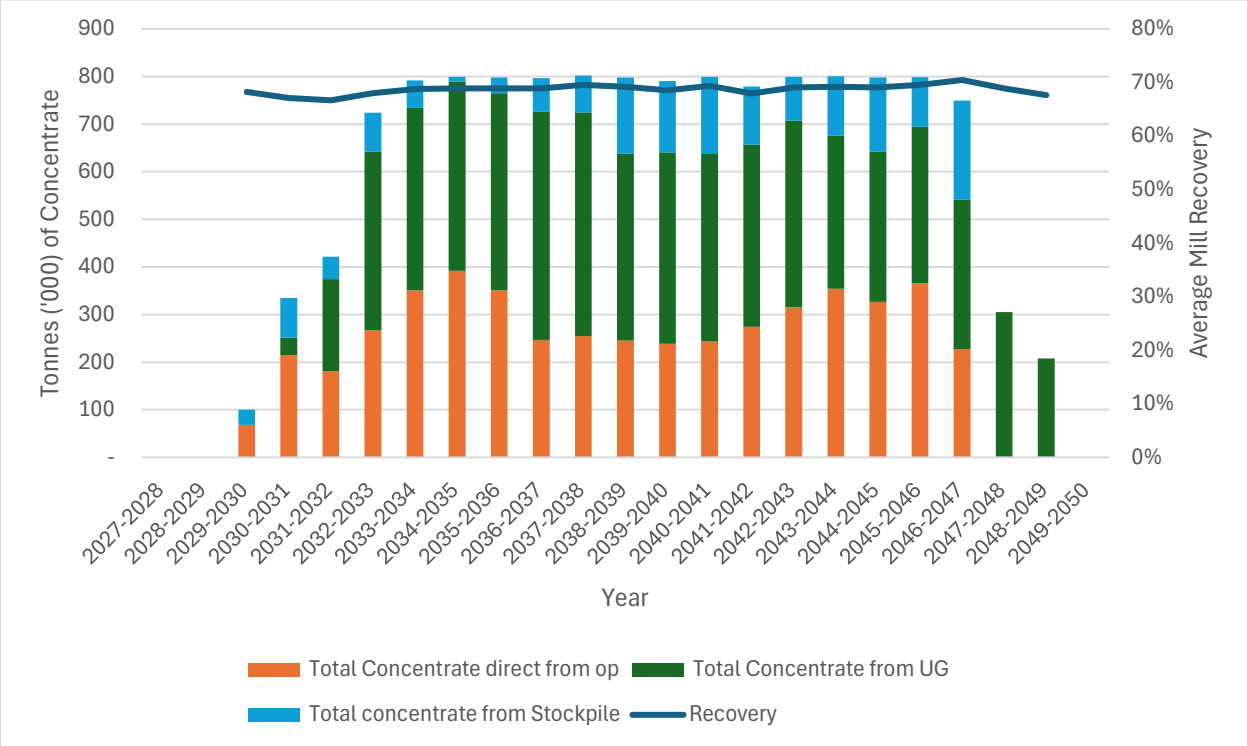


Figure 1.7: Concentrate Production Schedule



Source: GMS, June 2025.

1.16.1 Open Pit

The OP operation will utilize a fleet of diesel-powered equipment, including drills, hydraulic shovels, and off-highway haul trucks. The Project consists of a single pit with two (2) stages, each of which will be mined using sub pits. In total, the operations will be executed in six (6) separate sub pits. The OP peak mining rate is 23.0 Mtpa over a Life-of-Mine (LOM) of 19 years, including the two (2) years of pre-production period. A total of 49.2 Mt at 1.12% Li_2O of ore will come from the open pit. A total of 216.7 Mt of material, including ore, waste rock and overburden, will be moved during the life of mine, resulting in an average stripping ratio of 3.4.

Open pit optimization was conducted using GEOVIA WhittleTM version 2022 (Pseudo Flow algorithm) to determine the optimal economic pit shell to guide the pit design process. Different revenue factors and constraints have been applied for the west and east parts of the pit.

1.16.1.1 West Pit

The West Pit is physically constrained by the location of the first dam construction. Prioritizing the mining of the West Pit allows early access to ore content while delaying most of the impacts on Lake 001. Table 1.5 presents the Whittle optimization results for the West Pit, based on Measured and Indicated Resources (M&I).

Table 1.5: M&I Push Back Selection @ USD 1,303/t Conc.

Parameters	Push Back 1	Push Back 2	West Pit Final Pit Shell
Shell Number	5	12	32
Shell RF	0.24	0.31	0.51
Shell Price	313	404	665
Total Tonnage (kt)	6,573	22,373	50,262
Waste Tonnage (kt)	3,621	14,854	35,015
Strip Ratio (W:O)	1.23	1.98	2.30
Ore Tonnage (kt)	2,952	7,519	15,247
Grade (%Li_2O)	1.35	1.21	1.08

1.16.1.2 East Pit

The East Pit represents the second phase of open-pit mining of the Shaakichiuwaanaan Project. It is also physically constrained by the dam located northeast of the ultimate pit shell selected. Numbers below are inclusive of the West and East Pits.

Table 1.6: M&I Phasing Selection @ USD 1,303/t Conc.

Parameters	Push Back 1	East Pit Final Pit Shell
Shell Number	9	31
Shell RF	0.28	0.50
Shell Price	365	652
Total Tonnage (kt)	33,233	181,802
Waste Tonnage (kt)	18,574	130,070
Strip Ratio (W:O)	1.27	2.51
Ore Tonnage (kt)	14,659	51,732
Grade (%Li_2O)	1.18	1.12

1.16.2 Underground

The selected underground mining method is long-hole open stoping (LHOS), employing transverse, longitudinal, or up-hole stoping configurations. From the total of 1,548 stopes defined as economic, 72 are longitudinal retreat (5%), one (1) is up-hole stoping (<1%), and the remaining 1,475 (95%) are transverse stoping. To reflect more realistic operational constraints and optimization, 23 half-stopes have been joined to adjacent stopes, resulting in 1,525 stopes in the life of mine. The stoping sequence will be ascending from an initial undercut. Generally, stopes will be drilled from an upper access and mucked from a lower access. Stopes within sill pillars will require the upper access to be redeveloped through backfill to restore drilling access. Stopes height ranges between 30 to 36 m, with a strike length of a maximum of 17 m and a width of 7 m for longitudinal stopes and 25 m for transverse stopes. A crown pillar of 85 m of rock has been left in place, based on available data and analysis. Specific engineering studies regarding, among others, geotechnical (soil), rock mass strength, structural domaining and hydrogeology will be needed to fully fulfill the “Regulation respecting occupational health and safety in mines Article 77” as the Project progresses towards execution.

The underground Mineral Reserve estimates are based on the final underground mine design and exclude stopes located within the crown pillar, within the Open pit and within uneconomic areas. The underground mine contains 40.3 Mt of material, including 35.1 Mt at 1.45% Li_2O of ore material and 5.1 Mt of waste rock

from development. The mine will produce mineralized material for an estimated 19 years, where it will reach full production three years after its first stope is excavated. The maximum production rate will reach 5,500 tpd, or 2 Mtpa.

Following the completion of the portal excavation and construction, the Project will transition into the development phase, beginning with the advancement of the main decline toward the main zone. It is planned to use a mining contractor for the portal ground support and the pre-production development period, which is estimated to be two (2) years. Following this phase, the owner will assume responsibility for all underground activities. This decision is mainly based on the choice to use Load-Haul-Dump (LHD) BEV equipment. Using such equipment from the beginning is not deemed efficient, considering the initial CAPEX requirements and operational ramp-up needs (battery management, maintenance ramp-up, UG and OP site maturity, power grid establishment and availability during mine development phase, etc.). A development ramp-up was implemented during the first two (2) months of the Project. This ramp-up accounts for typically more challenging ground conditions within the first vertical 20 m in the James Bay area and the operational ramp-up of workforce and maintenance.

A total of 96.7 km of lateral development and 1.4 km of vertical development will be done over the life of mine.

For the Shaakichiuwaanaan Mine, 50-tonnes underground battery electric vehicles (BEVs) haul trucks have been selected for material handling to surface. It was determined that ramp haulage coupled with loading bays was economically preferable to other means of ore handling, such as conveyors or a dedicated haulage ramp. In this design, most drawpoint entrances are within 175 m of a loading bay. It is planned to use adjacent drawpoints as temporary remuck areas to optimize loading cycles.

1.17 Recovery Methods

The mineral processing facility is designed to produce spodumene concentrate from ROM. The facility will include ROM stockpiling, comminution, beneficiation, dewatering, and load-out areas. The processing facility employs a gravity-based beneficiation circuit, consisting of a two-stage DMS process for both coarse and fine size fractions. The design of the process will be executed utilizing two identical parallel process trains that could be operated independently of one another. Each process train will account for half (50%) of the crushing, beneficiation, and dewatering capacity.

The mineral processing facility has an overall availability and utilization of 81% and is designed to nominally process 5,000,000 dry tonnes per year (tpa) of mineralized material. When processing feed grades of

1.31 wt.% Li_2O, the concentrator can produce up to 827,530 tpa of spodumene concentrate with a grade of 5.50 wt.% Li_2O achieving 69.5% Li_2O recovery.

The key process areas of the mineral processing facility are:

- ROM stockpiles.
- Crushing circuit (with primary, secondary, and tertiary crushing).
- Crushed feed stockpile.
- Coarse DMS circuit.
- Fine DMS circuit.
- Recrush DMS circuit.
- Magnetic separation and final product handling.
- Fines bypass + middlings dewatering and handling.
- Final reject handling.

1.18 Project Infrastructure

The Project is close to significant existing infrastructure, including an all-season road and Hydro-Québec's power grid.

Main site facilities include the concentrator and underground mine buildings, garages for mining fleets, light vehicles, and highway trucks, as well as administrative offices and lab, dry rooms, warehouses, and auxiliary buildings.

In addition, the main site will include waste rock piles and tailings management systems, complete with ditching and pond systems for water management. Lake water intake and water treatment plants are also included.

A new 120 kV transmission powerline will be constructed to connect the site to Hydro-Québec's power grid, located approximately 55 km away.

Other facilities include an emulsion plant, explosive storage magazines, fuel storage pads, refuelling stations, and a permanent workers' camp to accommodate construction and operational personnel.

Highway trucks will transport spodumene concentrate approximately 834 km along the existing road network to the Transshipment Centre located in Matagami, QC. Once in Matagami, the product will be transferred to railcars for rail transport to the Port of Grande-Anse via Canadian National's Railway network. The Matagami Transshipment Centre will be upgraded with storage infrastructure and material handling equipment.

1.19 The Project's Market Studies and Contracts

The lithium spodumene market is characterized by dynamic pricing, shaped by a confluence of multifaceted factors. While price volatility is expected to persist in the medium term, a price in the range of USD 1,100-USD 1,350 per tonne for 5.5% spodumene concentrate FOB Australia is reasonable given recent technical reports, consensus forecast, Benchmark Intelligence report, and the backdrop of the Q2-Q3 2025 market conditions.

Based on the spodumene price assessment presented in Chapter 19, the long-term price of USD 1,221/tonne (SC 5.5%, FOB Australia basis), equivalent to USD 1,332 for 6% Li_2O spodumene concentrate, is used in this FS (Table 1.7).

Table 1.7: Spodumene Concentrate Price Used

Product	Price (USD/t)
Spodumene concentrate at 5.5% Li_2O (FOB Australia)	1,221

In January 2025, the Company entered into a binding offtake term sheet with Volkswagen's 100%-owned vertically integrated battery manufacturer, PowerCo SE (PowerCo), to supply 100,000 tonnes of spodumene concentrate (SC5.5 target) per year over a 10-year term (PMET, 2024) (PMET, 2025).

As of the effective date of this Technical Report, no contracts have been executed with vendors to purchase equipment or contractors to construct the Project.

1.20 Environmental Studies, Permitting and Social or Community Impact

1.20.1 Introduction

PMET started collecting baseline environmental data on the Shaakichiuwaanaan Property in 2022 and has continued its inventory efforts to date with a full program designed to support an Environmental and Social Impact Assessment. The environmental data collected to date includes the following components:

- Noise.
- Surface water and sediments.
- Hydrology.
- Surficial deposits and hydrogeology.
- Geochemistry.
- Bathymetry.
- Terrestrial fauna, including herpetofauna, avian fauna, chiropterans small and large mammals.
- Vegetation and wetlands.
- Species at risk.
- Fish and fish habitat.

With a view to maintaining a strong and ongoing relationship with the Indigenous and non-Indigenous groups (front-line regional players operating in the political, land use, geographic, social, financial, environmental, and technical spheres) potentially affected by the Shaakichiuwaanaan Project, PMET wishes to set up adapted, concerted information and consultation processes, and establish mutual collaboration and partnership agreements. To this end, PMET has been working with the communities and stakeholders to develop a consultation, communication and mobilization plan that includes ongoing Project updates. This plan first aims to gather the concerns and interests of both Indigenous and non-Indigenous groups, particularly those relating to environmental issues, land use, employment, training opportunities, service provision, potential collaborations, etc. Through this approach, PMET seeks to understand the opinions and concerns of Indigenous groups and stakeholders, and to openly discuss and record its activities. The Company encourages open dialogue, both formally and informally, to give the involved communities the opportunity to express their opinions and concerns about the Project. The outcome of these discussions will enable the Project to address its concerns and interests and optimize its social acceptability.

The Project is subject to both the provincial environmental and social impact assessment (ESIA) and review procedure of the JBNQA, and the federal Impact Assessment examination procedure. Numerous permits and authorizations will also be required in Québec and Canada to build and operate the proposed mine. Concerning the provincial procedure, a Preliminary Information Statement was sent to MELCCPF in November 2023 to officially begin the process. On April 5, 2024, the MELCCFP confirmed that the Project was subject to the environmental and social impact assessment procedure and issued a directive for the completion of the impact study. On the federal side, an Initial Project Description was submitted to the Impact Assessment Agency of Canada (IAAC) in February 2025. Following a public consultation period and a series of questions, the Project was formally designated as being subject to the federal Impact Assessment process in May 2025. Tailored Impact Assessment Guidelines were published in August 2025.

1.20.2 Geochemistry

Static testing was conducted on a total of 354 samples, including 316 waste rock samples, 25 spodumene pegmatite (ore) samples, and 9 tailings samples representing different tailings processing streams. The waste rock samples were selected based on estimated lithological proportions, spatial representativeness, including variability along strike and at depth, and geochemical variability based on identified constituents of potential concern (COPCs), particularly arsenic, antimony, and lithium. All samples were tested for Acid-base Accounting (ABA), Synthetic Precipitation Leaching Procedure (SPLP) and CTEU-9 leaching tests were conducted on all samples, with some samples selected for additional testing with Toxicity Characteristic Leaching Procedure (TCLP).

The kinetic testing program was implemented to quantify the rates of metal leaching and acid generation to better assess the risk for metal leaching and acid under site-specific conditions. Kinetic testing included the continuation of 16 humidity cell tests (HCTs) on waste rock and ore, including one (1) duplicate for data QA/QC. The HCTs were initiated in 2023, and the results from the first 40 weeks of kinetic testing were presented by BBA / Vision Geochemistry (2024a). Updated reporting of the HCT data, up to 100 weeks, was presented by Vision Geochemistry (2025). In early 2025, six (6) column tests with tailings material, including one (1) duplicate, were initiated to assess their risk for COPCs. Results of the column testing up to 20 weeks were reported by Vision Geochemistry (2025). The tested tailings material included floats, middlings, magnetics, bypass and a master composite.

Geochemical modelling of Stockpile 001 and Stockpile 002 was executed to better understand the evolution and COPC peak concentration estimates for arsenic, antimony and lithium, during mine operations and into mine closure (Vision Geochemistry, 2025). The models were designed to predict the evolution of COPC concentrations in waste rock and tailings contact water at Stockpile 001 and Stockpile 002, to support the design of mitigation measures and treatment systems. Previously reported stockpile modelling

results presented by BBA / Vision Geochemistry (2024b) showed that co-deposition of all waste rock in the same stockpiles results in contact water with arsenic concentrations exceeding the MELCCFP resurgence in surface water criteria, and therefore segregation of ultramafic waste rocks was recommended. However, recent modelling results by Vision Geochemistry (2025) incorporated the updated FS design dimensions of the stockpiles and determined that a more conservative segregation protocol was required to prevent arsenic leaching. The revised segregation protocol included the complete separation of ultramafic lithology, paired with the separation of all material with arsenic concentrations above 30 ppm, to maintain arsenic and antimony concentrations below the CVAA and MDMER/D019 criteria in Stockpile 001. Conversely, leaching conditions were identified at Stockpile 002, with arsenic concentrations exceeding the MDMER/D019 criteria by up to two (2) orders of magnitude, primarily driven by the contribution of the tailings. Similarly, antimony and, to a lesser extent, lithium are expected to leach from Stockpile 002, largely contributed from the tailings.

Continued and additional geochemical testing beyond the FS is recommended to improve the understanding of the geochemical risk associated with mine waste materials.

1.20.3 Reclamation and Closure

The closure of all project infrastructure aims to ensure a responsible transition to post-mining land use, in compliance with Quebec regulations and in close collaboration with Indigenous communities. The site rehabilitation strategies will be implemented in three (3) phases: progressive restoration during operations, dismantling of facilities after operations, followed by decontamination and restoration of site infrastructure, and finally, post-closure maintenance and monitoring.

Industrial facilities will be dismantled, materials will be managed according to the principles of reduction, reuse, and recycling, and surfaces will be revegetated. Certain roads, such as the one connecting to the Trans-Taiga Road, will remain accessible to the public and Cree communities. The Matagami transshipment site will remain under the jurisdiction of the town. Discussions are ongoing to transfer ownership of facilities such as the Indigenous Cultural Centre and the power line to the community, with dismantling costs excluded from the project budget.

The closure plan for mining infrastructures includes specific strategies for each component: pit, waste rock piles, tailings, and overburden stockpiles. Waste rock stockpiles will be managed according to their potential for contaminant generation:

- Stockpile 001, composed of low-risk waste rock, will be reprofiled and restored with a vegetated cover suitable for low-risk waste rock. If the geochemical classification changes, a new cover design will be required.

- Stockpile 002, which will contain high-risk or acid-generating waste rock, will be capped with a 1.5-metre-thick impermeable clay layer to limit infiltration into underlying materials and prevent contaminant generation, followed by a 20-cm topsoil layer to support vegetation growth.
- Open pit will be allowed to flood naturally, forming a stable lake, secured with a berm, and connected to the local hydrographic network, with aquatic habitats created.
- Overburden and organic soil piles (001, 004, 005) will be used for cover construction and subsequently restored. Once the soils are utilized, the remaining footprints will be scarified and revegetated. These measures aim to ensure the environmental stability of the site, promote ecological reintegration of the land, and meet the expectations of stakeholders, particularly local and Indigenous communities.
- Access roads will be decommissioned, drainage structures removed, and surfaces recontoured and revegetated.
- Water management ponds will be backfilled once water quality is confirmed, following sediment characterization and appropriate disposal. Dikes of the stockpile water collection ponds will be breached to prevent water accumulation at the base of the structures.

1.20.4 Monitoring and Maintenance Programs

Upon completion of closure activities, monitoring programs will be implemented to assess geochemical and geotechnical stability, as well as the performance of cover systems, ensuring that restored areas do not pose risks of contamination or subsidence. An environmental monitoring program will confirm that restoration objectives have been met. According to Directive 019 (2025 edition) from MELCCFP, post-operation and post-closure environmental monitoring requirements include:

- Effluent Monitoring: Final effluents must be monitored weekly during the restoration phase, including water quality sampling, flow rate and pH measurements, with data submitted to MELCCFP.
- Groundwater and Runoff Management: Groundwater will be sampled twice annually during restoration and post-restoration phases. Runoff from stockpiles will be treated until water quality objectives are achieved.

Post-Restoration Monitoring: Once the site is restored, breaches will be created in the collection pond dikes, resulting in four (4) effluents to be monitored monthly for 20 years (during the open water season). Groundwater will continue to be monitored twice per year, and mine water and final effluents will be monitored eight times annually. The effectiveness of revegetation will also be assessed.

1.21 Capital and Operating Costs

1.21.1 Capital Cost

The capital costs were obtained from equipment proposals for major, high-value and long-lead items where specified and quoted specifically for the Project by the consultants responsible for the area. Equipment prices for minor items were sourced from recent Projects or from G Mining Services, or from other consultants' current database.

Capital costs are divided into three (3) categories: initial capital costs, expansion capital and sustaining capital costs during life of mine. Total costs for the Project are evaluated at $2,914.7M. Note that the amount excludes a pre-production revenue of $101.7M during the Phase 1 construction and the reclamation and closure cost of $248.4M.

The total of the initial cost for the Shaakichiuwaanaan Project is estimated to $1.497.7M, including contingencies and indirect costs and excluding the $101.7M of pre-production revenue. The initial capital cost is made up of Phase 1 (open-pit mine) costs and 23% of Phase 2 (underground mine).

The expansion capital includes the costs related to the construction and development of the underground mine, the camp expansion, the second train of the processing plant and their related indirect costs and contingencies. The expansion capital will occur during the first 2 years of production from June 2030 to March 2032 (Years 1 and 2). All other capital expenses after Year 2 are included in the sustaining capital. The expansion capital cost was estimated to be $480.5M.

Sustaining capital costs include all expenditures necessary to sustain operations throughout the LOM. Sustaining costs start at Year 1 until the end of the mining operations and were estimated to amount to $936.4M over the LOM.

Table 1.8 provides an overview of the capital costs (pre-production, expansion and sustaining) on a cumulative basis for the life of the Project.

Note that Table 1.8 does not consider the amount of $248.4M of the closure cost.

Table 1.8: Project Total Capital Cost Summary

Capital Expenditure	Phase 1 OP Initial Capital Cost ($M)	Phase 2 UG Initial Capital Cost ($M)	Initial Capital Cost ($M)	Phase 2 UG Expansion Capital Cost ($M)	Total Devel. Capital Cost ($M)	LOM Sustaining Capital Cost ($M)	Total Capital Cost ($M)
100 - Infrastructure	124.9	-	**124.9**	24.8	**149.7**	30.8	**180.5**
200 - Power and Electrical	173.8	-	**173.8**	46.2	**220.0**	25.0	**245.1**
300 - Water Management	128.2	-	**128.2**	18.7	**146.9**	100.5	**247.4**
400 - Surface Operations	18.6	-	**18.6**	-	**18.6**	11.9	**30.5**
500 - Mining	120.0	99.1	**219.1**	36.4	**255.5**	550.5	**806.0**
600 - Process Plant	217.3	20.1	**237.4**	167.0	**404.4**	-	**404.4**
700 - Construction Indirect	262.8	0.1	**262.9**	123.8	**386.7**	-	**386.7**
800 - General Services / Owner's Cost	99.8	4.7	**104.5**	13.4	**117.9**	31.6	**149.6**
900 - Pre-production, Start-up, Comm.	73.3	9.3	**82.6**	1.5	**84.1**	186.1	**270.2**
Total Capital Expenditures Excluding Contingency	**1,218.7**	**133.3**	**1,352.0**	**431.8**	**1,783.8**	**936.4**	**2,720.2**
990 - Contingency	130.7	15.0	**145.7**	48.7	**194.4**	-	**194.4**
Total Capital Expenditures	**1,349.4**	**148.3**	**1,497.7**	**480.5**	**1,978.2**	**936.4**	**2,914.6**
Less: Pre-Prod. Credit Net of TC / RC & Royalties	(101.7)	-	**(101.7)**	-	**101.7**	-	**(101.7)**
Total Capex Net of Pre-Production Crédit	**1,247.7**	**148.3**	**1,396.0**	**480.5**	**1,876.5**	**936.4**	**2,813.0**

The overall capital cost estimate developed in this Feasibility Study generally meets the AACE Class 3 requirements and has an accuracy range of between -10% and +20%. The capital cost estimate for this study forms the basis for the approval of further development of the Project.

1.21.2 Operating Cost

The operating cost estimate (Opex) is based on first principles calculation. No cost escalation or contingency has been included within the operating cost estimate.

The total LOM operating cost for the Shaakichiuwaanaan mine site was estimated at $6,785M, as detailed in Table 1.9.

Table 1.9: Total LOM Operating Cost at Site

Parameters	Unit Cost ($/t conc)	Operating Cost ($M)
Open Pit Mining	104.9	1,390
Underground Mining	215.2	2,852
Processing	91.2	1,208
G&A	100.7	1,335
Total	**511.9**	**6,785**

The concentrate transportation costs from site to Grande-Anse are estimated at $217.2/t.

Table 1.10 summarizes the unit operating costs per tonne of concentrate.

Table 1.10: Unit Operating Cost

Operating Costs	Unit Cost ($/t conc)
Mining Cost	320.1
Processing Cost	91.2
Site Administration	100.7
Cash Operating Cost at Site(1)	**511.9**
Concentrate Transport	217.2
Total Cash Operating Cost (FOB Grande-Anse)(2)	**729.1**
Sustaining Capital	70.65
All-in Sustaining Cost(3)	**799.8**

**Note:*
(1): Cash operating cost at site includes mining, processing and site administration expenses calculated on an SC5.5 basis. This is a non-GAAP financial measure, and when expressed per tonne, non-GAAP ratios. Refer to the "Important Notice" for further information on these measures.
(2): Total cash operating cost (FOB Grande-Anse) includes mining, processing, site administration, and product transportation to Grande-Anse calculated on an SC5.5 basis. This is a non-GAAP financial measure, and when expressed per tonne, non-GAAP ratios. Refer to the "Important Notice" for further information on these measures.
(3): All-in sustaining costs (AISC) includes mining, processing, site administration, and product transportation costs to Grande-Anse and sustaining capital over the LOM per unit of concentrate produced during the LOM and excludes Royalties. This is a non-GAAP measure, and when expressed per tonne, a non-GAAP ratio. Refer to the "Important Notice" for further information on these measures.

1.22 Economic Analysis

The pre-tax base case financial model results in an internal rate of return (IRR) of 19.9% and a net present value (NPV) of $2,514M with a discount rate of 8%. The pre-tax payback period is 4.9 years. On an after-tax basis, the base case financial model results in an internal rate of return of 18.1% and an NPV of $1,594M with a discount rate of 8%. The after-tax payback period is 4.7 years. Table 1.11 shows the financial analysis summary.

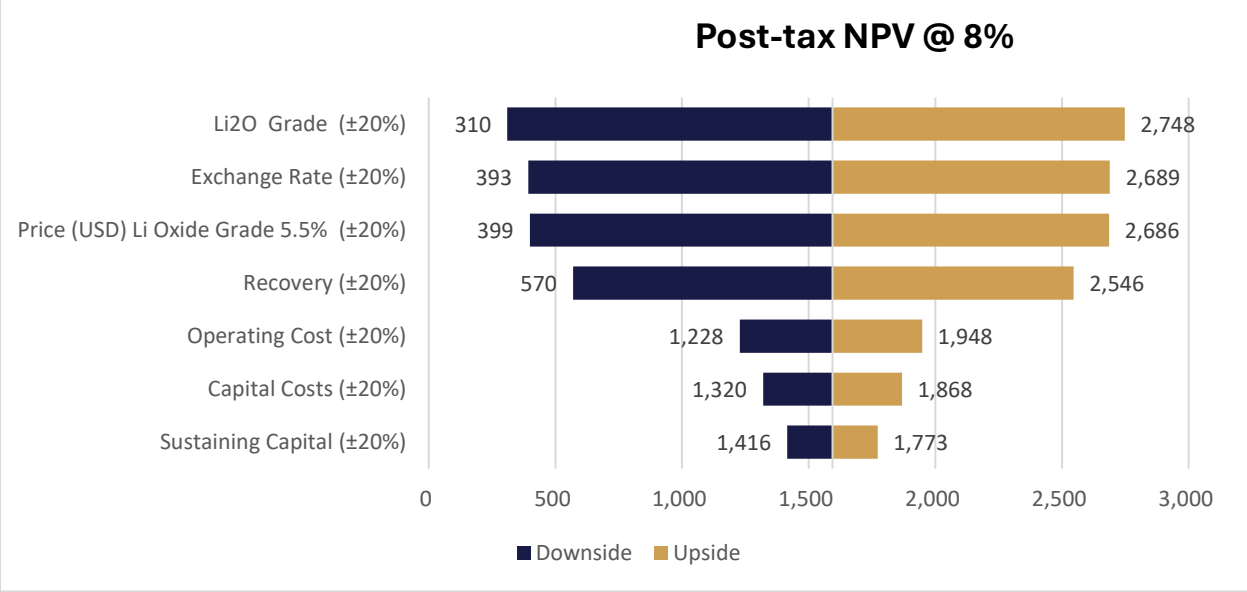
Table 1.11: Financial Analysis Summary

Description		CA$ M	US$ M
Pre-Tax	**Discount Rate**		
	0%	8,358	6,237
	5%	3,995	2,981
	8%	**2,514**	**1,876**
	10%	1,805	1,347
	15%	640	478
	Pre-Tax IRR	19.9%	
	Payback Period	4.9 years	

Description		CA$ M	US$ M
After-Tax	**Discount Rate**		
	0%	5,418	4,043
	5%	2,581	1,926
	8%	**1,594**	**1,190**
	10%	1,115	832
	15%	312	233
	After-Tax IRR	18.1%	
	Payback Period	4.7 years	

The Project is most sensitive to the exchange rate, grade, spodumene concentrate price and process recovery, as illustrated in the tornado graph in Figure 1.8, while it is much less affected by operating costs and capital expenditures.

Figure 1.8: Tornado Graph for the Sensitivity Analysis (+/-20%)



1.23 Adjacent Properties

The Company holds the dominant land position with respect to greenstone belt in the region; however, the Property is fully surrounded by other properties held over multiple mineral exploration companies targeting various mineral commodities.

1.24 Organization and Project Schedule

1.24.1 Organization

The Shaakichiuwaanaan Project will be executed by an Integrated Project Management Team (IPMT) using a self-perform approach, supported by contractors for specialized tasks and peak needs. The IPMT will manage infrastructure, plant construction, and mine development, applying rigorous QA/QC systems. Operations staff will join during construction for a structured handover. Owner-led services and mining teams will ensure continuity into operations, targeting 20% local workforce. Full operational control transfers after achieving 65% nameplate production.

1.24.2 Project Schedule

The Shaakichiuwaanaan Lithium Project schedule is structured across management and project levels, with a Level 1 high-level schedule defining milestones.

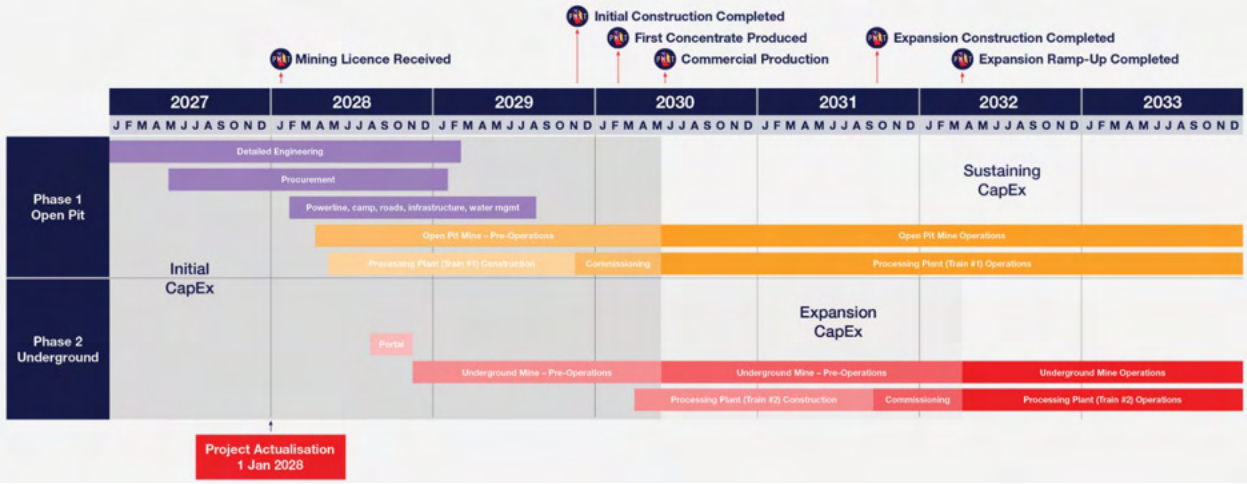
Phase 1 (Jan 2027–May 2030) covers open-pit mine development, permitting, procurement, and site infrastructure, with major construction in 2029 and commercial production by May 2030.

Phase 2, beginning Q3 2028, focuses on underground mine development and process plant expansion through March 2032.

The schedule includes procurement, equipment mobilization, utilities, and permanent facilities construction. Commissioning begins Q3 2029, ramping up through May 2030 to validate performance and ensure readiness for full operations.

Figure 1.9 focuses on the construction period and timeline per phase 1 (open pit) and phase 2 (underground), and their relation to initial, expansion and sustaining costs; also defining the milestone dates of the project.

Figure 1.9: Timeline for Construction Phases 1 and 2



1.25 Interpretations and Conclusions

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) and Form 43-101F1. The objective of this Feasibility Study Report is the evaluation of the technical feasibility and economic viability of the Project, notably the CV5 Spodumene Pegmatite's development based on a hybrid open pit and underground mining scenario and a process plant with a design capacity of 5.1 Mtpa and related infrastructure. This NI 43-101 Technical Report confirms the technical feasibility and economic viability based on an open pit mining and underground operation with a nominal throughput of 801,600-t of 5.5% Li_2O production per year over a 15-year life-of-mine (LOM) plus a total of 4-year at 400,000-t of 5.5% Li_2O production. It is recommended to advance the Project to the Detailed Engineering and Construction and Development phase.

1.25.1 Project Risks and Opportunities

The most significant risks identified with the Project are delays in permit approval due to environmental considerations, labour availability and employment and remote site location for spodumene and workforce transportation.

The above-noted risks can be attenuated with adequate engineering, planning, and operation mitigation procedures in place. Risks which are beyond the control of the Project proponents are much more difficult to anticipate and mitigate.

Opportunities remain to improve the economics, timing and/or permitting potential of the Project. The main opportunities noted are:

- Bulk sampling for better knowledge of the reserve and the geomechanical ground characterization.
- Continue evaluation of other commodities, i.e. Tantalum and Caesium.
- Improve logistics cost of concentrate transportation, road, train and port.

1.26 Recommendations

Based on the results of the technical and financial analyses of this Feasibility Study (FS), which demonstrates positive project economics, GMS recommends proceeding with the basic engineering for the Project and initiate limited notice to proceed (LNTP) with suppliers for critical items and equipment of long delivery during 2027 prior to the full construction release once project financing is finalized and the main permits are issued.

Table 1.12 summarizes the proposed budget of $83.1M to advance the project as budgeted in the feasibility in 2027 and to progress into the basic engineering stage and perform the recommendations discussed in this section to achieve commercial production by the end of May 2030. An additional $17.8M is required to the project budget to pursue geology, exploration and geochemical work.

Table 1.12: Cost Estimate Associated with Recommendations

Description	Amount ($M)
Down payment for camp, water management and other infrastructure.	18.6
LNTP for long-lead items Progress on power connection to HQ, for transmission line and main transformer procurement	33.6
Down payment on equipment and contractor	3.8
Progress basic engineering and initiate detailed engineering	16.4
Owners Cost	2.6
Contingency 11%	8.1
Total Project Costs for 2027	**83.1**
Geology and Exploration(1)	15.0
Geochemistry Testing(1)	0.5
Metallurgical Testwork(1)	0.3
Exploration Camp Improvement[1]	2.0
Total Anticipated Costs	**100.9**

Note:
(1): This cost is excluded from the Project Feasibility Study Capital Estimate.

2. INTRODUCTION

PMET Resources Inc. (PMET" or the "Company) retained G Mining Services Inc. (GMS) as lead consultant, with contributions from Primero Group Americas Inc. (Primero), AtkinsRéalis, BBA Inc., Paterson & Cooke, Alius Mine Consulting, WSP, Mailloux Hydrogéologie, Vision Geochemistry, and GCM Expert, to prepare an independent Feasibility Study (FS) and Technical Report for the CV5 Pegmatite at the Shaakichiuwaanaan Property (the Property). The Property, formerly referred to as the "Corvette Project", is located in the Eeyou Istchee James Bay region of Québec, Canada.

PMET Resources (formerly Patriot Battery Metals) is a leading hard-rock lithium exploration company advancing its 100%-owned Shaakichiuwaanaan Property in Quebec's Eeyou Istchee James Bay region. The Company's common shares trade on the Toronto Stock Exchange (TSX:PMET), Australian Stock Exchange (ASX:PMT), OTCQX market (OTCQBX: PMETF) and Frankfurt market (FSE: R9GA).

The head office of PMET Resources Inc. is located at:

1801 McGill College, Suite 900,
Montréal, Québec,
H3A 1Z4, Canada

On October 5, 2023, PMET Resources Inc. incorporated a wholly owned Québec-based subsidiary, Lithium Innova Inc. (Innova), which is the sole registered holder of the Exclusive Exploration Rights comprising the Shaakichiuwaanaan Property. This Technical Report is submitted by PMET Resources in its capacity as the owner of Innova.

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) and Form 43-101F1. The objective of this report and the FS is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit and underground mine, including processing facilities and related infrastructure. This Report provides operating and capital costs estimations and an economic analysis of the Project.

The MRE was prepared following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and in accordance with CIM Guidelines (2019) for Estimation of Mineral Resources and Reserves. The effective date of the mineral resource estimation is June 20, 2025. It is based, for the most part, on a drilling database and supporting geological information.

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014). As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste at a grade of 0% Li_2O.

The intention of this Technical Report is to provide sufficient, clear, and unambiguous scientific and technical information relating to the Project as of the Effective Date. The Qualified Persons (QPs) understand that a copy of this Report will be filed with the Canadian Securities Commissions and made publicly available.

As of the Effective Date of this Report, the QPs are not aware of any litigation potentially affecting the Project. The QPs did not verify the legality or terms of underlying agreements related to ownership, permits, Binding Offtake Term Sheet, licences, royalties, or other contracts between PMET Resources Inc. and third parties.

The qualified persons (QP) of this Technical Report are the following:

- Mr. Carl Michaud, P.Eng., MBA, Vice President of Mining Engineering at G Mining Services.
- Mr. Pascal Droz, P.Eng., Electrical Engineering Director / Project Manager at G Mining Services.
- Mr. Todd McCracken, P.Geo., Director Mining & Geology at BBA Inc.
- Mr. Ryan Cunningham, P.Eng., M. Eng., Process Engineer at Primero.
- Mr. Sebastien Guido, P.Eng., M. Sc., Senior Engineer, Rock Mechanics at Alius Mine Consulting.
- Mr. Hugo Latulippe, P.Eng., Principal Mining Engineer at BBA Inc.
- Mr. Ryan Smilovici, P.Eng., Process Engineer at Paterson & Cooke.
- Mr. Philip Addis, P.Eng., Principal Tailings Engineer at AtkinsRéalis.
- Mr. Antoine Cogulet, P.Eng., Team Lead, Mine Water Management at AtkinsRéalis.
- Mr. Holman Tellez, P.Eng., Sr. Expert Hydraulics / Hydrology at AtkinsRéalis.
- Ms. Nathalie Fortin, Eng., M. Env., Vice-President, Environmental Management Earth Sciences and Environment at WSP.
- Ms. Genevieve Marchand, P.Eng., M.Sc., PMP, Project Director and Mining Hydrogeology Specialist at AtkinsRéalis.

- Mr. Neal Sullivan, Ph.D., P.Geo., President & Principal Geochemist at Vision Geochemistry.
- Ms. Sandra Pouliot, P.Eng., M.Sc. A, PMP, Senior Mining Environment Engineer and Sustainable Mining at AtkinsRéalis.
- Mr. Michel Mailloux, P.Eng., P.Geo., M.Sc., Sr. Hydrogeologist and Owner at Mailloux Hydrogéologie.
- Mr. Charles Gagnon, P.Eng., M.Sc., Sr. Ventilation Engineer and Owner at CGM Expert.

2.1 QP Responsibility

The QPs are entirely independent of the issuer (PMET Resources) as described in Section 1.5 of the NI 43-101 standard of disclosure for mineral projects. The QPs involved in the mandate do not hold an interest in the issuer or its related entities.

The following QPs have contributed to the writing of this Report and have provided QP certificates. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed with figures, tables and portions of Chapters 1 (Summary), 2 (Introduction), 21 (Capital and Operating Costs), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References).

The QPs responsible for each section of the Technical Report are mentioned in Table 2.1

Table 2.1: Summary of Qualified Persons

Qualified Person	Company	Title	Report Sections
Carl Michaud, P.Eng., MBA	G Mining Services Inc	Vice President of Mining Engineering	1.1, 1.2, 1.31.14, 1.15, 1.20, 1.21, 1.26, 2, 3.2, 15, 16 (except 16.2,16.3, 16.5.6, 16.5.7.4), 21.3 (except 21.3.3), 22, 24.3, 25.1, 25.3, 25.4, 25.10, 25.11, 25.14, 26.2, 26.6
Pascal Droz, P.Eng.	G Mining Services Inc	Electrical Engineering Director / Project Manager	1.17, 1.24, 3, 18.1, 18.2, 18.3, 18.4, 18.5.1, 18.8, 18.9, 21.1, 24.1, 24.2, 25.9, 25.13, 26.4.1, 26.4.8, 26.4.9, 26.4.10, 26.7

Qualified Person	Company	Title	Report Sections
Todd McCracken, P.Geo.	BBA Inc.	Director Mining & Geology	1.4 to 1.11, 1.13, 1.22, 1.23, 3.1, 3.3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24.4, 25.2, 26.1
Ryan Cunningham, P.Eng., M.Eng.	Primero Group Americas Inc.	Process Engineer	1.12, 1.16, 13, 17, 18.5 (except 18.5.1, 18.5.6), 21.3.3, 25.5, 26.3
Sebastien Guido, P.Eng., M.Sc.	Alius Mine Consulting Inc.	Senior Project Engineer, Rock Mechanics	16.2
Hugo Latulippe, P.Eng.	BBA Inc.	Principal Mining Engineer	1.18, 19, 25.12,
Ryan Smilovici, P.Eng.	Paterson & Cooke	Process Engineer	16.5.7.4, 18.5.6
Philip Addis, P.Eng.	AtkinsRéalis	Principal Tailings Engineer	18.6, 18.7.2.1.4 to 18.7.2.1.6, 20.4, 25.7.1
Antoine Cogulet, P.Eng.	AtkinsRéalis	Team Lead, Mine Water Management	18.7 (excl. hydrology and hydraulics; and including: pumping, piping, treatment plants and side wide water balance), 20.7.2, 26.4.3, 26.4.4
Holman Tellez, P.Eng.	AtkinsRéalis	Sr. Expert Hydraulics / Hydrology	18.7 (excl. pipe and pump system design and treatment plants; and incl. site water balance), 20.5, 26.4.3
Nathalie Fortin, Eng., M. Env.	WSP	1. Vice-President Environmental Management Earth Sciences and Environment	1.19, 20.1 (except 20.1.1.7), 20.2, 20.3, 20.9, 20.10, 20.11, 20.12, 25.8, 26.5
Genevieve Marchand, P.Eng., M.Sc., PMP	AtkinsRéalis	Project Director and Mining Hydrogeology Specialist	20.1.1.7, 20.6, 25.6, 26.4.2
Neal Sullivan, Ph.D., P.Geo.	Vision Geochemistry	President & Principal Geochemist	20.7, 26.4.5
Sandra Pouliot, P.Eng., M.Sc. A, PMP	AtkinsRéalis.	Senior Mining Environment Engineer and Sustainable Mining	20.8, 21.2, 25.7.4, 26.4.6,
Michel Mailloux, P.Eng., P.Geo., M.Sc.	Mailloux Hydrogéologie	Sr. Hydrogeologist and Owner	16.3
Charles Gagnon, P.Eng., M.Sc.	CGM Expert	Sr. Ventilation Engineer and Owner	16.5.6

2.2 Effective Date

The FS is derived using the Company's mineral resources estimate effective as June 20, 2025 (the MRE).

The Effective Date of the Technical Report is October 20, 2025.

The Issue Date of the Technical Report is November 14, 2025.

2.3 Previous Technical Reports

NI 43-101 Technical Report on the Corvette Property, Quebec, Canada, Awk Geological Consulting Ltd., Calgary, Alberta, June 27, 2022 - effective date April 21, 2022.

NI 43-101 Technical Report, Mineral Resource Estimate for the CV5 pegmatite, Corvette Property, BBA Engineering Ltd., September 8, 2023 - effective date July 30, 2023.

NI 43-101 Technical Report, Preliminary Economical Assessment for the Shaakichiuwaanaan Project, James Bay Region, Quebec, Canada, BBA Engineering Ltd., September 12, 2024 - effective date August 21, 2024.

NI 43-101 Technical Report, Mineral Resource Estimate for the Shaakichiuwaanaan Project, BBA Engineering Ltd., August 29, 2025 - effective date June 20, 2025.

2.4 Use of Non-GAAP Financial Measures

This report includes non-IFRS financial measures and non-IFRS financial ratios. The Company believes that these measures provide additional insight, but these measures are not standardized financial measures prescribed under IFRS and therefore should not be confused with, or used as an alternative for, performance measures calculated according to IFRS. Furthermore, these measures should not be compared with similarly titled measures provided or used by other issuers.

The non-IFRS financial measures and non-IFRS financial ratios used in this news release and common to the mining industry are defined below: Cash operating costs at site and cash operating costs at site per tonne: Cash operating costs at site is a non-IFRS financial measure which includes mining, processing, and site administration. Cash operating costs at site per tonne is a non-IFRS financial ratio which is calculated as cash operating costs at site divided by anticipated production expressed in tonnes. These measures capture the important components of the Company's anticipated production and related costs

and are used to indicate anticipated cost performance of the Company's operations. Total cash operating costs (DAP Grande-Anse as POL) and total cash operating costs per tonne (DAP Grande-Anse as POL): Total cash operating costs (DAP Grande-Anse as POL) is a non-IFRS financial measure which includes mining, processing, site administration, and product transportation to Grande-Anse. Total cash operating costs (DAP Grande-Anse as POL) per tonne is a non-IFRS financial ratio which is calculated as total cash operating costs (DAP Grande-Anse as POL) divided by anticipated production expressed in tonnes. These measures capture the important components of the Company's anticipated production and related costs and are used to indicate anticipated cost performance of the Company's operations. All-in sustaining cost (AISC) and AISC per tonne: All-in sustaining cost is a non-IFRS financial measure which includes mining, processing, site administration, and product transportation to Grande-Anse and sustaining capital but excludes royalties. All-in sustaining cost per tonne of spodumene concentrate is a non-IFRS financial ratio which is calculated as all-in sustaining cost divided by anticipated production expressed in tonnes. These measures capture the important components of the Company's anticipated production and related costs and are used to indicate anticipated cost performance of the Company's operations.

2.5 Sources of Information

This Report is based in part on internal company reports, maps, published government reports, company letters and memoranda, and public information, as listed in Chapter 27 (References). Sections from reports authored by other consultants may have been directly quoted or summarized in this Report and are so indicated, where appropriate.

The QPs have no known reason to believe that any of the information used to prepare this Report and evaluate the Mineral Resources presented herein is invalid or contains misrepresentations. The QPs have sourced the information for this Report from the collection of documents listed in Chapter 27 (References).

2.6 Agreements, Mineral Tenure, Surface Rights and Royalties

The issuer provided details regarding mining titles, royalty agreements, environmental liabilities, mineral agreement and permits. The QPs are not qualified to offer any legal opinion on property titles, ownership, or potential litigation.

2.7 Site Visit

Qualified Person	Site Visit Scope	Dates
Carl Michaud, P.Eng., MBA	Mining, surface installation and topography	June 10 to 12, 2025
Pascal Droz P.Eng.	Surface infrastructure, roads and topography	June 10 to 12, 2025
Todd McCracken, P.Geo.	Geology, core review	April 7 to 11, 2023, June 4 to 7, 2024
Ryan Cunningham, P.Eng., M.Eng.	Site visit not required for process at this stage	
Sebastien Guido, P.Eng., M.Sc.	Core logging	Oct. 22 to 25, 2024
Hugo Latulippe, P.Eng.	Site visit not required for market study at this stage	
Ryan Smilovici, P.Eng.	Site visit not required for paste backfill at this stage	
Philip Addis, P.Eng.	Surface installation, lakes and topography	Oct. 16 to 17, 2024
Antoine Cogulet, P.Eng.	Water management and topography	June 10 to 12, 2025
Holman Tellez, P.Eng.	Water management and topography	Oct. 16 to 17, 2024
Nathalie Fortin, Eng., M. Env.	Site visit not required for environment at this stage	
Genevieve Marchand, P.Eng., M.Sc., PMP	Water management and hydrology	June 10 to 12, 2025
Neal Sullivan, Ph.D., P.Geo.	Site visit not required for geochemistry at this stage	
Sandra Pouliot, P.Eng., M.Sc. A., PMP	Site visit not required for mine closure at this stage	
Michel Mailloux, P.Eng., P.Geo., M.Sc.	Site visit not required for mine dewatering at this stage	
Charles Gagnon, P.Eng., M.Sc.	Site visit not required for underground ventilation at this stage	

2.8 List of Abbreviations and Units of Measurement

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system. Unless otherwise indicated, all references to “$”, “CA$” or “CAD” in this Report are to Canadian dollars and references to “US$” or “USD” are to US dollars. A foreign exchange conversion rate of US dollar of USD 1 / CAD 1.34 has been used over the LOM. A list of the main abbreviations and terms used throughout this Technical Report is presented in Table 2.2. Unless otherwise specified, the source for tables and figures is GMS, 2025.

Table 2.2: Table of Abbreviations

Abbreviation	Description
$, CA$ or CAD	Canadian Dollar (examples of use: CA$2.5M / $2.5M)
$/t	Dollars per tonne
%	Percent
°C	Degrees Celsius
3D	Three Dimensional
a	Annum (year)
AARQ	Atlas des amphibiens et reptiles du Québec
AAS	Atomic Absorption Spectroscopy
Actlabs	Activation Laboratories Ltd.
Ag	Silver
AI	Artificial Intelligence
ALS Canada	ALS Canada Ltd.
ANFO	Ammonium Nitrate Fuel Oil (explosive)
AR	Augmented Reality
ARD	Acid Rock Drainage
As	Arsenic
ATV	Acoustic Televiewer
Au	Gold
BEV	Battery Electric Vehicle
BWi	Bond Ball Work Index
Calc.	Calculation
Capex	Capital Cost Estimate
CBHSSJB	Cree Board of Health and Social Services of James Bay
CDA	Guidelines of the Canadian Dam Association
CDC	Map Designated Claim (from the French “claim désigné sur carte”)
CDPNQ	Centre de données sur le patrimoine naturel du Québec
CFM	Cubic Feet per Minute
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	Centimetre
CN	Canadian National Railway

Abbreviation	Description
CNG	Cree Nation Government
CNSC	Canadian Nuclear Safety Commission
Co	Cobalt
COG	Cut-off Grade
Company	Lithium Innova Inc., a subsidiary of PMET Resources Inc..
CP	Competent Person
Cr	Chromium
CRM	Certified Reference Material
CRRNTBJ	Commission Régionale sur les Ressources Naturelles et le Territoire de la Baies-James
Cs	Caesium
CS	Cross-section
CTA	Cree Trappers' Association
Cu	Copper
CWi	Crushing Work Index
d	Day (24 hours)
DDH	Diamond Drill Hole
DEF	Diesel Exhaust Fluid
deg. or °	Angular Degree
DSO	Deswik Stope Optimizer
DFO	Fisheries and Oceans Canada
DGPS	Differential Global Positioning Systems
DMS	Dense Media Separation
ECCC	Environment and Climate Change Canada
EIJBRG	Eeyou Istchee James Bay Regional Government
EPCM	Engineering, Procurement and Construction Management
EPMA	Electron Probe Micro-Analysis
EQA	Environment Quality Act
ESIA	Environmental and Social Impact Assessment
et al.	and others
EV	Electric Vehicles

Abbreviation	Description
FCI	Félicie – Corvette Ouest – Island Lake – Properties
FEL	Front-End Loader
FeSi	Ferrosilicon
FIFO	Fly-In Fly-Out
FOB	Free On Board
g	Gram
G&A	General and Administration
GAAP	Generally Accepted Accounting Principles
GDP	Gross Domestic Product
GESTIM	Québec Mineral Tenure System
GOH	Gross Operating Hours
GSC	Geological Survey of Canada
H	Height
h	Hour (60 minutes)
ha	Hectare
HDPE	High Density Polyethylene
HLS	Heavy Liquid Separation
HQ	Hydro-Québec
IAA	Impact Assessment Act
IAAC	Impact Assessment Agency of Canada
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ID	Identification
ID^2	Inverse Distance Square
IFRS	International Financial Reporting Standards
in. or ″	Inch
INAA	Instrumental Neutron Activation Analysis
IP-Resistivity	Induced Polarization and Resistivity
IRA	Inflation Reduction Act

Abbreviation	Description
IRR	Internal Rate of Return
ISAQ	Inventaire des sites archéologiques du Québec
ISQ	Institut de la statistique du Québec
ISRM	International Society for Rock Mechanics
JBNQA	James Bay and Northern Québec Agreement
K	Potassium
K	Thousand ($)
kg	Kilogram
km	Kilometre
km^2	Kilometre Square
kN	Kilonewton
kPa	Kilopascal
kt	Kilotonne
ktpa	Kilotonne per Annum (year)
kV	Kilovolt
kWh	Kilowatt per Hour
L	Litre
LA	Laser Ablation
LA by ICP-MS	Laser Ablation by Inductively Coupled Plasma Mass Spectrometry
LCE	Lithium Carbonate Equivalent
LCT	Li-Cs-Ta (Lithium-Caesium-Tantalum)
LEMVQ	Liste des espèces désignées menacées ou vulnérables au Québec
LG	Lerchs-Grossmann
LG-2	La Grande Rivière Airport
LG-4	La Grande-4
LGA	La Grande Alliance
LHD	Load Haul Dump (loaders)
Li	Lithium
Li_2CO_3	Lithium Carbonate
Li_2O	Lithium Oxide
LiDAR	Light Detection and Ranging

Abbreviation	Description
LLDPE	Linear Low-Density Polyethylene
LOI	Letter of Intent
LOM	Life of Mine
m	Metre
m^2	Square Metre
m^3	Cubic Metre
Ma	Mega Annum (million annum)
MAC	Mining Association of Canada
Max.	Maximum
MCC	Ministère de la Culture et des Communications du Québec
MDDEP	Ministère du Développement durable, de l'Environnement et des Parcs
MDMER	Metal and Diamond Mines Effluent Regulation
MELCCFP	Ministère de l'Environnement, de la Lutte contre les changements climatiques de la Faune et des Parcs (previously MDDEP)
MERN	Ministère de l'Énergie et des Ressources naturelles
mesh	US Mesh
MFFP	Ministère des Forêts, de la Faune et des Parcs
Mg	Magnesium
mg	Milligram
Min.	Minimum
ML	Metal Leaching
mm	Millimetre
MM	Mineralized Material
Mo	Molybdenum
MOU	Memorandum of Understanding
MRE	Mineral Resource Estimate
MRNF	Ministère des Ressources Naturelles et des Forêts
Mt	Million Tonnes
MTC	Matagami Transshipment Centre
MTOs	Material Take Offs
Mtpa	Million Tonne per Annum (year)

Abbreviation	Description
MW	Megawatt
Na	Sodium
Ni	Nickel
NI 43-101	Canadian National Instrument
NN	Nearest Neighbour
No.	Number
NOH	Net Operating Hours
NOWC	Net Operating Work Capital
NPV	Net Present Value
NQ	Normal Quality
NRCan	Natural Resources Canada
NSR	Net Smelter Royalty
NTS	National Topographic System
NTS	Not To Scale
OK	Ordinary Kriging
OP	Open Pit
Opex	Operating Cost Estimate
OTV	Optical Televiewer
OVB	Overburden
oz	Troy Ounce
Pb	Lead
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PGEs	Platinum-Group Elements
PMF	Probable Maximum Flood
PMP	Probable Maximum Precipitation
ppm	Parts per Million
Primero	Primero Group Americas Inc.
PSD	Particle Size Distribution
Q'	Rock Tunneling Quality Index
Q1	First Quarter

Abbreviation	Description
Q2	Second Quarter
Q3	Third Quarter
Q4	Fourth Quarter
QA/QC	Quality Assurance / Quality Control
Q-Method	Rock Mass Quality
QP	Qualified Person
Rb	Rubidium
RBQ	Régie du bâtiment du Québec
RCM	Regional Country Municipality
RF	Revenue Factor
ROM	Run-Of Mine
RP	Retention Pond
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
SARA	Species at Risk Act
SC	Spodumene Concentrate
SEDAR+	System for Electronic Document Analysis and Retrieval
SG	Specific Gravity
SGS	SGS Canada Inc.
SO_4	Sulphate
SPD	Deswik Strategic Pit Design tool
SPLP	Synthetic Precipitation Leaching Procedure
SQ	Sûreté du Québec
st	Short Ton (2,000 lb) (ton)
Std Dev.	Standard Deviation
t	Tonne (1,000 kg) (metric tonne)
Ta	Tantalum
Ta_2O_5	Tantalum Oxide
TCLP	Toxicity Characteristic Leaching Procedure
TCR	TCR
TMF	Tailings Management Facility

Abbreviation	Description
tpa	Tonnes per Annum (year)
tpd	Tonnes per Day
tph	Tonnes per Hour
tpm	Tonnes per Metre (1,000 kg) (metric ton)
TSS	Total Suspended Solids
TY	Transfer Yard
UG	Underground
UGAF	Fur-Bearing Animal Management Units (unités de gestion des animaux à fourrure)
US$ or USD	United States Dollar (examples of use: US$2.5M)
USMCA	USA-Mexico-Canada Trade Agreement
UTM	Universal Transverse Mercator
Var Comp	Variability Composite
Virginia	Virginia Gold Mines
W	Width
WBS	Work Breakdown Structure
WEC	Work Element Coding
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
y	Year
Zn	Zinc

3. RELIANCE ON OTHER EXPERTS

3.1 Introduction

The Qualified Persons (QPs) have relied on information provided by independent experts with respect to mineral rights, surface rights, property agreements, royalties, taxation, commodity pricing, and the spodumene concentrate off-take agreement terms. The QPs consider this reliance reasonable, as the experts are assumed to hold the necessary education, professional designations, and relevant experience on matters addressed in this Technical Report.

The conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of preparation of this Report.
- Assumptions, conditions, and qualifications set forth in this Report.
- Data, reports, and opinions supplied by PMET and other third-party sources.

The QPs believe that the underlying assumptions in the information provided are factual and accurate and that the resulting interpretations are reasonable. To the extent applicable, the QPs have relied on such data and have no reason to believe that any material facts have been withheld. In their professional judgement, the QPs have taken appropriate steps to ensure that the information relied upon is sound and, accordingly, do not disclaim responsibility for the content of this Report.

The QPs have assumed that all information and technical documents referenced in Chapter 27 (References) are accurate and complete in all material respects. While the QPs have reviewed the available documentation, its accuracy and completeness cannot be guaranteed. The QPs reserve the right, but are not obligated, to revise this Report and its conclusions should additional information become available after the effective date.

3.2 Taxation

Carl Michaud, P.Eng., QP, has fully relied on, and disclaims responsibility for, information supplied by PMET staff and experts retained by PMET for information related to taxation as applied to the financial model. This information is used in support of the financial analysis in Chapter 22 (Economic Analysis).

3.3 Mineral Tenure and Surface Rights

Todd McCracken, P.Geo., relied on Darren L. Smith, P.Geo., Executive and Vice President Exploration for PMET, for matters pertaining to mineral concessions, surface rights, and mining leases as disclosed in Chapter 4. For the purpose of this Report, specifically Section 4.2 (Mineral Disposition), the QP has relied on registered title information available on the *Ministère des Ressources naturelles et des Forêts* (MRNF, Ministère des Ressources naturelles et des Fôrets, 2023). This information was last accessed on August 4, 2025. While the title documents were reviewed for this Report, this Report does not constitute, nor is it intended to represent a legal, or any other opinion as to title.

The authors wish to emphasize that they are QPs only in respect of the areas in this Technical Report identified in their “Certificates of Qualified Persons” submitted with this Technical Report to the Canadian Securities Administrators. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this Technical Report by any third party is at the party’s sole risk.

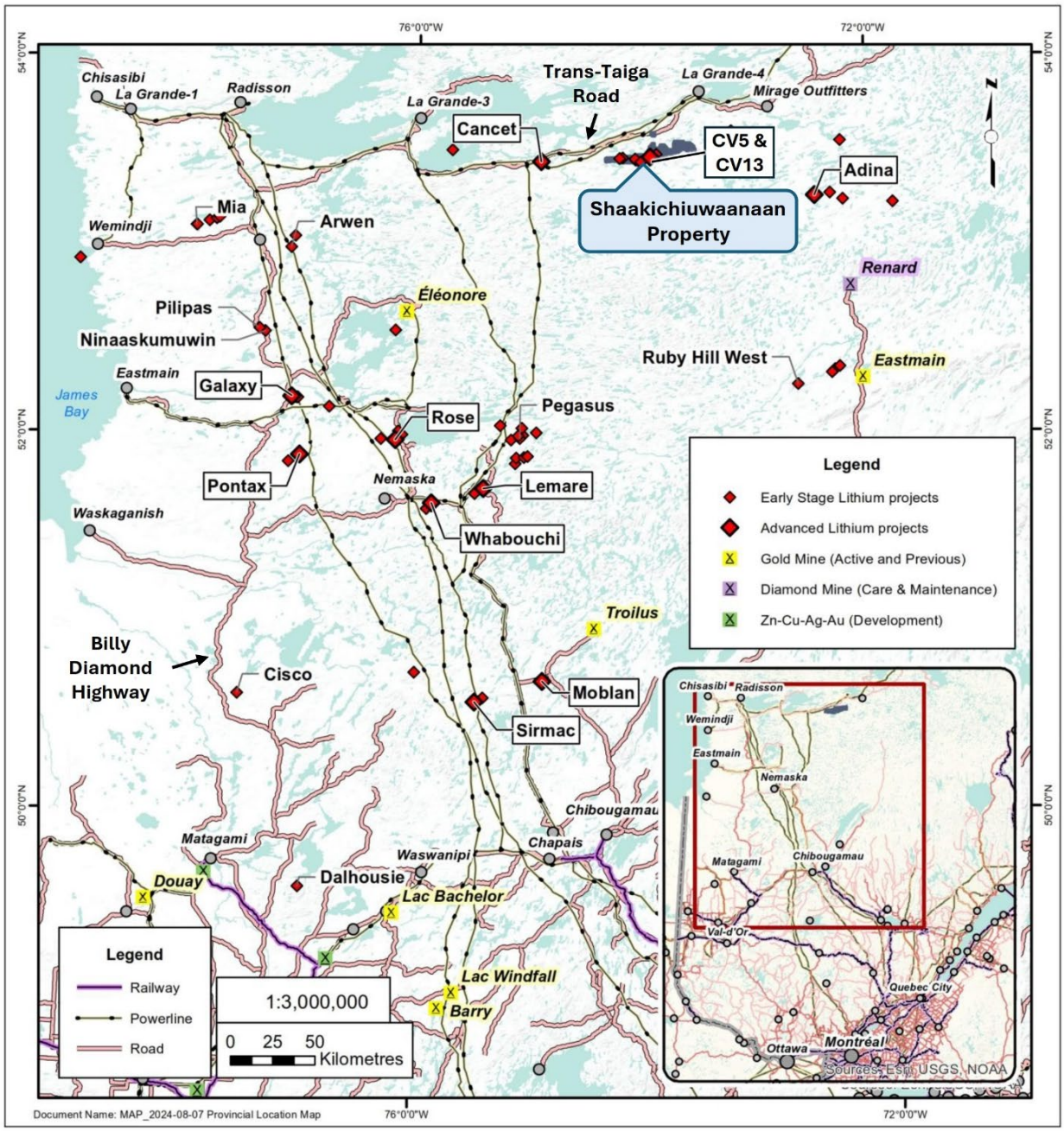
4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Shaakichiuwaanaan Property is located in the Eeyou Istchee James Bay region of Québec, Canada, and is centred on 53°32'00" N, 73°55'00" W, within NTS Sheets 33G08, 33G09, 33H05 and 33H12. The Property is situated approximately 220 km east of Radisson, Québec, and 240 km north-northeast of Nemaska, Québec. The Property consists of two (2) primary claim groups, one straddling KM-270 of the Trans-Taiga Road, and the second with its northern border located directly south of KM-270, approximately 5.8 km from the Trans-Taiga Road and powerline infrastructure corridor (Figure 4.1). The La Grande 4 (LG-4) hydroelectric dam complex is located approximately 30 km north-northeast of the Property. The CV5 Spodumene Pegmatite is located central to the Property, approximately 13 km south of KM-270 on the Trans-Taiga Road, 14 km south of the powerline, and 50 km southwest of the LG-4 dam complex. The CV13 Spodumene Pegmatite is located ~3 km west-southwest of CV5.

The Property is situated on Category III Land within the Eeyou Istchee Cree Territory (Cree Nation of Chisasibi, and Cree Nation of Mistissini), as defined under the James Bay and Northern Québec Agreement (JBNQA). The Eeyou Istchee James Bay Regional Government is the designated municipality for the region, including the Property.

Figure 4.1: Property Location



4.2 Mineral Disposition

The Property is comprised of 463 Exclusive Exploration Rights (EER) (formerly known as CDC – *claim désigné sur carte*, meaning map-designated claim – mineral claims) that cover an area of approximately 23,710 ha (Figure 4.2 to Figure 4.6). The terms “EER” and “claim(s)” are used interchangeably throughout this Report.

The Property is further divided into claim blocks that reflect the various claim acquisitions by the Company – Corvette Main (172 claims), Corvette East (83 claims), FCI East (28 claims), FCI West (83 claims), Deca-Goose (31 claims), Felix (20 claims), KCG (7 claims), JBN-57 (39 claims) – and collectively form two (2) distinct claim groupings (Figure 4.2).

The principal and largest claim grouping of the Shaakichiuwaanaan Property extends dominantly east-west for approximately 51 km and includes the Corvette Main, Corvette East, FCI East, FCI West, Deca-Goose, Felix, and JBN-57 claim blocks. The second and smallest claim grouping includes the KCG claim block and straddles the Trans-Taiga Road at KM-270.

4.3 Tenure Rights

In the province of Québec, the *Mining Act* governs the management of Mineral Resources and the granting of exploration rights for mineral substances during the exploration phase. It also regulates the granting of rights pertaining to the use of these substances during the mining phase. Finally, the *Mining Act* establishes the rights and obligations of the holders of mining rights to ensure maximum development of Québec's mineral resources.

The Québec mineral tenure system (GESTIM) allows individuals and corporations to acquire mineral rights situated on crown and private land. While an EER gives the holder the exclusive right to explore for minerals, it does not grant surface access to private property; separate permission from landowners is still required. Once an EER is registered through GESTIM's online map designation portal (i.e., online staking), the EER is in good standing for an initial 3-year period, followed by 2-year periods thereafter. Upon the end of each EER period, known as the 'Expiry Date', the EER may be renewed indefinitely, subject to applicable renewal fee payments and work expenditure requirements being completed and filed.

The Property EER status was verified using GESTIM (https://gestim.mines.gouv.qc.ca/) by the QP. As of August 4, 2025, the Shaakichiuwaanaan Property consists of 463 mineral titles that cover an area of approximately 23,710 ha and extend dominantly east-west for approximately 51 km as a nearly continuous, single claim block (Figure 4.2). All 463 EERs that comprise the Property are registered 100% in the name of Lithium Innova Inc., a wholly owned subsidiary of PMET Resources Inc., with the *Ministère des Ressources naturelles et des Forêts* (MRNF). A detailed list of the Shaakichiuwaanaan EERs is presented in Appendix A. The QP has not verified the legal titles to the Property or any underlying agreement(s) that may exist concerning the licences or other agreement(s) between third parties and is not aware of any potential restriction to the Company's legal title.

The 463 EERs that comprise the Property were acquired between July 2016 and May 2024 through a combination of option agreements (i.e., claim / EER acquisition agreements) for the initial Corvette block (DG Resource Management and three (3) individuals), FCI (O3 Mining), Deca-Goose (Canadian Mining House, and one (1) individual), Felix (Canadian Mining House), KCG (Canadian Mining House) and JBN-57 (Azimut Exploration) blocks, as well as directly through online map designation (akin to staking). All option agreements for the claim groups that comprise the Property have fully vested with the Company holding 100% interest through Lithium Innova Inc., subject to underlying royalties as described in Section 4.4.

All 463 EERs that comprise the Shaakichiuwaanaan Property are in good standing with term expiry dates ranging from January 22, 2026, to November 30, 2027, with renewals pending on multiple claims. As of August 4, 2025, EER expiry dates, work expenditure credits on file, work expenditure requirements, and renewal fees, for each claim's respective current term, are presented in Appendix A. The QP makes no further assertion regarding the legal status of the Property. The Property has not been legally surveyed to date, and, to the QP's knowledge, no requirement to do so exists.

Figure 4.2: Property Claim Blocks

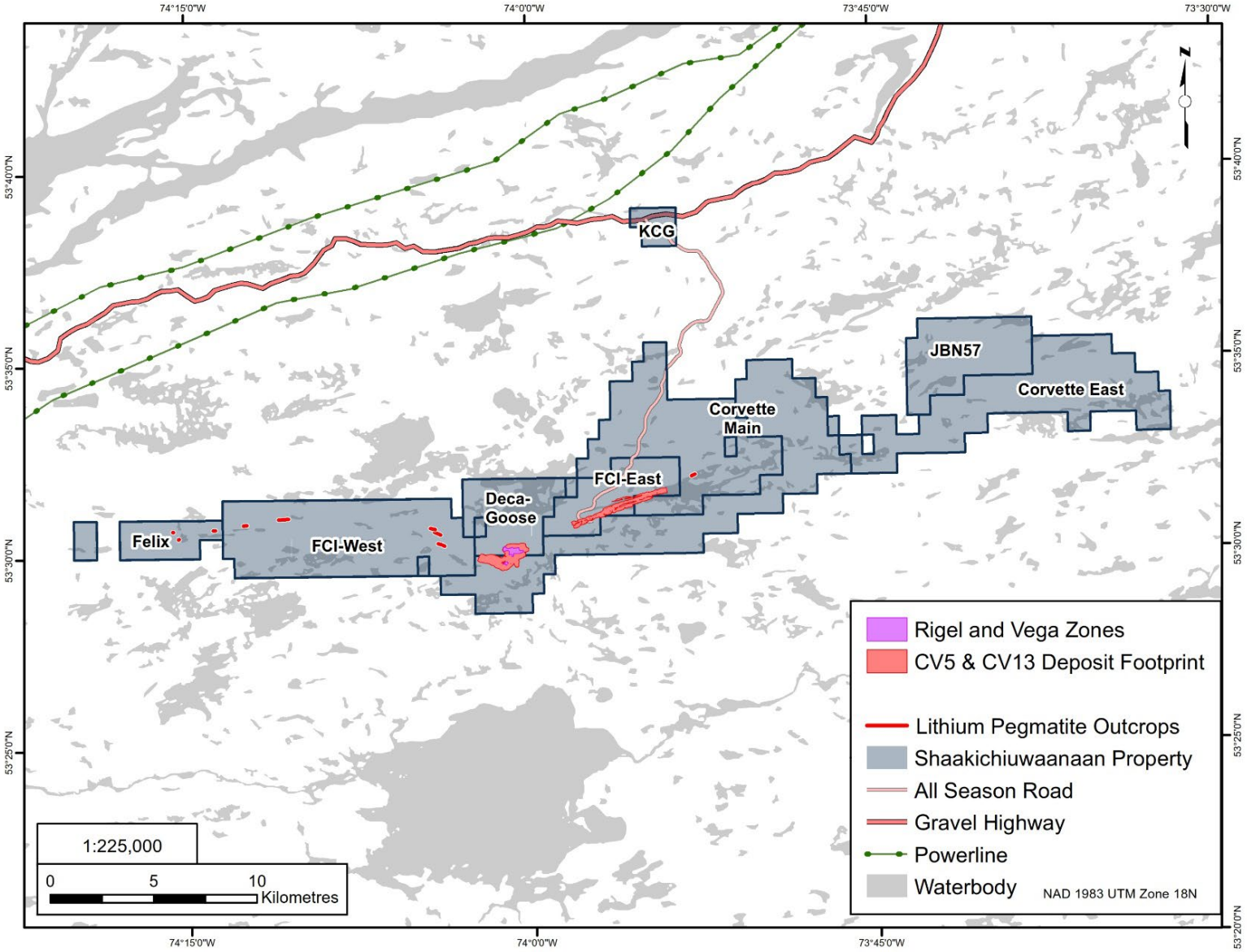


Figure 4.3: Property Claims (West)

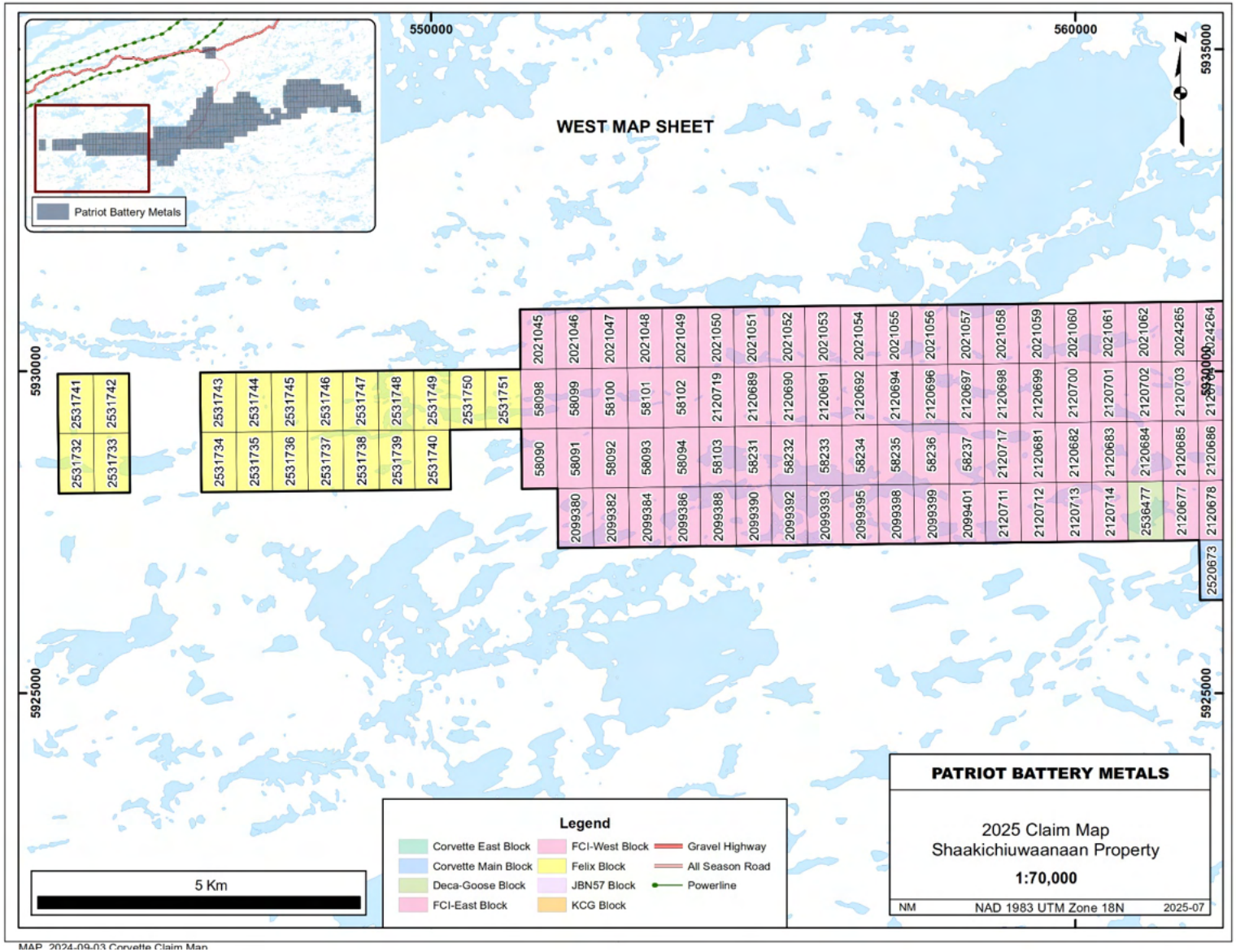


Figure 4.4: Property Claims (Central)

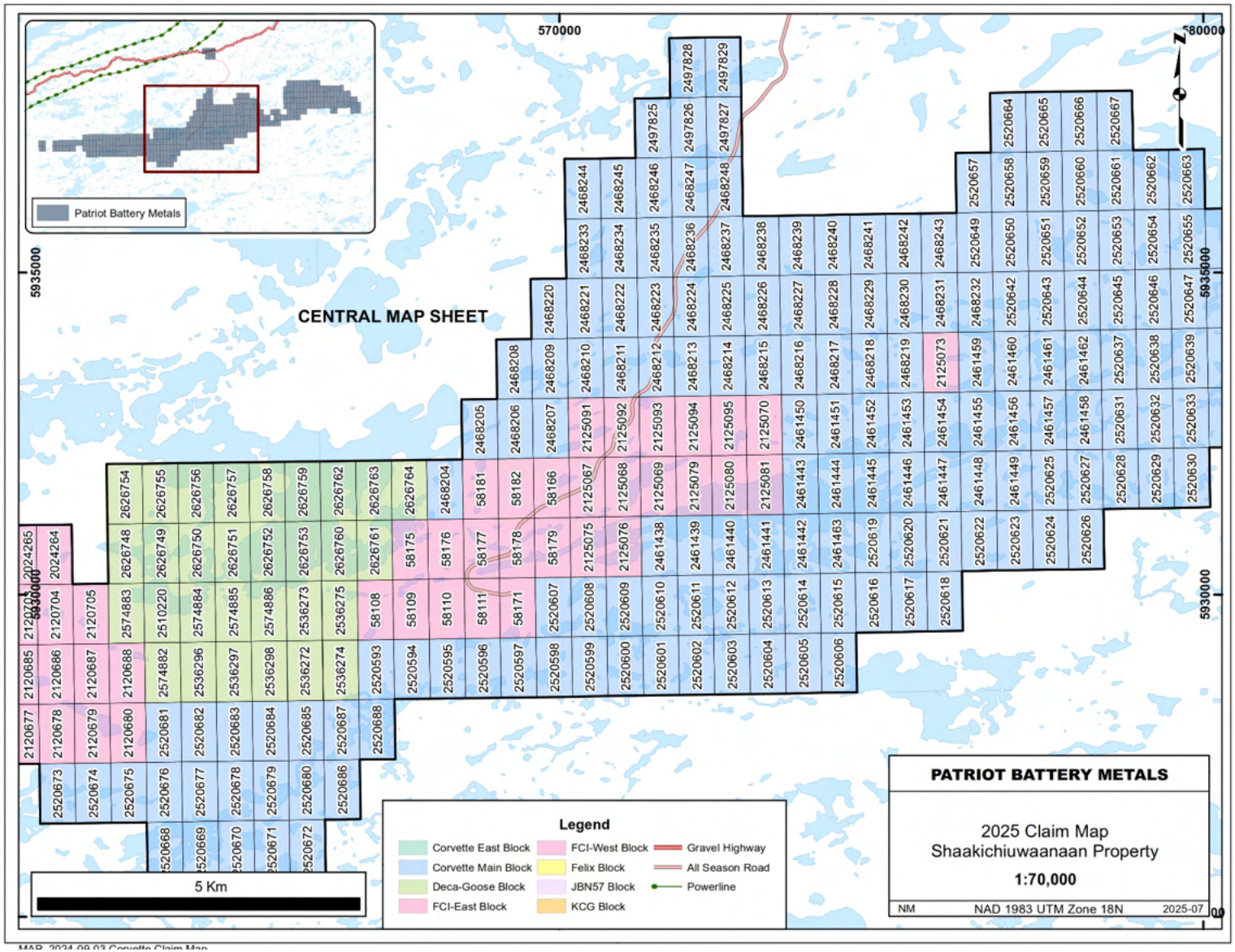


Figure 4.5: Property Claims (East)

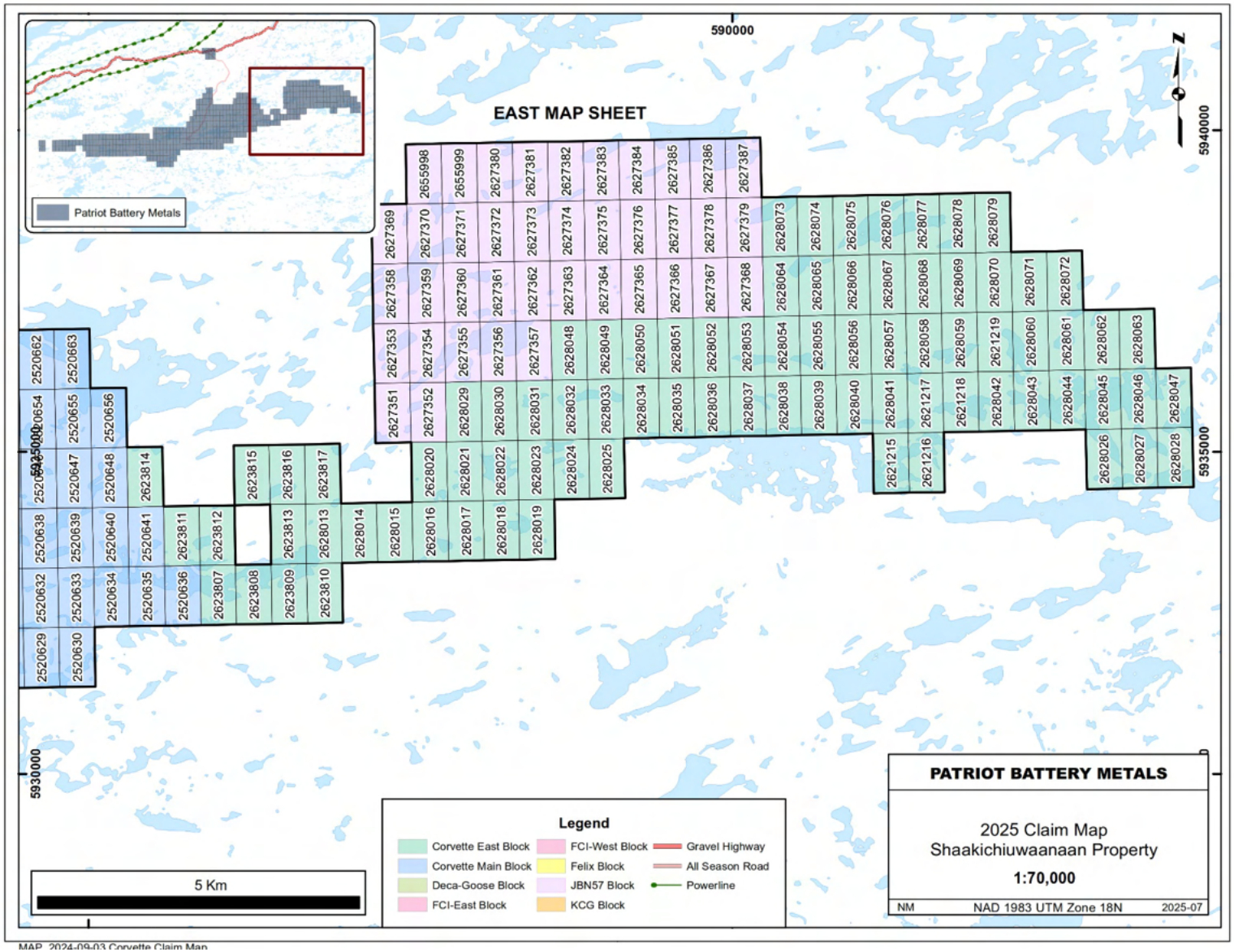
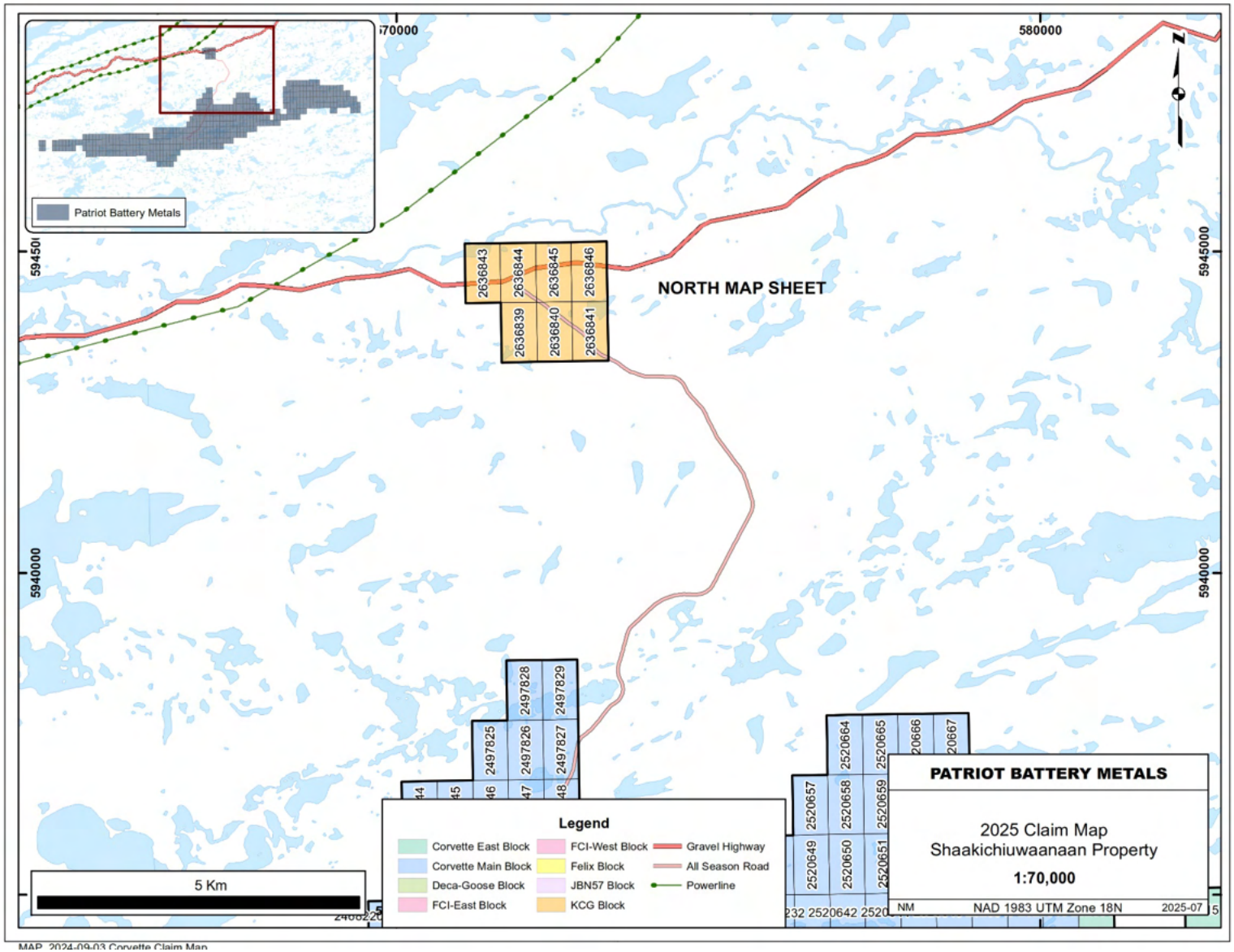


Figure 4.6: Property Claims (North)



4.4 Royalties and Related Information

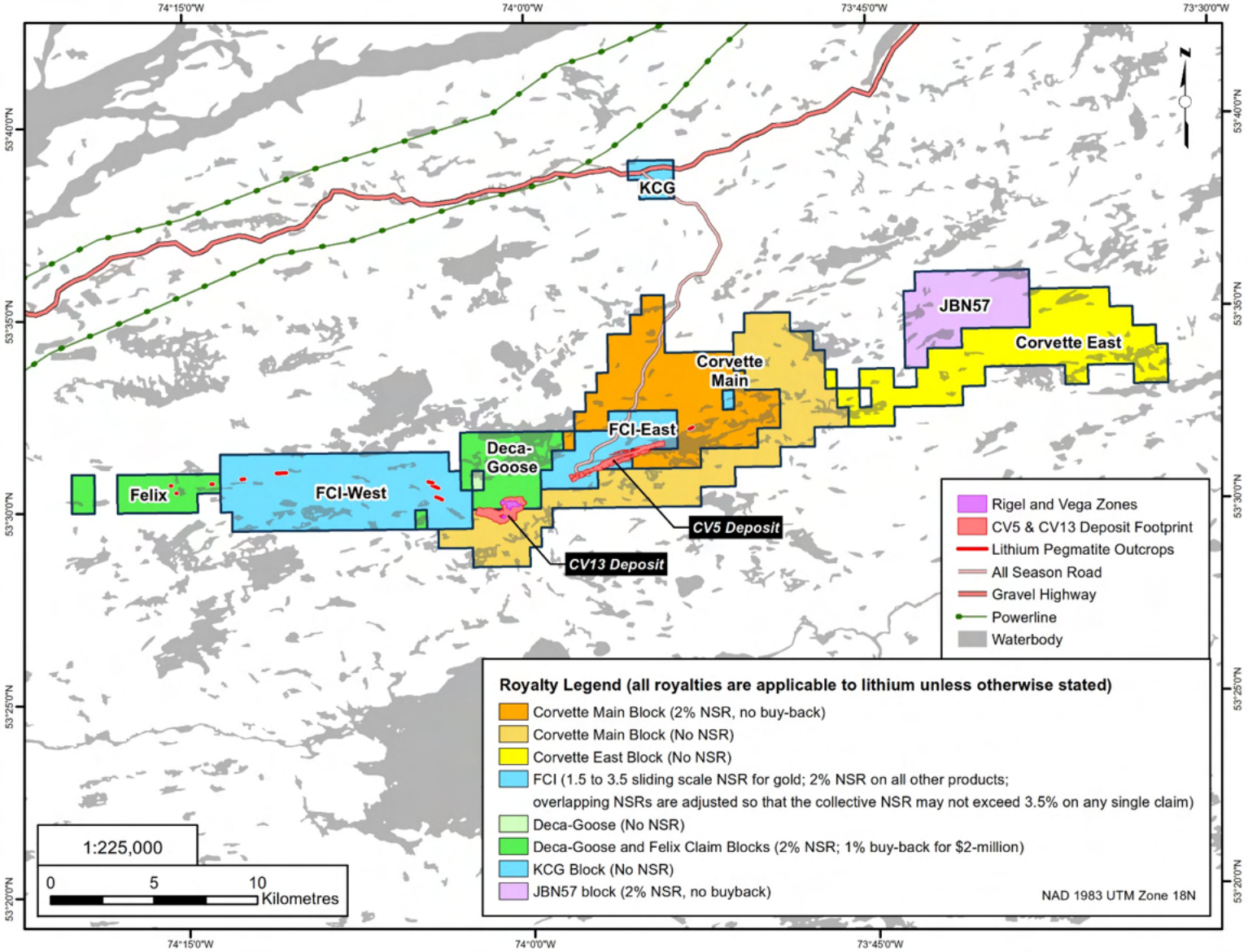
The Property is subject to various royalty obligations pursuant to the claim acquisition agreement for each respective claim block that comprises the Property (Figure 4.7). Of the 463 claims that comprise the Property, 276 are subject to a net smelter royalty (NSR). All NSRs include lithium, caesium, and tantalum, unless otherwise stated; specifically:

- **Corvette Main claim block**: 76 of 172 claims are subject to a 2% NSR held by DG Resource Management Ltd., a private company. There is no buy-back provision.
- **FCI East and West claim blocks**: All 111 claims are subject to an NSR held by OR Royalties Inc. (formerly known as Osisko Gold Royalties Inc.), which is dependent on commodity type and level of production. With respect to the production of precious metals, the claim block is subject to a 1.5% to 3.5% sliding scale NSR. This royalty is primarily based on the amount of production: 1.5% on the first 1M oz; 2.5% on the next 1M oz; and 3.0% on the next 1M oz and above. The remaining 0.5% royalty is based on the spot gold price starting at USD 1,000/oz and reaching the maximum at USD 2,000/oz.

  A 2.0% NSR royalty is present on all other products; provided, however, that if there is an existing royalty applicable on any portion of the claim block, then the percentages noted above (i.e., the sliding scale NSR) shall, as applicable, be adjusted so that the aggregate maximum royalty percentage on a claim shall not exceed, and therefore be capped, to 3.5% at any time. There is no buy-back provision for the NSR on the FCI East and West claim blocks.
- **Deca-Goose and Felix claim blocks**: 50 of 51 claims are subject to a 2% NSR held by 9219-8845 Québec Inc. (d.b.a. Canadian Mining House), a private Québec-based company, of which the Company retains the option of buying back one-half of the NSR for $2,000,000.
- **JBN-57 claim block**: All 39 claims are subject to a 2% NSR held by Azimut Exploration Inc. There is no buy-back provision.

The CV5 Lithium-Caesium-Tantalum (LCT) Pegmatite MRE straddles the Corvette Main and FCI East claim blocks and, therefore, is subject to a 2% NSR split between DG Resource Management Ltd. and OR Royalties Inc. The CV13 LCT Pegmatite MRE, as is currently defined, is subject to a 2% royalty (held by d.b.a. Canadian Mining House) over the northern portions of its eastern and western limbs. The Rigel Caesium Zone is located on royalty-free ground staked directly by the Company. The Vega Caesium Zone is subject to a 2% NSR (half buyback for $2 M) held by d.b.a. Canadian Mining House. The CV4, CV8, CV9, CV10, CV12, CV14, and CV15 LCT pegmatites are subject to a 2% royalty (Figure 4.7).

Figure 4.7: Net Smelter Royalty (NSR) per Claim Block



4.5 Permits

The provincial ministries through which permits and authorizations are issued for normal exploration activities are the *Ministère de l'Environnement, de la Lutte contre les changements climatiques de la Faune et des Parcs* (MELCCFP) and the MRNF. Normal exploration activities such as prospecting, rock sampling, channel sampling, and soil sampling do not require specific authorization from the ministries, as they are effectively granted when the claim is acquired. Authorizations for activities such as ground geophysical surveys (if line-cutting is required), trenching, and drilling are obtained from the MRNF due to the deforestation typically required. Additionally, as of May 6, 2024, an *Autorisation pour travaux d'exploration à impacts* (ATI) is required to carry out exploration work utilizing hydraulic-powered machinery. This includes trenching / stripping, bulk sampling, and exploration drilling (excluding civil engineering, hydrology, and geotechnical). As part of the ATI process, the Company must detail engagement with local communities, with respect to its planned activities, in its submission to the MRNF. The ATI, which takes approximately 2 to 6 weeks to process by the MRNF, covers a 2-year period and may be extended or reapplied for.

Activities such as drilling being completed over lake ice, lake water, or in wetlands require a Declaration of Conformity from the MELCCFP, which is typically a 30-day process. A Request for Review from Fisheries and Oceans Canada (DFO) is also required for any drilling activities completed within a water body. Authorizations from the various ministries are also required for the construction of temporary or permanent camps. In addition, for certain activities such as camp construction, a permit from the EIJBRG may also be required.

The Company currently holds permits / authorizations from the MELCCFP, MRNF, and DFO to carry out surface and drill exploration on the Property. Additionally, the Company holds a lease from the MRNF on an area immediately south of KM-270 of the Trans-Taiga Road for an exploration camp including staging (i.e., laydown), core processing, and storage areas (collectively, Camp Shaakichiuwaanaan). The Company holds various permits from the MRNF, MELCCFP, and EIJBRG for the construction and operation of Camp Shaakichiuwaanaan. Several authorizations from the MELCCFP have been obtained for drinking water and wastewater treatment for the permanent camp, and future requests will be filed accordingly. The Company also holds various authorizations from the ministry for the construction and maintenance of an all-season road extending south from KM-270 of the Trans-Taiga Road to the southwest side of the CV5 Pegmatite.

A formal notification is required to be submitted to the local municipality and landowner(s) at least 30 days prior to the commencement of exploration activities. Industry best practice also demands that a notification be submitted to the local Cree Nation and Tally-Person(s) to ensure they are informed of pending activities and presented with the appropriate contact information. The Property is situated on Category III Land within

the Eeyou Istchee Cree Territory (Cree Nation of Chisasibi, and Cree Nation of Mistissini), as defined under the JBNQA. The EIJBRG is the designated municipality for the region, including the Property. The Company has submitted notifications to the applicable municipality and stakeholders outlining its 2025 mineral exploration plans for the Property, and also meets regularly in Chisasibi, QC, with representatives of the Cree Nation of Chisasibi, including the local Tally Person's family.

The Cree Nations have requested that exploration activities in the region be paused for goose harvesting season, typically between mid-April and mid-May each year. However, with road access from the Trans-Taiga Road directly to CV5, drilling operations may be able to continue throughout this period with the approval of the local Tally-Person.

4.6 Environmental Liabilities

Potential environmental liabilities at the Property include an exploration camp at KM-270 of the Trans-Taiga Road, an all-season road and associated borrow pits, and exploration access trails in certain drill areas. If the Project were not to move forward, this road and access trails may have to be reclaimed, the exploration camp disassembled, and the area reclaimed.

The QPs are not aware of any additional environmental liabilities beyond the normal disturbance related to surface exploration.

4.7 Other Relevant Factors

The QP is not aware of any additional significant factors or risks that may affect access, title, or the right or ability to perform work on the Shaakichiuwaanaan Property. The Property does not overlap any atypically sensitive environmental areas or parks, or historical sites, to the knowledge of the Company. There are no known hindrances to operating at the Property, apart from the goose harvesting season (typically mid-April to mid-May) when the communities request that helicopter flying not be completed, and, potentially, wildfires depending on the season, scale, and location.

The Property lies within Category III lands of the Eeyou Istchee Cree Territory, which are open to exploration subject to the notifications mentioned above. The territory falls under the JBNQA, which is a modern land claims agreement that sets out a structured process and mechanisms for resource management and development, as well as indigenous peoples' consultation. The James Bay region of Québec currently has one (1) active mine, the Éléonore Gold Mine held by Dhilmar Ltd., as well as the Renard Diamond Mine held by Stornoway Diamonds (Canada) Inc., which was put on care and maintenance in October 2023.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Trans-Taiga all-season gravel road and Hydro-Québec's 735 kV power line trends east-west through the region, within approximately 6 km of the northern border of the Company's largest claim grouping (Figure 5.1). The Trans-Taiga Road connects approximately 210 km to the west of the Shaakichiuwaanaan Property to the Billy-Diamond Highway (Route 109) at KM-541, also known as the James Bay Road, which extends north to Radisson and south to Matagami, where it connects to Québec's regional road and railroad network.

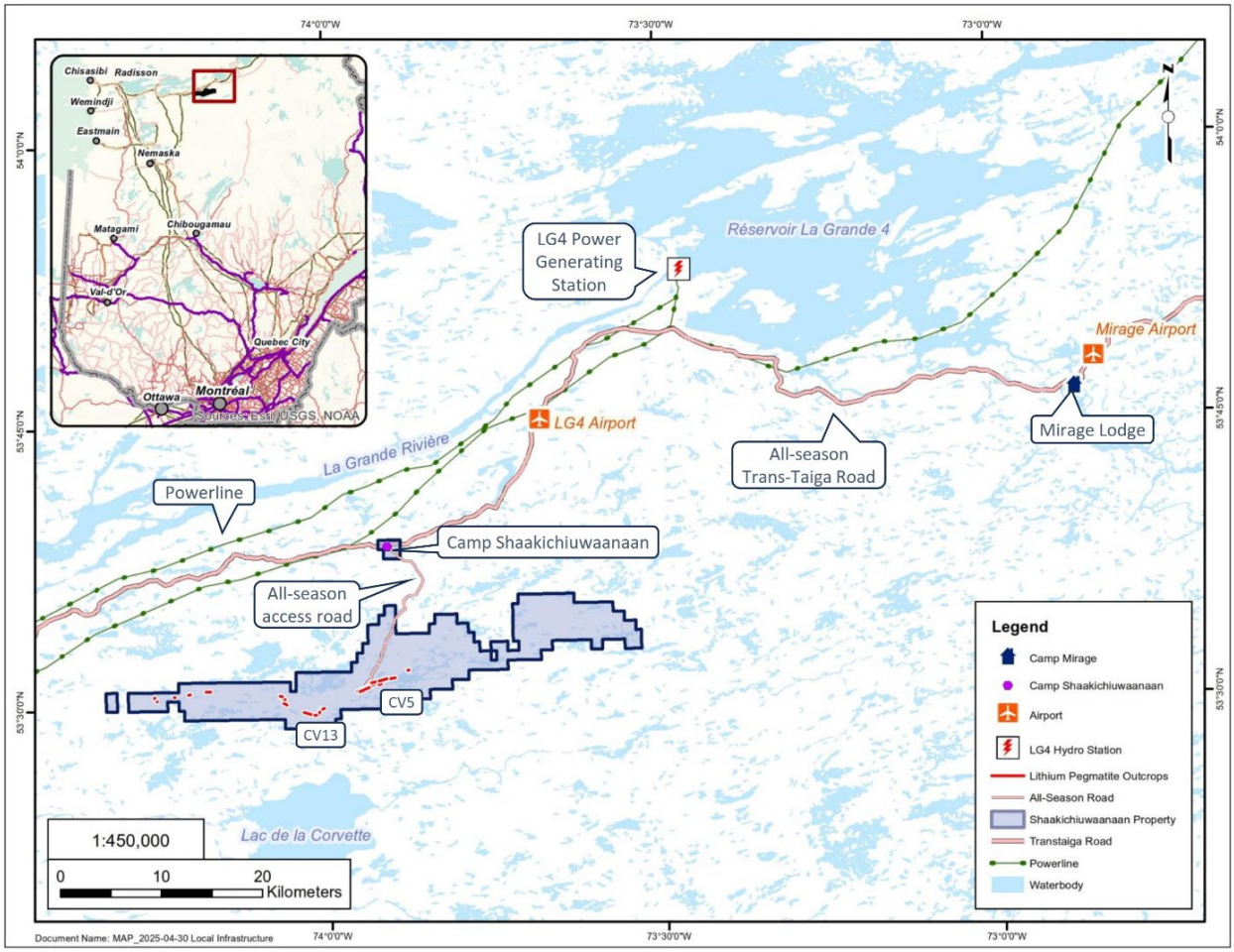
The CV5 Pegmatite at the Property is accessible year-round by an all-season road – constructed by the Company – which extends south from KM-270 of the Trans-Taiga Road and, therefore, is connected to the regional provincial road network. The CV13 Pegmatite is located ~3 km along strike from the CV5 Pegmatite and may be accessed by winter road from CV5. Additionally, the Property may be accessed by float plane or helicopter, and by snowmobile in the winter months.

Continued development of the transportation network in the James Bay Region of Québec is under active consideration as the area continues to attract significant mineral exploration and development interest. For example, La Grande Alliance (LGA) is a memorandum of understanding between the Cree Nation Government and the Government of Québec to plan and execute a 30-year infrastructure program that aims to facilitate the transportation of people and goods and increase the value of natural resources by lowering their transportation costs. La Grande Alliance will act as a hub for organizing and overseeing the development of infrastructure, in the common interest of communities, First Nations, and public and private enterprises seeking to establish, consolidate or harmonize their presence in the territory (LGA, 2022a).

Part of this regional infrastructure program includes a potential railroad extension running north from Matagami, Québec, to KM-541 of the Billy Diamond Highway at the turn-off of the Trans-Taiga Road. Additionally, the programs include plans for an extension of Highway 167, north from the Renard Diamond Mine and connecting to the Trans-Taiga, east of LG-4 Hydroelectric complex. Both development Projects have a projected timeline of six (6) to 15 years (LGA, 2022b).

The James Bay Region and the area of the Property are also covered by the mandate of the *Société du Plan Nord*. The *Société du Plan Nord* is an arm of the Québec Government that is mandated to support the sustainable development of Québec's territory north of the 49th parallel and includes infrastructure and mineral development (Government of Québec, 2022).

Figure 5.1: Local Infrastructure



5.2 Climate

The Property is located in a sub-arctic climate region. Average annual temperatures, precipitation, and snowfall (Weather Spark, 2020) are recorded at the La Grande Rivière Airport (also referred to as “LG-2), near Radisson, Québec, located approximately 220 km west of the Property, within the James Bay Region (Government of Canada, 2022). Over the course of the year, the temperature typically varies from -27°C to 20°C, with rare extremes of -35°C and 26°C. Snow covers the ground from mid-October to late May, typically limiting field work in the winter period to drilling and geophysics.

5.3 Local Resources

Exploration of the Property has been based out of Mirage Adventure Lodge, located at KM-358 on the Trans-Taiga Road, as well as Shaakichiuwaanaan Camp, located on the Company’s KCG claim block at KM-270 of the Trans-Taiga Road (Figure 5.1).

Mirage Lodge is located approximately 50 km to the east-northeast of the Property, and 75 km east-northeast of the CV5 Pegmatite. The lodge provides accommodation, meals, bulk fuel (gas, diesel, Jet A), a local airstrip, as well as internet access. A regional ground transportation service provider, Kepa Transport, provides weekly ground shipping services direct from Val-d'Or to Mirage and vice versa.

Camp Shaakichiuwaanaan is located within the Property's most northern claim block (KCG), approximately 13 km directly north of the CV5 Pegmatite, adjacent to the south side of KM-270 of the Trans-Taiga Road. The camp was constructed by the Company to support ongoing exploration and development activities at the Property, with operations beginning in January 2024. The camp has a current capacity of 88 people, with an expansion of up to 150 people planned. The Company also holds a lease covering the area of the camp, which provides additional space for storage, core processing, exploration laydown / staging, helicopter pads, and equipment maintenance. In May 2025, the Company consolidated its base of activities to camp, including core processing, with Mirage primarily only used for crew changes through its airstrip since that time.

Radisson, with a population of ~470 people, is the closest community accessible by road from Shaakichiuwaanaan Camp and is located approximately 220 km west of the Property and 245 km west of the Shaakichiuwaanaan camp. Radisson is serviced regularly by scheduled flights through the adjacent LG-2 Airport and is the closest airport to the Property with regularly scheduled flights. The Cree communities of Wemindji and Chisasibi are each located approximately 325 km west of the Property. Both Wemindji and Chisasibi host a larger array of service providers to the region and are serviced by regularly scheduled flights. Radisson, Wemindji, and Chisasibi, as well as Mirage, are accessible by road with connections to the main provincial network. Therefore, any supplies not available from these locations may be obtained by road from Val-d'Or. Val-d'Or, and the entire Abitibi region, has a long active mining history with significant labour experience to support the Project.

In addition to access by road from nearby communities, charter aircraft may be used to access the La Grande-3 (KM-100) and La Grande-4 (KM-292) airstrips located along the Trans-Taiga Road. Although these airstrips were constructed primarily to service Hydro-Québec, they are under active transition to allow for consistent public use. The Company expects to have regular access to the La Grande-4 (LG-4) airstrip for regularly scheduled charters to support exploration and development activities in the future.

5.4 Infrastructure

At the Property, infrastructure currently consists of a permanent camp (Camp Shaakichiuwaanaan) owned by the Company at KM-270 of the Trans-Taiga Road (south side) within the KCG claim block. The camp has been constructed to support continued all-season mineral exploration and development at the Property.

Additionally, the Company has completed construction of an approximately 20 km long all-weather road extending south from KM-270 of the Trans-Taiga Road to the CV5 Pegmatite at the Property (Figure 5.1). Various exploration access trails have also been constructed at the Property to support ground-based drilling activities.

The CV5 Pegmatite is located approximately 13 km to the south of the regional and all-weather Trans-Taiga Road, approximately 14 km south of a regional 735 kV power line, approximately 30 km south-southwest of the LG-4- airstrip, and approximately 50 km south-southwest of the LG-4 hydroelectric generating station, owned and operated by Hydro-Québec. The Tilly substation is located approximately 1.5 km south of the LG-4 dam. The LG-4 complex (KM-292 of the Trans-Taiga Road) has an installed capacity of 2,779 MW (Hydro-Québec, 2022). The majority of Québec's power is produced from a series of hydroelectric generating stations located along this east-west trending infrastructure corridor. Therefore, the infrastructure is well maintained, bridges are rated for high-tonnage traffic, and the Trans-Taiga Road is accessible year-round. This power infrastructure allows Québec to have electricity costs 49% lower than in the G7 countries on average (Investissement Québec, 2023).

5.5 Physiography

The Property topography consists of forested, gently rolling hills, drainages, and muskeg swamps between approximately 260 m and 350 m elevation, typical of the James Bay Region. Snow cover typically occurs from mid-October to late May. Vegetation is characteristic of the Boreal Vegetation Zone in Québec and consists mainly of black spruce, lesser alder, poplar, birch, and various shrubs. This region is typically inhabited by moose, woodland caribou, and black bears, as well as numerous smaller mammals.

6. HISTORY

Unless otherwise noted, the mineral exploration history discussed herein pertains to the principal claim group of the Shaakichiuwaanaan Property, consisting of the Corvette Main, Corvette East, FCI West, FCI East, Deca-Goose, Felix, and JBN-57 claim blocks. No significant or focused mineral exploration, mineral showings, or drill holes have been documented on the KCG claim block, which was primarily acquired for ease of access and use for a mineral exploration camp (Camp Shaakichiuwaanaan).

6.1 Prior Ownership

The Shaakichiuwaanaan Property is extensive, with the principal claim block covering a general east-west trend of more than 50 km. For this reason, differing areas of the Property have been assessed by numerous companies since the 1950s. The following is a summary of the more pertinent historical ownership documented for the Property based on field work completed.

The earliest documented mineral exploration work in the area dates back to the late 1950s. Tyrone Mines Ltd. (a subsidiary of Phelps Dodge Corporation) prospected for base metals in 1959 and dug five (5) trenches. Their work led to the discovery of a copper showing (1.15% Cu over 2.1 m in channel) in trench TR-9.

In 1997, Virginia Gold Mines Inc. (Virginia) acquired an extensive land position in the region (Félicie – Corvette Ouest – Island Lake – properties, collectively “FCI), which overlapped a large portion of the present-day Property. The Property was optioned several times in subsequent years; however, the ownership of the claim group was retained by Virginia.

In 2015, Virginia merged with Osisko Gold Royalties. During a subsequent restructuring, the FCI claims were transferred to a newly established entity called Osisko Exploration James Bay, held by Osisko Mining Inc. Several claims within the FCI claim group subsequently lapsed in the years that followed. In 2019, Osisko Mining Inc. spun out some of its assets into a new company called O3 Mining Inc., which, at that time, held the mineral rights to the FCI claims. In late 2018, the FCI East block (28 claims) was optioned to 92 Resources Corp (subsequently restructured to Gaia Metals Corp. on October 17, 2019, to Patriot Battery Metals Inc. on June 10, 2021, and to PMET Resources Inc. on September 18, 2025) for up to 75% interest, subject to certain terms and conditions. The agreement was later amended in early 2019 to also include the FCI West block (83 claims) and subsequently, in early 2022, where the Company acquired / purchased the remaining 25% interest in both the FCI East and West claim blocks.

The initial 76 claims of the present-day Corvette Property (part of the Corvette Main block) were staked in 2016, via map designation, for their lithium potential. The claims were staked by DG Resource Management Ltd., and a 100% interest subsequently vended to 92 Resources Corp (subsequently restructured to Gaia Metals Corp. on October 17, 2019, and again to PMET Resources Inc. on June 10, 2021). The claim position was subsequently expanded by the Company via map designation in summer 2018 (96 claims, part of the Corvette Main claim block), and again in fall 2021 (83 claims, the 'Corvette East' claim block).

In early 2022, the Property was further expanded through option agreements with one (1) individual for Deca-Goose (1 claim), and with Canadian Mining House for the Deca-Goose (31 claims) and Felix (20 claims) claim blocks. In May 2023, the Company acquired the KCG claim block (7 claims) situated along the Trans-Taiga Road, directly north of the other claim blocks that comprise the Property. The KCG claims were acquired from Canadian Mining House to allow for ease of access and use for the purposes of a camp to support mineral exploration of the Property. Finally, in May 2024, the Company acquired the JBN-57 claim block from Azimut Exploration Inc.

The present-day Shaakichiuwaanaan Property is comprised of 463 claims, totalling 23,710 ha, with Lithium Innova Inc., a wholly owned subsidiary of PMET Resources Inc., recorded as the 100% registered title holder with the MRNF.

6.2 Previous Exploration and Development

The following section discusses the historical mineral exploration that has overlapped the present-day Shaakichiuwaanaan Property. The QP notes that surface rock sample assays (i.e., grab, and often chip), as historically documented, are selective by nature and represent a point location and, therefore, may not necessarily be fully representative of the mineralized horizon sampled. Further, not all historical documentation provides a complete data set of sampling results (surface or drill), nor details of sampling approach, for a particular program and, therefore, any interpretation of the data should be understood within this context. Where stated, the values presented herein for the historical work are those that define the formal mineral showing / prospect locality and additional information is provided as practical / available.

In the late 1950s, Tyrone Mines Ltd. completed a work program that overlapped the present-day Property that included reconnaissance prospecting and trenching (pit blasting). This work resulted in the discovery of several Cu-Au-Ag showings including the Tyrone T-9 Showing with 3.19% Cu, 0.82 g/t Au, 38.4 g/t Ag in outcrop and 1.15% Cu over 2.1 m in channel, and the Lac Smokycat-SO Showing with sample grades including 1.75% Cu, 1.47 g/t Au, and 40.5 g/t Ag, and 0.76% Cu, 0.20 g/t Au, and 97.7 g/t Ag; located on the present-day FCI West claim block (Ekstrom, 1960 - GM10515).

From the 1950s through to 1997, the Property area was subject to limited exploration work only, including various regional mapping surveys by the federal and provincial governments as well as airborne magnetic and electromagnetic surveys. A NI 43-101 technical report completed in 2014 by Virginia and their option partner at the time, Komet Resources Inc., provides a good summary of the exploration over the area through 2013 (Quellette & Vachon, 2014 - GM68359). A NI 43-101 technical report completed by the Company in 2022 provides additional summary information on historical exploration through April 2022 (Knox, 2022). The following is a brief summary of exploration over the last few decades, which includes excerpts, and paraphrases from these two (2) technical reports.

In 1974, a regional lake bottom sediment survey was completed with multiple samples collected over the present-day Property; however, determining exact locations is challenging (Pride, 1978 - GM34044).

In 1996, Phelps Dodge Corporation completed a helicopter-borne magnetic and electromagnetic survey north of Corvette Lake followed by a short program of geological mapping (MRNF, 2023) and (Johnson, 1996 - GM56869).

In 1997, Virginia acquired an extensive land position in the region, which overlapped a large portion of the present-day Property. The focus was base and precious metals exploration (211 rock samples collected) led to the discovery of the Golden Gap Showing (32.7 g/t Au in outcrop, and 14.3 g/t Au over 2 m in channel), as well as two (2) copper-zinc showings (Bambic, 1997) and (Schmidt, Samson, William-Jones, & Smith, 2017). As part of the field work in 1997 and 1998, Virginia resampled the historical Tyrone Mines' trenches, as well as completed geological mapping, prospecting, and rock-till-soil sampling on the Property. In 1998, Virginia discovered the Golden East Showing (20.3 g/t Au in grab sample), the Felix Showing (three (3) samples ranging from 0.11% to 1.20% Cu and up to 0.35 g/t Au and 9.9 g/t Ag), in addition to completing regional mapping on portions of the present-day Felix claim block (de Chavigny, 1999 - GM 56161). Follow-up work in 1999 led to additional gold discoveries near Golden East with Deca-1 to Deca-4 (1.91 g/t Au over 5 m in channel and 6.91 g/t Au in grab sample), Goose-1 (1.98 g/t Au), and Goose-2 (3.74 g/t Au) showings, which overlap the Company's Deca-Goose claim block (McCracken & Cunningham, 2023). Further sampling at the Golden Gap Showing returned 5.76 g/t Au over 3 m. In 2000, the Sericite Showing was discovered (1.89% Cu, 0.3 g/t Au, 150 g/t Ag, and 1.45% Zn) and in 2001, the first drill holes on the Property were completed, targeting the Golden Gap Showing. Circa 1,400 surface rock samples were collected across the present-day Property over the 1997 through 2000 exploration programs.

The Property was optioned several times in subsequent years with additional groundwork completed each time, including further drilling, prospecting, mapping, soil sampling, as well as ground magnetic and IP surveys, which overlapped the Property to various extents (McCracken & Cunningham, 2023), (Demers & Blanchet, 2001), (Roy & Archer, 2010 - GM65536) and (Quellette & Vachon, 2014 - GM68359). In 2005,

the Félicie Showing was re-discovered (formerly the Lac Magin-Sud Showing, initially discovered in 1959) characterized by a sulphide bearing quartz-feldspar dyke with a grab sample assay of 5.54 g/t Au, >100 g/t Ag, 1.86% Cu, 1.56% Pb, and 4.94% Zn (Demers & Blanchet, 2001).

The drilling programs completed included holes at the Sericite Showing (302 m over two (2) holes in 2013), the Lac Bruno boulder field (391 m over three holes in 2007), Golden Gap (combined total of 5,267 m in 24 holes; between 2001 and 2013) and the Deca-Goose area (325 m over three (3) holes in 2001). The best historical precious metals drill intercept is from Golden Gap with 10.48 g/t Au over 7 m, obtained in 2007 (drill hole FCI-07-003). In addition to drill hole FCI 07 003, numerous other holes at Golden Gap returned nil to moderate precious metals mineralization, including 1.62 g/t Au over 2.5 m (IL-01-01), 0.27 g/t Au over 15 m and 1.35 g/t Au over 4 m (IL 01 02), and 0.59 g/t Au over 11.4 m (IL-01-03). At Golden East, a single drill hole was completed (IL-01-04) and returned 0.46 g/t Au over 1.0 m. Two (2) drill holes were completed at the Deca-1 area and returned 1.10 g/t Au over 1.0 m (IL-01-05), and 0.72 g/t Au over 1.0 m (IL-01-06).

Between 1997 and 2013, the dominant focus was precious metals, with a secondary focus on base metals. No exploration for LCT pegmatite was completed.

Some of the main surface mineral occurrences documented historically on the Property are summarized in Table 6.1 below, as well as Figure 6.1, Figure 6.2 and Figure 6.3.

Table 6.1: Surface Showing Highlights from Historical Work on the Property

Showing / Prospect	Year Discovered	Source	Cu (%)	Au (g/t)	Ag (g/t)	Zn (%)
Lac SmokyCat-SO	1957	Outcrop	1.75	1.47	40.5	
Lac de la Corvette	1959	Outcrop	0.70	0.02	19.1	
Tyrone-T9	1959	Outcrop	3.19	0.82	38.4	
Golden Gap	1997	Outcrop		108.90		
Golden East	1998	Outcrop		21.20		
Lac Long	1998	Outcrop	1.37	n/a	15.2	
Felix	1998	Outcrop	1.20	0.35	9	
Deca-1 to Deca-4	1999	Outcrop		6.91		
Goose-1	1999	Outcrop		1.98		
Goose-2	1999	Outcrop		3.74		
Sericite	2000	Outcrop	1.89	0.30	150	1.45

Showing / Prospect	Year Discovered	Source	Cu (%)	Au (g/t)	Ag (g/t)	Zn (%)
Bonoeil	2009	Outcrop	1.40	n/a	n/a	
Smith-Lac Magin	2010	Outcrop	0.65	0.64	25	

**Note: Surface rock sample assays (i.e., grab / chip), as historically documented, are selective by nature and represent a point location and, therefore, may not necessarily be fully representative of the mineralized horizon sampled. The sample assays presented are those most commonly associated with the showing.*

In 2008 and 2009, the Property was flown with high-resolution magnetics by the *Ministère de l'Énergie et des Ressources naturelles* (MERN, now the MRNF) over the course of a multi-year campaign, covering a large area of the James Bay Region (D'Amours, 2011 - DP 2011-08) and (Goldak Airborne Surveys, 2009 - DP 2009-01). The surveys were flown at a spacing of 250 m with tie lines at 2.5 km and provides a base data set of magnetics over the entire Property.

In 2016, the Company (then under the name of 92 Resources Inc.) acquired an initial claim position in the area (part of the present-day Corvette Main claim block). The claims were acquired, in part, because of the words "*cristaux de spodumène*" in pegmatite that was noted in an outcrop description (RO-IL-06-023) from a 2006 exploration program carried out by Virginia Mines (Demers & Blanchet, 2001). The description of the mineral spodumene indicated lithium pegmatite.

In recent years, the area has seen a renewed focus of exploration including geological mapping (Goutier, et al., 2021 - RP 2020-01), base metals sampling (Romain & Larivière, 2021 - GM 72626) and lithium pegmatite sampling (Azimut Exploration, 2024 - GM 73760), some of which overlapped with the eastern half of the Property (Corvette Main, Corvette East, and JBN-57 claim blocks).

Mineral exploration by the Company began in 2017 and is summarized in Chapter 9 of this Report. In June 2023, the Company completed a maiden MRE (lithium and tantalum) for the CV5 LCT Pegmatite (McCracken & Cunningham, 2023),. An updated MRE (lithium and tantalum) for the Shaakichiuwaanaan Property, including both the CV5 and CV13 LCT pegmatites, was announced in August 2024 (lithium and tantalum) (McCracken, et al., 2024) and again in May 2025 (lithium, tantalum, and gallium) (PMET Resources, 2025b).

The Shaakichiuwaanaan Mineral Resource Estimate (lithium, caesium, tantalum, and gallium) was updated again on July 20, 2025, (McCracken & Cunningham, 2025) with an Effective Date of June 20, 2025, and is the current MRE for the Project. The current MRE is the fourth for the Shaakichiuwaanaan Project and includes the addition of caesium at the Rigel and Vega zones within the CV13 Pegmatite.

6.3 Historical Mineral Resources

There are no known historical Mineral Resources or Reserves on the Property.

6.4 Production

There is no known historical production on the Property.

Figure 6.1: Historical Work Summary (West)

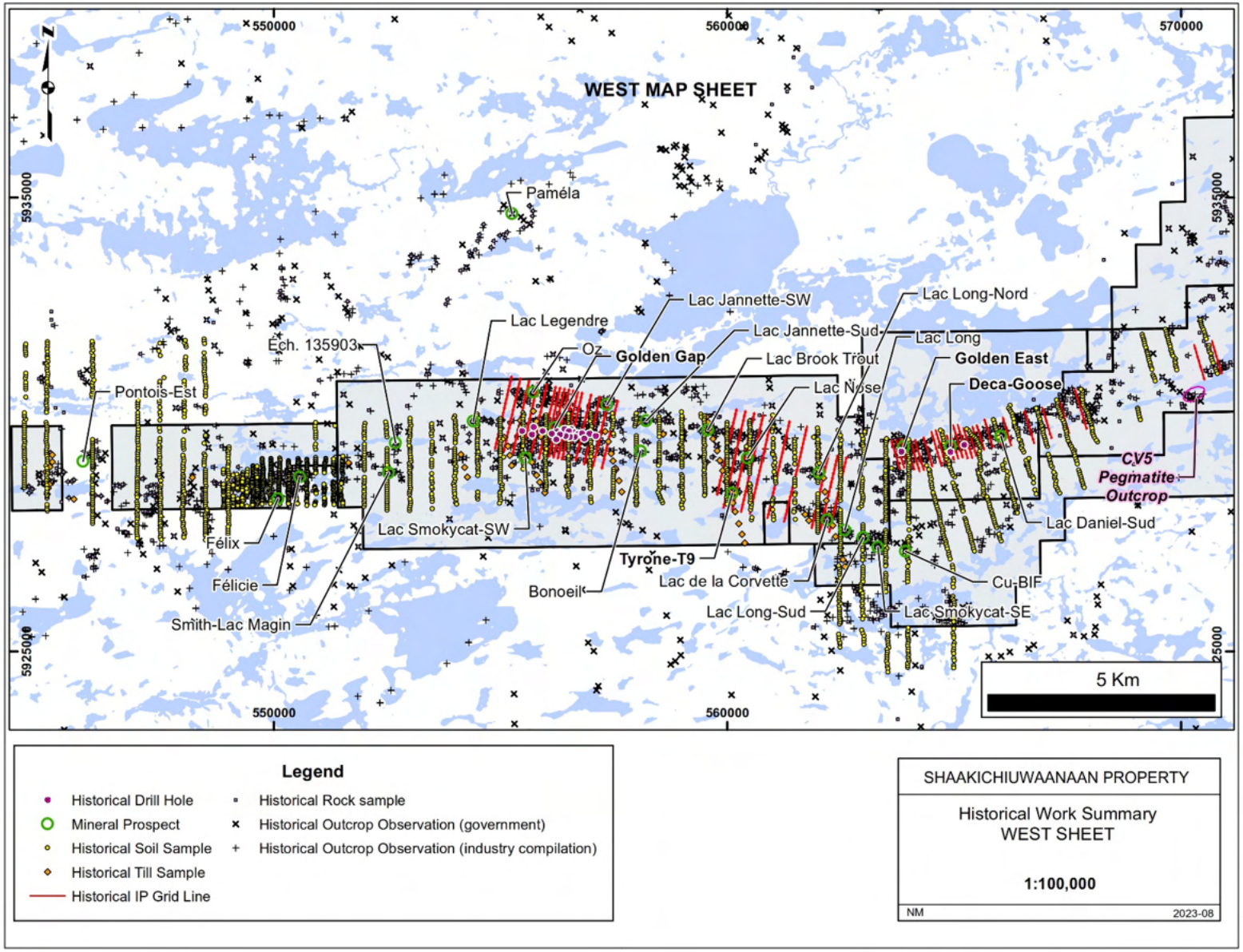


Figure 6.2: Historical Work Summary (East)

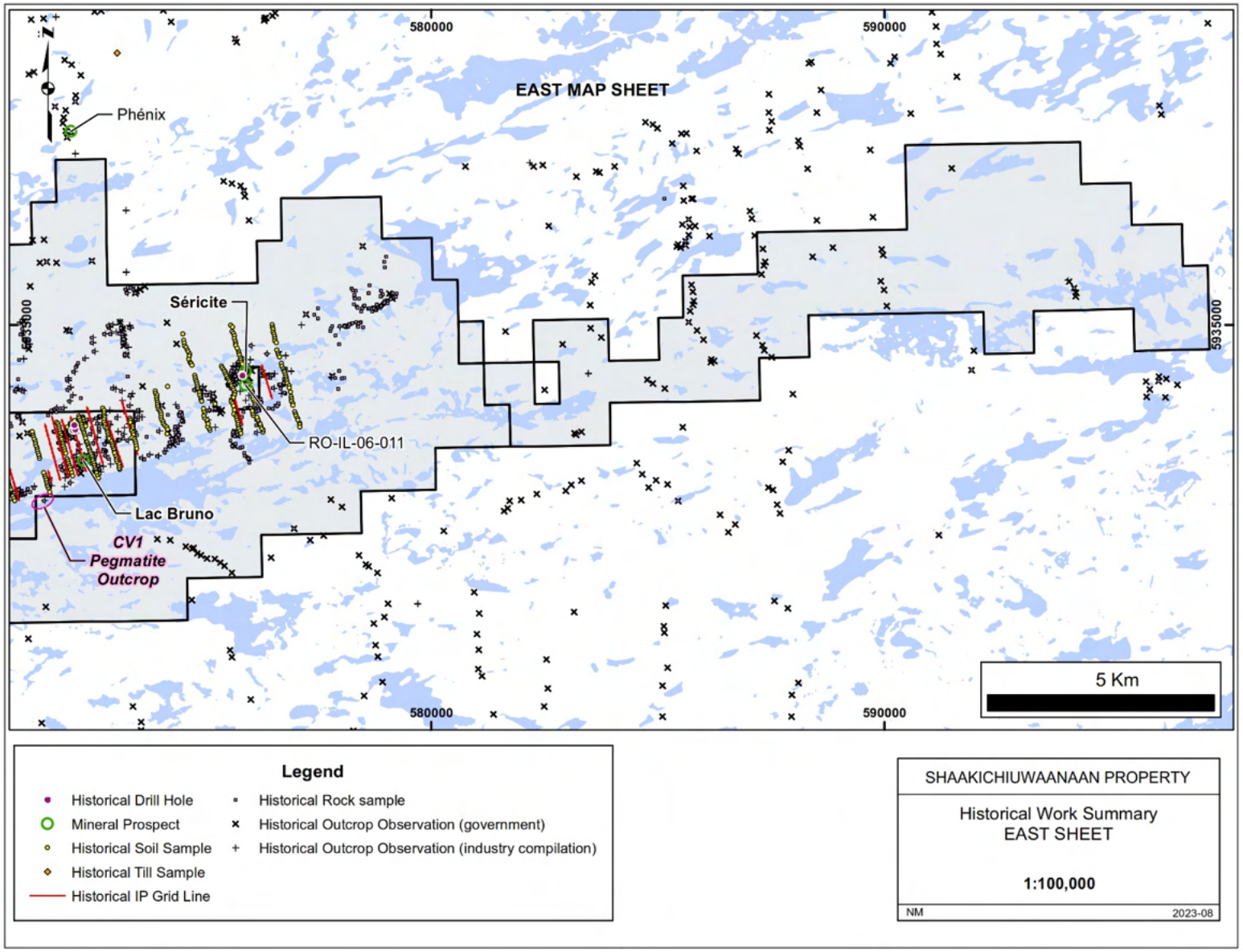
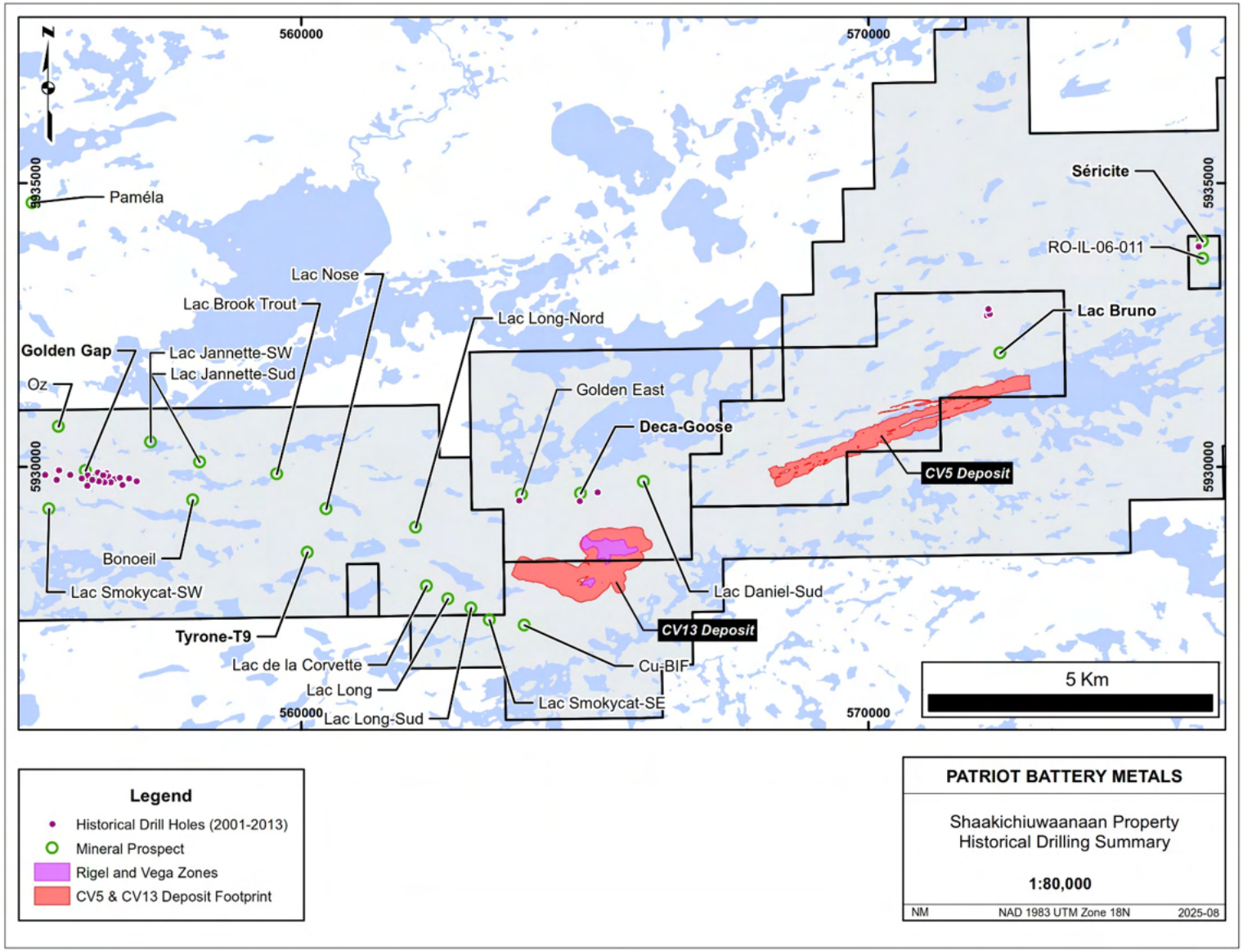


Figure 6.3: Historical Drill Hole Summary



7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Shaakichiuwaanaan Property is situated within the Archean Superior Province of the Canadian Shield, which extends from Manitoba to Québec and covers approximately 750,000 km^2 of Québec. Within the region, the Superior Province is divided into four (4) distinct sub-provinces based on their lithological, metamorphic, geophysical, and structural characteristics: Opatica, Nemiscau, Opinaca, and La Grande (Figure 7.1). The Property is situated within the central portions of the volcano-plutonic, La Grande sub-province, proximal to the Opinaca sub-province to the south. The region is considered to have strong exploration potential for a variety of commodities, including base and precious metals, and lithium (LCT pegmatite).

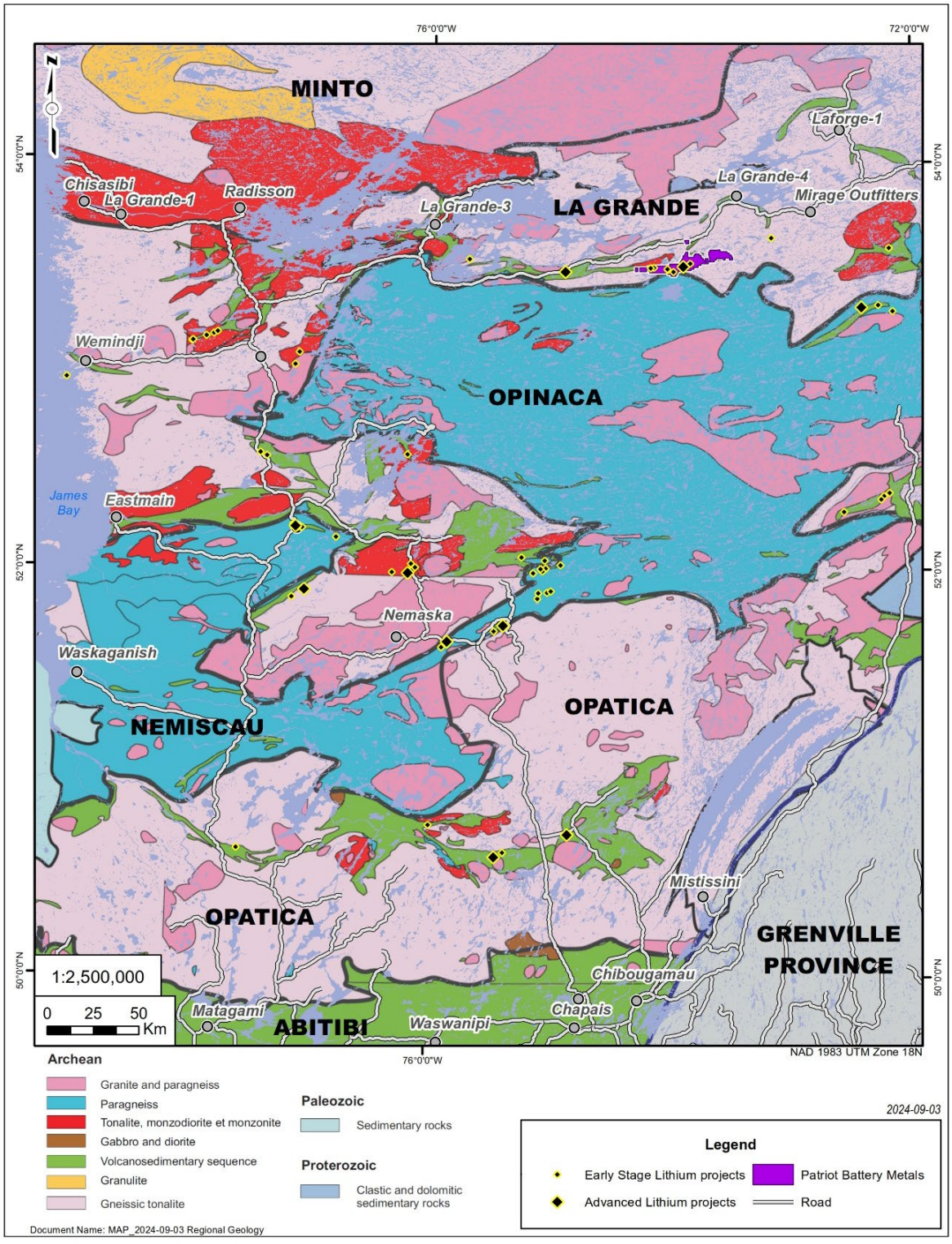
The La Grande sub-province is a volcano-plutonic assemblage oriented parallel to the Wemindji-Caniapiscau structural corridor (Houle, 2004). It consists of two (2) main domains (Percival, et al., 2012): the Eastmain River Belt (Upper and Lower) and the La Grande River Belt. The Property is situated within the La Grande River (Greenstone) Belt, characterized by a volcano-sedimentary sequence. This belt occupies the older, more evolved, northern domain (Houle, 2004; Percival, et al., 2012) and is comprised of two (2) supracrustal volcanic sequences (2750-2730 Ma) and interstratified metasediments. The lower basalt sequence sits unconformably atop the Mesoarchean basement (3360-2790 Ma) and locally overlies U-bearing pebble conglomerate, quartz arenite and minor carbonate (Roscoe & Donaldson, 1988; Goutier & Dion, 2004). The upper sequence is a result of crustal assimilation by komatiitic liquids. It is made up of felsic to intermediate volcanics, komatiite, volcanoclastic rocks, and iron formation capped by basalt and high-Mg andesite. This is a typical assemblage, especially in the Guyer-LG-4 sector (St-Seymour & Francis, 1988; Lucas & St-Onge, 1998).

The La Grande sub-province borders the plutonic Minto sub-province to the north and is bounded to the south by the metasedimentary and plutonic Opinaca sub-province (Lucas & St-Onge, 1998; Houle, 2004; Percival, 2007). Collectively, the La Grande and Opinaca sub-provinces host a significant number of the known spodumene pegmatite occurrences in Québec.

Regional metamorphism increases from greenschist facies in the centre of La Grande outwards to amphibolite facies in the north and southeast (Card, 1986; Houle, 2004). Steeply dipping structural trends transition from E-W in the southwest to NE-SW within northern La Grande, most of which developed between 2700 Ma and 2680 Ma (Percival, et al., 2012). A series of Proterozoic dykes, 2740-2680 Ma plutonic rocks, and the Paleoproterozoic Sakami Formation (siliciclastic infilled grabens) punctuates the

Archean rocks (Houle, 2004; Percival, et al., 2012). Rich Ni-Cu occurrences, often with associated PGE and Cr, have also been found in komatiitic flows and ultramafic intrusions in the region (Houle, 2004).

Figure 7.1: Regional Geology



7.2 Property Geology

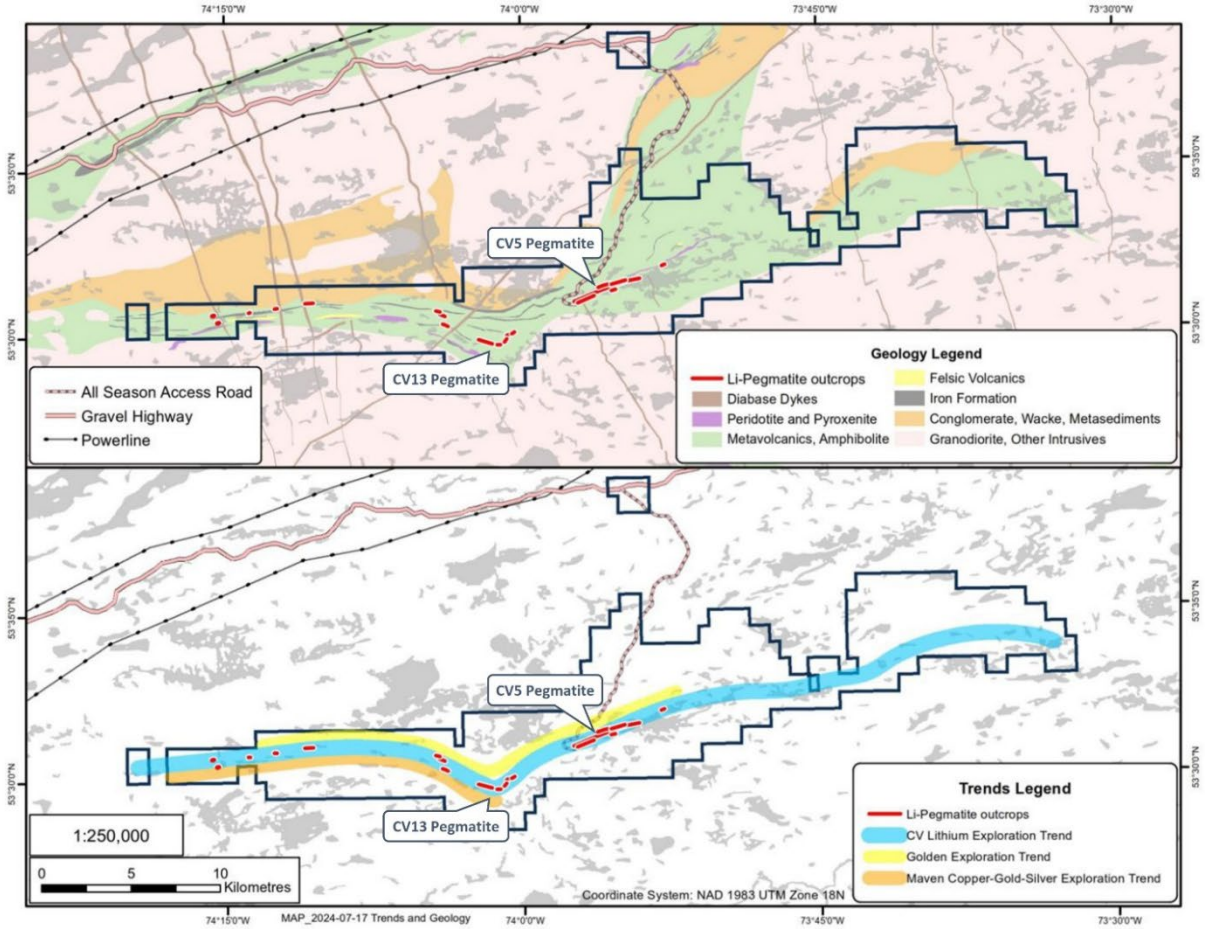
The Property overlies a large portion of the Lac Guyer Greenstone Belt, considered part of the larger La Grande River Greenstone Belt, and is dominated by volcanic and sedimentary rocks metamorphosed up to amphibolite facies (Figure 7.2). The Property's principal claim group is dominantly host to rocks of the Guyer Group (amphibolite, iron formation, intermediate to mafic volcanics, peridotite, pyroxenite, komatiite, as well as felsic volcanic tuffs). The amphibolite and metasedimentary rocks that trend east-west (generally moderately to steeply south-dipping) through this region are bordered to the north by the Magin Formation (conglomerate and wacke) and to the south by an assemblage of tonalite, granodiorite, and diorite, in addition to metasediments of the Marbot Group (conglomerate, wacke). Several regional-scale Proterozoic gabbroic dykes also cut through portions of the Property (Lac Spirt Dykes, Senneterre Dykes). The KCG claim block, located to the north of the principal claim group, is situated within the Bezier Suite (monzodiorite and granodiorite), and outside the Guyer Group.

The LCT pegmatites on the Property, including those at CV5 and CV13, are hosted predominantly within amphibolites, metasediments, and ultramafic rocks of the Guyer Group within the principal claim group.

The geological setting is primarily prospective for gold, silver, base metals, platinum group elements, lithium, caesium, and tantalum over several different deposit styles including orogenic gold (Au), volcanogenic massive sulphide (Cu, Au, Ag), komatiite-ultramafic (Au, Ag, PGE, Ni, Cu, Co), and LCT pegmatite. Additionally, in recent years, LCT pegmatites have been targeted as a potential primary or by-product source for Rb and Ga.

Exploration of the Property has outlined three (3) primary mineral exploration trends (Figure 7.2), crossing dominantly east-west over large portions of the Property's principal claim group – Golden Trend (gold), Maven Trend (copper, gold, silver), and CV Trend (LCT Pegmatite). The Golden Trend is focused on the northern areas of the Property, the Maven Trend in the southern areas, and the CV Trend "sandwiched" between. Historically, the Golden Trend has received the exploration focus followed by the Maven Trend. However, the identification of the CV Trend and the numerous lithium-tantalum pegmatites discovered to date represents a previously unknown LCT pegmatite district that was first recognized in 2016/2017 by Dahrouge Geological Consulting Ltd. and the Company.

Figure 7.2: Property Geology and Mineral Exploration Trends



The CV LCT Pegmatite Trend is currently recognized as an approximate 1-km wide and 25+ km long corridor, which is host to numerous distinct LCT pegmatite occurrences, and extends in a general east-west direction across the central portions of the Felix, FCI West, Deca-Goose, FCI East, and Corvette Main claim blocks. The trend is interpreted to extend across the majority of the principal claim group of the Property (~50 km); however, large areas remain to be explored for LCT pegmatite. The LCT pegmatites along this trend may outcrop as isolated high-relief 'whale-back' landforms or relatively low-relief to flat landforms (Figure 7.3, Figure 7.4, Figure 7.6, and Figure 7.7).

To date, nine (9) distinct lithium pegmatite clusters have been discovered along this trend at the Property: CV4, CV5, CV8, CV9, CV10, CV12, CV13, CV14 and CV15. Each of these clusters includes multiple lithium pegmatite outcrops in close proximity and oriented along the same local trend and has been grouped to simplify the exploration approach and discussion. Given the proximity of some lithium pegmatite outcrops to each other at these various clusters, as well as the shallow till cover, it is probable that some of the outcrops may reflect a discontinuous surface exposure of a single, larger pegmatite 'outcrop' subsurface.

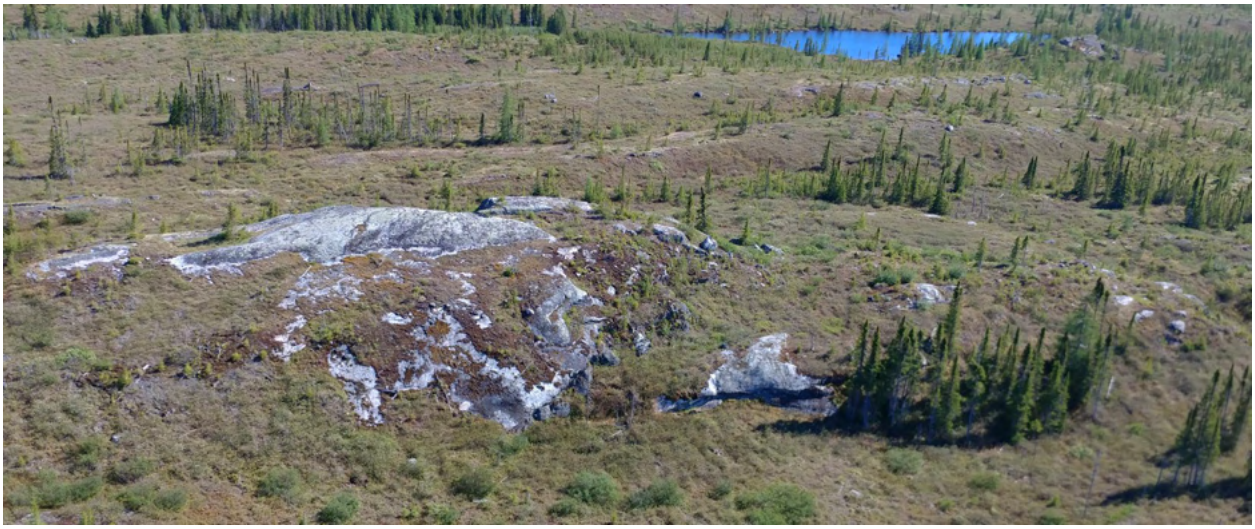
To date, the LCT mineralization discovered on the Property has been confined to the CV Trend. The Consolidated Mineral Resource Estimate (Consolidated MRE) for the Project, which includes the CV5 and CV13 pegmatites, represents the principal area of the trend. The CV5 and CV13 pegmatites are situated along the same geological trend, with strike lengths of approximately 4.6 km and 2.5 km, respectively, as defined by drilling to date. Both pegmatites remain open and are separated by a distance of roughly 2.6 km. The Consolidated MRE covers approximately 6.9 km of the ~7.1 km defined pegmatite trend, which also remains open along strike.

The local geology and mineralization of each known spodumene pegmatite cluster at the Property are further discussed in Section 7.4 (Mineralization).

Figure 7.3: 'Whale-Back' Spodumene Pegmatite Landform at CV13



Figure 7.4: 'Whale-Back' Spodumene Pegmatite Landform at CV5



7.3 Structural Geology

The Property overlies a large portion of the Lac Guyer Greenstone Belt, which is considered part of the larger La Grande Greenstone Belt within the La Grande sub-province and is dominated by volcanic rocks metamorphosed up to amphibolite facies. The La Grande sub-province underwent multiple tectonic deformation events, which are responsible for the formation of kilometre-scale thrust faults and folds within the volcano-sedimentary units and basement. The deformation included three (3) Archean episodes of ductile deformation and several Neoarchean to Paleoproterozoic episodes of brittle deformation (Goutier, et al., 2002); (Bandyayera, Burniaux, & Morfin, 2011); (Bandyayera, Burniaux, & Chapon, 2013).

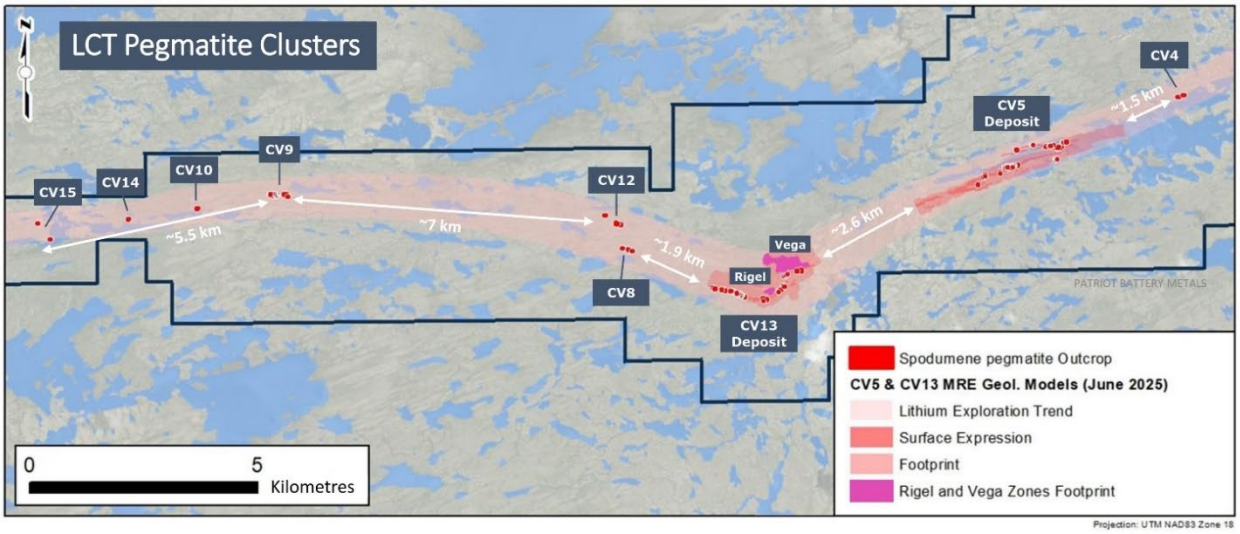
Within the Property, the Guyer Group domain is bound by two (2) shear zones which, in general, have an east-west extension – the Pontois-Sud Fault, a reverse shear zone in the north, and the Nochet Shear Zone in the south. The apparent fabric of the Guyer Group rocks is generally parallel to the shear zone contacts (west-east) and moderately to steeply south dipping (50-89 degrees). No major folds are known on the Property to this date, and only small-scale folding in outcrop and drilling has been documented.

The CV5 and CV13 pegmatites are non-concordant to the regional fabric and dip steeply northerly (CV5) or dip moderately northerly to flat-lying (CV13). The CV13 Pegmatite is coincident with a large-scale regional flexure and is evident in airborne magnetic data. The CV9 Pegmatite is currently interpreted to have a steep northerly dip and a moderate plunge to the east-southeast. There are no apparent indications that the CV5, CV13 and CV9 pegmatites have undergone any significant deformation. Through 2024, no other pegmatite clusters had been drill tested at the Property apart from a single, short hole at CV12.

7.4 Mineralization

The Property's geological setting is prospective for orogenic gold (Au), volcanogenic massive sulphide (Cu, Au, Ag), komatiite-ultramafic (Au, Ag, PGE, Ni, Cu, Co), and LCT pegmatite (Li, Cs, Ta). The following includes a discussion of the LCT pegmatite occurrences and mineralization at the Property (Figure 7.5). Additionally, in recent years, LCT pegmatites have been targeted as a potential primary or by-product source for Rb and Ga.

Figure 7.5: LCT Pegmatite Clusters at the Property



7.4.1 CV Trend (LCT Pegmatite)

The lithium-caesium-tantalum mineralization at the Property is observed to occur within quartz-feldspar LCT pegmatites, which may outcrop as high-relief 'whale-back' landforms as well as low-relief landforms. The pegmatite is often very coarse-grained and off-white in appearance, with darker sections commonly composed of smoky quartz (impure SiO_2), and occasional muscovite ($(KAl_2(AlSi_3O_{10})(F, OH)_2$) and tourmaline (dravite / schorl, $NaFe_3Al_6(BO_3)_3Si_6O_{18}(OH)_4$), and lighter sections composed of dominantly feldspars (albite and microcline, Na, K, $AlSi_3O_8$). Minor accessory and trace minerals may include beryl ($Be_3Al_2Si_6O_{18}$), chlorite ($(Fe, (Mg, Mn)_5,Al)(Si_3Al)O_{10}(OH)_8$), tantalite ($(Fe, Mn)(Ta, Nb)_2O_6$), lepidolite ($K(Li, Al)_3(Al, Si, Rb)_4O_{10}(F, OH)_2$), and apatite ($Ca_5(PO_4)_3(OH, F, Cl)$).

Lithium (Li)

Spodumene ($LiAlSi_2O_6$) is the dominant lithium mineral identified at all the lithium occurrences documented to date at the Property. Spodumene crystals range in size from centimetre-scale to metre-scale and have approached 2 m in length in drill core at CV5 and CV13. The colour of the spodumene crystals ranges from

cream to light grey-green over the CV5 and CV13 Pegmatite area, to a more whitish colour in the pegmatites to the west (CV8, CV9, CV10 and CV12). In rare cases, a purple variety (kunzite) has been identified at several clusters. Spodumene mineralization is commonly associated with smoky quartz and is most evident- in drill core (Figure 7.8, Figure 7.9, Figure 7.10, Figure 7.11, and Figure 7.19); however, it may still occur as isolated crystals in feldspar-rich pegmatite. Therefore, lithium (i.e., spodumene) content tends to be highest with higher contents of quartz and, correspondingly, lower with higher contents of feldspar. These two (2) mineral assemblages manifest as a 'high-grade' versus 'low-grade' zonation within the pegmatite at CV5.

Minor localized lepidolite ($K(Li, Al)_3(Al, Si, Rb)_4O_{10}(F, OH)_2$) and elbaite (a lithium tourmaline group mineral ($Na(Li, Al)_3Al_6(BO_3)_3Si_6O_{18}(OH)_4$) have been observed in a small number of lithium pegmatite outcrops as well as in drill holes. No significant occurrences of lithium phosphate minerals (Li_2PO_4) or petalite ($Li\ Al\ Si_4O_{10}$) have been documented to date in the pegmatites at the Property.

Variably altered spodumene, typically identified as cookeite ($LiAl_5Si_3O_{10}(OH)_8$), has been described occasionally in drill core, mostly commonly associated with fracture zones. Holmquistite – a lithium-bearing mineral of the amphibole group ($Li_2(Mg, Fe)_3(Al, Fe_3+)_2Si_8O_{22}(OH)_2$) – has been observed in the immediately adjacent host amphibolite, thus indicating a metasomatic replacement event involving lithium mobilized from the pegmatite syn. / post emplacement.

Caesium (Cs))

Of the nine (9) LCT pegmatite clusters identified at the Property to date, three (3) (CV5, CV13 and CV12) have documented pollucite ($(Cs, Na)_2Al_2Si_4O_{12}\cdot 2H_2O$) mineralization in drill hole and/or channel sampling. Pollucite is the optimal host mineral for caesium, which typically occupies up to 30% of the mineral crystal lattice. The occurrence of pollucite in LCT pegmatites is extremely rare and represents the most fractionated components of the system.

The Rigel and Vega caesium zones – nested entirely within the CV13 Pegmatite – are marked by significant occurrences of pollucite-hosted caesium (Figure 7.21 and Figure 7.22). The pollucite is typically centimetre- to decimetre-scale, presenting as clear to whitish-grey in colour with common late-stage veining of white pollucite or spodumene, or purple lepidolite, as well as common white flecks. It may be significantly more difficult to ascertain with the naked eye compared to spodumene. The pollucite also commonly occurs with significant amounts of spodumene (lithium) and tantalite (tantalum).

Mineralogy completed to date confirms that assays >0.50% Cs_2O are dominantly a result of the pollucite present. However, assays <0.50% Cs_2O have a higher probability of caesium-bearing mica minerals contributing materially to the caesium deportment.

Tantalum (Ta)

In addition to lithium and caesium, the pegmatites at Shaakichiuwaanaan typically carry a significant tantalum component, present in the form of tantalite. The tantalite is typically sub-millimetre to millimetre in scale and often very difficult to discern with the naked eye; however, it may occur in up to half-centimetre clusters with a medium black to dark brown colour.

As is common in LCT pegmatites, the tantalum mineralization is zoned within the wider pegmatite body and often overlaps with lithium and caesium mineralized zones. Further, very high-grade tantalum zones may also be coincident with very high-grade lithium and/or caesium zones.

Gallium (Ga)

The primary source of gallium today (>90%) is from bauxite, an aluminum-bearing mineral. The gallium may substitute for aluminum in these deposits and may be extracted from the spent liquor when concentrations have reached sufficient levels. However, in recent years, LCT pegmatites have been targeted as a potential new source of gallium, leveraging the same principle of substitution.

In LCT pegmatites, the gallium also substitutes for aluminum in feldspar ((Ca, Na, K) (Al, Si) $AlSi_2O_8$) and spodumene ($LiAlSi_2O_6$) minerals. During mineral processing for lithium in LCT pegmatites, a spodumene concentrate is produced, which may also host gallium at potentially higher grades than the pegmatite as a whole. Therefore, there is a potential opportunity for recovery of gallium post dissolution of the spodumene when the gallium is then also present in solution.

Rubidium (Rb)

Geochemically, the LCT pegmatites at the Property commonly contain elevated levels of rubidium compared to less differentiated granites, confirming their strongly differentiated signature – typical for LCT pegmatites. Whole rock rubidium contents are commonly 1,000 ppm to 4,000 ppm Rb and not uncommonly may exceed 1%. Most of the rubidium in the LCT pegmatites at Shaakichiuwaanaan occurs from substitution of potassium in feldspar, substitution of potassium or caesium in mica (e.g., lepidolite), and substitution of caesium in pollucite. Rubidium is commercially recovered from pollucite and lepidolite; however, recovery from feldspar has not yet been commercialized.

Given the significant pollucite mineralization at the Rigel and Vega zones, these two (2) areas are considered of strong interest for rubidium. There are no current Mineral Resources for rubidium defined at the Property.

7.4.1.1 CV5 Pegmatite

The CV5 Pegmatite is the largest single occurrence of LCT pegmatite at the Property identified to date. It is located centrally to the Property, approximately 13 km south of KM-270 on the Trans-Taiga Road (Figure 5.1). It had been delineated to within approximately 1.5 km of the CV4 Spodumene Pegmatite cluster to the east, and to within approximately 2.6 km of the CV13 Spodumene Pegmatite cluster to the west (Figure 7.5).

At surface, CV5 is exposed as a series of discontinuous spodumene pegmatite outcrops spanning a corridor of approximately 2.25 km long x 0.5 km wide. Outcrops range in size from ~1-3 m in size to ~175 m long x ~15 m to 30 m wide (CV1 outcrop) and ~220 m long x 20 m to 40 m wide (CV5 outcrop) (Figure 7.6 and Figure 7.7). Spodumene mineralization at CV5 is comprised of decimetre- to metre-scale crystals, typically off-white to grey in appearance (Figure 7.8 through Figure 7.11).

Figure 7.6: Main Outcrop at CV5 Pegmatite (Looking Westerly)



Figure 7.7: Main Outcrop at the CV5 Pegmatite (Looking Northerly)



Figure 7.8: Spodumene Crystal at the CV5 Pegmatite



Figure 7.9: Strongly Fractured, Pinkish Weathered Spodumene Crystals in Matrix of White Feldspar and Grey Quartz at the CV5 Pegmatite



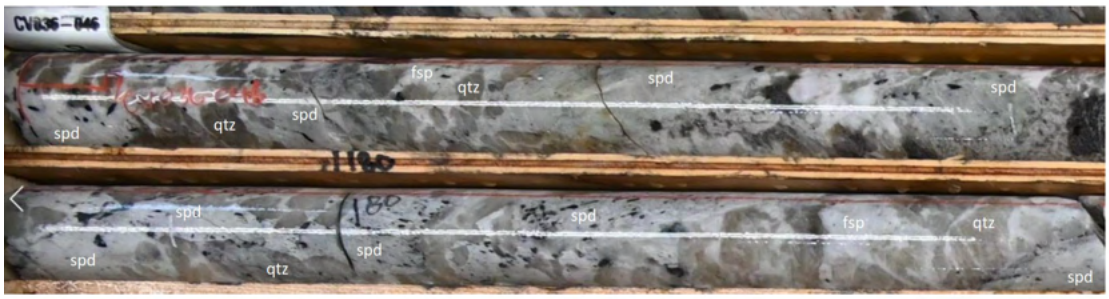
Figure 7.10: Coarse-grained Spodumene Mineralization in Drill Holes CV22-035 and 036



Blow-up of green box below illustrating coarse grained spodumene crystals

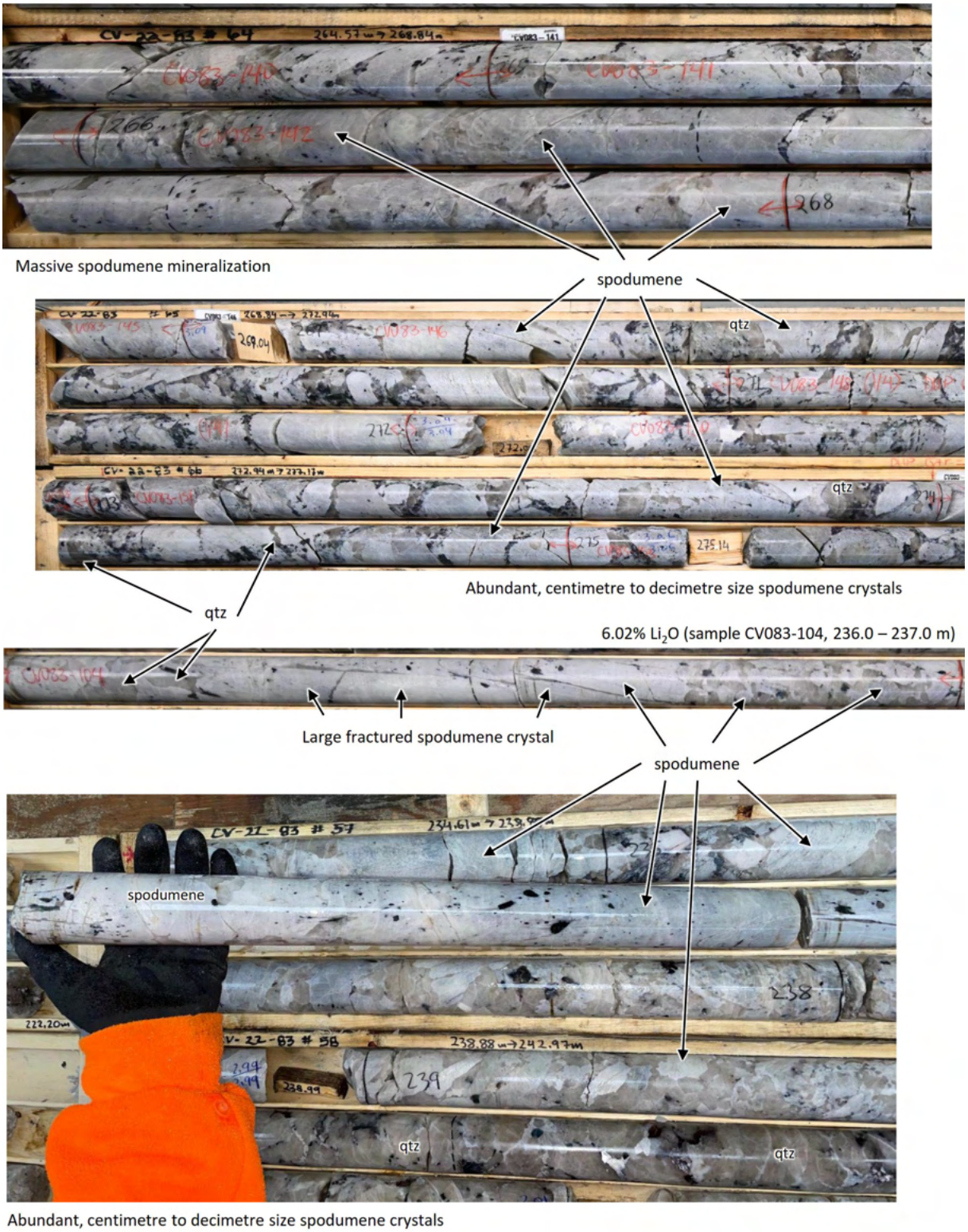


High-grade lithium mineralized drill intersection in CV22-035 – **3.29% Li_2O** and 177 ppm Ta_2O_5 **over 10.0 m** (202.5 m to 212.5 m – red box) within a wider zone of **1.25% Li_2O** and 118 ppm Ta_2O_5 **over 96.9 m** (126.1 m to 223.0 m)



Course grained spodumene (spd) in quartz (qtz) feldspar (fsp) pegmatite in drill hole CV22-036

Figure 7.11: Coarse-grained Spodumene Mineralization from Nova Zone in Drill Hole CV22-083



A portion of the known CV5 Pegmatite is situated beneath an unnamed shallow glacial lake (labelled Lake 001 in this Feasibility Study). This lake is typically <3-10 m deep with a maximum lake depth of ~18 m to 20 m in the very eastern areas of CV5. Standard geological interpretation in greenstone belts dictates that pegmatites should not be expected to be present under topographic lows (e.g., lakes). This is because they are resistive to chemical erosion by nature of their mineralogy and igneous formation and, therefore, should preferentially form topographic highs relative to their host amphibolite, metasediment, and ultramafic rocks. However, the Company's exploration approach interprets a process by which the coarse grain size and well-developed cleavage of spodumene (and to a lesser extent feldspars) offer small cracks that may be exploited by overlying glaciers to fracture at a larger scale and 'pluck' out and move large pegmatite blocks as the glacier advances. The result is a pegmatite topographic low, which was later infilled with water as the glacier receded, leaving behind what we find today at CV5. This interpretation is supported by several kilometre-long dispersion trains of up to car-sized pegmatite boulders in the down-ice direction.

To date, at the CV5 Pegmatite, multiple individual spodumene pegmatite dykes have been geologically modelled. However, a vast majority of the CV5 Mineral Resource is hosted within a single, large, principal spodumene pegmatite dyke, which is flanked on both sides by multiple, subordinate, sub-parallel trending dykes (Figure 7.12, Figure 7.13 and Figure 7.14). The CV5 Pegmatite, including the principal dyke, is modelled to extend continuously over a lateral distance of at least 4.6 km and remains open along strike at both ends and to depth along a large portion of its length. The width of the currently known mineralized corridor at CV5 is approximately ~500 m, with spodumene pegmatite intersected at depths of more than 450 m in some locations (vertical depth from surface). The pegmatite dykes at CV5 trend west-southwest (approximately 250°/070° using the right-hand rule), and therefore dip northerly, unlike the host amphibolites, metasediments, and ultramafic rocks, which dip moderately in a southerly direction.

The principal spodumene pegmatite dyke at CV5 ranges from <10 m to more than 125 m in true width, and may pinch and swell aggressively along strike, as well as up and down dip. It is primarily the thickest at near-surface to moderate depths (<225 m), forming a relatively bulbous, elongated shape, which may flare to surface and to depth variably along its length – see simplified geological cross-sections in Figure 7.15, Figure 7.16, and Figure 7.17, as well as those presented in the June 2023 MRE (McCracken & Cunningham, 2023) for additional context. As drilling has focused on the principal dyke, the immediate CV5 corridor has not been adequately drill tested, and it is interpreted that additional subordinate pegmatite lenses are situated proximal, especially in the south-central areas of the deposit. The pegmatites that define CV5 are relatively undeformed and very competent, although they likely have some meaningful structural control with several structures and faulting documented.

The CV5 Spodumene Pegmatite displays internal fractionation along strike and up / down dip, which is evidenced by variation in mineral abundance, including felspar, quartz, spodumene, and tantalite. This is

highlighted by the high-grade Nova Zone, which has been traced over a strike length of at least 1.1 km and includes multiple drill intersections ranging from 2 m to 25 m (core length) at >5% Li_2O, within a significantly wider mineralized zone of >2% Li_2O (Figure 7.11 and Figure 7.14).

Figure 7.12: Plan View of CV5 Pegmatite Geological Model – All Lenses

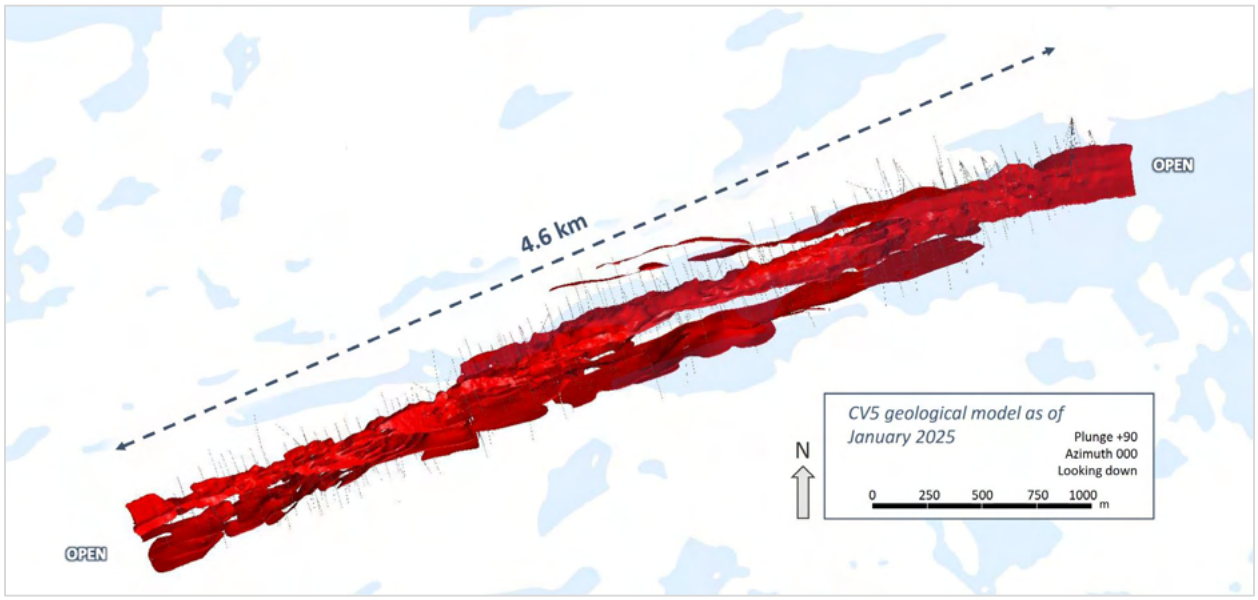


Figure 7.13: Inclined View of CV5 Pegmatite Geological Model Looking Down Dip (70°) – All Lenses

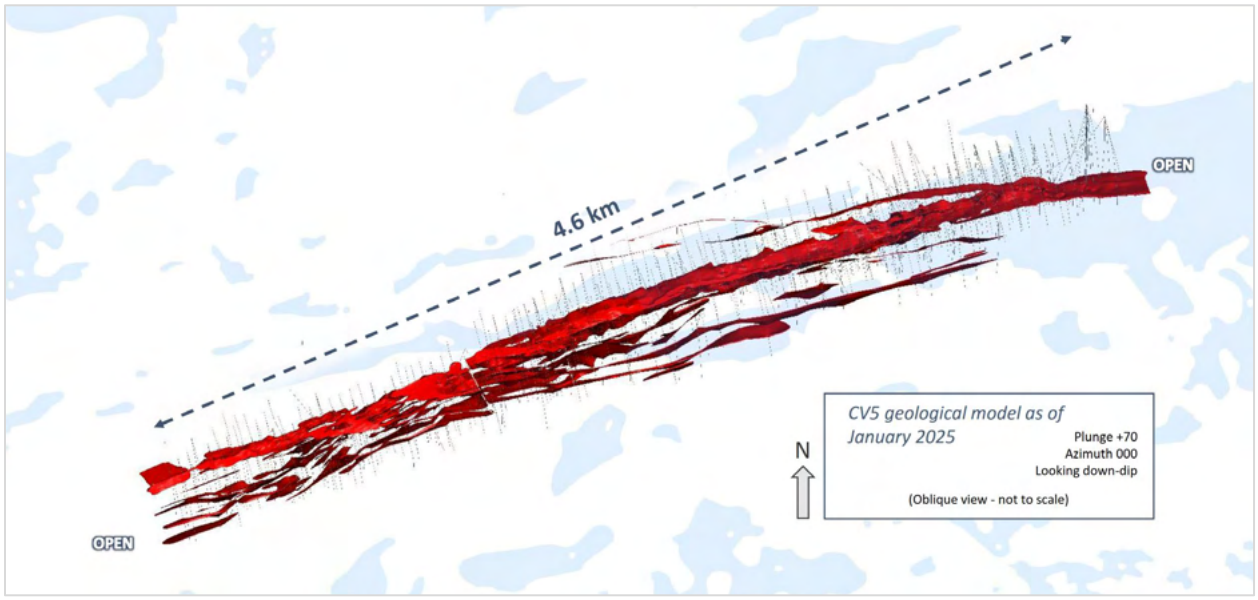
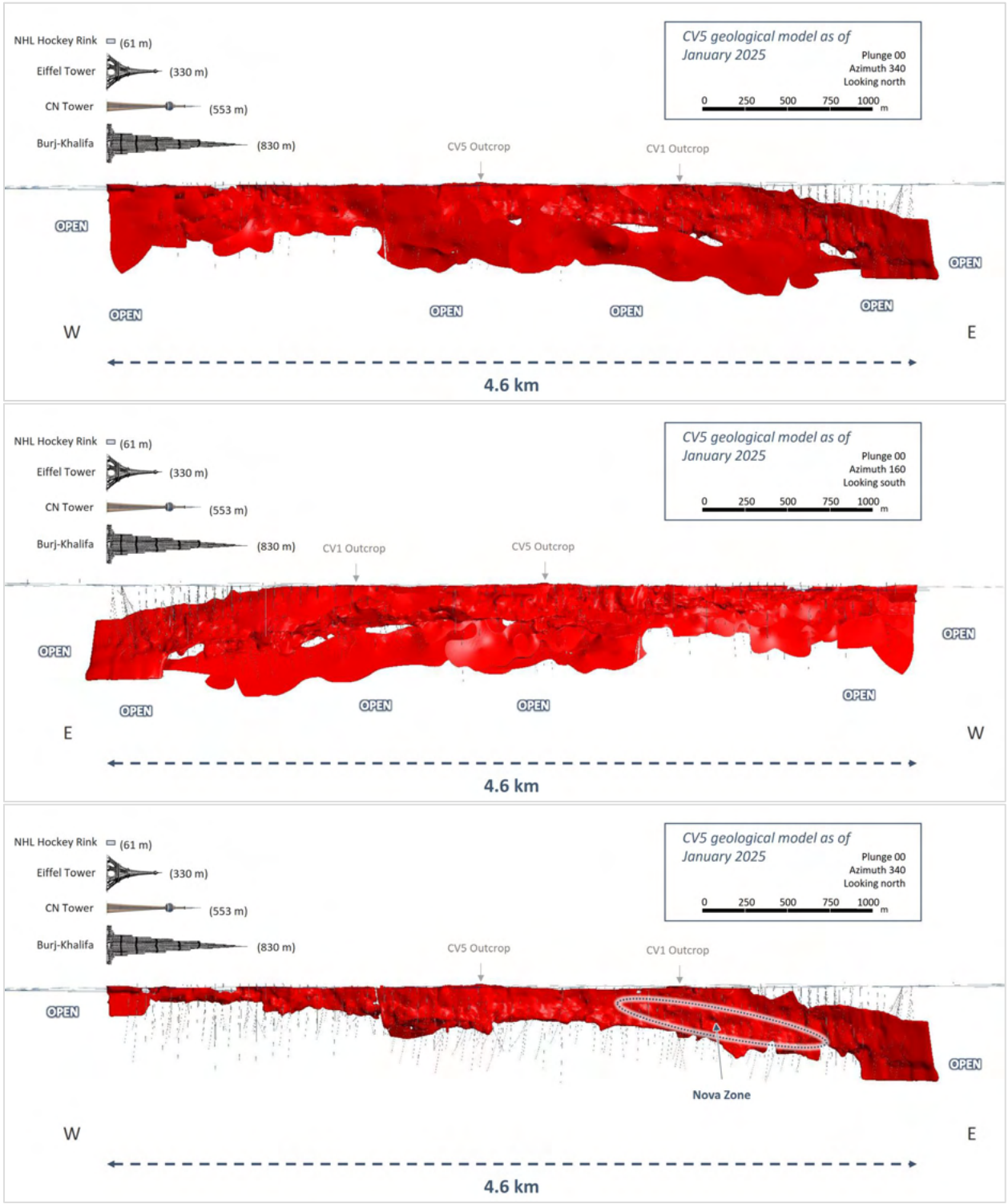


Figure 7.14: Side View of CV5 Geological Model



**Note: Looking north (340°), all lenses (top); looking south (160°), all lenses (middle); looking north (340°), principal pegmatite only (bottom).*

Figure 7.15: Simplified Cross-Section of CV5 Pegmatite Geological Model

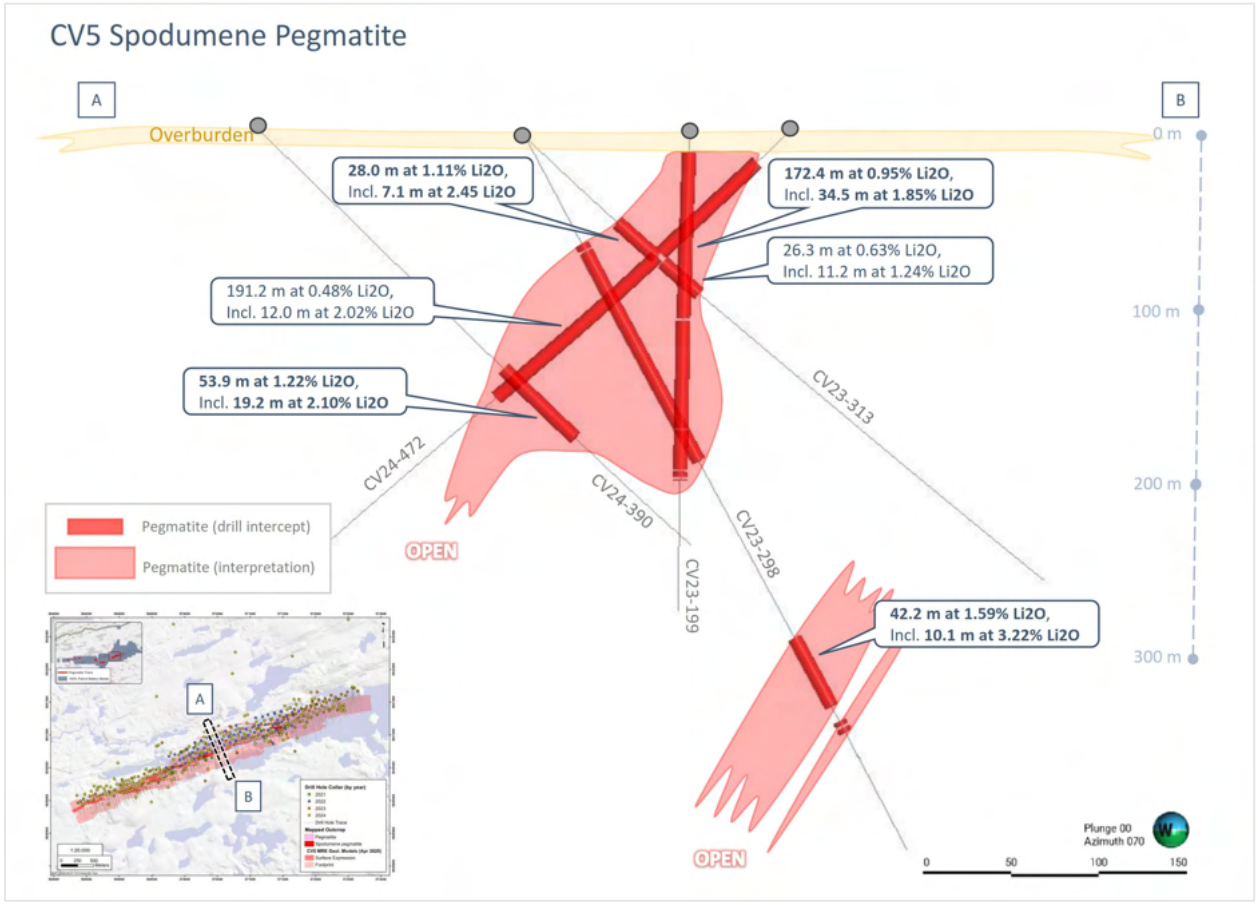


Figure 7.16: Simplified Cross-Section of CV5 Pegmatite Geological Model

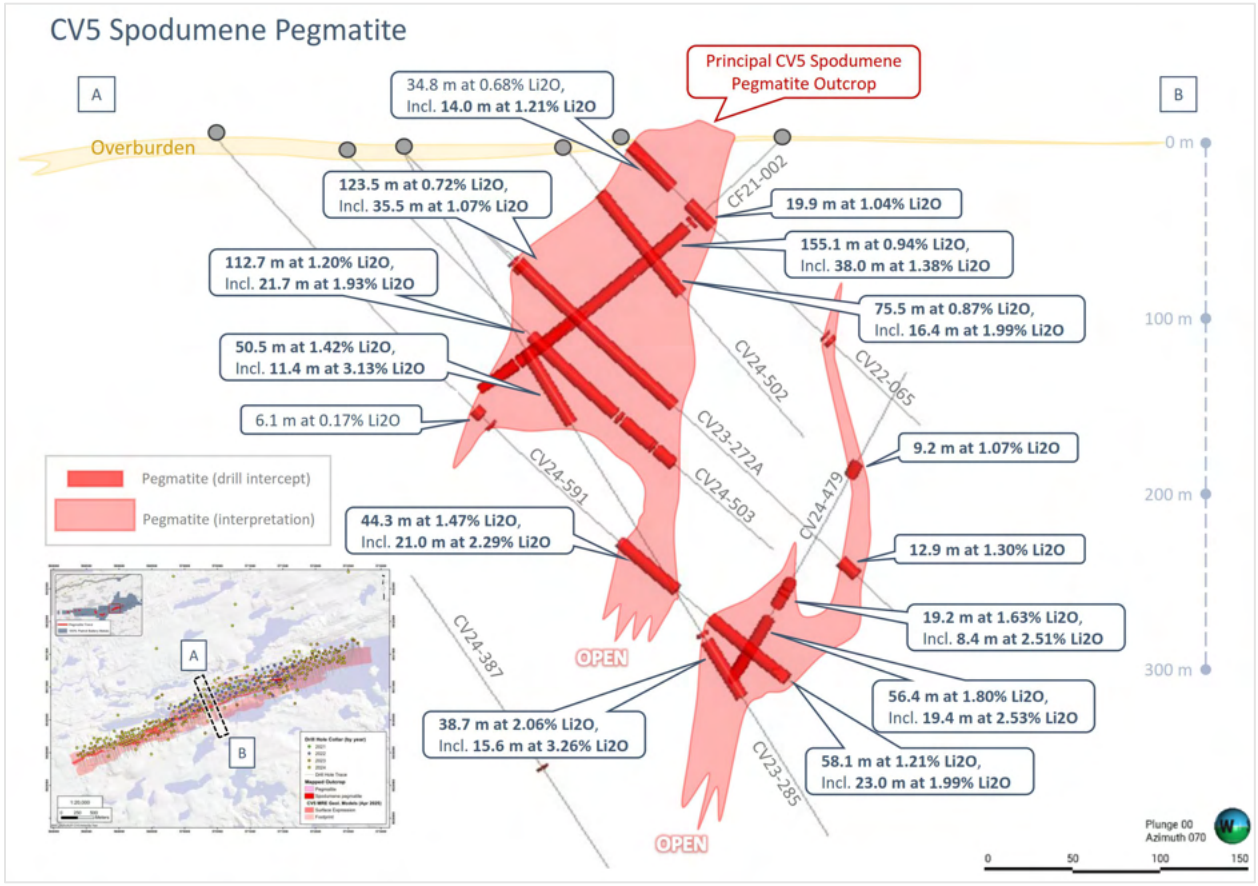
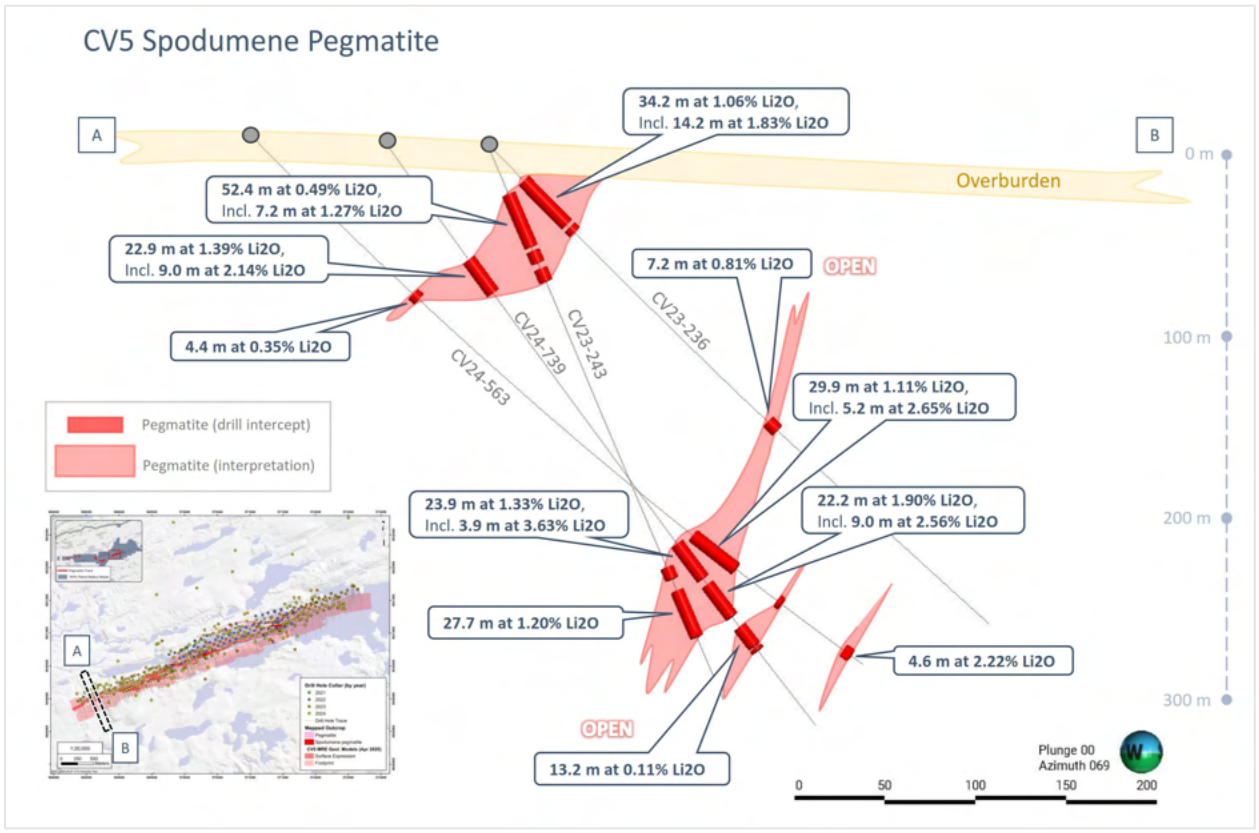


Figure 7.17: Simplified Cross-Section of CV5 Pegmatite Geological Model



7.4.1.2 CV13 Pegmatite

The CV13 Pegmatite, discovered in 2022, is located near the centre of the Property at the apex of an interpreted regional structural flexure. It is situated approximately 2.6 km from the CV5 Pegmatite to the northeast and approximately 1.9 km and 2.4 km, respectively, from the CV8 and CV12 Spodumene Pegmatites to the northwest (Figure 7.5).

The two (2) largest outcrops are approximately 70 m long by 12 m wide and 100 m long by 15 m wide, situated approximately 300 m apart, with the largest outcrop coincident with the apex of the regional structural flexure (Figure 7.18). The pegmatite outcrops define two (2) contiguous trends, totalling approximately 2.6 km in combined strike length. The pegmatite contacts are poorly exposed on the northern and southern edges, although, where exposed, they are often in contact dominantly with amphibolite, followed by ultramafic (undifferentiated), and/or wacke lithologies of the Guyer Group.

At the CV13 Pegmatite, surface mapping and drilling completed to date interpret a series of flat-lying to moderately dipping (northerly), sub-parallel trending LCT pegmatite bodies, of which three (3) appear to dominate (Figure 7.20). The pegmatite bodies are coincident with the apex of a regional structural flexure,

whereby the pegmatite manifests a west arm trending ~290° and an east arm trending ~230°. Drilling to date indicates the east arm includes significantly more pegmatite stacking compared to the west and also carries a significant amount of the overall CV13 Pegmatite tonnage and lithium grade. The Rigel and Vega caesium zones are nested entirely within the CV13 Pegmatite and are marked by significant occurrences of pollucite.

The CV13 Pegmatite ranges in true thickness from <5 m to more than 40 m and extends continuously over a collective strike length of approximately 2.6 km, along its west and east arms. The CV13 Pegmatite, which includes all proximal pegmatite lenses, remains open along strike at both ends and to depth along a significant portion of its length. Spodumene mineralization has been traced more than 400 m down-dip; however, due to the typically shallow dips in the pegmatite bodies, it is only ~200 m vertical depth from the surface.

Spodumene at CV13 is commonly centimetre- to decimetre-scale with rare metre-size crystals, the crystals becoming most evident on freshly cut faces (Figure 7.19). Variable grain sizes are observed in several outcrops. The spodumene is generally white to light-grey with common light-green, weakly chlorite-altered crystals.

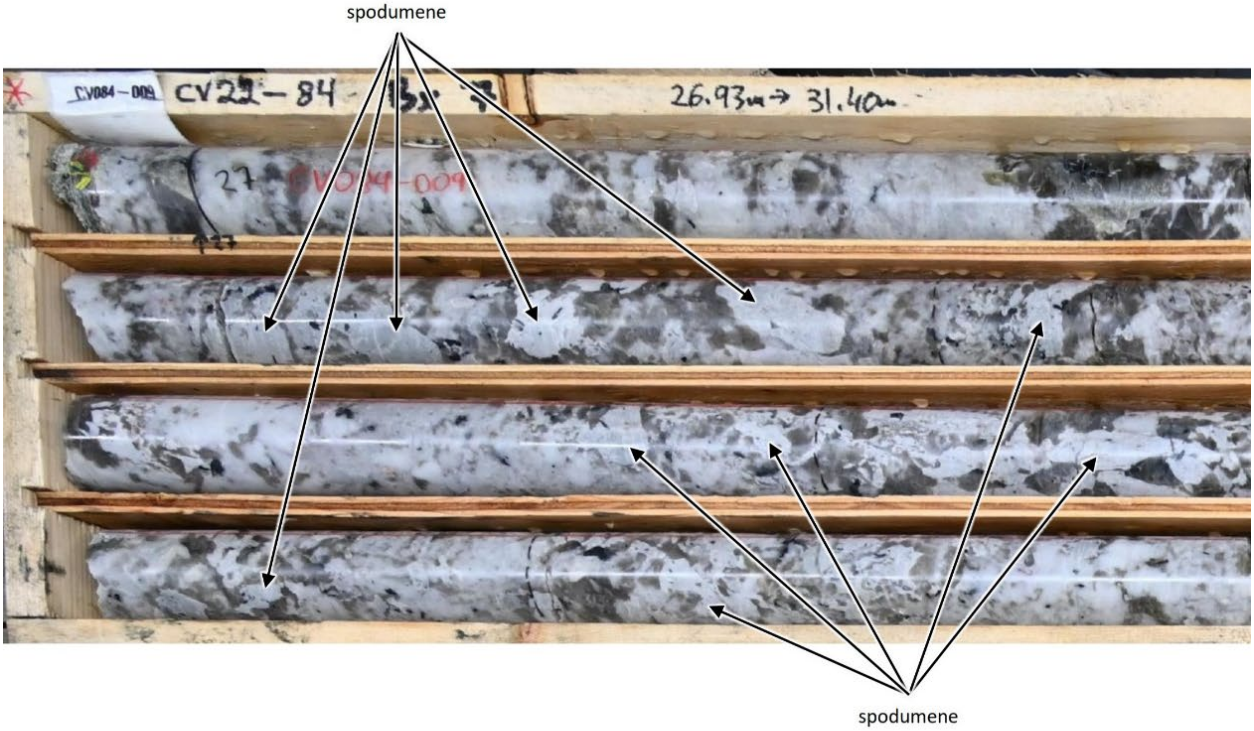
The CV13 Pegmatite displays internal fractionation along strike and up / down dip, similar to CV5. This is highlighted at CV5 by the high-grade (lithium) Nova Zone and at CV13 by the high-grade (lithium) Vega Zone, each situated at the base of their respective pegmatite lenses, and traced over a significant distance with multiple drill hole intercepts (core length) ranging from 2 m to 25 m (CV5) and 2 m to 10 m (CV13) at >5% Li_2O, respectively, each within a significantly wider mineralized zone of >2% Li_2O. The Vega Zone is situated approximately 6 km south-west and along the geological trend of the Nova Zone. Both zones share several similarities, including lithium grades and very coarse decimetre to metre-sized spodumene crystals. However, both pegmatite zones have distinct orientations, whereby the Vega Zone is relatively flat-lying to shallow dipping, covering an area of approximately 380 m by 220 m, while the Nova Zone is steeply dipping to vertical with a strike length of at least 1.1 km.

Based on the local geological trends as supported by geophysics, the CV13 Spodumene Pegmatite is interpreted to be part of a much larger LCT pegmatite system at the Property, potentially extending from the most easterly identified CV4 Cluster, and continuing westerly through the CV5 and CV8/12 clusters. Collectively, this area of the CV Lithium Pegmatite trend extends nearly 15 km, of which 6.9 km is confirmed by drilling to be continuous spodumene pegmatite hosting defined Mineral Resources. The scale of LCT pegmatite present along this trend suggests a deeply rooted and common ‘plumbing’ system and source of the lithium mineralized bodies discovered to date.

Figure 7.18: Aerial View of the Spodumene Pegmatite Outcrop at CV13 (Looking Northeasterly)

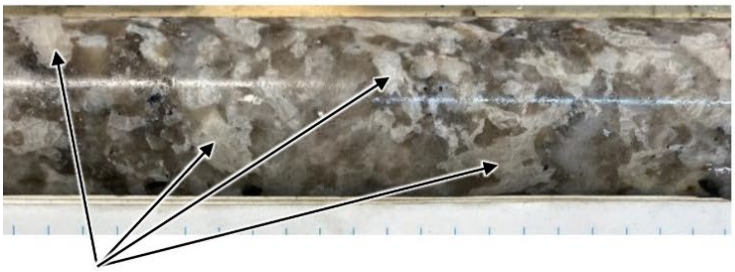


Figure 7.19: Coarse-grained Spodumene Crystals in Drill Core from CV13





Spodumene crystal at 138.5 m in CV22-084

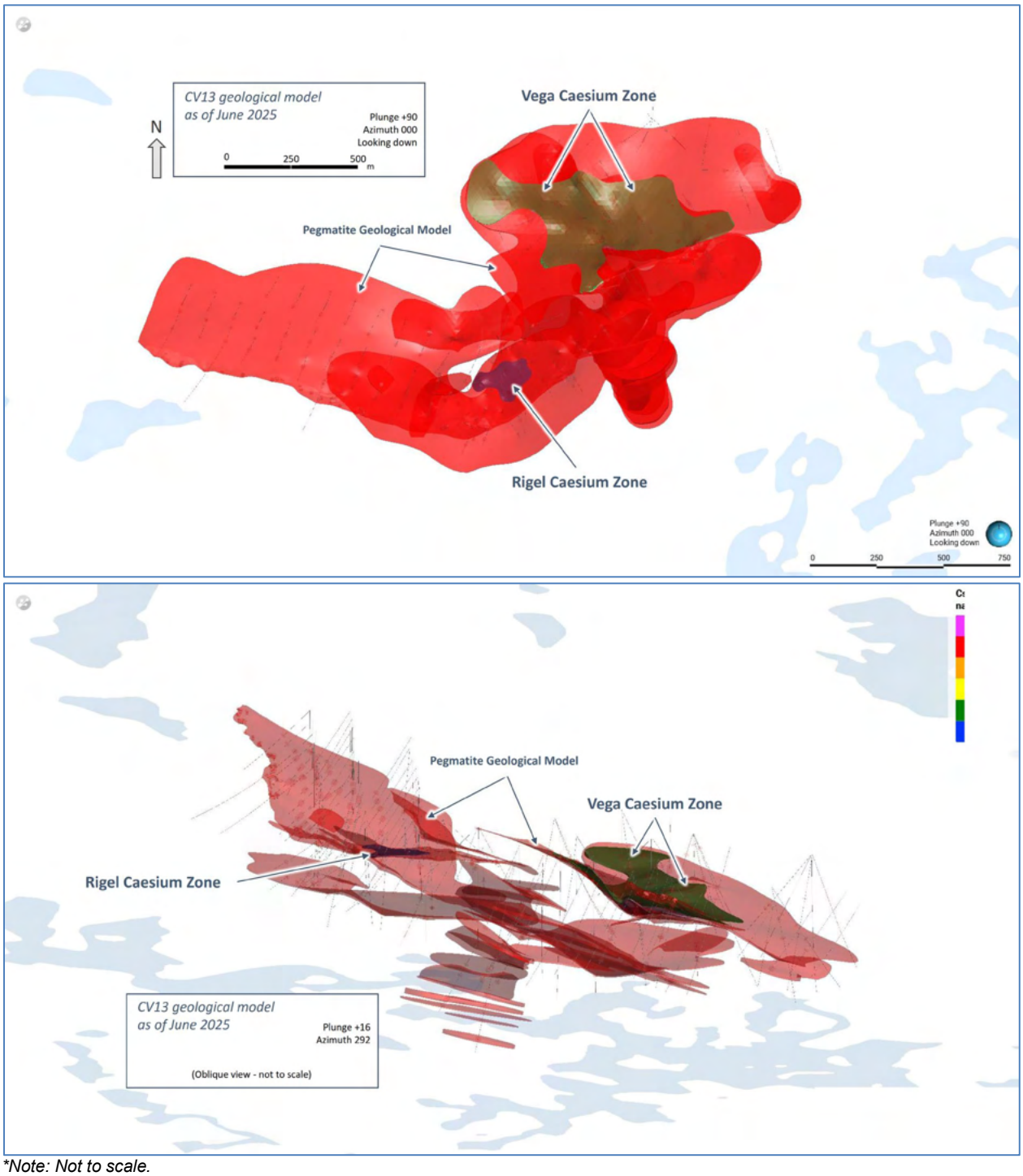


Spodumene crystals at 29.5 m in CV22-104



Massive spodumene crystal at 31.3 m in CV22-092

**Figure 7.20: Plan View of CV13 Pegmatite Geological – All Lenses (top);
Oblique View of CV13 Pegmatite Geological Model – All Lenses (bottom)**



**Note: Not to scale.*

7.4.1.2.1 Rigel and Vega Caesium Zones

The Rigel and Vega caesium zones – nested entirely within the CV13 Pegmatite – are marked by significant occurrences of pollucite-hosted caesium. These two (2) zones have been geologically sub-modelled using a 0.50% Cs_2O grade constraint. The grade constraint is supported by mineralogical analysis that confirms pollucite as the predominant caesium-bearing mineral above this threshold. These geological sub-models acted as hard boundaries within the wider CV13 Pegmatite body.

Using the 0.5% Cs_2O grade constraint, the footprint of caesium mineralization at Rigel has been traced over a general area of at least 200 m x 100 m and consists of a single, shallow-dipping lens at a depth of ~50 m with a true thickness of <2 m to ~6 m. At the Vega Zone, the footprint of the caesium mineralization has been traced over a general area of at least 800 m x 250 m and consists of two (2) proximal flat-lying lenses, at a depth of ~110 m, with a true thickness of <2 m and up to ~10 m and ~6 m, respectively.

The pollucite is typically centimetre- to decimetre-scale, presenting as clear to whitish-grey in colour with common late-stage veining of white pollucite or spodumene, or purple lepidolite, as well as common white flecks (Figure 7.21 and Figure 7.22). The pollucite also commonly occurs with significant amounts of spodumene (lithium) and tantalite (tantalum).

Simplified cross-sections of the Rigel and Vega caesium zones are presented in Figure 7.23 and Figure 7.24.

Figure 7.21: Pollucite Mineralization in High-Grade Caesium Drill Intersection at ~64.5 m Depth in Drill Hole CV23-271 at the Rigel Zone, CV13 Pegmatite



**Note: Interval grades 22.69% Cs2O over 1.0 m (64.0 m to 65.0 m) with XRD-Rietveld reporting a pollucite content over the interval of 58%.*

Figure 7.22: Pollucite with Lepidolite Veining (purple) in Grey Quartz Matrix from 139.3 m to 139.5 m in Drill Hole CV24-520 (Vega Zone), Within a Wider Zone of Caesium Mineralization Grading 7.39% Cs_2O over 7.1 m



Figure 7.23: Simplified Cross-section of the Rigel Caesium Zone Geological Model at the CV13 Pegmatite

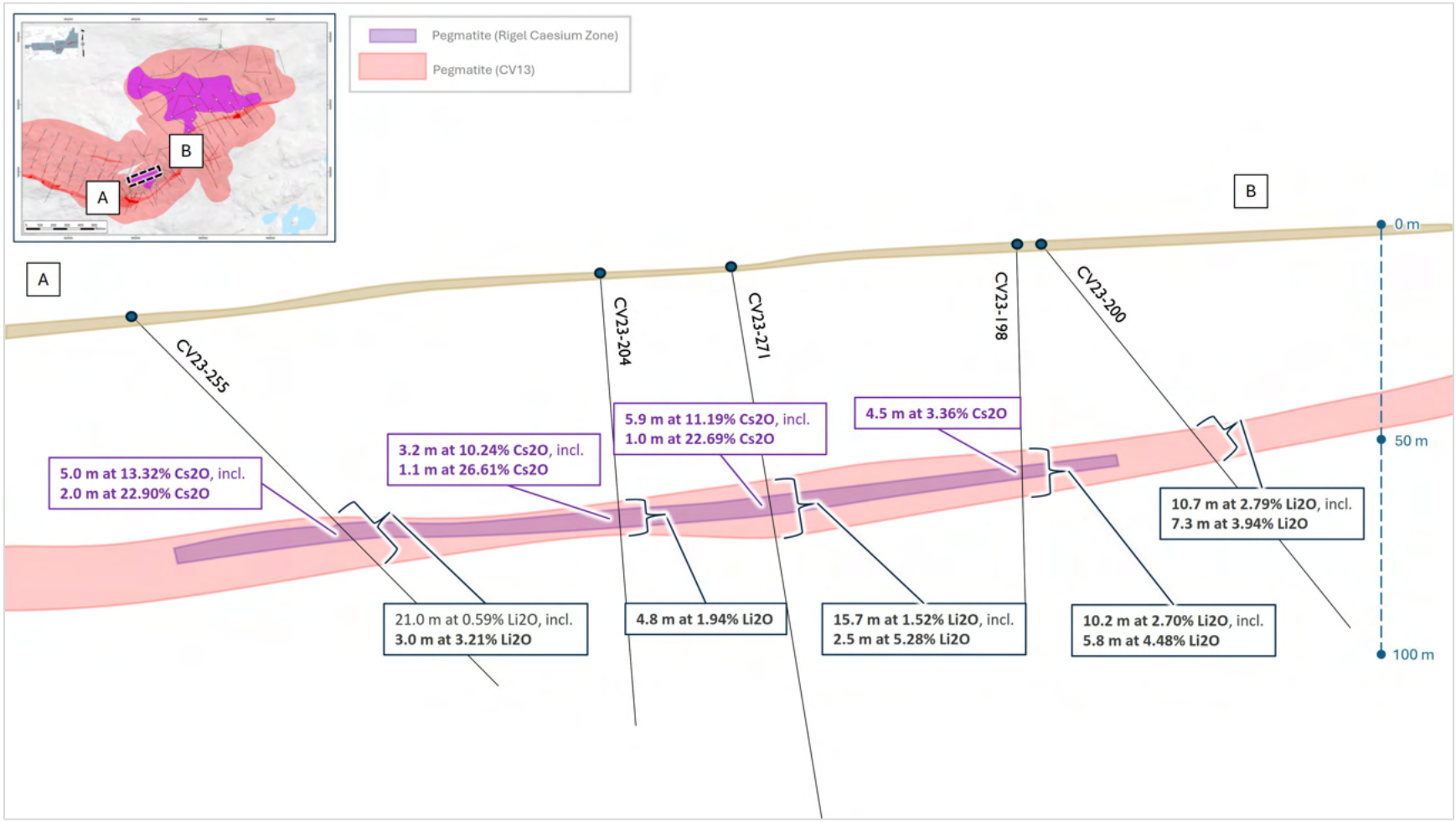
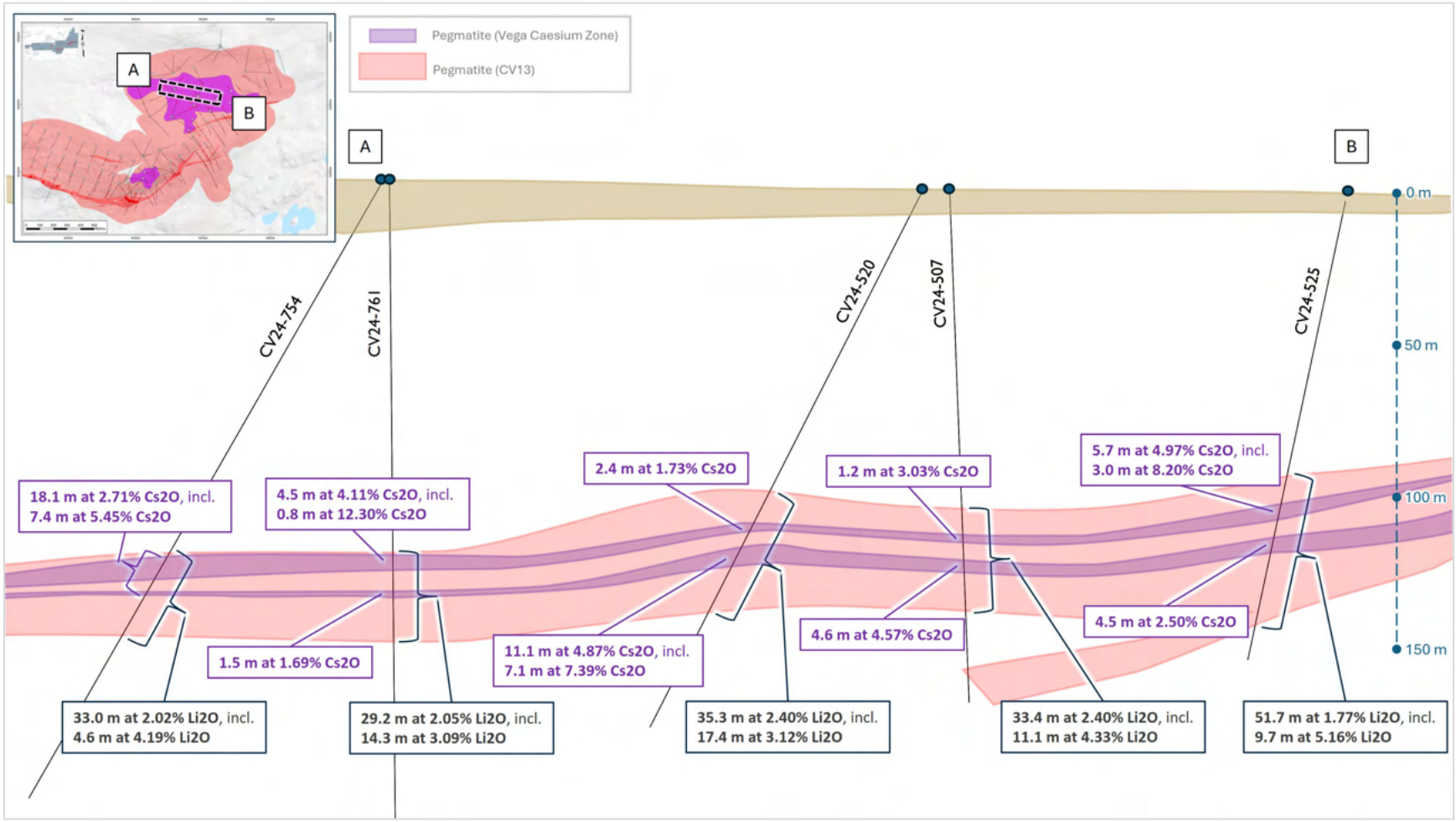


Figure 7.24: Simplified Cross-section of the Vega Caesium Zone Geological Model at the CV13 Pegmatite



7.4.1.3 Other LCT Pegmatites

In addition to the CV5 and CV13 pegmatites, seven (7) other notable LCT pegmatite clusters have been documented on the Property through 2024 – CV4, CV8, CV9, CV10, CV12, CV14, and CV15 (Figure 7.5). To date, the Shaakichiuwaanaan Consolidated MRE includes only the CV5 and CV13 pegmatites.

Each of the other seven (7) pegmatite clusters is characterized by multiple spodumene pegmatite outcrop occurrences with analysis of grab samples returning at least 0.90% Li_2O. The pegmatites are similar in nature to CV5 and CV13 in terms of bulk mineral assemblage, including coarse-grained spodumene crystals at typically centimetre- to decimetre-scale. These pegmatite clusters are also commonly anomalous to well-mineralized in tantalum. Pollucite has also been described, and confirmed by assay, at the CV12 Pegmatite. Further description of these other seven pegmatite occurrences is presented in the previous Technical Report for the Property (McCracken, et al., 2024) and in previous news disclosures by the Company.

7.4.2 Maven Trend (Copper, Gold, Silver)

The Maven Copper-Gold-Silver Exploration Trend is an approximately 10+ km long corridor, which hosts numerous Cu-Au-Ag showings and extends in a general east-west direction across the southern portion of the FCI West claim block and onto the Shaakichiuwaanaan claim block.

Mineralization at Maven consists of quartz-sulphide lenses / veins / veinlets hosted within mafic / amphibolite rock types or silicate iron formation, as well as interpreted epigenetic remobilization of mineralization within shear zones. Minor occurrences of ultramafic rocks are also documented. At the showings, semi-massive to disseminated sulphides are dominated by pyrrhotite and chalcopyrite, with common quartz and variable to absent pyrite and/or sphalerite. The nature of the mineralization has been broadly interpreted to be associated with the volcanogenic massive sulphide style (i.e., exhalative in nature).

7.4.3 Golden Trend (Gold)

The Golden Trend is an approximate 10+ km long corridor, which hosts several Au showings and extends in a general east-west direction across the northern portion of the FCI West claim block and Deca-Goose claim block. The primary mineral occurrence on the trend is the Golden Gap Prospect, which has returned 3 g/t to 108 g/t Au in outcrop and 10.48 g/t Au over 7 m in a drill hole. Mineralization along the trend occurs dominantly in quartz-rich gossanous units.

8. DEPOSIT TYPES

The primary target and deposit model for the Shaakichiuwaanaan Property are LCT pegmatites (Figure 8.1). These generally have granitic or alaskitic compositions. Major constituent minerals are quartz, feldspar (albite, orthoclase, microcline) along with lesser amounts of muscovite and lithium-bearing minerals, such as spodumene. Mafic minerals are generally minor constituents, including biotite, tourmaline, garnet, or cordierite. Oxide and sulphide minerals are rare. These pegmatites are often coarse-grained, frequently with finer-grained, sometimes graphitic margins. Other elements sometimes associated with lithium include caesium, tantalum, beryllium, phosphorus, and rare earths (Cerny & Ercit, 2005). Lithium-bearing minerals are most commonly spodumene, petalite, and lepidolite. Tantalum-bearing minerals include pyrochlore and columbite-tantalite. Caesium-bearing minerals include lepidolite and pollucite where fractionation is most extreme.

Most LCT pegmatites are hosted by metamorphosed supracrustal rocks in the upper greenschist to lower amphibolite metamorphic grades. LCT pegmatite intrusions generally are emplaced late during orogeny, with emplacement being controlled by pre-existing structures. Typically, they are located near evolved, peraluminous granites (i.e., S-Type) and leucogranites from which they are inferred to be derived by fractional crystallization. In cases where a parental granite pluton is not exposed, one is inferred to lie at depth. These pegmatite melts are enriched in fluxing components including water (H_2O), fluorine (F), phosphorus (P) and boron (B), which depress the solidus temperature, lower the density, and increase rates of ionic diffusion. This enables pegmatites to form thin dikes and massive crystals despite having a felsic composition and temperatures that are significantly lower than ordinary granitic melts. LCT pegmatites crystallize at low temperatures between about 350°C and 550°C, and in a very short time from days to years (Bradley, McCauley, & Stillings, 2017).

LCT pegmatites are major sources lithium production and accounted for roughly 58% of the world's lithium production in 2022 (Bird, 2023). The balance of lithium production is from brines, predominantly in Chile. Total global lithium production for all sources in 2022 was estimated at between 0.69 million tonnes (Mt) to 0.77 Mt of lithium carbonate equivalent (LCE) (Bird, 2023). In 2025, estimated global lithium supply from primary sources is forecasted to be 1.16 Mt LCE, of which ~74% will be from hard rock deposits and ~26% from brines (Benchmark Minerals, 2025).

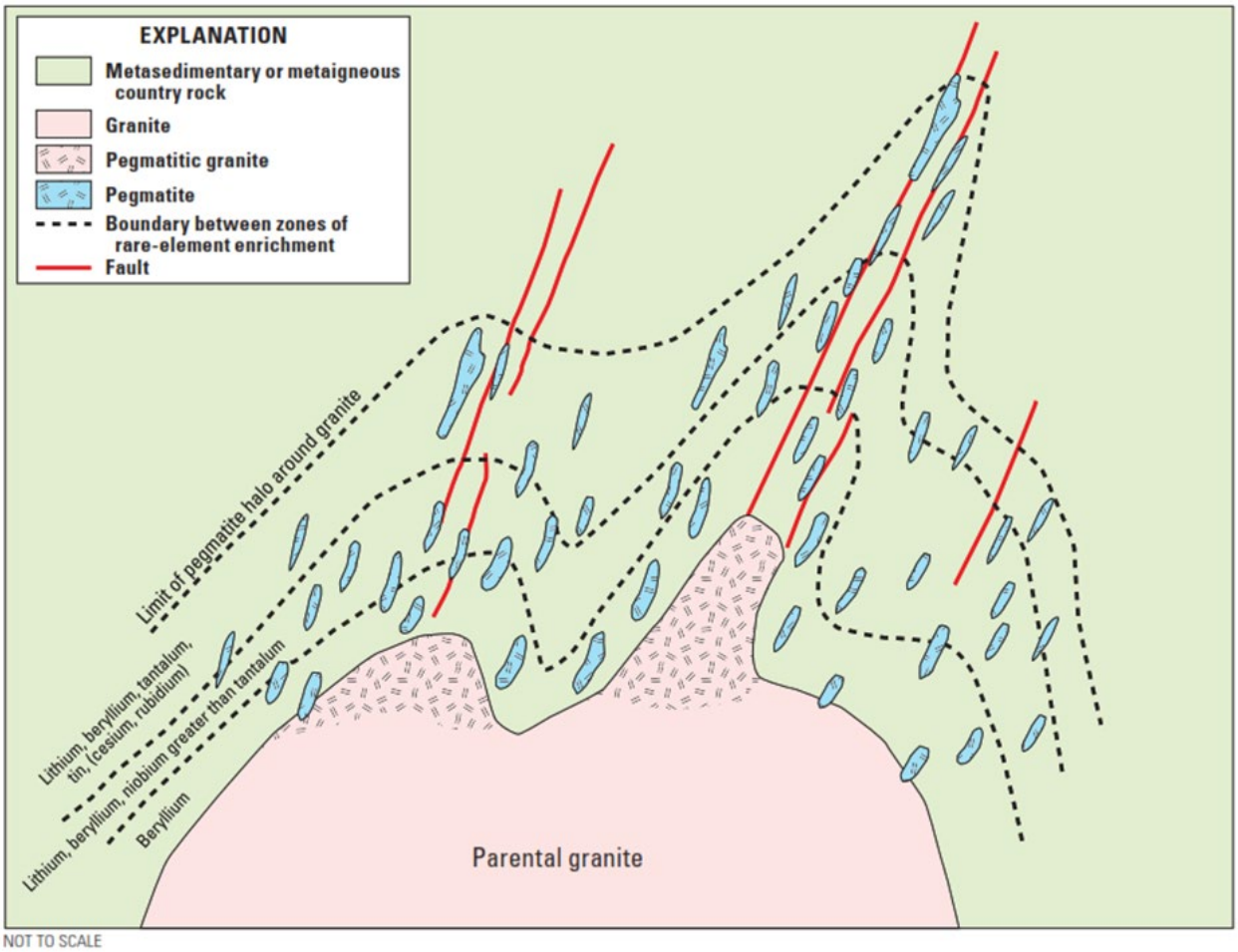
Australia, with multiple producing LCT pegmatites, dominates global lithium production accounting for roughly 80% of the spodumene market. Some of the largest LCT pegmatite deposits globally include Greenbushes, Pilgangoora, and Wodgina in Western Australia, Goulamina in Mali, and Manono in the Democratic Republic of Congo. Where present in high-enough concentrations, tantalum is commonly recovered as a by-product of lithium pegmatite operations.

Mineral deposits of pollucite-hosted caesium pegmatite are extremely rare globally and represent the most fractionated component of LCT pegmatite systems and are effectively the only primary economic source of caesium globally. Economic deposits of pollucite-hosted caesium pegmatite are typically on a smaller scale of <10 kt to 350 kt in size compared to deposits of lithium pegmatite that typically range in the millions of tonnes in size (<10 Mt and rarely over 100 Mt). Globally, it is estimated only three (3) mines have historically produced pollucite: Tanco (Canada), Bikita (Zimbabwe), and Sinclair (Australia). Caesium is also recovered as a secondary product from lepidolite in pegmatite mining operations in Asia.

Depending on the size and attitude of the pegmatite, a variety of mining techniques are used, including artisanal surface mining, open pit surface mining, small underground workings, and large underground operations using room-and-pillar design. In favourable circumstances, what would otherwise be gangue minerals (quartz, potassium feldspar, albite, and muscovite) can be mined along with lithium and/or tantalum as co-products (Bradley, McCauley, & Stillings, 2017).

Exploration and assessment of LCT pegmatites rely on a number of considerations. In remote areas, such as the James Bay region, where exploration has been historically minimal, the key criteria are an orogenic hinterland setting, appropriate regional metamorphic grades, and the presence of evolved granites and common granitic pegmatites. New LCT pegmatites are most likely to be found near known deposits. Pegmatites tend to show a regional mineralogical and geochemical zoning pattern with respect to the inferred parental granite, with the greatest enrichment in more distal pegmatites. Mineral-chemical trends in common pegmatites that can point toward an evolved LCT pegmatite include increasing rubidium in potassium feldspar, increasing lithium in white mica, increasing manganese in garnet, and increasing tantalum and manganese in columbite-tantalite. Most LCT pegmatite bodies show a distinctive internal zonation featuring four zones: border, wall, intermediate (where lithium, caesium, and tantalum are generally concentrated), and core. This zonation is expressed both in cross-section and map view; therefore, what may appear to be a common pegmatite may instead be the edge of a mineralized body (Bradley, McCauley, & Stillings, 2017).

Figure 8.1: LCT Pegmatite Deposit Model



Source: Bradley, McCauley, & Stillings, 2017.

9. EXPLORATION

The Company's non-drilling exploration activities at the Shaakichiuwaanaan Property include surface mapping and rock sampling, prospecting, channel sampling, ground and airborne geophysics, and remote sensing surveys. The focus has been predominantly on LCT pegmatite, although significant base and precious metal exploration has also been completed. A summary of these activities follows and is presented in Figure 9.1, Figure 9.2, and Figure 9.3, with further detail presented in 2025 (McCracken & Cunningham, 2025). All drill exploration activities completed by the Company are presented in Chapter 10 (Drilling).

The Qualified Person (QP) notes that prospecting surface rock samples (grab / chip) and associated assays, as discussed herein, are selective by nature and represent a point location and, therefore, are not necessarily representative of the mineralized horizon sampled.

9.1 2017 Through 2020

In 2017, the Company completed a short reconnaissance program, collecting three (3) surface grab samples from the outcrop described by Virginia Mines, and confirmed the presence of coarse-grained spodumene in two (2) sub-parallel trending pegmatites – the 'CV1' outcrop (described historically by Virginia as hosting "*cristaux de spodumène*), and the 'CV2' outcrop. The sampling returned 0.80% and 3.48% Li_2O, and 1.22% Li_2O from the CV1 and CV2 pegmatites, respectively, along with anomalous tantalum (Smith D. L., 2018 - GM70744). The CV1 and CV2 outcrop form part of the present CV5 LCT Pegmatite.

The Company expanded upon the work in 2018 with additional surface prospecting and rock sampling, which resulted in the discovery of new LCT pegmatite outcrops, including the CV4 LCT Pegmatite cluster – grab sample assays of 1.61% Li_2O (CV5 area) and 0.74% Li_2O (CV4), respectively (Smith D. L., 2019) (Smith D. L., 2019). In addition, channel sampling was completed across the CV1 and CV2 pegmatite outcrops at the CV5 Pegmatite. Highlights from the channel sampling include 2.28% Li_2O and 208 ppm Ta_2O_5 over 6 m (sample CV1-CH03) and 1.54% Li_2O and 136 ppm Ta_2O_5 over 8 m (sample CV1-CH01).

In July 2019, the Company expanded its scope of exploration with a stronger focus on base and precious metals with lithium (± tantalum) of secondary interest due to the declining market environment for those commodities at the time. The field work included prospecting of historical base and precious metal showings and prospects (e.g., Golden Gap, Lac Bruno, Tyrone T9, etc.) as well as completion of a soil sampling grid extending northeast of the Lac Bruno boulder field (Smith D. L., 2021 - GM72176).

A total of 680 rock samples and 211 soil samples were collected during the 2019 program and resulted in the discovery of new occurrences of gold (West Golden Gap, New Lac Bruno), copper-gold-silver (Elsass, Lorraine, Black Forrest, Hund), and lithium-tantalum (pegmatite outcrops CV5 through CV11), as well as further understanding of known targets (Smith D. L., 2020 - GM71564), (Smith D. L., 2021 - GM72176) (Smith D. L., 2020 - GM71564), (Smith D. L., 2021 - GM72176), and (Smith D. L., 2019 - GM71513)). Sample results ranged from nil to 11.9 g/t Au, nil to 171 ppm Ag, nil to 8.15% Cu, nil to 4.72 Li_2O, and nil to 1,011 ppm Ta_2O_5. Sample assay highlights of outcrop occurrences include: 3.63% Cu, 0.64 g/t Au, and 52.3 g/t Ag (Elsass), 8.15% Cu, 1.33 g/t Au, and 171 g/t Ag (Lorraine), 3.28% Cu, 0.78 g/t Au, and 30.1 g/t Ag (Hund), 1.13% Cu, 0.05 g/t Au, and 19.5 g/t Ag (Black Forrest), 2.81 g/t Au (West Golden Gap), 1.4 g/t Au (New Lac Bruno), 0.68% Cu, 0.11 g/t Au, and 5.3 g/t Ag (Lac Farley), 4.06% Li_2O and 564 ppm Ta_2O_5 (CV5 Pegmatite), 4.44% Li_2O and 205 ppm Ta_2O_5 (CV8 Pegmatite), 4.72% Li_2O (CV9 Pegmatite), and 1.33% Li_2O and 255 ppm Ta_2O_5 (CV10 Pegmatite).

The exploration completed by the Company between 2017 and 2019 outlined three primary exploration trends, crossing roughly east-west over large portions of the Property – the Maven Trend (copper, gold, silver), Golden Trend (gold), and CV LCT Pegmatite Trend (lithium, caesium, tantalum) (see Figure 7.2). A detailed review and discussion through 2019 of the individual mineral occurrences that comprise the Maven, Golden, and CV exploration trends is presented in (Smith D. L., 2021 - GM72176) and (Smith D. L., 2019 - GM71513).

No field work was completed in 2020 by the Company; however, desktop work was advanced and included a reinterpretation (by Dynamic Discovery Geoscience and Dahrouge Geological Consulting) of historical induced polarization and resistivity surveys (IP-Resistivity) and airborne magnetic survey data. A major finding of the work indicates that the majority of the follow-up drill holes to test the historical 10.5 g/t Au over 7 m drill intercept at the Golden Gap Prospect did not test the mineralized zone's potential strike extension to the east and rather is interpreted to have followed a secondary trend (Gaia Metals Corp., 2020). Therefore, the data indicates significant potential for follow-up drilling at Golden Gap remains.

9.2 2021

Exploration continued in 2021 and focused on the Maven Trend and the CV Trend ahead of initial diamond drilling, which followed in the fall. Airborne and surface work included geological mapping and rock sampling, ground-based induced-polarization and resistivity survey, airborne magnetic survey, and a remote sensing survey (Smith, Mickelson, & Blu, 2023 - GM73402). The geophysical and remote sensing surveys were used to further develop both LCT pegmatite and base and precious metal targets at the Property.

During the surface sampling program, a total of 164 grab / chip samples were collected across the Property, predominantly on the FCI West claim block. The most significant result of the 2021 mapping and rock sampling program was the recognition of the CV12 LCT Pegmatite cluster, where numerous lithium pegmatite outcrops were discovered. LCT pegmatite at CV12 was initially discovered in 2019 and characterized by one (1) sample that graded 0.27% Li_2O; however, this was significantly expanded upon during the 2021 follow-up. A total of 11 grab samples were collected in 2021 from the CV12 Pegmatite and associated trend analytical results ranging from nil to 5.98% Li_2O and 49 ppm to 1,478 ppm Ta_2O_5, with an average of 2.83% Li_2O and 438 ppm Ta_2O_5.

In addition, two (2) lithium-tantalum mineralized boulder samples were discovered east-southeast of the CV12 and CV8 pegmatites with grab samples assays of 2.69% Li_2O and 198 ppm Ta_2O_5, and 2.20% Li_2O and 265 ppm Ta_2O_5, respectively. Based on glacial ice movement in the region, the discovery indicates additional yet to be discovered pegmatite outcrop is present to the northeast, and on strike with the Company's Deca-Goose claim block.

Prospecting along the Maven Trend, completed to refining initial drill targets, returned multiple samples consistent with area showings. Six (6) samples were collected, exceeding 1% Cu to a high of 3.53% Cu, 3.15 g/t Au, and 46.4 g/t Ag from a chalcopyrite-quartz amphibolite at the Tyrone-T9 Showing.

9.3 2022

Based on the successful lithium pegmatite exploration in 2021, the 2022 exploration campaign reoriented firmly towards LCT pegmatite (i.e., lithium) with only minor base and precious metals work completed. Exploration included prospecting and rock sampling, surface outcrop mapping, channel sampling, and a light detection and ranging (LiDAR) and orthophoto survey (Smith, Schmidt, Mickelsen, & Ullrich, 2024 - GM73931).

In August 2022, Group PHB (Perron, Hudon, Belanger Inc.) completed a LiDAR and digital photogrammetric (orthophoto) survey over the entirety of the Shaakichiuwaanaan Property. The survey would serve as tight topographical control for future geological modelling based on drill hole data, as well as generate a significant number of targets for ground truthing.

A total of 236 surface rock samples were collected over the course of the 2022 program, and more than 70 spodumene pegmatite outcrops were mapped across the Property. Minor sampling was completed along the Maven Trend as well as along the Golden Trend and focused on confirmation sampling of historical showings situated on the recently acquired Deca-Goose and Felix claim blocks. Assay results were generally in line with historical sampling.

A large focus of the 2022 surface exploration was on mapping and prospecting of the local trends at the various CV LCT pegmatite clusters that had been identified to date at the Property – CV4, CV5, CV8, CV9, CV10 and CV12. Multiple new LCT pegmatite outcrops were identified with grab / chip sampling returning results in line with previous sampling. Outcrop channel sampling was also completed and returned 1.5 m at 1.12% Li_2O (CV4), 5.6 m at 1.93% Li_2O (CV8), 15.0 m at 0.46% Li_2O (CV9), and 21.9 m at 0.80% Li_2O; 7.7 m at 1.46% Li_2O; 10.1 m at 1.09% Li_2O; 1.1 m at 3.24% Cs_2O; and 3.3 m at 1.58% Cs_2O (CV12).

The most significant result of the 2022 surface exploration was the discovery of the CV13 LCT Pegmatite cluster, situated between the CV8 and CV12, and CV5 LCT Pegmatite clusters. The CV13 Pegmatite cluster is characterized by two (2) contiguous trends of spodumene pegmatite outcrops, totalling more than 2 km in combined strike length, situated within the apex of a regional structural flexure. A total of 38 pegmatite surface grab / chip samples were collected at the cluster, of which 14 assayed >1% Li_2O to a peak of 3.73% Li_2O. Outcrop channel sampling followed with results including 14.2 m at 1.17% Li_2O (CH22-025/026), 13.1 m at 1.57% Li_2O (CH22017), and 10.5 m at 1.53% Li_2O (CH22-018/19).

9.4 2023

Surface exploration in 2023 included an orientation IP-Resistivity geophysical survey over a large portion of the CV5 Pegmatite, a ground magnetic survey over the CV5 to CV13 corridor, a ground gravity orientation survey, as well as geological mapping and rock sampling, prospecting, and channel sampling. Additionally, an airborne magnetic and radiometric survey was completed over the Corvette Main, FCI East, and Felix claim blocks (Smith, Delporte, & Mickelsen, 2025 - GM74460).

The results of the ground orientation IP-Resistivity and gravity surveys were inconclusive with respect to identifying the principal pegmatite body at CV5; however, the methods may have merit in identifying certain geological contacts as well as further defining the local pegmatite trend. The airborne magnetic survey was flown at the sample spacing as the prior surveys (50 m, helicopter) and merged with the prior datasets. The radiometric data, which was collected alongside the magnetic data, has thus far proven to be inconclusive in terms of vectoring for lithium pegmatite at the Property.

Over the summer-fall period, a surface exploration program was completed and included detailed geological mapping at the CV5 Pegmatite, channel sampling at the CV13 Pegmatite, and prospecting and rock sampling over regional areas of the Property. The program was impacted by the regional forest fires over the period, which prevented access to the Property for a significant amount of the field season.

A total of 474 surface rock grab / chip samples were collected over the course of the 2023 surface program. The most significant result of the 2023 surface exploration was the discovery of the CV14 LCT Pegmatite

cluster, situated approximately 1.5 km along the geological trend of the CV10 LCT Pegmatite. Two (2) grab samples assayed 0.94% Li_2O and 0.86% Li_2O, with the primary outcrop approximately 33 m by 9 m in size. The discovery highlighted an approximate 3.6 km long prospective trend extending from CV9 through CV10 to CV14.

Outcrop channel sampling was also completed in 2023 at the CV13 LCT Pegmatite. A total of 147 m of channel samples were collected, with results including 13.4 m at 1.22% Li_2O; 6.4 m at 1.44% Li_2O; and 5.4 m at 1.93% Li_2O.

9.5 2024

In 2024, non-drill-related exploration by the Company included a surface exploration program of detailed geological mapping at the CV5 and CV13 pegmatites, channel sampling at multiple LCT pegmatite clusters, and regional prospecting. A LiDAR and orthophoto survey, and heliborne magnetic and radiometric survey were also completed in 2024 over the JBN-57 claim block. With the completion of these surveys, the Company now has complete magnetic, LiDAR, and orthophoto data coverage over the entire Property.

A total of 647 surface rock grab / chip samples were collected over the course of the 2024 surface program. The most significant result of the 2024 surface exploration was the discovery of the CV15 LCT Pegmatite cluster, situated approximately 1.9 km southwest and along geological trend from CV14, and collectively outlines a larger ~5.5 km long prospective trend extending from the CV9 LCT Pegmatite cluster to CV15, now referred to as the Mickel Trend (Figure 7.5). The CV15 discovery consists of multiple pegmatite outcrops spread over an approximate 400 m x 200 m area, with the largest measuring ~7 m x 6 m in size and remains open in all directions. Outcrop grab sample assays include 2.11% Li_2O, 1.55% Li_2O, and 1.02% Li_2O. Additionally, grab sample assays from nearby boulders returned grades of 3.10% Li_2O and 3.02% Li_2O.

In addition to the CV15 discovery, the Company also discovered a new LCT pegmatite outcrop ~525 m along strike from CV8 with sample assays including 2,282 ppm Ta_2O_5, significantly extending the local prospective LCT pegmatite trend to nearly 800 m. The 2024 program was also successful in outlining several other new targets based on anomalous to mineralized (Li-Ta) outcrops and boulders.

9.6 2025

Through 2025 to the Issue Date of this report, non-drill related exploration by the Company included a surface exploration program of detailed geological mapping at the CV5 and CV13 pegmatites, channel sampling at multiple LCT pegmatite clusters, and regional prospecting. Additionally, a seismic survey was

completed over the CV5 area to further constrain overburden thickness and bedrock topography. The surface exploration program began in June 2025, and no results have been reported to date by the Company.

9.7 Lithium Pegmatite Surface Sampling Summary

9.7.1 CV5 Pegmatite

CV5 was the first LCT pegmatite to be sampled at the Property. Approximately 60 grab / chip rock samples of pegmatite outcrop have been collected at CV5 through 2024, with results ranging from below detection limit (bdl) to 7.32% Li_2O, bdl to 2,490 ppm Ta_2O_5, and bdl to 0.18% Cs_2O. Channel sampling, collectively totalling 179.4 m, was completed in 2018, 2022 and 2024, with highlights including 23.6 m at 1.06% Li_2O, 11.0 m at 1.36% Li_2O, and 8.0 m at 1.54% Li_2O.

9.7.2 CV13 Pegmatite

Approximately 44 grab / chip rock samples of pegmatite outcrop have been collected at CV13 through 2024, with results ranging from bdl to 3.73% Li_2O, bdl to 1,016 ppm Ta_2O_5, and bdl to ≥1.06% Cs_2O (upper detection limit of analytical package). Channel sampling, collectively totalling 424.8 m, was completed in 2022, 2023 and 2024, with highlights including 15.0 m at 1.19% Li_2O, 14.2 m at 1.17% Li_2O, 13.1 m at 1.57% Li_2O, 13.4 m at 1.22% Li_2O, and 10.5 m at 1.53% Li_2O. Additionally, high-grade caesium was returned in channel sampling – 3.0 m at 9.43% Cs_2O, including 1.0 m at 22.41% Cs_2O.

9.7.3 CV4 Pegmatite

Approximately seven (7) grab / chip rock samples of pegmatite outcrop have been collected at CV4 through 2024, with results ranging from bdl to 2.00% Li_2O, 63 ppm to 548 ppm Ta_2O_5, and bdl to 0.06% Cs_2O. Channel sampling, collectively totalling 4.4 m, was completed in 2022 with results of 1.5 m at 1.12% Li_2O and 1.9 m at 0.58% Li_2O.

9.7.4 CV8 Pegmatite

Approximately six (6) grab / chip rock samples of pegmatite outcrop have been collected at CV8 through 2024, with results ranging from 0.01% to 6.72% Li_2O, 6 ppm to 397 ppm Ta_2O_5, and bdl to 0.07% Cs_2O. Channel sampling, collectively totalling 8.4 m, was completed in 2022 with results of 5.6 m at 1.93% Li_2O and 2.8 m at 1.74% Li_2O.

9.7.5 CV9 Pegmatite

Approximately 29 grab / chip rock samples of pegmatite outcrop have been collected at CV9 through 2024, with results ranging from bdl to 4.71% Li_2O, 15 ppm to 401 ppm Ta_2O_5, and bdl to 0.08% Cs_2O. Channel sampling, collectively 17.1 m, was completed in 2022 with highlights including 15.0 m at 0.46% Li_2O.

9.7.6 CV10 Pegmatite

Approximately six (6) grab / chip rock samples of pegmatite outcrop have been collected at CV10 through 2024, with results ranging from 0.11% to 1.88% Li_2O, 133 ppm to 255 ppm Ta_2O_5, and bdl to 0.68% Cs_2O. No channel sampling has been completed at CV10 through 2024.

9.7.7 CV12 Pegmatite

Approximately 21 grab / chip rock samples of pegmatite outcrop have been collected at CV12 through 2024, with results ranging from bdl to 5.98% Li_2O, and bdl to 1,478 ppm Ta_2O_5, and bdl to 0.17% Cs_2O. Channel sampling, collectively 84.1 m, was completed in 2022 with highlights including 21.9 m at 0.80% Li_2O, 7.7 m at 1.46% Li_2O, and 10.1 m at 1.09% Li_2O. Additionally, strong caesium mineralization was returned in channel sampling – 1.1 m at 3.24% Cs_2O, and 3.3 m at 1.58% Cs_2O.

9.7.8 CV14 Pegmatite

Approximately 12 grab / chip rock samples of pegmatite outcrop have been collected at CV14 through 2024, with results ranging from 0.01% to 0.94% Li_2O, bdl to 88 ppm Ta_2O_5, and bdl to 0.07% Cs_2O. Channel sampling, collectively 77.4 m, was completed in 2024 and returned 6.2 m at 0.16% Li_2O.

9.7.9 CV15 Pegmatite

Approximately eight (8) grab / chip rock samples of pegmatite outcrop have been collected at CV14 through 2024, with results ranging from 0.01% to 2.12% Li_2O, 51 ppm to 252 ppm Ta_2O_5, and bdl to 0.06% Cs_2O. No channel sampling was completed at CV15 through 2024.

Figure 9.1: Company Exploration Summary Through 2024 (West)

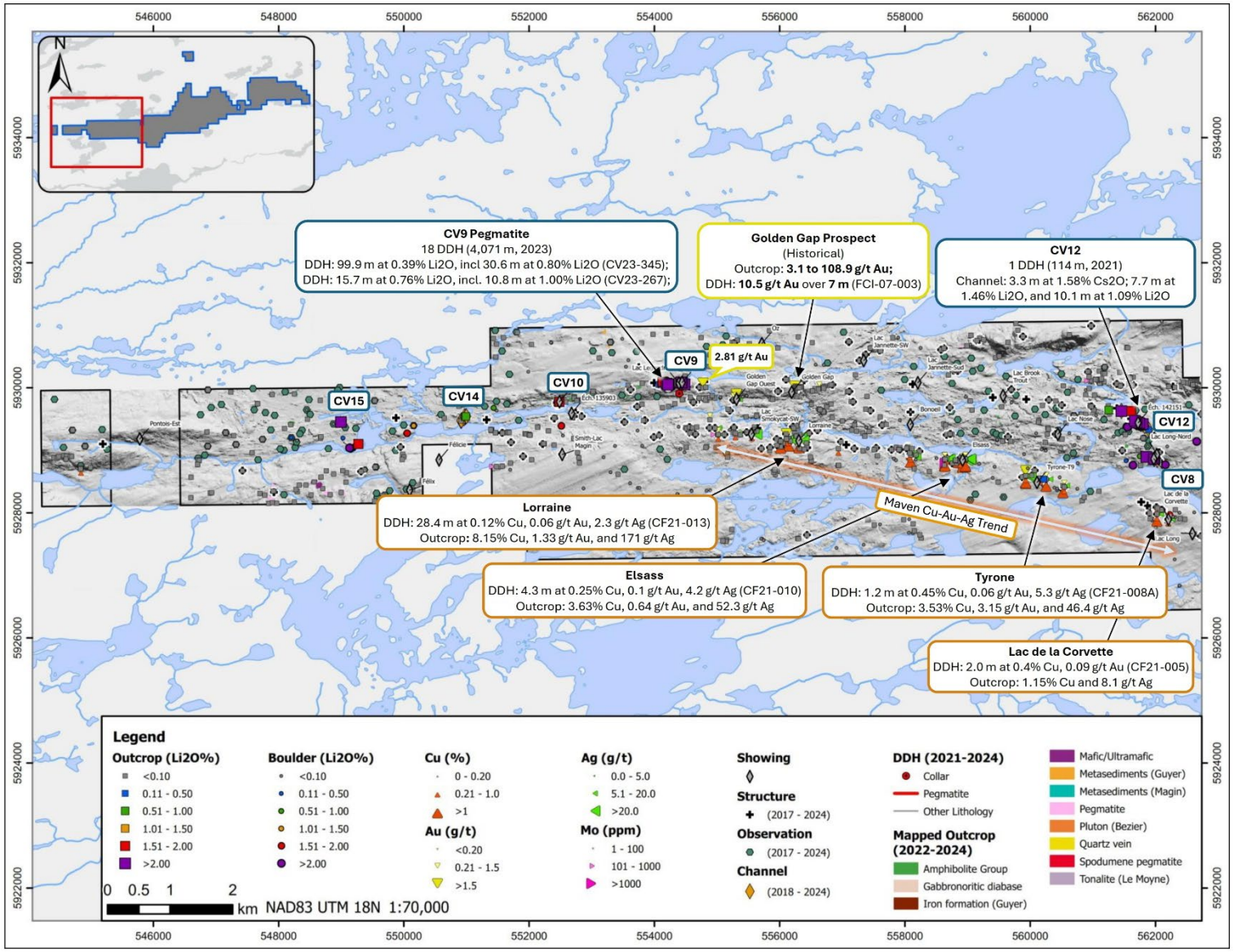


Figure 9.2: Company Exploration Summary Through 2024 (Central)

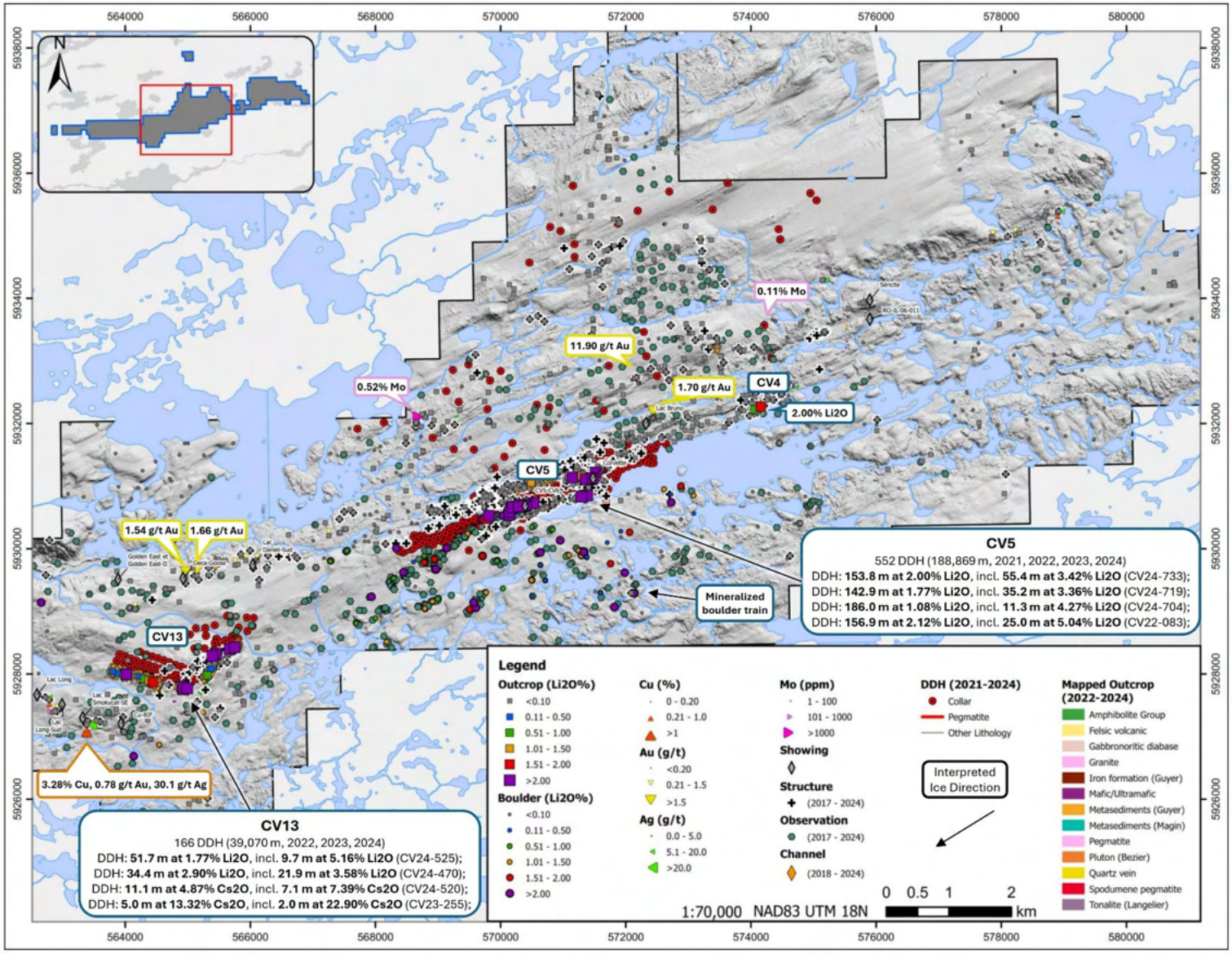
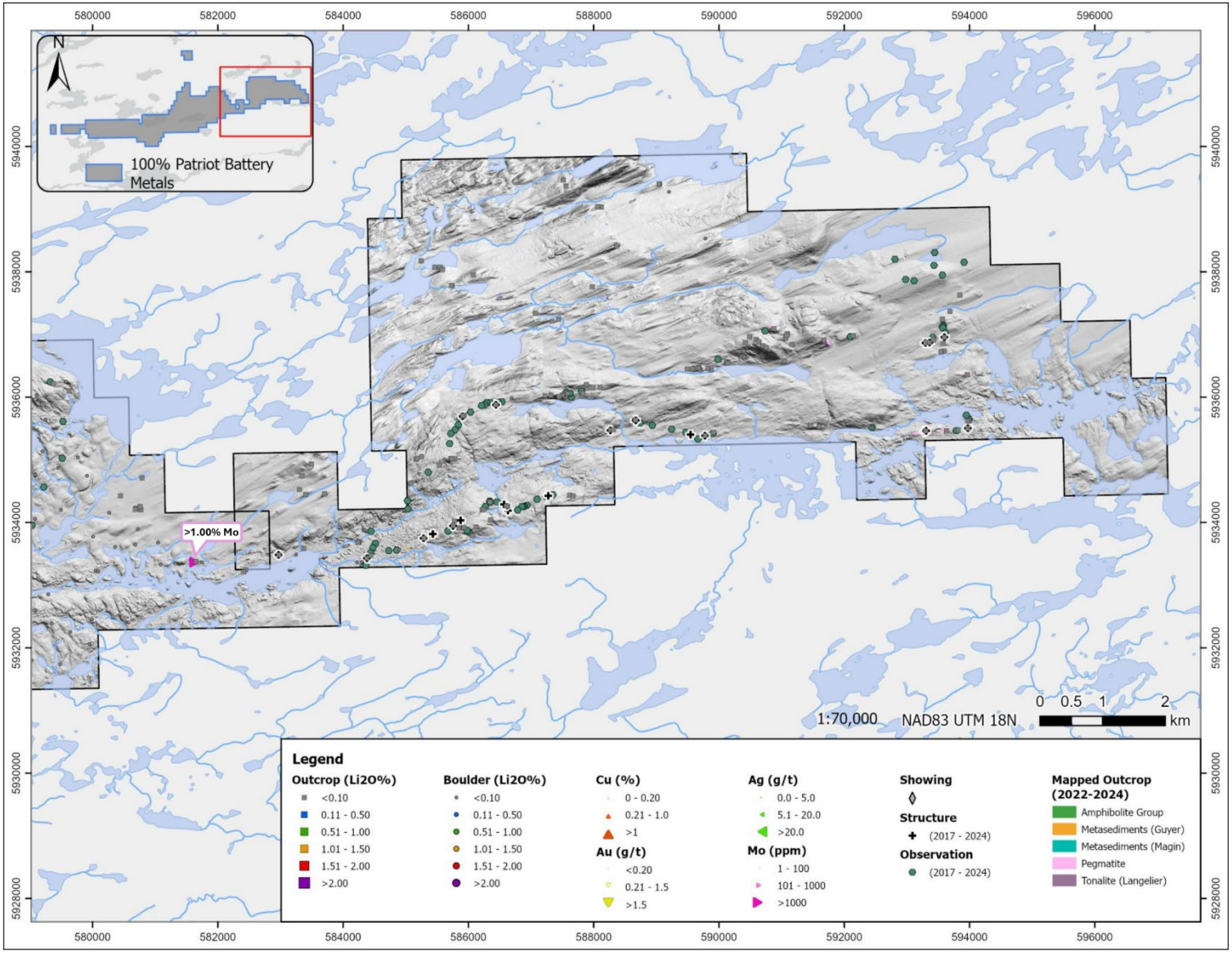


Figure 9.3: Company Exploration Summary Through 2024 (East)



10. DRILLING

10.1 Drilling Campaigns

The Company completed drilling at the Property in 2021 (Maven and CV trends), 2022 (CV Trend), 2023 (CV Trend, Camp), 2024 (CV Trend), and 2025 (CV Trend). Drilling through April 2024 is described in detail in the previous technical reports completed on the Property, including drill hole coordinates and results (Knox, 2022); (McCracken & Cunningham, 2023); and (McCracken, et al., 2024). The following sections provide a summary of the 2021, 2022, 2023 and 2024 (through April) drilling programs for context and completion. A detailed description of the subsequent drilling at the Property is presented below. To date, the Company has not announced any results of the drilling completed in 2025.

The Shaakichiuwaanaan database includes 800 diamond drill holes (DDH) and three (3) rotary drill holes completed over the 2021, 2022, 2023, and 2024 programs (through hole CV24-787), for a collective total of 235,061 m, as well as outcrop channels totalling 800 m. The Shaakichiuwaanaan Consolidated MRE (and host geological models), which includes the CV5 and CV13 pegmatites only, are supported by 720 DDH of NQ (predominant) or HQ size, completed over the 2021, 2022, 2023 and 2024 (through the end of 2024 – drill hole CV24-787) programs, for a collective total of 227,703 m, as well as 604 m of outcrop channels. This equates to 555 holes (188,695 m) and 179 m of outcrop channels at CV5, and 165 holes (39,008 m) and 425 m of outcrop channels at CV13 (Table 10.1). Included within the CV13 dataset are 32 holes totalling 7,808 m, completed over the 2022, 2023 and 2024 programs, as well as 7 m of channels, which were used to support the Vega and Rigel Caesium Zone MRE and geological models.

A plan view drill hole location map for all holes completed by the Company at the Property to date is in Figure 10.1. Plan view drill hole location maps for all holes completed by the Company that have informed the Shaakichiuwaanaan geological model and MRE (i.e., through CV24-787), in addition to channels, are presented in Figure 10.2 and Figure 10.3.

Table 10.1: Company Drill Hole Summary Through 2024 at the Property

Year	Target	No. Holes	Metres	Comments
2021	Maven	10	1,176	
	CV5 Pegmatite	4	758	
	CV12 Pegmatite	1	114	
2022	CV5 Pegmatite	76	23,951	
	CV13 Pegmatite	14	2,647	

Year	Target	No. Holes	Metres	Comments
2023	CV5 Pegmatite	168	58,460	
	CV13 Pegmatite	74	14,917	
	CV9 Pegmatite	18	4,071	
	Shaakichiuwaanaan Camp	7	915	To support construction
2024	CV5 Pegmatite	304	105,701	
	CV13 Pegmatite	78	21,507	
	North of CV5 Pegmatite	49	844	To support CV5 development
Total Property		**803**	**235,061**	
Total CV5 Pegmatite		552	188,869	
Total CV13 Pegmatite		166	39,070	
Total CV9 Pegmatite		18	4,071	
Total CV12 Pegmatite		1	114	

All drill holes from 2021 through 2024 were completed by Fusion Forage Drilling Ltd. of Hawkesbury, Ontario, except for a single water-well drill hole at Shaakichiuwaanaan Camp completed in 2023 by Puisatiers de Delisle Inc. The 2021 and 2022 programs, as well as the summer-fall 2023 program, utilized exclusively helicopter transportable drill rigs. However, the winter / spring 2023 and 2024 programs utilized a combination of helicopter transportable and skid-mounted drill rigs due to the construction of a temporary winter road, and later an all-season road, extending from the Trans-Taiga Road to the CV5 Pegmatite.

To date, no oriented drill coring has been completed; however, downhole optical and acoustic televiewer surveys have been completed on multiple holes at the CV5 and CV13 pegmatites to assess overall structure. This data has guided the geological model supporting the MRE as well as subsequent refinement.

With respect to the 2021, 2022, 2023 and 2024 drilling programs as discussed herein, there were no drilling, sampling or recovery factors identified that could materially impact the accuracy and reliability of the results presented herein. No detailed evaluation has been completed on drilling completed subsequent to the July 2025 Consolidated MRE, the current MRE for the Project.

Figure 10.1: Drill Holes Completed at Shaakichiuwaanaan Property's Principal Claim Grouping (through to CV24-787)

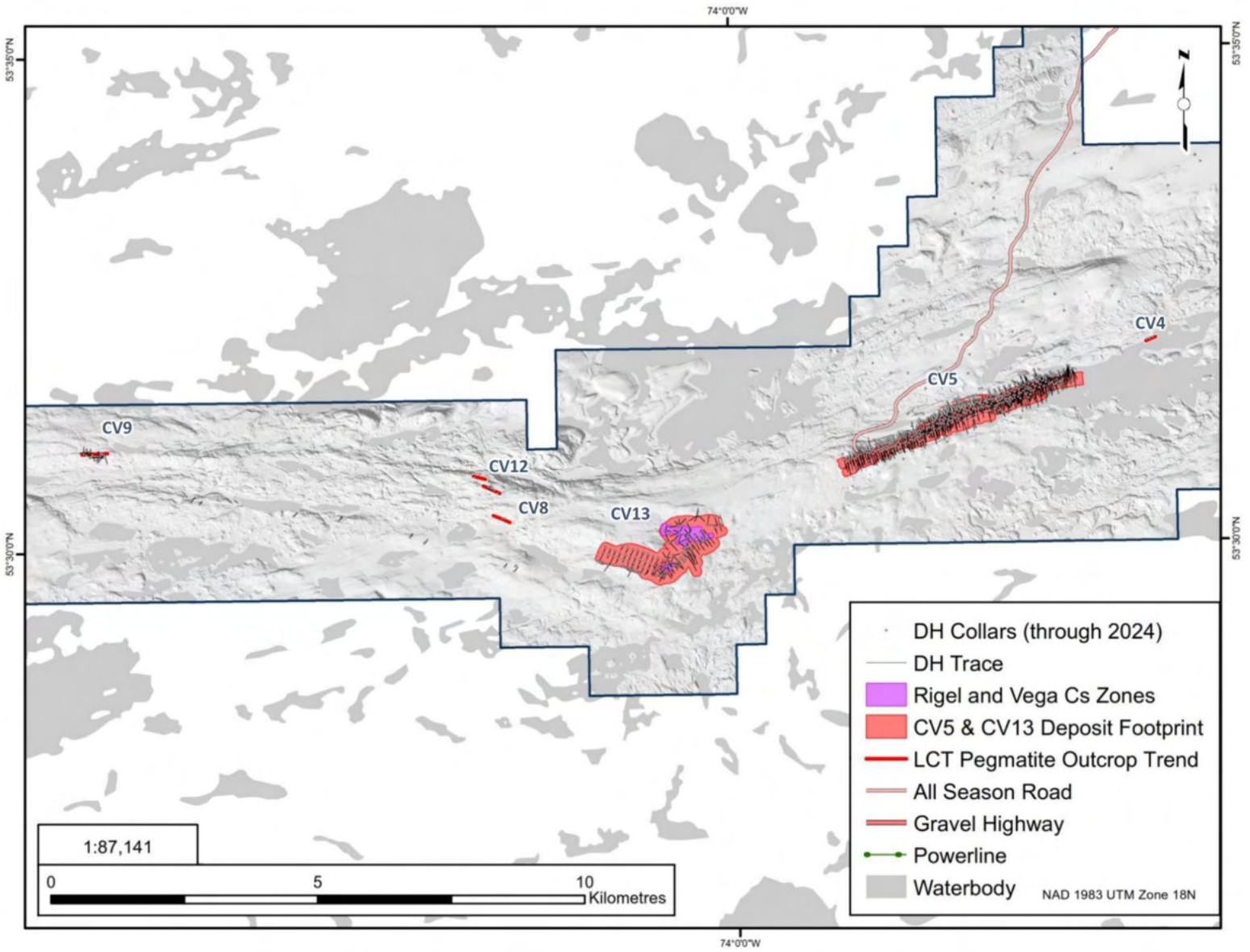


Figure 10.2: Drill Holes and Channels Completed at CV5 LCT Pegmatite Through 2024

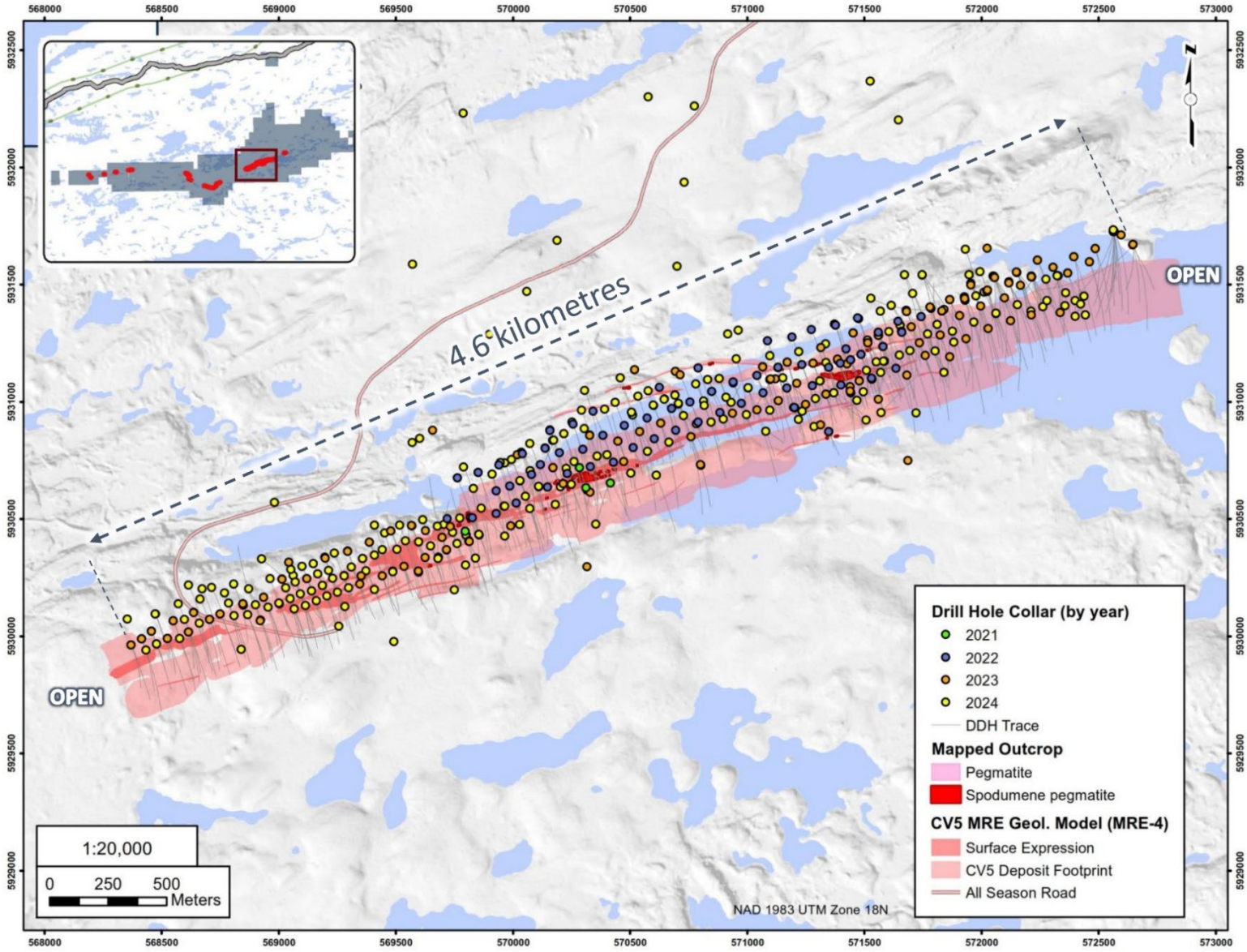
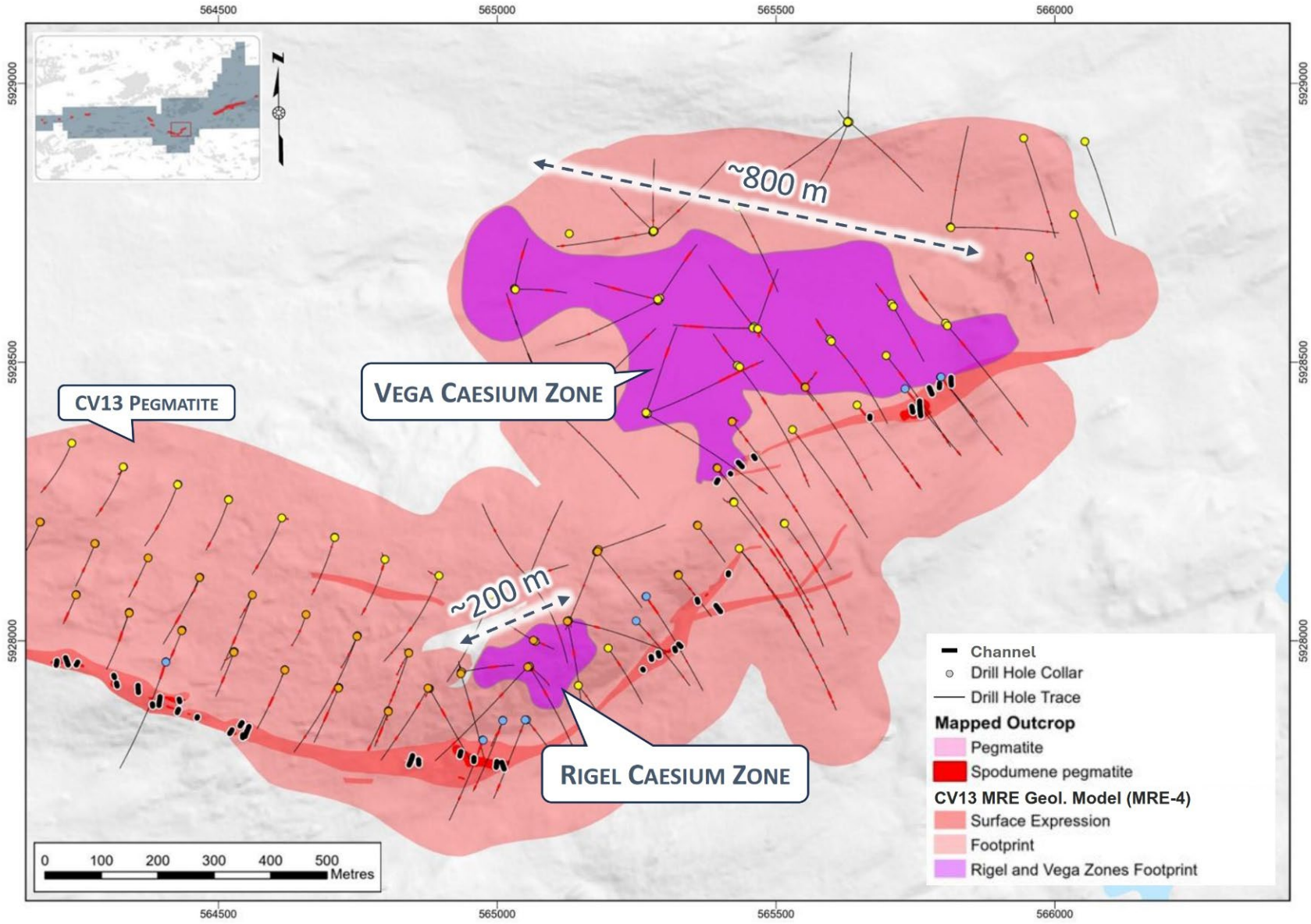


Figure 10.3: Drill Holes and Channels Completed at CV13 LCT Pegmatite Through 2024



10.1.1 2021 Drill Program

The Company completed a diamond drilling program on the Property in September – October 2021. The program included 15 NQ size DDH, totalling 2,048 m, and was split over two (2) prospective trends – the CV LCT Pegmatite Trend (872 m over five (5) holes) and the Maven Cu-Au-Ag Trend (1,176 m over 10 holes). The drilling program (drill holes CF21-001 through CF21-014) marked the first documented drilling along the Maven Trend, as well as for LCT pegmatite on the Property (Figure 10.1, Figure 10.2, and Figure 9.1).

The primary objective of the LCT pegmatite drilling in 2021 at the CV Trend was to test if the two (2) main spodumene (lithium) pegmatite outcrops at CV5 continued to depth. The drilling was very successful with results including:

- 148.7 m at 0.93% Li_2O and 114 ppm Ta_2O_5, including 73.0 m at 1.09% Li_2O and 108 ppm Ta_2O_5 (CF21-001, the 'discovery hole').
- 154.1 m at 0.94% Li_2O and 118 ppm Ta_2O_5, including 38.0 m at 1.38% Li_2O and 160 ppm Ta_2O_5 (CF21-002).
- 59.1 m at 1.23% Li_2O and 194 ppm Ta_2O_5, including 33.0 m at 1.80% Li_2O and 264 ppm Ta_2O_5 (CF21-003).
- 63.6 m at 0.64% Li_2O and 231 ppm Ta_2O_5, including 30.0 m at 1.13% Li_2O and 180 ppm Ta_2O_5 (CF21-004).

Drilling in 2021 at the Maven Trend tested geophysical and surface-derived targets at the Lac de la Corvette, Tyrone-T9, Elsass, and Lorraine showings / prospects. The program returned anomalous to moderate grades over several drill holes (Figure 9.1), including individual sample highs comparable to prior surface results – 3.1 m of 0.34% Cu, 0.21 g/t Au, and 6.7 g/t Ag within a larger interval of 28.4 m of 0.12% Cu, 0.06 g/t Au, and 2.3 g/t Ag (CF21-013, Lorraine), and 0.2 m of 2.12% Cu, 0.26 g/t Au, and 25.4 g/t Ag (CF21-008A, Tyrone-T9). Mineralization consists of visible chalcopyrite present as stringers and disseminations.

10.1.2 2022 Drill Program

The Company completed a diamond drilling campaign on the Property throughout 2022, which included winter / spring and summer / fall programs (through drill hole CV22-104). Collectively, the program included 90 NQ size DDH, totalling 26,598 m – 76 holes totalling 23,951 m at the CV5 Pegmatite and 14 holes totalling 2,647 m at the CV13 Pegmatite (Figure 10.2 and Figure 10.3).

The primary objective of the drilling was the delineation of the CV5 Pegmatite in support of a maiden MRE and initial drill testing of the CV13 Pegmatite discovered earlier in the year. The drilling at both pegmatites was very successful, with results including:

At the CV5 Pegmatite:

- 156.9 m at 2.12% Li_2O, 0.11% Cs_2O, and 181 ppm Ta_2O_5, including 25.0 m at 5.04% Li_2O, 0.38% Cs_2O, and 270 ppm Ta_2O_5, or 5.0 m at 6.36% Li_2O, 0.42% Cs_2O, and 216 ppm Ta_2O_5 (CV22-083).
- 52.2 m at 3.34% Li_2O, 0.08% Cs_2O, and 229 ppm Ta_2O_5, including 15.0 m at 5.10% Li_2O, 0.14% Cs_2O, and 314 ppm Ta_2O_5 (CV22-093).
- 131.2 m at 1.96% Li_2O, 0.06% Cs_2O, and 421 ppm Ta_2O_5, including 57.0 m at 2.97% Li_2O, 0.09% Cs_2O, and 185 ppm Ta_2O_5 (CV22-100).
- 159.7 m at 1.65% Li_2O, 0.12% Cs_2O, and 193 ppm Ta_2O_5, including 37.0 m at 3.05% Li_2O, 0.20% Cs_2O, and 207 ppm Ta_2O_5 (CV22-042).
- 152.8 m at 1.22% Li_2O, 0.14% Cs_2O, and 138 ppm Ta_2O_5, including 66.0 m at 1.51% Li_2O, 0.18% Cs_2O, and 100 ppm Ta_2O_5 (CV22-030).

At the CV13 Pegmatite:

- 22.6 m at 1.56% Li_2O, 0.11% Cs_2O, and 240 ppm Ta_2O_5, including 6.0 m at 3.19% Li_2O, 0.25% Cs_2O, and 270 ppm Ta_2O_5 (CV22-092).
- 22.4 m at 1.28% Li_2O, 0.05% Cs_2O, and 124 ppm Ta_2O_5 (CV22-077) – collared in LCT pegmatite.
- 15.6 m at 1.50% Li_2O, 0.06% Cs_2O, and 113 ppm Ta_2O_5 (CV22-081) – collared in LCT pegmatite.
- 17.3 m at 1.41% Li_2O, 0.05% Cs_2O, and 91 ppm Ta_2O_5, including 8.0 m at 2.09% Li_2O, 0.07% Cs_2O, and 133 ppm Ta_2O_5 (CV22-104).

A major development from the 2022 drilling campaign was the recognition of a continuous high-grade zone at CV5, termed the ‘Nova Zone’ (see Figure 7.14). At the end of the 2022 program, the Nova Zone had been delineated over a strike length of approximately 350 m (later extended to at least 1.1 km). This included intersections in drill holes CV22-017 (40.7 m at 3.01% Li_2O), CV22-042 (37.0 m at 3.04% Li_2O), CV22-066 (38.0 m at 2.17% Li_2O, including 2.0 m at 6.41% Li_2O), and CV22-083 (25.0 m at 5.04% Li_2O, including 5.0 m at 6.36% Li_2O), CV22-093 (15.0 m at 5.10% Li_2O), and CV22-100 (57.0 m at 2.97% Li_2O).

10.1.3 2023 Drill Program

The Company completed a diamond drilling campaign (plus a single rotary drill hole) on the Property throughout 2023, which included 267 NQ (predominant) and HQ size holes, totalling 78,363 m (CV23-105 through CV23-365). This included drill holes at the CV5, CV13 and CV9 LCT pegmatites, as well as hydrogeological holes at the Company's Shaakichiuwaanaan Camp situated on the south side of the Trans-Taiga Road, approximately 13 km directly north of the CV5 Pegmatite.

The primary objective of the drilling was continued delineation of the CV5 and CV13 pegmatites, initial drill testing of the CV9 pegmatite, as well as collecting hydrogeological information to support a preliminary hydrological model for the CV5 area and camp.

10.1.3.1 CV5 Pegmatite

Drill result highlights for lithium from 168 holes (58,460 m) completed at the CV5 Pegmatite in 2023 include:

- 83.7 m at 3.13% Li_2O, 0.06% Cs_2O, and 235 ppm Ta_2O_5, including 19.8 m at 5.28% Li_2O, 0.09% Cs_2O, and 283 ppm Ta_2O_5, and 5.1 m at 5.17% Li_2O, 0.11% Cs_2O, and 287 ppm Ta_2O_5 (CV23-105).
- 139.2 m at 1.26% Li_2O, 0.09% Cs_2O, and 106 ppm Ta_2O_5, including 36.2 m at 1.74% Li_2O, 0.08% Cs_2O, and 112 ppm Ta_2O_5 (CV23-190).
- 130.3 m at 1.56% Li_2O, 0.14% Cs_2O, and 185 ppm Ta_2O_5, including 52.7 m at 2.45% Li_2O, 0.28% Cs_2O, and 168 ppm Ta_2O_5 (CV23-132).
- 122.6 m at 1.89% Li_2O, 0.10% Cs_2O, and 175 ppm Ta_2O_5, including 8.1 m at 5.01% Li_2O, 0.24% Cs_2O, and 274 ppm Ta_2O_5 (CV23-138).
- 101.2 m at 1.59% Li_2O, 0.03% Cs_2O, and 246 ppm Ta_2O_5, including 8.8 m at 5.20% Li_2O, 0.12% Cs_2O, and 303 ppm Ta_2O_5 (CV23-141).
- 108.0 m at 2.44% Li_2O, 0.24% Cs_2O, and 277 ppm Ta_2O_5, including 16.0 m at 4.08% Li_2O, 0.35% Cs_2O, and 206 ppm Ta_2O_5 (CV23-181).
- 172.4 m at 0.95% Li_2O, 0.10% Cs_2O, and 121 ppm Ta_2O_5, including 34.5 m at 1.85% Li_2O, 0.09% Cs_2O, and 121 ppm Ta_2O_5 (CV23-199).
- 133.9 m at 1.21% Li_2O, 0.10% Cs_2O, and 154 ppm Ta_2O_5, including 41.5 m at 1.52% Li_2O, 0.17% Cs_2O, and 199 ppm Ta_2O_5, and 42.2 m at 1.59% Li_2O, 0.06% Cs_2O, and 93 ppm Ta_2O_5 (CV23-298).

The drilling at CV5 was very successful and extended the strike length of CV5 to 4.6 km (drill hole to drill hole), up from 2.2 km at the end of 2022, with it remaining open. The program also provided the final set of drill hole data informing a maiden Mineral Resource Estimate for the CV5 Pegmatite, announced in July 2023 (PMET, 2023a), which established the CV5 Pegmatite as a world-class lithium pegmatite, ranking largest in the Americas and in the top 10 globally at the time.

The Nova Zone, first discovered during the 2022 drilling program, was also expanded over the course of the 2023 drilling program (January through April) to at least 1.1 km. At the Nova Zone, geological modelling supports a continuous spodumene mineralized zone of variable thickness, at grades of 2% to over 5% Li_2O, occurring between vertical depths of approximately 125 m to 325 m. The high-grade zone includes a higher-grade sub-zone that is an approximate 3 m to 25 m thick (core length) band of 5%+ Li_2O spodumene pegmatite traced over at least a 200 m strike length.

Additionally, caesium overlimit analysis was completed in early 2025 on samples from several holes completed in 2023 at CV5. Using the overlimit analysis, results for caesium drill intersections completed in 2023 at CV5 include:

- 10.4 m at 1.77% Li_2O, 1.30% Cs_2O, and 240 ppm Ta_2O_5, including 4.0 m at 2.06% Li_2O, 2.02% Cs_2O, and 341 ppm Ta_2O_5 (CV23-117).
- 2.0 m at 3.50% Li_2O, 5.24% Cs_2O, and 187 ppm Ta_2O_5 (CV23-219).

10.1.3.2 CV13 Pegmatite (Lithium)

Drill result highlights for lithium from 74 holes (14,917 m) completed at the CV13 Pegmatite in 2023 include:

- 12.7 m at 2.46% Li_2O, 0.18% Cs_2O, and 147 ppm Ta_2O_5, including 7.6 m at 3.82% Li_2O, 0.28% Cs_2O, and 145 ppm Ta_2O_5 (CV23-191).
- 8.0 m at 2.86% Li_2O, 0.14% Cs_2O, and 378 ppm Ta_2O_5, including 4.3 m at 5.03% Li_2O, 0.24% Cs_2O, and 261 ppm Ta_2O_5 (CV23-195).
- 10.2 m at 2.70% Li_2O, 1.52% Cs_2O, and 265 ppm Ta_2O_5, including 5.8 m at 4.48% Li_2O, 2.63% Cs_2O, and 265 ppm Ta_2O_5 (CV23-198).
- 28.7 m at 1.49% Li_2O, 0.04% Cs_2O, and 94 ppm Ta_2O_5, including 20.4 m at 2.03% Li_2O, 0.04% Cs_2O, and 91 ppm Ta_2O_5 (CV23-311).
- 19.2 m at 1.74% Li_2O, 0.05% Cs_2O, and 98 ppm Ta_2O_5 (CV23-215).
- 22.5 m at 1.10% Li_2O, 0.04% Cs_2O, and 69 ppm Ta_2O_5, including 15.2 m at 1.57% Li_2O, 0.03% Cs_2O, and 83 ppm Ta_2O_5 (CV23-300).

- 16.1 m at 1.54% Li_2O, 0.06% Cs_2O, and 59 ppm Ta_2O_5, including 7.2 m at 2.57% Li_2O, 0.04% Cs_2O, and 68 ppm Ta_2O_5 (CV23-319).

A significant development from the drilling at CV13 was the identification of a new high-grade zone located near surface (~40-50 m vertical depth) near the apex of the pegmatite. Additionally, drill hole CV23-195 returned two (2) samples assaying greater than 6% Li_2O, including 1.2 m at 6.41% Li_2O (PMET, 2023b). This high-grade zone is partially coincident with the Rigel Caesium Zone.

10.1.3.3 CV13 Pegmatite (Caesium)

A caesium overlimit analysis was completed in early 2025 on several holes completed in 2023 at CV13. Using the overlimit analysis, drill result highlights for caesium from 74 holes (14,917 m) completed at the CV13 Pegmatite in 2023 include:

Rigel Caesium Zone:

- 5.9 m at 1.07% Li_2O, 11.19% Cs_2O, and 3,261 ppm Ta_2O_5, including 1.0 m at 0.48% Li_2O, 22.69% Cs_2O, and 110 ppm Ta_2O_5 (CV23-271).
- 5.0 m at 0.24% Li_2O, 13.32% Cs_2O, and 1 ppm Ta_2O_5, including 2.0 m at 0.44% Li_2O, 22.90% Cs_2O, and 1 ppm Ta_2O_5 (CV23-255).
- 3.2 m at 2.89% Li_2O, 10.23% Cs_2O, and 814 ppm Ta_2O_5, including 1.1 m at 0.23% Li_2O, 26.61% Cs_2O, and 1 ppm Ta_2O_5 (CV23-204).
- 4.5 m at 4.19% Li_2O, 3.36% Cs_2O, and 333 ppm Ta_2O_5 (CV23-198).

The Rigel Zone, within the CV13 Pegmatite, was first intersected in a drill hole in 2023; however, caesium overlimit analysis was not received until early 2025. Subsequent modelling using the 0.5% Cs_2O grade constraint has outlined a footprint of caesium mineralization at Rigel over a general area of at least 200 m x 100 m and consists of a single, shallow-dipping lens at a depth of ~50 m with a true thickness of <2 m to ~6 m. The zone is located at the apex of the regional structural flexure controlling CV13 and is partially coincident with the high-grade lithium zone discovered in drill holes CV23-191 and CV23-195 described above. The Rigel Caesium Zone is described in further detail in Chapter 7, including a cross-section (see Figure 7.23).

Additionally, for reference, 3.0 m at 9.43% Cs_2O, including 1.0 m at 22.41% Cs_2O, was returned from a channel (CH23-069) completed at the Vega Caesium Zone in 2023.

Through 2023, the CV13 Pegmatite had been traced by drilling over an approximate 2.3 km strike length and remained open along strike at both ends and to depth.

10.1.3.4 CV9 Pegmatite

Drill result highlights from 18 holes (4,071 m) completed at the CV9 Pegmatite in 2023 include:

- 99.9 m at 0.39% Li_2O, 0.03% Cs_2O, and 69 ppm Ta_2O_5, including 30.6 m at 0.80% Li_2O, 0.03% Cs_2O, and 75 ppm Ta_2O_5 (CV23-345).
- 15.7 m at 0.76% Li_2O, 0.03% Cs_2O, and 81 ppm Ta_2O_5, including 10.8 m at 1.00% Li_2O, 0.03% Cs_2O, and 79 ppm Ta_2O_5 (CV23-267).
- 7.7 m at 1.35% Li_2O, 0.02% Cs_2O, and 55 ppm Ta_2O_5 (CV23-333).

The program was the initial drill testing of the spodumene pegmatite outcrops that define the CV9 Pegmatite at surface, with a primary objective to determine the geometry and orientation of the pegmatite system. A total of 18 holes (4,071 m) were completed.

The results are encouraging and confirm that widespread spodumene mineralization is present at depth at CV9. The pegmatite intersected in drill hole at CV9 is variably mineralized (typically <5% to 15% spodumene content), with strong grades (>1% Li_2O) demonstrated over 7 m to 10+ m intervals in addition to wider and more moderately mineralized zones (e.g., 30.6 m at 0.80% Li_2O in CV23-345). High grades of spodumene pegmatite were also intercepted with multiple holes returning individual sample grades over 2% Li_2O, including a peak sample high of 4.28% Li_2O (over 0.6 m) in CV23-345 – the last drill hole of the program at CV9.

10.1.4 2024 Drill Program

The Company completed a diamond drilling campaign (plus two (2) rotary drill holes) on the Property throughout 2024 that included 431 NQ (predominant) and HQ-size holes, totalling 128,052 m (CV24-366 through CV24-787). This included drill holes at the CV5 and CV13 pegmatites, as well as holes north of CV5 at potential waste rock pile locations. (Figure 10.1, Figure 10.2, and Figure 10.3).

The primary objectives of the drilling were continued delineation of the CV5 Pegmatite to support conversion of Mineral Resources from the Inferred category to the Indicated category, to complete geotechnical, geomechanical, and hydrogeological drilling in support of CV5 development, and continue step-out delineation of the CV13 Pegmatite.

10.1.4.1 CV5 Pegmatite

Drill result highlights for lithium from 304 holes (105,701 m) completed at the CV5 Pegmatite in 2024 include:

- 126.3 m at 1.66% Li_2O, 0.13% Cs_2O, and 158 ppm Ta_2O_5, including 54.9 m at 2.50% Li_2O, 0.19% Cs_2O, and 129 ppm Ta_2O_5 (CV24-374).
- 124.9 m at 1.72% Li_2O, 0.04% Cs_2O, and 138 ppm Ta_2O_5, including 13.4 m at 4.04% Li_2O, 0.05% Cs_2O, and 175 ppm Ta_2O_5 (CV24-473).
- 122.5 m at 1.42% Li_2O, 0.06% Cs_2O, and 138 ppm Ta_2O_5, including 35.8 m at 2.15% Li_2O, 0.07% Cs_2O, and 170 ppm Ta_2O_5 (CV24-405).
- 135.7 m at 1.02% Li_2O, 0.05% Cs_2O, and 106 ppm Ta_2O_5, including 44.7 m at 2.03% Li_2O, 0.03% Cs_2O, and 69 ppm Ta_2O_5 (CV24-410).
- 112.7 m at 1.20% Li_2O, 0.08% Cs_2O, and 86 ppm Ta_2O_5, including 21.7 m at 1.93% Li_2O, 0.04% Cs_2O, and 99 ppm Ta_2O_5 (CV24-503).
- 100.8 m at 1.97% Li_2O, 0.08% Cs_2O, and 125 ppm Ta_2O_5, including 69.8 m at 2.52% Li_2O, 0.08% Cs_2O, and 113 ppm Ta_2O_5 (CV24-392).
- 90.2 m at 1.29% Li_2O, 0.05% Cs_2O, and 108 ppm Ta_2O_5, and 48.5 m at 1.25% Li_2O, 0.12% Cs_2O, and 121 ppm Ta_2O_5 (CV24-377).
- 94.9 m at 1.10% Li_2O, 0.04% Cs_2O, and 123 ppm Ta_2O_5, including 26.1 m at 2.16% Li_2O, 0.05% Cs_2O, and 99 ppm Ta_2O_5 (CV24-378).
- 66.8 m at 2.56% Li_2O, 0.19% Cs_2O, and 220 ppm Ta_2O_5, including 46.9 m at 3.53% Li_2O, 0.26% Cs_2O, and 188 ppm Ta_2O_5, or 16.1 m at 5.02% Li_2O, 0.46% Cs_2O, and 243 ppm Ta_2O_5 (CV24-401A).
- 63.7 m at 2.68% Li_2O, 0.23% Cs_2O, and 160 ppm Ta_2O_5, including 35.6 m at 3.79% Li_2O, 0.38% Cs_2O, and 202 ppm Ta_2O_5 (CV24-404).

The program was successful in further delineation of the CV5 Spodumene Pegmatite, with results generally in line with expectations and prior results. Several new mineralized pegmatite veins were encountered at depth at various locations at CV5 and highlight the strong potential remaining for additional discovery. As the focus was on infill, no step-out holes at CV5 were completed in 2024, and, therefore, the delineated strike length remained at 4.6 km.

Additionally, several drill holes completed in 2024 returned significant caesium mineralization with overlimit analysis completed on several samples in 2025. Results with overlimit analysis include:

- 9.0 m at 2.02% Li_2O, 1.20% Cs_2O, and 194 ppm Ta_2O_5, including 1.5 m at 2.22% Li_2O, 5.03% Cs_2O, and 219 ppm Ta_2O_5 (CV24-651).
- 7.5 m at 3.85% Li_2O, 1.29% Cs_2O, and 200 ppm Ta_2O_5, including 1.5 m at 3.93% Li_2O, 3.90% Cs_2O, and 143 ppm Ta_2O_5 (CV24-404).
- 0.8 m at 1.79% Li_2O, 13.04% Cs_2O, and 205 ppm Ta_2O_5 (CV24-627).

10.1.4.2 CV13 Pegmatite (Lithium)

Drill result highlights for lithium from 78 holes (21,507 m) completed at the CV13 Pegmatite in 2024 include:

- 51.7 m at 1.77% Li_2O, 1.06% Cs_2O, and 105 ppm Ta_2O_5, including 9.7 m at 5.16% Li_2O, 0.33% Cs_2O, and 86 ppm Ta_2O_5 (CV24-525).
- 34.4 m at 2.90% Li_2O, 0.46% Cs_2O, and 124 ppm Ta_2O_5, including 21.9 m at 3.58% Li_2O, 0.25% Cs_2O, and 133 ppm Ta_2O_5 (CV24-470).
- 33.4 m at 2.40% Li_2O, 0.89% Cs_2O, and 100 ppm Ta_2O_5, including 11.1 m at 4.33% Li_2O, 0.82% Cs_2O, and 97 ppm Ta_2O_5; and 17.6 m at 1.89% Li_2O, 0.28% Cs_2O, and 119 ppm Ta_2O_5, including 5.6 m at 3.40% Li_2O, 0.11% Cs_2O, and 96 ppm Ta_2O_5 (CV24-507).
- 39.5 m at 1.18% Li_2O, 0.40% Cs_2O, and 155 ppm Ta_2O_5, including 12.9 m at 3.06% Li_2O, 0.92% Cs_2O, and 75 ppm Ta_2O_5 (CV24-498).
- 27.1 m at 1.02% Li_2O, 0.08% Cs_2O, and 135 ppm Ta_2O_5, including 7.6 m at 2.39% Li_2O, 0.18% Cs_2O, and 175 ppm Ta_2O_5 (CV24-513).
- 32.1 m at 0.78% Li_2O, 0.04% Cs_2O, and 95 ppm Ta_2O_5, including 10.7 m at 2.17% Li_2O, 0.04% Cs_2O, and 81 ppm Ta_2O_5 (CV24-499).

Along the western arm, the pegmatite was extended down-dip over 400 m. Moreover, along the eastern arm, the high-grade Vega Lithium Zone was discovered. At the end of the program, the Vega Lithium Zone had been delineated to be relatively flat-lying to shallow dipping and near-surface (starting at ~100 m vertical depth from surface), covering an area of at least 700 m by 250 m with an interpreted true thickness of ~8 to 30+ m, hosted within a wider moderately to strongly mineralized pegmatite body.

10.1.4.3 CV13 Pegmatite (Caesium)

Drill result highlights for caesium from 78 holes (21,507 m) completed at the CV13 Pegmatite in 2024 include:

Vega Caesium Zone:

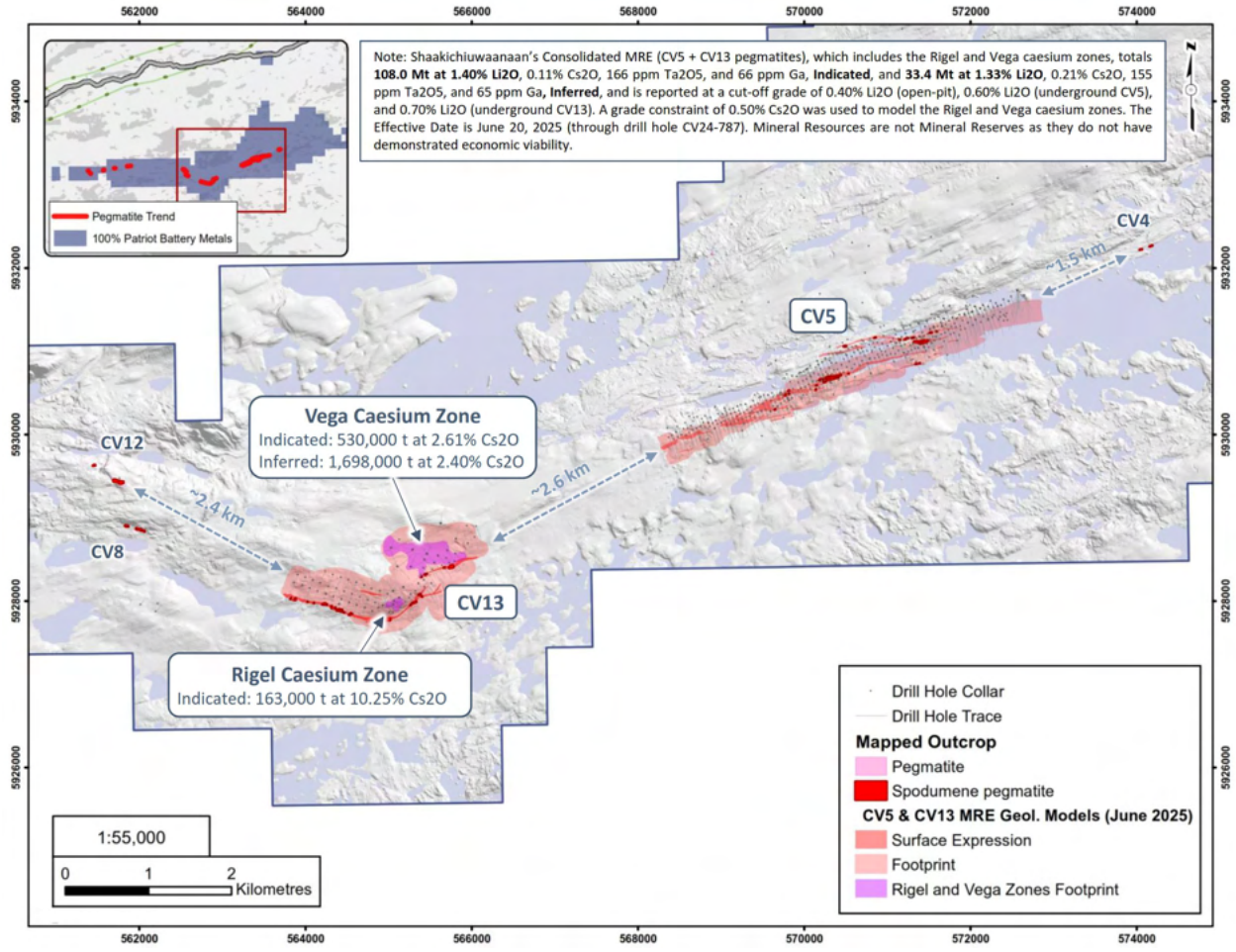
- 18.1 m at 1.89% Li_2O, 2.71% Cs_2O, and 288 ppm Ta_2O_5, including 7.5 m at 0.99% Li_2O, 5.45% Cs_2O, and 286 ppm Ta_2O_5 (CV24-754).
- 11.1 m at 2.09% Li_2O, 4.87% Cs_2O, and 1,116 ppm Ta_2O_5, including 7.1 m at 0.96% Li_2O, 7.39% Cs_2O, and 103 ppm Ta_2O_5 (CV24-520).
- 5.7 m at 0.99% Li_2O, 4.97% Cs_2O, and 61 ppm Ta_2O_5, including 3.0 m at 1.16% Li_2O, 8.20% Cs_2O, and 30 ppm Ta_2O_5 (CV24-525).
- 9.6 m at 2.08% Li_2O, 1.59% Cs_2O, and 371 ppm Ta_2O_5, including 4.4 m at 3.55% Li_2O, 2.34% Cs_2O, and 354 ppm Ta_2O_5 (CV24-579).

In addition to the discovery of the Vega Lithium Zone, the 2024 drilling at CV13 also delineated the Vega Caesium Zone. The Vega Caesium Zone is coincident with the Vega Lithium Zone, whereby the caesium enrichment occupies two (2) extensive bands, with the primary band up to 10 m thick, within the high-grade zone of lithium in the wider CV13 Pegmatite body.

Subsequent modelling using a 0.5% Cs_2O grade constraint has outlined a Vega Caesium Zone footprint over a general area of at least 800 m x 250 m and consists of two (2) proximal flat-lying lenses, at a depth of ~110 m, with a true thickness of <2 m and up to ~10 m and ~6 m, respectively. The Vega Caesium Zone is described in further detail in Chapter 7, including a cross-section (see Figure 7.24).

Through the end of the 2024 program, the CV13 Pegmatite had been traced by drilling over a strike length of at least 2.5 km (drill hole to drill hole) and remains open along strike at both ends and to depth along a significant portion of its length. Mineralized pegmatite at CV13 has been traced to within approximately 2.6 km of the CV5 Pegmatite to the northeast and 2.4 km of the CV12 Pegmatite to the northwest (Figure 10.4).

Figure 10.4: Drill Holes Completed Through 2024 at the CV5 and CV13 LCT Pegmatites



10.1.5 2025 Drilling Program

In January 2025, the Company commenced a drilling campaign on the Property with three (3) primary objectives:

- Extend the high-grade Vega Lithium Zone (CV13 Pegmatite) westward along an interpreted structural corridor toward the CV12 Pegmatite.
- Test the highly prospective CV5 to CV13 pegmatite corridor.
- Complete condemnation drilling at two (2) proposed waste rock pile locations in support of the ongoing Feasibility Study at the CV5 Pegmatite.

In 2025 through June 30th, the Company had completed a total of 28,088 m (121 holes) of drilling at the Shaakichiuwaanaan Project. This includes 14,461 m (49 holes) at CV13, 1,955 m (5 holes) at the CV5-CV13 corridor, 5,464 m (36 holes) at the waste rock piles for condemnation, and 6,208 m (31 holes)

at the CV12 Pegmatite. No drill results from the 2025 drilling campaign have been reported to date by the Company. The 2025 Drilling activities at the Property concluded in October.

10.2 Structure

To date, no oriented drill coring has been completed; however, downhole optical and acoustic televiewer surveys have been completed on multiple holes to assess the overall structure of the CV5 and CV13 LCT pegmatites. The surveying was completed by DGI Geoscience Inc. over multiple periods. The data has guided the geological model supporting the Shaakichiuwaanaan Consolidated Mineral Resource Estimate.

10.3 Collar Survey

Each drill hole collar (CF21-001 through CV24-787) was surveyed with an RTK tool (Topcon GR5 or Trimble Zephyr 3), with some minor exceptions that were surveyed using a handheld GPS (Garmin GPSMAP 64s) only. All collar survey data has been validated by the Project geologists on site, and by the database lead.

10.4 Downhole Deviation Survey

Downhole deviation surveys for each drill hole were completed with a Devico DeviGyro tool (2021 and 2024 holes), Reflex Gyro Sprint IQ tool (2022, 2023 and 2024 holes), Axis Champ Gyro (2023 and 2024 holes), or Reflex OMNI Gyro Sprint IQ (2024 holes). Survey shots were typically continuous at approximately 3-5 m intervals. The use of the gyro tool system negated potential deflection issues arising from minor but common pyrrhotite within the host rock units. All collar and downhole deviation data have been validated by the Project geologists on site, and by the database lead.

10.5 Core Logging and Sampling Procedures

Procedures at the drill followed industry best practices with drill core placed in either 4 or 5-foot-long, typically flat, square-bottom wooden boxes, with the appropriate hole and box ID noted and block depth markers placed in the box. Core recovery typically exceeds 90%. Once full, the box was fibre-taped shut with wooden lids at the drill and box slung directly to Mirage Lodge for processing (2021) or north by helicopter to a laydown area on the Trans-Taiga Road (KM-270 or KM-277), where they were then transported by truck to Mirage Lodge for processing (2022 and 2023). In 2023 (winter) and 2024, the core was also transported by winter / all-season road from the Property to Mirage Lodge for processing.

Upon receipt at the core shack, the core box information was confirmed, and all drill cores were pieced together, oriented to maximum foliation. The core was then metre-marked, geotechnically logged (TCR, RQD, ISRM, and Q-Method (since mid-winter 2023)), alteration logged, structure logged, geologically logged (rock type), and sample logged and marked on an individual sample basis. The logging of drill core was qualitative by nature and included estimates of spodumene grain size, inclusions, and model mineral estimates. The drill core was then, prior to sampling, wet- and dry-photographed for a digital record of all cores received in the core shack.

These logging practices meet current industry standard practices and are of appropriate detail to support a Mineral Resource estimation. All protocols employed are considered appropriate for the sample type and nature of mineralization and are considered the optimal approach for maintaining representativeness in sampling. Further details are provided below.

10.5.1 2021 Drill Program

Core sample collection was guided by lithology, mineralogy, and textural changes, as determined during geological logging (i.e., by a geologist). As target mineralization / rock type would typically be visible to the naked eye – chalcopyrite for the Maven and pegmatite for the CV Trend – a protocol was set whereby the sampling could continue at least 10 m on either side of the visually identified mineralized zone, with the geologist able to extend sampling at their discretion. Sample lengths targeted 1.0 m within a mineralized zone and were extended to 1.5 m outside of mineralized zones. If target mineralized sample zones were interfingered with interpreted unmineralized zones over short intervals, the entire section was sampled. All pegmatite encountered in the drill hole was sampled, irrespective of perceived mineralization.

Samples that were marked were cut in half using a core saw, with one half collected for analysis and the other half remaining in the core box for reference. Where a duplicate sample was indicated, the half core remaining in the box was cut in half again, producing two (2) quarter-core pieces with one collected for analysis and the other remaining in the core box for reference. In addition to quarter-core duplicates, the QA/QC program included systematic insertion of quartz blanks and certified reference materials.

Samples collected for analysis were placed in a labelled heavy-duty plastic sample bag with the corresponding sample tag. The bags were closed with zip ties and catalogued before being packaged in labelled and sealed rice sacs, which were placed into a pallet-sized heavy-duty super-sacs (i.e., neoprene bulk bags), ready to be transported to the lab. The pallets of samples were loaded onto regularly scheduled truck shipments from Mirage Lodge by Kepa Transport and transported by ground to Activation Laboratories in Ancaster, Ontario. Samples were tracked during shipment along with the chain of custody documentation.

Upon arrival at the laboratory, the samples were cross-referenced with the shipping manifest to confirm all samples were accounted for and had not been tampered with.

All unsampled 2021 drill core remaining in the core boxes was either flown back to the Property for long-term storage (cross-stacked) or placed in long-term storage at the Company's Camp Shaakichiuwaanaan. All analytical reject and pulp material is currently stored at Camp Shaakichiuwaanaan.

10.5.2 2022, 2023, 2024, & 2025 Drill Programs

For the 2022, 2023, 2024 and 2025 drilling programs, current as of the date of this Report, the protocols outlined for the 2021 drilling program were continued with only minor adjustments and refinements. Additionally, all drilling in 2022, 2023, 2024 and 2025 focused solely on LCT pegmatite, with no base or precious metal targets drill tested.

Core sample collection was guided by lithology, mineralogy, and textural changes, as determined during geological logging (i.e., by a geologist). All pegmatite intervals were sampled in their entirety (half-core), whether spodumene / pollucite mineralization was noted or not (to ensure an unbiased sampling approach), in addition to ~1 m to 3 m of sampling into the adjacent host rock (dependent on pegmatite interval length) to "shoulder" the sampled pegmatite. The geologist may extend this shoulder distance at their discretion based on logging observations. If target mineralized sample zones were interfingered with interpreted unmineralized zones over short intervals, the entire section was sampled. All pegmatite encountered in the drill hole was sampled, irrespective of perceived mineralization.

The targeted minimum individual sample length was typically 0.3 m to 0.5 m, and the maximum sample length was typically 2.0 m. Targeted individual pegmatite sample lengths are 1.0 m to 1.5 m. Additionally, samples of the host, non-pegmatite rock unit(s) were collected at systematic intervals (one (1) sample every ~20 m) throughout the hole, in addition to samples of interest as determined by the logging geologist.

All sample-marked drill core was saw-cut using an Almonte automatic core saw, with one (1) half-core collected for assay, and the other half-core remaining in the box for reference. Where a duplicate sample was indicated (collected through only hole CV23-365), the half core remaining in the box was cut in half again, producing two (2) quarter-core pieces with one collected for analysis and the other remaining in the core box for reference. In addition to quarter-core duplicates, the QA/QC program included systematic insertion of quartz blanks and certified reference materials.

A new addition to the protocol in 2022 was the systematic collection of specific gravity (SG) measurements using the water immersion method. SG measurements were collected for the entire half-core sample interval at a rate of approximately one sample every 4 m to 6 m and over each rock type encountered.

Samples collected for analysis were placed in a labelled heavy-duty plastic sample bag with the corresponding sample tag. The bags were closed with zip ties and catalogued before being packaged in labelled and sealed rice sacs, which were placed into a pallet-sized heavy-duty super-sacs (i.e., neoprene bulk bags), ready to be transported to the lab.

For the 2022 drill core, the pallets of samples were loaded onto regularly scheduled truck shipments from Mirage Lodge, by third-party service provider Kepa Transport, and transported by ground to SGS Canada Laboratories in Lakefield, Ontario (vast majority), Sudbury, Ontario (CV22-028, 029, 030), or Burnaby, British Columbia (CV22-031, 032, 033, and 034). Samples were tracked during shipment along with the chain of custody documentation. Upon arrival at the laboratory, the samples were cross-referenced with the shipping manifest to confirm all samples were accounted for and had not been tampered with.

For the 2023 drill core, the pallets of samples were shipped ‘on-demand’ by ground transport, by the drill contractor (Forage Fusion Drilling), directly to SGS Canada’s laboratory in Lakefield, Ontario (CV23-105, 106 and 107), and Val d’Or, Québec (CV23-108 through 365). Samples were tracked during shipment along with the chain of custody documentation. Upon arrival at the laboratory, the samples were cross-referenced with the shipping manifest to confirm all samples were accounted for and had not been tampered with.

For the 2024 and 2025 drill core, the pallets of samples were shipped ‘on-demand’ by ground transport, by the drill contractor (Forage Fusion Drilling), directly to SGS Canada’s laboratory in Radisson, Québec, or Val d’Or, Québec. Samples were tracked during shipment along with the chain of custody documentation. Upon arrival at the laboratory, the samples were cross-referenced with the shipping manifest to confirm all samples were accounted for and had not been tampered with.

All unsampled drill core remaining in the core boxes is in storage at Camp Shaakichiuwaanaan on the Property. All analytical rejects and pulp material are currently in temporary storage at SGS Canada’s lab facilities or in long-term storage at the Camp Shaakichiuwaanaan on the Property.

10.6 Qualified Person's Opinion

The QP is of the opinion that the drilling and logging procedures and protocols employed by the Company meet acceptable industry standards and are sufficient to support geological and Mineral Resource modelling.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

11.1.1 Rock and Channel Sampling Programs

Channel sampling followed best industry practices with a 3 cm to 5 cm wide, saw-cut channel completed across the pegmatite as practically possible, perpendicular to the interpreted pegmatite strike. Samples were collected at 0.5 m to ~1 m contiguous intervals with the channel bearing noted, and GPS coordinates (RTK tool) were collected at the start and end points of the channel.

The rock type and mineralogy of each channel sample were logged on site at the time of collection. Channel samples were not geotechnically logged by nature; however, channel recovery was effectively 100%.

All rock and channel samples collected for analysis (2022, 2023, and 2024) were placed in a labelled heavy-duty plastic sample bag with the corresponding sample tag and closed with zip ties. Samples were transported by road or helicopter to Camp Shaakichiuwaanaan or Mirage Lodge, catalogued, and packaged in labelled and sealed rice sacks for transport to the analytical laboratory. Samples were then shipped directly from Camp Shaakichiuwaanaan or Mirage Lodge to SGS Canada's laboratory in Lakefield, ON (2022), or Val-d'Or, QC (2023 and 2024), using a third-party (2022) or dedicated service provider contracted by the Company (2023 and 2024). The Company largely relied on internal laboratory quality assurance / quality control (QA/QC) for its surface rock samples; however, the occasional certified reference material (CRM) and blank were submitted with sample batches. For the 2023-2024 channel samples, a protocol was followed, which included systematic insertion of blanks and CRMs in sample batches submitted to the laboratory.

Upon receipt at the SGS Canada laboratory (from spring 2023 onwards), each sample was sorted and catalogued. An updated standard drill core sample preparation was then completed, which included drying at 105°C, crushing to 90% passing 2 mm (instead of the prior 75% passing 2 mm), riffle split 250 g, and pulverizing 85% passing 75 microns (package PRP89).

SGS Canada laboratory used, during the surface exploration, programs that have the relevant accreditations (ISO 17025) and that are independent of the Company.

11.1.2 2024 Drilling Program (April 2024 to December 2024, Holes CV24-527 to CV24-787)

Core samples collected from the 2024 drill holes completed subsequent to those included in the 2024 MRE (i.e., CV24-527 to CV24-787) were shipped to SGS Canada's laboratory in Val-d'Or, Québec, or Radisson, Québec, for sample analysis preparation.

Upon receipt at the laboratory, each sample was sorted and catalogued. An updated standard drill core sample preparation was then completed, which included drying at 105°C, crushing to 90% passing 2 mm (instead of the prior 75% passing 2 mm), riffle split 250 g, and pulverizing 85% passing 75 microns (package PRP89).

The primary laboratory (SGS Canada) used for the 2024 core analysis is a commercial laboratory with the relevant accreditations (ISO 17025) and is independent of the Company.

11.2 Analytical Procedure

11.2.1 2024 Drill Program

Subsequent to the 2024 MRE, all the 2024 drilling program core sample pulps were shipped by air, from SGS Canada's Val-d'Or, Québec, or Radisson, Québec, preparation facility to SGS Canada's laboratory in Burnaby, British Columbia, where the samples were homogenized and subsequently analyzed for multi-element (including Li and Ta) using sodium peroxide fusion with ICP-AES/MS finish (codes GE_ICP91A50 and GE_IMS91A50). The analytical package had a relatively high detection limit for Li (5%), so overlimit analyses were not required. Overlimits, where requested, were submitted for analysis by acid digestion for alkaline metals with AAS[1] finish (code GC_AAS49C) for Cs and Rb, borate fusion with XRF finish (code GC_XRF76V) for Cs and Ta, and sodium peroxide fusion with ICP-MS finish (Code GC_IMS93A50V) for Rb.

The primary laboratory (SGS Canada) used for the 2024 core analysis is a commercial laboratory with the relevant accreditations (ISO 17025) and is independent of the Company and vendor (SGS, 2022).

11.2.2 Previous Years

Analytical procedures and sample preparation methods used in previous years' drilling programs are described in detail in the two (2) previous MRE technical reports (McCracken & Cunningham, 2023);

[1]AAS: Atomic Absorption Spectroscopy

(McCracken, et al., 2024); both are similar to the current methodologies. The main difference was for the 2021 program, when samples were submitted to and analyzed by Activation Laboratories in Ancaster, Ontario, and underwent their own procedures. For 2021, the samples were analyzed for multi-element (including lithium) by four-acid digestion with ICP-OES finish (package 1F2) or by sodium peroxide fusion with ICP-OES / ICP-MS finish (package UT7). Any samples returning >8,000 ppm Li by 1F2 were reanalyzed for Li by code 8-4 Acid ICP Assay. Additionally, all samples were analyzed for tantalum by INAA[2] (code 5B). Where Au was requested, it was determined by fire assay (package 1A2B-30).

11.3 Quality Assurance / Quality Control

11.3.1 Channel Sample Program

A total of 430 samples, totalling 218.4 m, were collected from channel samples during the 2024 program and included in the 2025 MRE. A protocol was followed, which included systematic insertion of blanks and CRMs in sample batches submitted to the laboratory.

11.3.2 2021 to 2024 Drilling Programs

Complete procedures and QA/QC protocols are described in detail in previous technical reports (McCracken & Cunningham, NI 43-101 Technical Report for Patriot Battery Metals. Mineral Resource Estimate for the CV5 Pegmatite, Corvette Property, 2023); (McCracken, et al., 2024). Industry's best practices have been applied to the Project from the beginning with insertion of blanks, CRM, quarter-core duplicates, pulp duplicates, and external pulp duplicates. In 2021, the laboratory was Activation Laboratories in Ancaster, Ontario. For subsequent programs, starting in 2022, SGS Canada has been the primary laboratory, and ALS Canada is used as the secondary laboratory for external checks. Table 11.1 shows the different QA/QC insertions for drilling programs from 2021 to 2024.

Table 11.1: QA/QC Insertions for Drilling Programs from 2021 to 2024

Drilling Program	CRM	Blank	Pulp Duplicate	External Pulp Duplicate	Quarter Core	Reject Duplicate
2021	70	-	4	164	-	-
2022	375	374	395	393	331	-
2023	341	338	357	357	304	-
2023 - 2024	966	1,042	1,116	1,115	-	494

[2] INAA: Instrumental Neutron Activation Analysis

11.3.3 2024 Drilling Program (Holes CV24-527 to CV24-787)

SGS Canada implements routine QA/QC protocols during internal analysis. These are routine procedures that consist of using pulp duplicates for repeat analysis and internal CRMs.

In addition to the standard internal laboratory QA/QC, the Company implemented a QA/QC protocol, following industry best practices, into the program. This protocol included systematic insertion of quartz blanks and CRMs into sample batches. Additionally, analysis of pulp-split and coarse-split (through hole CV23-365 only) sample duplicates was completed at the primary laboratory (SGS Canada) to assess analytical precision at different stages of the laboratory preparation process, and pulp-split duplicates were prepared at the primary laboratory for subsequent check analysis and validation at an external (secondary) laboratory (ALS Canada).

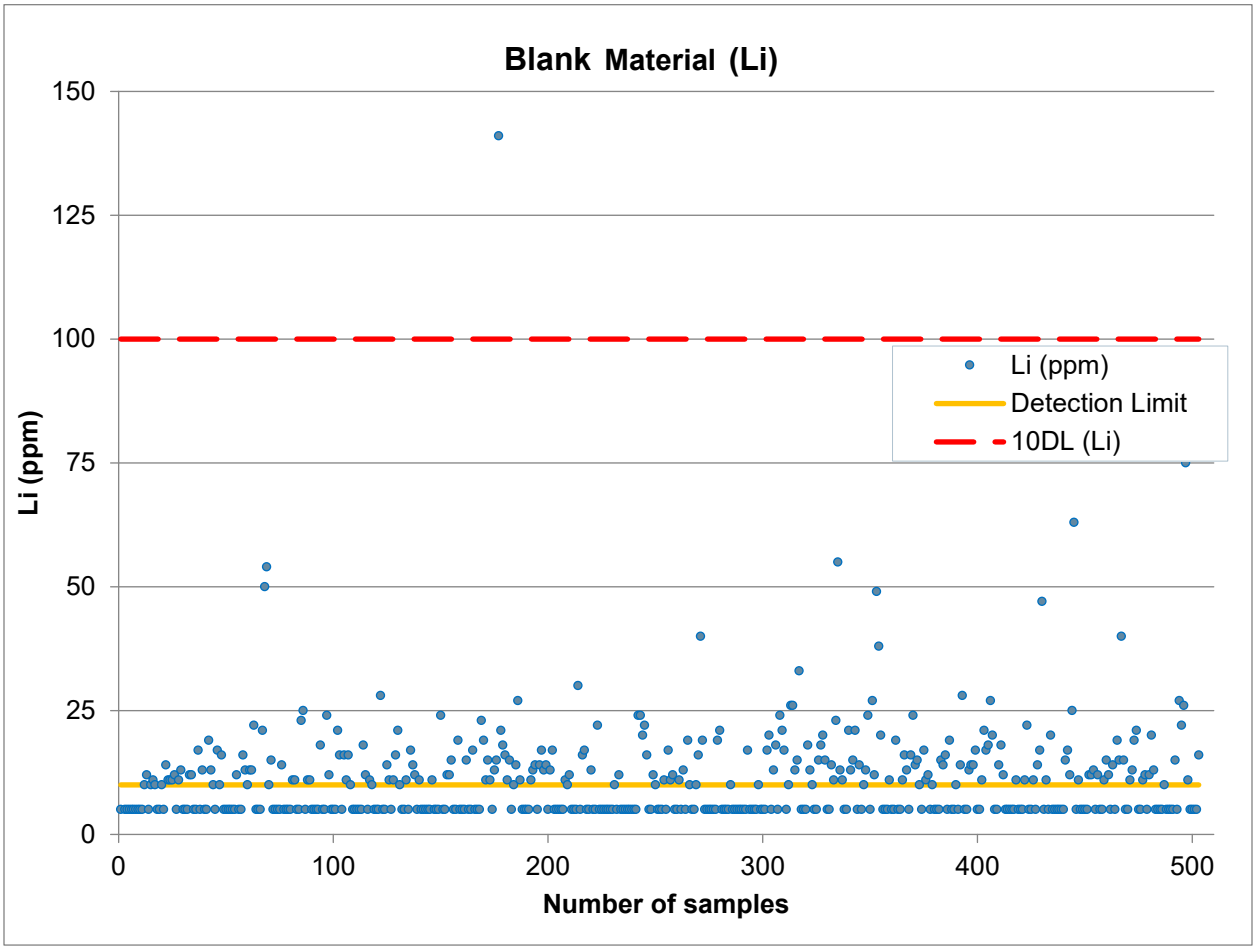
Throughout the program, the Company followed the same QA/QC protocols in place from the prior program, with the exception of quarter-core and course-split duplicates no longer being collected after CV23-190 and CV23-365, respectively. A review of the existing data set determined that the quarter-core and coarse-split duplicates were no longer required as part of its QA/QC protocols.

11.3.3.1 Blanks

Blanks consisted of an approximate 0.4 kg to 0.5 kg sample of ‘coarse silica blank material’ from OREAS, at a size of approximately 0.5 cm to 1 cm per piece silica blank. A total of 503 quartz blanks were submitted as control samples over the 2024 drilling program.

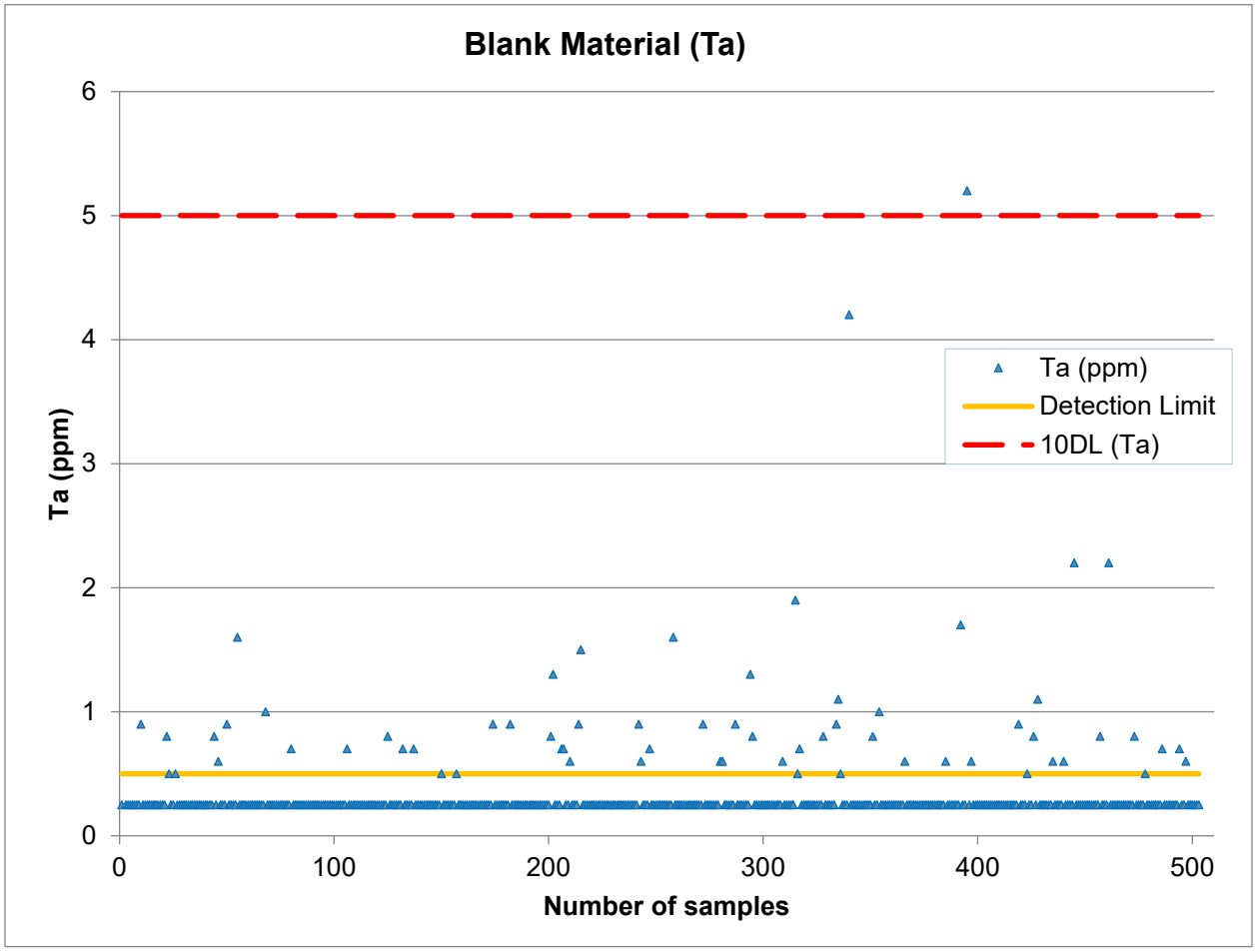
For lithium (Figure 11.1), with a lower detection limit of 10 ppm with the GE_ICP91A50 method, the trend was around 11 ppm for the blank. At 3x the lower detection limit (30 ppm), a warning was issued, and at 10x the detection limit (100 ppm), the assay failed. When an assay failed, a request to re-assay five (5) samples before and after was made to the laboratory. For this drilling program, one (1) sample failed the lithium 10x detection limit threshold.

Figure 11.1: Blank Sample Results (Li) from the 2024 Drilling Campaign



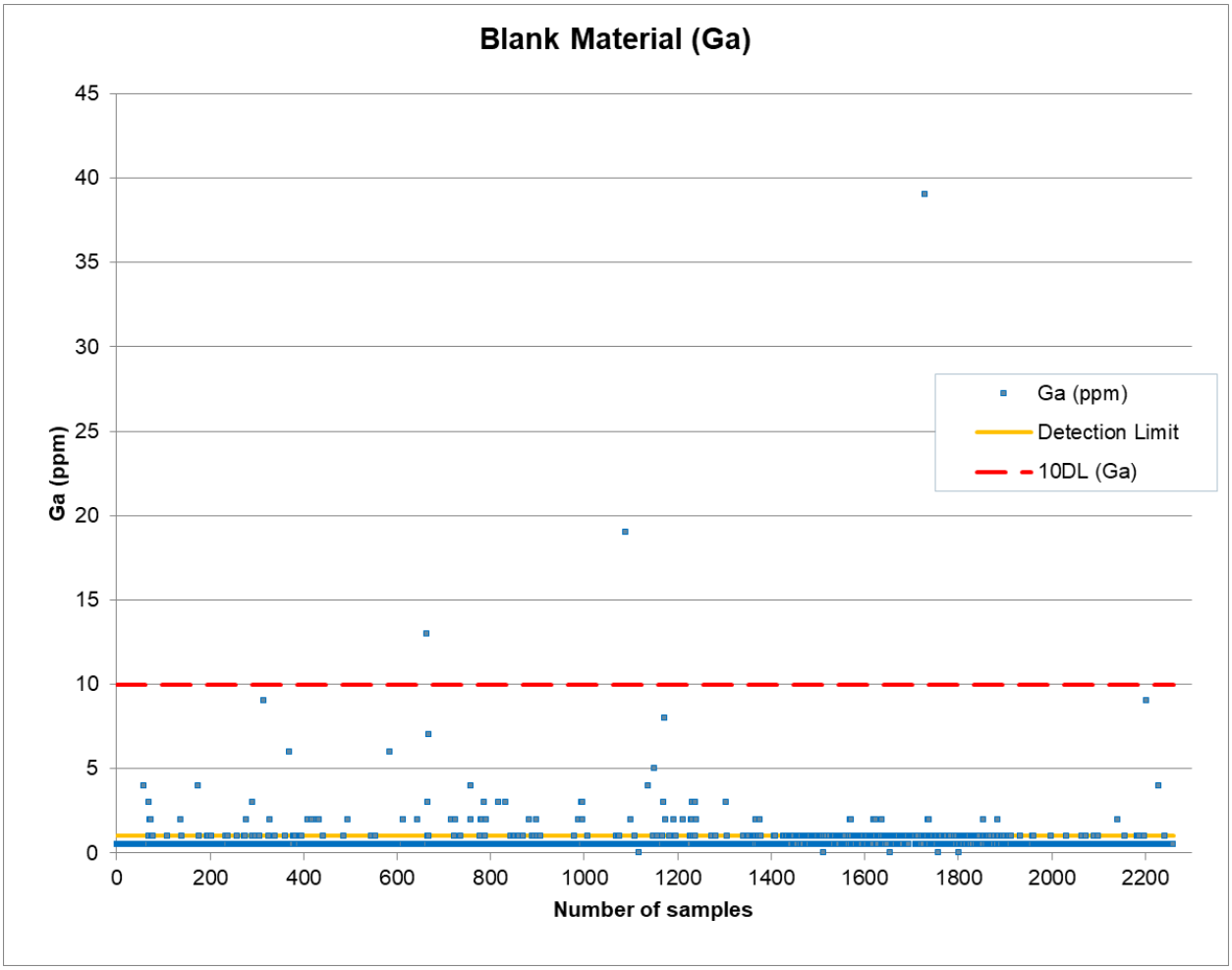
For tantalum with a detection limit of 0.5 ppm with the GE_IMS91A50 method, the decision was to set the failed values at 10 times the detection limit (5 ppm). When an assay failed, a request to re-assay five (5) samples before and after was made to the laboratory. For this drilling program, one (1) sample failed the tantalum 10x detection limit threshold.

Figure 11.2: Blank Sample Results (Ta) from the 2024 Drilling Campaign



Since gallium is being estimated for the first time in this current Technical Report, the blank sample results presented are from the 2022 to 2024 drilling programs (Figure 11.3). For gallium with a detection limit of 1 ppm with the GE_IMS91A50 method, the decision was to set the failed values at 10 times the detection limit (10 ppm). For the 2022 to 2024 drilling programs, only three (3) samples failed the gallium 10x detection limit threshold on a total of 2,260.

Figure 11.3: Blank Sample Results (Ga) from 2022-2024 Drilling Campaigns



The blank material was not appropriate to be used for QA/QC protocols for caesium, as approximately 20% of the blanks exceeded 10x the detection limit (1 ppm). An adjustment to the threshold for caesium should be considered.

11.3.3.2 Certified Reference Materials

Several CRMs were used during the 2024 drilling program – AMIS0342, AMIS0355, AMIS0565, OREAS 751, OREAS 752 and OREAS 753 (Table 11.2). In its procedure, PMET Resources' geologists set a warning when an element was ± 2 standard deviations (Std Dev.), and the element was considered failed if above ± 3 standard deviations. Failing rate is minimal for the 2024 drilling program.

Since gallium and caesium are being estimated for the first time on the Project in this current Technical Report, the CRM results from the 2022 to 2024 drilling programs were reviewed and are presented in Table 11.3.

Table 11.2: Certified Reference Materials Used in the 2024 Drilling Program

Lithium (Li)									
Standard (CRM)	**Standard Supplier**	**Laboratory**	**Certified Li Value (ppm)**	**Quantity Inserted**	**Mean Grade (Li ppm)**	**Lower Process Limit (CRM - 3SD)**	**Upper Process Limit (CRM - 3SD)**	**Failed (Outliers)**	**(%) Passing Quality Control**
AMIS0342	AMIS	SGS	1,612	55	1,710	1,362	1,863	0	100.0
AMIS0355	AMIS	SGS	7,268	165	7,511	6,014	8,522	0	100.0
AMIS0565	AMIS	SGS	5,424	105	5,579	4,722	6,126	0	100.0
Oreas751	OREAS	SGS	4,680	52	4,661	4,170	5,190	0	100.0
Oreas752	OREAS	SGS	7,070	45	7,075	6,440	7,700	0	100.0
Oreas753	OREAS	SGS	10,200	37	10,032	9,510	10,890	0	100.0
Total				**459**				**0**	**100.0**
Tantalum (Ta)									
Standard (CRM)	**Standard Supplier**	**Laboratory**	**Certified Ta Value (ppm)**	**Quantity Inserted**	**Mean Grade (Ta ppm)**	**Lower Process Limit (CRM - 3SD)**	**Upper Process Limit (CRM - 3SD)**	**Failed (Outliers)**	**(%) Passing Quality Control**
AMIS0342	AMIS	SGS	169	55	171	118	220	0	100.0
AMIS0355	AMIS	SGS	214	165	223	88	340	0	100.0
AMIS0565	AMIS	SGS	46	105	46	4	88	0	100.0
Oreas751	OREAS	SGS	28	52	27	18	38	0	100.0
Oreas752	OREAS	SGS	41	45	40	36	46	1	97.8
Oreas753	OREAS	SGS	20	37	20	16	24	0	100.0
Total				**459**				**1**	**99.8**

Gallium (Ga)									
Standard (CRM)	**Standard Supplier**	**Laboratory**	**Certified Ga Value (ppm)**	**Quantity Inserted**	**Mean Grade (Ga ppm)**	**Lower Process Limit (CRM - 3SD)**	**Upper Process Limit (CRM - 3SD)**	**Failed (Outliers)**	**(%) Passing Quality Control**
AMIS0342	AMIS	SGS	-	-	-	-	-	-	-
AMIS0355	AMIS	SGS	75	165	74	66	84	0	100.0
AMIS0565	AMIS	SGS	44	105	43	37	52	1	99.0
Oreas751	OREAS	SGS	19	52	19	15	23	0	100.0
Oreas752	OREAS	SGS	18	45	18	15	21	0	100.0
Oreas753	OREAS	SGS	17	37	16	13	20	0	100.0
Total				**404**				**1**	**99.8**
Caesium (Cs)									
Standard (CRM)	**Standard Supplier**	**Laboratory**	**Certified Cs Value (ppm)**	**Quantity Inserted**	**Mean Grade (Cs ppm)**	**Lower Process Limit (CRM - 3SD)**	**Upper Process Limit (CRM - 3SD)**	**Failed (Outliers)**	**(%) Passing Quality Control**
AMIS0342	AMIS	SGS	-	-	-	-	-	-	-
AMIS0355	AMIS	SGS	259.0	165	266	231	288	3	98.2
AMIS0565	AMIS	SGS	-	-	-	-	-	-	-
Oreas751	OREAS	SGS	48.5	52	49	43	54	2	96.2
Oreas752	OREAS	SGS	66.0	45	68	55	77	0	100.0
Oreas753	OREAS	SGS	62.0	37	62	49	75	0	100.0
Total				**299**				**5.0**	**98.3**

Table 11.3: Certified Reference Materials Used in the 2022 to 2024 Drilling Programs for Ga and Cs

Gallium (Ga) 2022-2024									
Standard (CRM)	**Standard Supplier**	**Laboratory**	**Certified Ga Value (ppm)**	**Quantity Inserted**	**Mean Grade (Ga ppm)**	**Lower Process Limit (CRM - 3SD)**	**Upper Process Limit (CRM - 3SD)**	**Failed (Outliers)**	**(%) Passing Quality Control**
AMIS0342	AMIS	SGS	-	645	-	-	-	-	-
AMIS0355	AMIS	SGS	75	649	73	66	84	22	96.6
AMIS0565	AMIS	SGS	44	666	42	37	52	9	98.6
Oreas148	OREAS	SGS	29	13	26	25	34	1	92.3
Oreas751	OREAS	SGS	19	52	19	15	23	0	100.0
Oreas752	OREAS	SGS	18	45	18	15	21	0	100.0
Oreas753	OREAS	SGS	17	37	16	13	20	0	100.0
Total			**0**	**2,107**				**32**	**98.5**
Caesium (Cs) 2022-2024									
Standard (CRM)	**Standard Supplier**	**Laboratory**	**Certified Cs Value (ppm)**	**Quantity Inserted**	**Mean Grade (Cs ppm)**	**Lower Process Limit (CRM - 3SD)**	**Upper Process Limit (CRM - 3SD)**	**Failed (Outliers)**	**(%) Passing Quality Control**
AMIS0342	AMIS	SGS	-	645	-	-	-	-	-
AMIS0355	AMIS	SGS	259.0	649	266	231	288	24	96.3
AMIS0565	AMIS	SGS	-	666	-	-	-	-	-
Oreas148	OREAS	SGS	311	13	298	272	350	0	100.0
Oreas751	OREAS	SGS	48.5	52	49	43	54	2	96.2
Oreas752	OREAS	SGS	66.0	45	68	55	77	0	100.0
Oreas753	OREAS	SGS	62.0	37	62	49	75	0	100.0
Total				**2,107**				**26**	**98.8**

As no commercial CRM or standard for caesium at near percent levels could be located, the Company has relied upon a combination of internal laboratory standards, pulp duplicates, and external pulp duplicates to assess the analytical accuracy of Cs geochemical analysis from the primary laboratory (SGS Canada). To further build upon this QA/QC approach, in 2025, the Company created its own internal standards using core samples from the Vega Caesium Zone. These internal standards (VCS-001 and VCS-002) were implemented starting June 2025 as part of its QA/QC protocol, where caesium-rich zones are anticipated based on core logging and geological modelling.

Further, the Company has completed Rietveld XRD and/or TIMA mineralogical analysis on more than 140 core samples from the CV13 Pegmatite targeting the Rigel and Vega caesium zones. The result of these analyses confirms pollucite as the dominant caesium-bearing mineral where the corresponding sample geochemical analysis is >0.5% Cs_2O. Further, the total estimated Cs content based upon the abundance of pollucite and other Cs-bearing minerals, as determined by mineralogical analysis, reconciles well with the Cs content determined by geochemical analysis at the primary laboratory (SGS Canada).

11.3.3.3 Pulp Duplicates

A total of 561 pulp duplicates were collected from the 2024 drilling program (Figure 11.4 and Figure 11.5). Since gallium and caesium are being estimated for the first time on the Project in this current Technical Report, the pulp duplicate results from the 2022 to 2024 drilling programs were reviewed and are shown in Figure 11.6 and Figure 11.7. The total number of pulp duplicates from 2022 to 2024 is 2,429.

Figure 11.4: Pulp Duplicates (Li) for the 2024 Program

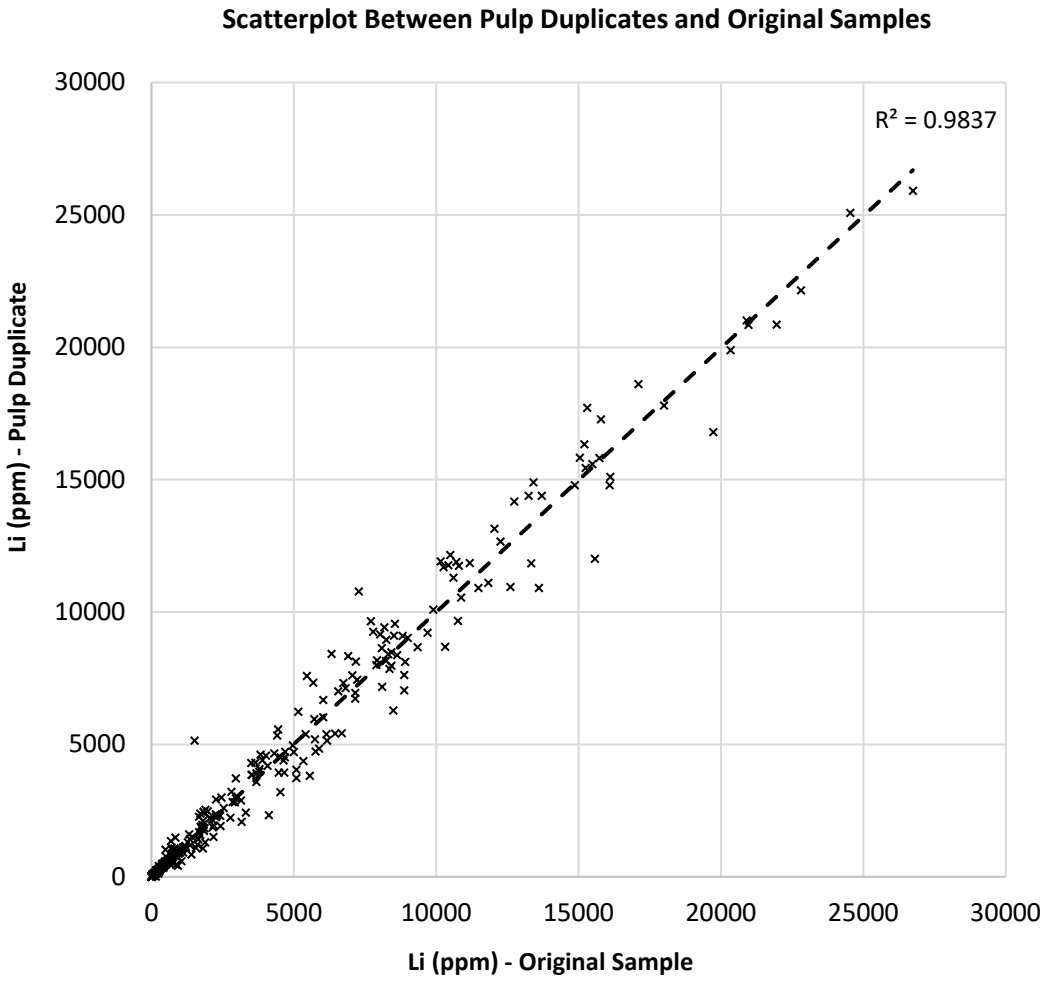


Figure 11.5: Pulp Duplicates (Ta) for the 2024 Program

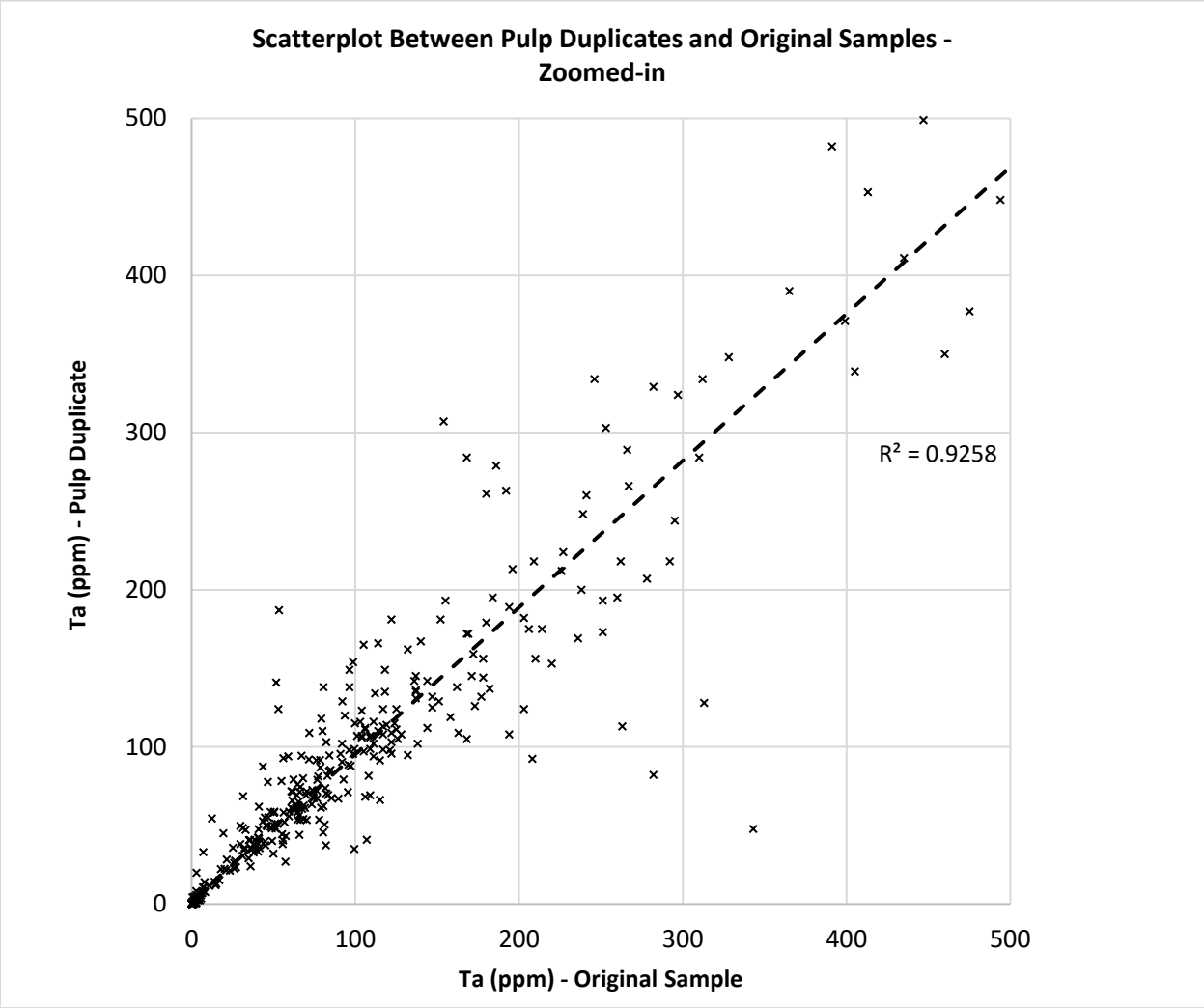


Figure 11.6: Pulp Duplicates (Ga) for the 2022 to 2024 Programs

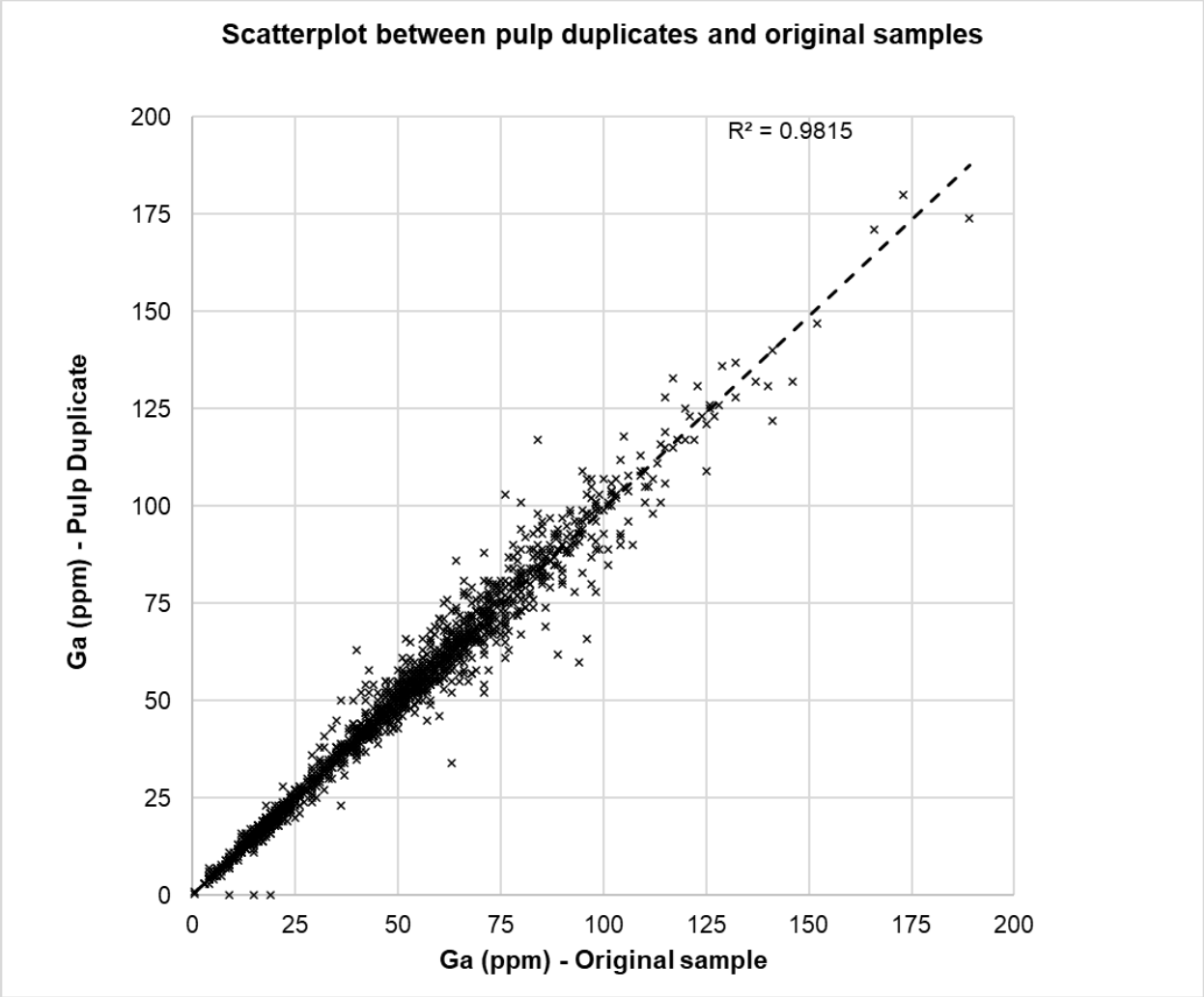
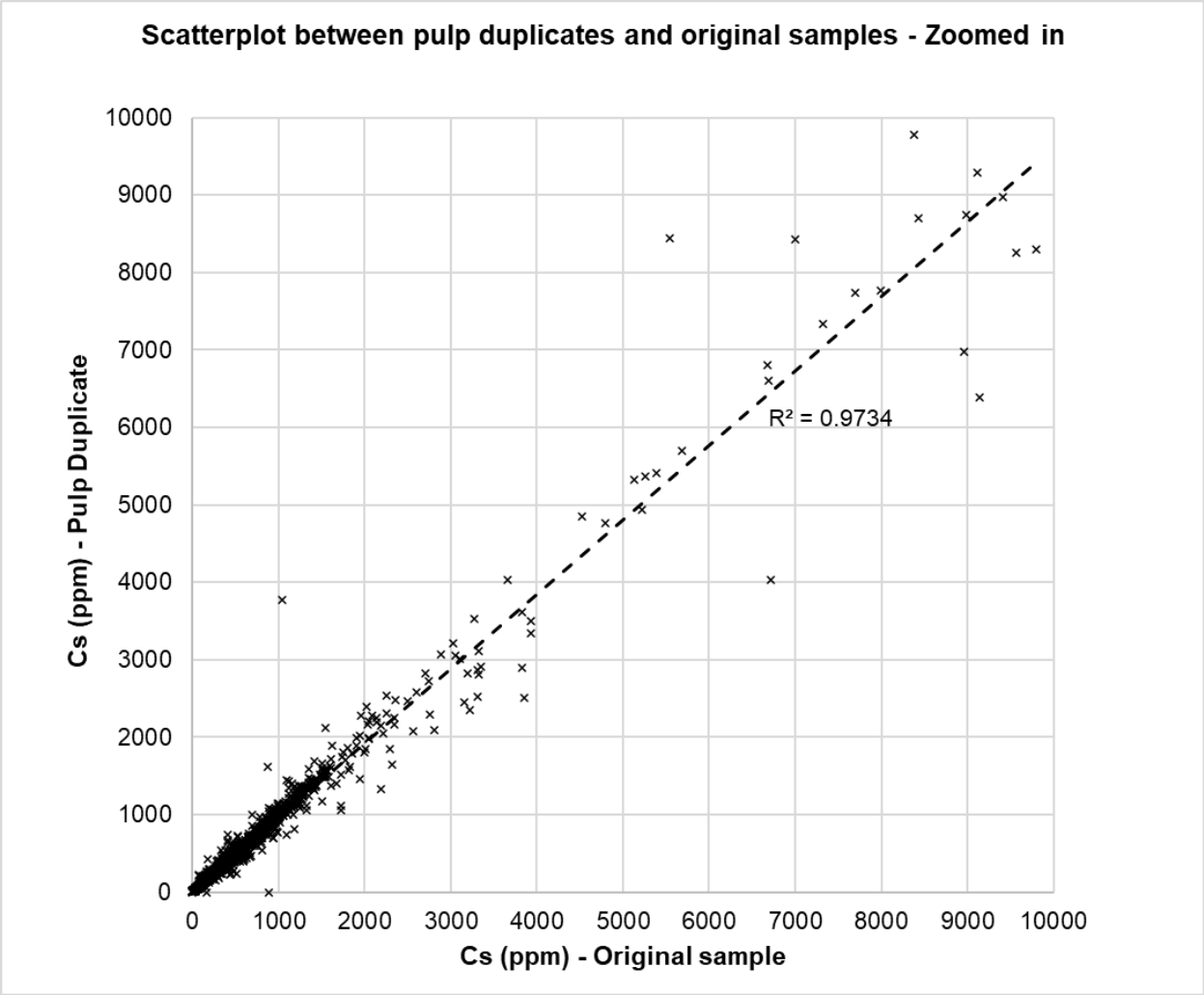


Figure 11.7: Pulp Duplicates (Cs) for the 2022 to 2024 Programs for Assays Less than 10,000 ppm



For the four (4) elements, the coefficient of determination is good, which shows a good reproducibility between the original samples and the duplicates.

11.3.3.4 External Pulp Duplicates (Secondary Lab Check)

A total of 561 pulp-split duplicates, created at the primary laboratory (SGS Canada) from core samples collected from the 2024 drilling program (at the CV5 and CV13 pegmatites), were submitted for check analysis to ALS Canada's Vancouver, British Columbia, laboratory. Upon receipt at ALS Canada, the pulp samples were homogenized via manual sheet rolling (package ROL-21) and analyzed by ICP-MS following a sodium peroxide fusion (package ME-MS89L). The external pulp duplicate for Li, Ta and Cs is shown in Figure 11.8, Figure 11.9 and Figure 11.10. Gallium is not analyzed by ALS Canada in the external pulp duplicates.

Since caesium is being estimated for the first time on the Project in this current Technical Report, the external pulp duplicate results from the 2022 to 2024 drilling programs were reviewed for a total of 2,325 external pulp duplicates.

Figure 11.8: External Pulp Duplicates (Li) for the 2024 Program

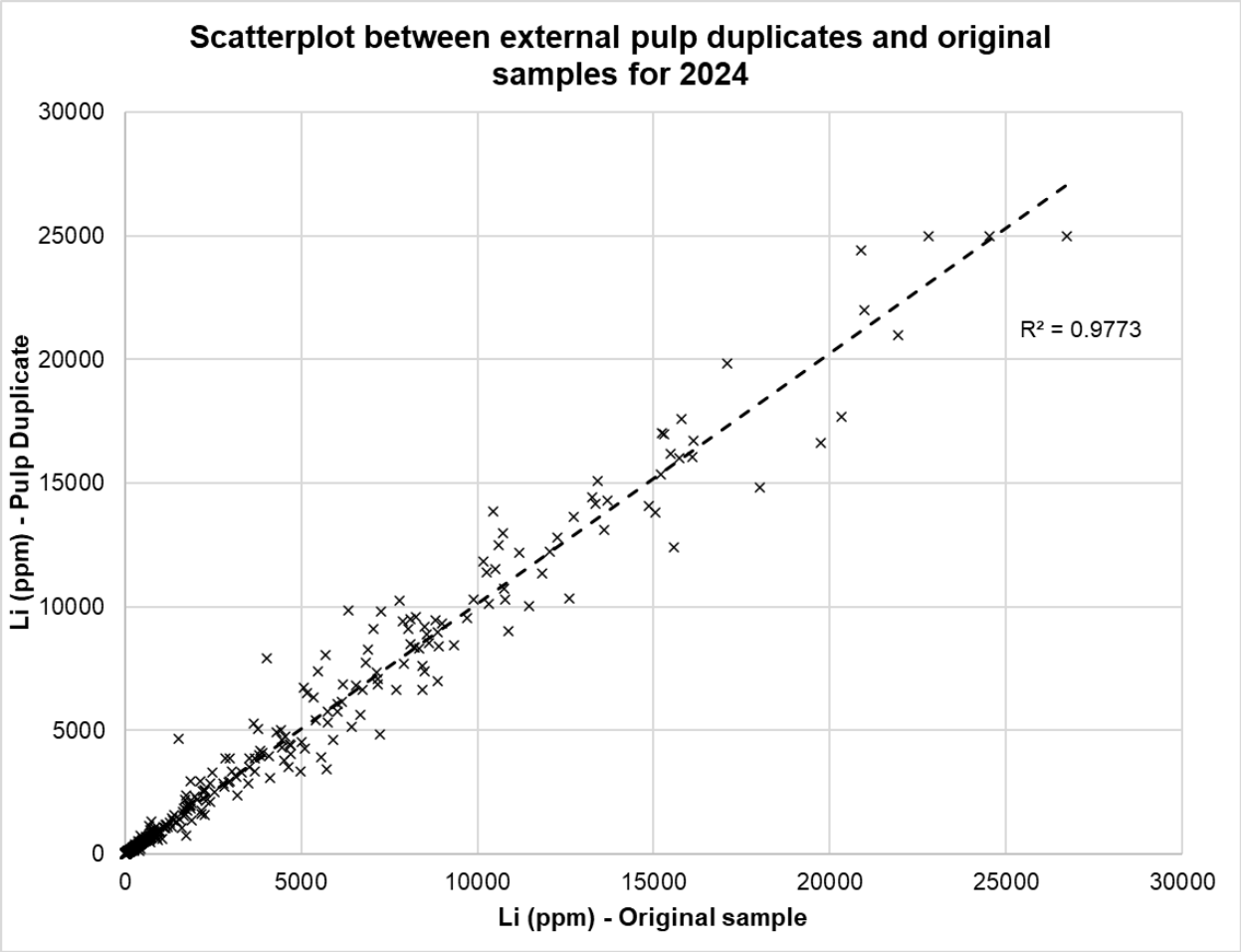


Figure 11.9: External Pulp Duplicates (Ta) for the 2024 Program

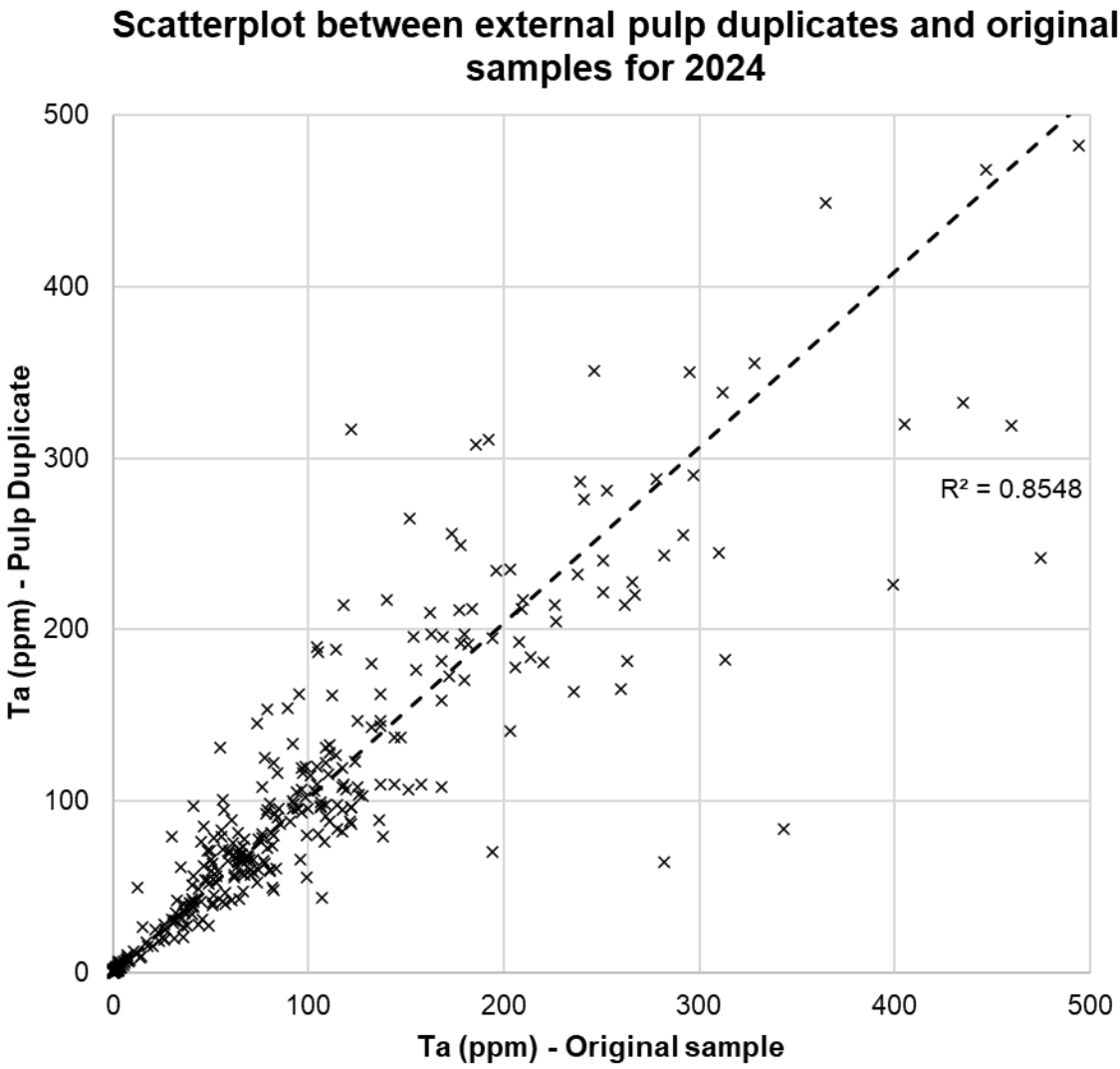
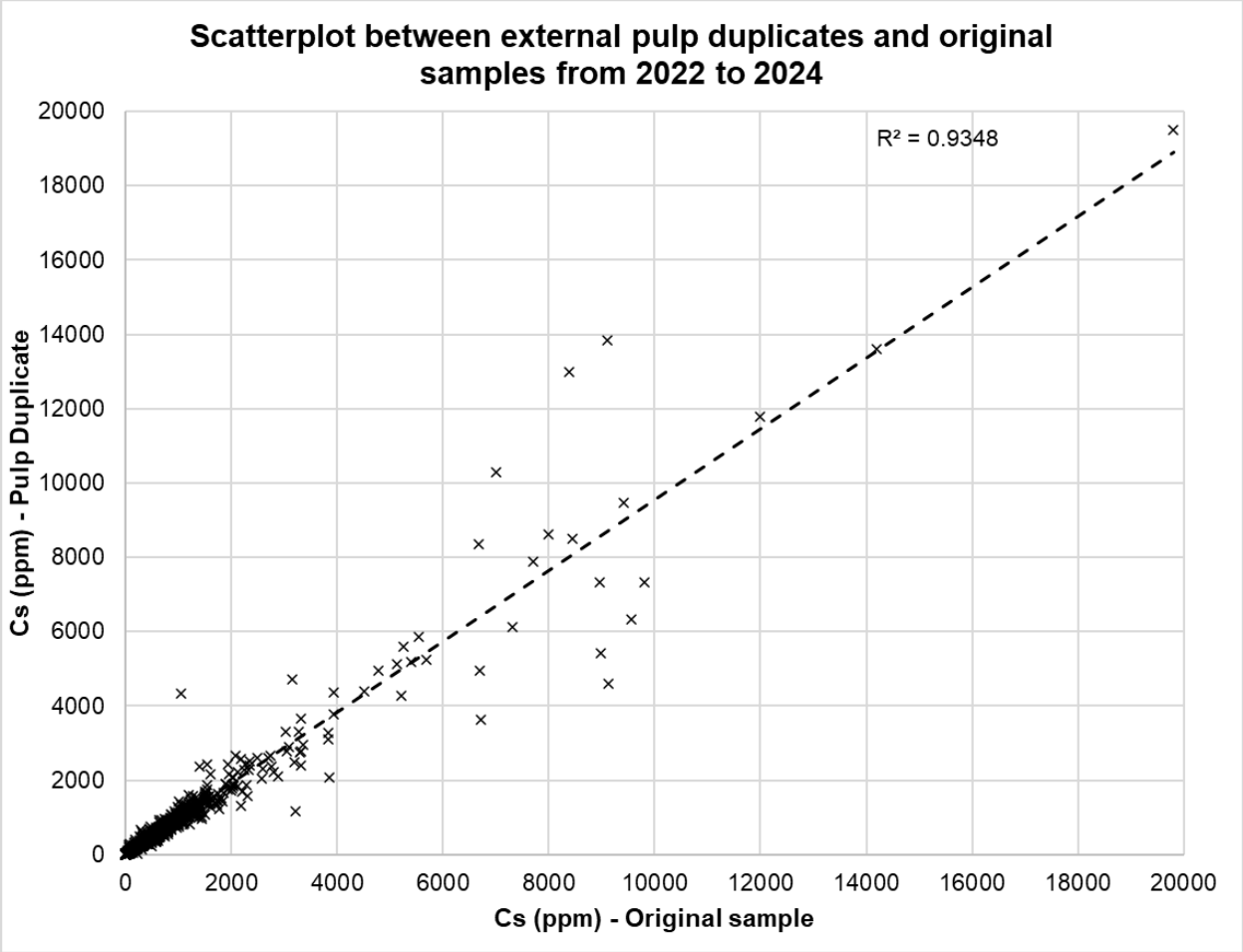


Figure 11.10: External Pulp Duplicates (Cs) for the 2022 to 2024 Programs (2,325 Samples; < 20,000 ppm)



For lithium and caesium, the coefficient of determination is good and shows excellent reproducibility between both laboratories. For tantalum, the correlation between both laboratories is acceptable with a positive bias to the secondary laboratory.

11.4 Sample Security

The Company followed industry-standard chain-of-custody methods and approaches for the 2024 core samples. Sample security and chain of custody for the drill core started with the removal of the core from the core barrel at the drill. Upon receipt of the core in the core shack, custody was transferred to the Company or its representatives for core processing. Once complete, drill core samples were shipped, typically weekly, by ground transport to SGS Canada's Val-d'Or, Québec, or Radisson, Québec, preparation facility.

All sample bags were catalogued upon receipt at the laboratory and cross-referenced with the Company's shipping manifest to ensure all samples had arrived. Additionally, upon receipt at the laboratory, all sample bags were assessed for signs of tampering.

The Company's sample security and chain of custody protocols included dates and a waybill / form documentation for each sample batch / shipment with respect to when they had departed the core shack area and when they had been received at the laboratory.

11.5 Qualified Person's Opinion

It is the QP's opinion that the sample preparation, security, and analytical procedures for channel and drill core sampling put in place by the Company meet acceptable industry standards and are sufficient to support geological and Mineral Resource modelling.

12. DATA VERIFICATION

12.1 Geology

12.1.1 Site Investigation

Mr. Todd McCracken, P.Geo. and QP, visited the Property from June 4 to 7, 2024, and previously from April 7 to 11, 2023. Mr. McCracken stayed at Mirage Lodge, which was the location of geological core logging facility at the time and visited the exploration camp (Shaakichiuwaanaan). Access to the Property was by chartered helicopter from camp.

Mr. McCracken examined the Project setting and outcrops and reviewed numerous drill collar sites and channels (Figure 12.1). CV13 and CV9 areas, which were not previously inspected in 2023, were examined as well as the CV5 area during the 2024 site visit.

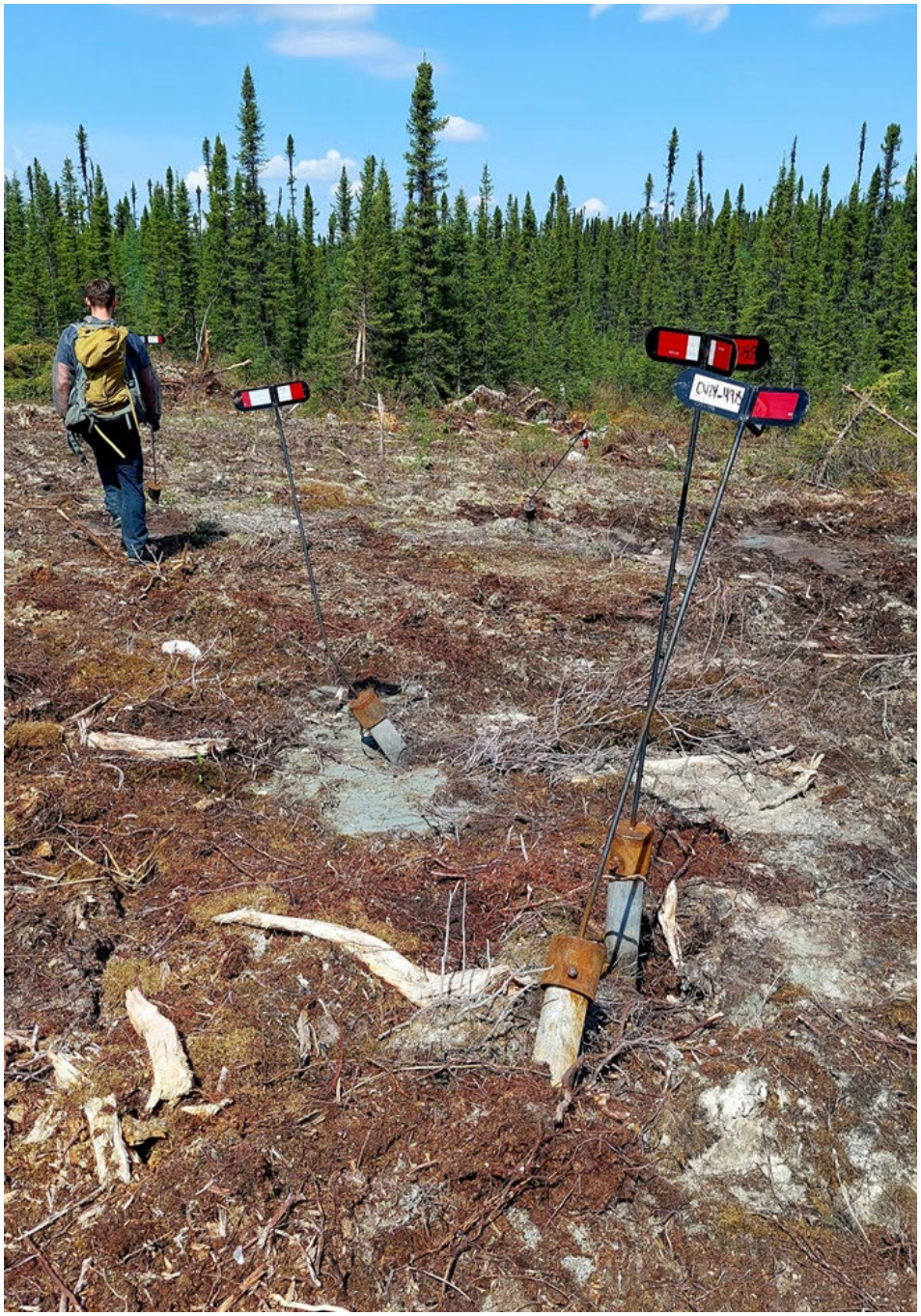
Figure 12.1: Channeled Outcrop Examined During the Site Visit



12.1.2 Drill Collar Validation

The QP confirmed the locations of 99 surface drill hole collars and eight (8) channel locations during the June 2024 site visit. The QP collected the collar locations using a handheld GPS unit (Figure 12.2). Less than 4% of the boreholes were outside the expected tolerance of a handheld GPS (±5 m).

Figure 12.2: Drill Collar Validation



12.1.3 Database Validation

The QP validated the digital database. The Company uses MX database software with scroll down menus that are integrated into Leapfrog Geo. The database was exported into a .CSV format to be validated by the QP.

Survey and collar data were verified. Assays were provided in PDF and CSV format direct from the analytical laboratory and were validated for Li, Cs, Ta, and Ga against the original assay certificates. No discrepancies between the Company's MX database and the original assays certificates were identified.

Due to the outcrops surface, some channel sample "collars" were moved vertically, usually less than 5 m, in Leapfrog so that the completed channel could be located below the topography.

12.1.4 Mining Validation

The mineralized material in the open pits and underground stopes design was estimated to validate the tonnage and grade between both models on in situ basis and were matching perfectly. The MRE block models were performed in Leapfrog, and the Mining block model was on Deswik software.

12.1.5 Qualified Person's Opinion

It is the QP's opinion that the data has been suitably validated. Mr. McCracken also believes that the sample database provided by the Company and validated by himself is suitable to support the MRE and engineering studies.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

PMET engaged Primero and SGS Canada in 2023 to assist with a metallurgical testwork program for the CV5 deposit. Testwork was completed at the SGS Lakefield Ontario facility. The scope of the program included both mineralogical characterization and metallurgical testwork. Both SGS and Primero are independent of the Company and are industry recognized in lithium pegmatite processing. The objectives of the metallurgical testwork program being to confirm the dominant lithium bearing mineral species for CV5 and evaluate the beneficiation performance of the deposit using a conventional spodumene DMS flowsheet. The target performance was the generation of a concentrate with a grade >5.5% Li_2O and <2.0% Fe_2O_3 while maximizing lithium recovery.

13.1.1 Testwork Overview

Testwork was carried out by SGS Canada at their Lakefield, Ontario, metallurgical testing facility. The testwork program was broken up into a series of projects summarized as the following:

- 19005-01 (SGS, 2023a) – Mineralogical characterization on 20 samples of drill core from the CV5 Pegmatite. Characterization work used a combination of TIMA-X (Quantitative SEM) Electron Probe Micro-Analysis (EPMA), Laser Ablation by Inductively Coupled Plasma Mass Spectrometry (LA by ICP-MS), X-ray diffraction (XRD) analysis, and chemical assays.
- 19005-02 (SGS, 2023b) – Initial dense media separation (DMS) testwork was completed on two (2) composites from CV5 Pegmatite. Heavy liquid separation (HLS) tests were done with material crushed to a top size of 6.35 mm and 9.5 mm respectively. Results from HLS results were used to determine specific gravity (SG) cut points for two DMS trails as well as a top size selection of 9.5 mm. Additionally four (4) bench scale flotation tests were completed on the DMS middlings and DMS bypass fraction (the -0.85 mm fraction) to identify future recovery opportunities for the Project.
- 19005-04 (SGS, 2023c) – Variability testing was completed using HLS and magnetic separation with a sample top size of 9.5 mm. A total of 11 variability composites sourced from CV5 were tested. Samples were chosen that exhibited a range of lithia (i.e., Li_2O) and iron (i.e., Fe_2O_3) grades. This work's focus was confirming a DMS-only flowsheet in the processing of CV5.
- 19005-06 (SGS, 2023d) – Semi-quantitative XRD and HLS testwork was completed on five (5) samples originating from the CV13 Pegmatite. This work was to investigate future opportunities in mining CV13.

- 19005-02A (SGS, 2024) – Variability testing was completed using HLS and magnetic separation with a sample top size of 9.5 mm and a bottom size of 0.60 mm. A composite sample representing the Project's starting open pit was tested via a two-size range (-9.5 mm to +3.3 mm and -3.3 mm to +0.60 mm) DMS and dry magnetic separation program. Eleven composites were tested from the CV5 Pegmatite and five (5) composites representing the host rock surrounding the CV5 Pegmatite. Size-by-size elemental assays as well as size-by-size HLS testwork were completed on the pegmatite samples to quantify the distribution / liberation nature of the spodumene. The host rock samples were analysed distinctly via HLS to assess and quantify where these external dilution materials would report to in DMS flowsheet.
- 19005-10 (SGS, 2025) – Size-by-size variability testing was completed on the CV5 deposit using HLS and magnetic separation for three (3) size fractions: coarse (9.5 mm to 3.3 mm), fine (3.3 mm to 1.7 mm), and ultrafine (1.7 mm to 0.60 mm), to quantify the size-by-size distribution and liberation characteristics of spodumene. Eight (8) composite samples were prepared for the program, four (4) representing open pit material and four (4) representing underground material, with each group including one composite-pegmatite, low-grade, high-grade, and dilution (host rock) sample. These were used to generate two (2) master composites: MC001, composed of the four (4) open pit composites, and MC002, composed of the four (4) underground composites, each blended in defined proportions. In addition to the size-by-size work, HLS was also conducted on re-crushed material at two (2) target sizes (-6.3 mm and -3.3 mm) to quantify the performance of a re-crush stream. Comminution tests were done to determine the Abrasion Index. DMS and magnetic separation tests were then completed on both MC001 and MC002 samples. The DMS concentrate underwent magnetic separation testing, the bypass underwent filtration testwork, the middlings and bypass streams were subjected to flotation testwork to assess the viability of a downstream wet plant, while the DMS magnetic rejects were evaluated using gravity separation to support potential tantalum recovery options.
- 17445-05 (SGS, 2025) – Standard bond ball mill grindability test and size distribution analysis on the DMS floats samples (CV5 Pegmatite) for paste plant design.
- Additionally, mineralogical analysis has continued to be completed in subsequent programs by SGS using XRD and TIMA methods. The mineralogical analysis has been completed on more than 550 samples (core composites, pulps, rejects) across the CV5 and CV13 pegmatites, including more than 120 samples from the Rigel and Vega caesium zones.
- 19005-07 (SGS, 2025c) – A DMS concentrate produced from the Zeppelin program's master drill core composite (CV5 Pegmatite) was advanced through a hydrometallurgical program to support flowsheet development for lithium hydroxide conversion. Steps included calcination, grinding, acid roasting, leaching, primary and secondary impurity removal, ion exchange, causticization, sodium removal, and final crystallization of lithium hydroxide monohydrate.

- 19005-14 (SGS, 2025b - Pending) – X-ray ore sorting testwork was completed on a drill core composite from the CV13 Pegmatite's Vega Caesium Zone (the "Saucer" program). As of the Issue Date of this Report, analytical results of the testwork have not yet been received.

13.1.2 Test Material

Approximately 880 kg of quarter-core NQ and 1,826 kg of half-core NQ of lithium-bearing pegmatite samples from CV5 have been used in the metallurgical test program. Additionally, 389 kg of half-core NQ samples, representing the host rock surrounding CV5, have been tested metallurgically. The length of drill core tested from CV5 Pegmatite sums up to approximately 1,136 m.

A list of the sources of samples is shown in Table 13.1, and a map of the location of each drill hole is shown in Figure 13.1.

Table 13.1: Sources of Samples for Metallurgical Programs Completed on CV5 Pegmatites

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
19005-01(1)	CF001-028	CF21-001	27	28	1	2.41	0.50	Mineralogical characterization
	CF001-089	CF21-001	80	81	1	2.17	0.76	
	CF001-103	CF21-001	91	92	1	2.37	2.27	
	CF001-119	CF21-001	106	107	1	0.02	0.15	
	CF001-129	CF21-001	114	115	1	0.02	0.10	
	CF001-148	CF21-001	131	132	1	1.08	0.48	
	CF001-177	CF21-001	157	158	1	0.43	0.90	
	CF001-188	CF21-001	166	167	1	3.53	0.29	
	CF001-227	CF21-001	203	204	1	5.23	0.85	
	CF001-229	CF21-001	204	205	1	1.59	0.33	
	CF002-047	CF21-002	105	106	1	0.00	0.07	
	CF002-094	CF21-002	145	146	1	1.85	0.27	
	CF002-099	CF21-002	150	151	1	0.24	0.36	
	CF002-110	CF21-002	159	160	1	3.85	0.39	
	CF002-150	CF21-002	195	196	1	1.23	0.85	
	CF002-177	CF21-002	218	219	1	1.03	0.39	
	CF002-188	CF21-002	227	228	1	2.05	0.86	

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
	CF004-020	CF21-004	48	49	1	2.17	0.49	
	CF004-043	CF21-004	69	70	1	2.67	0.50	
	CF004-051	CF21-004	76	77	1	0.17	2.89	
19005-02(1)	CF21-001Met	CF21-001	26	99	73	1.16	0.75	Initial mineral processing program (HLS, DMS, magnetic separation, flotation)
			142	173	31			
			200	213	14			
	CF21-002Met	CF21-002	79	132	53	1.05	0.31	
			145	179	34			
			189	203	14			
			209	233	24			
19005-04(1)	HLS-COMP-001	CV22-017	226	236	10	1.40	1.10	HLS screening across CV5
	HLS-COMP-002	CV22-019	133	140	7	1.16	0.83	
			141	144	3			
	HLS-COMP-003	CV22-025	28	38	10	1.14	0.80	
	HLS-COMP-004	CV22-035	139	143	4	1.33	0.68	
			157	163	6			
	HLS-COMP-005	CV22-038	233	243	10	1.68	0.81	
	HLS-COMP-006	CV22-040	331	341	10	0.67	0.59	
	HLS-COMP-007	CV22-042	180	190	10	1.57	0.57	
	HLS-COMP-008	CV22-048	396	398	2	1.35	0.34	
			418	423	6			
			424	426	2			
	HLS-COMP-009	CV22-052	183	193	10	2.04	0.31	
	HLS-COMP-010	CV22-054	55	65	10	1.20	0.29	
	HLS-COMP-011	CV22-070	180	190	10	2.73	0.42	

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
19005-02A[(2)]	COMP-001	CV23-160A	64	75	11	1.38	0.63	Size-by-size HLS testing, these composites were used in a DMS testing. Sample designed to represent the pegmatite from the first 4 years of the Project.
		CV23-160A	92	101	9			
		CV23-160A	176	190	13			
		CV23-161	87	96	10			
		CV23-161	116	126	10			
		CV23-161	139	149	10			
		CV23-172	106	114	7			
		CV23-172	150	169	19			
		CV23-176	93	107	14			
		CV23-176	115	126	11			
		CV23-176	164	172	8			
		CV23-190	28	36	8			
		CV23-190	99	114	15			
		CV23-190	134	146	12			
	COMP-002	CV23-182	159	190	30	0.71	0.83	
	COMP-003	CV23-184	167	173	6	1.68	0.53	
		CV23-184	184	196	12			
		CV23-184	206	219	13			
	COMP-005	CV23-120	280	291	11	0.30	0.60	HLS Variability Testing
		CV23-120	307	310	4			
	COMP-006	CV23-130	202	219	17	1.01	1.22	
	COMP-007	CV23-120	253	268	14	0.43	0.94	
	COMP-008	CV23-189	227	234	7	0.05	0.90	
	COMP-009	CV23-140	335	339	4	0.75	3.06	
		CV23-115	314	318	4			
	COMP-010	CV23-134	128	140	11	0.16	0.37	
		CV23-134	143	147	4			
	COMP-011	CV23-182	97	110	13	0.14	0.89	
	COMP-012	CV23-120	310	320	10	0.22	2.24	
		CV23-160A	327	331	4			

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
19005-10	COMP-001	CV23-199	76	88	12	2.26	1.19	Size-by-size HLS testing for three (3) size fraction DMS (coarse, fine, ultrafine). HLS recrush testing at two (2) size fractions. Comminution testing. DMS testing. Magnetic separation testing. Filtration testing. Flotation testing. Gravity separation testing.
		CV23-199	148	155	7			
		CV23-199	165	180	15			
		CV23-241	153	164	11			
		CV23-241	167	176	8			
		CV23-241	188	195	7			
		CV24-377	189	200	11			
		CV24-377	210	220	11			
		CV24-377	236	243	6			
		CV24-414	119	137	19			
		CV24-414	142	151	9			
		CV24-414	159	166	7			
		CV24-503	167	178	12			
		CV24-503	217	232	14			
		CV24-503	238	247	10			
	COMP-002	CV23-230	47	81	34	1.38	0.78	
	COMP-003	CV24-467	160	169	9	3.66	1.15	
		CV24-467	184	195	11			
	COMP-005	CV22-066	187	197	10	2.30	0.92	
		CV22-066	212	220	8			
		CV22-066	256	266	10			
		CV23-106	295	306	11			
		CV23-106	328	335	7			
		CV23-106	339	348	9			
		CV24-386	179	189	10			
		CV24-386	216	227	11			
		CV24-386	506	519	13			
		CV24-400	154	164	10			
		CV24-400	172	186	13			
		CV24-400	191	203	12			

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
		CV24-473	185	196	11			
		CV24-473	218	237	19			
		CV24-473	273	288	15			
	COMP-006	CV22-042	174	206	32	1.24	0.54	
	COMP-007	CV22-017	190	198	8	3.43	0.72	
		CV22-017	204	213	9			
		CV22-017	219	226	7			

Notes:
(1) All samples in test program 19005-01, 19005-02, 19005-04 are quarter-core NQ.
(2) All samples in test program 19005-02A and 19005-10 are half-core NQ.

Figure 13.1: Location of CV5 Spodumene Pegmatite Drill Core Collected HLS and DMS Samples

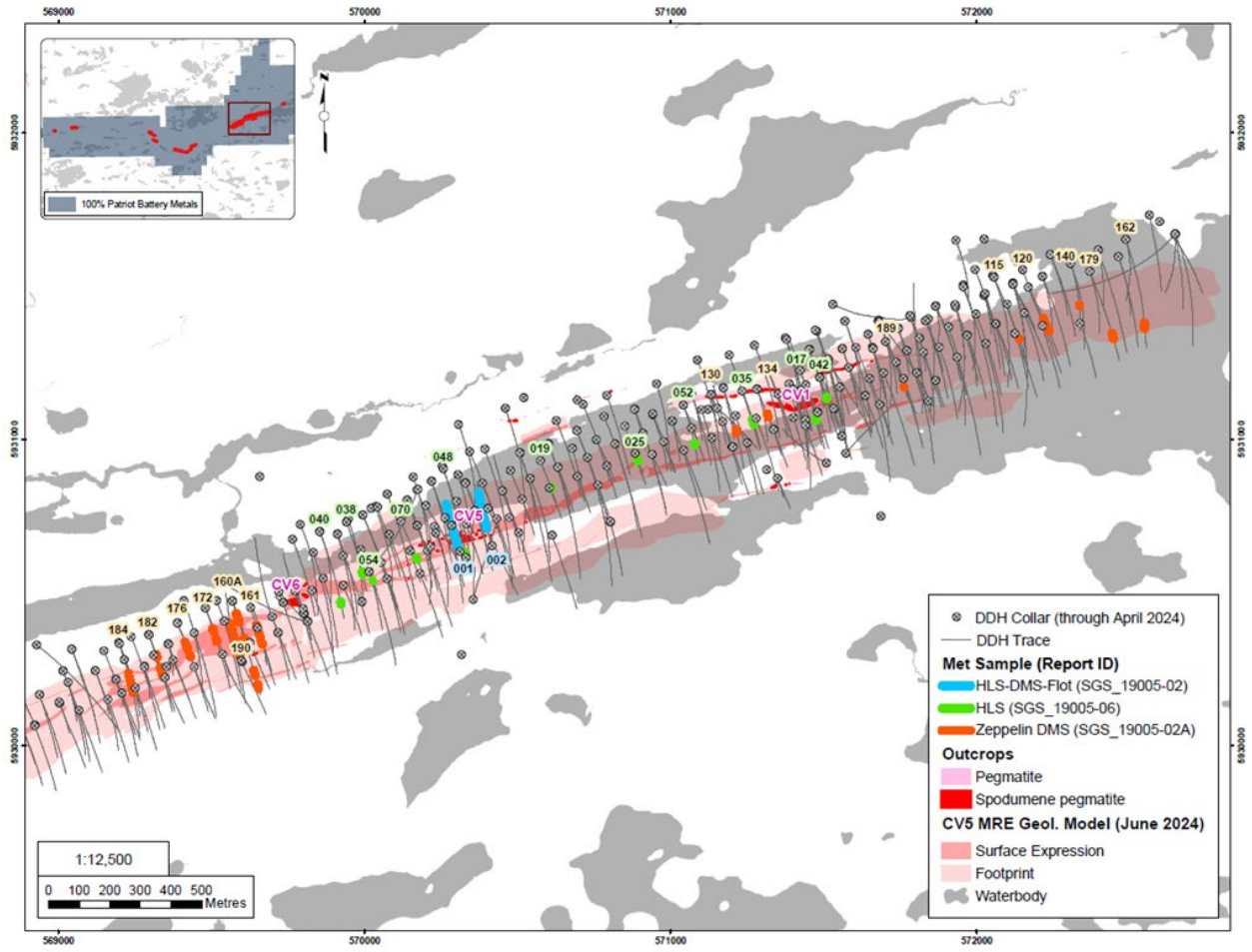


Table 13.2: Metallurgical Programs Samples Completed on Host Rock Surrounding CV5 Sources

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
19005-02A(1)	COMP-004	CV23-160A	59	62	3	0.11	8.38	Size-by-size HLS testing, dilution material used in DMS testing
		CV23-160A	190	193	3			
		CV23-161	84	87	3			
		CV23-161	166	171	5			
		CV23-172	103	106	3			
		CV23-172	121	124	3			
		CV23-176	129	132	3			
		CV23-176	160	164	4			
		CV23-182	193	209	16			
		CV23-184	146	149	3			
		CV23-184	228	231	3			
		CV23-190	23	27	4			
		CV23-190	165	168	3			
	COMP-013	CV23-160A	394	415	21	0.02	13.10	HLS Variability Testing
	COMP-014	CV23-162	380	405	25	0.04	1.78	
	COMP-015	CV23-160A	331	350	20	0.01	10.40	
	COMP-016	CV23-179	310	337	27	0.04	3.59	
19005-10	COMP-004	CV23-199	10	14	4	0.23	8.84	Size-by-size HLS testing for three (3) size fraction DMS (coarse, fine, ultrafine). HLS recrush testing at two (2) size fractions. Comminution testing. DMS testing. Magnetic separation testing. Filtration testing. Flotation testing. Gravity separation testing.
		CV23-199	184	186	2			
		CV23-230	28	31	4			
		CV23-230	122	148	26			
		CV23-241	75	89	14			
		CV23-241	203	206	3			
		CV24-377	243	246	3			
		CV24-414	62	80	18			
		CV24-414	166	169	3			
		CV24-467	123	144	21			
		CV24-467	233	236	3			
		CV24-503	128	152	24			

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
		CV24-503	263	269	6			
	COMP-008	CV22-017	165	167	2	0.33	4.19	
		CV22-017	236	239	3			
		CV22-042	152	154	2			
		CV22-042	207	210	3			
		CV22-066	181	186	5			
		CV22-066	274	277	4			
		CV23-106	285	288	2			
		CV23-106	359	363	4			
		CV24-386	151	155	4			
		CV24-386	523	527	4			
		CV24-400	136	141	4			
		CV24-400	209	213	4			
		CV24-473	179	181	2			

**Note: (1) All samples in test program 19005-02A and 19005-10 are half-core NQ.*

Although CV13 currently is not part of the material considered for the PEA, metallurgical testing (HLS) was conducted on drill core from the CV13 Pegmatite. A list of the source of samples used in testing of the pegmatite around CV13 is shown in Table 13.3.

Table 13.3: Sources of Samples for Metallurgical Programs Completed on Pegmatite from CV13

Test Program Number	Metallurgical Sample ID	Hole ID	From (m)	To (m)	Interval (m)	% Li_2O	% Fe_2O_3	Purpose
19005-06(1)	HLS-COMP-012	CV22-077	10	20	10	1.42	0.22	HLS screening across CV13
	HLS-COMP-013	CV22-092	33	43	10	0.84	0.19	
	HLS-COMP-014	CV22-103	24	33	10	1.25	0.33	
	HLS-COMP-015	CV22-082	28	33	5	1.25	0.21	
		CV22-084	29	34	5			
	HLS-COMP-016	CV22-085	167	175	8	0.95	0.33	

**Note: (1) All samples in test program 19005-06 are quarter-core NQ.*

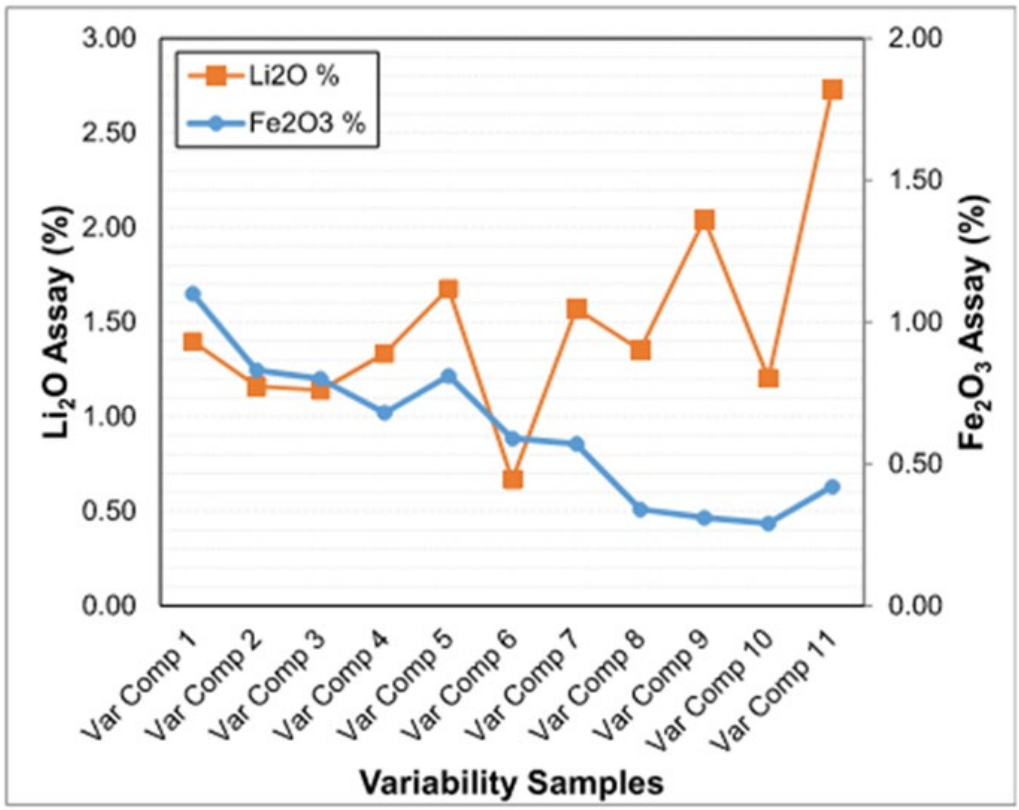
13.1.2.1 Head Sample Characterization

Head assays of the eleven CV5 variability composite samples ranged from 0.67%–2.73% Li_2O and 0.29%-1.10% Fe_2O_3. All samples contained rubidium and caesium, with averages of 2,927 g/t Rb and 980 g/t Cs, respectively.

Var Comp 1 through5, 9 and 10, all returned Ta_2O_5 above the detection limit of 0.01% Ta_2O_5. Some samples contained Ta_2O_5 at levels up to 0.03%. There is an opportunity to consider recovery of tantalite as the flowsheet is developed.

Head grades of Li_2O and Fe_2O_3 for the variability composite samples are presented in Figure 13.2.

Figure 13.2: Variability Composite Head Grades



A total of 16 composite samples, along with a master composite, were analyzed based on chemical assays by size to determine lithium deportment. Figure 13.3 summarizes the lithium distribution across particle sizes for all composites, while Table 13.4 provides the associated lithium and iron grades.

The majority of the lithium is found in the coarser size fractions (+1.70 mm), ranging from approximately 65% to 96% of the total lithium. Comp-003 (from test program 19005-02A), which was a high-grade spodumene bearing pegmatite, has the highest lithium distribution at 96% of the lithium in the +1.70 mm size fraction, while Comp-008 (also from test program 19005-02A) had the lowest lithium grade in the coarser fractions at 65% of lithium in the +1.70 mm size fraction.

Figure 13.3: Size by Assay - Lithium Distribution

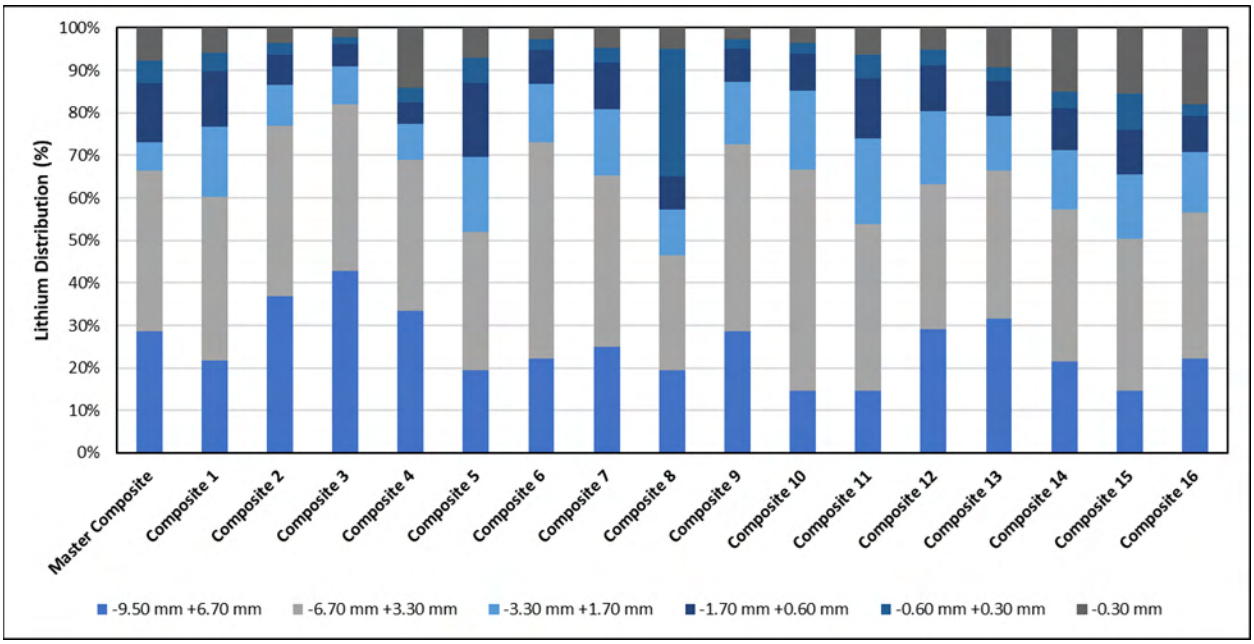


Table 13.4: Lithium (Li_2O) and Iron (Fe_2O_3) Grade Distribution with Size in CV5

Samples	Feed		-9.50 mm +6.70 mm		-6.70 mm +3.30 mm		-3.30 mm +1.70 mm		-1.70 mm +0.60 mm		-0.60 mm +0.30 mm		-0.30 mm	
	Li_2O Grade	Fe_2O_3 Grade	Li_2O Grade	Fe_2O_3 Grade	Li_2O Grade	Fe_2O_3 Grade	Li_2O Grade	Fe_2O_3 Grade	Li_2O Grade	Fe_2O_3 Grade	Li_2O Grade	Fe_2O_3 Grade	Li_2O Grade	Fe_2O_3 Grade
19005-02A -Master Comp			1.76	1.88	1.33	1.91	1.20	1.44	1.10	1.37	0.86	1.26	0.60	2.78
19005-02A - Comp-001	1.38	0.63	1.18	0.45	1.38	0.63	1.33	0.66	1.27	0.66	1.01	0.70	0.77	0.76
19005-02A - Comp-002	0.71	0.83	0.67	0.69	0.69	0.65	0.75	0.62	0.77	0.82	0.62	1.15	0.45	1.09
19005-02A - Comp-003	1.68	0.53	1.59	0.43	1.46	0.54	1.59	0.56	1.59	0.59	1.16	0.84	0.97	0.80
19005-02A - Comp-004	0.11	8.38	0.11	8.52	0.11	8.48	0.11	7.92	0.10	7.81	0.12	7.42	0.12	9.52
19005-02A - Comp-005	0.30	0.60	0.17	0.48	0.32	0.55	0.37	0.78	0.47	0.68	0.45	0.75	0.41	0.91
19005-02A - Comp-006	1.01	1.22	1.23	1.09	1.53	1.13	0.92	1.23	0.82	1.27	0.71	1.39	0.58	1.57
19005-02A - Comp-007	0.43	0.94	0.47	0.54	0.45	0.81	0.39	0.99	0.37	0.90	0.30	0.92	0.30	1.25
19005-02A - Comp-008	0.05	0.90	0.06	1.53	0.06	0.89	0.06	0.85	0.06	1.03	0.06	1.03	0.07	1.34
19005-02A - Comp-009	0.75	3.06	0.86	3.40	0.99	2.77	0.86	3.20	0.69	3.10	0.45	2.72	0.34	3.19
19005-02A - Comp-010	0.16	0.37	0.12	0.22	0.28	0.28	0.21	0.30	0.14	0.25	0.12	0.29	0.13	0.31
19005-02A - Comp-011	0.14	0.89	0.10	0.65	0.15	0.61	0.16	0.72	0.15	0.98	0.13	1.21	0.10	1.32
19005-02A - Comp-012	0.22	2.24	0.39	2.11	0.26	2.19	0.30	2.07	0.24	2.27	0.19	2.57	0.17	3.50
19005-02A - Comp-013	0.02	13.10	0.02	13.20	0.01	13.20	0.01	12.90	0.01	12.80	0.01	12.20	0.01	13.20
19005-02A - Comp-014	0.04	1.78	0.03	1.39	0.03	1.91	0.03	1.57	0.03	1.80	0.03	2.09	0.04	2.22
19005-02A - Comp-015	0.01	10.40	0.01	10.40	0.01	10.40	0.01	10.30	0.01	10.30	0.02	10.50	0.01	10.10
19005-02A - Comp-016	0.04	3.59	0.05	3.71	0.05	3.43	0.05	3.45	0.05	3.40	0.05	3.60	0.06	4.11

13.1.2.2 Mineralogy

Mineralogical characterization testwork was carried out on 20 samples of drill core from the CV5 Pegmatite (CF21-001, 002, and 004) using a combination of TIMA-X (Quantitative SEM) EPMA, Laser Ablation by Inductively Coupled Plasma Mass Spectrometry (LA by ICP-MS), XRD analysis, and chemical assays. Generally, spodumene was found to be the dominant lithium mineral species across all samples.

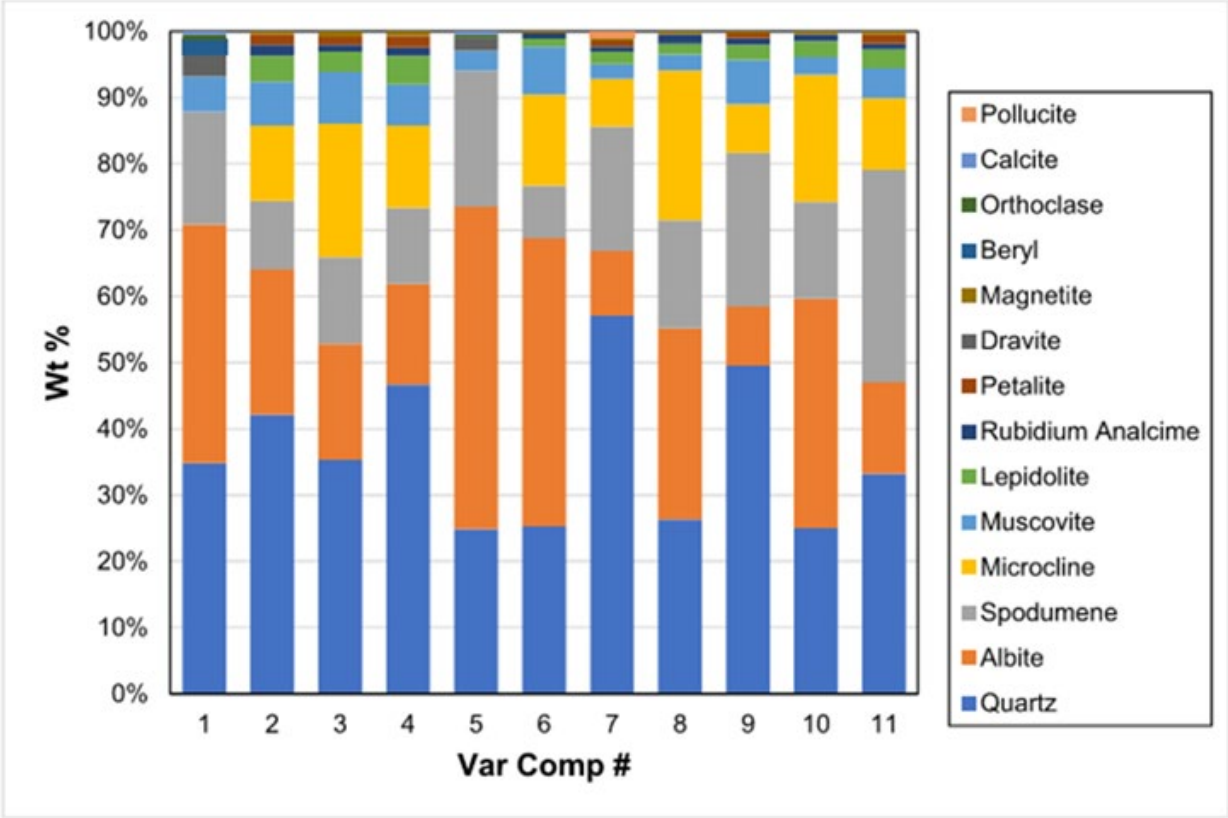
Semi-Quantitative XRD was performed on the 11 CV5 HLS variability composite samples (19005-02), the 16 CV5 pegmatites and non-pegmatites of the 19005-02A program, and the MC001 and MC002 samples (19005-10).

Spodumene was identified as the dominant lithium-bearing mineral, with spodumene content varying from 7.9%–32.1%. Spodumene content corresponded with Li_2O assay results with Var Comp 6 (0.67% Li_2O, 7.9% spodumene) and Var Comp 11 (2.73% Li_2O, 32.1% spodumene) having the lowest and highest Li_2O and spodumene content, respectively.

Results indicate that the main gangue minerals in the samples are quartz with albite, comprising combined proportion of 47% to 74% within the 11 samples. Minor quantities of lepidolite (up to 3.5%) and moderate quantities of muscovite (up to 17.1%) were also present. Occurrences of up to 16% tourmaline were identified in CV5 samples via TIMA-X.

XRD results for the variability composite samples are indicated in Figure 13.4.

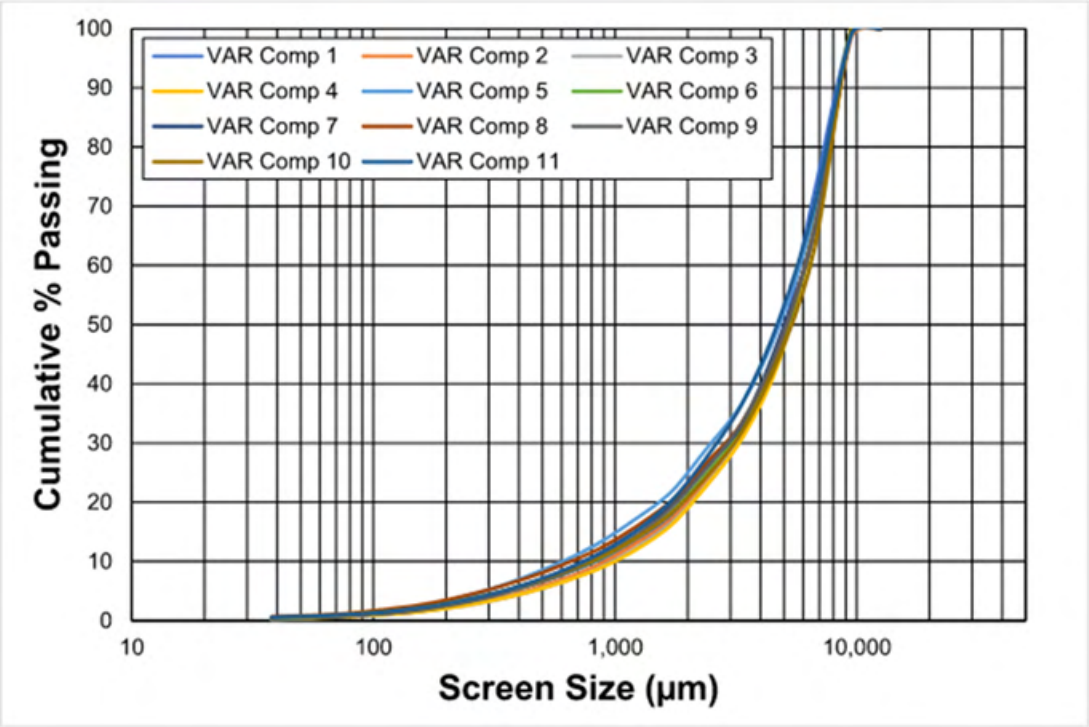
Figure 13.4: XRD Results on Variability Composite Samples



13.1.2.3 Particle Size Distribution

Particle size analysis was performed on 0.5 kg subsamples from each CV5 variability composite sample after crushing to -9.5 mm in the laboratory. All variability composites had a similar particle size distribution with P80 values from 7.5 mm–8.0 mm. Particle size distributions for the variability composite samples are indicated in Figure 13.5.

Figure 13.5: Particle Size Distributions for Variability Composite Samples



13.1.2.4 Comminution Tests

13.1.2.4.1 Standard Bond Abrasion Test

Comminution testwork was conducted by SGS to determine the Abrasion Index (Ai) of the material using standard Bond abrasion testing. Five (5) composites were tested: One (1) from test program 19005-02, using a mixture of CF21-001Met and CF21-002Met, and four (4) for testwork program19005-10, using Comp 001 (open pit composite-pegmatite), Comp 004 (open pit host rock), Comp 005 (underground composite pegmatite), and Comp 008 (underground host rock), representing both pegmatite and host rock from open pit and underground deposits.

The results of each test are summarized in Table 13.5. As expected, the host rock exhibited a lower Ai compared to the pegmatite, which may help explain the presence of iron in the DMS bypass streams. Additionally, the underground composites reported a higher Ai than their open pit counterparts. This testwork will support more accurate modelling of crushing performance and predicted wear.

Table 13.5: Standard Bond Abrasion Test Results

Test Program Number	Sample	Abrasion Index (Ai)
19005-02	CF21-001Met and CF21-002Met	0.445
19005-10	Comp 001	0.498
	Comp 004	0.384
	Comp 005	0.551
	Comp 006	0.419

13.1.2.4.2 Standard Bond Rod Mill Grindability

Grindability testwork was completed by SGS in program 19005-02 on a pegmatite feed composite, using a mixture of CF21-001Met and CF21-002Met. The Bond Rod Mill Work Index was measured to be 14.2 kWh/tonne (metric). The results are shown are summarized in Table 13.6

Table 13.6: Standard Bond Rod Mill Grindability Test Results

Test Program Number	Sample	Bond Rod Mill Work Index
19005-02	CF21-001Met and CF21-002Met	14.2

13.1.2.4.3 Standard Bond Ball Mill Grindability

Grindability testwork on a pegmatite sample was performed by SGS in program 19005-02. The sample was a spodumene bearing pegmatite using a mixture 50:50 of sample CF21-001Met and CF21-002Met. The Bond Ball Mill Work Index of this material was measured to be 16.0 kWh/tonne (metric). Grindability testwork on a sample that represented the expected tailings generated from the future concentrator was performed as part of the paste fill design. The testwork material was a combination of the DMS floats and DMS bypass and was combined in proportions that represented the expected tailings of the design (based on the PEA mass balance). The sample was generated during the SGS test program 19005-02A DMS testwork. The sample had a Bond Ball Mill Work Index of 12.6 kWh/tonne (metric). The Bond Ball Mill grindability results are shown in Table 13.7.

Table 13.7: Standard Bond Ball Mill Grindability Test Results

Test Program Number	Sample	Bond Rod Mill Work Index
19002-02	CF21-001Met and CF21-002Met	16.0
19005-02A(1)	Comp-001, 002, 003 and 004	12.6

**Note: (1) This test was done using material from 19005-02A, this test was part of the design of the paste backfill design.*

13.1.2.5 Heavy Liquid Separation

13.1.2.5.1 SGS - 19005-04

A 1.5 kg subsample of each -9.5 mm variability composite was screened at 0.85 mm. The -0.85 mm fraction was set aside and HLS testwork at a single SG cut point of 2.85 was performed on the 9.5 mm +0.85 mm fraction. All HLS sink and float products and the -0.85 mm fractions were assayed.

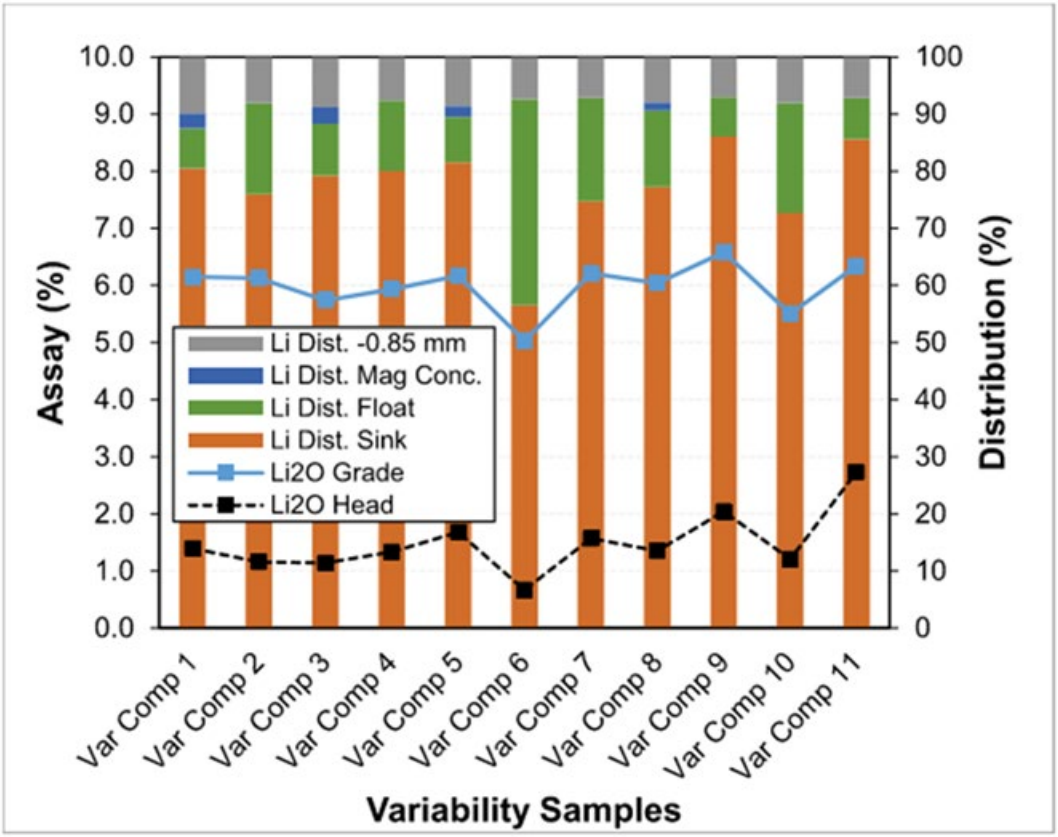
HLS sinks results indicate Li_2O grades of 5.03%–6.58% and HLS Stage Li recoveries of 61.0%–92.5%. Nine (9) of the 11 variability composites achieved a concentrate grade of >5.5% Li_2O and Global HLS recovery >72%.

Var Comp 3 recorded 5.35% Li_2O grade and 90.1% Li recovery, while Var Comp 6 recorded 5.03% Li_2O grade and 60.1% Li recovery. There appears to be a strong potential for these samples to achieve >5.5% Li_2O grade at marginally lower recoveries using a higher SG cut point than 2.85.

Fe_2O_3 grades were recorded as 0.52%–1.79% before any magnetic separation was applied.

Global HLS results are indicated in Figure 13.6.

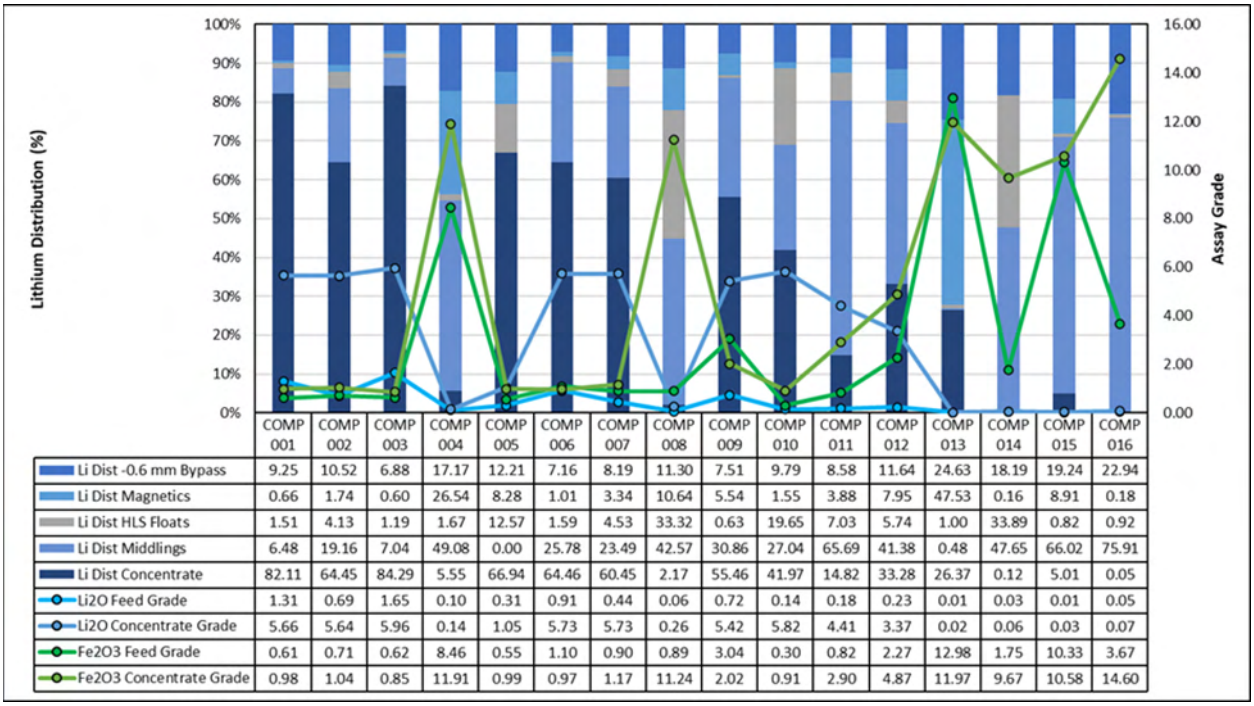
Figure 13.6: Global HLS Results for Variability Composite Samples



13.1.2.5.2 SGS - 19005-02A

HLS tests were conducted on all 16 composites at a crush size of 9.50 mm, and a bottom size of 0.60 mm. Dry magnetic separation was then performed on the sink products to remove iron-rich material. Figure 13.7 illustrates the subsequent lithium distributions and assays.

Figure 13.7: Test Program 19005-02A HLS Testwork Results



	COMP 001	COMP 002	COMP 003	COMP 004	COMP 005	COMP 006	COMP 007	COMP 008	COMP 009	COMP 010	COMP 011	COMP 012	COMP 013	COMP 014	COMP 015	COMP 016
Li Dist -0.6 mm Bypass	9.25	10.52	6.88	17.17	12.21	7.16	8.19	11.30	7.51	9.79	8.58	11.64	24.63	18.19	19.24	22.94
Li Dist Magnetics	0.66	1.74	0.60	26.54	8.28	1.01	3.34	10.64	5.54	1.55	3.88	7.95	47.53	0.16	8.91	0.18
Li Dist HLS Floats	1.51	4.13	1.19	1.67	12.57	1.59	4.53	33.32	0.63	19.65	7.03	5.74	1.00	33.89	0.82	0.92
Li Dist Middlings	6.48	19.16	7.04	49.08	0.00	25.78	23.49	42.57	30.86	27.04	65.69	41.38	0.48	47.65	66.02	75.91
Li Dist Concentrate	82.11	64.45	84.29	5.55	66.94	64.46	60.45	2.17	55.46	41.97	14.82	33.28	26.37	0.12	5.01	0.05
Li2O Feed Grade	1.31	0.69	1.65	0.10	0.31	0.91	0.44	0.06	0.72	0.14	0.18	0.23	0.01	0.03	0.01	0.05
Li2O Concentrate Grade	5.66	5.64	5.96	0.14	1.05	5.73	5.73	0.26	5.42	5.82	4.41	3.37	0.02	0.06	0.03	0.07
Fe2O3 Feed Grade	0.61	0.71	0.62	8.46	0.55	1.10	0.90	0.89	3.04	0.30	0.82	2.27	12.98	1.75	10.33	3.67
Fe2O3 Concentrate Grade	0.98	1.04	0.85	11.91	0.99	0.97	1.17	11.24	2.02	0.91	2.90	4.87	11.97	9.67	10.58	14.60

The composites (COMP-001 to COMP-003) demonstrated sufficient liberation at a 9.50 mm crush size, producing spodumene concentrates with >5.50% Li_2O and <1.20% Fe_2O_3 while achieving 64% to 85% global lithium recovery (including -0.60 mm fines bypass). Specifically, COMP-001 and COMP-003 showed uniform spodumene crystal sizes and excellent liberation, with only 6% to 7% of global lithium distribution in the middlings fraction. In contrast, COMP-002 (low-grade pegmatite) exhibited less uniform spodumene crystals sizes, with 19% of global lithium distribution in the middlings. The host rock composite (COMP-004) indicated that approximately 49% of global lithium distribution reported to the HLS middlings.

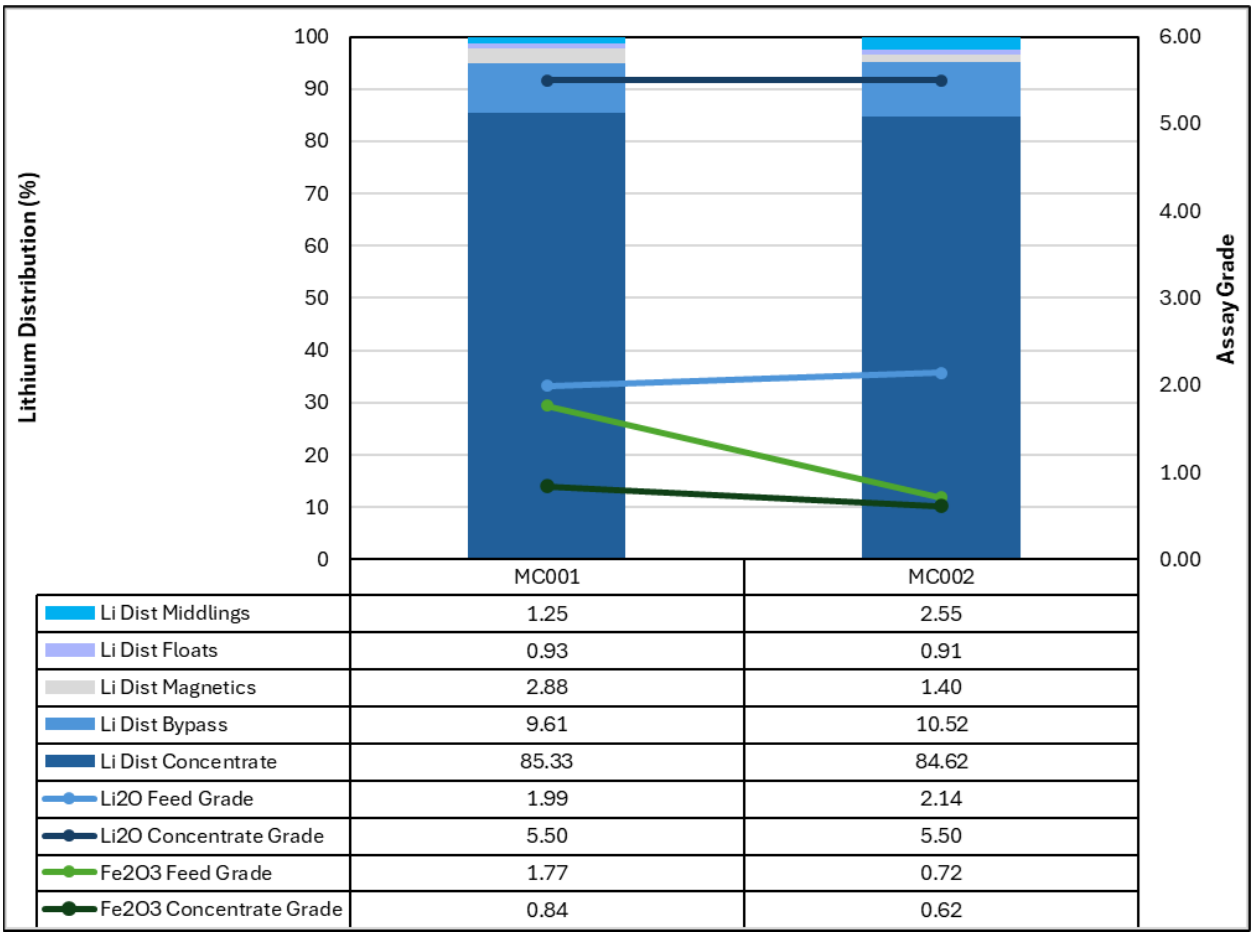
The variability samples were tested to understand the impact of different lithologies on the process responded with high degree of variability, as seen in Figure 13.7. COMP-006, 007, 009 and 010 achieved >5.50% Li_2O with 42% to 64% global lithium recovery. The lithium reporting to middlings increased from 6% to 19% (global lithium distribution) for the samples (COMP-001 to 003) to upwards of 30%, indicating that the lithium grain size in the variability samples may be smaller and are less uniformed in size, further highlighting the sensitivities identified in the first four (4) composites.

13.1.2.5.3 SGS – 19005-10

HLS tests were conducted on the two (2) created samples to represent the open pit and underground composites, MC001 and MC002, respectively. The samples were crushed to a top size of 9.5 mm, and testing was carried out across three (3) size fractions, coarse (9.5 mm to 3.3 mm), fine (3.3 mm to 1.7 mm),

and ultrafine (1.7 mm to 0.60 mm). This approach allowed for evaluation of staged recovery potential by further splitting the previous fine fraction (3.3 mm to 0.60 mm) into distinct fine and ultrafine streams. The objective was to assess whether this refinement would improve lithium recovery and better align with equipment sizing and circuit design considerations, where splitting the fine fraction was advantageous. Dry magnetic separation was then performed on the sink products to remove iron-rich material. Figure 13.8 illustrates the subsequent lithium distributions and assays.

Figure 13.8: HLS Testwork Results

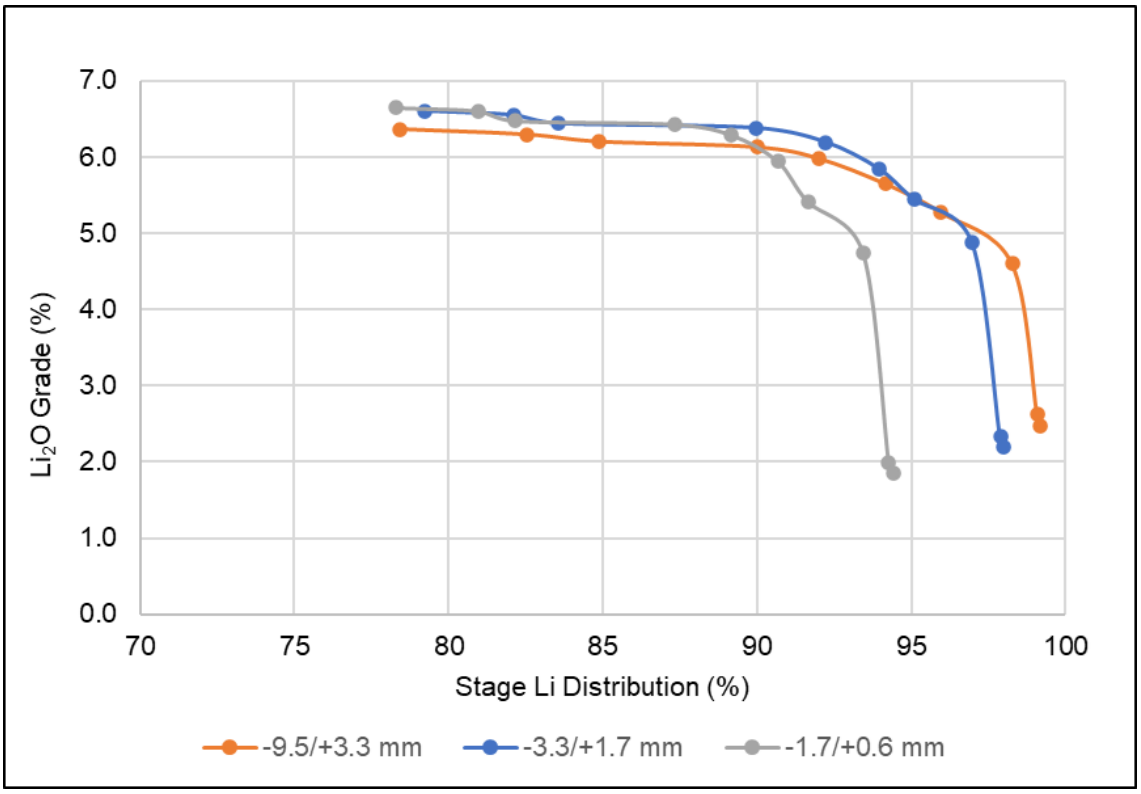


	MC001	MC002
Li Dist Middlings	1.25	2.55
Li Dist Floats	0.93	0.91
Li Dist Magnetics	2.88	1.40
Li Dist Bypass	9.61	10.52
Li Dist Concentrate	85.33	84.62
Li2O Feed Grade	1.99	2.14
Li2O Concentrate Grade	5.50	5.50
Fe2O3 Feed Grade	1.77	0.72
Fe2O3 Concentrate Grade	0.84	0.62

HLS testwork on the master composites demonstrated strong performance, producing interpolated concentrates at 5.5% Li_2O at approximately 85% global lithium recovery. The elevated recovery is primarily attributed to the high lithium head grades of the MC001 and MC002 samples, with the majority of lithium losses reporting to the bypass fraction. The open pit composite (MC001), which had a higher Fe_2O_3 content, exhibited greater losses to the magnetic separation stage, while the underground composite (MC002) reported more lithium to the middlings fraction.

The test program also aimed to assess the impact of splitting the traditional fine size fraction (-3.3 mm to +0.6 mm) into two (2) subfractions: fine (-3.3 mm to +1.7 mm) and ultrafine (-1.7 mm to +0.6 mm). As shown in Figure 13.9, the ultrafine fraction returned the lowest stage recovery of the three (3) tested size ranges. However, performance remained strong, achieving 91% lithium recovery at a concentrate grade of 5.5% Li_2O, supporting its potential inclusion in the design basis.

Figure 13.9: Stage HLS Results



13.1.2.6 Magnetic Separation

13.1.2.6.1 SGS - 19005

Magnetic separation was carried out on HLS sink products for Var Comp 1, 3, 5 and 8 with a dry belt magnetic separator at a field strength of 10,000 Gauss. Sinks products were screened at 3.3 mm with magnetic separation performed separately on the +3.3 mm and -3.3 mm fractions to increase separation efficiency. The magnetic and non-magnetic +3.3 mm and -3.3 mm fractions were then combined and assayed.

Results indicate all four (4) of the variability composite samples achieved the target grades of >5.5% Li_2O and <2.0% Fe_2O_3. Up to 3.2% Li was reported to the Magnetic Separation Mags across the four (4) tests.

Table 13.8: Magnetic Separations Results for Variability Composite Samples 1, 3, 5 & 8

HLS Products	Weight (g)	%	Li_2O	Fe_2O_3	Distribution (%)	
					Li	Fe_2O_3
Var Comp 1 Sink Before Mag	208	23.6	5.53	1.79	92.2	43.4
Var Comp 1 Sink Non-Mag	181.1	20.54	6.15	0.61	89.29	12.9
Var Comp 1 Sink Mag	27	3.1	1.35	9.67	2.9	30.5
Var Comp 1 Float	673	76.	0.14	0.72	7.78	56.6
Feed (Calc.)	**881**	**100**	**1.41**	**0.97**	**100**	**100**
Var Comp 3 Sink Before Mag	146	15.7	5.35	1.58	90.1	31.9
Var Comp 3 Sink Non-Mag	132	14.2	5.74	1.14	87.0	20.7
Var Comp 3 Sink Mag	14.7	1.58	1.87	5.55	3.16	11.2
Var Comp 3 Float	784	84.3	0.11	0.63	9.89	68.1
Feed (Calc.)	**931**	**100**	**0.93**	**0.78**	**100**	**100**
Var Comp 5 Sink Before Mag	230	25.2	5.93	1.10	91.4	43.5
Var Comp 5 Sink Non-Mag	215	23.7	6.17	0.71	89.2	26.4
Var Comp 5 Sink Mag	14.4	1.58	2.26	6.90	2.17	17.1
Var Comp 5 Float	680	74.8	0.19	0.48	8.64	56.5
Feed (Calc.)	**910**	**100**	**1.64**	**0.64**	**100**	**100**
Var Comp 8 Sink Before Mag	205	22.1	5.81	0.97	85.5	36.1
Var Comp 8 Sink Non-Mag	193	20.9	6.04	0.62	83.9	21.7
Var Comp 8 Sink Mag	11.5	1.24	1.89	6.94	1.56	14.4
Var Comp 8 Float	721	77.9	0.28	0.49	14.5	63.9
Feed (Calc.)	**926**	**100**	**1.50**	**0.60**	**100**	**100**

13.1.2.6.2 Changsha Research Institute of Mining and Metallurgy (CRIMM) – 19005-10

Magnetic separation testwork was completed by CRIMMs on the 19005-10 DMS concentrate samples using bench-scale wet drum separators capable of treating coarse material. Samples were subjected to a staged magnetic separation process at increasing field strengths (7,000 Gs, 8,000 Gs, and 9,000 Gs), across three (3) feed size fractions: +0.65 mm to -3.3 mm, +3.3 mm to -6.3 mm, and +3.3 mm to -9.5 mm.

The test results demonstrated that the CRIMMs unit performed effectively across the full tested range, including the coarsest fraction initially thought unsuitable for magnetic separation. Finer fractions achieved the highest iron rejection but showed lower selectivity, while coarser material retained more lithium but was

less effective at rejecting Fe_2O_3. The combined product (fine and coarse non-magnetics) had total Li_2O losses of less than 2% and iron rejection of approximately 45%. This testwork demonstrated the feasibility of processing the coarsest fraction by magnetic separation, supporting its inclusion in the final flowsheet design.

Table 13.9: CRIMMs Magnetic Separations Results at 9,000 Gs

Products	Mass Yield (%)	Li_2O Grade (%)	Fe Grade (%)	Li_2O Loss (%)	Fe Rejection (%)	Selectivity
+0.65 -3.3 mm Feed	100	3.46	2.98			
+0.65 -3.3 mm Non-Mags	66.5	4.94	1.14	5.01	74.53	14.88
+3.3 mm -6.3 mm Feed	100	3.19	2.98			
+3.3 mm -6.3 mm Non-Mags	69.9	4.44	1.27	2.64	70.06	26.54
+3.3 mm -9.5 mm Feed	100	3.25	3.03			
+3.3 mm -9.5 mm Non-Mags	84.9	3.78	2.22	1.11	37.75	34.01
Combined +0.65 -9.5 mm Feed	100	3.29	3.02			
Combined +0.65 -9.5 mm Non-Mags	81.1	3.98	2.03	1.92	45.37	23.65

13.1.2.7 Dense Media Separation

13.1.2.7.1 SGS - 19005-02

Dense media separation was carried out on a single 143 kg sample of CV5 Spodumene Pegmatite material with a head grade of 1.05% Li_2O and 0.55% Fe_2O_3 (the sample was the combination of sample CF21-001Met and CF21-002Met). The material was tested through a two-stage DMS flow sheet twice with better performance being achieved with a first pass cut point of 2.70 SG and a second pass cut point of 2.85 SG. Second pass DMS sinks (spodumene concentrate) was passed over a magnetic separator for iron removal. The flowsheet is displayed in Figure 13.14.

The DMS test returned a spodumene concentrate grading 5.77% Li_2O and 0.62% Fe_2O_3 and a 79% Li recovery. DMS spodumene concentrate from this testwork is shown in Figure 13.10 and Figure 13.11.

**Figure 13.10: Spodumene Concentrate (DMS + non-magnetic fractions)
5.8% Li_2O and 0.60% Fe_2O_3 at 79% Recovery**



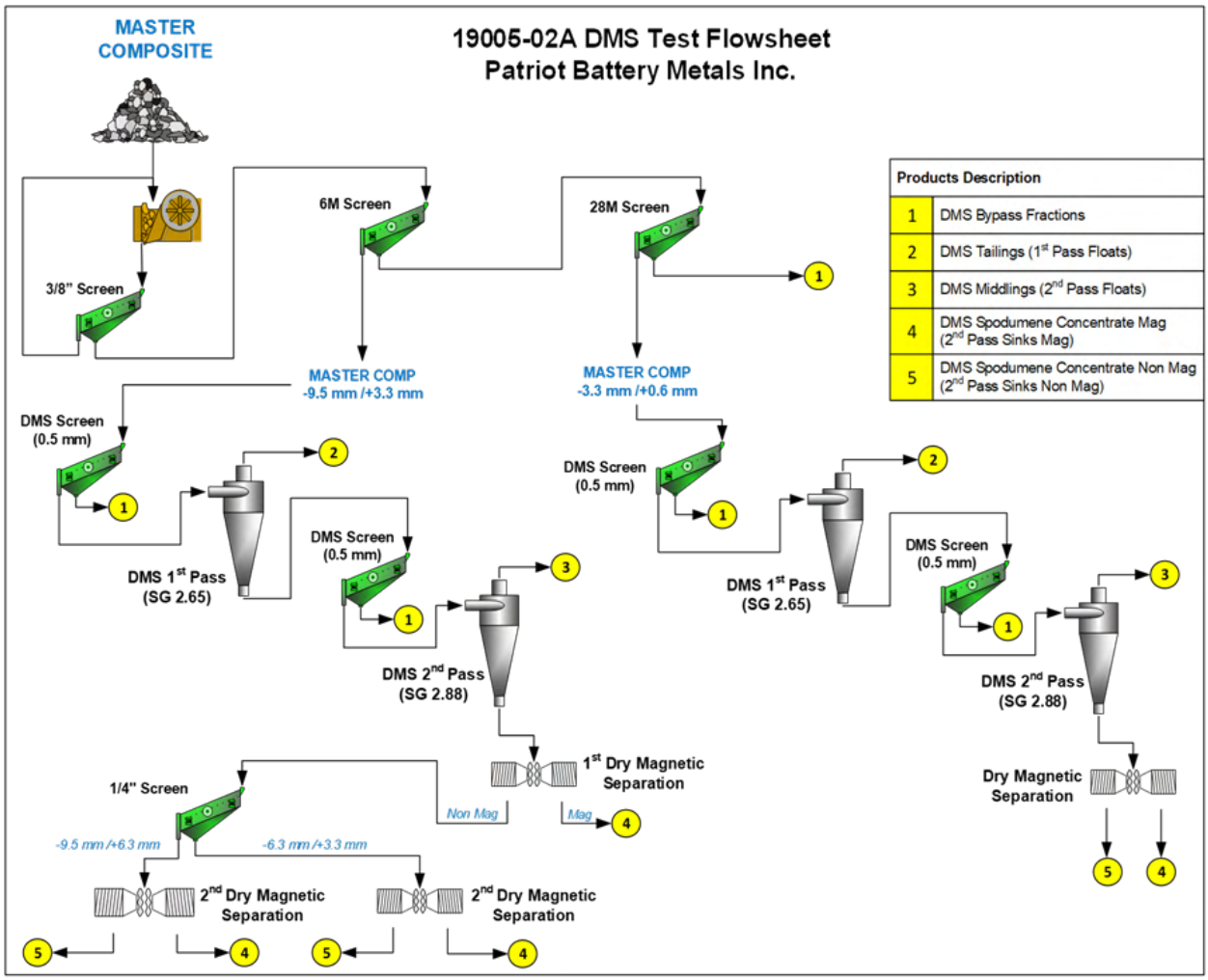
**Figure 13.11: Close-up of Spodumene Concentrate (DMS + non-magnetic fractions)
5.8% Li_2O and 0.60% Fe_2O_3 at 79% Recovery**



13.1.2.7.2 SGS - 19005-02A

Dense media separation (DMS) was conducted on the master composite sample (~373 kg) with a head grade of 1.05% Li_2O and 1.88% Fe_2O_3 (the sample was composed of Comp-001, 002, 003 and 004 at 65%, 10%, 10% and 15%, respectively). The material was processed through a two-stage DMS flowsheet, with a first pass cut point of 2.65 SG and a second pass cut point of 2.88 SG. Dry magnetic separation was performed on the second pass sinks to remove iron-rich material. Figure 13.12 presents the tested flowsheet.

Figure 13.12: DMS Testwork Flowsheet



The test results achieved the final concentrate grade of 6.21% Li_2O and 0.60% Fe_2O_3 at 69% global lithium recovery. Table 13.10 summarizes the test results for the master composite.

Table 13.10: Master Comp DMS Testwork Results

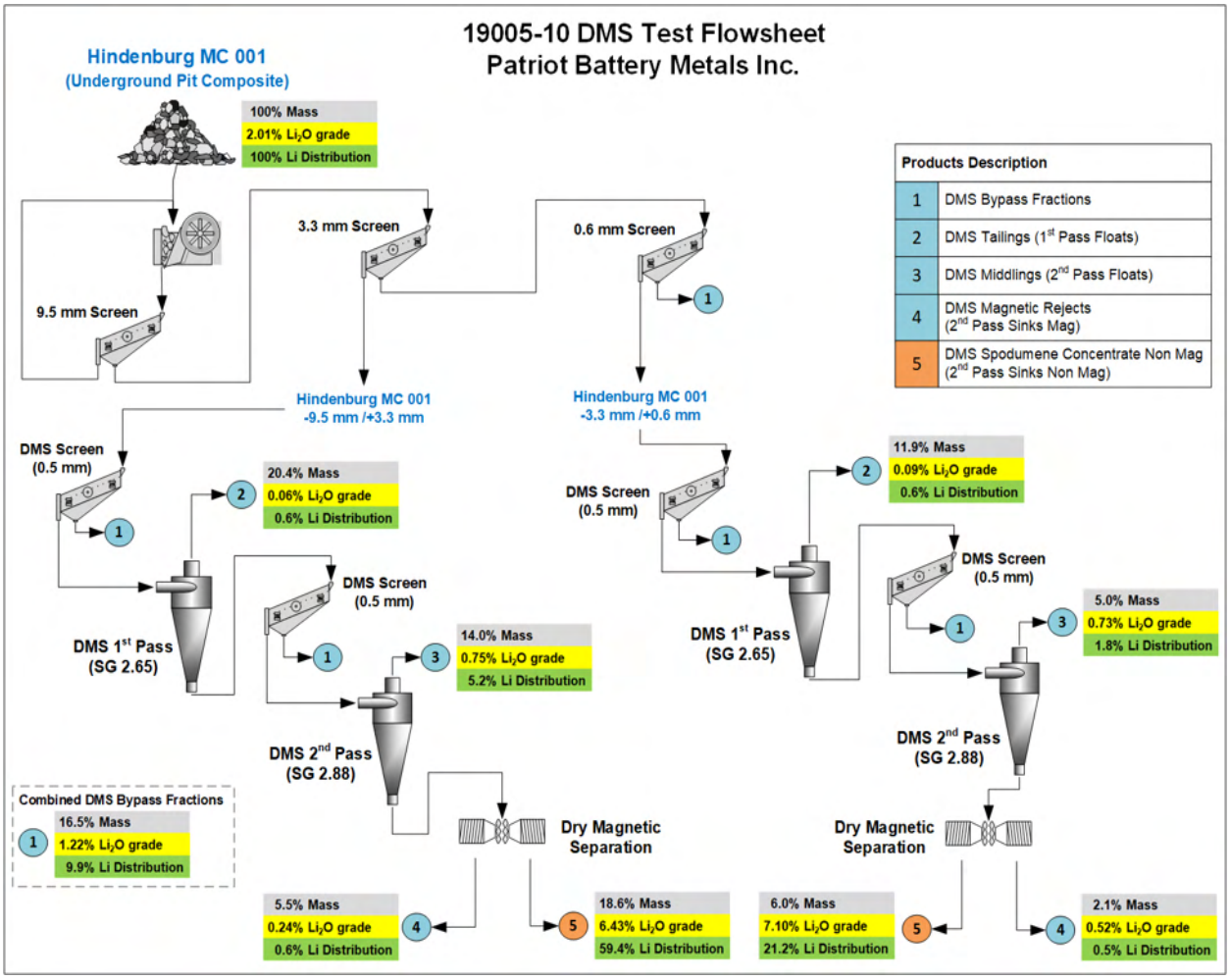
Description	Mass Distribution	Li_2O Grade	Fe_2O_3 Grade	Li_2O Distribution	Fe_2O_3 Distribution
DMS Feed	100.00	1.05	1.80	100.00	100.00
-0.6 mm Fines Bypass	15.91	0.67	2.38	10.07	21.00
DMS Floats	37.14	0.06	0.26	1.98	0.26
DMS Middlings	27.79	0.51	1.86	13.51	28.73
Magnetic Tailings	7.39	0.72	10.02	5.05	41.07
DMS Concentrate	11.77	6.21	0.60	69.39	3.94

Table 13.10 shows that ~23% of the global lithium distribution is accounted for by fines bypass and middlings, with only minor losses in the DMS floats. These results align with the HLS findings and support the use of a DMS flowsheet moving forward.

13.1.2.7.3 SGS-19005-10

Dense media separation was conducted on the master composites MC001 and MC002, which were processed individually through a two-stage DMS flowsheet. The open pit MC001 had a head grade of 2.01% Li_2O and 1.86% Fe_2O_3 while underground MC002 reported 2.28% Li_2O and 0.74% $Fe2O_3$. The material was processed through a two-stage DMS flowsheet, with a first pass cut point of 2.65 SG and a second pass cut point of 2.88 SG. Dry magnetic separation was performed on the second-pass sinks to remove iron-rich material. Figure 13.13 presents the tested flowsheet.

Figure 13.13: DMS Testwork Flowsheet



The MC001 test results achieved the final concentrate grade of 6.59% Li_2O and 0.81% Fe_2O_3 at 80.6% global lithium recovery. The MC001 test results achieved the final concentrate grade of 6.60% Li_2O and 0.49% Fe_2O_3 at 81.9% global lithium recovery. Table 13.11 and Table 13.12 summarizes the test results for the two (2) runs.

Table 13.11: MC001 DMS Testwork Results

Description	Mass Distribution	Li_2O Grade	Fe_2O_3 Grade	Li_2O Distribution	Fe_2O_3 Distribution
DMS Feed	100.00	2.01	1.86	100.00	100.00
DMS Bypass	16.46	1.22	2.55	9.95	22.56
DMS Floats	32.30	0.07	0.28	1.19	4.86
DMS Middlings	18.98	0.75	1.23	7.04	12.51
Magnetic Tailings	7.63	0.32	12.05	1.20	49.39
DMS Concentrate	24.63	6.59	0.81	80.62	10.67

Table 13.12: MC002 DMS Testwork Results

Description	Mass Distribution	Li_2O Grade	Fe_2O_3 Grade	Li_2O Distribution	Fe_2O_3 Distribution
DMS Feed	100	2.28	0.74	100	100
DMS Bypass	13.96	1.51	1.19	9.23	22.54
DMS Floats	35.79	0.09	0.27	1.38	13.15
DMS Middlings	19.72	0.78	0.75	6.75	20.04
Magnetic Tailings	2.22	0.79	8.50	0.77	25.59
DMS Concentrate	28.31	6.60	0.49	81.87	18.68

The results show that 15–17% of the global lithium distribution is accounted for by the fines bypass and middlings, with only minor losses to the floats and magnetic fractions. These outcomes are consistent with the HLS program and support the case for a DMS-only flowsheet, given the high overall lithium recovery. The DMS testwork also enabled several downstream investigations: the DMS bypass and middlings were used for flotation testwork, the bypass stream supported vacuum filtration and thickening trials, the middlings were processed in a separate HLS program to evaluate a potential re-crush circuit at various size fractions, and the magnetic rejects were tested in a gravity separation circuit to assess the potential for tantalum recovery.

13.1.2.8 Recrush Heavy Liquid Separation

Following the DMS testwork, the coarse middlings stream was identified as the tailings fraction with the second-highest lithium content outside of the bypass. For sample MC001, the coarse middlings accounted for 5.23% of the overall Li_2O distribution. To evaluate the potential for additional recovery, SGS conducted HLS re-crush testwork on this fraction.

The test program involved recrushing the coarse middlings to two (2) target sizes: -6.3 mm and -3.3 mm. Material finer than 0.65 mm was screened out as additional bypass. Each size fraction was then tested independently to assess lithium recovery potential. The -6.3 mm recrush achieved a stage Li_2O recovery of 36.15%, while the finer -3.3 mm fraction yielded a higher recovery of 52.16%.

Although finer crushing generated more bypass fines, the improved recovery from the -3.3 mm fraction resulted in a greater impact on global recovery. Incorporating the -3.3 mm recrush circuit into the DMS flowsheet increased total Li_2O recovery by 2.67%. The results of the HLS testwork are presented in Table 13.13.

Table 13.13: MC001 HLS Recrush Results

Description	Mass Distribution	Li_2O Grade	Fe_2O_3 Grade	Li_2O Distribution	Fe_2O_3 Distribution
DMS Concentrate	24.63	6.59	0.81	80.62	10.67
DMS Coarse Middlings	14.00	0.75	1.11	5.23	8.35
Stage -6.3 mm Recrush	0.75	5.50	0.67	36.15	2.88
Stage -3.3 mm Recrush	1.03	5.50	0.49	52.16	2.89
DMS Concentrate with -6.3 mm Recrush	25.4	6.56	0.80	82.32	10.83
DMS Concentrate with -3.3 mm Recrush	25.7	6.55	0.79	83.29	10.83

13.1.2.9 Flotation

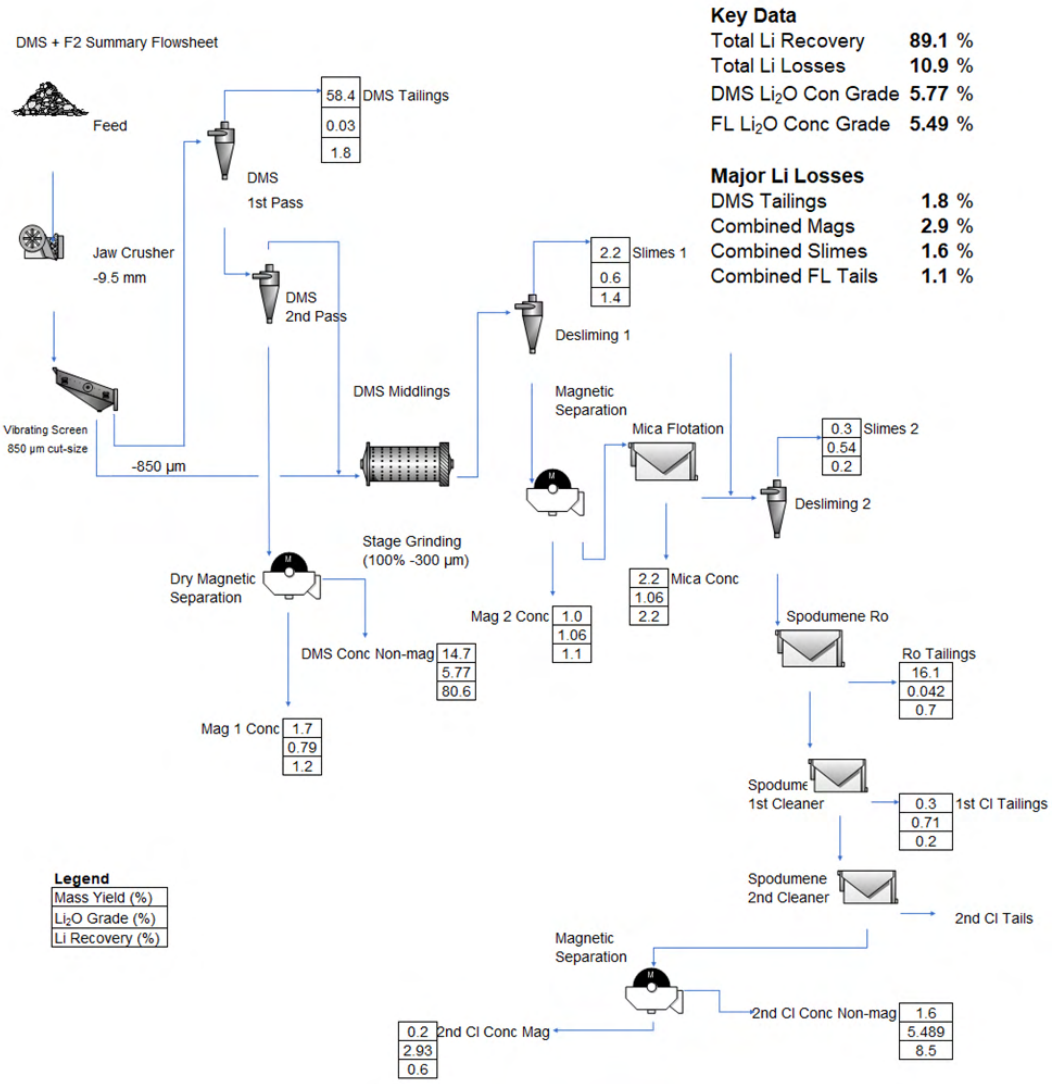
13.1.2.9.1 SGS - 19005-02A

Although a DMS-only flowsheet is the preferred process route for CV5, preliminary flotation testwork was conducted to understand the amenability of CV5 material to flotation.

Four (4) 2-kg sample of second stage DMS floats (commonly referred to as the DMS middlings) combined with the bypass fraction (i.e., the -0.85 mm fraction screened out prior to DMS) was ground to 100% passing 300 µm. The material was de-slimed, processed through a magnetic separator with mica flotation of non-magnetic fraction, de-slimed again (to also increase solids concentration for spodumene conditioning), then processed through spodumene flotation consisting of a rougher cell and two (2) stage cleaning cells, with a final magnetic separator to remove iron from second cleaner concentrate.

The bench scale flotation tests increased the global lithium recovery by 10.1% above the results achieved by the DMS testwork alone (DMS alone achieved 79.0%, while the hybrid circuit achieved 89.1%). Flotation spodumene concentrate returned a grade of 5.49% Li_2O and 0.40% Fe_2O_3. The overall DMS and flotation testwork flowsheet is shown in Figure 13.14.

Figure 13.14: Combined DMS and Flotation Testwork Flowsheet



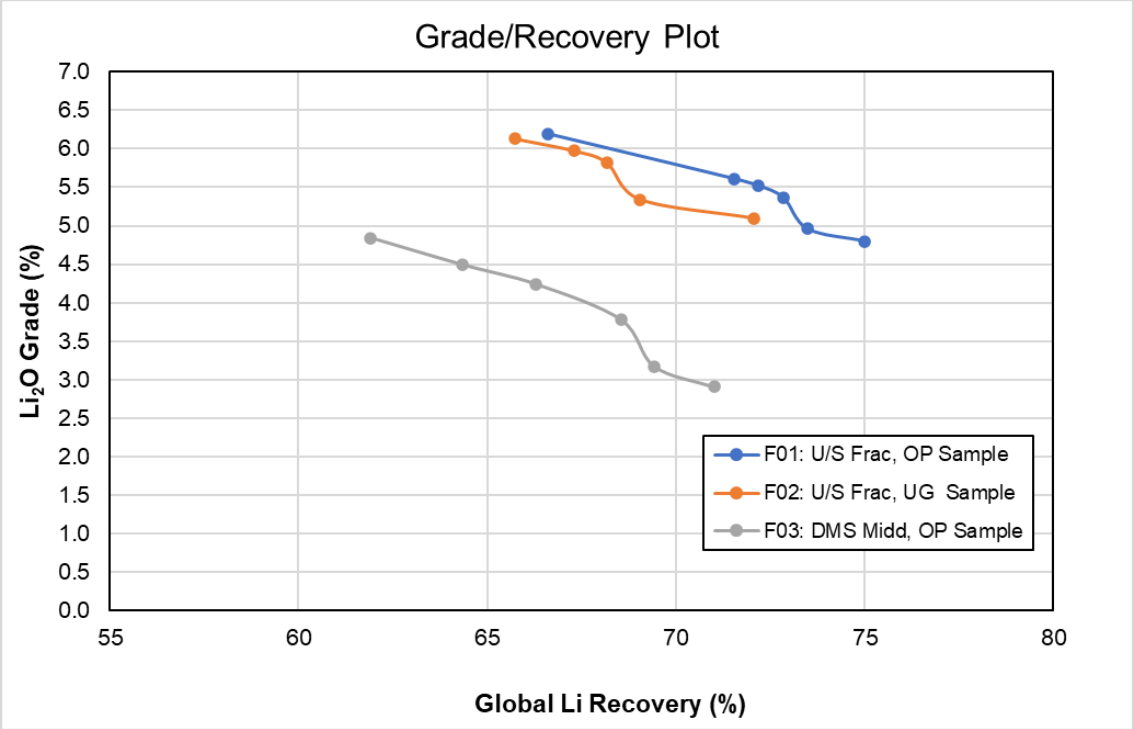
13.1.2.9.2 SGS - 19005-10

Following the 19005-10 DMS test, 2-kg samples of MC001 DMS bypass fraction (F01) and middling fraction (F03), as well as the MC002 DMS bypass fraction (F03), were used for spodumene flotation testing. The samples were ground to 100% passing 250 µm, de-slimed, processed through a magnetic separator with mica flotation of non-magnetic fraction, scrubbed and de-slimed again (to also increase solids

concentration for spodumene conditioning), then processed through spodumene flotation consisting of a rougher cell and three (3) stage cleaning cells, with a final magnetic separator to remove iron from the concentrate.

The results of the flotation testwork are displayed in Figure 13.15. The points from right to left represent the rougher and scavenger concentrate, rougher concentrate, 1st cleaner concentrate, 2nd cleaner concentrate, 3rd cleaner concentrate and non-magnetic 3rd cleaner concentrate. The bypass flotation circuit produced Li_2O concentrate product with a grade >6% with recoveries between 65-68%. The addition of the flotation circuit to treat the bypass material of the sample led to an increase in global Li_2O recovery of ~6.5%. The DMS middlings fraction did not meet the grade target of 5.5% Li_2O.

Figure 13.15: 19005-10 Spodumene Flotation Testwork Results



13.1.2.10 Filtration

Filtration and settling testwork was conducted by SGS on the bypass from the Open Pit (MC001) and Underground (MC002) DMS tests. The objective was to assess pre-thickening and filtration of the fines stream, with and without flocculant. Settling tests were completed at 10% w/w solids to compare natural settling versus flocculated conditions, while filtration tests were conducted at two setpoints, 32% and 40% w/w.

Both samples showed excellent settling behaviour, with rapid settling rates and high underflow densities achievable in all cases. Slightly cloudy overflow was observed in unflocculated conditions, but overall performance aligned with expectations for this material type. Filtration tests on the thickened slurry (with and without flocculant) yielded high filtrate quality, with form times ranging from 14 to 349 seconds, filtration capacities between 303 and 1,479 L/m²h, and final cake moistures of 13.2-18.7%. As shown in Figure 13.16, flocculation had a clear benefit in terms of reducing form time and improving throughput. Based on this testwork, the proposed design incorporates a pre-thickening step ahead of vacuum filtration.

Figure 13.16: Summary of Filtration Tests

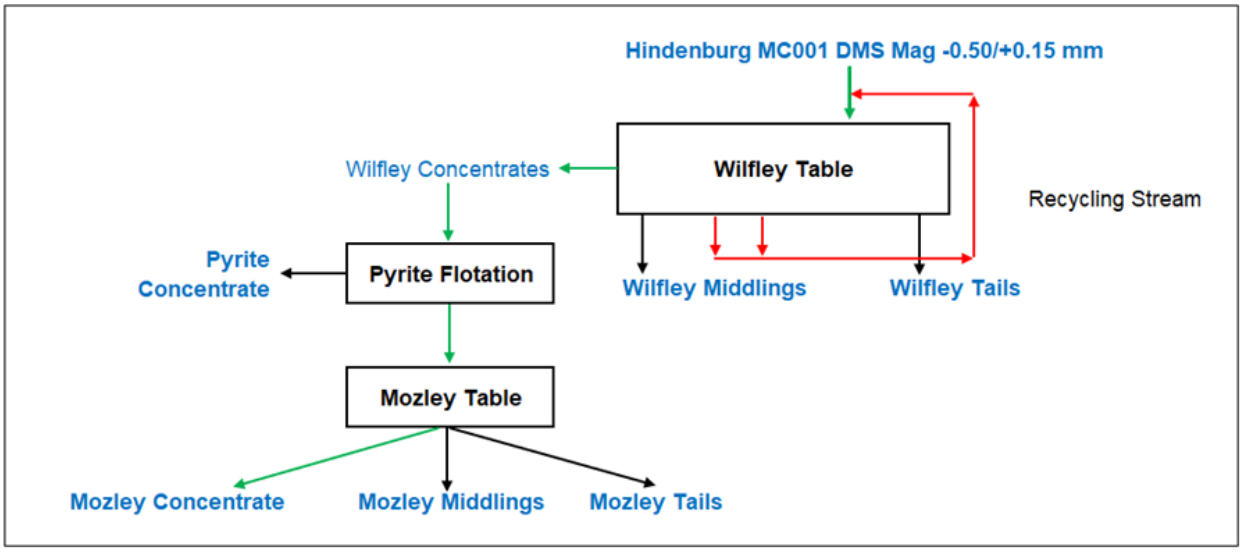
	MC001		MC001		MC002		MC002	
	40%		32%		40%		32%	
Test ID	1	2	1	2	1	2	3	4
Actual cake thickness, m	0.013	0.014	0.013	0.014	0.013	0.014	0.014	0.014
Filter area m^{2}	0.01		0.01		0.01		0.01	
Actual cake volume, L	0.13	0.140	0.13	0.14	0.13	0.140	0.14	0.14
Cake wet weight, kg	0.228	0.216	0.228	0.224	0.227	0.227	0.229	0.229
Cake dry weight, kg	0.187	0.187	0.187	0.189	0.187	0.188	0.186	0.188
Wet cake solids,%, S	82.2	86.8	82.1	84.4	82.4	82.7	81.3	82.2
Wet cake density, kg/L	1.747	1.539	1.749	1.593	1.74	1.617	1.633	1.631
Dry cake density, kg/L	1.435	1.335	1.436	1.345	1.434	1.338	1.327	1.341
Filtrate volume, L	0.23	0.255	0.36	0.36	0.235	0.235	0.36	0.36
Filter loading, L/m^2, L= (Volume/Area)	35.9	39.4	48.9	49.9	36.4	37.4	49.9	49.9
Form time, h	0.061	0.004	0.05	0.023	0.097	0.054	0.093	0.042
Dry time, h	0.023	0.023	0.023	0.023	0.023	0.023	0.023	0.023
Form/Dry Time Ratio	2.66	0.16	2.17	1	4.22	2.36	4.04	1.83
Total filtration time (form+dry), h	0.084	0.027	0.073	0.046	0.12	0.077	0.116	0.065
Filtrate TSS, mg/L	30	<10	21	19	13	<10	24	28
* Calculated filter throughput versus test conditions								
Vacuum level, form, inches mercury	20	20	20	20	20	20	20	20
Vacuum level, dry, inches mercury	14	14	20	20	19	19	20	20
Feed solids, %wt	40	40	40	40	40	40	0	0
Final wet cake moisture, %wt	17.8	13.2	17.9	15.6	17.6	17.3	18.7	17.8
Dry solids capacity, $kg/m^2 h$	222	701	256	409	155	243	160	288
Filter capacity, L/m^2h	427	1479	670	1085	303	485	430	766
* From raw test data / not for sizing any particular type of filter. Conversion: 0.098 bar = 2.896" Hg = 1 mH_2O = 3.28 ft H_2O = 1.42psi								

13.1.2.11 Gravity Separation

Gravity separation testwork was performed by SGS to assess the feasibility of producing a saleable tantalite concentrate from non-product streams of the DMS plant. The testwork was conducted on the MC001 magnetic rejects sample from the DMS program, selected due to its relatively high tantalum grade (240 ppm). The test feed was initially crushed to below 0.500 mm and screened into two (2) size fractions: coarse (-0.50 mm +0.15 mm) and fine (-0.15 mm). Both fractions were first processed via gravity separation using a Wilfley shaking table, followed by a Mozley table. However, after the Wilfley table stage, a high

concentration of pyrite was observed in the gravity concentrate. To address this, a pyrite flotation step was introduced between the two (2) gravity stages. The details of the testwork flowsheet are presented in Figure 13.17.

Figure 13.17: SGS Gravity Separation Flowsheet



The testwork demonstrated promising separation performance, with both size fractions yielding similar Ta_2O_5 recoveries. When the Mozley middling products are included, the overall recovery improves significantly, raising the combined stage recovery from 54.2% to 74.6%. Although this reduces the overall concentrate grade (from 18.6% to 9.54%), it is not expected to materially impact product value, as Ta_2O_5 concentrate pricing is generally based on contained Ta_2O_5. The results are summarized in Table 13.14.

Table 13.14: Stage Gravity Separation Results of MC001 DMS Magnetic Rejects

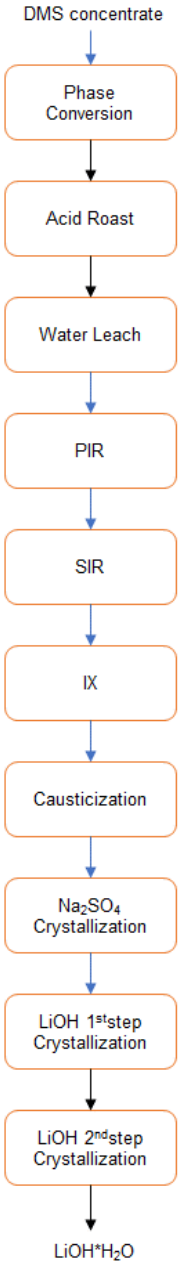
Size Fraction	Product	Mass (%)	Ta_2O_5 Grade (%)	Ta_2O_5 Distribution (%)
-0.50 + 0.15 mm	Mozley Conc	0.08	20.6	50.8
	Mozley Conc + Midds	0.18	12.5	74.4
-0.15 mm	Mozley Conc	0.06	15.8	61.7
	Mozley Conc +Midds	0.20	6.26	75.0
Combined-0.50 mm	Combined Mozley Conc	0.07	18.6	54.2
	Combined Mozley Conc + Midds	0.19	9.54	74.6

The stage gravity separation results demonstrated strong performance; however, the global recovery of Ta_2O_5 remains low at 10.1%. This is primarily due to the limited contribution of the DMS magnetic rejects stream to the overall mass balance. To improve global Ta_2O_5 recovery, additional non-product streams of the DMS beyond the magnetic rejects should be considered. The DMS middlings and DMS bypass fractions represent significant opportunities, together accounting for over 51% of the site's tantalum distribution (24.1% and 27.0%, respectively). Although the magnetic rejects show good potential, further testwork is recommended to evaluate the gravity response of these two (2) additional streams and better define the global recovery potential of an integrated tantalum circuit.

13.1.2.12 Lithium Hydroxide Conversion Test

A DMS spodumene concentrate produced from the CV5 Pegmatite was advanced through a testwork program to support flowsheet development for the conversion of spodumene concentrate into lithium hydroxide. The concentrate was prepared as part of SGS's 19005-02A test program, using a DMS-only flowsheet with magnetic separation, based on a master composite drill core sample representing anticipated early mine life. This master composite incorporated a 15% host-rock dilution to reflect a conservative mining scenario. The overall test program flowsheet is presented in Figure 13.18.

Figure 13.18: Lithium Conversion Testwork Flowsheet

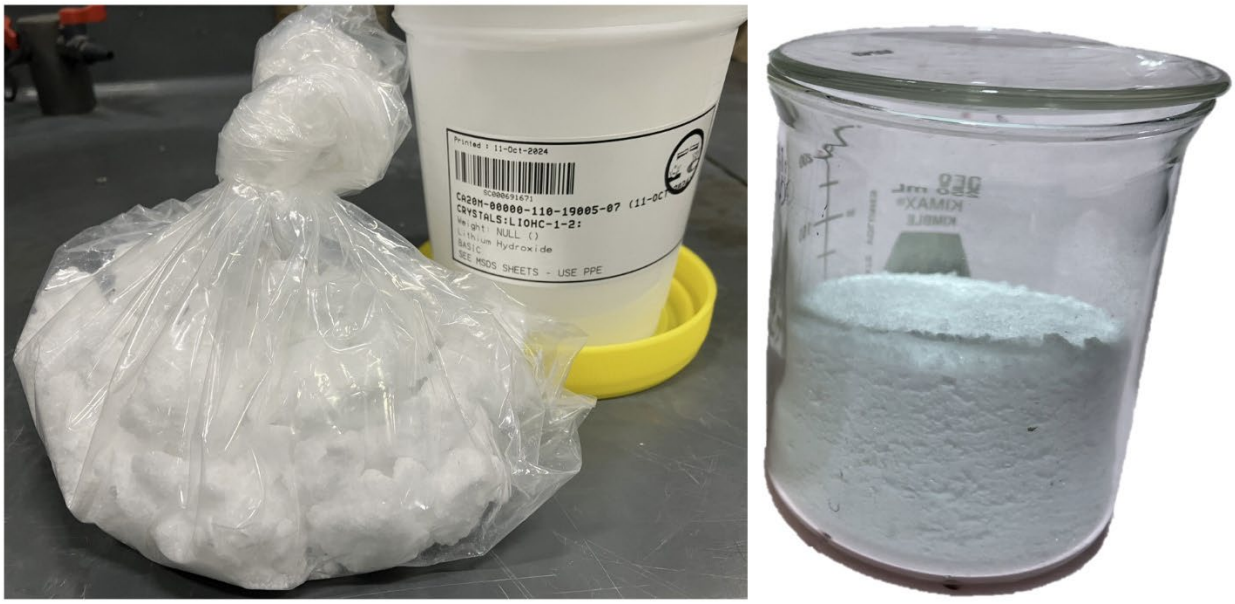


The spodumene concentrate (6.2% Li_2O, 0.60% Fe_2O_3) was calcined at 1,050°C and successfully converted α- to β-spodumene, and subsequent sulphuric acid roasting achieved ~98% lithium extraction into solution. Water leaching at mild conditions maintained high recoveries, with overall lithium extraction through leach and impurity removal stages measuring 87%. Primary impurity removal (PIR) was accomplished by selective lime precipitation with air oxidation, removing the bulk of iron and aluminum while minimizing lithium losses. Secondary impurity removal (SIR) further reduced calcium and magnesium

by >98% with negligible co-precipitation (<0.1% lithium loss). Residual calcium and magnesium were further reduced by ion exchange to concentrations below 1 mg/L.

The purified lithium sulphate solution was then causticized with sodium hydroxide to form lithium hydroxide, with excess sodium removed via Glauber's salt crystallization. Lithium hydroxide monohydrate was recovered in a two-step crystallization sequence, producing a high-purity, battery-grade product shown in Figure 13.19.Figure 13.19

Figure 13.19: On Spec Battery-Grade Lithium Hydroxide Monohydrate Product



This program demonstrated that spodumene concentrate from CV5 can be successfully processed into lithium hydroxide using a conventional sulphate-based flowsheet, with high lithium recoveries and product quality supportive of battery-grade specifications.

13.2 Basis for Recovery and Throughput Estimates

The CV5 HLS variability testwork was used as the basis for determining a feed grade vs. recovery trend to be used to assist with Mineral Resource Estimate cut-off grades.

To allow for changes in recovery that may be exhibited when scaling up from laboratory testwork to expected recovery in an operating DMS plant, the HLS testwork data were adjusted.

Recovery via HLS is “ideal” compared to recovery by a DMS cyclone in an operating plant. A reduction in recovery is expected and an “offset” of 2% to 10% from HLS results is typically observed. This “offset” is dependent on factors such as lithology, liberation, and quantity of near SG composite particles.

Specific items that account for scale down are the following:

- The laboratory crush size is typically coarse compared to the benchmark modelled and operating crushed mineralized material PSDs. Testwork particle sizing indicates 8% to 12% passing 0.65 mm, with 14% to 18% passing 0.65 mm typically expected in operation. This increase in bypass material results in a loss of recovery.
- HLS results are based solely on the density of the particle. The density of the particle is still an important factor in estimating performance in a DMS processes; however, the size of the particle is also a factor. As such, if the size range of the material reporting a DMS processes is too large, the risk of the smaller heavy particles reporting to the floats is increased. Common practice is to split the material into multiple size fractions to mitigate this fact, but even so, there are some in recovery losses associated with this phenomenon.

The predicted grade vs. recovery curve in Figure 13.20 considers the testwork recovery values from each variability composite sample and leverages the results to offset the ideal HLS testwork results. Taking into consideration the scale-up inefficiencies from HLS to DMS and finer size distribution changes within the plant compared to laboratory results, Primero can utilize its extensive experience in DMS operations to model the predicted recovery curve.

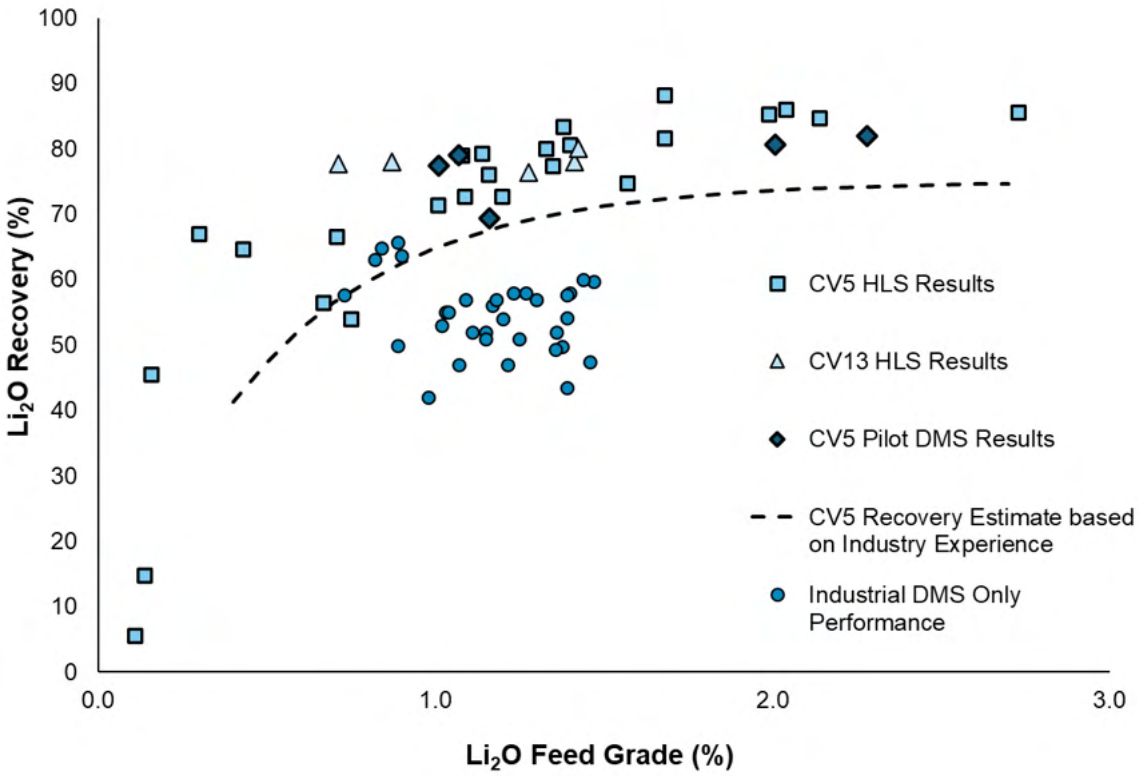
The predicted grade vs. recovery data for the HLS results is shown in the equation below:

$$Recovery\ \% = Max\ Recovery\ \% \times \left(1 - e^{-C(Li_2O\ Feed\ Grade\ \%)}\right)$$

$$Recovery\ \% = 75\ \% \times \left(1 - e^{-1.995(Li_2O\ Feed\ Grade\ \%)}\right)$$

This trend indicates that recoveries of approximately 70%–75% Li_2O at feed grades above 1.4% Li_2O, recoveries above approximately 60% Li_2O are possible at feed grades above 0.9% Li_2O and recoveries of approximately 50% Li_2O up to 60% Li_2O are possible at feed grades above 0.7% Li_2O.

Figure 13.20: Metallurgical Testwork Recovery Results and Industry-based Recovery Estimates for 3x Size Range DMS Process Plant



This relationship is dependent on both the spodumene grain size and grain size distribution in the deposit. Mineralogy with finer grained and/or wider grain size distributions will have less spodumene reporting to the concentrate stream at a given size. To date, the Shaakichiuwaanaan Deposit has demonstrated a crystal size and distribution that shows good liberation characteristics at 9.5 mm.

13.3 Processing Factors and Deleterious Elements

The variability samples tested to date indicate that the CV5 and CV13 pegmatites are amenable to a DMS-only spodumene concentrator flowsheet fed at a coarse crush size. To be amenable to this style of flowsheet, spodumene needs to be the predominant mineral that hosts lithium, and it needs to generally exhibit coarse crystal sizes so that it can successfully be liberated and recovered at coarse particle sizes (i.e., approximately 4 mm to 10 mm).

Data to date indicates that coarse spodumene lithology dominates CV5 and CV13 at grades, mostly, above 1.0% Li_2O. However, if significant quantities of finer lithology, alternate mineralogy, or low-grade material (<0.6% Li_2O) are identified, this material will require testing to confirm its expected behaviour in a

DMS plant. Lithology with finer spodumene crystals may not liberate successfully, which may result in reduced recovery or grade.

Some of the drilling and samples tested to date have been identified to have low levels of lepidolite. If samples are identified with higher quantities of lithium in non-spodumene mineral species, such as lepidolite or petalite, lower recoveries to DMS concentrate may be expected. This is due to the SG of minerals such as petalite being closer to gangue mineral SG when compared to the SG of spodumene.

High concentrations of some gangue minerals can complicate spodumene concentration by DMS. Higher concentrations of iron bearing minerals from within the pegmatite such as tourmaline or some micas can result in higher iron grades in the final DMS concentrate. For most of the HLS tests conducted so far, iron grades in the concentrate were satisfactorily low or able to be controlled using magnetic separation.

Higher iron can also potentially come from host rock species such as amphibolite, anorthosite, or meta-sediment. If unacceptable levels of host rock dilution are encountered, they can be mitigated by conservative mining methods to reduce dilution, ROM blending, or magnetic separation. Ore sorting to reject dilution is also potentially an option if required. The bulbous structure of the CV5 Pegmatite is fortunate and may help when aiming to maintain low concentrations of host rock in the DMS plant feed.

Mica can misreport to the concentrate in the DMS due to its platelike shape. High concentration of mica in the process feed can result in recoveries or grades being lower than expected. To mitigate this, levels of mica will be tracked and, if required, up flow classification can be added to the flowsheet to reject mica ahead of DMS.

The crushed mineralized material particle is important for global recovery estimates. It is typical to expect 14% to 18% of mass to report to the -0.65 mm and not be processed by DMS. If finer PSDs are generated due to the manner in which mining or crushing is conducted, the quantity of fines produced can increase significantly to as high as 30% passing -0.65 mm, resulting in reduced global DMS recovery. This risk can be reduced by appropriate mining and crushing circuit design and operation.

Like the point raised prior, an alternative mechanism for higher proportion of -0.65 mm can come from the nature of the rock. In cases where the rock exhibits high friability (the size of particles that a rock breaks into, high friability results, is many fine particles), the amount of bypass is higher. This is typical of materials that have experienced large degrees of weathering. To date, there has been little evidence of high amounts of weathering in the Shaakichiuwaanaan Deposit. Samples from the surface of the deposit and/or from around faults can be tested for their friability in the future.

13.4 Conclusions

Mineralogical, DMS, HLS, flotation, filtration, and gravity separation testwork was carried out by SGS Canada at their Lakefield, Ontario, facility using drill core composite samples from the CV5 Pegmatite, CV5 Host Rock, and CV13 Pegmatite. The following conclusions have been made:

- Testwork supports a DMS-only process flowsheet to produce a spodumene concentrate grade of >5.5% Li_2O and <2% Fe_2O_3. Testwork Li_2O recoveries of 70% to 85% were achieved for HLS testwork (for feed grade in the range 1.0% to 1.5% Li_2O, respectively).
- Testwork completed on CV5 includes five (5) DMS tests and 26 Heavy Liquid Separation (HLS) and magnetic separation tests. The HLS and magnetic separation tests were conducted using 26 composites from across the CV5 Deposit.
- Coarse spodumene was found to be the dominant lithium mineral species across all samples with minor quantities of lepidolite (values range between 0% to 4.3% with an average of 0.98%) and moderate quantities of mica (values range between 0% to 17.1% with an average of 6.50%) observed.
- Three (3) pilot DMS tests (cyclone diameter of 250 mm) were completed. Table 13.15 summarises the global Li_2O feed grades (before fines screening), global lithium recoveries and the Li_2O and Fe_2O_3 grades of the concentrates achieved. These results strongly support adopting a DMS-only process flowsheet.

Table 13.15: Pilot DMS Results

DMS Feed Li_2O Grade (%)	Global DMS Lithium Recovery (%)	Concentrate Li_2O Grade (%)	Concentrate Fe_2O_3 Grade (%)
1.01	77.4	5.64	0.55
1.07	79.0	5.77	0.62
1.16	69.4	6.21	0.60

- Fe_2O_3 grades in HLS concentrates were in the range 0.52%–1.79% and after magnetic separation was applied to 15 of the 24 composites, all concentrates were <2% Fe_2O_3.
- The 26 CV5 HLS variability test results were adjusted to more appropriately represent recoveries expected in an operating DMS plant. After fitting a trend to this data, it indicates:
  - Recoveries of 70%–75% Li_2O expected at feed grades above 1.4% Li_2O.
  - Recoveries of 60%–70% Li_2O expected at feed grades of 0.9%–1.4% Li_2O.
  - Recoveries of 50%–60% Li_2O are possible at feed grades of 0.7%–0.9% Li_2O.

- Flotation was performed on sample composed of DMS middlings (second stage DMS floats) combined with the DMS bypass fraction (i.e., -0.85 mm). The global Li_2O recovery was improved from 79.0% (the DMS-only recovery) to 89.1% (DMS followed by flotation). Flotation spodumene concentrate returned a grade of 5.49% Li_2O and 0.40% Fe_2O_3. Flotation shows promise to potentially be added to a DMS-only plant at some stage in the future once operational.
- HLS testwork confirmed strong lithium recoveries when splitting the traditional fines fraction into fines and ultrafines, validating the current flowsheet design.
- Reprocessing of the coarse middlings via recrush and HLS demonstrated a notable recovery benefit. The -3.3 mm fraction achieved a stage recovery of 52.16% and increased global Li_2O recovery by 2.67%, supporting its inclusion in the final flowsheet design.
- CRIMMs magnetic separation trials on DMS concentrate demonstrated effective iron rejection across all size fractions tested, including the coarse size fraction (-9.5+3.3 mm) initially thought unsuitable for magnetic separation.
- Thickening and filtration tests on both open pit and underground fines composites demonstrated fast settling rates and high-density underflow, with or without flocculant. Filtration tests showed excellent filtrate clarity and strong filtration capacities, supporting the use of a pre-thickening step prior to vacuum filtration, confirming that pressure filtration is not required.
- Gravity separation testwork on the DMS magnetic rejects stream produced promising Ta_2O_5 recoveries, particularly when Mozley middlings were included, boosting stage recovery from 54.2% to 74.6%. Although the global recovery remains modest at ~10%, the work highlights strong potential for further recovery from middlings and bypass streams.

13.5 Recommendations

As the Project advances, further testwork is recommended to further estimate future performance and ensure optimized results with the DMS-only flowsheet. These tests are:

- Comminution testwork for crushing (i.e., Bond Crushing Work Index “CWi). This work serves to confirm the crusher sizing and provides an indication of the size distribution feeding the plant.
- Additional magnetic separation testwork, particularly in the coarse concentrates following the results from CRIMMs. Further larger scale testwork is recommended to evaluate the unit's ability to operate at elevated throughputs and quantify the potential performance impacts of overloading the unit. Even if performance declines slightly under higher loads, this may still be a preferable option compared to bypassing the coarse fraction entirely, particularly since the coarse material performed well when included in testing.

- Due to the width and orientation of the CV5 Pegmatite lenses, the expected dilution of the plant feed is expected to be relatively low. However, there may be opportunities to maximize the extraction of spodumene concentrate from the deposit if parts of the deposit with higher dilution are directed to an ore sorting processing solution. Ore sorting testwork is planned for the next phase of testwork.
- Given the heavy reliance of a DMS flowsheet on the particle size distribution, it is recommended to characterise spodumene samples representative of a Run of Mine material generated from blasting activities. Samples generated from crushed drill core (basis of most exploratory testwork) does not generate the same size distribution. These samples have less fines in them. Therefore, a blasted bulk sample would validate the size distribution used in the design (which assumes more fines than that measured in the lab).

Opportunities exist for processing the bypass fraction of the DMS circuit (i.e., the minus 0.65 mm). As such, the following testwork is recommended to assess these opportunities:

- Gravity recovery testwork to determine if a tantalite recovery circuit should be included in the initial flowsheet to recover the high values of tantalum in the feed. The bypass fraction represents a large distribution of the available tantalum and following confirmation from testwork can potentially be processed without additional grinding.
- Processing of the 0.65 mm to 0.25 mm fraction without grinding can generate higher global recoveries. It is recommended to test technologies that can processes this size range such as coarse flotation (upflow / tetter-bed settlers with flotation modifying the settling characteristics of the spodumene) or gravity technologies (e.g., spirals).

14. MINERAL RESOURCE ESTIMATES

14.1 Resource Estimate

The QP completed a Mineral Resource Estimate of the CV5 and CV13 pegmatites (Li, Ta, Cs, Ga) at the Shaakichiuwaanaan Property. The effective date of the resource is June 20, 2025.

14.1.1 Database

PMET Resources maintains all drill data for the Shaakichiuwaanaan Property in the MX database. Header, surveys, assays, lithology, and geotechnical logging information are saved in the database. The final database information in CSV format was provided to the QP on June 20, 2025

The MRE for the CV5 Pegmatite area is supported by 555 diamond drill holes (DDH) of NQ (predominant) or HQ size, totalling a collective 188,695 m, and 179 m of channels. At CV13, the MRE is supported by 165 DDH (39,008 m) and 425 m of channels. The Rigel and Vega MRE and geological models, which are a subset within the CV13 Pegmatite, are supported by 6 DDH of NQ size totalling 1,228 m (Rigel) and by 26 diamond drill holes of NQ size totalling 6,580 m, and 7 m of channels (Vega).

The drilling includes programs in 2021, 2022, 2023 and 2024 (through the end of 2024; CV24-787). The resource estimation was conducted using Leapfrog Edge™ version 2024.1.

14.1.2 Specific Gravity

Up to hole CV24-787, PMET Resources collected a total of 24,675 samples from 719 drill holes for specific gravity (SG) measurements. The same methodology and regression function were used for CV5 and CV13.

PMET Resources used the following procedure to determine the average SG for each mineral domain:

- Samples were selected for SG measurement after the core was cut.
- The full length of each sample was measured for SG.
- Once the scale was calibrated, the sample was weighed dry.
- The sample was then weighed while submerged and saturated in tap water.
- The following equation was used to determine the SG:

Wd = Dry Weight, Ws = Submerged Weight

Figure 14.1 illustrates the SG measuring setup employed by PMET Resources during this round of SG data collection. Results are presented in Table 14.1. A linear regression formula based on the Li_2O and B_2O_3 percentage was used to calculate the density for all the pegmatites (Figure 14.2). The regression function is:

$$SG = 0.0674*(Li_2O\%+B_2O_3*0.81)+2.6202$$

Non-pegmatite blocks were assigned a fixed SG based on the field measurement median value of their respective modelled lithology.

Figure 14.1: SG Measurement Setup Employed by PMET Resources

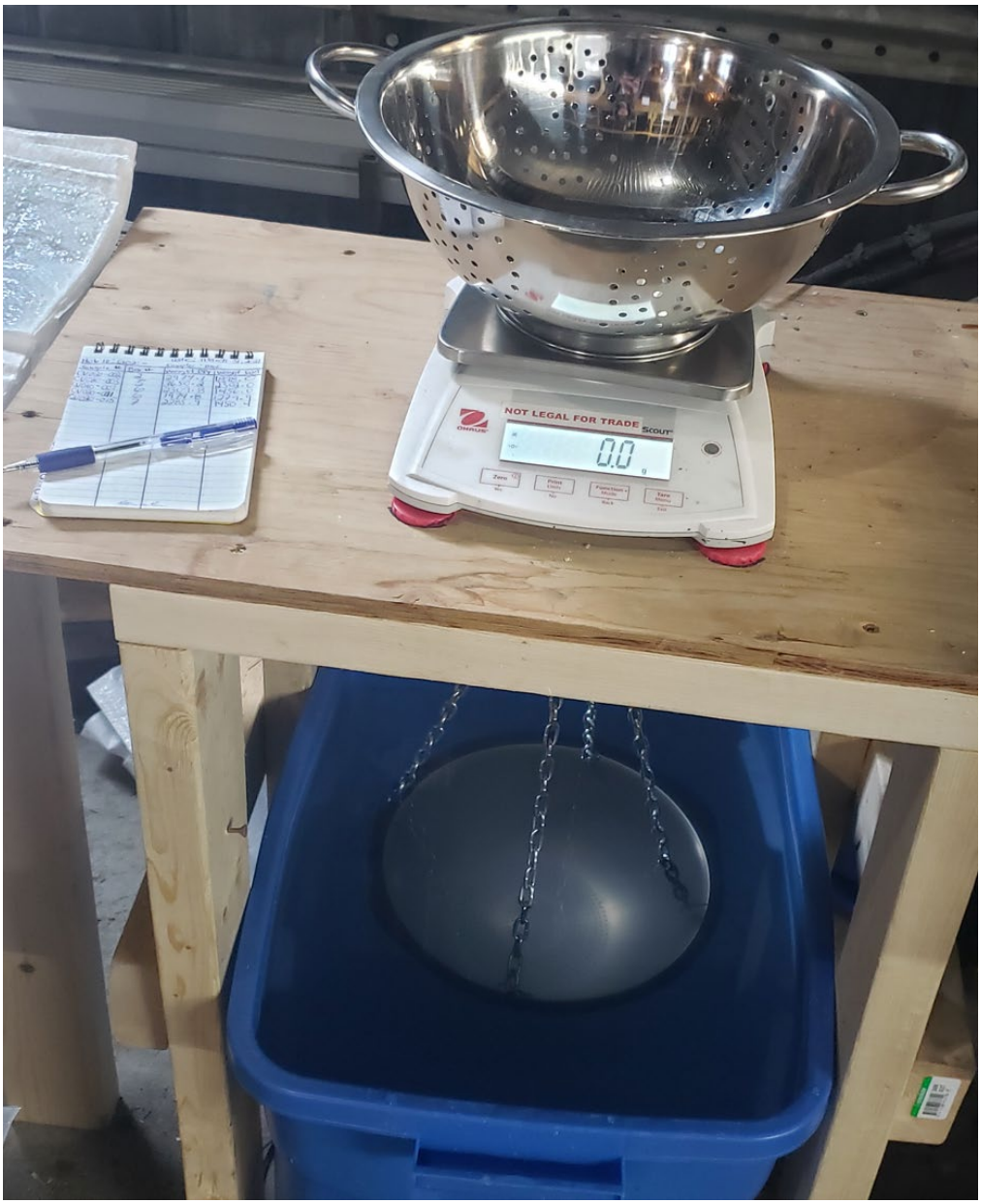
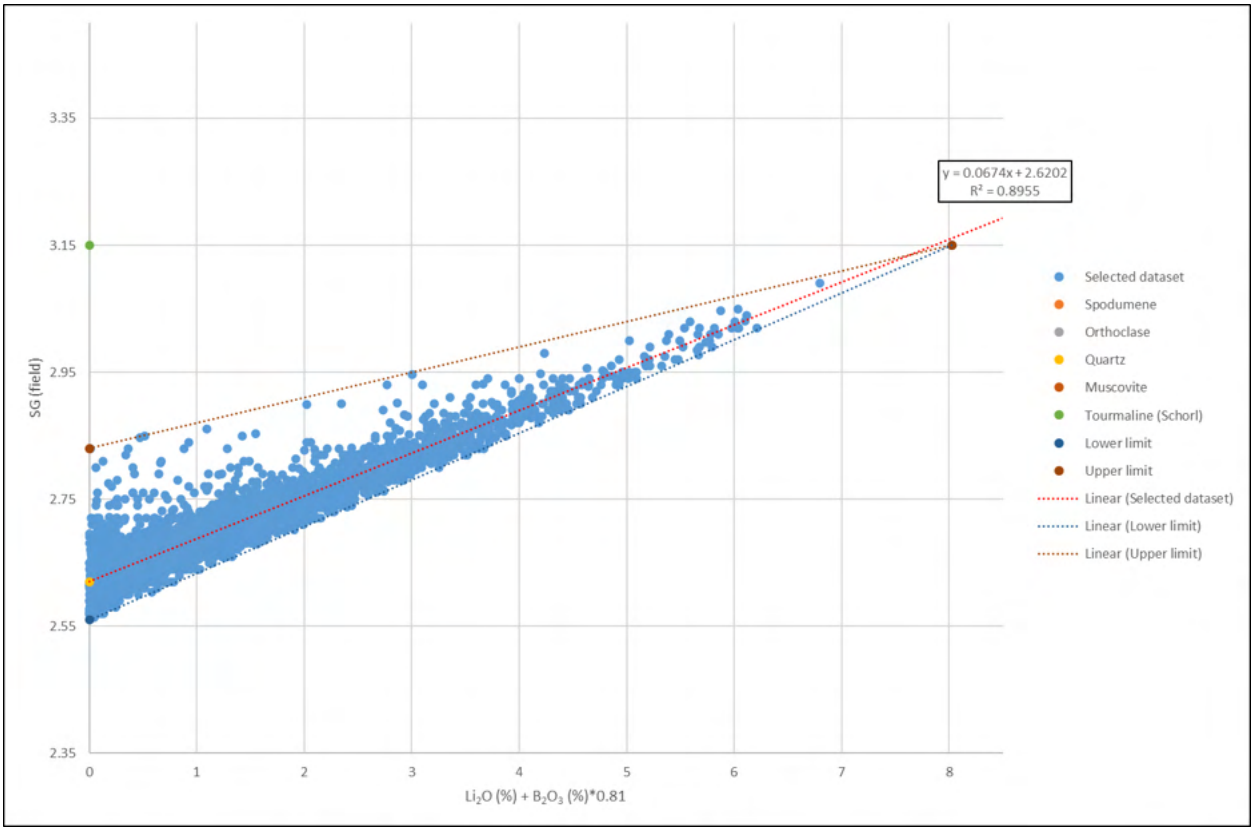


Table 14.1: MRE Specific Gravity Summary

Lithology	CV5 Rock Density	CV13 Rock Density
Pegmatite	Linear Regression Curve	Linear Regression Curve
Amphibolite	2.99	3.01
Ultramafic	2.94	3.02
Diabase	2.89	-
Metasediment	2.75	2.82
Iron Formation (Garnet + FeOx)	2.95	-
Iron Formation (Marbot)	2.84	-
Water	1.00	-
Overburden	2.00	2.00

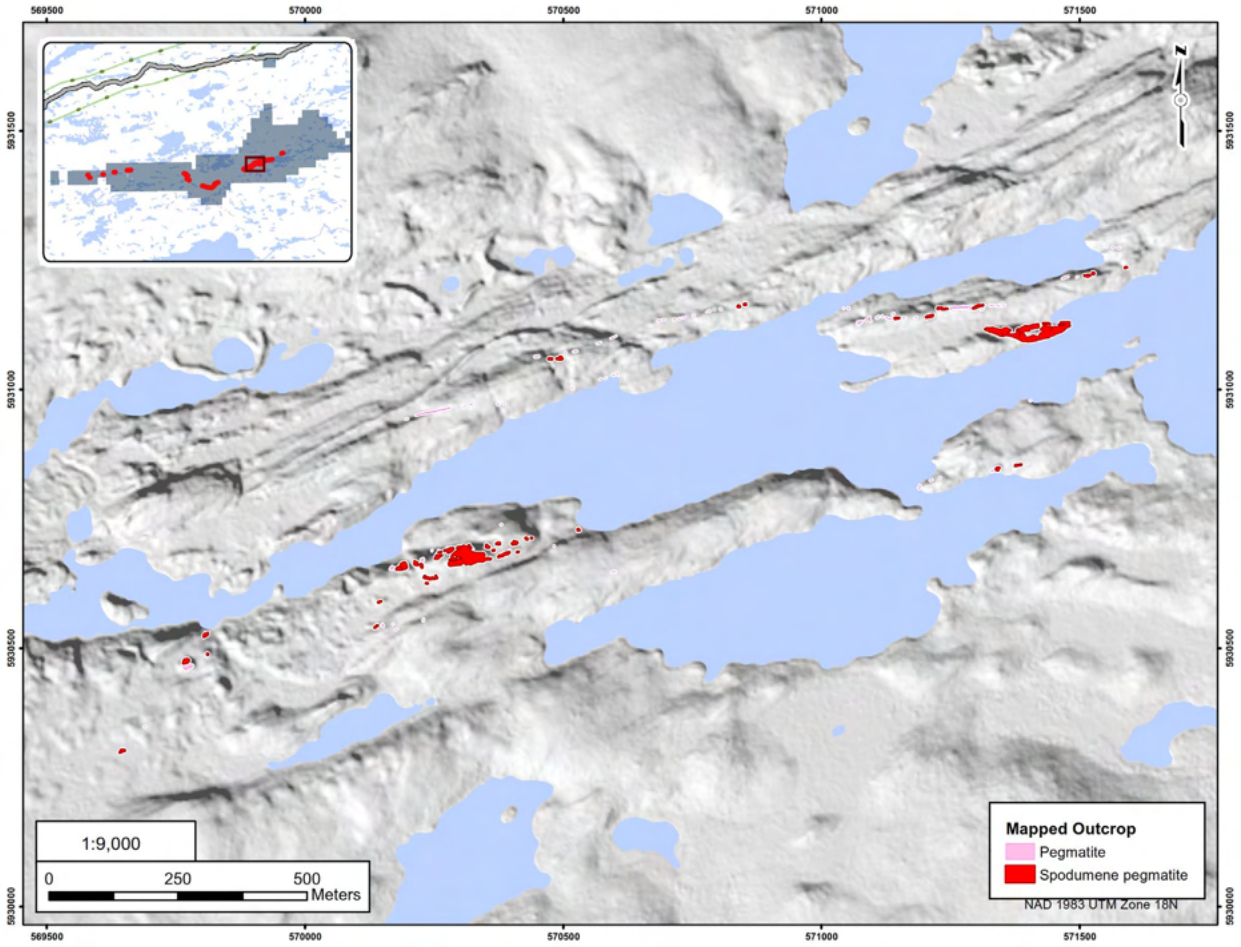
Figure 14.2: Regression Function SG Derivation



14.1.3 Topography Data

PMET Resources completed a property-wide LiDAR and orthophoto survey in August 2022, which provides high-quality topographic control. The quality and accuracy of the topographic controls are considered adequate for advanced stage exploration and development, including an MRE.

Figure 14.3: LiDAR Topographic Image with Spodumene Pegmatite Outcrops at CV5



14.1.4 Geological Interpretation

Three-dimensional wireframe models of mineralization were developed in Leapfrog Geo™ by PMET Resources and its geological consultants, respecting the guidelines and recommendations of the QP. The wireframes were based on the geological interpretation of the zones as distinct domains and not strictly on grade intervals. 3D modelling of the mineralized zones and interpretation were also based on structural data measurements and the regional trend. Zones were modelled with a combination of implicit and explicit modelling. Control lines were used to constrain the volume in specific orientations.

The wireframes extend at depth, below the deepest DDH and laterally. The resource model did not estimate grades into the full volume of the wireframes due to constraints on interpolation parameters.

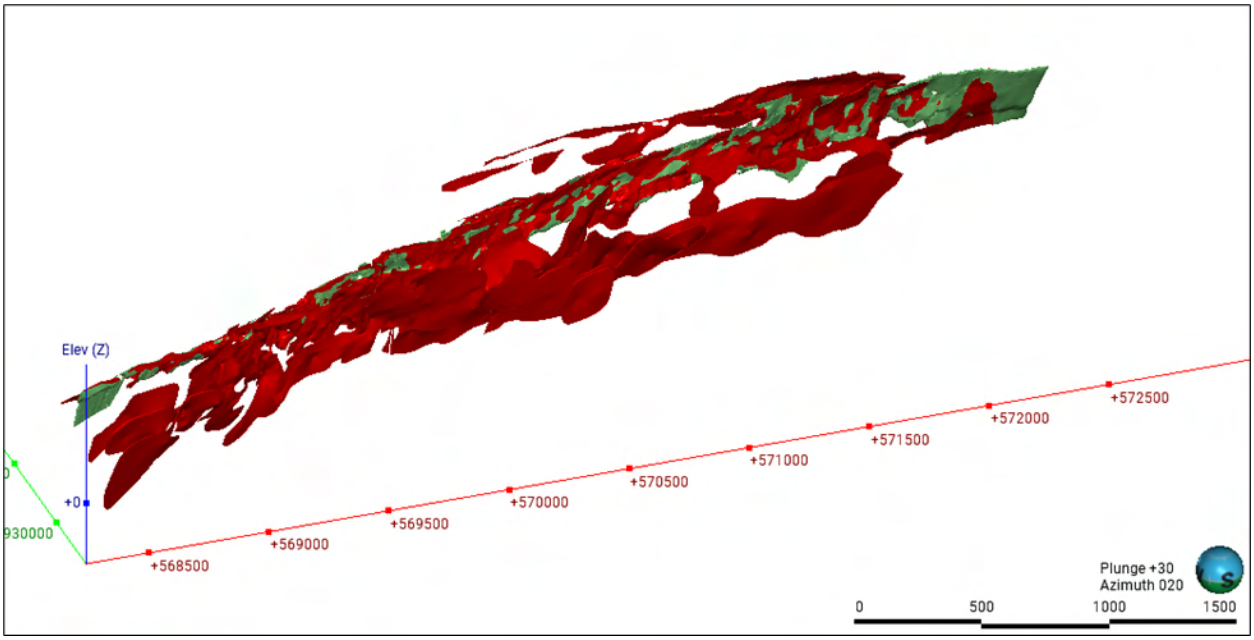
The non-assayed intervals were assigned half the detection limit value. The QP believes that non-assayed material should not be assigned a zero value, as this does not reflect the true value of the material. Each domain was modelled using the same principal assumptions and methodology.

14.1.4.1 CV5

The mineralized zones (Figure 14.4) were broken down into 11 different domains. The CV5 principal pegmatite was divided into two (2) domains (spodumene-rich (1) and feldspar-rich (2)). Nine (9) mostly parallel pegmatite dykes were geologically modelled for the MRE (CV5_110; CV5_120; CV5_130; CV5_140; CV5_150; CV5_160; CV5_170; CV5_180; and CV5_190).

The other units of the 3D model consist of amphibolite, ultramafic, metasediment, diabase, iron formation, overburden, and water.

Figure 14.4: CV5 Mineralized Zones



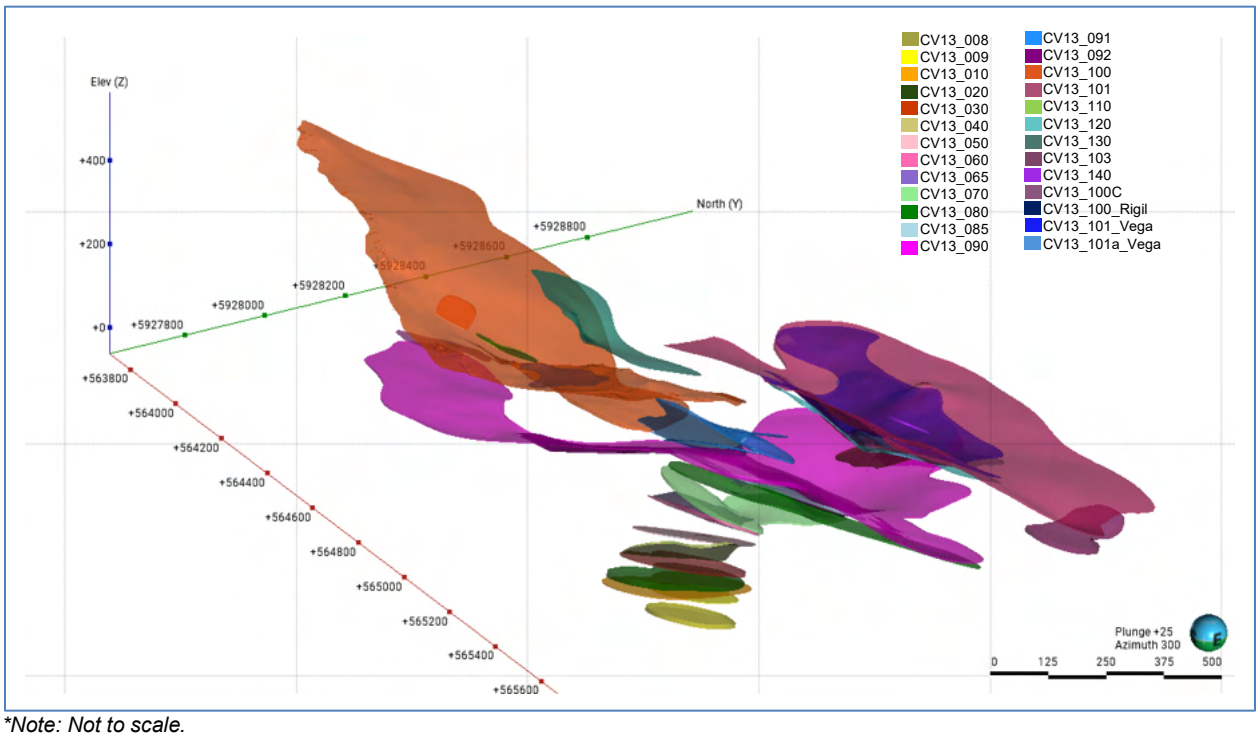
**Note: Red domains are spodumene-rich, and the green domain represents the feldspar-rich domain of the CV5 principal pegmatite (image not to scale).*

14.1.4.2 CV13

The CV13 mineralized zones (Figure 14.5) were broken down into 23 different domains. Domains are mostly subparallel to each other.

The other units of the 3D model consist of amphibolite, ultramafic, metasediment and overburden.

Figure 14.5: Mineralized Zones of CV13



**Note: Not to scale.*

14.1.5 Exploratory Data Analysis

14.1.5.1 Assays

CV5

The 11 domains included in the MRE were sampled by a total of 22,567 assays. The assay intervals within each mineral domain were captured using the evaluation function in Leapfrog Geo™. These intervals were reviewed to ensure all the proper assay intervals were properly captured. Table 14.2 summarizes the basic statistics for the assays (Li_2O%, Ta_2O_5 ppm, Cs_2O%, and Ga ppm) intervals for each of the mineral domains at CV5 on the Property.

Table 14.2: Assays Summary by Domain (CV5; length-weighted)

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
Spod-Rich Peg	Li_2O (%)	12,692	0.00	6.88	1.48	1.26
	Cs_2O (%)	12,472	0.00	13.04	0.09	0.20
	Ta_2O_5 (ppm)	12,692	0.31	25,520.99	162.41	289.48
	Ga (ppm)	12,672	0.50	187.00	66.80	22.13
Felds Peg	Li_2O (%)	5,313	0.00	5.93	0.21	0.44
	Cs_2O (%)	5,245	0.00	3.46	0.06	0.09
	Ta_2O_5 (ppm)	5,313	0.31	11,291.51	159.96	295.11
	Ga (ppm)	5,303	0.50	189.00	50.32	17.40
CV5_110	Li_2O (%)	8	0.01	0.52	0.12	0.18
	Cs_2O (%)	8	0.01	0.03	0.02	0.01
	Ta_2O_5 (ppm)	8	41.27	232.01	106.57	80.12
	Ga (ppm)	8	47.00	77.00	55.88	7.27
CV5_120	Li_2O (%)	229	0.00	3.78	0.34	0.68
	Cs_2O (%)	229	0.00	5.31	0.09	0.38
	Ta_2O_5 (ppm)	229	0.31	2,119.83	137.52	196.44
	Ga (ppm)	229	1.00	110.00	52.49	17.24
CV5_130	Li_2O (%)	689	0.00	4.34	0.67	0.94
	Cs_2O (%)	689	0.00	1.72	0.06	0.11
	Ta_2O_5 (ppm)	689	0.31	3,039.32	129.24	234.74
	Ga (ppm)	689	9.00	214.00	55.89	17.78
CV5_140	Li_2O (%)	684	0.00	5.66	0.83	1.06
	Cs_2O (%)	684	0.00	2.52	0.07	0.17
	Ta_2O_5 (ppm)	684	0.31	914.60	120.31	105.55
	Ga (ppm)	679	4.00	125.00	57.38	18.06
CV5_150	Li_2O (%)	926	0.00	6.84	0.82	0.97
	Cs_2O (%)	926	0.00	5.45	0.10	0.30
	Ta_2O_5 (ppm)	926	0.31	2,313.98	138.14	142.20
	Ga (ppm)	920	7.00	149.00	57.28	18.28
CV5_160	Li_2O (%)	1,160	0.00	6.12	1.03	1.18
	Cs_2O (%)	1,160	0.00	7.23	0.09	0.25

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
	Ta_2O_5 (ppm)	1,160	0.31	2,422.66	161.37	180.32
	Ga (ppm)	1,157	0.50	196.00	58.50	21.97
CV5_170	Li_2O (%)	233	0.00	4.66	0.82	1.13
	Cs_2O (%)	233	0.00	0.71	0.06	0.09
	Ta_2O_5 (ppm)	233	0.31	2,270.02	174.70	177.36
	Ga (ppm)	231	2.00	173.00	58.94	19.21
CV5_180	Li_2O (%)	598	0.00	4.91	0.80	1.00
	Cs_2O (%)	598	0.00	2.47	0.09	0.21
	Ta_2O_5 (ppm)	598	0.31	2,434.87	152.18	195.08
	Ga (ppm)	588	3.00	132.00	50.45	20.31
CV5_190	Li_2O (%)	35	0.00	1.34	0.20	0.36
	Cs_2O (%)	35	0.00	0.25	0.06	0.07
	Ta_2O_5 (ppm)	35	12.09	1,464.10	231.37	305.98
	Ga (ppm)	35	23.00	85.00	57.43	11.15

CV13

The 23 domains included in the MRE were sampled by a total of 3,382 assays. The assay intervals within each mineral domain were captured using the evaluation function in Leapfrog Geo™. These intervals were reviewed to ensure that all appropriate assay intervals were correctly captured. Table 14.3 summarizes the basic statistics for the assay (Li_2O%, Ta_2O_5 ppm, Cs_2O%, and Ga ppm) intervals for each mineral domain on the Property. Table 14.4 summarizes the basic statistics for the assay (Cs_2O%) intervals for each mineral subdomain at CV13 on the Property, including the caesium-rich domains corresponding to the Rigel (CV13_100) and Vega (CV13_101) zones.

Table 14.3: Assay Summary by Domain (CV13; Length-weighted)

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV13_008	Li_2O (%)	9	0.00	0.12	0.02	0.03
	Cs_2O (%)		0.00	0.01	0.01	0.00
	Ta_2O_5 (ppm)		20.27	151.42	85.33	39.25
	Ga (ppm)		47.00	72.00	63.13	6.98

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV13_009	Li_2O (%)	2	0.01	0.03	0.01	0.02
	Cs_2O (%)		0.02	0.03	0.03	0.01
	Ta Ta_2O_5 (ppm)		17.34	46.89	30.92	20.90
	Ga (ppm)		34.00	42.00	37.68	5.66
CV13_010	Li_2O (%)	4	0.01	0.03	0.02	0.01
	Cs_2O (%)		0.00	0.03	0.02	0.01
	Ta_2O_5 (ppm)		31.02	92.93	58.26	27.07
	Ga (ppm)		39.00	51.00	44.50	5.83
CV13_020	Li_2O (%)	19	0.01	0.29	0.07	0.07
	Cs_2O (%)		0.00	0.08	0.03	0.03
	Ta_2O_5 (ppm)		5.13	243.00	52.15	53.34
	Ga (ppm)		20.00	61.00	42.92	12.53
CV13_030	Li_2O (%)	10	0.01	0.11	0.06	0.03
	Cs_2O (%)		0.00	0.05	0.02	0.02
	Ta_2O_5 (ppm)		1.59	145.31	59.92	39.25
	Ga (ppm)		20.00	63.00	45.19	13.17
CV13_040	Li_2O (%)	21	0.01	0.25	0.07	0.06
	Cs_2O (%)		0.00	0.07	0.02	0.02
	Ta_2O_5 (ppm)		0.85	616.66	105.33	137.75
	Ga (ppm)		17.00	74.00	48.39	18.43
CV13_050	Li_2O (%)	18	0.00	1.09	0.14	0.27
	Cs_2O (%)		0.01	0.10	0.05	0.03
	Ta_2O_5 (ppm)		17.71	197.82	81.72	58.00
	Ga (ppm)		34.00	69.00	51.60	9.82
CV13_060	Li_2O (%)	16	0.01	0.18	0.05	0.05
	Cs_2O (%)		0.00	0.06	0.03	0.02
	Ta_2O_5 (ppm)		11.72	285.74	95.77	81.37
	Ga (ppm)		20.00	88.00	49.70	16.66
CV13_065	Li_2O (%)	9	0.00	0.21	0.08	0.08
	Cs_2O (%)		0.00	0.00	0.00	0.00
	Ta_2O_5 (ppm)		0.31	271.08	101.01	83.63

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
	Ga (ppm)	8	49.00	75.00	58.41	8.35
CV13_070	Li_2O (%)	26	0.00	0.19	0.02	0.03
	Cs_2O (%)		0.00	0.16	0.02	0.03
	Ta_2O_5 (ppm)		0.31	440.82	82.10	86.57
	Ga (ppm)	23	17.00	79.00	53.27	12.81
CV13_080	Li_2O (%)	105	0.00	2.25	0.19	0.39
	Cs_2O (%)		0.00	0.31	0.03	0.04
	Ta_2O_5 (ppm)		0.31	1,817.00	124.34	220.66
	Ga (ppm)	104	24.00	122.00	53.66	15.86
CV13_085	Li_2O (%)	26	0.02	2.31	0.22	0.50
	Cs_2O (%)		0.00	0.52	0.06	0.11
	Ta_2O_5 (ppm)		7.82	252.77	115.56	76.08
	Ga (ppm)		22.00	83.00	47.56	14.51
CV13_090	Li_2O (%)	363	0.00	4.05	0.46	0.81
	Cs_2O (%)		0.00	1.41	0.06	0.11
	Ta_2O_5 (ppm)		0.31	1,163.71	93.60	111.86
	Ga (ppm)	358	10.00	133.00	52.66	18.63
CV13_091	Li_2O (%)	56	0.00	2.05	0.33	0.56
	Cs_2O (%)		0.00	3.82	0.18	0.67
	Ta_2O_5 (ppm)		0.31	1,257.73	139.38	237.16
	Ga (ppm)	54	24.00	208.00	70.87	41.46
CV13_092	Li_2O (%)	10	0.03	0.16	0.08	0.05
	Cs_2O (%)		0.01	0.40	0.08	0.13
	Ta_2O_5 (ppm)		116.25	382.20	200.61	77.97
	Ga (ppm)		49.00	91.00	61.61	13.71
CV13_100	Li_2O (%)	1,590	0.00	6.82	0.92	1.07
	Cs_2O (%)		0.00	26.61	0.19	1.51
	Ta_2O_5 (ppm)		0.31	12,455.22	120.86	422.75
	Ga (ppm)	1,586	0.50	196.00	61.43	22.22

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV13_100C	Li_2O (%)	11	0.02	0.32	0.09	0.09
	Cs_2O (%)		0.01	0.12	0.06	0.04
	Ta_2O_5 (ppm)		20.76	168.51	72.20	50.86
	Ga (ppm)		39.00	82.00	53.40	13.44
CV13_101	Li_2O (%)	971	0.00	7.11	1.21	1.67
	Cs_2O (%)		0.00	26.61	0.43	1.39
	Ta_2O_5 (ppm)		0.31	8,431.70	187.37	399.60
	Ga (ppm)		0.50	250.00	65.98	34.24
CV13_103	Li_2O (%)	10	0.00	1.03	0.25	0.38
	Cs_2O (%)		0.00	0.07	0.03	0.03
	Ta_2O_5 (ppm)		1.47	291.84	156.18	78.17
	Ga (ppm)		27.00	72.00	51.03	12.75
CV13_110	Li_2O (%)	11	0.01	1.77	0.35	0.60
	Cs_2O (%)		0.00	1.48	0.21	0.39
	Ta_2O_5 (ppm)		1.34	266.20	95.47	89.86
	Ga (ppm)		13.00	78.00	53.74	19.72
CV13_120	Li_2O (%)	54	0.01	4.60	0.69	1.29
	Cs_2O (%)		0.00	6.20	0.34	0.99
	Ta_2O_5 (ppm)		2.69	691.14	171.72	132.45
	Ga (ppm)		14.00	130.00	60.81	24.03
CV13_130	Li_2O (%)	35	0.00	2.96	0.49	0.83
	Cs_2O (%)		0.00	6.95	0.56	1.43
	Ta_2O_5 (ppm)		0.31	923.15	109.22	169.62
	Ga (ppm)		16.00	106.00	58.77	20.83
CV13_140	Li_2O (%)	4	0.02	1.83	0.50	0.91
	Cs_2O (%)		0.02	0.16	0.07	0.07
	Ta_2O_5 (ppm)		61.91	244.22	144.34	88.40
	Ga (ppm)		50.00	70.00	57.04	9.68

Table 14.4: Cs_2O Assays Summary by Refined Domain (CV13; length-weighted)

Domain	Refined Domain	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV13_100	Rigel_Cs: CV13_Cs_05P_100	Cs_2O (%)	19	0.67	26.61	8.95	8.80
CV13_101	Vega_Cs: CV13_Cs_05p_101	Cs_2O (%)	65	0.01	26.61	2.73	3.38
	Vega_Cs: CV13_Cs_05p_101a		29	0.03	13.04	3.12	3.08

14.1.5.2 Compositing

Compositing of all the assay data within the various domains was completed on downhole intervals, honouring the interpretation of the geological solids. Statistics indicate that a majority of the samples were collected at 1 m intervals.

Compositing was done in Leapfrog Edge™ using a 1 m interval. For residual length less than 0.5 m, it was redistributed equally within the domain. Table 14.5 summarizes the statistics for CV5 domains after compositing, while the statistics for CV13 are summarized in Table 14.6. For the caesium-enriched subdomains (Rigel and Vega), compositing was done in Leapfrog Edge™ using a 0.5 m interval. For residual length less than 0.25 m, it was redistributed equally within the domain.

Table 14.7 summarizes the statistics for the CV13 caesium enriched subdomains (Rigel and Vega) after compositing.

Table 14.5: Compositing Summary by Domain for CV5

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
Spod-Rich	Li_2O (%)	14,654	0.00	6.84	1.48	1.16
	Cs_2O (%)	14,438	0.00	6.02	0.09	0.17
	Ta_2O_5 (ppm)	14,654	0.31	18,742.75	162.41	241.92
	Ga (ppm)	14,551	3.40	184.98	66.79	20.28
Felds Pegmatite	Li_2O (%)	6,153	0.00	4.53	0.21	0.39
	Cs_2O (%)	6,088	0.00	3.05	0.06	0.08
	Ta_2O_5 (ppm)	6,153	0.31	10,856.02	159.96	267.64
	Ga (ppm)	6,138	1.99	162.74	50.29	16.30

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV5_110	Li_2O (%)	11	0.01	0.49	0.12	0.16
	Cs_2O (%)	11	0.01	0.03	0.02	0.01
	Ta_2O_5 (ppm)	11	41.37	228.35	106.57	74.41
	Ga (ppm)	11	48.68	68.65	55.88	5.46
CV5_120	Li_2O (%)	275	0.00	3.26	0.34	0.63
	Cs_2O (%)	275	0.00	2.63	0.09	0.30
	Ta_2O_5 (ppm)	275	0.31	2,028.25	137.52	183.22
	Ga (ppm)	275	1.00	110.00	52.49	16.10
CV5_130	Li_2O (%)	799	0.00	4.34	0.67	0.87
	Cs_2O (%)	799	0.00	1.72	0.06	0.11
	Ta_2O_5 (ppm)	799	0.31	2,878.13	129.24	221.82
	Ga (ppm)	799	9.35	198.66	55.89	16.18
CV5_140	Li_2O (%)	847	0.00	5.51	0.83	1.00
	Cs_2O (%)	847	0.00	2.52	0.07	0.15
	Ta_2O_5 (ppm)	847	0.31	837.86	120.31	96.76
	Ga (ppm)	802	8.00	118.23	57.18	17.01
CV5_150	Li_2O (%)	1,100	0.00	6.05	0.82	0.89
	Cs_2O (%)	1,100	0.00	4.86	0.10	0.27
	Ta_2O_5 (ppm)	1,100	0.31	1,848.12	138.14	125.15
	Ga (ppm)	1,062	7.00	136.96	57.14	16.69
CV5_160	Li_2O (%)	1,299	0.00	5.12	1.03	1.08
	Cs_2O (%)	1,299	0.00	6.00	0.09	0.23
	Ta_2O_5 (ppm)	1,299	0.31	2,172.77	161.37	168.29
	Ga (ppm)	1,294	0.50	196.00	58.50	20.11
CV5_170	Li_2O (%)	249	0.00	4.42	0.82	1.05
	Cs_2O (%)	249	0.00	0.71	0.06	0.08
	Ta_2O_5 (ppm)	249	0.31	1,652.13	174.70	152.16
	Ga (ppm)	245	2.00	118.28	58.94	17.02
CV5_180	Li_2O (%)	655	0.00	4.29	0.80	0.92
	Cs_2O (%)	655	0.00	2.42	0.09	0.20
	Ta_2O_5 (ppm)	655	0.31	2,195.44	152.18	175.97

Zone	Element	Count	Minimum	Maximum	Mean	Std Dev.
	Ga (ppm)	629	3.00	124.10	50.16	19.08
CV5_190	Li_2O (%)	17	0.00	1.06	0.20	0.33
	Cs_2O (%)	17	0.01	0.22	0.06	0.07
	Ta_2O_5 (ppm)	17	12.09	1,169.54	231.37	289.16
	Ga (ppm)	17	23.00	81.00	57.43	9.17

Table 14.6: Compositing Summary by Domain for CV13

Name	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV13_008	Li_2O (%)	10	0.00	0.08	0.02	0.02
	Cs_2O (%)		0.00	0.01	0.01	0.00
	Ta_2O_5 (ppm)		47.26	151.42	85.33	34.71
	Ga (ppm)		56.44	72.00	63.13	5.08
CV13_009	Li_2O (%)	2	0.01	0.02	0.01	0.01
	Cs_2O (%)		0.02	0.03	0.03	0.01
	Ta_2O_5 (ppm)		17.34	44.50	30.92	19.21
	Ga (ppm)		34.00	41.35	37.68	5.20
CV13_010	Li_2O (%)	5	0.01	0.03	0.02	0.01
	Cs_2O (%)		0.00	0.03	0.02	0.01
	Ta_2O_5 (ppm)		31.02	92.93	58.26	22.43
	Ga (ppm)		40.06	51.00	44.50	5.28
CV13_020	Li_2O (%)	24	0.01	0.24	0.07	0.06
	Cs_2O (%)		0.00	0.08	0.03	0.03
	Ta_2O_5 (ppm)		5.86	218.48	52.15	43.69
	Ga (ppm)		24.99	61.00	42.92	10.35
CV13_030	Li_2O (%)	14	0.01	0.10	0.06	0.03
	Cs_2O (%)		0.00	0.05	0.02	0.01
	Ta_2O_5 (ppm)		12.33	145.31	59.92	36.13
	Ga (ppm)		22.80	63.00	45.19	11.95
CV13_040	Li_2O (%)	23	0.01	0.17	0.07	0.05
	Cs_2O (%)		0.00	0.06	0.02	0.02
	Ta_2O_5 (ppm)		0.85	611.40	105.33	132.33

Name	Element	Count	Minimum	Maximum	Mean	Std Dev.
	Ga (ppm)		17.00	74.00	48.39	17.68
CV13_050	Li_2O (%)	17	0.00	0.75	0.14	0.24
	Cs_2O (%)		0.01	0.09	0.05	0.03
	Ta_2O_5 (ppm)		20.83	173.26	81.72	50.08
	Ga (ppm)		40.00	69.00	51.60	8.70
CV13_060	Li_2O (%)	17	0.01	0.18	0.05	0.05
	Cs_2O (%)		0.00	0.06	0.03	0.02
	Ta_2O_5 (ppm)		12.23	264.55	95.77	73.04
	Ga (ppm)		20.00	88.00	49.70	16.08
CV13_065	Li_2O (%)	9	0.00	0.18	0.08	0.07
	Cs_2O (%)		0.00	0.00	0.00	0.00
	Ta_2O_5 (ppm)		0.31	271.08	101.01	80.00
	Ga (ppm)	7	49.14	70.63	58.41	7.54
CV13_070	Li_2O (%)	34	0.00	0.10	0.02	0.02
	Cs_2O (%)		0.00	0.08	0.02	0.02
	Ta_2O_5 (ppm)		0.31	422.73	82.10	79.54
	Ga (ppm)	30	17.00	79.00	51.29	14.32
CV13_080	Li_2O (%)	110	0.00	2.13	0.19	0.34
	Cs_2O (%)		0.00	0.21	0.03	0.03
	Ta_2O_5 (ppm)		0.31	1,817.00	124.34	203.90
	Ga (ppm)	108	28.77	120.65	53.66	13.77
CV13_085	Li_2O (%)	31	0.02	2.31	0.22	0.46
	Cs_2O (%)		0.00	0.52	0.06	0.09
	Ta_2O_5 (ppm)		10.85	252.77	115.56	70.59
	Ga (ppm)		25.00	83.00	47.56	13.02
CV13_090	Li_2O (%)	398	0.00	4.05	0.46	0.76
	Cs_2O (%)		0.00	1.10	0.06	0.10
	Ta_2O_5 (ppm)		0.31	1,163.71	93.60	105.88
	Ga (ppm)		13.26	129.99	52.54	17.59
CV13_091	Li_2O (%)	48	0.00	1.72	0.33	0.49
	Cs_2O (%)		0.00	3.82	0.18	0.59

Name	Element	Count	Minimum	Maximum	Mean	Std Dev.
	Ta_2O_5 (ppm)		4.64	1,257.73	139.38	221.89
	Ga (ppm)		25.00	202.93	70.74	39.73
CV13_092	Li_2O (%)	13	0.03	0.16	0.08	0.05
	Cs_2O (%)		0.01	0.40	0.08	0.12
	Ta_2O_5 (ppm)		116.25	382.20	200.61	64.56
	Ga (ppm)		49.00	85.44	61.61	12.44
CV13_100	Li_2O (%)	1,367	0.00	6.82	0.92	0.97
	Cs_2O (%)	1,346	0.00	2.86	0.06	0.13
	Ta_2O_5 (ppm)	1,367	0.31	11,093.06	120.86	374.69
	Ga (ppm)	1,362	0.50	196.00	61.46	20.07
CV13_100C	Li_2O (%)	10	0.03	0.30	0.09	0.09
	Cs_2O (%)		0.01	0.11	0.06	0.04
	Ta_2O_5 (ppm)		20.76	133.73	72.20	45.98
	Ga (ppm)		39.00	69.78	53.40	11.57
CV13_101	Li_2O (%)	1,083	0.00	7.01	1.21	1.55
	Cs_2O (%)	967	0.00	5.30	0.14	0.30
	Ta_2O_5 (ppm)	1,083	0.31	8,431.70	187.37	364.19
	Ga (ppm)	1,083	0.50	250.00	65.98	31.55
CV13_103	Li_2O (%)	13	0.00	1.03	0.25	0.30
	Cs_2O (%)		0.00	0.07	0.03	0.02
	Ta_2O_5 (ppm)		76.85	278.93	156.18	66.38
	Ga (ppm)		39.08	72.00	51.03	9.10
CV13_110	Li_2O (%)	11	0.02	1.38	0.35	0.49
	Cs_2O (%)		0.00	1.15	0.21	0.34
	Ta_2O_5 (ppm)		2.13	210.67	95.47	78.14
	Ga (ppm)		13.50	78.00	53.74	18.74
CV13_120	Li_2O (%)	63	0.02	4.19	0.69	1.16
	Cs_2O (%)		0.00	6.20	0.34	0.90
	Ta_2O_5 (ppm)		6.59	691.14	171.72	118.67
	Ga (ppm)		15.00	118.41	60.81	21.22

Name	Element	Count	Minimum	Maximum	Mean	Std Dev.
CV13_130	Li_2O (%)	50	0.00	2.78	0.49	0.79
	Cs_2O (%)		0.00	6.42	0.56	1.25
	Ta_2O_5 (ppm)		0.31	709.46	109.22	150.84
	Ga (ppm)	43	16.00	100.91	58.28	20.24
CV13_140	Li_2O (%)	5	0.02	1.73	0.50	0.74
	Cs_2O (%)		0.02	0.16	0.07	0.06
	Ta_2O_5 (ppm)		61.91	201.83	144.34	61.86
	Ga (ppm)		50.00	70.00	57.04	8.71

Table 14.7: Cs_2O (%) Compositing Summary in Cs-Enriched Zones CV13 (composites of 0.5 m)

Domain	Field	Count	Min.	Max.	Mean	Std Dev.
Rigel: CV13_Cs_05P_100	Cs_2O (%)	40	0.67	26.61	8.95	8.44
Vega: CV13_Cs_05p_101		152	0.01	25.81	2.73	3.29
Vega: CV13_Cs_05p_101a		81	0.03	13.04	3.12	3.02

14.1.5.3 Grade Capping

Composited assay data for each domain were examined individually to assess the amount of metal that is biased from high-grade assays. A combination of geostatistical methods, probability plots and cumulative frequency plots was used to assist in the determination of whether grade capping was required on each element in each domain.

The QP elected to apply a variable top cut by element by domain group. Table 14.8 and Table 14.9 summarize the results of the capping for CV5 and CV13, respectively. Figure 14.6 is an example to show the capping justification for CV5, and Figure 14.7 is an example to show the capping justification for CV13. Capping was done on composites.

The same geostatistical methods were applied to the caesium-enriched subdomains (Rigel and Vega), and no capping was required for Cs_2O. Table 14.9 summarizes the results of the capping for the refined caesium model at CV13.

Table 14.8: Grade Capping Summary by Domain for CV5

Domain	Field	Sample	Uncut	COV	Uncut	Max.	Min.	Capping	Number	%	Metal	Cut		
		Count	Mean		Median			Value	Capped	Capped	Loss (%)	Mean	COV	Median
Spod-Rich	Li_2O (%)	14,654	1.48	0.78	1.24	6.84	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Cs_2O (%)	14,654	0.09	1.92	0.06	6.02	0	3.5	10	0.07	0.75	0.09	1.74	0.06
	Ta_2O_5 (ppm)	14,654	162.71	1.49	115.17	18,742.75	0.31	3,000	7	0.05	0.89	161.26	1.12	115.17
	Ga (ppm)	14,654	66.32	0.32	64.48	184.98	0.01	150	7	0.05	0.01	66.31	0.32	64.48
Feldspar-Rich	Li_2O (%)	6,153	0.21	1.9	0.08	4.53	0	3.5	10	0.16	0.3	0.21	1.87	0.08
	Cs_2O (ppm)	6,153	0.06	1.32	0.05	3.05	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Ta_2O_5 (ppm)	6,153	160.43	1.68	99.36	10,856.02	0.31	1,500	29	0.47	3.68	154.45	1.22	99.36
	Ga (ppm)	6,153	50.19	0.33	47.71	162.74	0.01	150	2	0.03	0.01	50.18	0.33	47.71
Veins (110 to 190)	Li_2O (%)	5,252	0.81	1.19	0.41	6.05	0	5	10	0.19	0.05	0.81	1.19	0.41
	Cs_2O (ppm)	5,252	0.08	2.6	0.05	6	0	3.5	3	0.06	0.98	0.08	2.4	0.05
	Ta_2O_5 (ppm)	5,252	143.2	1.12	107.6	2,878.13	0.31	1,200	19	0.36	1.78	140.83	0.97	107.6
	Ga (ppm)	5,252	54.96	0.36	55.06	198.66	0.01	150	2	0.04	0.03	54.94	0.36	55.06

Table 14.9: Grade Capping Summary by Domain for CV13

Domain	Field	Sample	Uncut	COV	Uncut	Max.	Min.	Capping	Number	%	Metal	Cut		
		Count	Mean		Median			Value	Capped	Capped	Loss (%)	Mean	COV	Median
CV13_100	Li_2O (%)	1,367	0.91	1.06	0.62	6.82	0.00	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Cs_2O (%)	1,346	0.06	0.00	0.04	2.86	0.00	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Ta_2O_5 (ppm)	1,367	120.86	3.10	65.68	11,093.06	0.31	3,000.00	3	0.22	5.55	114.72	2.03	65.68
	Ga (ppm)	1,367	61.18	0.33	58.95	196.00	0.01	150.00	3	0.22	0.06	61.14	0.33	58.95
CV13_101	Li_2O (%)	1,083	1.21	1.28	0.36	7.01	0.00	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Cs_2O (%)	967	0.14	0.00	0.08	5.30	0.00	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Ta_2O_5 (ppm)	1,083	187.50	1.93	108.82	8,431.70	0.31	3,000.00	2	0.18	3.38	181.23	1.43	108.82
	Ga (ppm)	1,083	65.98	0.48	61.46	250.00	0.50	150.00	14	1.29	0.36	65.74	0.47	61.46
Other Domains Combined	Li_2O (%)	907	0.34	1.96	0.06	4.19	0.00	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Cs_2O (%)	907	0.11	4.03	0.03	6.42	0.00	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	Ta_2O_5 (ppm)	907	107.07	1.20	74.09	1,817.00	0.31	1,200.00	2	0.22	0.71	106.32	1.13	74.09
	Ga (ppm)	872	54.01	0.35	52.00	202.93	13.26	150.00	3	0.34	0.25	53.89	0.34	52.00

Table 14.10: Grade Capping Summary by Refined Domain for Cs_2O at CV13

Domain	Field	Sample Count	Uncut Mean	COV	Uncut Median	Max.	Min.	Capping Value	Number Capped	% Capped	Metal Loss (%)	Cut Mean	Cut COV	Cut Median
CV13_Cs2O_100	Cs_2O (%)	40	8.95	0.94	4.87	26.61	0.67	n/a	n/a	n/a	n/a	n/a	n/a	n/a
CV13_Cs2O_101		152	2.73	1.21	1.70	25.81	0.01	n/a	n/a	n/a	n/a	n/a	n/a	n/a
CV13_Cs2O_101a		81	3.12	0.97	2.14	13.04	0.03	n/a	n/a	n/a	n/a	n/a	n/a	n/a

Figure 14.6: CV5 Capping Justification on Li_2O for Spodumene-Rich Domain (Zone 100)

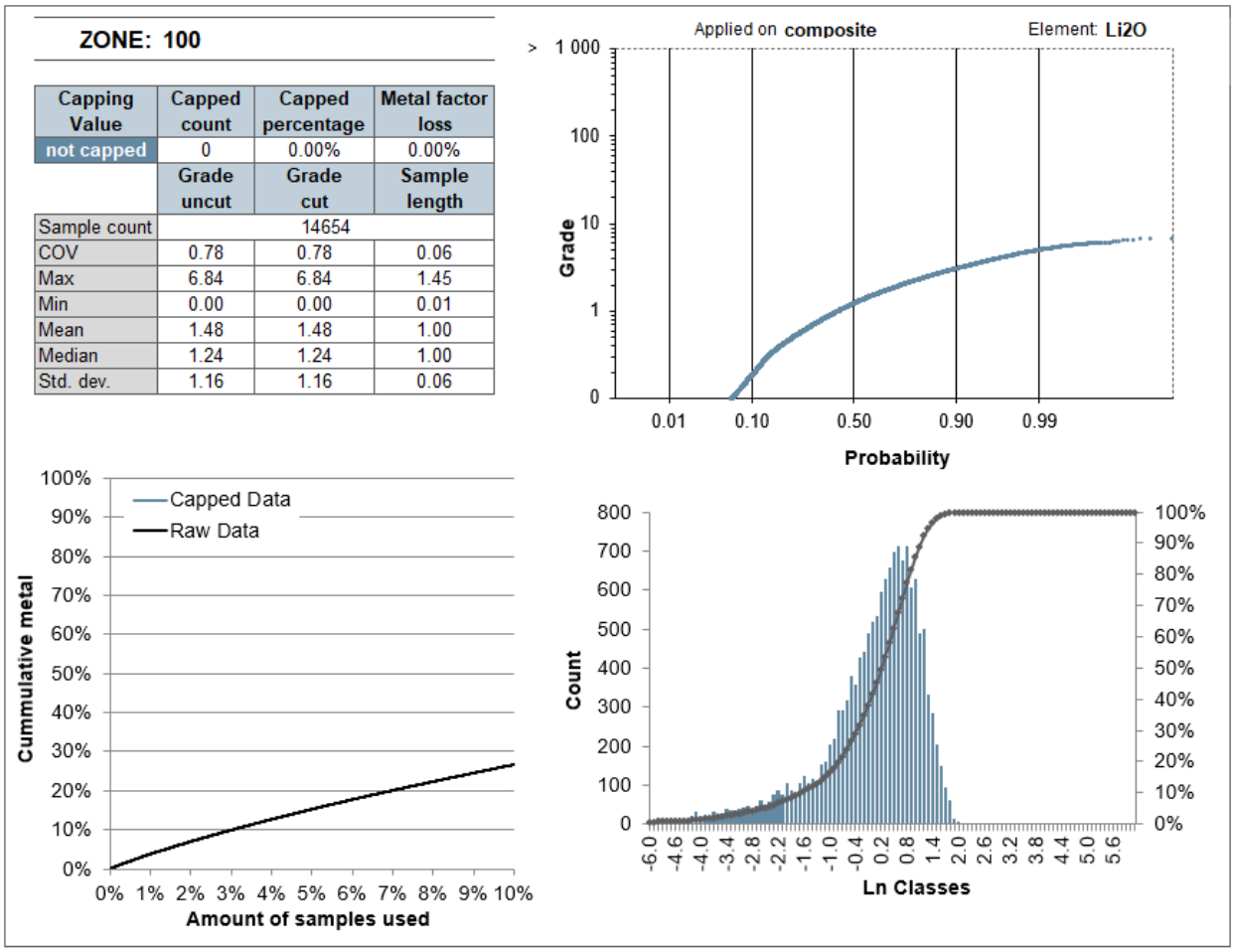
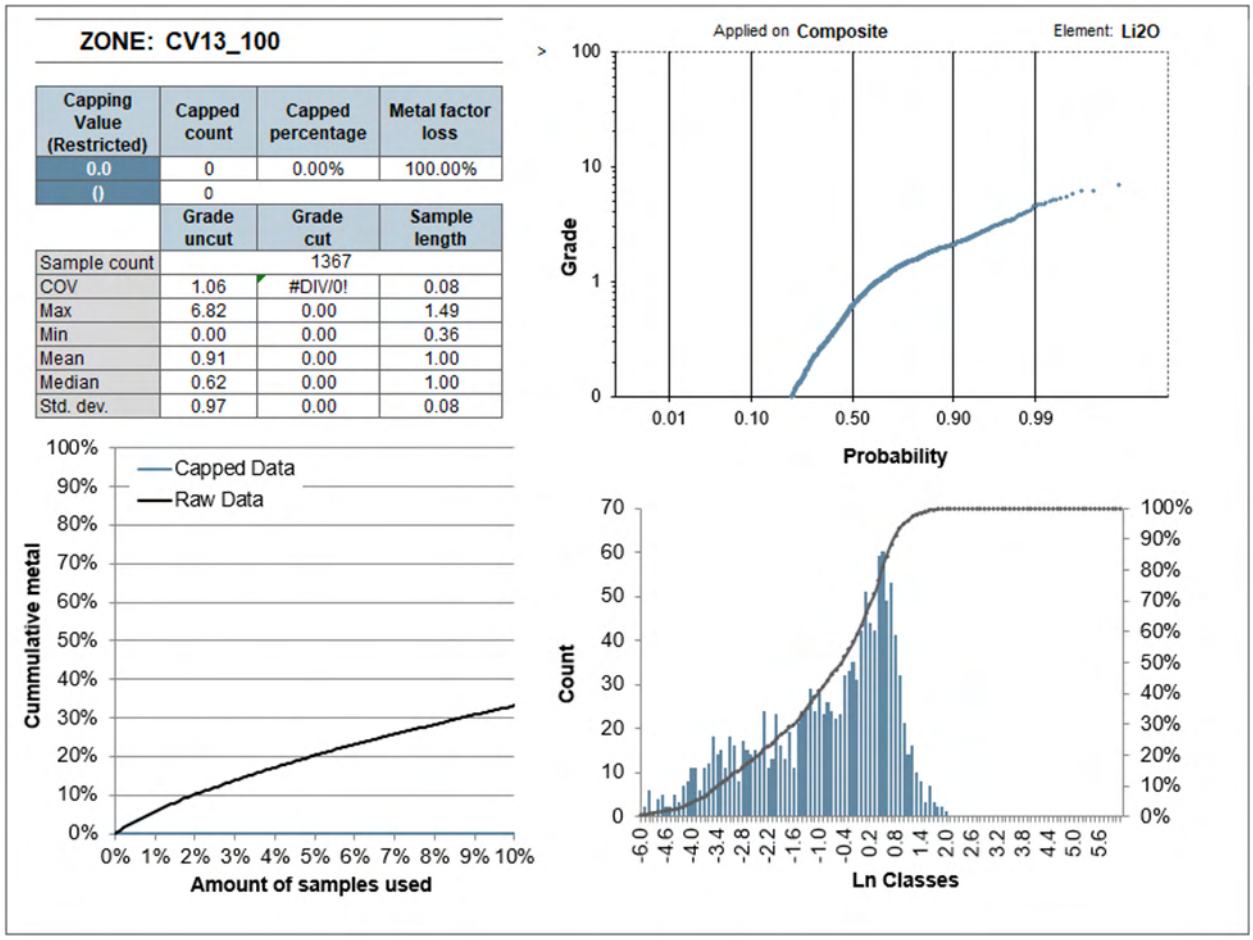


Figure 14.7: CV13 Capping Justification on Li_2O for CV13_100 Domain

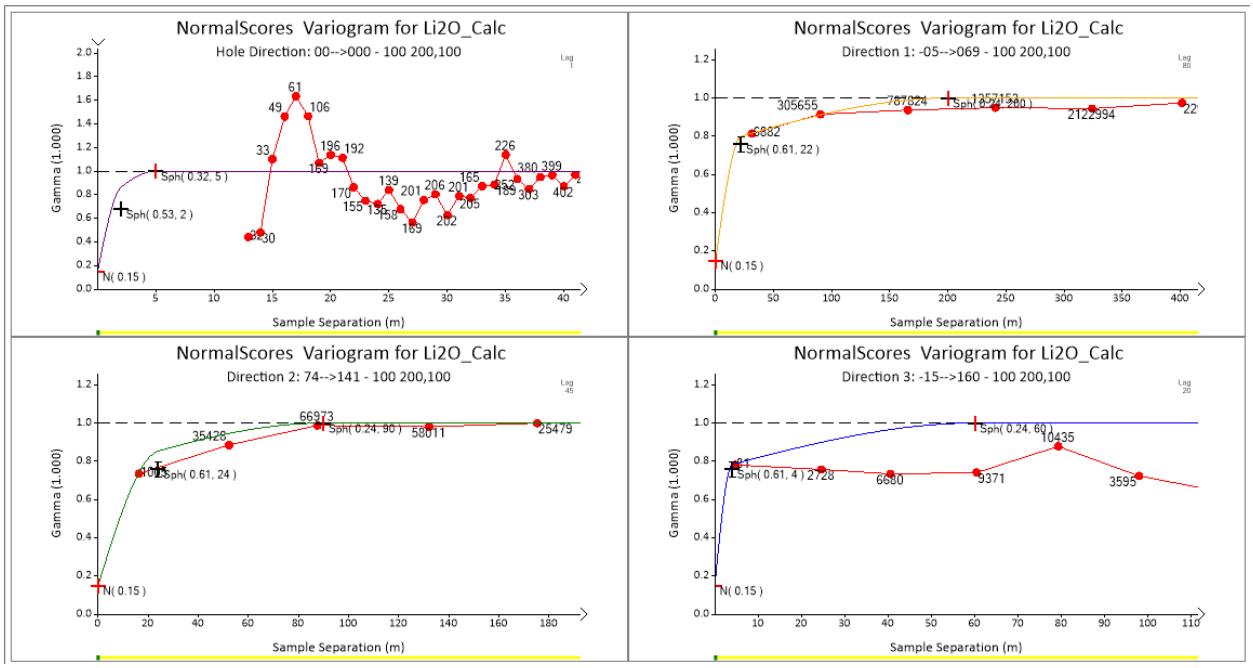


14.1.5.4 Spatial Analysis

CV5

Variography was done both in Leapfrog Edge™ and Supervisor. For Li_2O, Cs_2O, Ta_2O_5 and Ga, a well-structured variogram model (example Figure 14.8) were obtained for the CV5 principal pegmatite and CV5_160. Estimation with ordinary kriging (OK) was done for the CV5 principal pegmatite and CV5_160 domains in Leapfrog Edge™.

Figure 14.8: Variogram Model for CV5 Zone 100 (Li_2O)



Other domains did not yield well-structured variograms. Therefore, the other elements were estimated using inverse distance squared (ID^2) for the remaining domains (110, 120, 130, 140, 150, 170, 180, 190), also using Leapfrog Edge™.

The QP is of the opinion that additional drilling and samples are required before kriging would be an effective estimation method for the other domains.

It was determined that the spodumene-rich variogram of the CV5 principal pegmatite could be used for the feldspar-rich domains. Table 14.11 summarizes the results for the variogram used for the CV5 principal pegmatite and CV5_160 domains.

Table 14.11: Variogram Summary for CV5

Element	Rock Code	Nugget	First Structure				Second Structure				Leapfrog Orientation		
			Sill	Range X (m)	Range Y (m)	Range Z (m)	Sill	Range X (m)	Range Y (m)	Range Z (m)	Dip	Dip Az.	Pitch
Li_2O	100-200	0.161	0.634	22	24	4	0.205	200	90	60	75	340	175
	160	0.172	0.492	60	43	26	0.336	215	110	45	75	340	175
Cs_2O	100-200	0.334	0.564	33	14	16	0.102	190	100	70	75	340	175
	160	0.353	0.547	37	40	14	0.1	220	100	45	75	340	175
Ta_2O_5	100-200	0.334	0.569	41	6	6	0.097	230	70	45	75	340	175
	160	0.216	0.563	85	31	23	0.221	190	100	45	75	340	175
Ga	100-200	0.153	0.601	20	11	5	0.246	100	90	35	70	340	175
	160	0.205	0.411	50	42	6	0.384	200	80	35	70	340	175

CV13

Geostatistical analysis did not yield a well-structured variogram. On CV13, Li_2O, Cs_2O, Ta_2O_5 and Ga were estimated using ID^2 in Leapfrog Edge™.

14.1.5.5 Resource Block Model

For CV5 and CV13, the block model was created in Leapfrog Edge™ for each mineral domain. The block model is rotated around the Z axis (Leapfrog Azimuth 340°), and interpolation was done on the parent cell.

A block size of 10 m x 5 m x 5 m was selected in order to accommodate a large-scale open-pit mining potential. Table 14.12 and Table 14.13 summarize details of the parent block model for CV5 and CV13, respectively.

Table 14.12: CV5 Parent Block Model Summary

Parameters	Data
Base Point X	568,190
Base Point Y	5,928,600
Base Point Z	440
Boundary Size X	5,760
Boundary Size Y	2,180
Boundary Size Z	705
Block Size (m)	10 x 5 x 5
Rotation (Z)	340°
Sub-block Count	4 x 4 x 4
Size in Blocks	576 x 436 x 141
Total No. Blocks	35,410,176

Table 14.13: CV13 Parent Block Model Summary

Parameters	Data
Base Point X	563,900
Base Point Y	5,926,800
Base Point Z	550
Boundary Size X	2,800
Boundary Size Y	1,800
Boundary Size Z	700
Block Size (m)	10 x 5 x 5
Rotation (Z)	340°
Sub-block Count	4 x 4 x 4
Size in Blocks	280 x 360 x 140
Total No. Blocks	14,112,000

14.1.5.6 Estimate Parameters for CV5

The CV5 principal pegmatite and CV5_160 estimated Li_2O, Cs_2O, Ta_2O_5, and Ga, using OK. The remaining eight pegmatite dykes' domains were estimated using ID^2. Table 14.14 shows the search ellipse parameters by domain.

Three (3) oriented search ellipsoids were used to select data and interpolate grades in successively less restrictive passes. The ellipse sizes and anisotropies were based on the variography, drill hole spacing, and pegmatite geometry. Estimations were completed using a multi-pass ellipse with a minimum / maximum composite required and maximum composites per drill hole. Table 14.15 shows the estimation criteria applied for each element.

Variable search ellipse orientations (dynamic anisotropy) were used to interpolate seven (7) of the parallel dykes. Spatial anisotropy of the dykes is respected during estimation using the Leapfrog Edge™ Variable Orientation tool. The search ellipse follows the trend of the central reference plane of each dyke.

Table 14.14: CV5 Search Ellipse Summary

Domain	Element	Ellipsoid Direction			Ellipsoid Ranges			Ellipsoid Ranges			Ellipsoid Ranges		
					1st Pass			2nd Pass			3rd Pass		
		Dip	Dip Azi.	Pitch	Max.	Int.	Min.	Max.	Int.	Min.	Max.	Int.	Min.
100 200	Li_2O	75	340	175	100	45	30	200	90	60	300	135	90
	Cs_2O	75	340	175	100	45	30	200	90	60	300	135	90
	Ta_2O_5	75	340	175	115	35	22.5	230	70	45	402.5	122.5	79
	Ga	75	340	175	50	45	17.5	100	90	35	200	180	70
110 120 130 140	Li_2O	Var.	Var.	Var.	107.5	55	22.5	215	110	45	322.5	165	67.5
	Cs_2O	Var.	Var.	Var.	107.5	55	22.5	215	110	45	322.5	165	67.5
	Ta_2O_5	Var.	Var.	Var.	95	50	22.5	190	100	45	285	150	67.5
	Ga	Var.	Var.	Var.	100	40	17.5	200	80	35	300	120	52.5
150	Li_2O	75	338	175	107.5	55	22.5	215	110	45	322.5	165	67.5
	Cs_2O	Var.	Var.	Var.	107.5	55	22.5	215	110	45	322.5	165	67.5
	Ta_2O_5	75	338	175	95	50	22.5	190	110	45	285	150	67.5
	Ga	75	338	175	100	40	17.5	200	80	35	300	120	52.5
160 170 180 190	Li_2O	Var.	Var.	Var.	107.5	55	22.5	215	110	45	322.5	165	67.5
	Cs_2O	Var.	Var.	Var.	107.5	55	22.5	215	110	45	322.5	165	67.5
	Ta_2O_5	Var.	Var.	Var.	95	50	22.5	190	110	45	285	150	67.5
	Ga	Var.	Var.	Var.	100	40	17.5	200	80	35	300	120	52.5

Table 14.15: CV5 Estimation Criteria Summary

Domain	Pass	Min. Number of Composites	Max. Number of Composites	Max. Number of Composites per DDH
All CV5 Domains	1	5	15	4
	2	5	15	4
	3	3	15	-

14.1.5.7 Estimate Parameters for CV13

All domains in CV13 for Li_2O, Cs_2O, Ta_2O_5, and Ga were estimated using ID^2. Table 14.16 shows the search ellipse parameters by domains, and Table 14.17 presents the parameters for the Cs2O-enriched domains. The estimation methodology used for CV13 was the same as that used for CV5. Table 14.18 shows the estimation criteria.

Table 14.16: CV13 Search Ellipse Summary

Domain	Element	Ellipsoid Direction			Ellipsoid Ranges			Ellipsoid Ranges			Ellipsoid Ranges		
					1st Pass			2nd Pass			3rd Pass		
		Dip	Dip Azi.	Pitch	Max.	Int.	Min.	Max.	Int.	Min.	Max.	Int.	Min.
CV13_008 CV13_009 CV13_010 CV13_020 CV13_030 CV13_040 CV13_050 CV13_060 CV13_065 CV13_070 CV13_080 CV13_085 CV13_090 CV13_140	Li_2O	var.	var.	var.	60	35	10	120	70	20	240	140	40
	Cs_2O	var.	var.	var.	60	35	10	120	70	20	240	140	40
	Ta_2O_5	var.	var.	var.	50	60	10	100	120	20	200	240	40
	Ga	var.	var.	var.	60	35	10	120	70	20	240	140	40
CV13_091 CV13_092 CV13_100 CV13_110 CV13_130 CV13_100C	Li_2O	var.	var.	var.	80	45	10	160	90	20	320	180	40
	Cs_2O	var.	var.	var.	80	45	10	160	90	20	320	180	40
	Ta_2O_5	var.	var.	var.	55	35	10	110	70	20	220	140	40
	Ga	var.	var.	var.	70	35	10	140	70	20	280	140	40
CV13_101 CV13_103 CV13_120	Li_2O	var.	var.	var.	60	50	20	120	100	40	240	200	80
	Cs_2O	var.	var.	var.	60	50	20	120	100	40	240	200	80
	Ta_2O_5	var.	var.	var.	35	30	20	70	60	40	140	120	80
	Ga	var.	var.	var.	60	35	20	120	70	40	240	140	80

Table 14.17: CV13 Estimation Criteria Summary

Domain	Element	Ellipsoid Direction			Ellipsoid Ranges			Ellipsoid Ranges			Ellipsoid Ranges		
					1st Pass			2nd Pass			3rd Pass		
		Dip	Dip Azi.	Pitch	Max.	Int.	Min.	Max.	Int.	Min.	Max.	Int.	Min.
CV13_Cs2O_100	Cs2O	var.	var.	var.	80	45	10	160	90	20	320	180	40
CV13_Cs2O_101	Cs2O	var.	var.	var.	60	50	20	120	100	40	240	200	80
CV13_Cs2O_101a	Cs2O	var.	var.	var.	60	50	20	120	100	40	240	200	80

Table 14.18: CV13 Estimation Criteria Summary

Domain	Pass	Min. Number of Composites	Max. Number of Composites	Max Number of Composites per DDH
All Domains	1	3	8	2
	2	3	8	2
	3	2	8	-

14.1.6 Resource Classification

The Shaakichiuwaanaan resource classification has been completed in accordance with the NI 43-101, and CIM Definition Standards for Mineral Resources and Reserves reporting guidelines. All reported Mineral Resources have been constrained by conceptual open pit and underground mineable shapes to demonstrate reasonable prospects for eventual economic extraction (RPEEE). As the Company is dual-listed on the ASX in Australia, the QP also considered the definitions of JORC 2012. The Company has reported the Mineral Resource Estimate (effective as of June 20, 2025) to ASX in accordance with the JORC Code 2012 (see news release dated July 20, 2025).

Other factors considered for the classification are:

- The QP's experience with LCT pegmatites.
- Spatial continuity based on assays within the drill holes.
- Understanding of the geology of the deposit.
- Drill hole and channel spacing, and the estimation runs required to estimate the grades in a block.

Blocks in the model were initially classified as Indicated when:

- They demonstrated geological continuity and a minimum thickness of 2 m for the pegmatite and 0.5 m for the caesium-enriched zones (Rigel and Vega).
- The drill spacing was 70 m or less, and when they met the minimum parameters of the estimation criteria.
- Blocks were estimated with at least two (2) drill holes (pass 1 or pass 2).
- There was grade continuity at the reported cut-off grade (COG).

Blocks in the model were initially classified as Inferred when:

- The drill spacing was between 70 m and 140 m, and when they met the minimum parameters of the estimation criteria.
- Geological continuity and a minimum thickness of 2 m, or 0.5 m for the caesium-enriched subdomains (Rigel and Vega), were also mandatory.
- There was grade continuity at the reported COG.

There are no measured classified blocks. Pegmatite dykes or extensions with a lower level of information / confidence were also not classified.

Classification shapes are created around contiguous blocks at the stated criteria, with consideration for the selected mining method. The Mineral Resource Estimate appropriately reflects the view of the QP.

No environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues are known to the author that may affect the estimate of Mineral Resources. Mineral Reserves can only be estimated on the basis of an economic evaluation that is used in a preliminary feasibility study or a feasibility study of a mineral project; thus, no reserves have been estimated. According to NI 43-101, Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.

14.1.7 Mineral Resource Tabulation

The resource reported is effective as of June 20, 2025, and has been tabulated in terms of pit and underground mining shapes. Both underground and open-pit conceptual mining shapes were applied as constraints to demonstrate RPEEE. Cut-off grades for open-pit constrained resources are 0.40% Li_2O for both CV5 and CV13, and for underground constrained resources, cut-off grades are 0.60% Li_2O for CV5 and 0.70% Li_2O for CV13. Mineral Resources are Inclusive of Mineral Reserves.

Table 14.19: Shaakichiuwaanaan Mineral Resource Estimate

Pegmatite	Classification	Mass	Li_2O	Cs_2O	Ta_2O_5	Ga	Contained LCE (Mt)
		t	%	%	ppm	ppm	
CV5 & CV13	Indicated	107,991,000	1.40	0.11	166	66	3.75
	Inferred	33,380,000	1.33	0.21	155	65	1.09

14.1.7.1 CV5 – MRE Details

The Mineral Resource constrained within the open pit and underground mining shapes meets the definition of “Reasonable Prospect of Eventual Economic Extraction”, even though a portion of the open pit is under a lake. The QP took the following factors into account when considering the RPEEE:

- The depth of water where a coffer dam would be required is less than 20 m.
- There is no commercial fishery on the lake.
- There are no houses, cottages, or lodges on the lake.

The detailed CV5 MRE is presented in Table 14.20.

Table 14.20: Detailed CV5 Mineral Resource Estimate

Conceptual Mining Constraint (COG)	Classification	Mass	Li_2O	Cs_2O	Ta_2O_5	Ga	Contained LCE (Mt)
		t	%	%	ppm	ppm	
Open Pit (0.4% Li_2O)	Indicated	97,757,000	1.39	0.09	163	66	3.35
Underground (0.6% Li_2O)		4,071,000	1.08	0.06	186	66	0.11
Total		**101,828,000**	**1.38**	**0.09**	**164**	**66**	**3.46**
Open Pit (0.4% Li_2O)	Inferred	5,745,000	1.16	0.09	163	61	0.17
Underground (0.6% Li_2O)		8,153,000	1.24	0.07	136	60	0.25
Total		**13,898,000**	**1.21**	**0.08**	**147**	**60**	**0.41**

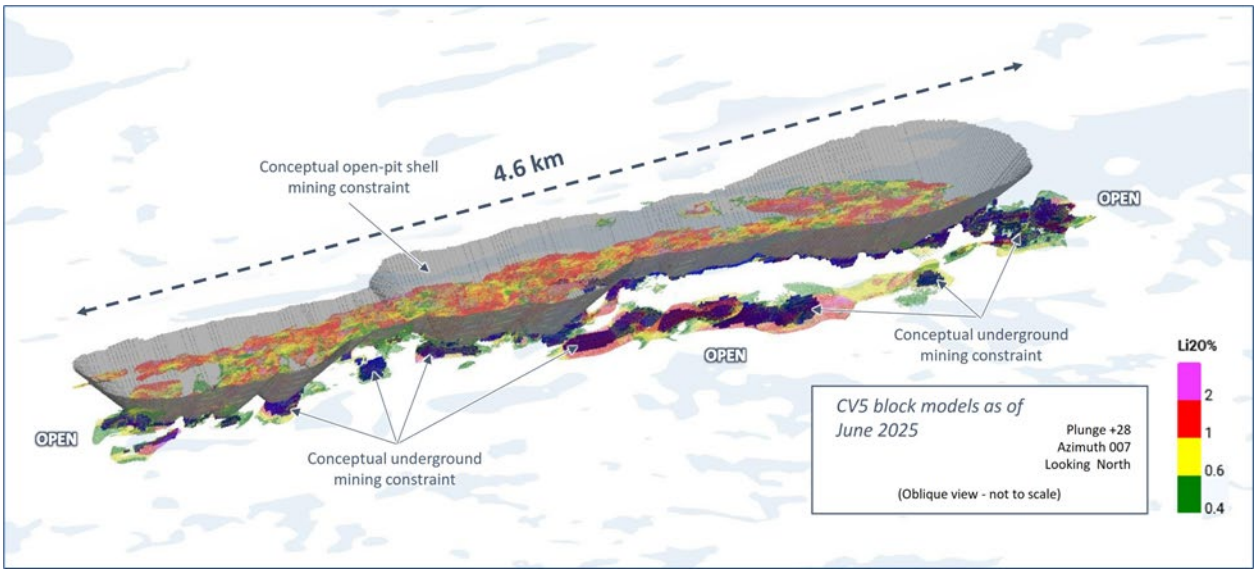
Table 14.21 summarizes the parameters used to develop the constraints and cut-off grades (UG and OP) for a reasonable prospect of economic extraction on CV5. The constraint parameters are provided primarily through benchmarking of similar projects and, therefore, are largely conceptual in nature and may change as development of the CV5 Pegmatite is studied. Lithium is the only payable metal utilized in determining the RPEEE parameters. All other elements reported (Cs_2O, Ta_2O_5 and Ga) are considered to have reasonable prospects for eventual economic extraction when mined and processed concurrently with the lithium resource.

Table 14.21: CV5 Parameters for Reasonable Prospects of Economic Extraction – OP & UG

Parameters	Unit	Open Pit	Underground
Mining Cost	$/t	5.47	68.66
Mining Cost OVB	$/t	4.00	
Processing Cost	$/t milled	14.91	
Tailing Management Cost	$/t milled	3.45	
G&A Cost	$/t milled	18.88	
Transport Cost	$/t conc.	226.74	
Mill Recovery	%	75*(1-EXP(-1.995*(Li_2O feed Grade%)))	
Concentrate Grade	%	5.5	
Exchange Rate	CAD/USD	0.7	
Concentrate Price	USD/t	1,500	
Royalty	%	2	
Revenue Factor		1	
Production Rate	Mtpa	0.8	
Discount Rate	%	8	
Pit Slope	°	45 to 53	
Li_2O Cut-off Grade	%	0.4	0.6
Li_2O Conversion		Li x 2.153	
LCE (i.e., Li_2CO_3) Conversion		Li_2O x 2.473	

The block model for the CV5 MRE is shown in Figure 14.9.

Figure 14.9: Oblique View of the CV5 Spodumene Pegmatite Block Model with Respect to Applied OP and UG Conceptual Mining Constraint Shapes



**Note: Not to scale.*

14.1.7.2 CV13 – MRE Details

The Mineral Resource constrained within the open pit and underground mining shapes meets the definition of the RPEEE. The detailed CV13 MRE is presented in Table 14.22.

Table 14.22: Detailed CV13 Mineral Resource Estimate

Conceptual Mining Constraint (COG)	Classification	Mass	Li_2O	Cs_2O	Ta_2O_5	Ga	Contained
		t	%	%	ppm	ppm	LCE (Mt)
Open Pit (0.4% Li_2O)	Indicated	5,996,000	1.89	0.60	201	76	0.28
Underground (0.7% Li_2O)		167,000	0.85	0.06	132	60	0.004
Total		6,163,000	1.86	0.59	199	76	0.28
Open Pit (0.4% Li_2O)	Inferred	18,020,000	1.44	0.32	168	70	0.64
Underground (0.7% Li_2O)		1,462,000	1.05	0.08	75	55	0.04
Total		19,482,000	1.41	0.30	161	69	0.68

Mineral Resources for the Rigel and Vega zones are hosted within subdomains of the CV13 Pegmatite's open pit conceptual mining shape and, therefore, are inclusive within the Consolidated MRE for CV5 and CV13 pegmatites. The Rigel and Vega zones were constrained using a 0.50% Cs_2O grade within lithium blocks greater than 0.4% Li_2O and are based on mineral processing analogues and mineralogical analysis supporting pollucite as the predominant caesium-bearing mineral present. Table 14.23 presents the MRE within the caesium-enriched zones of Rigel and Vega. Some portions of blocks at Vega and Rigel that were below the 0.4% Li_2O COG but above 0.5% Cs_2O grade were captured in the resource wireframes for continuity and included in the Consolidated MRE and the MRE within the caesium-enriched zones.

Table 14.23: Mineral Resources at Rigel and Vega Caesium Zones within the CV13 Pegmatite

Caesium Zone	Classification	Tonnes	Li_2O	Cs_2O	Ta_2O_5	Contained Cs_2O
		(t)	(%)	(%)	(ppm)	(t)
Rigel	Indicated	163,000	1.78	10.25	646	16,708
	Inferred	–	–	–	–	–
Vega	Indicated	530,000	2.23	2.61	172	13,833
	Inferred	1,698,000	1.81	2.40	245	40,752
Rigel + Vega	Indicated	693,000	2.13	4.40	283	30,541
	Inferred	1,698,000	1.81	2.40	245	40,752

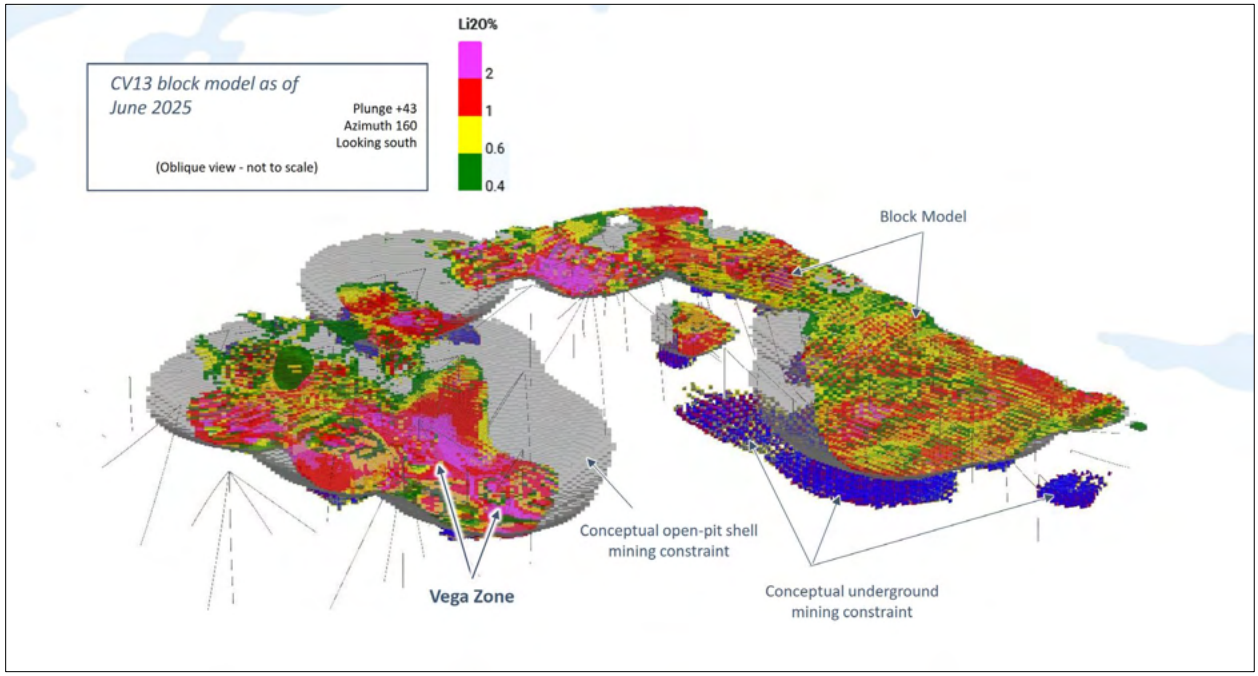
Table 14.24 summarizes the parameters used to develop the constraints and cut-off grades (UG and OP) for a reasonable prospect of economic extraction on CV13. The constraint parameters are provided primarily through benchmarking of similar projects and, therefore, are largely conceptual in nature and may change as development of the CV13 Pegmatite is studied. Lithium is the only payable metal utilized in determining the RPEEE parameters. However, other elements reported (Cs_2O, Ta_2O_5 and Ga) are considered to have reasonable prospects for eventual economic extraction when mined and processed concurrently with the lithium resource.

Table 14.24: Parameters for Reasonable Prospect of Economic Extraction – OP & UG

Parameters	Unit	Open Pit	Underground
Mining Cost	$/t	7.47	100
Mining Cost OVB	$/t	4.00	
Processing Cost	$/t milled	14.91	
Tailing Management Cost	$/t milled	3.45	
G&A Cost	$/t milled	18.88	
Transport Cost	$/t conc.	226.74	
Mill Recovery	%	75*(1-EXP(-1.995*(Li_2O feed Grade%)))	
Concentrate Grade	%	5.5	
Exchange Rate	CAD/USD	0.7	
Concentrate Price	USD/t	1,500	
Royalty	%	2	
Revenue Factor		1	
Production Rate	Mtpa	0.8	
Discount Rate	%	8	
Pit Slope	°	45	
Li_2O Cut-off Grade	%	0.4	0.7
Li_2O Conversion		Li x 2.153	
LCE (i.e., Li_2CO_3) Conversion		Li_2O x 2.473	

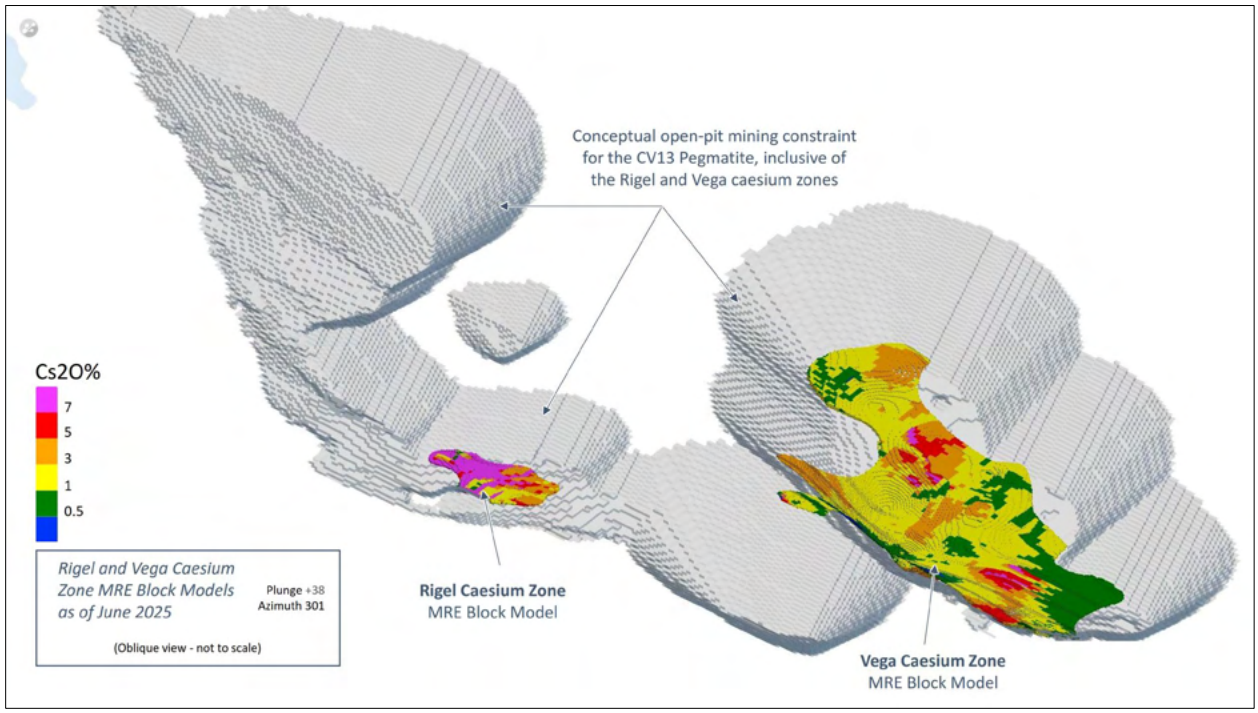
Open-pit and underground constrained MRE for the CV13 Pegmatite block model is shown in Figure 14.10.

Figure 14.10: CV13 Open Pit and Underground Constraints to MRE Block Model



*Note: Not to scale.

Figure 14.11: CV13 Open Pit Constraint to Rigel and Vega Caesium Zone MRE Block Model



Note: Not to scale.

14.1.8 Model Validation

CV5 and CV13 models were validated using the following three (3) methods:

- Visual comparison of colour-coded block model grades with composite grades on the section.
- Comparison of the global mean block grades for OK (when applicable), ID^2, nearest neighbour (NN), and composites.
- Swath plots.

14.1.8.1 Visual Validation

Visual comparisons of block model grades with composite grades for each zone show a reasonable correlation between values. Figure 14.12 and Figure 14.13 present results for Li_2O. Figure 14.14 and Figure 14.15 present results for Cs_2O. No significant discrepancies were apparent from the sections reviewed.

Figure 14.12: CV5 Comparison (composites vs block model Li_2O; eastern portion)

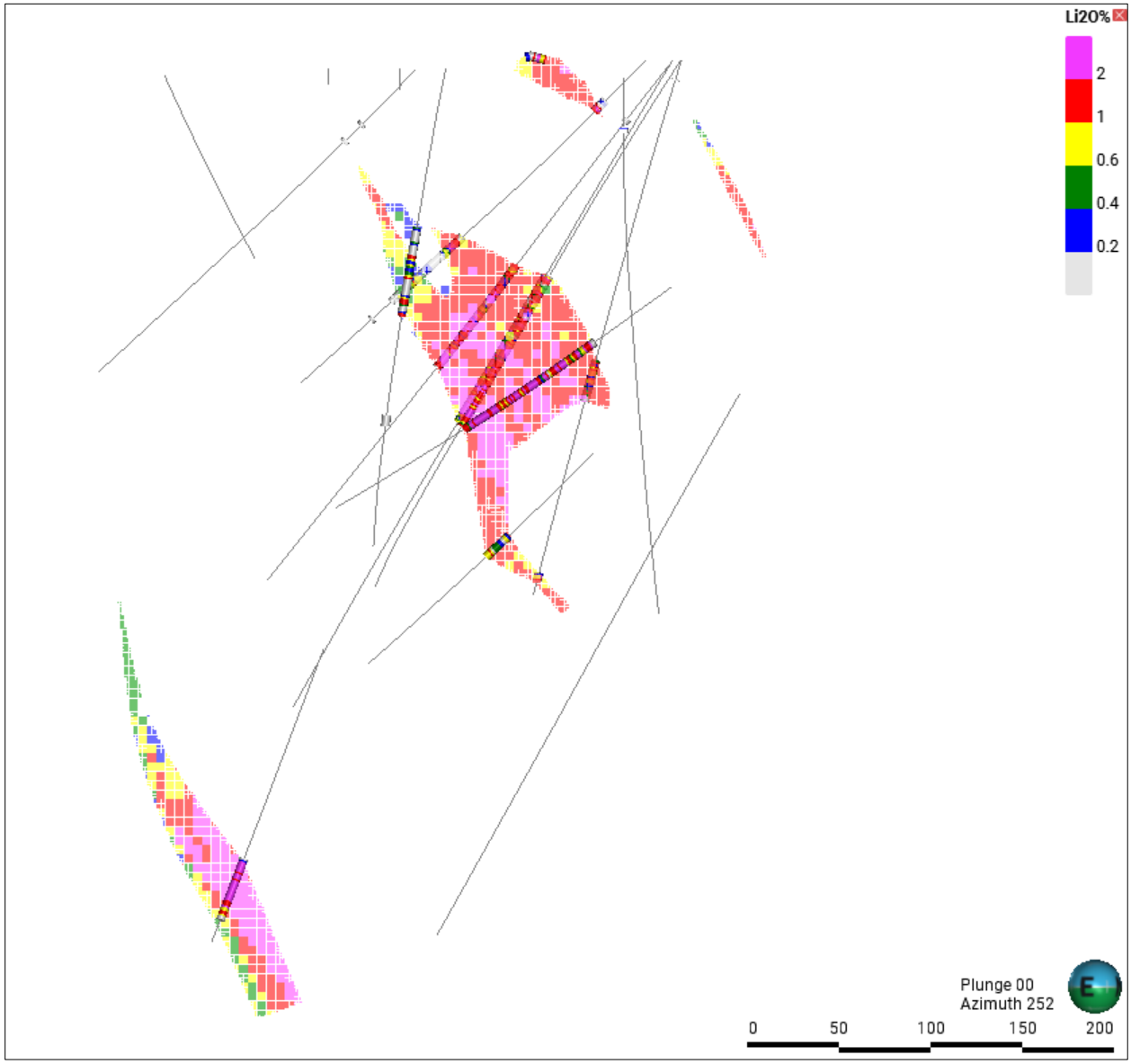


Figure 14.13: CV13 Comparison (composites vs block model Li_2O; Vega portion)

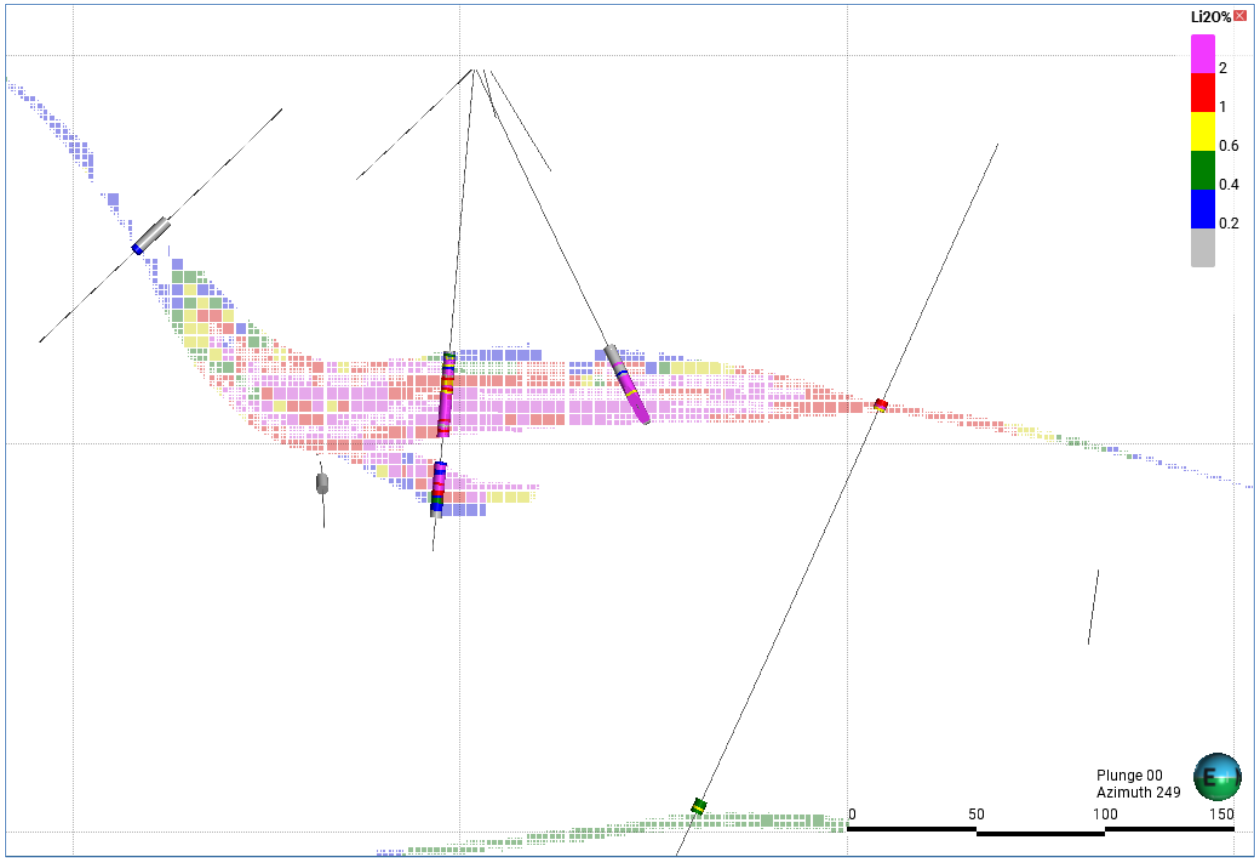


Figure 14.14: CV13 Comparison
(composites vs block model Cs_2O; CV13_101 with Vega Cs-enriched zones)

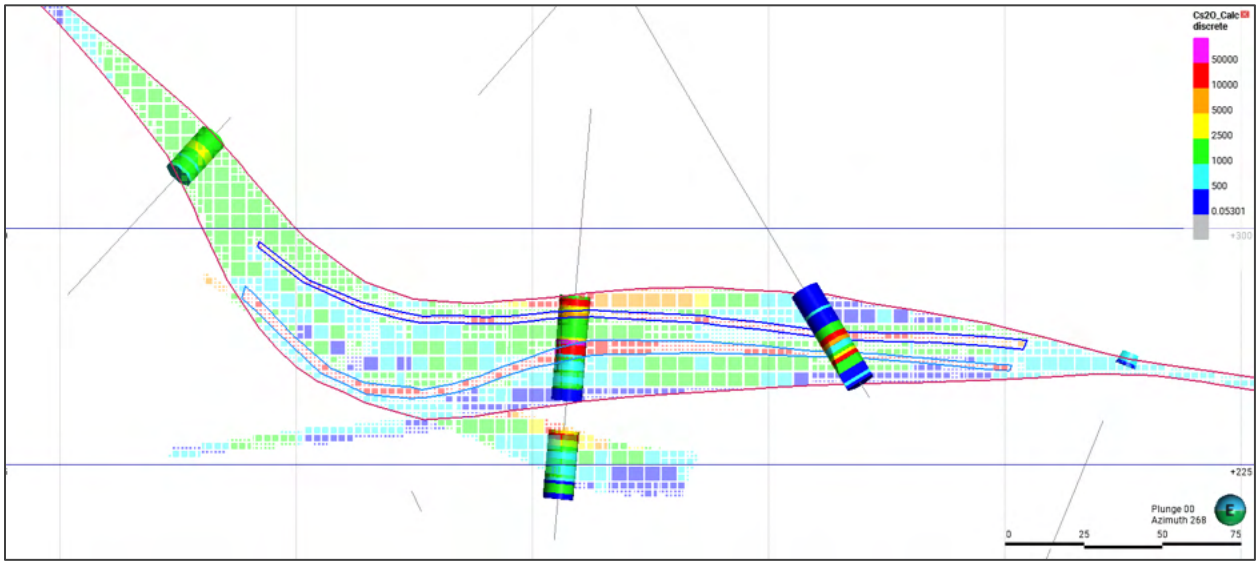
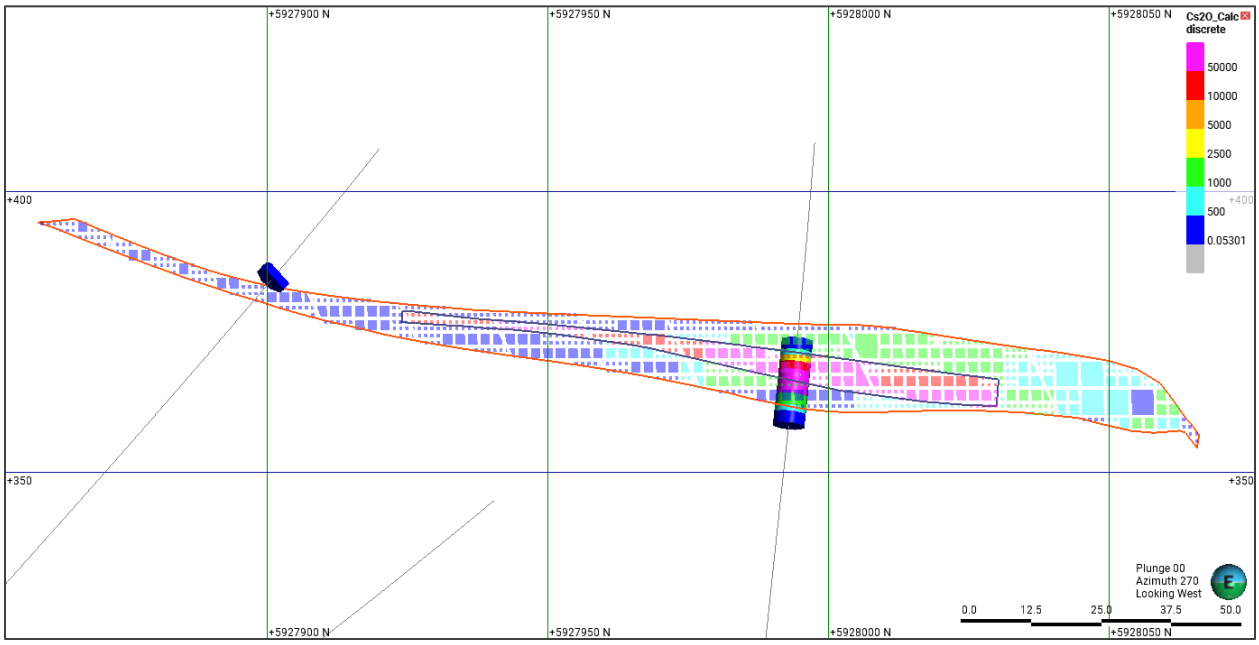


Figure 14.15: CV13 Comparison
(composites vs block model Cs_2O; CV13_100 with Rigel Cs-enriched zone)



14.1.8.2 Statistics Comparison

CV5 and CV13 block model statistics were compared between estimation methods, OK, NN, and ID^2 model values and the capped composite of the drill hole data. Table 14.25 shows this comparison of the estimates for the three (3) estimation method calculations for each zone 100 (CV5 and CV13). All three (3) estimation methods yield similar results for Li_2O, with the anticipated smoothing resulting from the estimation process. Comparisons were made using all blocks without a cut-off grade.

Table 14.25: Comparison of Estimation Method Statistics Between Composites, NN, ID^2 and OK on Li_2O (%) for CV5 and CV13

	CV5 (Zone 100)				CV13 (Zone 100)			
	Comp (Li_2O)	NN (Li_2O)	ID2 (Li_2O)	OK (Li_2O)	Comp (Li_2O)	NN (Li_2O)	ID2 (Li_2O)	OK (Li_2O)
Number	14,654	1,44,997	1,446,849	1,447,997	1,367	495,939	467,519	467,519
Mean	1.48	1.41	1.43	1.41	0.92	0.61	0.83	0.82
Median	1.24	1.15	1.30	1.31	0.63	0.25	0.70	0.71
CV	0.78	0.82	0.47	0.41	1.05	1.36	0.77	0.73

14.1.8.3 Swath Plots

Swath plots comparing estimation results with composites in three (3) directions (Easting, Northing and Elevation) were generated and reviewed for each domain. Figure 14.16 and Figure 14.17 are examples of a swath plot in the Easting and Elevation direction, respectively. There are good correlations of the results between the three (3) estimation methods with the expected smoothing of the kriging results.

Figure 14.16: Li_2O Swath Plot in X Direction (CV5 Zone 100)

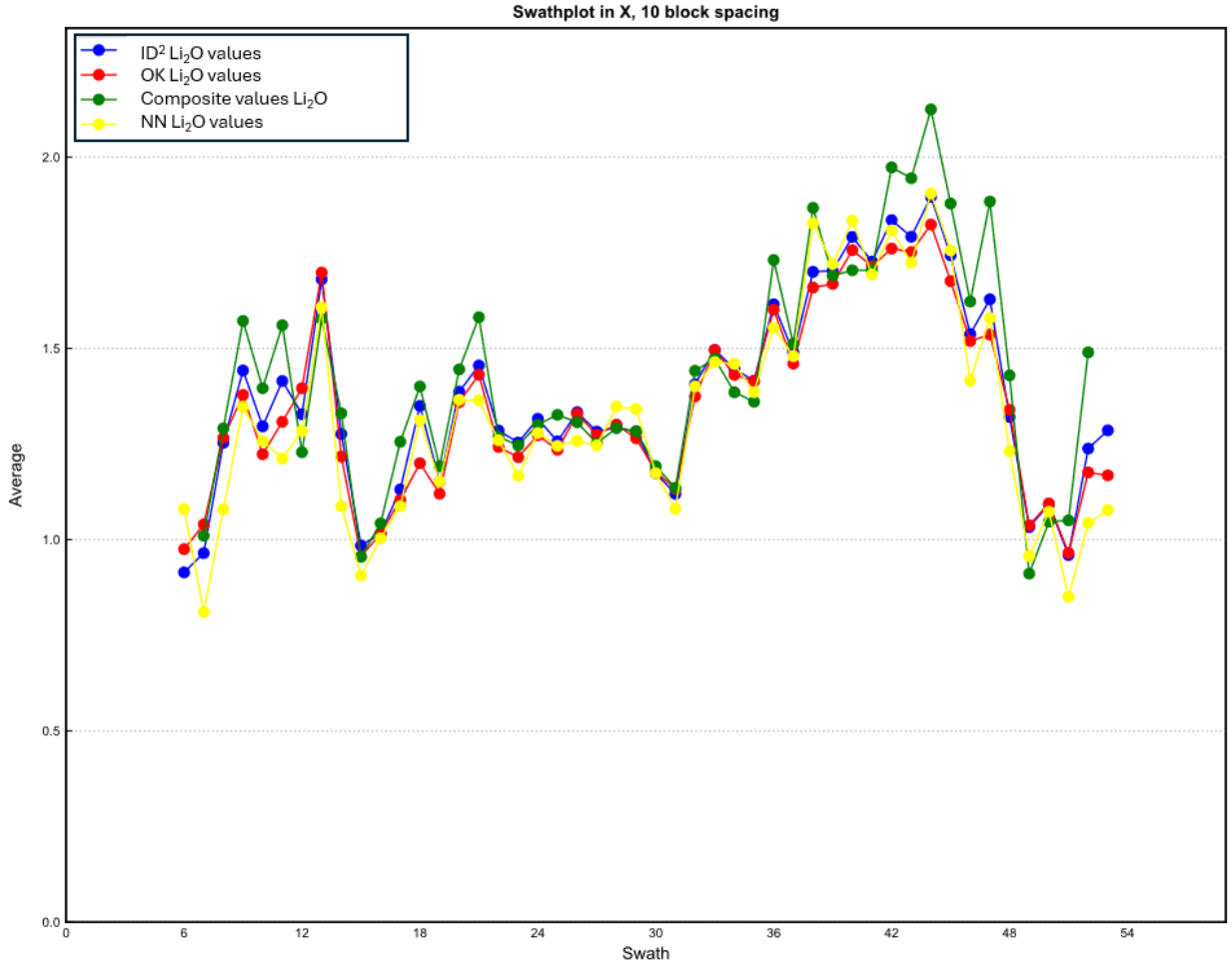


Figure 14.17: Li_2O Swath Plot in Elevation (Z) Direction (CV5 Zone 100)

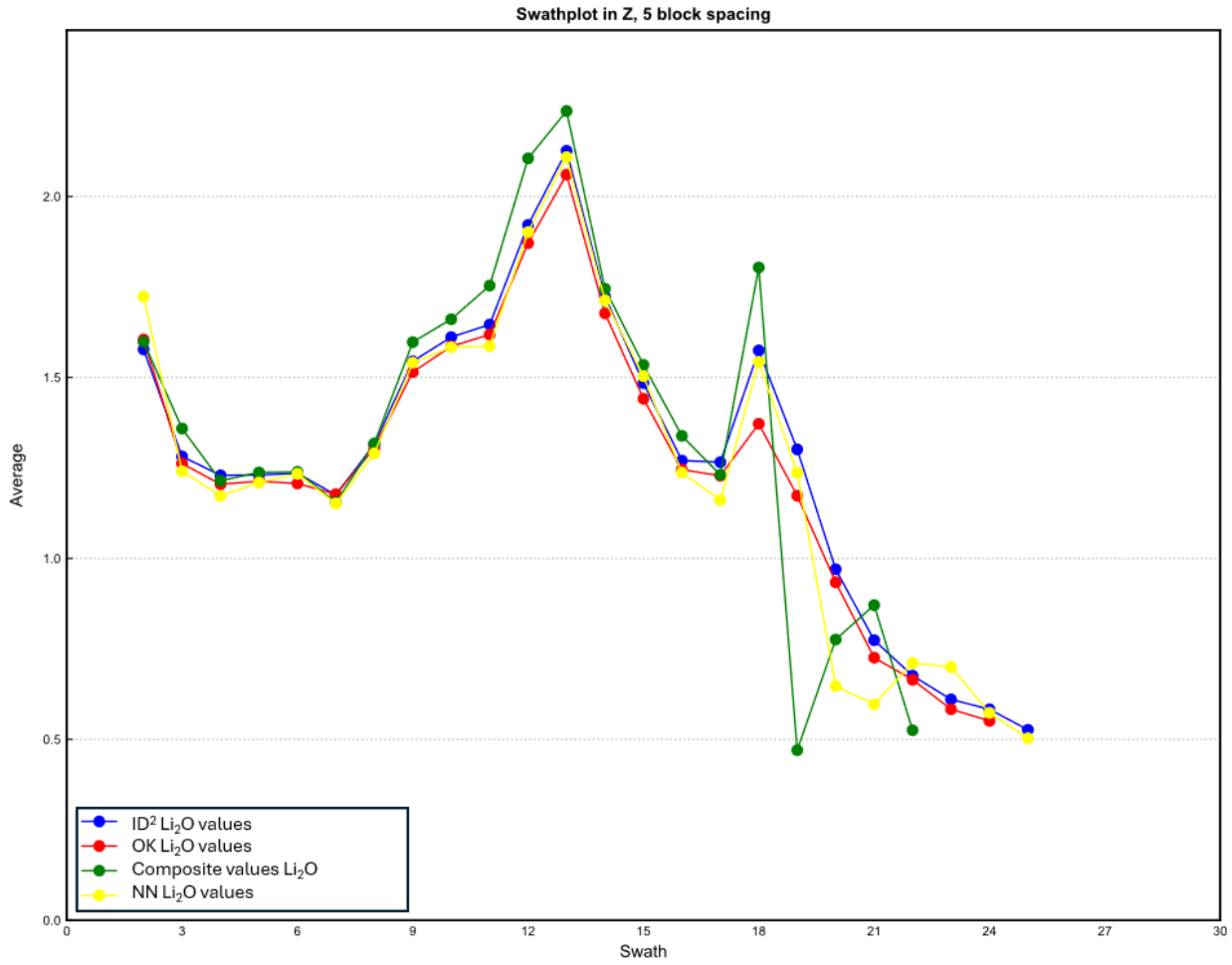


Figure 14.18: Li_2O Swath Plot in Easting (X) Direction (CV13 Zone 101)

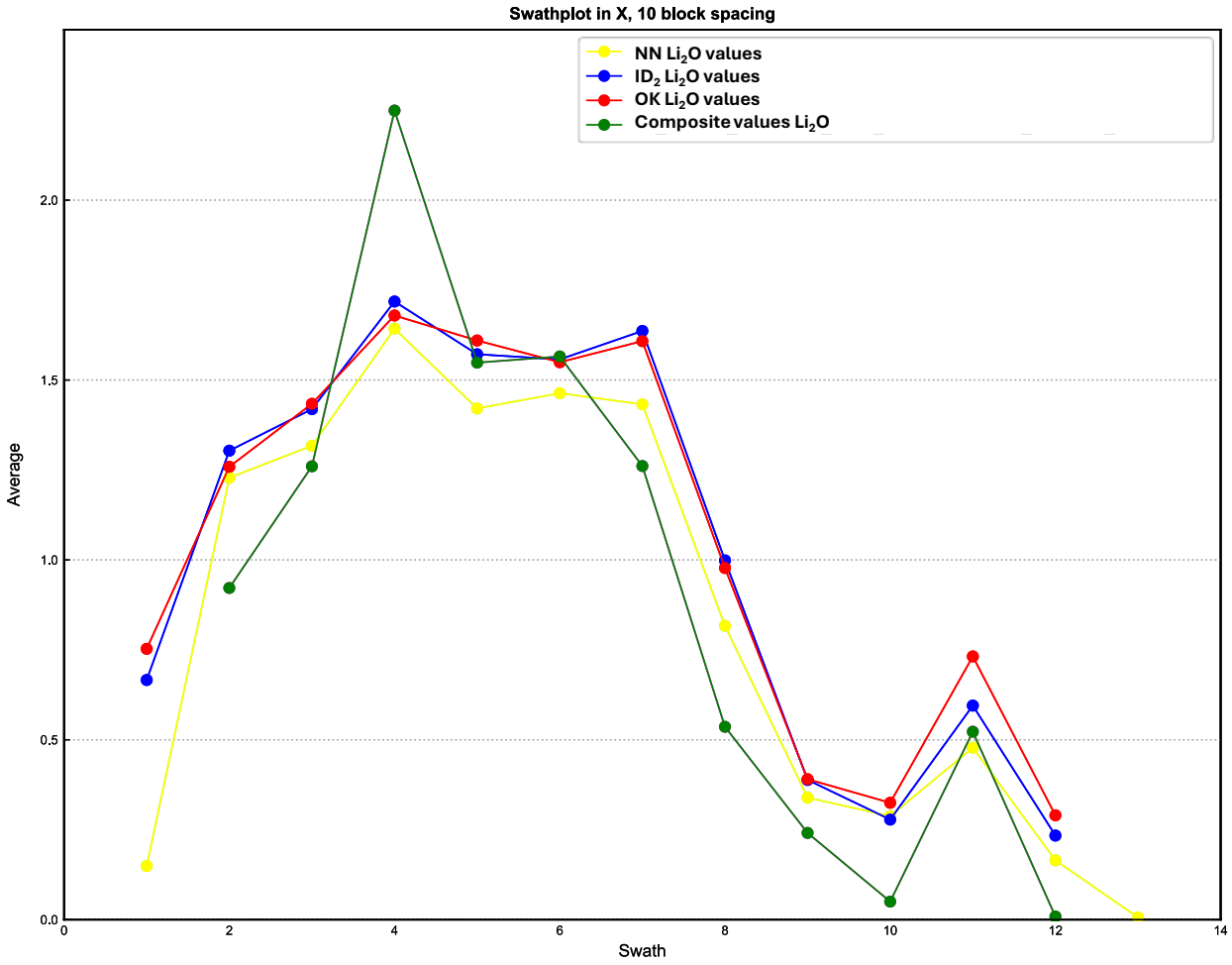
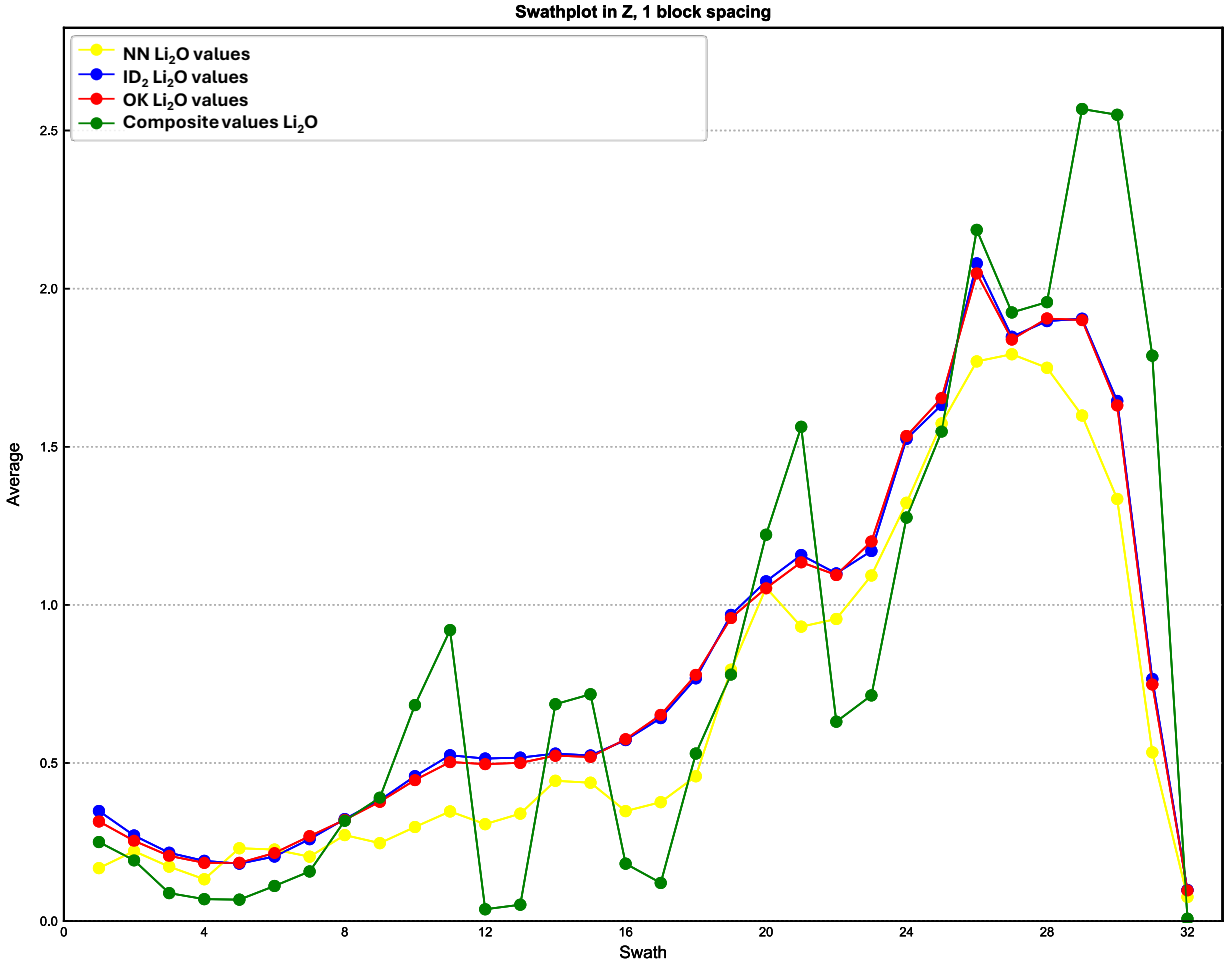


Figure 14.19: Li_2O Swath Plot in Elevation (Z) Direction (CV13 Zone 101)



14.1.9 Sensitivity Analysis

The sensitivity analysis for the Shaakichiuwaanaan MRE is presented as the sum of the open-pit and underground constrained and classified resources at the same cut-off (Table 14.26). The sensitivity analysis by cut-off grade defines significant tonnage at very high-grade, primarily reflecting the Nova Zone at CV5 and the Vega Zone at CV13 (Figure 14.20; Figure 14.21).

Table 14.26 should not be interpreted as a Mineral Resource. The table presents the sum of the open-pit and underground constrained and classified resources at the same cut-off. The data is presented to demonstrate the Mineral Resource tonnage and grade sensitivity to various cut-off grades.

Table 14.26: Sensitivity Analysis for Shaakichiuwaanaan MRE

	CV5 Spodumene Pegmatite				CV13 Spodumene Pegmatite			
	Indicated		Inferred		Indicated		Inferred	
Cut-off grade (%)	Tonnes ≥ cut-off	Average grade (Li_2O) ≥ cut-off (%)	Tonnes ≥ cut-off	Average grade (Li_2O) ≥ cut-off (%)	Tonnes ≥ cut-off	Average grade (Li2O) ≥ cut-off (%)	Tonnes ≥ cut-off	Average grade (Li2O) ≥ cut-off (%)
0.1	119 190 000	1.21	15 120 000	1.13	6 680 000	1.74	22 280 000	1.27
0.2	109 390 000	1.30	14 710 000	1.16	6 510 000	1.78	21 140 000	1.33
0.3	104 540 000	1.35	14 260 000	1.19	6 320 000	1.83	20 210 000	1.38
0.4	101 450 000	1.38	13 690 000	1.22	6 100 000	1.88	19 300 000	1.43
0.5	98 570 000	1.41	13 070 000	1.26	5 850 000	1.94	18 220 000	1.48
0.6	95 710 000	1.43	12 280 000	1.30	5 590 000	2.01	17 070 000	1.55
0.7	92 100 000	1.46	11 300 000	1.36	5 330 000	2.07	15 910 000	1.61
0.8	87 030 000	1.50	10 230 000	1.42	5 090 000	2.13	14 620 000	1.69
0.9	80 870 000	1.55	9 290 000	1.48	4 870 000	2.19	13 390 000	1.76
1	73 450 000	1.62	8 250 000	1.54	4 630 000	2.26	12 120 000	1.85
1.1	65 580 000	1.68	7 230 000	1.61	4 390 000	2.32	10 830 000	1.94
1.2	57 490 000	1.76	6 260 000	1.69	4 150 000	2.39	9 630 000	2.04
1.3	49 640 000	1.84	5 240 000	1.77	3 910 000	2.46	8 540 000	2.15
1.4	42 290 000	1.92	4 150 000	1.89	3 670 000	2.53	7 580 000	2.25
1.5	35 760 000	2.01	3 430 000	1.98	3 400 000	2.62	6 650 000	2.36
1.6	30 050 000	2.10	2 830 000	2.07	3 130 000	2.71	5 870 000	2.47
1.7	25 190 000	2.19	2 340 000	2.16	2 850 000	2.82	5 190 000	2.57
1.8	21 000 000	2.27	1 950 000	2.25	2 630 000	2.91	4 590 000	2.68
1.9	17 360 000	2.36	1 580 000	2.34	2 450 000	2.99	4 100 000	2.78
2	14 260 000	2.45	1 320 000	2.42	2 270 000	3.07	3 700 000	2.87

**Note: Errors may occur in totals due to rounding.*

Figure 14.20: Shaakichiuwaanaan Mineral Resource Grade-Tonnage Curves for CV5

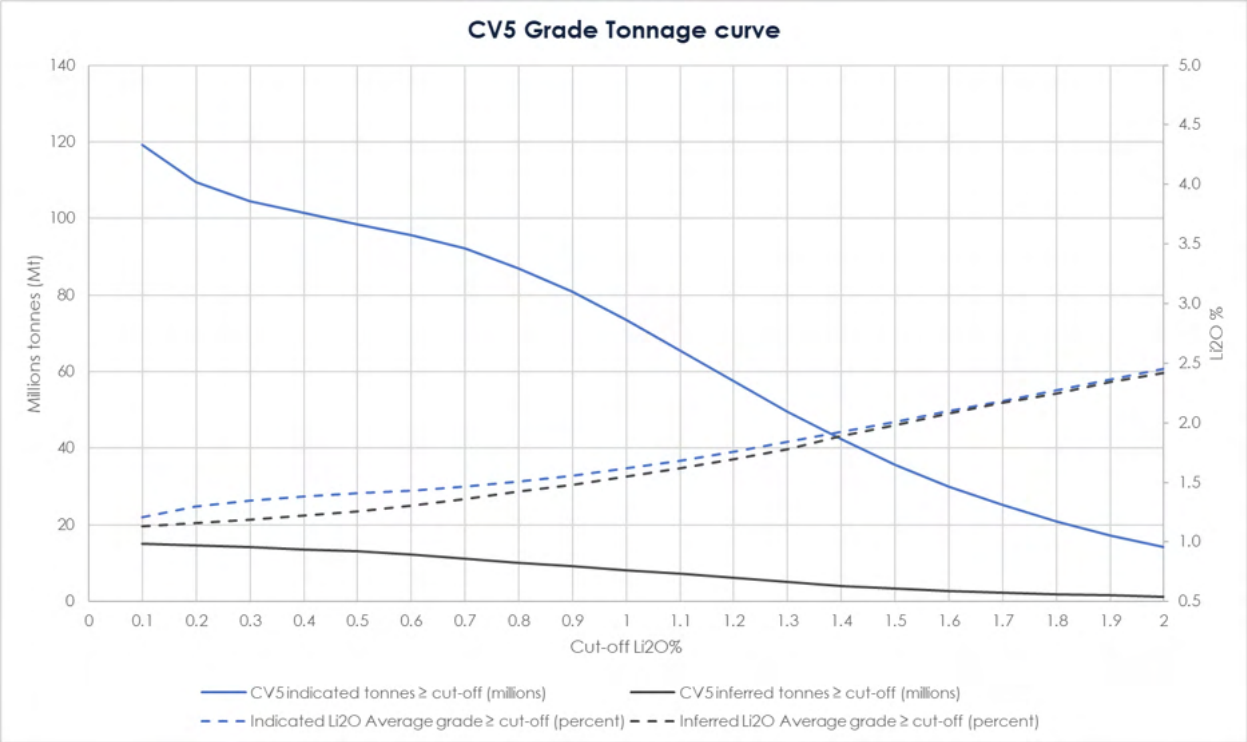
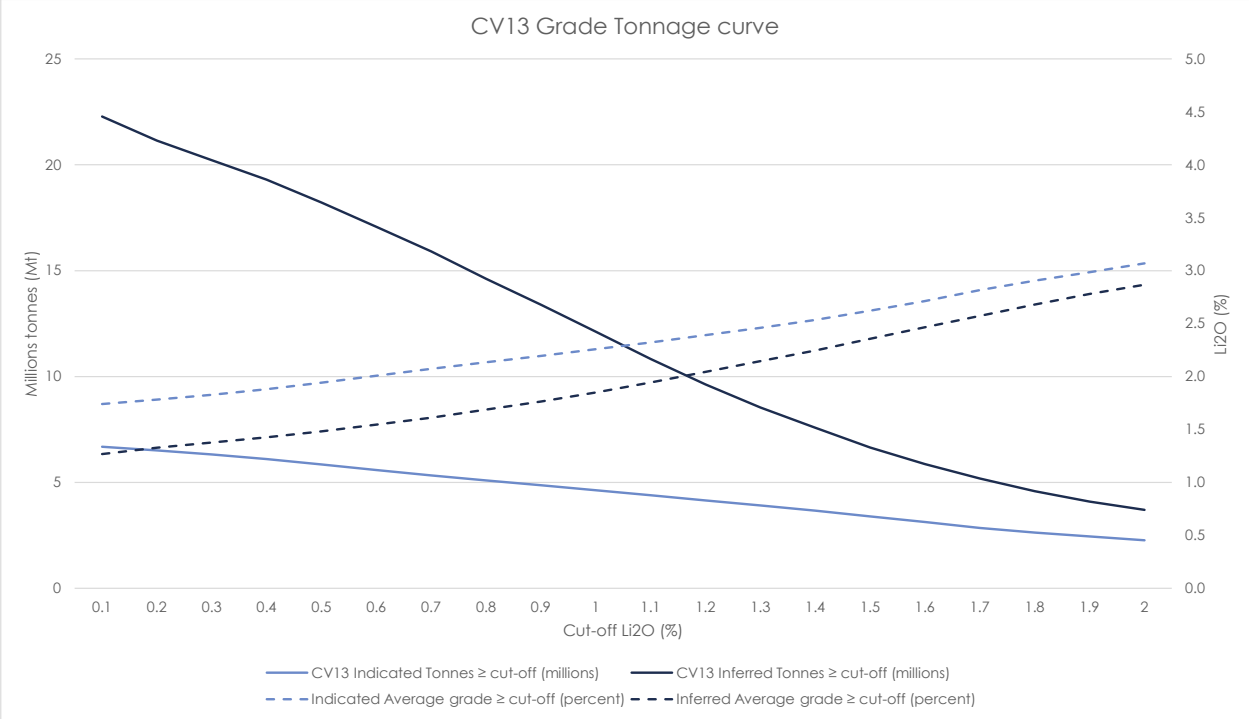


Figure 14.21: Shaakichiuwaanaan Mineral Resource Grade-Tonnage Curves for CV13



14.1.10 Previous Estimates

The third MRE for the Project (MRE-3) (Table 14.27), with an effective date of January 6, 2025, is no longer considered current. It included the CV5 and CV13 pegmatites (PMET Resources, 2025b).

Table 14.27: Shaakichiuwaanaan Mineral Resource Estimate (January 6, 2025)

Pegmatite	Classification	Mass	Li_2O	Ta_2O_5	Ga	Contained LCE (Mt)
		t	%	ppm	ppm	
CV5 & CV13	Indicated	107,955,000	1.40	166	66	3.75
	Inferred	33,280,000	1.33	156	65	1.09

The change in the current Mineral Resource is attributed to a number of factors:

- Additional in-fill drilling on CV13 to further understand the deposit and increase confidence.
- The addition of new elements (Cs) in the resource.

The second MRE for the Project (MRE-2) (Table 14.28) with an effective date of August 21, 2024, is no longer considered valid. It included the CV5 and CV13 pegmatites (McCracken, et al., 2024).

Table 14.28: Shaakichiuwaanaan Mineral Resource Estimate (August 21, 2024)

Pegmatite	Classification	Tonnes	Li_2O (%)	Ta_2O_5 (ppm)	Contained Li_2O (Mt)	Contained LCE (Mt)
CV5 & CV13	Indicated	80,130,000	1.44	163	1.15	2.85
	Inferred	62,470,000	1.31	147	0.82	2.03

The maiden MRE for the Project (MRE-1) (Table 14.29) with an effective date of June 25, 2023, is no longer considered valid. It included the CV5 Pegmatite only (McCracken & Cunningham, 2023).

Table 14.29: Mineral Resource Statement (June 25, 2023)

Pegmatite	Classification	Tonnes	Li_2O (%)	Ta_2O_5 (ppm)	Contained Li_2O (Mt)	Contained LCE (Mt)
CV5	Inferred	109,242,000	1.42	160	1.55	3.84

15. MINERAL RESERVE ESTIMATES

15.1 Introduction

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).

In line with these standards, the Mineral Reserves are based solely on Indicated Mineral Resources. Inferred Mineral Resources have not been included in the Mineral Reserve. Any Inferred Resources that fall within the mine design envelope have been treated as waste and assigned a grade of 0% Li_2O for the purposes of mine planning and economic evaluation.

The Shaakichiuwaanaan Project hosts a Probable Mineral Reserve, which includes the CV5 Pegmatite only, of 84.3 Mt at 1.26% Li_2O for 2.62 Mt of contained lithium carbonate equivalent (LCE), as summarized in Table 15.1.

It should be noted that there are no Proven Reserves; only Probable Reserves are declared.

Table 15.1 Shaakichiuwaanaan Project Mineral Reserve

Area	Category	Tonne (Mt)	Grade (%Li_2O)	Contained Li_2O (kt)	Contained Lithium (Li) (kt)	Contained LCE (kt)
Open Pit	Proven	0.0	0.0	0.0	0.0	0.0
	Probable	49.2	1.12	551.9	256.4	1,364.7
Underground	Proven	0.0	0.0	0.0	0.0	0.0
	Probable	35.1	1.45	508.0	236.0	1,256.0
Total (Open Pit + Underground)	Proven	0.0	0.0	0.0	0.0	0.0
	Probable	84.3	1.26	1,059.9	492.4	2,620.7
	Proven and Probable	**84.3**	**1.26**	1,059.9	**492.4**	2,620.7

**Notes on Mineral Reserves:*

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).*
2. *The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. As such, the Mineral Reserves are based on Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.*
3. *Mineral Reserves are estimated using a long-term lithium price of USD 1,303/t of spodumene concentrate @ 5.5% Li_2O and an exchange rate CAN/USD of 1.32.*

4. *The Qualified Person for the estimate is Carl Michaud, P.Eng., MBA. The estimate has an Effective Date of September 11, 2025.*
5. *The Mineral Reserves for open pit are estimated using a cut-off grade of 0.40% Li_2O. Open pit marginal material containing a grade above 0.37% Li_2O is also included within this statement. Mineral Reserves for underground stoping are estimated using a cut-off grade of 0.70%. Underground development tonnages containing material above 0.37% Li_2O are also included in the statement.*
6. *The following mill recovery equation was used in the cut-off grade recovery:*

$$Mill\ Recovery = \frac{75 * (1 - e^{-1.995 * Li2O\ feed\ Grade})}{100}$$

7. *The open pit strip ratio is 3.40, and the dilution factor is 2.0% based on the smallest mining unit (SMU). The OP mine mining recovery is 97%.*
8. *The underground mine average external dilution factor is 12.7% including 3.9% for backfill dilution and 8.8% for ELOS dilution.*
9. *For the underground Mineral Reserves, a minimum mining width of 5 m was applied with a mining recovery of 90% for all stopes, while 100% extraction was assumed for all development mining.*
10. *Contained lithium oxide (Li_2O), lithium, and lithium carbonate equivalent (LCE) are reported without accounting for metallurgical recovery*
11. *Total may not sum due to rounding; rounding followed the recommendations of the NI 43-101.*

15.2 Open Pit

15.2.1 Mineral Resource Block Model

The Mineral Resource block model for the CV5 Pegmatite was imported into Deswik CAD™ software as a single block model. The Mineral Resource block model, developed by BBA, is sub-blocked to a resolution of 2.5 m x 1.25 m x 1.25 m with a parent block of 10 m x 5 m x 5 m. For the purpose of open pit optimizations, the Mineral Resource block model provided was regularized and reblocked to a uniform block size of 10 m x 5 m x 5 m to meet the requirements of the optimization software.

15.2.2 Open Pit Optimization

Open pit optimization was conducted using GEOVIA WhittleTM version 2022 to determine the optimal economic pit shell to guide the pit design process. This software employs the Pseudoflow algorithm, which operates on a block model of the orebody by progressively constructing lists of related blocks that should or should not be mined. The algorithm uses the economic value of the blocks to define a pit outline that maximizes the total economic value, while adhering to geotechnical constraints such as pit slope angles (defined as structure arcs) and physical constraints represented by “heavy blocks” within the software.

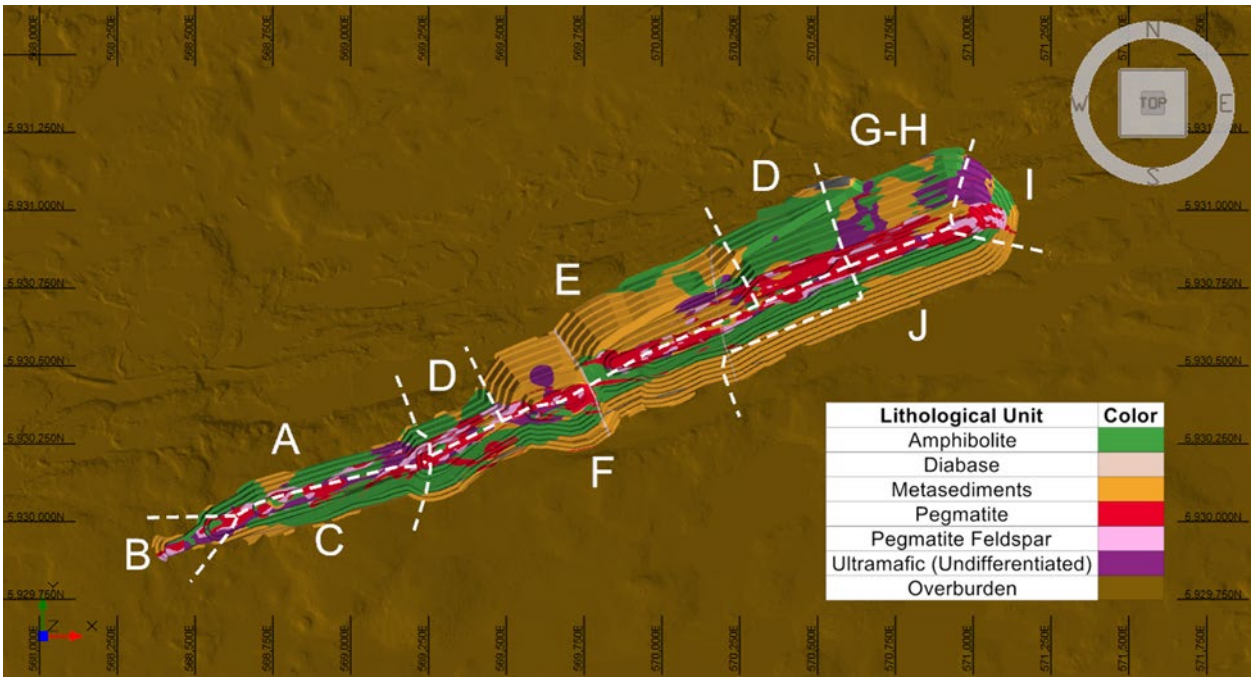
To enable the use of the Pseudoflow algorithm, the 10 m x 5 m x 5 m block model was converted to 10 m x 10 m x 10 m for the creation of structural arcs. The pit optimization process then established the ultimate pit limits (pit shell), providing the basis for the ultimate pit design. Only Indicated Resources were considered in the optimization in accordance with NI 43-101 disclosure standards.

15.2.3 Slope Recommendations

A geotechnical study was completed by Alius Mine Consulting, establishing design parameters varying by geological zone and lithology. The geotechnical analysis incorporates considerations of rock mass characteristics as well as the orientation and spacing of structural discontinuities (e.g., joints, faults, bedding) across different sectors and lithological units.

Details regarding the geotechnical assessment and resulting design parameters are presented in Section 16 of this report. The geotechnical domains defined for the purpose of pit slope design are illustrated in Figure 15.1.

Figure 15.1: Geotechnical Domains by Lithology and Orientation (from Alius, 2025)



Source: GMS, June 2025.
**Note: Not to scale.*

The pit is subdivided into three (3) main areas based on geometrical, lithological, and geomechanical characteristics. Significant differences are observed between lithologies, particularly within the ultramafic group. The geotechnical recommendations presented are based on an analysis of televiewer data collected from 2021 to 2024, results from direct shear tests, Uniaxial Compressive Strength Tests (UCS), and Brazilian tensile strength tests. The geotechnical benches design criteria are summarized in Table 15.2 and Table 15.3.

Depending on the face orientations and lithologies, the Bench Face Angle (BFA) varies between 40° to 75°. For walls higher than 160 m, the design includes a geotechnical bench 7.5 m wider than the standard bench width. A 10 m-wide transition bench was incorporated at the contact between the overburden and the bedrock.

Table 15.2: Geotechnical Benches Design Criteria

Geotechnical Bench Requirements	
Geotechnical Bench	7.5 m wider than standard bench width of the sector. Spaced vertically at 160 m intervals
At the transition between overburden and bedrock	10 m wide catch bench.

Table 15.3: Detailed Slope Design Parameters

Area	Wall	Lithology	Bench Height (m)	Bench Width (m)	Bench Face Angle, Degrees	Inter-Ramp Angle "IRA", Degrees
West	North	AMP	20	10.5	68	47
	West	AMP+ PEG+ UMU	20	8.5	75	55
	South	AMP	20	11	75	51
Central	North	META	20	8.5	50	38
	North	AMP	20	8.5	59	44
	South	META	20	11	75	51
	South	AMP	20	11	75	51
East	North	UMU	20	8.5	40	32
		AMP	20	8.5	47	36
	East	AMP+ PEG+ UMU+META	20	8.5	75	55
	South	META	20	12	75	49
		AMP	20	12	75	49

15.2.4 Mining Dilution and Mining Recovery

G Mining Services developed a proprietary dilution algorithm to simulate realistic mining conditions for open pit operations. The approach uses a 1-metre dilution skin based on the smallest mucking unit (15 m^3 bucket) and the average block width (5 m). For each ore block adjacent to a waste block, the grade and density of the ore block are recalculated by incorporating the contribution of the dilution skin from adjacent waste

blocks, which can include mineralized material below the cut-off grade. This results in more conservative but operationally realistic block grades. The algorithm is applied to the regularized block model with 10 m x 5 m x 5 m cells. A mining recovery factor of 97% is also used to account for ore Recovery. The impact of both dilution and ore recovery is fully reflected in the reported Mineral Reserves and considered in the pit design and economic evaluations.

15.2.5 Pit Optimization Parameters and Cut-Off-Grade

A summary of the pit optimization parameters is presented in Table 15.4 based on a nominal mining throughput of 3.0 Mt/yr. The lithium price used for the Mineral Reserve estimation is set at USD 1,303/t spodumene concentrate at 5.5% Li_2O concentrate. A 2% NSR that is applicable to the open pit component has been incorporated into the block model. The reference mining cost was derived from the previous PEA study and updated to reflect current labour and fuel prices, resulting in a cost of $5.00/t for overburden and $6.81/t for ore and waste. An incremental cost of $0.05/t per 10 m bench is also included.

The total ore material-based cost, including processing, G&A, power and energy, and tailing management, is estimated at $35.70/t milled.

Metallurgical recoveries are evaluated by the following formula:

$$\text{Mill Recovery} = \frac{75 * (1 - e^{-1.995 * Li2O\ feed\ Grade})}{100}$$

Using the ore material-based cost and the mill recovery, a grade of 0.37% Li_2O is calculated. With the addition of the mining costs presented in Table 15.4, the cut-off grade is 0.40% Li_2O. Mining costs include dewatering, technical services support. blasting, loading material into haul trucks, and hauling. This cut-off grade value has been used for the Mineral Reserves statement calculation.

Table 15.4: Economic Optimization Parameters

Parameters	Units	Value
Economic Parameters		
Discount Rate	%	8.0
Exchange Rate	CAN/USD	1.32
Lithium Price at 5.5% Li_2O Concentrate	USD/t	1,303
Transport Cost	CAD/t wet concentrate	226.22
Royalty Rate	%	2.0
Processing Assumptions		
Plant Throughput	t/yr	3,000,000
Plant Recovery	%	Varies
Ore-Based Costs		
Processing	CAD/t ore milled	14.91
General & Administration Costs	CAD/t ore milled	17.47
Power and Energy	CAD/t ore milled	1.41
Tailing	CAD/t ore milled	1.91
Total Ore-Based Cost	**CAD/t ore milled**	**35.70**
Cut-Off Grade Recovered		
Marginal Material Cut-Off Grade	*%Li_2O*	0.37
Cut-Off Grade	*%Li_2O*	0.40
Mining Costs		
Overburden Mining Cost	CAD/t mined	5.00
Mining Cost ORE	CAD/t mined	6.81
Mining Cost WASTE	CAD/t mined	6.81
Incremental Bench Cost	CAD/10 m bench	0.05

15.2.6 Pit Optimization Results

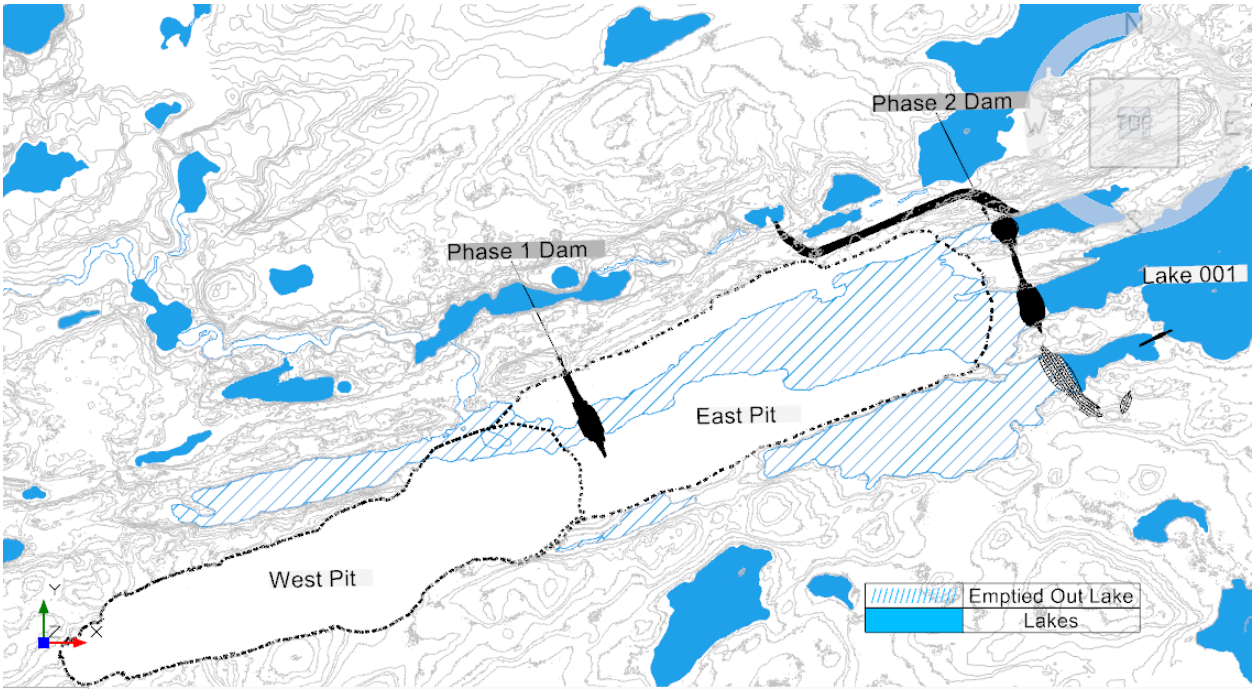
The results of the Whittle™ nested pit shell analysis are presented in Sections 15.2.6.1 and 15.2.6.2 and are based exclusively on Indicated Resources (M&I). The nested pit shells generated a range of revenue factors, which were applied to scale up and down the base case selling price.

The Project is naturally divided into two (2) zones referred to as the West Pit and East Pit.

The West Pit encompasses the portion of the deposit that is the least affected by Lake 001 and is located on the western side of the deposit.

- The East Pit covers the remaining portion of the deposit, as illustrated in Figure 15.2.

Figure 15.2: West and East Pits Delimitation



Source: GMS, June 2025.
**Note: Not to scale.*

15.2.6.1 West Pit

The West Pit is physically constrained by the location of the first dam construction. Prioritizing the mining of the West Pit allows early access to ore content while delaying most of the impacts on Lake 001. Table 15.5 presents the Whittle optimization results for the West Pit, based on Indicated Resources (M&I).

Table 15.5: M&I Pit by Pit Results @ USD 1,303/t Conc.

Pit Shell	Best Case Disc. @ 8% (CAD M)	Specified Disc. @ 8% (CAD M)	Worst Case Disc. @ 8% (CAD M)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Waste Tonnage (kt)	Grade (%Li_2O)	Conc. (kt)
1	383	383	383	4,417	1,981	1.23	2,436	1.40	503
2	428	428	428	4,995	2,236	1.23	2,760	1.39	565
3	493	492	492	5,899	2,642	1.23	3,257	1.37	658
4	527	526	526	6,443	2,872	1.24	3,571	1.36	709
5	537	535	535	6,573	2,952	1.23	3,621	1.35	724
6	609	607	606	8,014	3,501	1.29	4,513	1.32	841
7	626	623	622	8,361	3,635	1.30	4,726	1.31	868
8	712	707	706	10,854	4,333	1.51	6,521	1.29	1,016
9	828	821	818	13,465	5,365	1.51	8,101	1.25	1,215
10	859	851	847	14,222	5,674	1.51	8,548	1.23	1,271
11	879	871	866	14,745	5,888	1.50	8,856	1.22	1,308
12	1,067	1,051	1,042	22,373	7,519	1.98	14,854	1.21	1,660
13	1,203	1,186	1,172	27,436	8,808	2.11	18,628	1.20	1,926
14	1,286	1,266	1,247	31,016	9,735	2.19	21,280	1.19	2,104
15	1,359	1,337	1,312	33,871	10,680	2.17	23,190	1.17	2,269
16	1,383	1,360	1,334	34,957	10,982	2.18	23,975	1.16	2,322
17	1,431	1,406	1,377	36,960	11,693	2.16	25,267	1.15	2,438
18	1,447	1,421	1,391	37,817	11,897	2.18	25,921	1.14	2,476
19	1,465	1,437	1,405	38,565	12,231	2.15	26,334	1.14	2,524
20	1,487	1,458	1,425	39,985	12,529	2.19	27,456	1.13	2,581

Pit Shell	Best Case Disc. @ 8% (CAD M)	Specified Disc. @ 8% (CAD M)	Worst Case Disc. @ 8% (CAD M)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Waste Tonnage (kt)	Grade (%Li_2O)	Conc. (kt)
21	1,492	1,462	1,428	40,246	12,607	2.19	27,639	1.13	2,594
22	1,501	1,470	1,436	40,861	12,735	2.21	28,126	1.13	2,617
23	1,506	1,475	1,439	41,154	12,836	2.21	28,318	1.13	2,632
24	1,525	1,492	1,453	42,495	13,243	2.21	29,252	1.12	2,692
25	1,551	1,515	1,470	44,357	13,748	2.23	30,608	1.11	2,769
26	1,553	1,517	1,471	44,521	13,792	2.23	30,729	1.11	2,776
27	1,572	1,534	1,483	45,990	14,210	2.24	31,780	1.10	2,836
28	1,582	1,543	1,489	46,864	14,412	2.25	32,452	1.09	2,868
29	1,587	1,547	1,492	47,314	14,503	2.26	32,812	1.09	2,882
30	1,595	1,554	1,495	48,120	14,681	2.28	33,439	1.09	2,909
31	1,610	1,566	1,503	49,610	15,047	2.30	34,562	1.08	2,963
32	1,617	1,572	1,506	50,262	15,247	2.30	35,015	1.08	2,989
33	1,619	1,574	1,507	50,437	15,299	2.30	35,138	1.08	2,996
34	1,623	1,577	1,508	50,813	15,416	2.30	35,397	1.07	3,011
35	1,626	1,580	1,510	51,223	15,506	2.30	35,717	1.07	3,024
36	1,631	1,584	1,512	51,927	15,635	2.32	36,292	1.07	3,044
37	1,634	1,587	1,512	52,397	15,745	2.33	36,652	1.07	3,059
38	1,636	1,589	1,513	52,747	15,801	2.34	36,946	1.07	3,067
39	1,653	1,602	1,518	55,521	16,290	2.41	39,230	1.06	3,140
40	1,656	1,605	1,520	55,986	16,389	2.42	39,597	1.06	3,154

Figure 15.3: M&I Pit by Pit Results @ USD 1,303/t Conc.

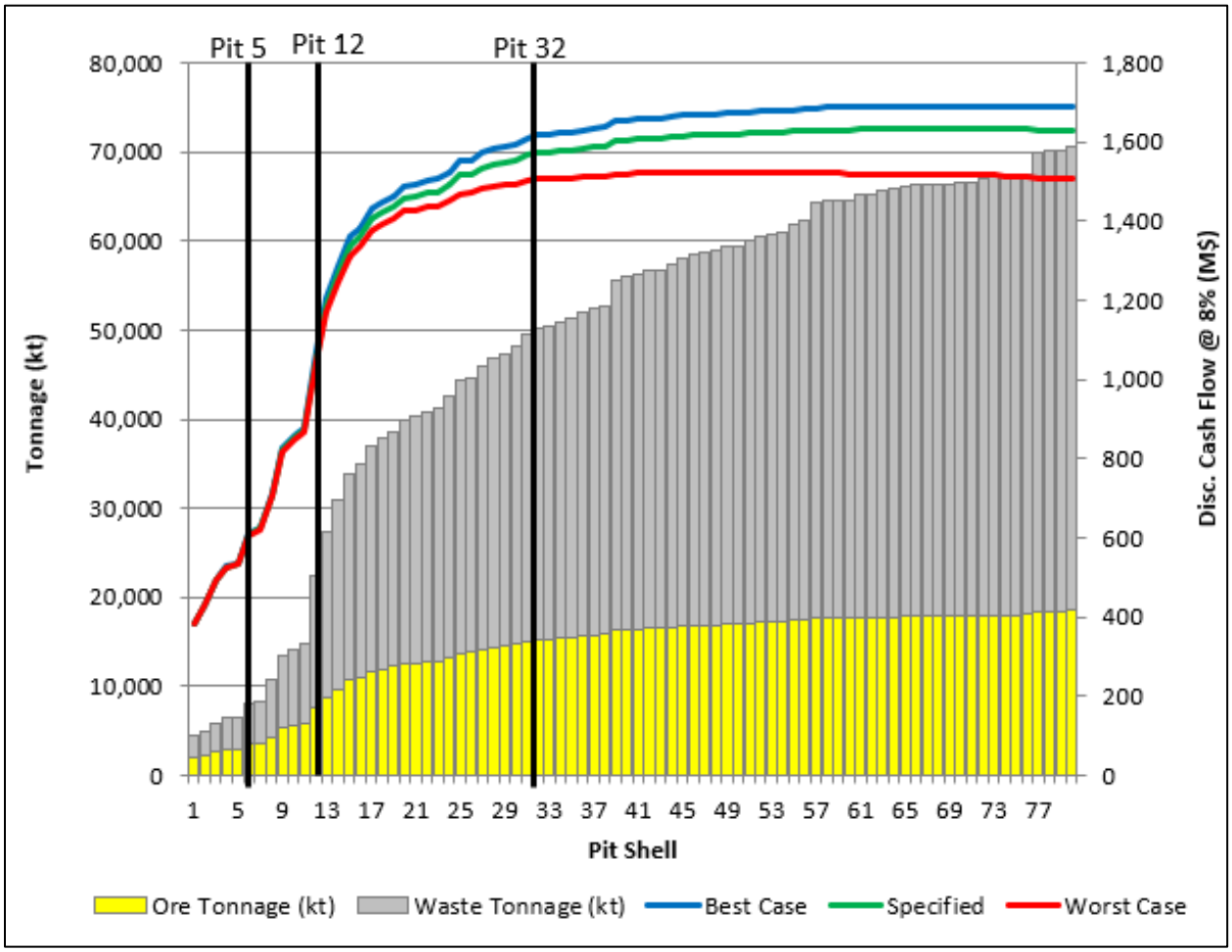


Table 15.6: M&I Pit Shell Selection @ USD 1,303/t Conc.

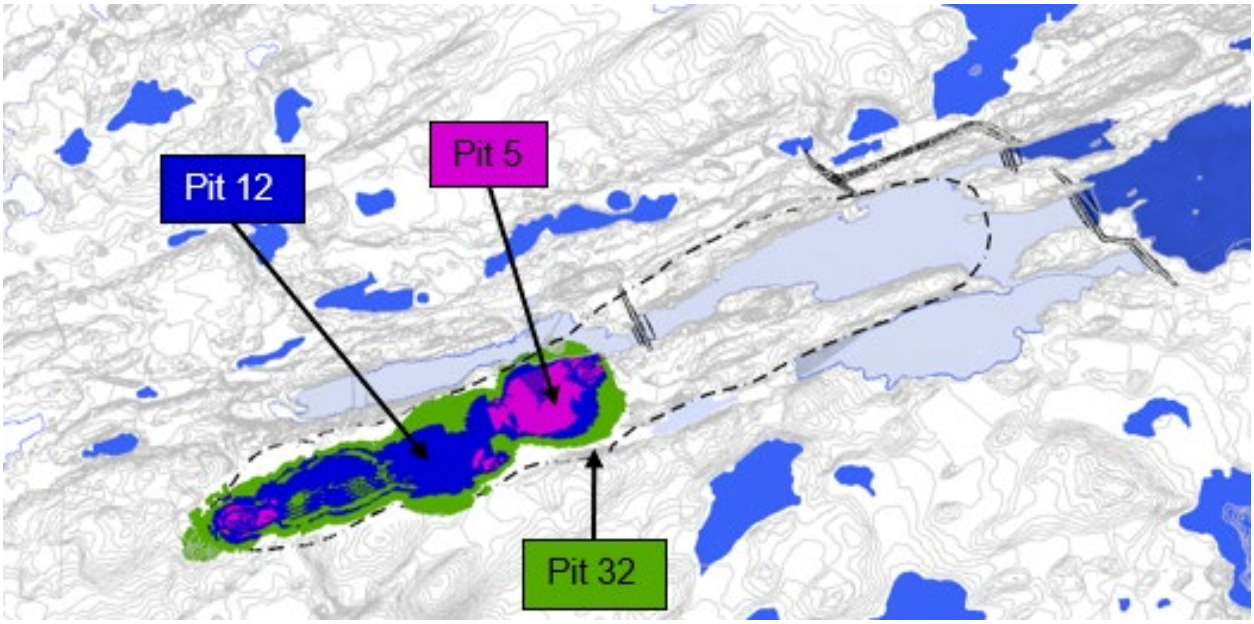
Parameters	Best	Spec.	Worst	Selection
Shell Number	53	53	52	32
Shell RF	0.72	0.72	0.71	0.51
Shell Price	938	938	925	665
Total Tonnage (kt)	60,749	60,749	60,469	50,262
Waste Tonnage (kt)	43,510	43,510	43,289	35,015
Strip Ratio (W:O)	2.52	2.52	2.52	2.30
Ore Tonnage (kt)	17,239	17,239	17,180	15,247
Grade (%Li_2O)	1.04	1.04	1.04	1.08

The pit shell #32, corresponding to a revenue factor of 0.51, was selected as it is believed it will bring the most value to the Project in the early years of the Project. This pit shell limits the stripping ratio early in the Project and corresponds to the early portion of the economic apex, subsequent pit shells providing a marginal increase in value to the Project.

15.2.6.1.1 Pushbacks West Pit

In Figure 15.4, pit shell #5 is presented in pink, pit shell #12 in blue, and the ultimate pit of the West Pit in green. For general understanding, the West Pit shell #5 will be referred to as West sub-pit 1A, the West Pit shell #12 will be referred as West sub-pit 1B and the West Pit shell #32 as the West Pit. The pushbacks have been selected at every interval where a significant increase in waste or size is noticed. A 60 m minimum mining distance between each pushback is maintained, ensuring operational efficiency and adherence to practical equipment maneuvering constraints.

Figure 15.4: West Pit Phasing



Source: GMS, June 2025.
**Note: Not to scale.*

Table 15.7: M&I Push Back Selection @ USD 1,303/t Conc.

Parameters	Push Back 1	Push Back 2
Shell Number	5	12
Shell RF	0.24	0.31
Shell Price	313	404
Total Tonnage (kt)	6,573	22,373
Waste Tonnage (kt)	3,621	14,854
Strip Ratio (W:O)	1.23	1.98
Ore Tonnage (kt)	2,952	7,519
Grade (%Li_2O)	1.35	1.21

15.2.6.2 East Pit

The East Pit represents the second phase in the mining of the Shaakichiuwaanaan Project. It is also physically constrained by the dam located northeast of the ultimate pit shell selected. Numbers below are inclusive of the West and East Pits.

Table 15.8: M&I Pit by Pit Results @ USD 1,303/t Conc.

Pit Shell	Best Case Disc. @ 8% (CAD M)	Specified Disc. @ 8% (CAD M)	Worst Case Disc. @ 8% (CAD M)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Waste Tonnage (kt)	Grade (%Li_2O)	Conc. (kt)
1	568	568	568	5,867	3,052	0.92	2,815	1.37	758
2	712	710	710	7,594	4,045	0.88	3,549	1.34	982
3	803	799	799	8,817	4,680	0.88	4,136	1.32	1,123
4	906	900	900	10,281	5,458	0.88	4,823	1.30	1,288
5	1,018	1,009	1,009	12,330	6,339	0.94	5,990	1.28	1,478
6	1,195	1,179	1,179	15,924	7,864	1.02	8,060	1.26	1,794
7	1,413	1,389	1,389	21,117	9,708	1.18	11,409	1.25	2,198
8	1,544	1,513	1,513	24,942	10,954	1.28	13,987	1.23	2,456
9	1,856	1,801	1,801	33,233	14,659	1.27	18,574	1.18	3,147
10	2,464	2,399	2,374	57,401	22,197	1.59	35,204	1.17	4,711
11	2,502	2,435	2,407	59,048	22,736	1.60	36,312	1.17	4,817
12	2,744	2,657	2,616	73,599	25,964	1.83	47,635	1.17	5,525
13	3,100	2,967	2,900	96,548	31,709	2.04	64,839	1.17	6,745
14	3,420	3,222	3,109	120,854	38,481	2.14	82,373	1.15	8,076
15	3,585	3,337	3,193	136,799	42,540	2.22	94,259	1.15	8,876
16	3,607	3,354	3,203	139,395	43,170	2.23	96,225	1.15	8,999
17	3,635	3,371	3,208	142,110	44,049	2.23	98,061	1.14	9,148
18	3,662	3,391	3,219	145,775	44,724	2.26	101,051	1.14	9,292
19	3,676	3,398	3,219	147,356	45,240	2.26	102,116	1.14	9,376
20	3,692	3,409	3,223	149,855	45,725	2.28	104,130	1.14	9,472

Pit Shell	Best Case Disc. @ 8% (CAD M)	Specified Disc. @ 8% (CAD M)	Worst Case Disc. @ 8% (CAD M)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Waste Tonnage (kt)	Grade (%Li_2O)	Conc. (kt)
21	3,701	3,416	3,228	151,304	46,013	2.29	105,291	1.14	9,529
22	3,709	3,420	3,230	152,625	46,300	2.30	106,325	1.14	9,581
23	3,717	3,426	3,232	153,954	46,547	2.31	107,407	1.14	9,628
24	3,729	3,431	3,229	155,869	47,054	2.31	108,815	1.13	9,708
25	3,790	3,463	3,210	168,463	49,215	2.42	119,248	1.13	10,124
26	3,795	3,466	3,208	169,678	49,422	2.43	120,256	1.13	10,163
27	3,806	3,470	3,197	171,870	49,993	2.44	121,877	1.13	10,250
28	3,819	3,476	3,188	174,788	50,533	2.46	124,254	1.13	10,345
29	3,820	3,476	3,187	175,123	50,604	2.46	124,519	1.13	10,355
30	3,825	3,476	3,180	176,286	50,829	2.47	125,457	1.12	10,392
31	3,844	3,483	3,164	181,802	51,732	2.51	130,070	1.12	10,552
32	3,847	3,483	3,159	182,686	51,968	2.52	130,718	1.12	10,586
33	3,852	3,484	3,156	184,165	52,220	2.53	131,945	1.12	10,628
34	3,857	3,486	3,151	185,745	52,516	2.54	133,229	1.12	10,675
35	3,858	3,486	3,148	186,281	52,631	2.54	133,650	1.12	10,692
36	3,863	3,487	3,141	187,933	52,905	2.55	135,028	1.12	10,737
37	3,865	3,486	3,136	188,569	53,035	2.56	135,534	1.12	10,755
38	3,871	3,488	3,126	191,169	53,345	2.58	137,824	1.12	10,813
39	3,896	3,487	3,052	203,217	54,840	2.71	148,376	1.11	11,087
40	3,901	3,490	3,041	205,709	55,209	2.73	150,500	1.11	11,147

Figure 15.5: M&I Pit by Pit Results @ USD 1,303/t Conc.

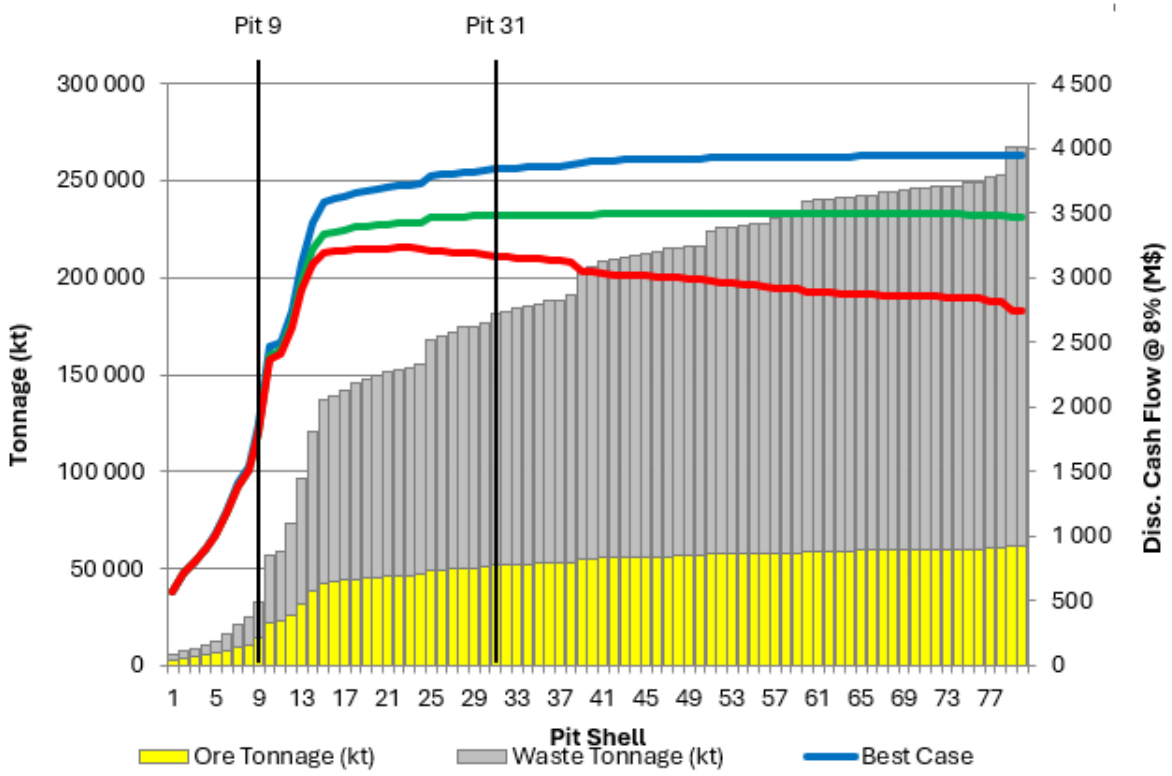


Table 15.9: M&I Final Pit Shell Selection @ USD 1,303/t Conc.

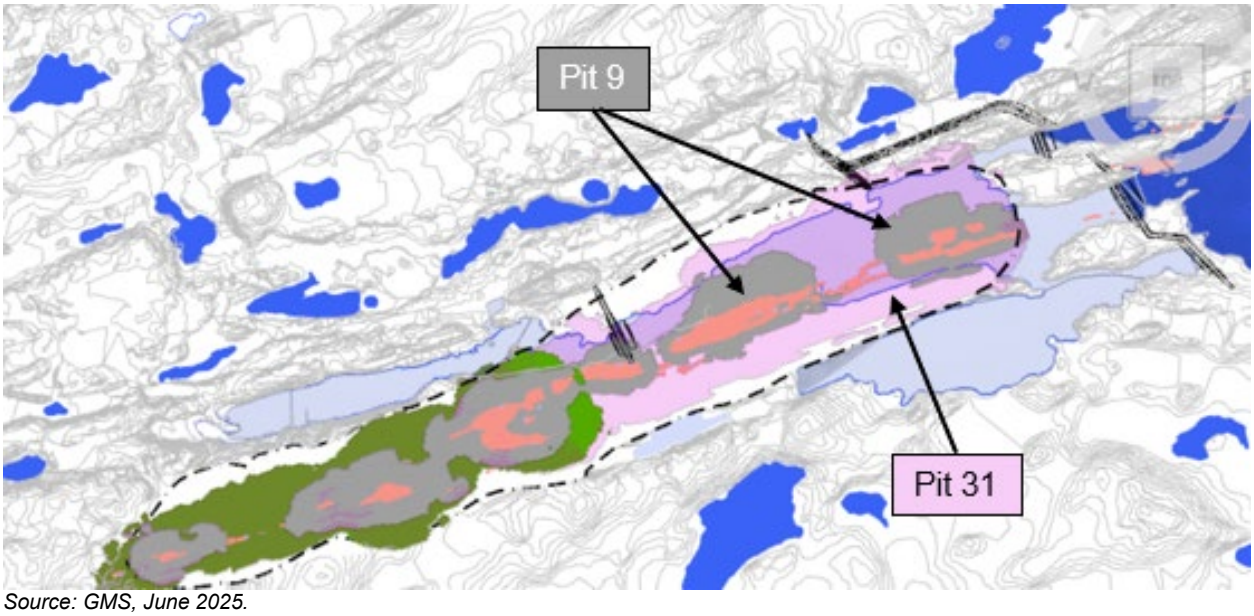
Shell Selection	Best	Spec.	Worst	Selection
Shell Number	53	51	23	31
Shell RF	0.72	0.70	0.42	0.50
Shell Price	938	912	547	652
Total Tonnage (kt)	225,938	224,092	153,954	181,802
Waste Tonnage (kt)	168,188	166,581	107,407	130,070
Strip Ratio (W:O)	2.91	2.90	2.31	2.51
Ore Tonnage (kt)	57,750	57,511	46,547	51,732
Grade (%Li_2O)	1.10	1.10	1.14	1.12

Pit shell #31, corresponding to a revenue factor of 0.50, was selected as it is considered to deliver the greatest overall value to the Project.

15.2.6.2.1 Pushbacks East Pit

In Figure 15.6, pit shell #9 is presented in grey and pit shell #31 is in pink. The East Pit shell #9 will be referred to as East Sub-Pit 3A and 3B, and the East Pit shell #31 as the East Pit. Only one (1) pushback was selected for the East Pit. A 60 m minimum mining distance between each pushback is maintained, ensuring operational efficiency and adherence to practical equipment maneuvering constraints. The numbers presented below are inclusive of the West and East Pits.

Figure 15.6: East Pit Phasing



Source: GMS, June 2025.
**Note: Not to scale.*

Table 15.10: M&I Phasing Selection @ USD 1,303/t Conc.

Parameters	Push Back 1
Shell Number	9
Shell RF	0.28
Shell Price	365
Total Tonnage (kt)	33,233
Waste Tonnage (kt)	18,574
Strip Ratio (W:O)	1.27
Ore Tonnage (kt)	14,659
Grade (%Li_2O)	1.18

15.2.7 Mine Design

15.2.7.1 Ramp Design Criteria

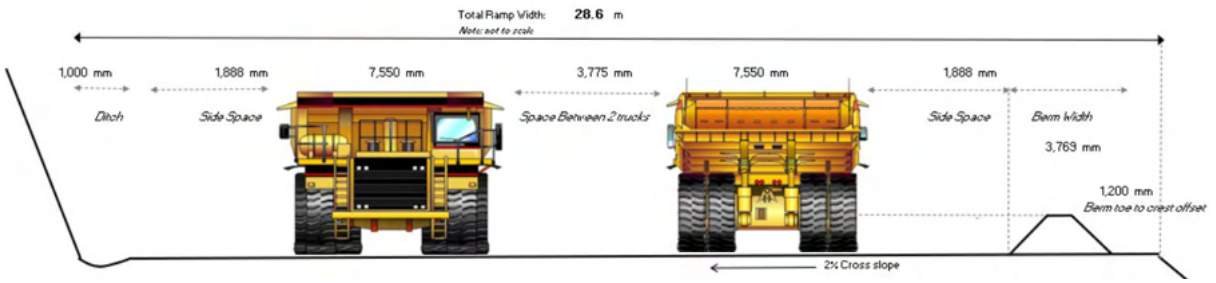
The ramps and haul roads are designed to accommodate the largest equipment considered, that being a 140-tonne class haul truck with a canopy width of 7.6 m.

In accordance with industry regulations and best practices:

- For double-lane traffic, a minimum travel width of three (3) times the width of the largest vehicle is used.
- For single-lane traffic, a minimum travel width of two (2) times the width of the largest vehicle is used.
- The ramp's maximum gradient is set at 10% to ensure safe and efficient hauling operations.

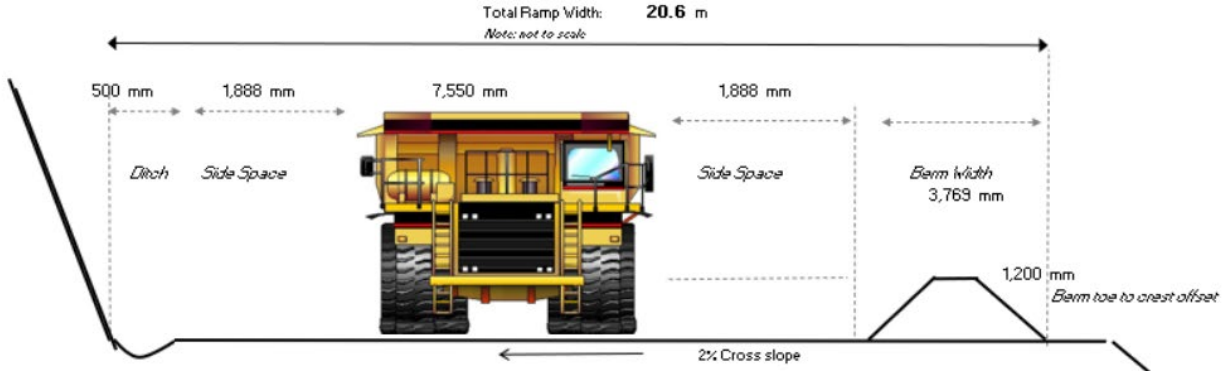
A safety berm along the outer edge of the ramp will be constructed with crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.53 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A ditch planned along the highwall will capture run-off from the pit wall surface and ensure proper drainage of the running surface. The ditch will be 1.0 m wide. To facilitate drainage of the roadway, a 2% cross slope on the ramp is planned. The double-lane ramp is 28.6 m wide, and the single-lane ramp is 20.6 m wide. Single-lane ramps are introduced at the pits' bottom when the benches start narrowing and when the mining rates will be significantly reduced. Double- and single-lane ramp configurations are shown in Figure 15.7 and Figure 15.8, respectively.

Figure 15.7: Double-Lane Ramp Design Criteria



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 15.8: Single-Lane Ramp Design Criteria



Source: GMS, June 2025.
**Note: Not to scale.*

15.2.8 Open Pit Mine Reserves

The Mineral Reserve and stripping ratio estimates are based on the final pit design presented in Section 16 of this report. Table 15.11 presents the Mineral Reserves contained within the final pit design. Waste tonnage shown in Table 15.12 includes Inferred Mineral Resource, which is treated as waste at a grade of 0% Li_2O in accordance with NI 43-101 guidelines.

The Probable Mineral Reserves are inclusive of mining dilution and ore loss. Mining dilution was applied to the block model using a dilution script. A general mining recovery of 97% was applied to the diluted ore tonnage to estimate recoverable reserves.

The total ore tonnage before dilution and ore loss is estimated at 49.78 Mt at an average grade of 1.14%. The dilution skin around ore blocks results in a dilution tonnage of 1.04 Mt with an average lithium grade of 0.05%. This dilution tonnage represents about 2% of the ore tonnage before dilution. The ore loss represents the isolated blocks surrounded by waste and is about 0.03 Mt at 0.45%.

This gives a total of 49.2 Mt of Probable Mineral Reserve with an average grade of 1.12% Li_2O for the open pit reserve component.

Table 15.11: Open Pit Mineral Reserves Statement

Open Pit Mineral Reserves Statement	Tonnage (Mt)	Grade (%Li_2O)
Ore Before Dilution and Ore Loss	49.8	1.14
Add Dilution	1.04	0.05
Less: Ore Loss	(0.03)	(0.45)
Less: Underground Interface Blocks	(0.10)	0.00
Add: Internal Mining Dilution	0.09	0.28
Unrecovered Probable Mineral Reserve	**50.8**	**1.12**

Table 15.12: Shaakichiuwaanaan Open Pit Mineral Reserves and Quantities

Final Pit Quantities	
Probable Mineral Reserve	
Recovered Probable Mineral Reserve	49.2 Mt
Grade (%Li_2O)	1.12%
Lithium Concentrate	6.7 kt
Waste Material (including Overburden)	
Waste Tonnage	167.5 Mt
Total Tonnage (ore + waste)	**216.7 Mt**
Strip Ratio	
3.40	

15.3 Underground Mining

15.3.1 Mineral Reserve Calculation Methodology

The Underground Mineral Reserves were estimated by applying appropriate mining and economic considerations to the Mineral Resource block model for the CV5 Pegmatite. The block model, developed by BBA, was imported into Deswik.CAD™ software as a single dataset with a parent block size of 10 m x 5 m x 5 m, sub-blocked to a minimum resolution of 2.5 m x 1.25 m x 1.25 m.

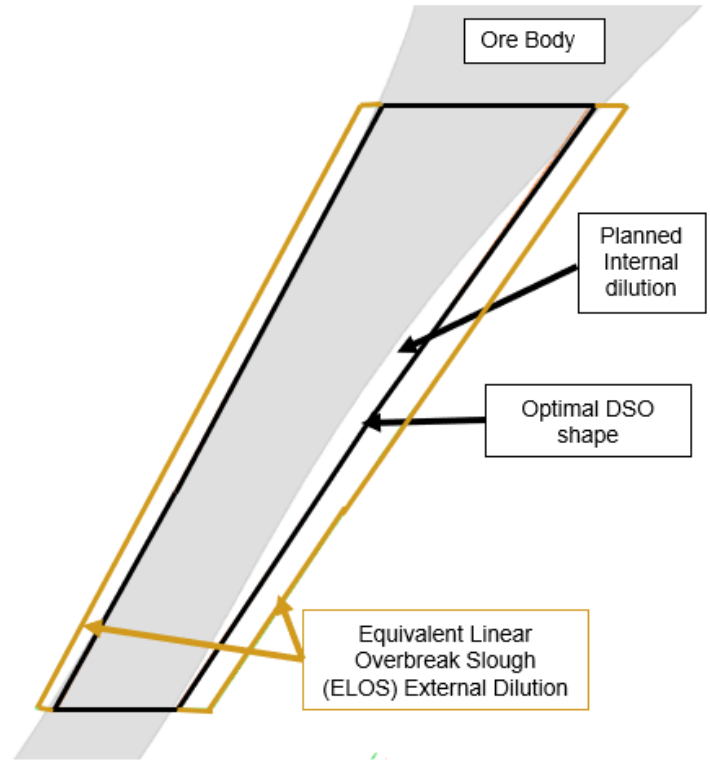
Only Indicated Mineral Resources are considered in the Mineral Reserves for the Shaakichiuwaanaan Lithium Project. Any Inferred Mineral Resources contained within the mine design are treated as waste at an assigned grade of 0% Li_2O.

A preliminary pass using Deswik's stope optimizer (DSO) was employed to identify potentially economic extraction zones using various stope optimization parameters and Equivalent Linear Overbreak Slough (ELOS) dilution assumptions on both the hanging wall and footwall. The resulting stope shapes were reviewed, and uneconomic areas were excluded. Stope shapes deemed to have a reasonable prospect for economic extraction were then adjusted to account for external backfill dilution and mining recovery. The resulting data were compiled to establish the Mineral Reserve estimate.

15.3.2 Dilution and Mining Losses

Dilution parameters were assigned to each stope to estimate the amount of additional material likely to be included during mining operations. An Equivalent Linear Overbreak Slough (ELOS) (Figure 15.9) of 0.75 m was applied to the hanging wall, and 1.25 m to the footwall when these walls are not located in ore, as presented in Section 16.

Figure 15.9: Equivalent Linear Overbreak Slough (ELOS) Schematic



Source: GMS, June 2025.
**Note: Not to scale.*

To account for dilution related to paste backfilling, additional factors were applied as follows:

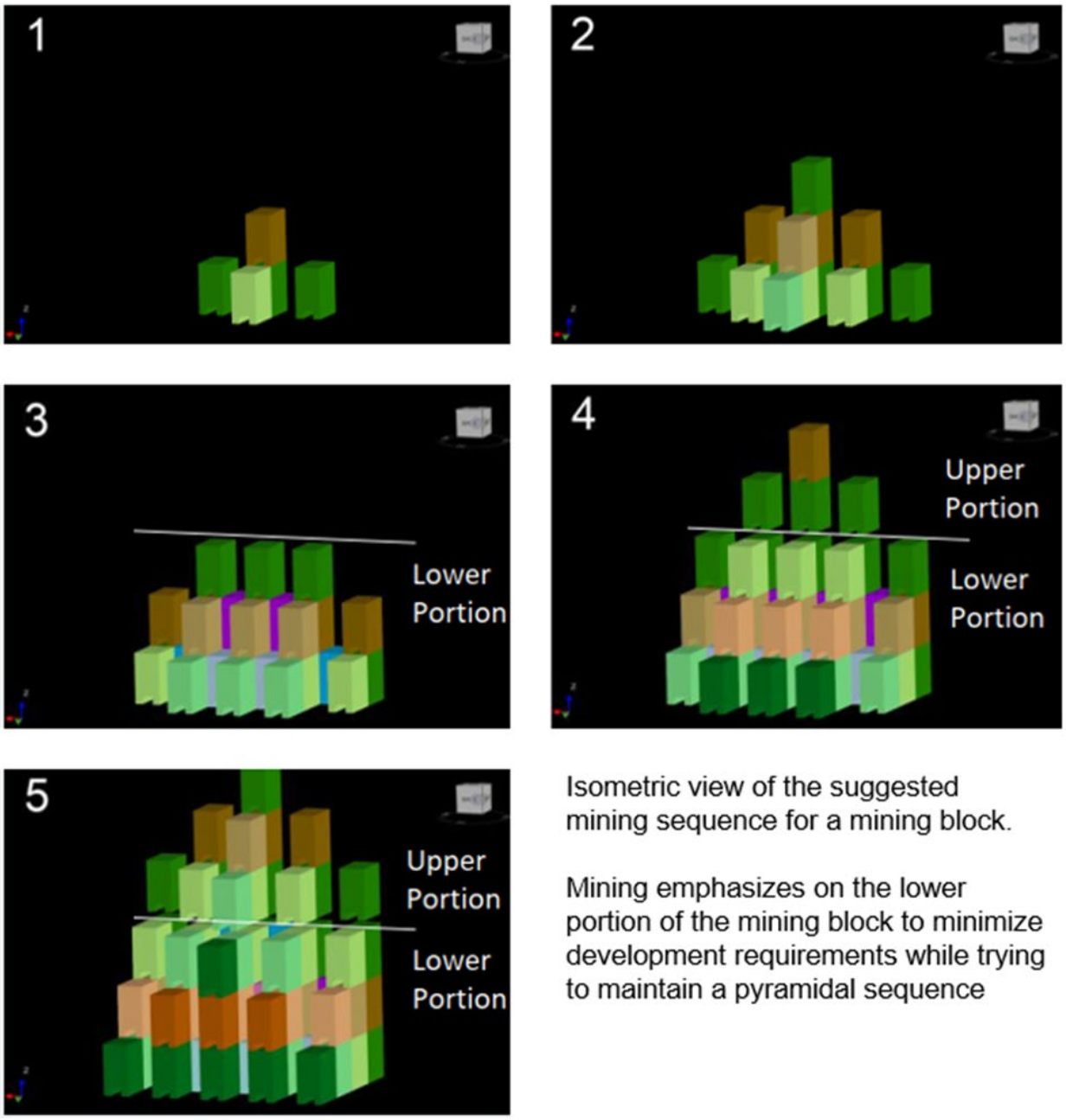
- Floor dilution of 0.5 m where load-haul-dump (LHD) equipment is expected to muck backfill from the top of the previously mined stope.
- Endwall dilution of 0.5 m, due to expected paste dilution from blasting transversely into the adjacent previously backfilled stope.
- Sidewall dilution of 1.0 m, typically applied to secondary stopes situated between two (2) previously mined primary stopes.
- A mining recovery factor of 90% was applied to all stopes.
- No dilution and a 100% mining recovery were applied to ore development.

Table 15.13 summarizes the various backfill dilution factors used based on the stope type, while Figure 15.10 illustrates the different transverse stope configurations considered, reflecting the range of possible mining scenarios.

Table 15.13: Underground Mine Backfill Dilution Parameters

Parameters	Floor (m)	Endwall (m)	Sidewalls (m)
Transverse Stoping			
Initial Primary Stope	0.5	0	0
Initial Secondary Stope	0.5	0	1.0
Subsequent Primary Stope	0.5	0.5	0
Subsequent Secondary Stope	0.5	0.5	1.0
Longitudinal Stoping			
Initial Stope	0.5	0	0
Subsequent Stope	0.5	0.5	0

Figure 15.10: Underground Mine Transverse Stope Types



Source: Alius, June 2025.
**Note: Not to scale.*

15.3.3 Underground Cut-Off Grades and Optimization Parameters

The break-even cut-off grade (COG) is the minimum grade at which the revenue from mined and processed material equals the total operating cost. It represents the economic threshold below which material is considered uneconomic. The COG is calculated based on commodity price, metallurgical recovery, and mining and processing costs.

To evaluate the potentially economical portion of the Mineral Resource Estimate, a break-even cut-off grade was calculated based on the selected mining method (Long Hole Open Stoping - LHOS). Table 15.14 outlines the key technical and economic parameters used for estimating the underground cut-off grade for the Mineral Reserves estimation. The referenced ore and mining costs were based on and derived from the previous PEA study.

Table 15.14: Underground Mine Cut-Off Calculation Parameters

Parameters	Units	Value
Spodumene Market Price	*USD/t Li_2O*	1,303
Exchange Rate	*CAD/USD*	1.32
Royalty Rate	*%*	2
Transportation & Insurance		
Transportation & Insurance	*CAD/wmt*	226.22
Concentrate Grade	*%*	5.50
Mill Recovery		
Plant Recovery	*%*	0.75*(1-EXP(-1.995*% Li_2O *100))
Ore-Based Costs		
Processing	*CAD/t ore*	14.91
General & Administration Costs	*CAD/t ore*	17.47
Power and energy	*CAD/t ore*	1.41
Rehandling	*CAD/t ore*	1.42
Total Ore-Based Cost	***CAD/t ore***	**35.21**
Mining Costs		
Mineralized Material Mined - Underground	*CAD/t mined*	68.66
All in UG mining cost	*CAD/t mined*	103.87
Cut-Offs		
Stoping Cut-Off Grade (No Sustaining Capital)	***% Li_2O***	**0.70**
Development Cut-Off Grade	***% Li_2O***	**0.37**

A zone-by-zone approach was applied to validate the economic viability of each area of the deposit by integrating zone-specific sustaining capital costs into the economic analysis. Multiple iterations were conducted using the stope optimizer tool within Deswik™ software to generate the most optimal stope

shapes. The stope geometry and cut-off grade parameters used in the optimization process are summarized in Table 15.15.

Table 15.15: Underground Mine Stope Optimizer Parameters

Parameters	Units	Value
Method	-	Vertical
Stope Height	m	Variable based on production levels hauling drift at 3%
Strike Length	m	17
Maximum Mining Width (HW to FW)	m	25
Minimum Mining Width (HW to FW)	m	5
HW Dilution	m	0.75
FW Dilution	m	1.25
Minimum / Maximum Dip	°	55/125
Smoothing: Average	-	Strike & Vertical
Cut-Off Grade	%	0.70

15.3.4 Underground Mineral Reserve

The underground Mineral Reserve Estimate is based on the final underground mine design presented in Section 16. Table 15.16 presents a summary of underground Mineral Reserves. Reported Raw DSO Results only include stopes East of 570822 UTM. Stopes located west of that coordinate were reviewed and are either mined by open pit or within uneconomic areas. Cleaned DSO Results excluded stopes located within the crown pillar, within the open pit and within uneconomic areas.

Table 15.16: Underground Mineral Reserves Statement

Steps	Tonnage (Mt)	Grade (%Li_2O)	Contained Lithium (Li) (kt)
Block Model Only with Indicated Mineral Resources	134.3	1.09	680.4
Raw DSO Results	45.3	1.45	305.4
Cleaned DSO Results	37.0	1.51	259.3
Reserves Stopes Without Development	35.0	1.51	245.8
Additional Backfill Dilution	1.4	0.00	0.0
Reserves Stopes with External Dilution Before Mining Recovery	36.4	1.45	245.3

Steps	Tonnage (Mt)	Grade (%Li_2O)	Contained Lithium (Li) (kt)
Mining Losses on Stopes	3.6	1.45	24.5
Mining Stopes Reserve (Diluted & Recovered)	32.8	1.45	220.8
Additional Development Ore	2.3	1.41	15.2
All Ore	35.1	1.45	236.0

15.3.5 Factors Which May Affect Mineral Reserves

Changes in the following factors and assumptions may affect the Mineral Reserve estimate:

- Geological complexity, geological interpretation, and Mineral Resource block modelling.
- Commodity prices, market conditions, and foreign exchange rate assumptions.
- Long-term consumables price assumptions.
- Operating cost assumptions.
- Sustaining capital costs.
- Rock quality, geotechnical constraints, dilution, and mining recovery factors.
- Hydrogeological assumptions.
- Metallurgical process recoveries.
- Mining recovery and dilution assumptions.
- Ability to meet and maintain permitting and environmental licence conditions and the ability to maintain the social licence to operate.

There are no other environmental, legal, title, taxation, socioeconomic, refining, political, or other relevant factors known to the Qualified Person (QP), which would materially affect the estimation of Mineral Reserves, other than those discussed in this Report. It is reasonably expected that all necessary government approvals will be obtained for the Project to proceed.

16. MINING METHODS

16.1 Summary

The CV5 Shaakichiuwaanaan Deposit consists of a large main pegmatite dyke, accompanied by several smaller dykes striking approximately east-west. Planned mining operations will incorporate both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The pegmatite dykes dip at approximately 80° to the north. The deposit extends over a strike length of approximately 4.6 km with mineralization modelled from surface to a depth of 650 m. A significant portion of the orebody is located under Lake 001, which will require partial dewatering to enable open pit operation. A trade-off study of several hybrid scenarios (OP/UG) was conducted to determine the optimal mining scenario. The following criteria and limitations on both technical and non-technical parameters were considered in selecting the preferred scenario:

- Minimizing impacts on Lake 001 reducing fish habitat compensation.
- Maintaining water diversion within the same watershed, with pit limits consistent with the previous PEA study.
- Economic and financial performance.

The dense media separation process plant has a nominal throughput of 5.1 Mtpa at peak capacity. The construction of the process plant will be carried out in two (2) stages, where Stage 1 will reach a nominal processing rate of 2.5 Mtpa at the beginning of Year 2. Year 2 is also the start of the expansion to add a second processing circuit of 2.5 Mtpa. The process plant will be in operation for approximately 19 years, following a four-month ramp-up period.

The OP operation will utilize a fleet of diesel-powered equipment, including drills, hydraulic shovels, and off-highway haul trucks. The Project consists of a single pit with two (2) stages, each of which will be mined using sub pits. In total, the operations will be executed in six (6) separate sub pits. The OP peak mining rate is 23.0 Mtpa over a Life-of-Mine (LOM) of 19 years including the pre-production period. A total of 49.2 Mt at 1.12% Li_2O of ore will come from the open pit. A total of 216.7 Mt of material, including ore, waste rock and overburden, will be moved during the life of mine, resulting in an average stripping ratio of 3.4.

The primary loading and hauling fleet will consist of 15 m^3 diesel-hydraulic shovels paired with 140-metric t off-highway mining trucks. The mining operation will be primarily owner-operated, with contractors responsible for overburden removal and explosives handling. Pre-production mining will extend over approximately 24 months, by providing construction material and removing overburden to establish initial access to the orebody.

A total of 10.5 Mt of waste and 1.5 Mt of ore will be mined during the pre-production and commissioning period. This waste will be used for construction purposes, while the ore will be stockpiled until the process plant commences operation.

The underground (UG) operation comprises a single mine accessible through a portal located near the run-of-mine (ROM) pad. The selected mining method is long-hole open stoping (LHOS) with cemented paste backfill (CPB), using both transverse and longitudinal stoping techniques.

The Life-of-Mine (LOM) for the UG mine is expected to be 21 years including construction, development, pre-production, and the full production period. Over this LOM, the UG mine is expected to be at full production for 16 years. A two-year pre-production period is planned to allow sufficient underground development to be completed to sustain full production. The UG mine is expected to achieve an average production rate of 5,475 tpd of ore, with 5,200 tpd from stope production and an average 275 tpd from lateral development. Development of the UG mine includes approximately 96.7 km of lateral and 1.4 km of vertical development to be excavated. A total of 35.1 Mt of ore is expected to be mined at an average diluted lithium oxide grade of 1.45% Li_2O. The primary production and development equipment fleet include 18-t Battery Electric Vehicle (BEV) load-haul-dump (LHD) coupled with 50-t BEV-powered underground mining trucks.

16.2 Geotechnical Considerations

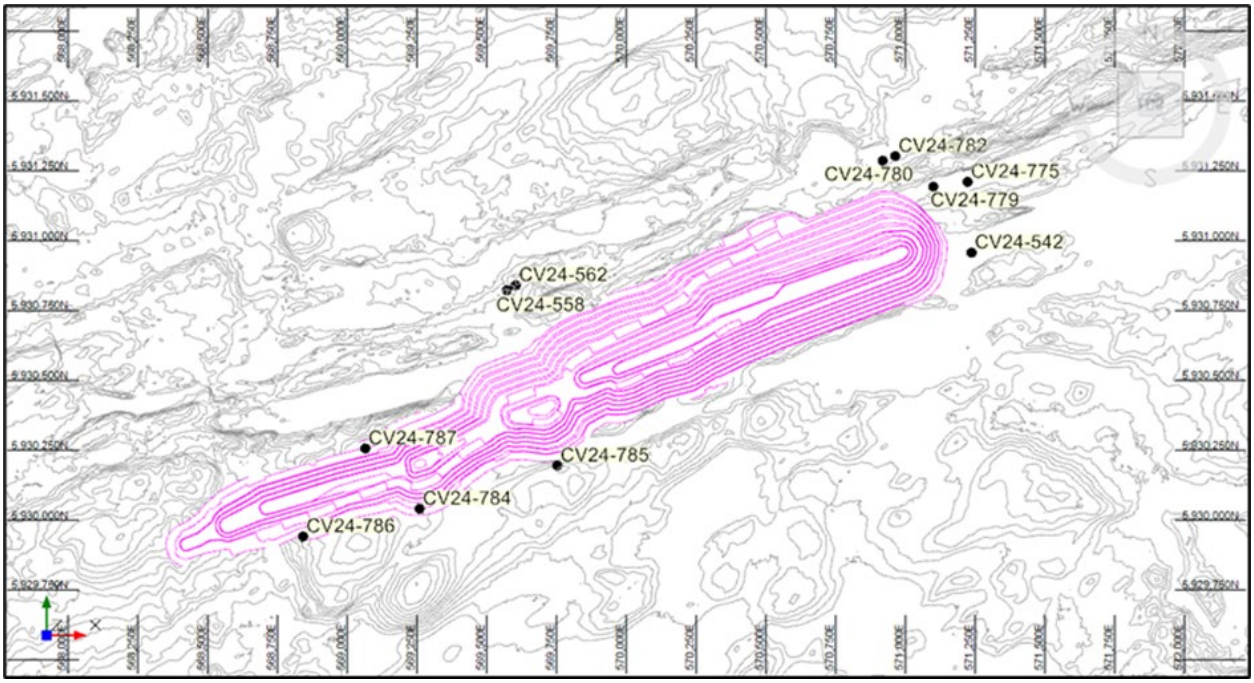
Alius Mine Consulting (Alius) conducted a geotechnical evaluation of the Project that included data from a geotechnical field investigation program, as well as a review of historical data intended to characterize the geotechnical conditions of the residual soils and rock mass in the vicinity of the open pit (OP) slopes and the underground (UG) mine. The various elements of the geotechnical evaluation and findings are presented in the following sections.

16.2.1 Geotechnical Field Campaigns

Geotechnical field campaigns were completed during the summer and autumn of 2024 for soil characterization purposes. The campaign included soil descriptions and classifications (BBA, 2024a) (BBA, 2024b) of drill holes across the Project site for various purposes (e.g., infrastructure, hydrogeology, overburden slopes around the pit), including 11 drill holes located within 250 metres of the open pit footprint.

The location of the 11 geotechnical drill holes are presented in Figure 16.1

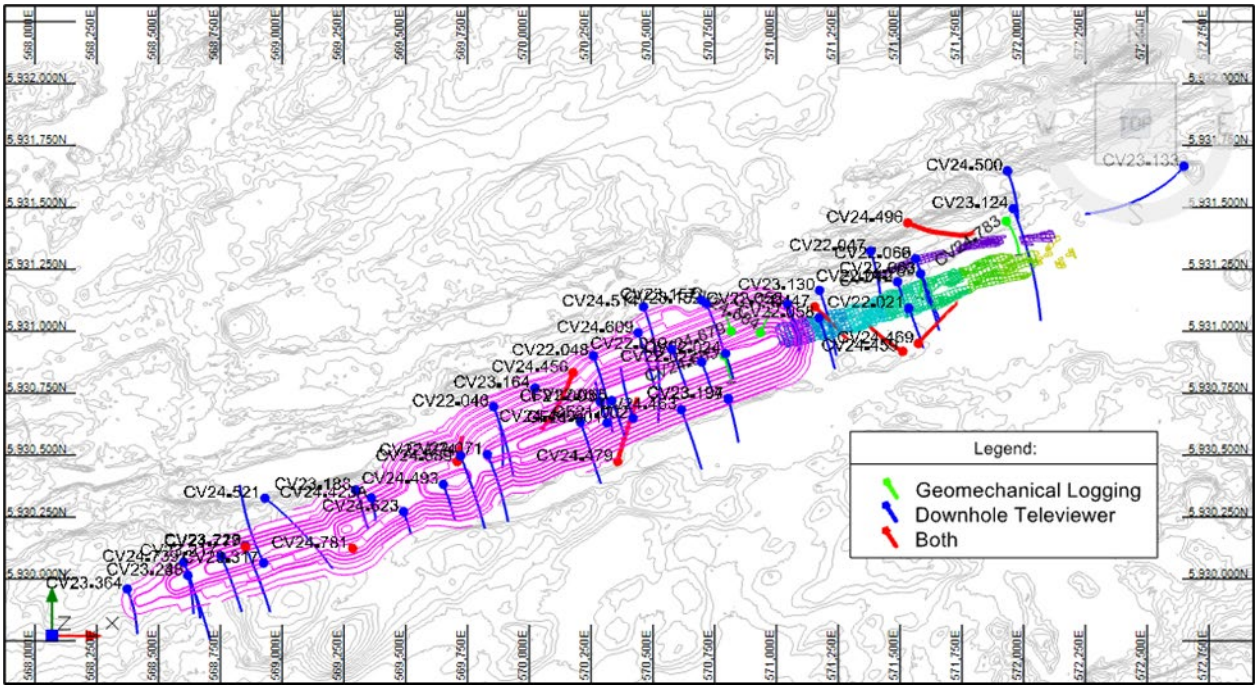
Figure 16.1: Plan View of Open Pit and Relevant Geotechnical (Soil) Drill Holes



Source: Alius, June 2025.
**Note: Not to scale.*

Geomechanical field campaigns were completed by BBA (2024c, 2024d) and Alius Mine Consulting in 2024 to characterize the rock mass. Geomechanical core logging was conducted on 14 drill holes, totalling 3,575 m. Downhole acoustic and/or optical televiewer surveys were used to obtain high-resolution structural imagery directly from the borehole walls of 65 drill holes. Drillholes used in geomechanical analyses are presented in Figure 16.2.

Figure 16.2: Plan View of Open Pit & Underground Mine and Geomechanical Drill Holes



Source: Alius, June 2025.
**Note: Not to scale.*

16.2.2 Soil and Rock Mass Characterization

In the area surrounding the open pit, residual soils are predominantly coarse-grained, with localized thin layers of silt. Overburden thickness is generally less than five (5) metres in the northwestern area of the pit, increasing to more than 20 metres in the southwestern area. Soil resistance was assessed using Standard Penetration Tests (SPT), and soil samples were collected from split tubes and sent to Englobe Corporation for laboratory testing. A total of 31 tests were conducted on the 11 relevant drill holes to determine particle size distribution and water content. The final interpreted strength properties of the main identified material layers, based on SPT results, are summarized in Table 16.1.

Table 16.1: Interpreted Strength Properties of Geotechnical Units

Geotechnical Unit	Interpreted Strength Properties	
	Friction Angle (°)	Cohesion (kPa)
Silt and Sand	28	0
Sand and Gravel	40	0
Till	48	0

Field data collection to characterize the rock mass included intact rock descriptions and discontinuity (bedding, foliation, joints, faults and shear zones) descriptions to derive GSI, RMR_{76} and Q' classifications. Relevant information included: RQD, fracture frequency, rock strength index and structure orientation (dip and dip direction) as well as point-load strength data. Representative samples of each rock domain were collected from the geomechanical drill holes for laboratory testing on intact rock and discontinuities. A total of 363 laboratory tests were conducted across all campaigns combined, including point-load testing (PLT), Brazilian tensile strength (BTS) tests, uniaxial compressive strength (UCS) tests, triaxial compressive strength (TCS) tests, and direct shear tests (DST) on discontinuities. Table 16.2 presents the intact rock strength properties of all relevant lithologies based on all available laboratory tests.

Table 16.2: Summary of Main Lithologies Intact Rock Strength Properties

Lithology	UCS		BTS		TCS		
	N	Average (MPa)	N	Average (MPa)	N	σ_{ci} (MPa)	m_i
Amphibolite	17	171	44	10	22	198	24.2
Metasediments	23	129	36	10	25	153	15.1
Pegmatite	14	70	20	7	18	120	24.7
Ultramafic Undifferentiated	17	63	26	8	14	58	12.3

**Notes*

1. ***N** Number of tests; σ_{ci} intact rock unconfined compressive strength; m_i Hoek-Brown intact rock constant.*

Table 16.3 summarizes the rock mass classifications for all lithological units (combined for the open pit and underground areas). Rock mass classifications were also assessed by lithology, position along strike and depth. The rock mass classifications were ultimately separated between the open pit and the underground area because of minor differences between the two (2) areas. The classifications were incorporated into stope sizing and dilution estimates, as well as ground support considerations (Sections 16.2.5.2 and 16.2.5.4, respectively).

Table 16.3: Rock Mass Classifications

Lithological Unit	Rock Mass Rating			Q' Index			GSI	
	$RMR_{76,P25}$	$RMR_{76,P50}$	Quality	Q'_{P25}	Q'_{P50}	Quality	GSI_{P25}	GSI_{P50}
Amphibolite	61	70	Good	11	18	Good	64	64
Metasediment	52	59	Fair	7	15	Fair – Good	56	64
Pegmatite	50	66	Fair – Good	8	17	Fair – Good	47	65
Ultramafic Undifferentiated	53	60	Fair	6	11	Fair – Good	61	65

**Notes:*
1. *P25 and P50 refer to the 25th and 50th percentiles of the cumulative distribution function.*
2. *Q' = RQD/J_n x J_r/J_a.*
3. *Q' Quality: Assuming J_w/SRF = 1.*

16.2.3 Stress State

As no in-situ stress measurements were conducted on site, stress states have been estimated based on two (2) nearby sites: Éléonore mine and Hydro-Quebec's LG2 Project. The major principal stress (σ_1) is assumed to be horizontal and oriented parallel to the orebody, with a magnitude of 1.48 times the vertical stress component (σ_v), with a locked-in value of 10 MPa. The intermediate principal stress (σ_2) is assumed to be horizontal and perpendicular to the orebody, with a magnitude of 0.7 times σ_v and a locked-in value of 7 MPa. The minor stress (σ_3), defined as the vertical stress, is calculated from the material weight. Table 16.4 summarizes the in-situ stress tensor considered for the Project.

Table 16.4: In-Situ Stress State

Horizontal / Vertical Stress Component	Principal Stress Component	Magnitude Gradient (MPa)	Orientation	
			Trend (°)	Plunge (°)
Horizontal max (σ_H)	Major (σ_1)	0.040 × Z + 10	070	00
Horizontal min (σ_h)	Intermediate (σ_2)	0.019 × Z + 7	160	00
Vertical (σ_v)	Minor (σ_3)	0.027 × Z	--	90

**Note: Z = depth below surface, in metres.*

16.2.4 Open Pit Geotechnical Considerations

The following sections present the methodology used to develop pit slope design guidelines.

16.2.4.1 Open Pit Geotechnical Domains

Open pit geotechnical domains are based on rock mass classifications, structural orientations (interpreted from stereographic Projections), laboratory-derived strength envelopes, and expected rock mass behaviour on mine design.

For the Project, which extends over 4 km, variations in structural context were identified along the strike within the same units. Differences were also noted for a lithological unit occurring in the footwall and in the hanging wall of a specific sector. For this reason, the geotechnical domains of the open pit are named as a concatenation of the pit area (i.e., West, Central, East), the Wall location (i.e., North, South, East, West) and the lithological unit. The domains are detailed in Table 16.5.

Table 16.5: Open Pit Geotechnical Domains

Geotechnical Category	Geotechnical Domain	Area - Location (wall)	Lithological Units
Overburden	OVB	All	Overburden
Rock	WAll-Amp	West – All Walls	AMP
	WW-APU	West – West Wall	AMP, PEG & UMU
	CN-Amp	Central – North Wall	AMP
	CN-Meta	Central – North Wall	META
	CS-Amp	Central – South Wall	AMP
	CS-Meta	Central – South wall	META
	EAll-Amp	East – All Walls	AMP
	EN-UMU	East – North Wall	UMU
	ES-Meta	East – South wall	META
	EE-AMPU	East – East Wall	AMP, META, PEG & UMU

**Notes:*
Lithological Units: AMP = amphibolite, META = metasediment, PEG = pegmatite, UMU = Ultramafic Undifferentiated.

Discontinuity sets were determined for each geotechnical domain, based on relevant drill holes. The major discontinuity sets identified were used in the open pit slope analyses, presented in Table 16.6. Some units presented additional joint sets, but these were considered to be minor and are thus not presented.

Table 16.6: Major Discontinuity Sets Identified per Geotechnical Domain in Open Pit Mine

Geotechnical Domain	Discontinuity Set 1		Discontinuity Set 2	
	Dip (°)	Dip Direction (°)	Dip (°)	Dip Direction (°)
WAll-Amp	70	157	22	347
WW-APU	72	157	17	2
CN-Amp	59	151	23	333
CN-Meta	49	164	-	-
CS-Amp	52	159	22	336
CS-Meta	55	152	21	356
EAll-Amp	45	159	23	357
EN-UMU	58	152	29	162
ES-Meta	55	151	29	342
EE-AMPU	47	160	23	349

**Notes: Values represent average pole.*

16.2.4.2 Slope Stability Analyses

For the overburden, slope stability assessments were conducted using 2D limit equilibrium analyses, assuming a frictional material (no cohesion), as well as the implementation of proactive water management measures to limit water pressure above the till layer.

Open pit benches were designed using a risk-based approach to address the limitations of a deterministic approach. By integrating geotechnical logging data with the proposed pit shell, probability of occurrence and the probability of sliding were derived for wall orientations (dip and dip direction) to determine the bench scale probability of failure (PoF). Using estimated failure volumes, catch bench widths (BW) were optimized to an established risk level to contain material spills, where risk was defined as a function of both PoF and proportion of spills exceeding containment capacity. Given that steeper benches typically generate larger failure volumes, narrower BWs may be insufficient for containment. Establishing a risk tolerance threshold enables the optimization of inter-ramp angles (IRA) in a structurally controlled rock mass, by balancing bench face angles (BFA) and bench widths, resulting in a more realistic design, as some failures may be operationally manageable. Proper mining practices and slope geometry, particularly adequate catch bench widths, are required to not compromise workspace safety by effectively containing spills. This risk-based approach for bench-scale analyses is appropriate when rock mass failure is expected to be structurally controlled (i.e., strong rocks), as is the case for the Project.

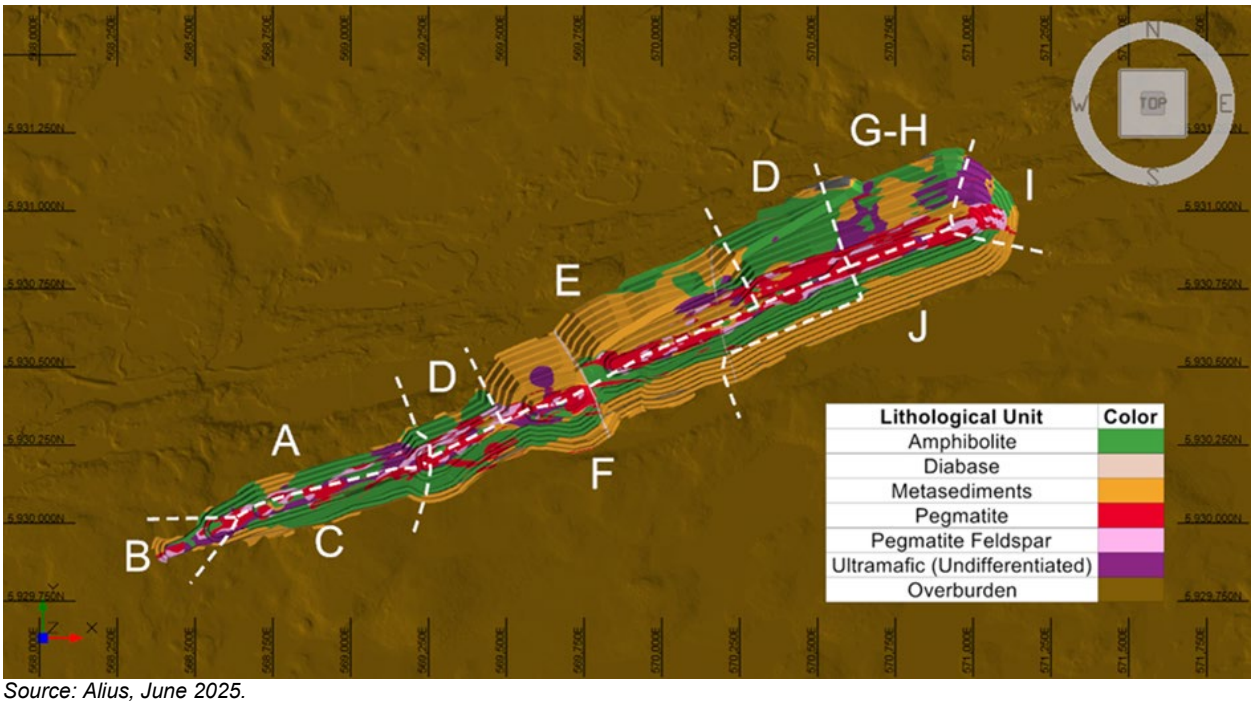
Using the recommended slope design guidelines, overall pit slope stability was assessed using 3D numerical modelling (finite-volume approach). The model incorporated the different lithologies, as well as the multiple pit phases to assess factor of safety, considering fully drained slopes.

16.2.4.3 Slope Design Guidelines

Design guidelines have been established for the various pit sectors, as shown in Figure 16.3. These guidelines include one (1) sector for the overburden (soil) and ten (10) sectors within the rock and are presented in Table 16.7.

Due to the overall excellent quality and strength of the rock, slope stability is expected to be controlled primarily by structural features rather than rock mass strength. Consequently, many domains have lower inter-ramp angles to mitigate risks of planar failures. Additionally, 16 m wide geotechnical berms are recommended in rock slopes at vertical intervals of 160 m, to provide additional design contingency regarding local variation of discontinuity sets.

Figure 16.3: Open Pit Design Sectors



Source: Alius, June 2025.
**Note: Not to scale.*

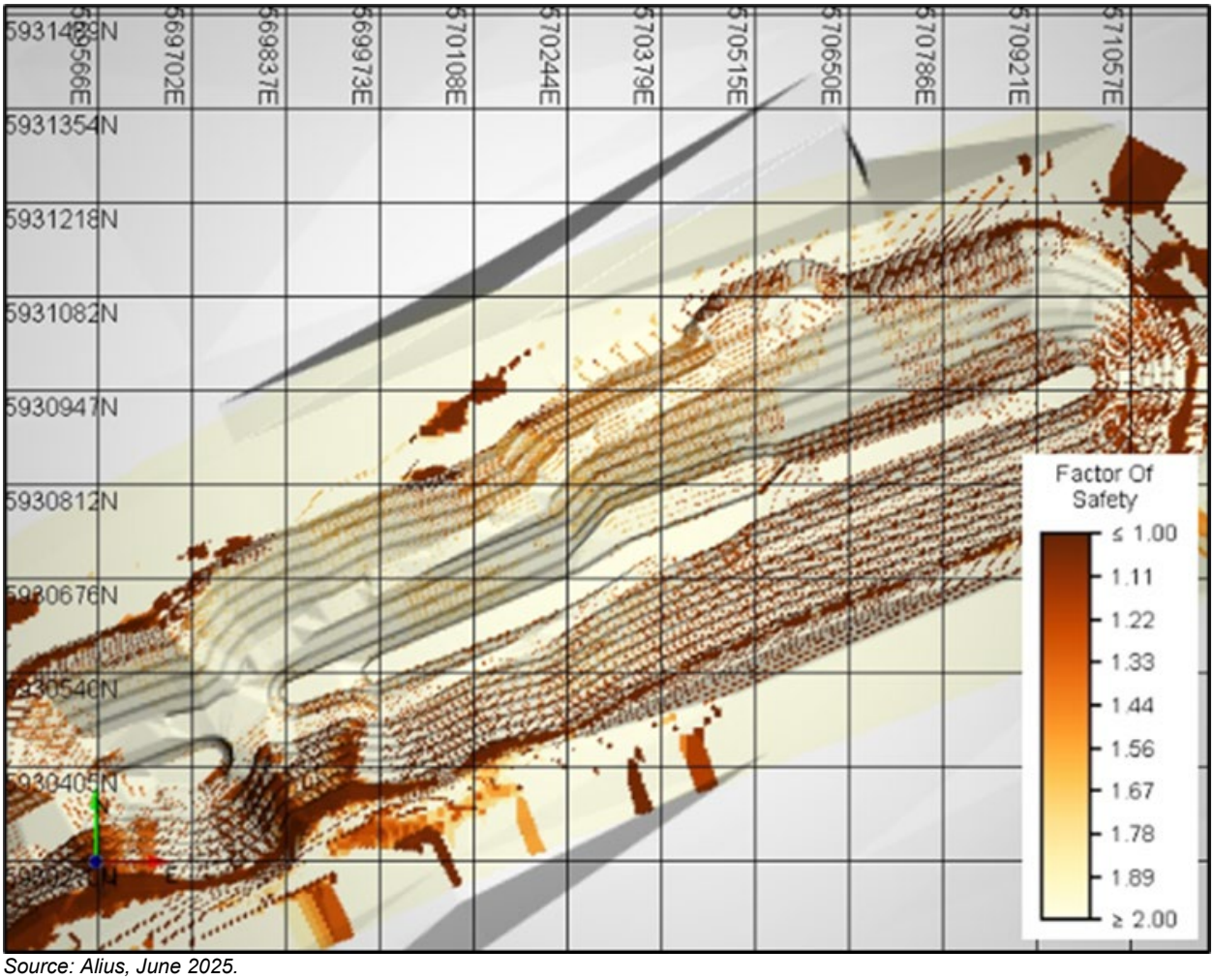
Table 16.7: Guidelines for Open Pit Design Sectors

Pit Sectors	Geotechnical Domains	Azimuth Range (°)	BH (m)	BW (m)	BFA (°)	IRA (°)	Geotechnical Berm
1	OVB	N/A	< 25	0.0	36	36	10 m between overburden and rock
A	WAll-Amp	040-110	20	10.5	68	47	16 m; every 160 m
B	WW-APU	275-040		8.5	75	55	
C	WAll-Amp	250-275		11.0	75	51	
D	CN-Amp	060-075		8.5	59	44	
E	CN-Meta	050-065		8.5	50	38	
F	CS-Amp, CS-Meta	240-260		11.0	75	51	
G	EAll-Amp	075-090		8.5	47	36	
H	EN-UMU	075-090		8.5	40	32	
I	EE-AMPU	090-220		8.5	75	55	
J	EAll-Amp, ES-Meta	220-245		12.0	75	49	

**Notes: BH: bench height; BW: bench width; BFA: bench face angle; IRA: inter-ramp angle.*

The open pit slope design guidelines were assessed through numerical modelling to verify the overall slope stability. Figure 16.4 presents the static safety factors (FoS) distribution Projected onto the final pit walls. All FoS values exceed 1.5 for the overall slope stability assessment. Figure 16.5 presents the anticipated rock mass damage on the final pit walls, with only minor damage observed on the northwestern section of the pit slopes.

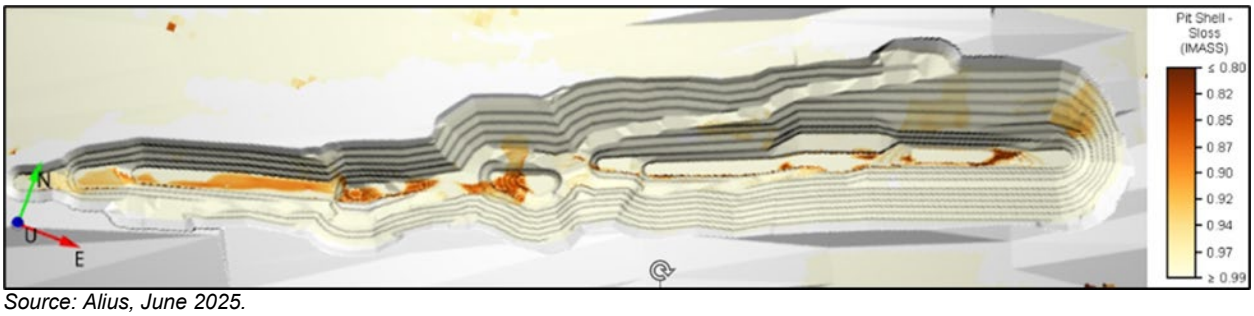
Figure 16.4: Static Factor of Final Pit Slopes' Safety from 3D Model



Source: Alius, June 2025.
**Note: Not to scale.*

It should be noted that the lower factor of safety (FoS) values observed in the figure are superficial artifacts resulting from limitations regarding mapping of results onto the final pit shell design, and do not reflect actual stability concerns within the pit wall.

Figure 16.5. Damage Indicator (Sloss) on Final Pit Slopes from 3D Model



Source: Alius, June 2025.
**Note: Not to scale.*

16.2.5 Underground Geotechnical Considerations

The following sections present the methodology used to develop underground mine design recommendations.

16.2.5.1 Underground Geotechnical Domains

The lower and upper bounds of rock mass classifications, corresponding to the first and second quartiles of the cumulative distribution functions (CDF), were used as input parameters for underground design. Given that multiple lithological units can occur within the hanging walls and footwalls of the stopes, all possible lithological combinations were evaluated. These combinations were subsequently considered in the design optimization process, weighted by their respective occurrence frequencies.

16.2.5.2 Stope Sizing and Dilution Estimates

Stope surface stability was assessed using the empirical stability charts developed by Potvin (1988) and Mawdesley et al. (2001), along with the ELOS method Clark (1998) for dilution estimates. These approaches consider rock mass classification, induced stress conditions, intact rock strength, joint orientations, as well as stope geometry (i.e., hydraulic radius) and orientation.

Analyses were performed for both transverse and longitudinal stopes with the following scenarios of material on walls:

- Back: Pegmatite.
- Endwalls: Pegmatite or paste.
- Footwall: Amphibolite, pegmatite, metasediment, ultramafic undifferentiated or paste.
- Hanging wall: Pegmatite, metasediment or ultramafic undifferentiated.

Table 16.8 summarizes the final stope dimensions and dilution estimates for the longitudinal and transverse stopes. Dilution estimates are presented as intervals, as they vary according to the stope configuration (e.g., primary vs. secondary, exposed lithology).

Table 16.8: Stope Dimensions and Dilution Estimates

Stope Dimensions	Longitudinal	Transverse
Vertical Height (m)	30 – 36	30 – 36
Strike Length (m)	17	17
Maximum Width (m)	7	25
ELOS – Hanging wall (m)	0.5 – 1.0	0.5 – 1.0
ELOS – Footwall dilution (m)	0.5	0 – 2.0
ELOS – Endwalls (m)	0 – 0.5 (assumed to be ore)	0 – 0.5 (assumed to be ore)
ELOS – Back (m)	0 (assumed to be ore)	1.0 – 2.0 (assumed to be ore)

16.2.5.3 Backfill

Stopes will be backfilled using cemented paste backfill (CPB) to avoid leaving rib pillars. CPB strengths are required to ensure lateral stability of the exposed surfaces (either endwalls, hanging walls or footwalls), based on the limit equilibrium method proposed by (Mitchell, Olsen, & Smith, 1982), (Li & Aubertin, 2012) and (Belem, Gélinas, & Oke, 2022). A strength of 240 kPa is required for longitudinal stopes, while a strength of 540 kPa is required for transverse stopes. During the first three (3) years of production, a reduction in the required paste backfill strength is proposed to align with typical engineering optimization practices, resulting in lower targeted factors of safety. Therefore, from year three (3) onward, the strength requirements are set at 210 kPa for longitudinal stopes and 440 kPa for transverse stopes.

Although rarely required, when development in backfill is needed to extract sill-level stopes, higher binder content will be necessary to ensure stability of the stope above. For underhand exposures, a strength varying from 1.2 to 1.6 MPa is recommended, depending on the stope size. A 2.0 MPa strength requirement is used for all underground stopes located in the vicinity of the Open pit. Table 16.9 and Table 16.10 present the cemented paste backfill strength requirements for the Project. Portland cement is used as binder in paste backfill.

Backfill barricades will be mostly built with waste material and covered with a 0.05 m (2 inches) thick shotcrete layer to ensure no leaking. Each barricade is estimated to be 310 cubic metres (m^3) of waste material.

Table 16.9: Cemented Paste Backfill Summary of Strength Requirements

Stope Type	Strength Requirements (kPa)	Curing Period to Reach Required Strength (days)	Binder % in Backfill Recipes
Longitudinal Retreated	210-240[2-3]	28[2]	Table 16.10
Longitudinal Retreated – Last Stope (minimum capacity if not possible to do URF)	100	14	1.8
Transverse Primary and Secondary	440-540[2-3]	28[2]	Table 16.10
Transverse Secondary – Last Stope (minimum capacity if not possible to do URF)	100	14	1.8
Pit-Stope Interaction Zone	2,000	56	9.2
Underhand Mining	Stope <15 mW: 1,200 Stope >15 mW: 1,600	56	Stope <15 mW: 6.7 Stope >15 mW: 8.0
Upper Mining[1]	100	14	1.8

**Notes:*

1. *Upper mining: when stopes are mined from a single drift beneath the stope.*
2. *Strength of 200 kPa required after 3 days of curing.*
3. *Refer to Table 16.10.*

Table 16.10: Strength Requirement for Longitudinal & Transversal Stopes Over the Years

Year of Operation	Longitudinal Retreated		Transverse Primary and Secondary	
	Strength Requirements (kPa)	Binder % in Backfill Recipes	Strength Requirements (kPa)	Binder % in Backfill Recipes
1	240	3.0	540	4.7
2	230	2.9	510	4.6
3	220	2.8	480	4.4
4 and forward	210	2.8	440	4.2

16.2.5.4 Ground Support

Permanent development is planned to be supported on a 1.2 × 1.2 m pattern, using 2.4 m resin rebars (back) and 1.5 m friction bolts (walls), with 4” × 4” 6-gauge mesh. 6-metre cable bolts are planned in intersections. Temporary development areas, such as stope draw points and ore sills, are planned to be supported with the same ground support as permanent development, given their extended operational life and proximity to stopes. Detailed ground support guidelines are provided in Table 16.11.

Back support for transverse stopes was evaluated for various widths, considering rock mass classification variability. Stopes less than 10 m wide do not require cablebolting or long support. However, for wider stopes – up to 25 m – support is required with longer cablebolts (up to 8 m). On average, all stope backs should be supported using 8 m cable bolts, installed on a 2 × 2 m pattern. Additionally, long-support provisions are advised for stope brows, with flexibility for use in other areas as needed. Finally, stopes with hanging wall dips of less than 60° should be cablebolted on the hanging wall. Again, detailed ground support guidelines are provided in Table 16.11.

Assessment of ground support for the portal development has not yet been analyzed in detail; however, this will be comprehensively addressed during the detailed engineering phase of the Project.

Table 16.11: Ground Support Guidelines

Excavation Type	Details	Typical Dimensions (m)		Max Span (m)	Back Support		Wall Support	
		Height	Width		Type	Details	Type	Details
Lateral	Development	5.5	5.5	7.5	Primary	RB 2.4 m; 1.2 × 1.2 m	Primary	SS 1.5 m; 1.2 × 1.2 m
					Secondary	SS 1.0 m; 1.2 × 1.2 m	Secondary	SS 1.0 m; 1.2 × 1.2 m
	Wide Development	5.5	8.0	10.0	Primary	RB 2.4 m; 1.2 × 1.2 m	Primary	SS 1.5 m; 1.2 × 1.2 m
					Secondary	RB 3.0 m; 1.2 × 1.2 m	Secondary	SS 1.0 m; 1.2 × 1.2 m
	Backfill Development	5.5	5.5	6.0	Primary	SC; 2 x 2” + WWM + SX 2.1 m; 1.2 x 1.2 m	Primary	SC; 2 x 2” + WWM + SS 1.8 m; 1.2 × 1.2 m (to floor)
Intersections	Typical (3-way)	5.5	5.5	10.0	Secondary	CB 6.0 m; 2.0 × 2.0 m	-	
	Large (4-way)	5.5	5.5	12.0	Secondary	CB 6.0 m; 1.8 × 1.8 m	-	
Vertical	Raise	-	<3.0	3.0	-		Primary	SS 1.8 m; 1.2 × 1.2 m
		-	<5.0	5.0	-		Primary	RB 2.4 m; 1.2 × 1.2 m
							Secondary	SS 1.0 m; 1.2 × 1.2 m
Stopes	Brow	-		Refer to stope guidelines	Secondary	9 × CB 5 m	-	
	Back	-			Secondary	Refer to stope guidelines	-	

**Notes:*

1. ***CB** 15 mm Cablebolt; **RB** Rebar 20 mm (#6); **SC 2x 2"** Two (2) passes of 2-inch shotcrete; **SS** Split Set 35 mm; **SX** 12 t Swellex; **WWM** welded wire mesh.*
2. *If secondary support is recommended, primary support must first be added (using the appropriate excavation type recommendation).For example, a 10-m intersection in the ramp requires back support of RB 2.4 m on a 1.2 × 1.2 m pattern (primary) followed by CB 6.0 m on a 2.0 × 2.0 m pattern (as second pass).*
3. *Lateral development: assumes wall support to 1.5 m from floor, unless otherwise specified.*
4. *Lateral development: assumes 1.5 × 2.7 m galvanized welded wire mesh (WWM), #6 gauge.*
5. *Vertical development: assumes chain-link or WWM.*
6. *Use 10 tonnes dome plates, except for cables (use 17 tonnes plates).*
7. *Raises that are inaccessible to personnel during and after development do not require support (e.g., drop raises or temporary ventilation raises).*

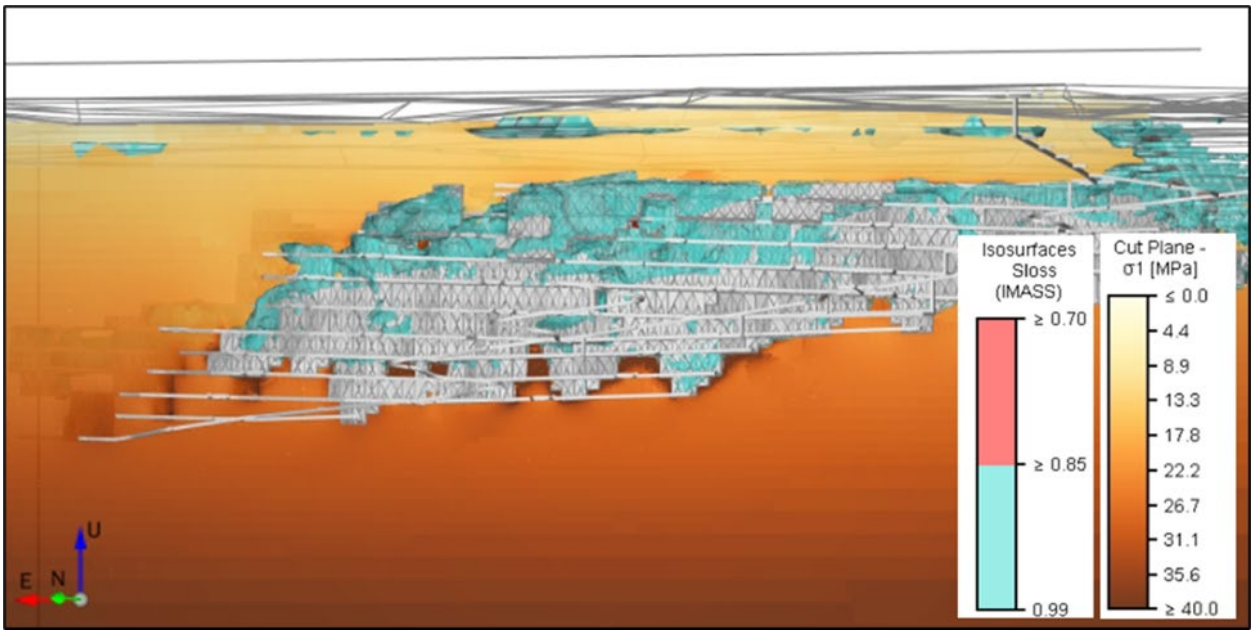
16.2.5.5 Mine Design Considerations

A 3D numerical model (using a finite-volume method) was developed to assess interactions between the preliminary open pit and underground designs and to establish stand-off distance guidelines for key infrastructures. Following completion of the final mine design, another detailed 3D model was constructed to review the mining sequence, identify areas of elevated stress, review the impact on crown pillar and assess potential seismic hazards. Both 3D numerical models were developed using FLAC3D and the IMASS (Itasca Model for Advanced Strain Softening) constitutive model was adopted to represent the rock mass response to stress changes induced by excavation. IMASS accounts for the progressive degradation of the rock mass, from intact rock to fully disaggregated material, while incorporating the effects of dilatancy and bulking during plastic deformation. Key findings are summarized below:

- Stand-Off Distances: Based on the numerical modelling results, a minimum distance of 20 m is recommended between the stopes and the haulage drifts. A minimum distance of 40 m is recommended with raises, and the ramp should maintain a stand-off distance of 100 m from the stopes.
- Sequencing and Stress: The underground operation will involve multiple and simultaneous mining sequences – mainly using a primary-secondary sequence – to maintain productivity and operational flexibility. Stress concentrations are expected on sill levels, in retreat zones between converging fronts, between lenses, and within waste pillars. These areas may experience spalling, local instabilities, and movement along geological discontinuities. Higher stress levels anticipated at depth may also result in fracturing and potential seismicity, although this is considered unlikely. Quantifying the exact stress magnitudes is currently limited by the absence of in-situ stress measurements.
- Seismic Hazard: Seismic hazard was assessed by analyzing seismic potency outputs from the 3D model. Due to the strong and stiff nature of the rock mass, the potential for seismic activity is not anticipated at this stage, although areas of converging mining fronts and areas near waste pillars could experience some deformation. At this stage, the current mine plan does not warrant the implementation of seismic monitoring or dynamic ground support. However, this may need to be revisited depending on the rock mass response as mining progresses.
- Mine Portal: The portal design has not been assessed in detail. However, this will be comprehensively addressed during the detailed engineering phase of the Project.

Figure 16.6 presents the major principal stress (σ_1) and the damage indicator (Sloss) from the 3D model at the end of 2049. The visible damage to isosurfaces on the figure represents the areas experiencing plastic deformations due to high stress concentrations.

Figure 16.6: Major Principal Stress (σ_1) and Damage Indicator (Sloss) Surrounding the Mined Stopes at Year 2049 From 3D Model



Source: Alius, June 2025.
**Note: Not to scale.*

16.2.5.5.1 OP and UG Interaction

An assessment of the stability of the OP-UG interaction was conducted. A 2D limit equilibrium model was developed using inputs from the optimized pit shell and stopes, and the lithological model. The 2D modelling aimed to evaluate slope stability under the assumption that all stopes are entirely backfilled. At most, approximately three (3) stopes in height, equivalent to about 90 m, are expected to be locally exposed on the pit wall. Following the models, a minimum backfill strength of 2.0 MPa is required for approximately six (6) stopes – or 100 m – behind the pit wall. The recommendations derived from the 2D analysis were validated using the 3D numerical model.

16.2.5.5.2 Crown Pillar

Following Quebec's Regulation respecting occupational health and safety in mines Article 77, an underground excavation is under the influence of a body of water when the excavation is less than 100 m (328.1 ft) from the rock contour below the body of water at its highest point. An underground excavation under the influence of a body of water may not begin without plans and specifications prepared by an engineer, accompanied by studies covering the following elements:

- A study of the surface of the site under which the surface pillars will be located.
- A soil distribution study.

- A study of the mechanical properties of the soil.
- A study of the mechanical properties of the rock to be excavated.
- A study of the hydrogeological conditions.
- The data necessary for a surface pillar verification and maintenance system.

Outside the zone of influence of a body of water, the crown pillar design was evaluated using the Scaled Span empirical method (Carter, 2014) combined with 3D numerical modelling. The empirical analysis, incorporating lower and higher bound input parameters, suggests a pillar thickness of 85 m. The numerical models indicate that plastic strain develops in the crown pillar, very late in the mining sequence (years 2046–2049). This deformation occurs after the extraction of stopes located near the top of the mining sequence, directly beneath the water body. Further investigations and refinement, including optimization of the mining sequence, will need to be completed to mitigate this risk.

Recommendations for using an 85 m crown pillar lies on available data and analysis. Specific engineering studies regarding, among others, geotechnical (soil), rock mass strength, structural domaining and hydrogeology will be needed to fully fulfill the “Regulation respecting occupational health and safety in mines Article 77” as the Project progresses towards execution. Furthermore, operational awareness mechanisms such as live feed monitoring and a trigger action response plan (TARP) are additional means by which the company can further ensure safe and sustainable operation, particularly when operations take place within 100 metres of the water body.

16.3 Hydrology and Hydrogeology

16.3.1 Hydrology

The Shaakichiuwaanaan Project site is located in the James and Hudson Bays drainage area (Region 09), within the heart of the Grande Rivière watershed, which extends over 800 km and covers more than 200,000 km^2. This watershed was significantly enlarged by the La Grande Hydroelectric Complex. South of the site, the Pontois River watershed (19,142 km^2) flows into the Grande Rivière about 50 km downstream. Locally, the Shaakichiuwaanaan Project site is scattered with numerous bodies of water, some of which overlap with the planned Project infrastructure. This is particularly true in the case of Lake 001, which will be impacted by the establishment of the necessary dams to mine CV5 Open Pit. More details regarding the dams and the surface hydrology are respectively presented in Sections 18 and 20.

16.3.2 Hydrogeology

Mailloux Hydrogéologie Inc. conducted an evaluation of the hydrogeological conditions for the proposed underground and open pit mine and developed a three-dimensional groundwater flow model. This section provides a summary of their findings.

Overburden in the proposed open-pit and planned underground mine area consists of glacial till (BBA, 2025), made up of sand with a variable proportion of fine particles (silt and clay). According to the drill hole logs prepared by BBA (BBA, 2024b), overburden thickness in the planned pit / UG mines area varies between 0.8 and 20.6 m. Overburden hydraulic conductivity values ranging from 2.9 X 10^{-8} to 3.9 X 10^{-6} m/s were measured in observation wells installed in the overburden. The geometric mean of permeability test values was 1.1 X 10^{-6} m/s (n = 8).

BBA conducted nine (9) geotechnical drillings on the ice covering Lake 001 (BBA, 2025a). Each drill hole was drilled to bedrock, allowing for an assessment of the stratigraphy of the unconsolidated deposits beneath the reservoir at the drilled locations. A layer of silt or clay was observed at the beginning of drilling in eight (8) out of the nine (9) drill holes. The observed thickness of the clay or silt layer ranges from 0.1 to 2.44 m. Beneath this layer, till is present. The total thickness of unconsolidated deposits at the drilling site varies between 0.8 and 8.8 m.

The hydraulic conductivity of the bedrock was assessed using 25 slug tests, 12 packer test profiles, and one pumping test conducted by BBA (BBA, 2025), (BBA, 2024a). The results indicate that hydraulic conductivity values generally decrease with depth and remain low, predominantly in the range of 10^{-8} to 10^{-9} m/s. The highest measured hydraulic conductivity value from packer test is 2.4×10^{-7} m/s. The pumping test was conducted in a 100 m depth well at a pumping rate between 26 and 40 L/min over a period of 6.5 day. At the conclusion of the test, the drawdown at the pumping well reached 76.45 m, with a negative limit (i.e., an increase in the drawdown rate) appearing after 100 hours. Pumping affected the water table only at the adjacent observation well (CV23-174), where the measured drawdown was limited to just 0.3 m. The significant drawdown observed at the pumping well, despite a relatively low average pumping rate of 30 L/min, along with the limited radius of influence and the presence of a negative limit, suggests that the tested rock has low permeability and a minimally developed fracture network. Transmissivity values calculated by BBA (BBA, 2025) from the pumping test data are 1.6 to 8.6 x 10^{-6} m²/sec, while hydraulic conductivity is estimated between 1.6 and 8.6 x 10^{-8} m/sec, considering a pumping well depth of 100 m.

BBA identified six (6) vertical faults near the pits and underground workings. Additionally, three (3) shear zones or faults were recorded in the Project area within the Sigeom database, one (1) of which is located

north of the pits. Five (5) hydraulic tests conducted at or near interpreted faults all indicated low hydraulic conductivity values (10^{-7} m/s or lower), consistent with observations across the site. To take a conservative approach for groundwater inflow estimation, despite field test results suggesting low permeability, the hydraulic conductivity of these faults was assumed to be ten times higher than the highest value measured during packer testing.

Groundwater infiltration into the proposed open pit and underground mine was assessed using a 3D numerical model representing the site's groundwater flow system, based on the conceptual framework outlined in this report. The model was calibrated under steady-state conditions to reflect current groundwater flow by comparing simulated groundwater levels with measured values from 2024. Groundwater inflow into the pit and underground workings was estimated considering the following conservative assumptions: These include assigning the hydraulic conductivity of faults a value ten times greater than the highest measured during packer testing, as previously discussed, and increasing the vertical hydraulic conductivity of the crown pillar by a factor of 10 to account for the possible formation of vertical fractures in this specific area.

Table 16.12 presents groundwater inflow results for the proposed pit and underground mine over different time periods. Groundwater inflow into the pit gradually increases over time, reaching 3,200 m³/day by the Project's completion in 2049. A similar trend is observed for the underground mine, where groundwater inflow reaches 4,900 m³/day in 2049.

Table 16.12: Groundwater Inflow Calculation – Base Case Scenario

Year	Groundwater Inflow into the Pit (m^3/day)	Groundwater Inflow into the Underground Working (m^3/day)
2032	1,200	2,600
2036	1,800	3,200
2040	2,600	4,300
2044	2,900	4,600
2049	3,200	4,900

16.3.2.1 Crown Pillar

Following Quebec's Regulation respecting occupational health and safety in mines, an underground excavation is under the influence of a body of water where the excavation is less than 100 m (328.1 ft) from the rock contour below the body of water at its highest point. An underground excavation under the influence

of a body of water may not begin without there being obtained from an engineer plans and specifications accompanied by studies covering the hydrogeological conditions.

The groundwater flow model and associated results in this section are based on a crown pillar thickness of 85 metres. As shown in Table 16.12, reducing the crown pillar thickness from 100 metres to 85 metres would still result in a manageable level of groundwater inflow into the underground mine and does not pose a significant risk, even under the conservative assumption regarding the vertical hydraulic conductivity of the crown pillar discussed earlier.

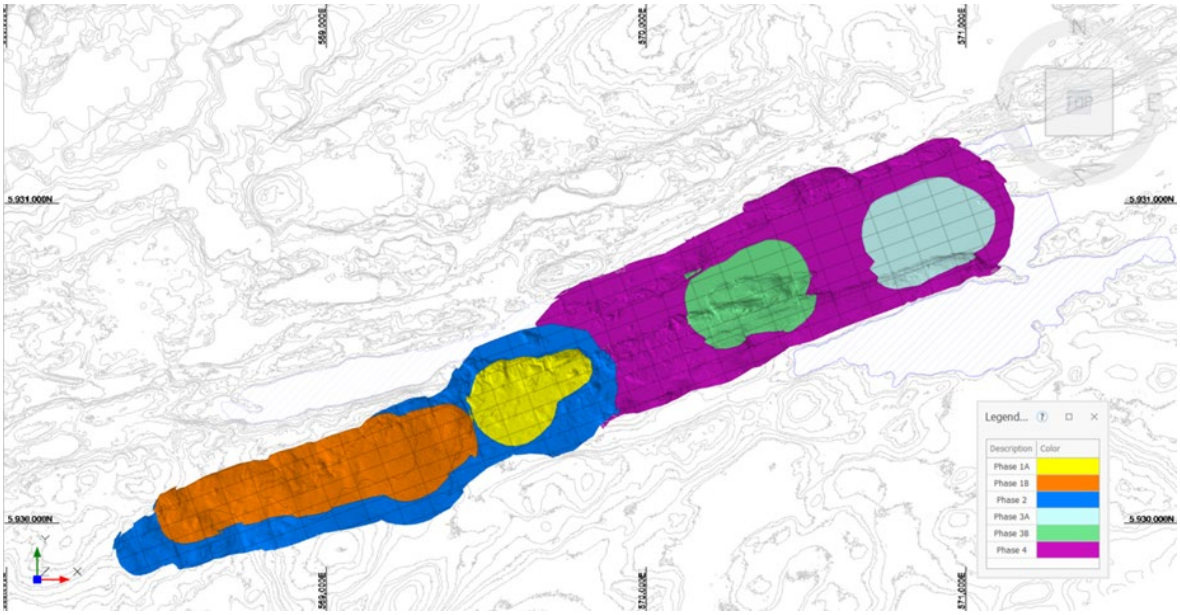
Recommendations for using an 85 m crown pillar lies on available data and analysis. Specific engineering studies regarding, among others, geotechnical (soil), rock mass strength, structural domaining and hydrogeology will be needed to fully fulfill the “Regulation respecting occupational health and safety in mines Article 77” as the Project progresses towards execution. Furthermore, operational awareness mechanisms such as live feed monitoring and a trigger action response plan (TARP) are additional means by which the company can further ensure safe and sustainable operation, particularly when operations take place within 100 metres of the water body.

16.4 Open Pit Mine

16.4.1 Mine Phases

The open pit mine designs were developed using optimal Whittle™ pit shells. The mining of the Shaakichiuwaanaan open pit is planned in six (6) phases. Figure 16.7 presents the final configuration of the pits, while Table 16.13 shows the details of each phase.

Figure 16.7: Open Pit Phase Limits - Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

Table 16.13: Physical Quantities per Open Pit Phase

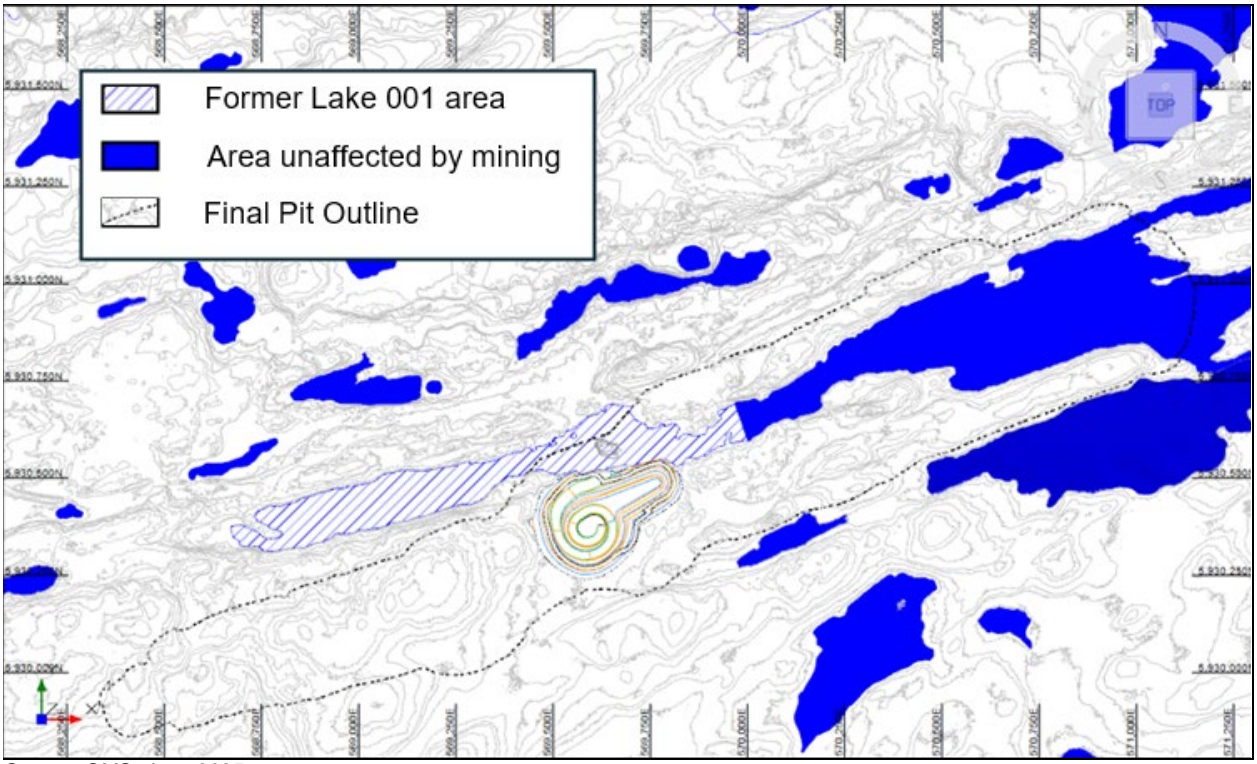
Mining Reserves by Phase	Units	West Area			East Area			TOTAL
		Phase 1A	Phase 1B	Phase 2	Phase 3A	Phase 3B	Phase 4	
Waste Tonnage	kt							
Overburden	kt	1,313	6,546	5,200	1,135	434	2,854	17,481
Rock	kt	1,718	4,643	32,855	4,360	4,832	101,566	149,975
Ore Tonnage	kt							
Ore[2]	kt	1,795	1,181	12,487	2,002	3,603	28,127	49,196
Total tonnage	kt	4,826	12,371	50,542	7,497	8,834	132,547	216,652
Strip Ratio (W:O)	W:O	1.69	9.47	3.05	2.74	1.48	3.71	3.40

**Notes on Physical Quantities:*
1. *Total may not sum due to rounding; rounding followed the recommendations of the NI 43-101.*
2. *Open pit marginal material containing grade above 0.37% Li_2O is also included within this statement.*

The open pit is located south of the process plant and is separated in six (6) different phases. The first two (2) phases, phases 1B and 1A, are focused on gathering waste for road construction and a small stockpile of ore to supply feed during the early months of the process plant ramp-up. Phase 2 is constrained by the location of the first dam. Focusing on the West Area of the open pit allows for more rapid ore production while delaying, as much as possible, impacts on Lake 001. Following Quebec's *Regulation respecting occupational health and safety in mines*, the width of the protective wall is maintained at more than 100 m (328.1 ft) from the nearest body of water at its highest point.

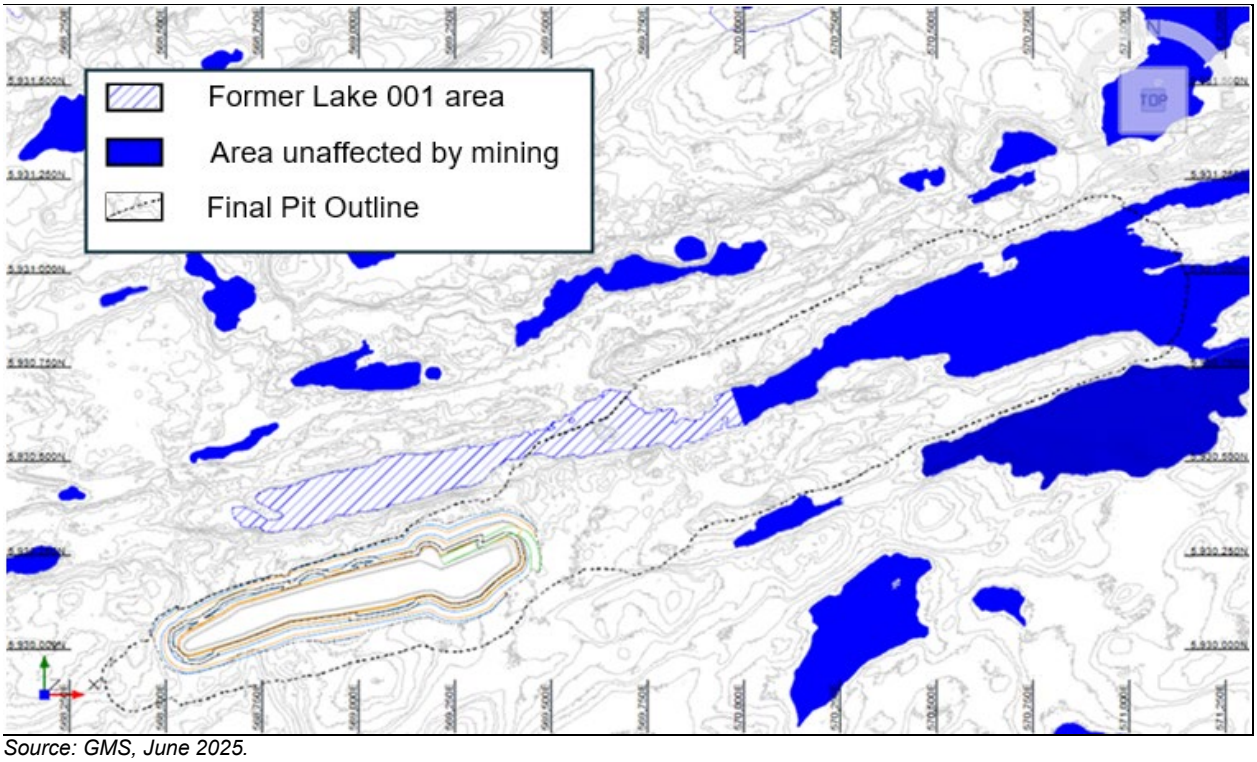
In each phase, the haulage ramp transitions from a double lane to a single lane on the lower benches, optimizing ore recovery at the bottom of the pit. For both areas, open pit exits are planned towards the north to provide the shortest haulage distance to the crusher and waste dumps. The different pit phases are presented in Figure 16.8 to Figure 16.13.

Figure 16.8: Phase 1A Open Pit Design – Plan View



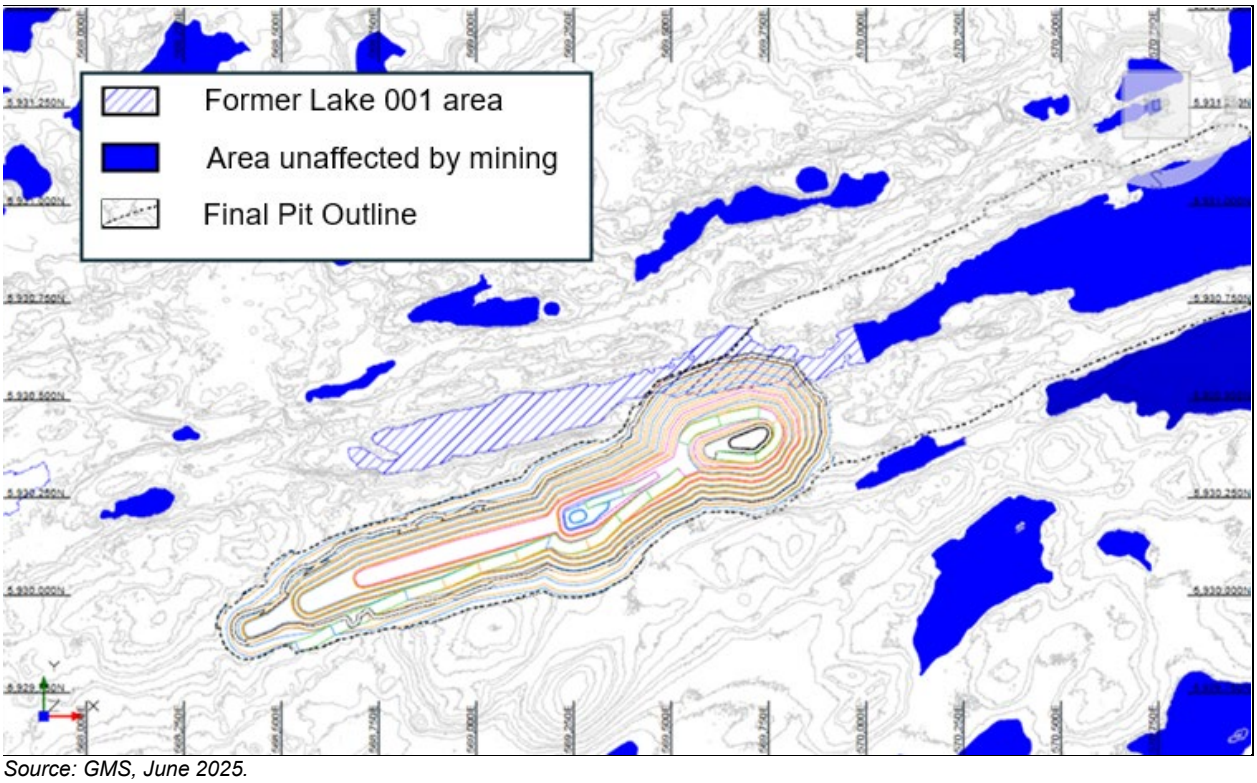
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.9: Phase 1B Open Pit Design -Plan View



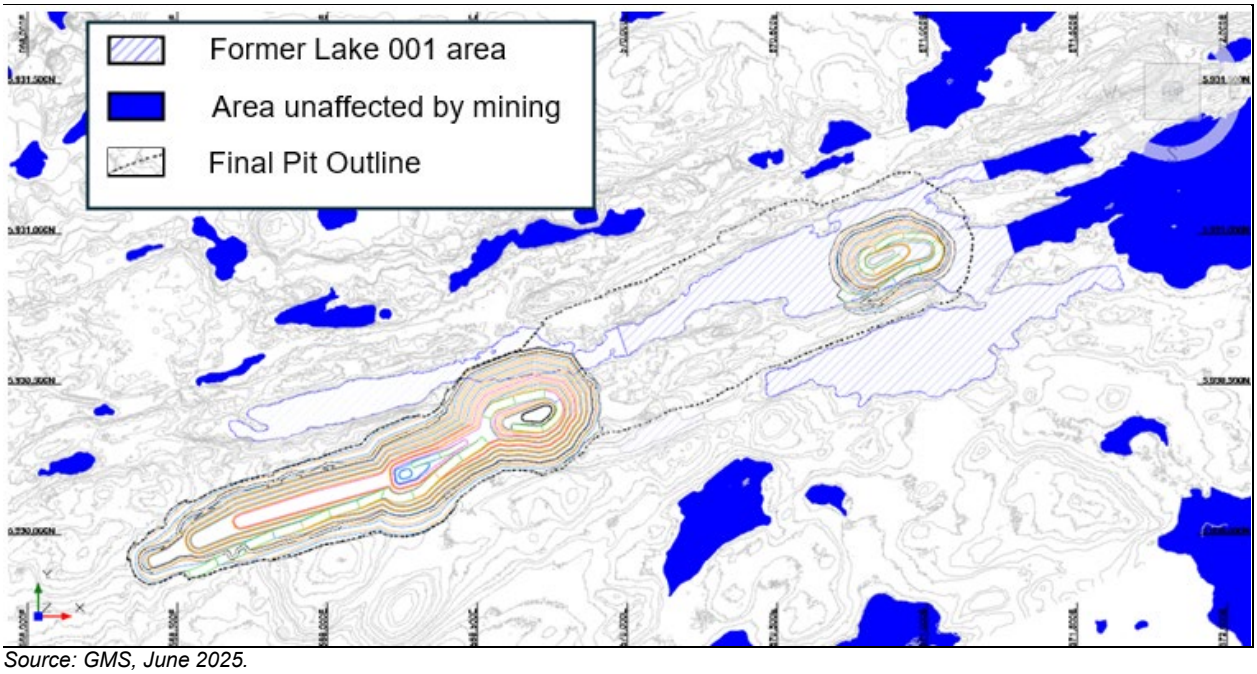
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.10: Phase 2 Open Pit Design – Plan View



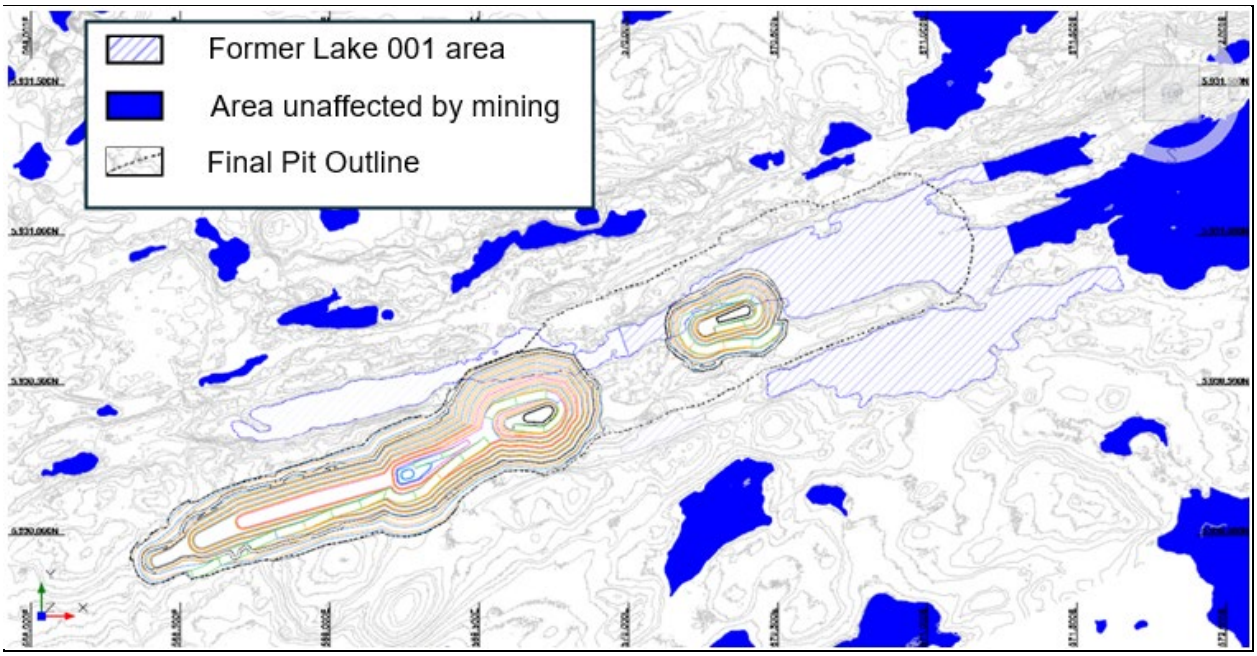
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.11: Phase 3A Open Pit Design – Plan View



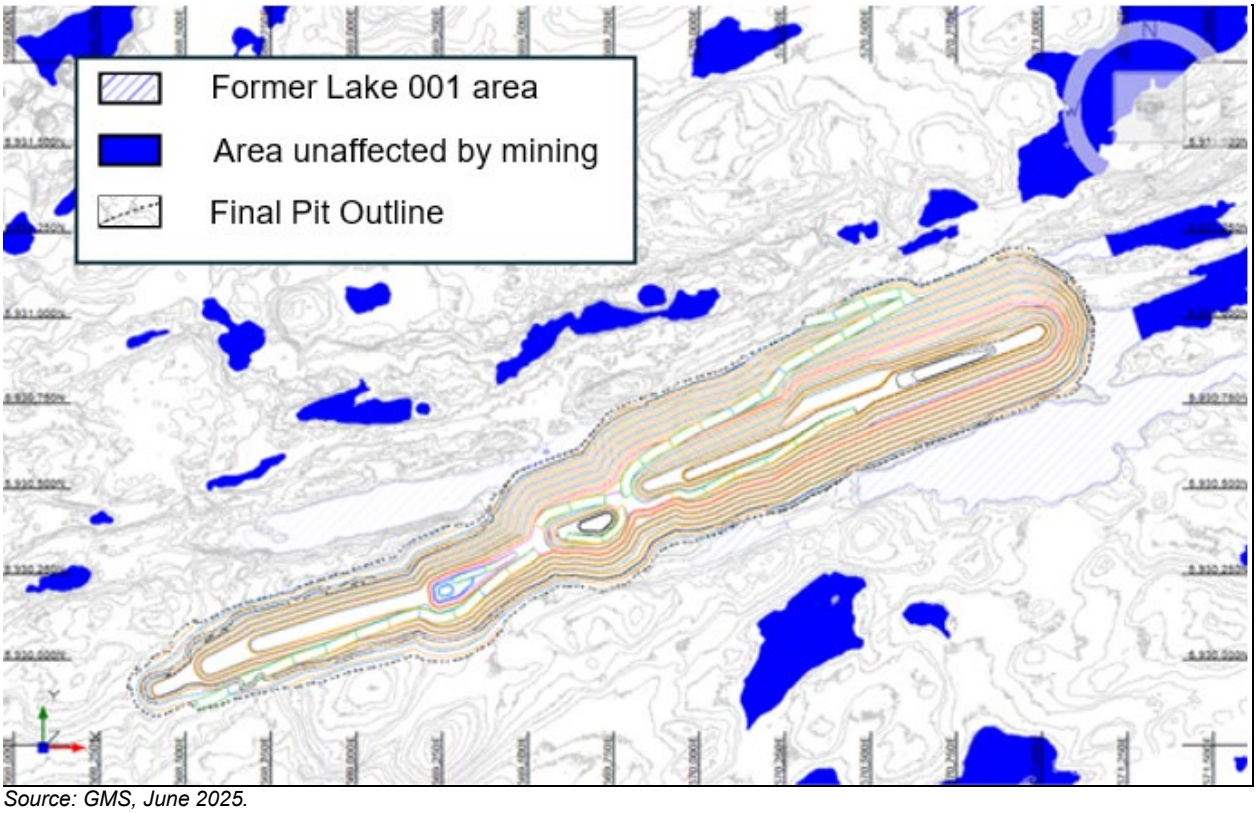
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.12: Phase 3B Open Pit Design – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.13: Phase 4 (Final Open Pit Design) – Plan View



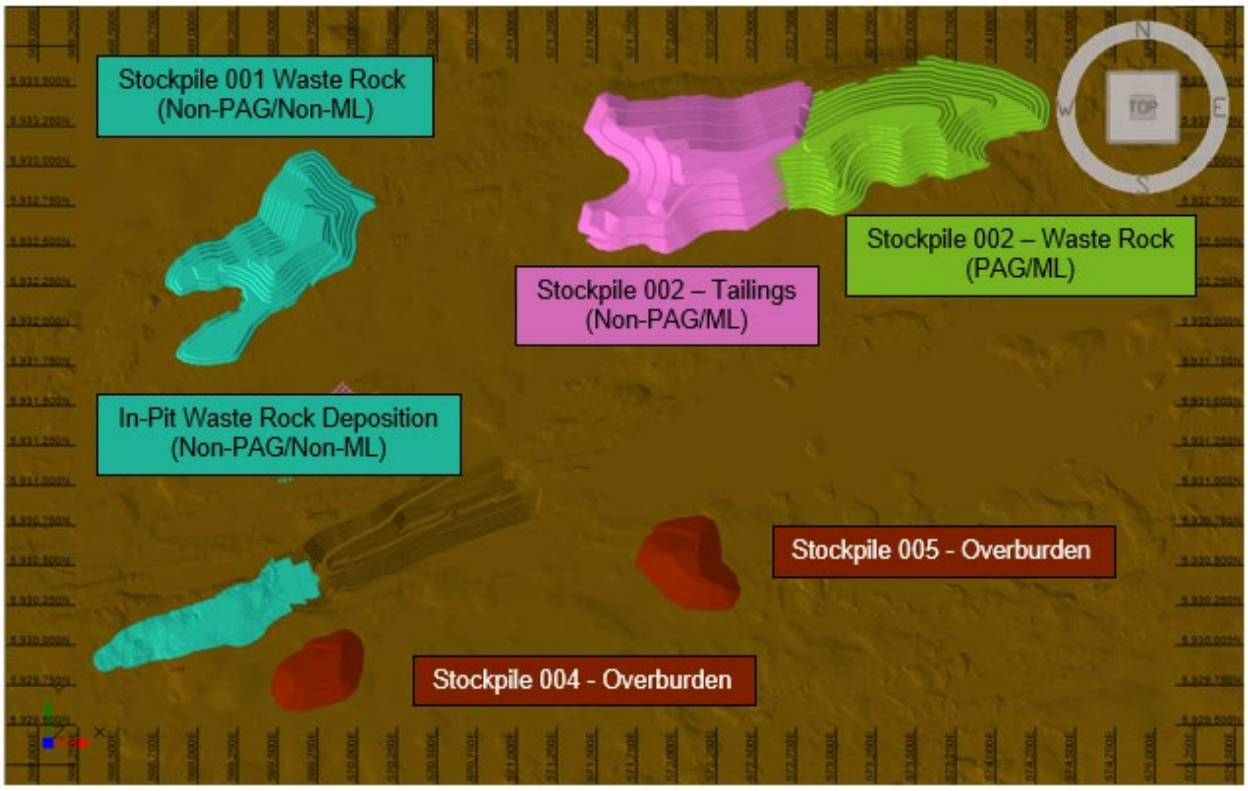
Source: GMS, June 2025.
**Note: Not to scale.*

16.4.2 Waste Rock Storage Facility

A total of 167 Mt of waste rock, including overburden, potentially acid-generating (PAG) waste and non potentially acid-generating waste (NPAG) will be stored in Stockpile 001, 002, 004, 005, and in-pit. Figure 16.14 illustrates the location of the waste storage facility (WSF). Aside overburden, which will be mined using smaller contractor equipment, all waste rock will be hauled to the WSF and dry stacked using 140-t off-highway mining trucks and tracked dozers. Stockpile 002 will be used to store both PAG and tailings material.

Guidelines for the design of the WSF slopes and berm widths were provided by AtkinsRéalis and are detailed in Section 18.

Figure 16.14: Waste Storage Facility – Plan View

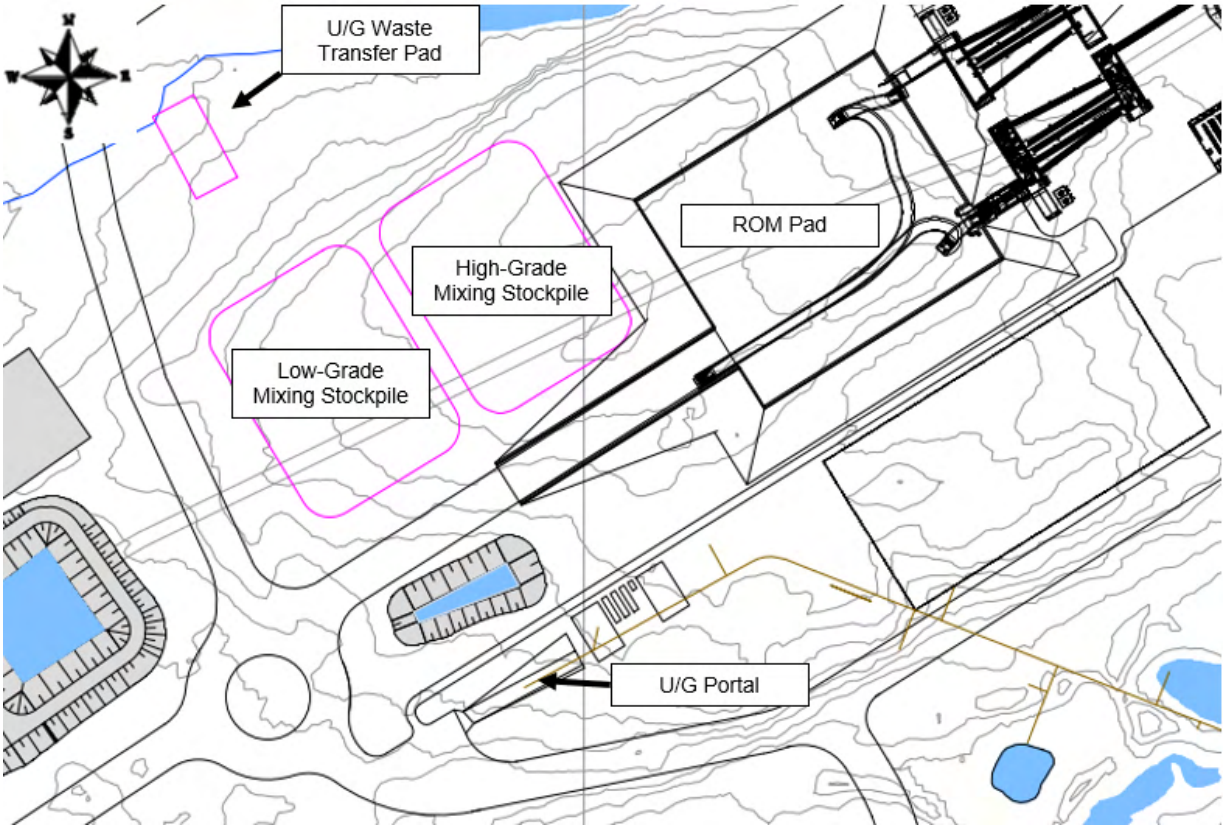


Source: GMS, June 2025.
**Note: Not to scale.*

16.4.3 Ore Stockpiles

Ore material will be stockpiled north of the pit, next to the primary crushers, to ensure a steady flow of ore material to the process plant and to minimize rehandling. Figure 16.15 illustrates the localization of the ore stockpiles, which will have a total capacity of up to 1.5 Mt. The stockpiles will receive ore from both open pit and underground mine. The ore dumping bins are not designed to receive direct dumping from underground haul trucks. Accordingly, underground trucks will dump onto the surface pad, and daily rehandling will be carried out using the mill feed loader.

Figure 16.15: Ore Stockpile Location – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

16.4.4 Mine Haul Roads

The haul roads from the pit to the WSF, the ore stockpile, the crusher and other facilities will primarily be constructed during the pre-production period. Additional haul roads will be built during operations as the pit and the WSF evolve, such as the access to overburden WSF 005. Over the Life-of-mine (LOM), approximately 13.3 km of double lane mine haul roads will be constructed outside of the pit.

16.4.5 Open Pit Production Schedule

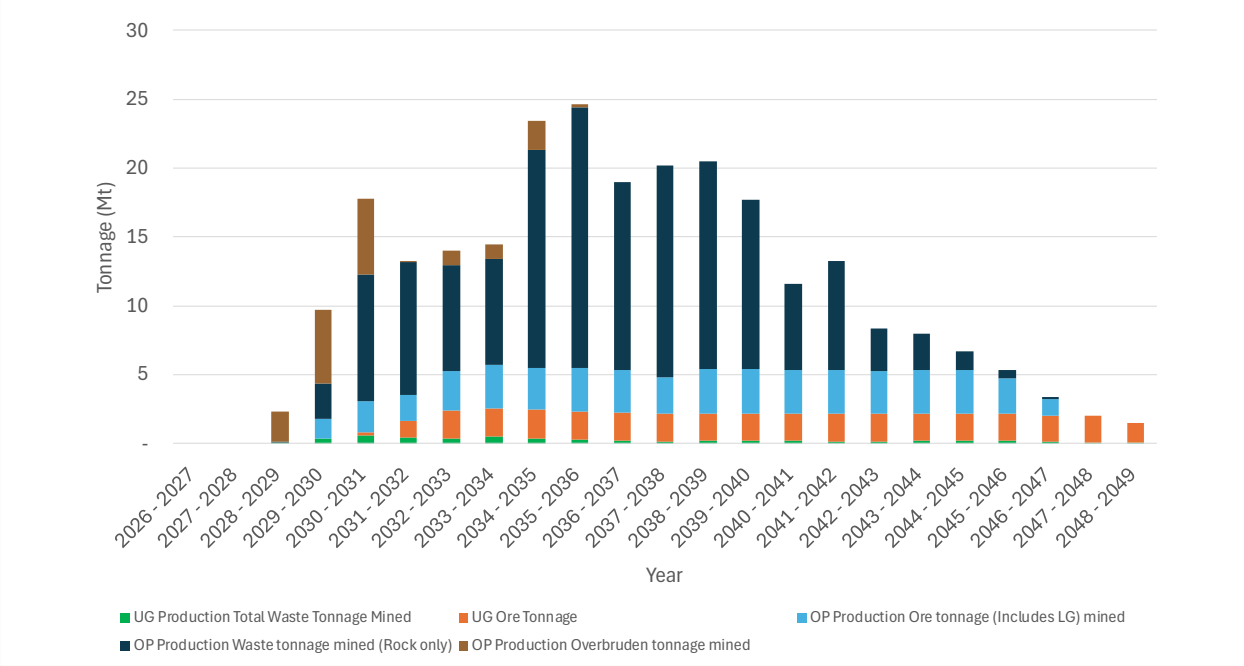
The LOM production schedule for the open pit mine was optimized using Minemax^{tm} Scheduler, an industry leading schedule optimizer powered by the CPLEX solver. Minemax^{tm} Scheduler is an automated mine scheduling tool which leverages multi-period optimization to determine the maximum net present value (NPV) while imposing various physical constraints and targets. The optimization includes mine sequencing, mining rate, stockpiling strategies and rehandling and fleet usage.

The strategic optimal plan generated by Minemax[tm] was then further detailed on monthly, quarterly and annual basis using Deswik[tm] to track material movements, stockpile inventory, process plant feed blending, waste movements, and equipment usage. A monthly schedule was implemented until commercial production to better capture details and opportunity optimizations. The first two (2) years of production were then modelled on a quarterly basis to provide greater detail of early production, as these initial years have the greatest impact on Project economics. The remaining schedule was prepared on a yearly basis. Both the open pit and underground mining schedules were integrated in Minemax[tm] Scheduler, to achieve optimal grade and concentrate distribution.

Open pit mining activities are planned over a duration of 17 years, with two (2) years of preproduction period. The mining rate will ramp up to reach a maximum rate of 23 Mtpa of total material movement in Year 6, after which it will decrease. Figure 16.16 presents the mining schedule by material type, excluding stockpile reclaim movement. Following Project's construction schedule, the production schedule is reported according to a production year that begins on July 1 and ends on June 30 of each year. Variations in the quantity of open pit tonnage mined are attributable to multiple factors:

- The maximum production of dry spodumene concentrate is set to 801,060 t per year basis, based on both open pit and underground tonnage contribution.
- A targeted blended head grade between 1.2% Li_2O and 1.4% Li_2O is maintained to ensure optimal process recovery.
- The contribution of the underground mine production to the process plant feed is variable, requiring the open pit schedule to adapt. Process plant throughput at peak capacity is capped at 5.1 Mtpa considering both open pit and underground tonnage contribution.

Figure 16.16: Open Pit and Underground Mined Tonnes

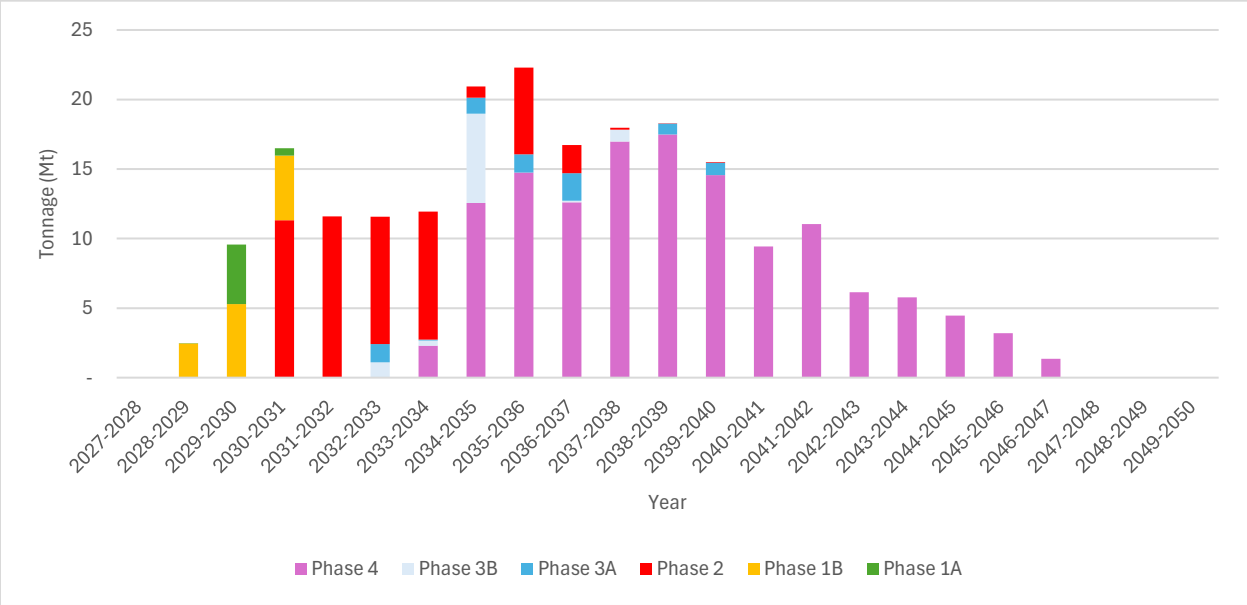


Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.17 presents the LOM tonnage mined by phase. In any given year up to four (4) phases may be mined simultaneously. The mine plan maintains a maximum sinking rate of 60 m per year per phase.

Details of mine production, including mined grades and material movement, are presented in Table 16.14. Figure 16.18 to Figure 16.24 illustrate the progression of the pit and waste storage facility over the Life of Mine (LOM).

Figure 16.17: Open Pit Mining Schedule by Phase (without stockpile reclaim)

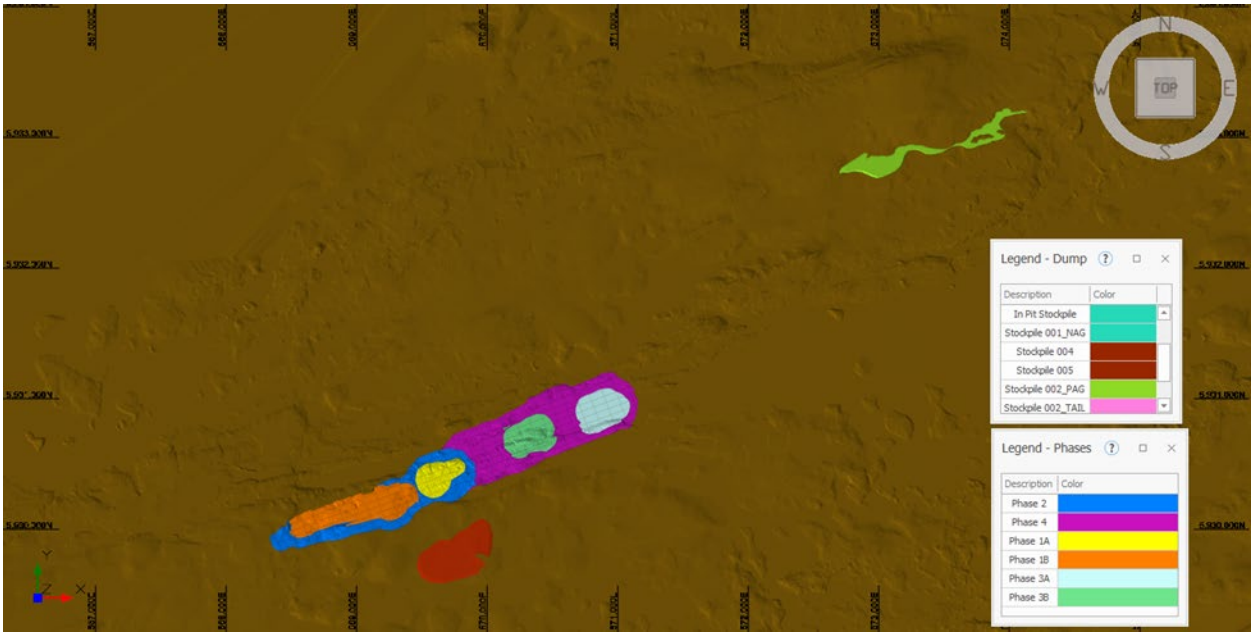


Source: GMS, June 2025.
**Note: Not to scale.*

Table 16.14: Open Pit Mining Schedule Summary

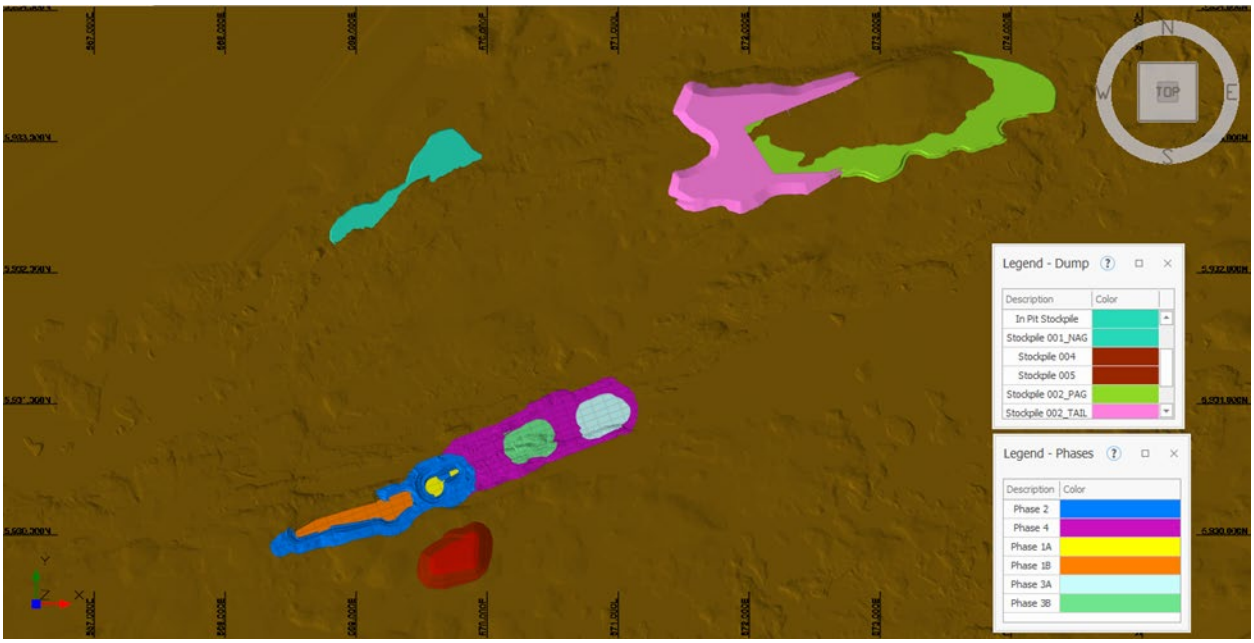
Description	Unit	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048
OP Waste	kt	167,456	-	2,210	7,933	14,725	9,702	8,746	8,754	17,908	19,131	13,659	15,312	15,115	12,335	6,252	7,912	3,045	2,628	1,354	586	149	-
Overburden Waste	kt	17,481	-	2,199	5,364	5,491	5	1,039	1,074	2,081	228	-	-	-	-	-	-	-	-	-	-	-	-
Rock Waste	kt	149,975	-	11	2,569	9,235	9,697	7,706	7,680	15,826	18,902	13,659	15,312	15,115	12,335	6,252	7,912	3,045	2,628	1,354	586	149	-
OP Ore Tonnage	kt	49,196	-	-	1,412	2,238	1,886	2,826	3,188	3,030	3,161	3,059	2,649	3,184	3,170	3,149	3,136	3,091	3,138	3,118	2,563	1,197	-
OP Ore Grade	%Li_2O	1.12	-	-	1.08	1.07	0.97	1.00	1.07	1.10	1.03	1.11	1.02	1.07	1.06	1.03	1.00	1.15	1.29	1.36	1.41	1.47	-
Strip Ratio (W:O)	W:O	3.4	-	-	5.6	6.6	5.1	3.1	2.7	5.9	6.1	4.5	5.8	4.7	3.9	2.0	2.5	1.0	0.8	0.4	0.2	0.1	-
Total OP Production Tonnage Mined (O, W, OVB)	kt	216,652	-	2,460	9,345	16,963	11,588	11,572	11,942	20,938	22,291	16,718	17,960	18,299	15,505	9,402	11,048	6,135	5,767	4,472	3,149	1,347	-

Figure 16.18: Mine Development End of Dec. 2028– Plan View



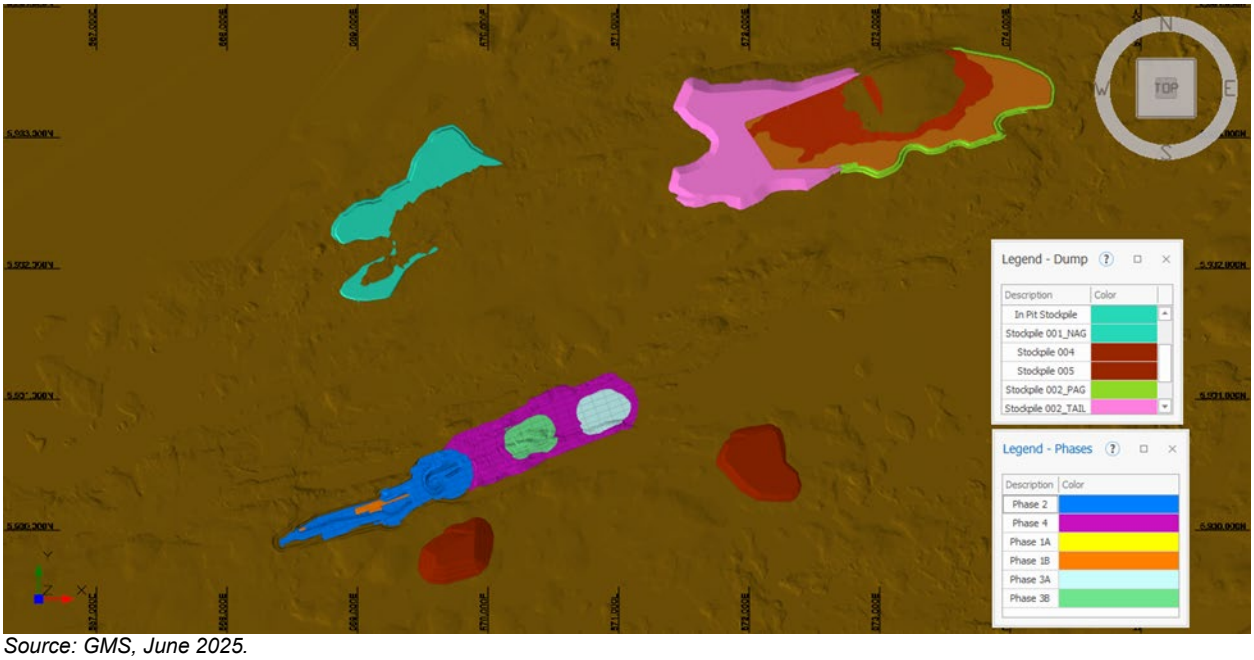
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.19: Mine Development End of June 2030– Plan View



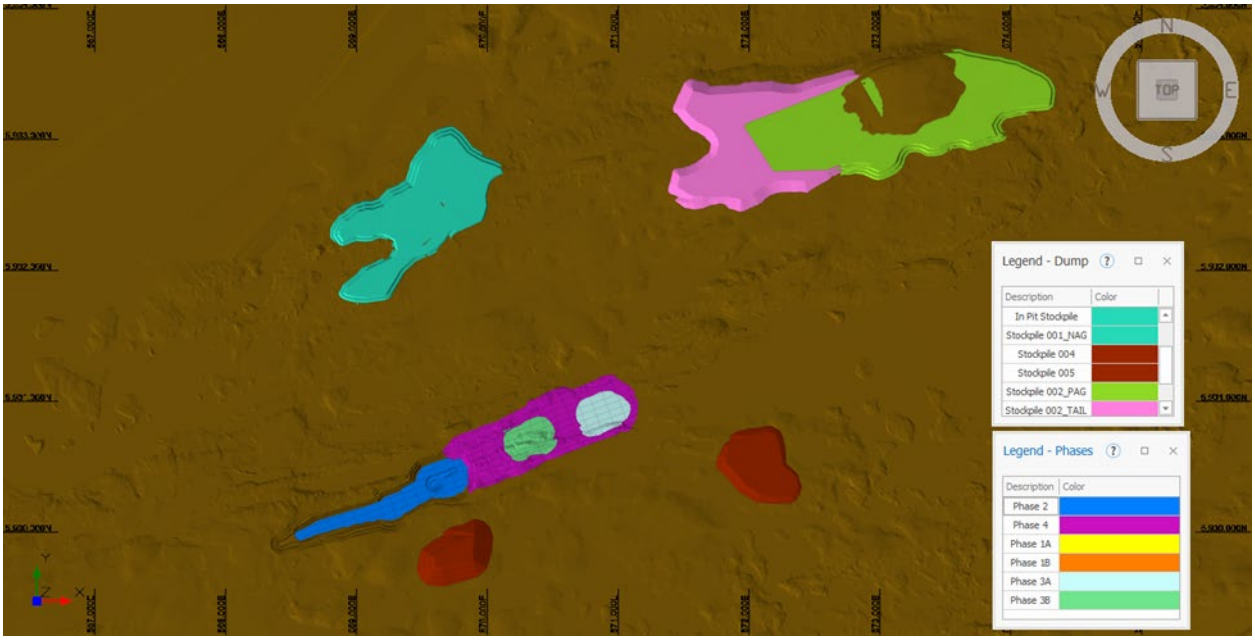
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.20: Mine Development End of Q1 2031– Plan View



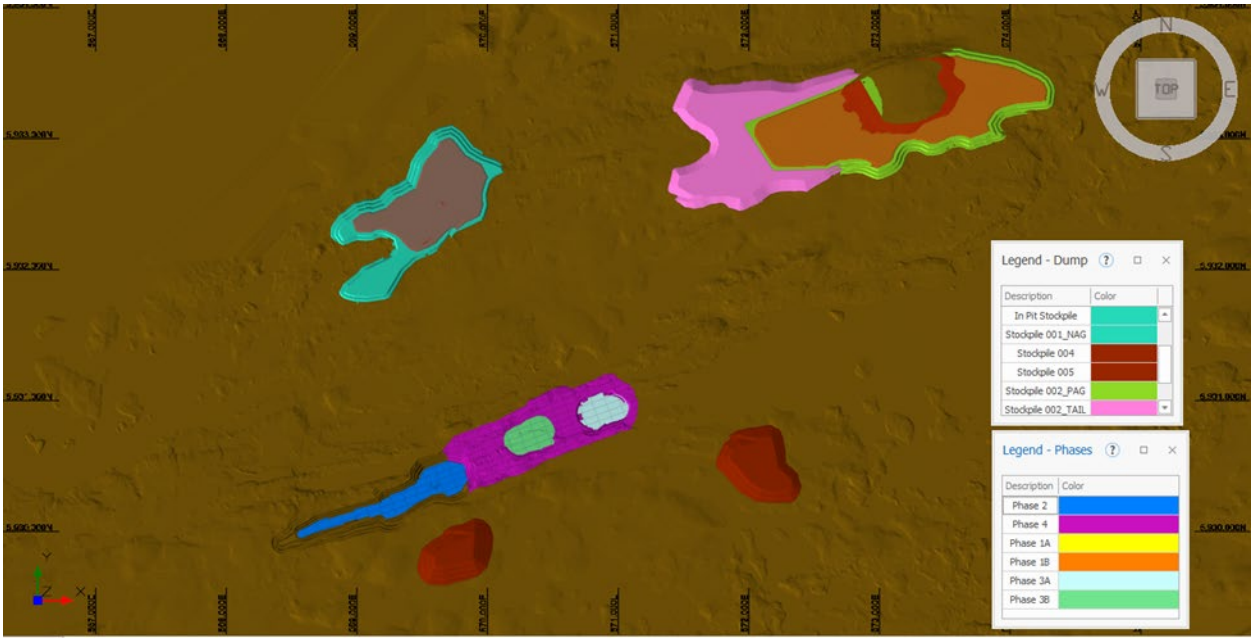
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.21: Mine Development End of Q1 2032 – Plan View



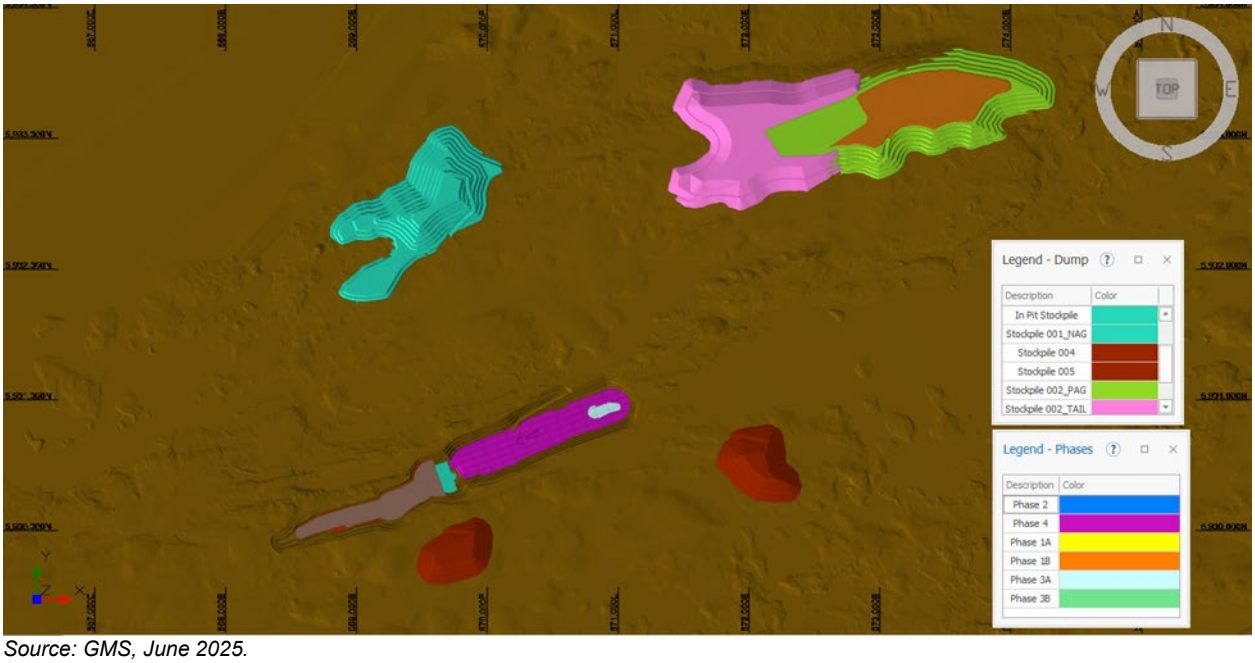
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.22: Mine Development End of Q2 2033 – Plan View



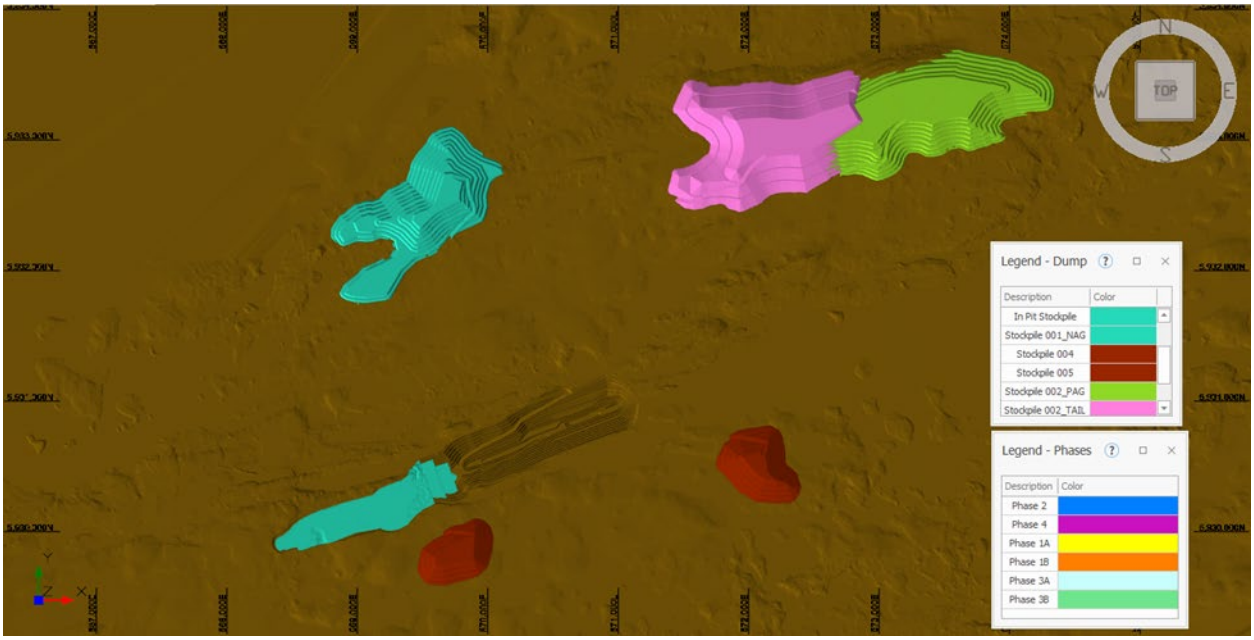
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.23: Mine Development End of Q3 2038 – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.24: Mine Development End of Q3 2047 (LOM) – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

16.4.6 Mine Operations and Equipment Selection

16.4.6.1 Drilling and Blasting

Only one (1) drilling pattern will be used in the open pit regardless of the material type (ore and waste). Being both hard rock, very few differences would be seen in terms of rock fragmentation if a different drilling pattern were used in different material. Single pass DTH drills will be used to ensure optimal productivity and efficiency. Both ore and waste will be drilled using 165 mm (6.5 in.) holes, 10 m high with 1.2 m subdrill. Table 16.15 describes all the different parameters for ore or waste.

Explosives and Blasting services will be supplied by a third-party provider who will be responsible for supplying and delivering explosives with emulsion trucks, load the holes, and initiate the blasts in safe conditions. The third party will provide all the equipment and workforce to operate the explosive plant as well as to carry out loading and blasting services. The owner's mining engineering department will be responsible for designing blast patterns.

In the case of the Shaakichiuwaanaan Project, only bulk or pre-packed emulsion is planned to be used for both development and production. Emulsion is not water-soluble and, compared to ammonium nitrate fuel oil (ANFO), generates very little dissolved ammonium nitrate in the environment when used according to

manufacturer instructions. The manufacture can also modify the emulsion formulae for the best application of rock breakage and dissolved ammonia.

Table 16.15: Drill and Blast Parameters

Drill & Blast Parameters		Ore Fresh Rock	Waste Rock
Drill Pattern		DTH	DTH
Explosive Density	g/cm^3	1.25	1.25
Hole Diameter	in	6.50	6.50
Diameter (D)	m	0.165	0.165
Burden (B)	m	5.00	5.00
Spacing (S)	m	5.00	5.00
Subdrill (J)	m	1.20	1.20
Stemming (T)	m	3.50	3.50
Bench Height (H)	m	10.0	10.0
Blasthole Length (L)	m	11.2	11.2
Pattern Yield			
Rock Density	t/bcm	2.72	2.90
BCM / Hole	bcm/hole	250	250
Yield per Hole	t/hole	680	725
Yield per Metre Drilled	t/m drilled	61	65
Explosive Column (LE)	m	7.70	7.70
Volume of Explosives / Hole	m^3	0.16	0.16
Weight of Explosives / Hole	kg	206.06	206.06
Powder Factor	kg/t	0.30	0.28
Powder Factor	kg/bcm	0.82	0.82

16.4.6.2 Loading

The loading fleet consists of two (2) 15 m^3 diesel-hydraulic shovels, and two (2) 22.9 m^3 diesel front-end loaders. The 15 m^3 shovel will be utilized for both waste and ore material loading, while the front-end loader will be primarily used to rehandle the ore stockpile, waste rehandling, tailings rehandling and to supplement ore or waste loading as needed. Table 16.16 presents the productivity assumption for the loading fleet. The number of units is represented in whole numbers for the purpose of these productivity assumptions.

Table 16.16: Loading Productivity for Ore Material

Loading Unit	Units	Shovel	Front-End Loader
Haulage Unit		**Mining Truck (140 tonnes)**	**Mining Truck (140 tonnes)**
Rated Truck Payload	*t*	139	139
Heaped Tray Column	*m³*	78	78
Bucket Capacity	*m³*	15.0	22.9
Bucket Fill Factor	*%*	90	90
In-situ Dry Density	*t/bcm*	2.72	2.72
Moisture	*%*	2	2
Loading Swell Factor	*%*	40	40
Hauling Swell Factor	*%*	50	50
Wet Loose Density	*t/lcm*	1.98	1.98
Actual Load per Bucket	*t*	26.75	40.91
Passes (decimal)	#	5.20	3.40
Passes (whole)	#	5.00	3.50
Actual Truck Wet Payload	*t*	134	143
Actual Truck Dry Payload	*t*	131	140
Actual Heaped Volume	*m³*	72	77
Payload Capacity	*%*	96	103
Heaped Capacity	*%*	93	99
Cycle Time			
Hauler Exchange	*min*	0.60	0.70
First Bucket Dump	*min*	0.10	0.10
Average Cycle Time	*min*	0.58	0.80
Load Time	*min*	3.02	2.80
Cycle Efficiency with Wait Time	*%*	75	75
Nb. of Trucks Loaded per hr	#	14.90	16.07
Productivity			
Productivity	*t/hr*	1,954	2,256
Effective Hours per Year	*hrs/y*	5,214	5,540
Annual Production Capacity	*kt/yr/unit*	10,188,758	12,499,973

16.4.6.3 Hauling

A trade-off study was completed to assess different haulage alternatives. Different sizes of off-highway mining trucks, conveyors, and in-pit Crusher (IPCC) as well as the utilization of trolley-assisted trucks to move both ore and waste were evaluated. To achieve the yearly targeted tonnages of ore and waste material, a uniform fleet of 140 t class off-highway mining trucks coupled with 15 m^3 diesel hydraulic shovels was selected as optimal pass matches. A market study of available and operational trolley assisted trucks in North America was completed. From discussions with OEMs, trolley-assisted trucks are traditionally available for 250 t-class trucks and larger. The use of larger trucks and a trolley require wider ramps, affecting the stripping ratio of the pit and necessitating higher up-front investments. The additional electricity needed from trolley-assisted trucks was also considered in the trade-off study.

In-pit crushers (IPCC) are strategically placed crushers installed for a long period in open pits which allow the conveying of rocks to surface. The system is composed of rock crushers and conveyors. Haul trucks dump into the in-pit crushers and the ore and waste are hauled out of the pit via conveyors. IPCC can reduce costs, reduce environmental footprint, and could lead to a smaller haul truck fleet and overall cost reduction. However, they are highly CAPEX-intensive, as crushers and conveyors need to be bought and installed as quickly as possible. The system is also less flexible than haul trucks as the management of potentially acid-generating (PAG) waste and non-potentially acid-generating waste (NPAG) as well as moving the system between pit phases 1 and 2 would be very cost and time intensive significantly decreasing several benefits of such a system.

After completing the trade-off study, the overall size of the operation, the pit design and phasing and the required investments to use either trolley-assisted trucks or IPCC justified the use of diesel haul trucks.

Most of the haulage will be performed by 140 t class off-highway mining trucks for both waste and ore material. Articulated trucks will be used by contractors for overburden removal, upper benches and difficult access roads. Ore material will be hauled to the crusher located outside of the pit, while waste will be hauled to the WSF.

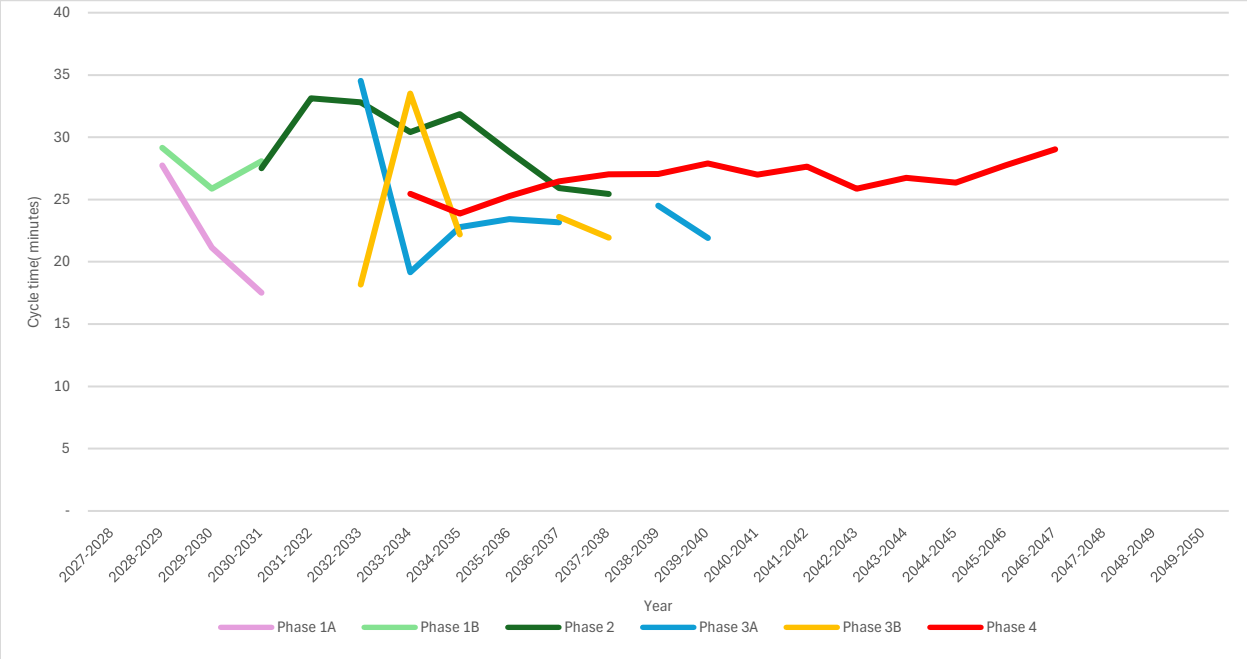
The truck requirements were calculated using Deswik^{tm} LHS (Landform and Haulage) software. This software links the mining schedule to the material movements, determines optimal haulage routes and simulates haulage times using rim pull data from the fleet. The following assumptions were applied when running the simulations.

- Speed limits of 50 km/h on roads and 30 km/h on benches.
- Speed limits of 30 km/h when loaded or going downhill.

- Average rolling resistance of 2%.
- Average loading time of three (3) minutes and dumping time of two (2) minutes Double-lane haul roads and two (2) one-way bridges.

Figure 16.25 illustrate the average cycle times for ore and waste material over the LOM. Cycle times generally increase as the pit deepens due to the longer uphill hauls required. However, they are also influenced by the distance the trucks must travel to the waste dump (including overburden). Plateaus or dips are normal as the stockpiling strategy is dynamic and includes in-pit dumping, which is close to the ore body. The cycle times presented include both loading and dumping times.

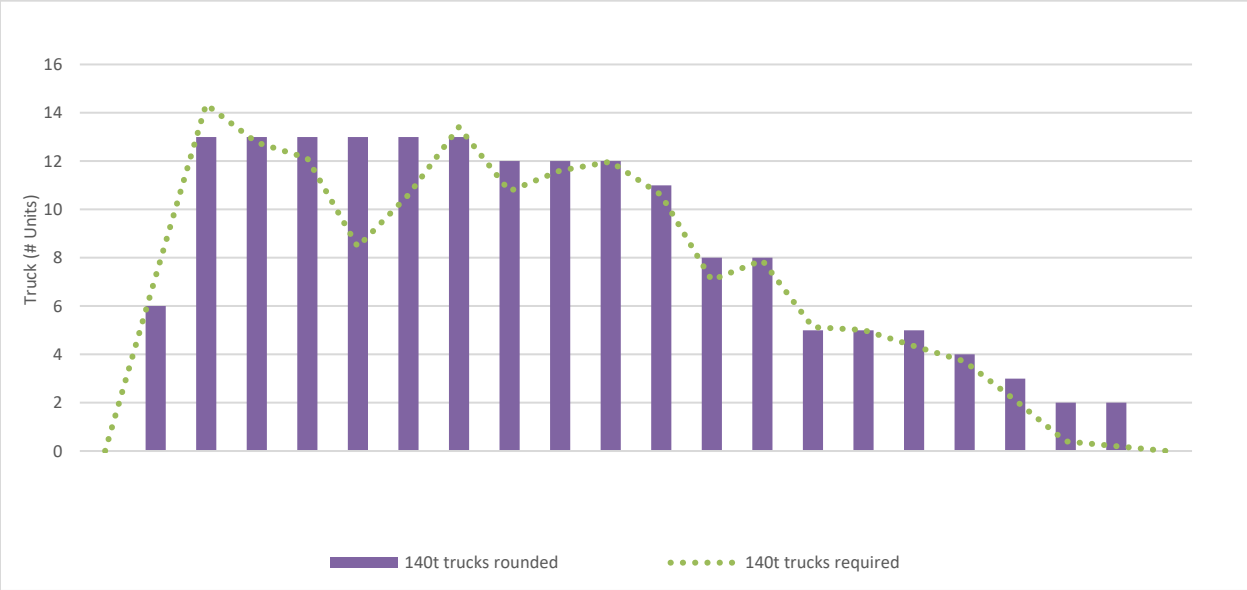
Figure 16.25: Open Pit Average Haulage Cycle Times by Phase



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.26 depicts the total truck requirements over the years. A total of 13 mining trucks will be required to maintain production at the peak mining rate.

Figure 16.26: Open Pit Truck Requirements



Source: GMS, June 2025.
**Note: Not to scale.*

16.4.6.4 Support Operations

Support equipment requirements are based on typical open pit mine operation and maintenance needs to safely support the loading, hauling, and drilling fleets.

Support equipment is planned for maintaining dump areas, stockpiles, pit floors, ditches and mine roads. The fleet of support equipment consists of the following:

- 3 x 600 HP dozers for dump maintenance.
- 2 x 18 ft blade motor graders for road upkeep.
- 1 x articulated water truck for dust suppression.
- 1 x 530 HP wheel dozer.
- Compactors for dry stack tailings.

16.4.6.5 Automation

The automated hauling system (AHS) is a highly specialized system that automates material movement in a mine. AHS usually includes haul trucks equipped with GPS, radar, and LiDAR for navigation and obstacle detection. This system is focused on hauling operations, but all vehicles and personnel on site must be tracked to ensure safe operation. An often-cited argument in favor of AHS is the possibility of decreasing

the number of people on site by replacing operators with an automated system. This could be a significant cost saver, especially in a fly-in fly-out mine where accommodation and transportation costs represent a fair percentage of the all-in personnel costs. However, it has been estimated that more than half of the operators would, in fact, be replaced by automation specialists and overhead support staff. These people are often leased from OEMs at a very high hourly rate, resulting in marginal cost savings.

Besides reducing costs for salaried employees and camp size reduction, an automated haulage system can provide considerable productivity gains in net operating hours (NOH). It has been estimated that when the AHS is fully installed and operating, an increase of up to 12% can be achieved in NOH. This is due to better utilization rates, availability, and effectiveness. For a relatively small, planned fleet of 13 trucks (Figure 16.26), the potential cost savings derived from increasing NOH are outweighed by the additional costs of implementing such systems, including a control room. Furthermore, all support equipment and light vehicles within production areas (open pit, ROM pad, haul roads, stockpiles) would need to be equipped with detection sensors, adding CAPEX and OPEX costs.

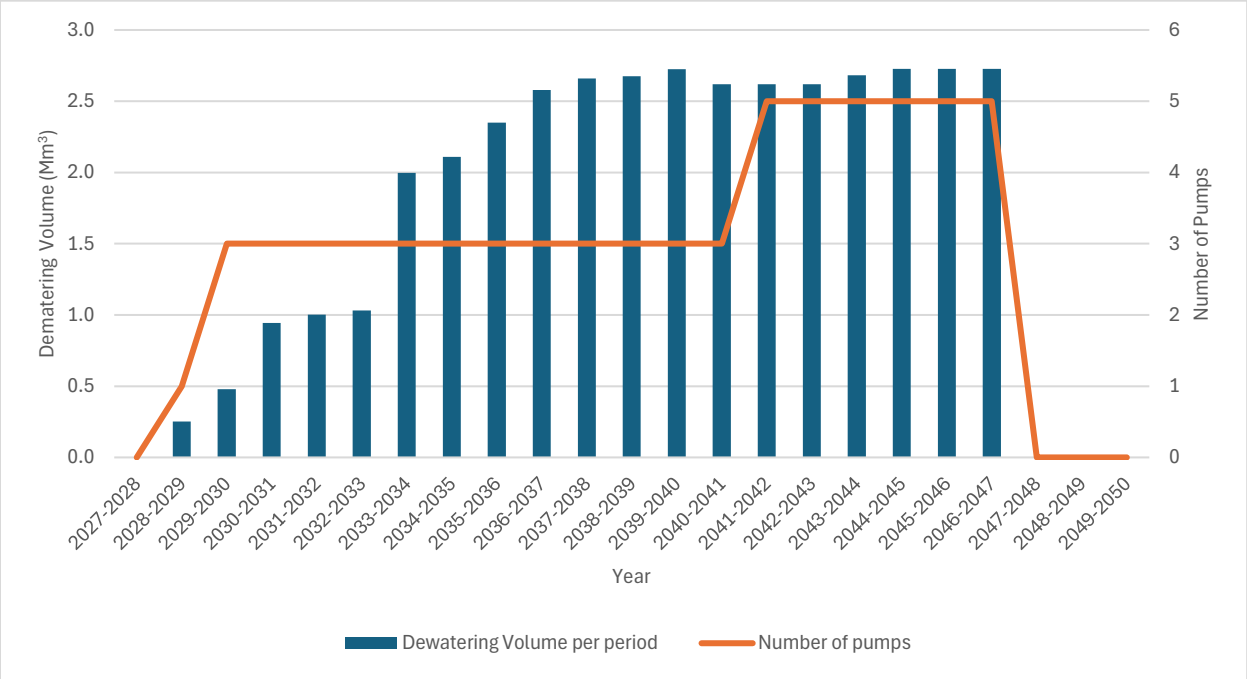
Additionally, the synergy between the underground fleet, the haul roads, and the ROM pad should be considered. This indicates that a portion of the UG fleet would also require detection sensors, resulting in increased CAPEX investments. All designs for haul roads and access routes must be evaluated to prevent unplanned interactions between automated and manual vehicles. Although not impossible, this is considered a significant challenge in such a compact mine site. At this stage, the implementation of an automated hauling system (AHS) is not recommended for the Project.

The automation of the drilling was also considered. However, considering that only two (2) OP drills are planned (Table 16.19), no real benefits were identified by increasing adding automation and increasing NOH. Having only one drill was deemed a risk for the operation. Furthermore, automated drilling often neglects the operational challenges to successfully implement re-drill procedure and casing insertions.

16.4.7 Open Pit Mine Dewatering

Shaakichiuwaanaan is surrounded by lakes and will need dewatering initially and during the production period. Dewatering during the operations will reach a maximum of 2.7 Mm^3 per year of water requiring up to five (5) pumps in series. To compile dewatering needs, runoff water, groundwater inflows and water derived from operations have been accounted for. Dewatering volumes and the quantity of pumps over time are presented in Figure 16.27.

Figure 16.27: Open Pit Dewatering Volumes and Quantity of Pumps Over Time



Source: GMS, June 2025.
**Note: Not to scale.*

16.4.8 Controlled Blasting Techniques and Mitigation of Environment Disturbance

A variety of major infrastructures (dams, lakes, administration buildings, process plant, roads, stockpiles, etc.) are located around the pit, making controlled blasting techniques necessary for the open pit. Three (3) main risk factors have been identified:

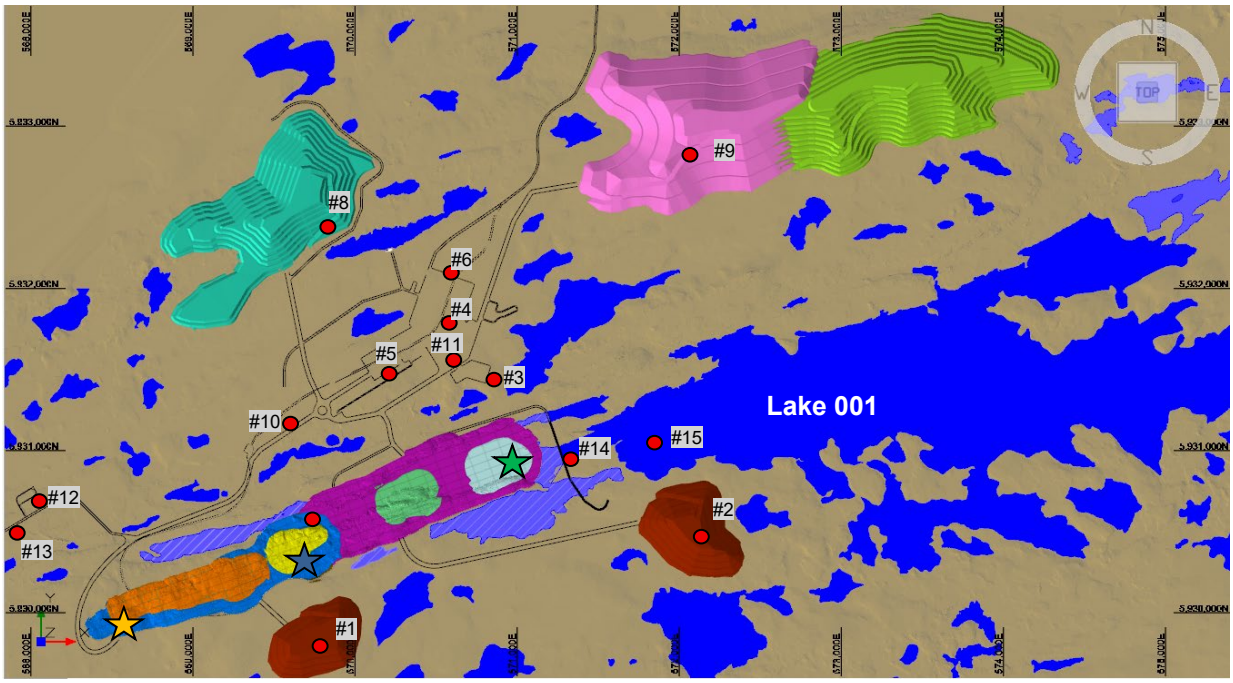
- Fish habitat impacts and limitations.
- Vibration and air blasts impacts.
- Fly rock.

Among controlled blasting techniques, pre-splitting, and blasting mattresses are preferred. Pre-splitting consists of closely spaced holes along the designed excavation limit. These holes are lightly charged and detonated simultaneously, or in groups separated by short delays. Firing the pre-split row creates a fracture along the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves.

Blasting mattresses are another type of controlled blasting technique. They are relatively large mats, made of recycled tires woven with steel cords, which absorbs shocks and prevent fly rocks. These will be used in the eastern part of the pit until the clearing distance is respected.

G Mining Services has performed a risk model considering fish habitat impacts, vibration and air blast impacts, and fly rock at specific points of interest around the pit in relation to three (3) strategic blasts in the pit, as shown in Figure 16.28.

Figure 16.28: Points of Interest – Risk Model



Overburden Stockpile 004	*#1*	*Offices*	*#6*	*Paste Plant*	*#11*
Overburden Stockpile 005	*#2*	*Dam Step 1*	*#7*	*Emulsion Plant/Magazine*	*#12*
Underground Mine Portal	*#3*	*Rock Waste Stockpile 001*	*#8*	*Explosive Mag*	*#13*
Garage	*#4*	*Rock Waste Stockpile 002*	*#9*	*Dam Step 2*	*#14*
Mill	*#5*	*Mine Run off Sedimentation Pond*	*#10*	*Lake 001*	*#15*
Blast Location No.1	★	*Blast Location No. 2*	★	*Blast Location No.3*	★

Source: GMS, June 2025.
**Note: Not to scale.*

16.4.8.1 Fish Habitat Impact

The “Guidelines for the Use of Explosives in or Near Canadian Fisheries Waters” (Hopky, 1998) mention nine (9) guidelines, of which the last three (3) have been evaluated as direct risks affecting the Shaakichiuwaanaan Open Pit mine:

- No explosive is to be knowingly detonated within 500 m of any marine mammal (or no visual contact from an observer using 7x35-power binocular). Note: Upon review of a proposal, the DFO Regional / Area authority may impose a greater avoidance distance, depending on the size of the charge or other Project specific or fishery resource conditions.

- No explosive is to be detonated in or near fish habitat that produces, or is likely to produce, an instantaneous pressure change (i.e., overpressure) greater than 100 kPa (14.5 psi) in the swim bladder of a fish.
- No explosive is to be detonated that produces, or is likely to produce, a peak particle velocity greater than 13 mm/s in a spawning bed during the period of egg incubation.

G Mining Services determined that, based on the proposed blasting patterns and varying rock lithologies, setback distances must be a minimum of 72.5 m to mitigate pressure changes (guideline 8) and 217.5 m to mitigate spawning bed incubation impacts (guideline 9). Distances between each blast location and the points of interest are presented in Table 16.17.

Table 16.17: Points of Interest and Blasting Location Distances

Points #	Points of Interest / Blasting Location	Blast Location No. 1	Blast Location No. 2	Blast Location No. 3
#1	Overburden Stockpile 004	389	1,421	1,362
#2	Overburden Stockpile 005	2,105	3,364	837
#3	Underground Mine Portal	1,085	2,230	808
#4	Garage	1,340	2,454	883
#5	Mill	1,729	2,840	1,014
#6	Offices	1,720	2,755	1,203
#7	Dam Step 1	278	1,526	1,140
#8	Rock Waste Stockpile 001	1,570	2,204	1,830
#9	Rock Waste Stockpile 002	3,068	4,285	1,803
#10	Mine Run off Sedimentation Pond	844	1,570	1,544
#11	Paste Plant	1,196	2,264	994
#12	Emulsion Plant / Magazine	2,176	1,065	3,497
#13	Explosive Mag	2,422	1,255	3,756
#14	Dam Step 2	1,730	3,004	347
#15	Lake 001	1,864	3,140	489

No distances presented in Table 16.17 are within the setback distances, meaning that nowhere during the mine operation would fish be affected by guidelines 8 and 9. Nonetheless, G Mining Services recommends further studies as blasting near any water bodies should be carefully assessed and mitigated to reduce impacts on aquatic wildlife as much as possible. Rock fracturing techniques with low environmental impact should be considered if required.

16.4.8.2 Vibration and Air Blasts

Ground vibration and air blasts are inevitable when using high impact explosives. According to the *Explosives Engineers Guide* produced by Dyno Nobel®, the maximum peak particle velocities for ground vibration should be less than 25 mm/sec for anywhere around industrial buildings and steel constructions. The maximum recommended peak particles velocities should be less than 5 mm/sec for areas such as dams. Airblasts are an airborne shock resulting from explosives detonation. In terms of impacts, the energy waves should be limited to a maximum of 0.7 kPa, resulting in minor damage.

G Mining Services modelled three (3) different situations in terms of confinement: a free face in highly structured rock, a free face in average rocks, and heavily confined rocks such as in box cuts. These three (3) situations have been calculated at the three blast locations and in regard to the points of interest 1 to 15.

According to the results, box-cuts blasting will be a source of disturbance, in terms of ground vibration and air blasts, and will need to be managed by using special blasting techniques.

16.4.8.3 Fly Rock

Fly rock refers to rock fragment that are Projected at greater distances from the blast. This is usually due to poor stemming and unplanned rock fracturing. G Mining Services has evaluated the minimum clearing distance for human and equipment for the given blasting pattern. A factor of safety of 2 is applied to the clearing distance of equipment and of 4 to the personnel's resulting in blasting radiuses of 430 m for equipment and 860 m for the personnel. This means that, without any fly rocks control devices, such as blasting mats, these distances must be respected.

Based on Table 16.17, blasts located in the eastern part of the East pit would need evacuation of personnel and equipment. The risk of fly rock on dams and key infrastructure should be avoided, leading G Mining Services to propose the systematic use of blasting mattress when both the personnel and equipment are in the blasting radiuses.

16.4.9 Mining Fleet Requirements

Table 16.18 summarizes the gross operating hours used to calculate equipment fleet requirements. The mine is expected to operate 20 hours per day for 355 days per year, allowing 10 days of scheduled outages. Additional delays and applied factors are considered in productivity calculations.

Additional equipment will be procured to facilitate maintenance activities and support operation, such as fuel and lube trucks, a forklift, a telehandler, a low-boy trailer and a tractor for moving the tracked equipment. Other small equipment, such as mechanical service trucks, generators and welding machines are also included.

Table 16.19 and Table 16.21 present the equipment purchase schedule for the Life of Mine (LOM).

Table 16.18: Equipment Usage Assumption

Equipment Usage Assumptions		Shovels	Loaders	Trucks	Drills	Pumps
Days in Period	days	365	365	365	365	365
Weather, Schedule Outages	days	10.0	10.0	10.0	10.0	10.0
Shifts per Day	shift/day	2.0	2.0	2.0	2.0	2.0
Hours per Shift	h/shift	12.0	12.0	12.0	12.0	12.0
Availability	%	85.0	85.0	85.0	85.0	90.0
Use of Availability	%	90.0	90.0	90.0	85.0	95.0
Utilization	%	76.5	76.5	76.5	72.25	85.5
Effectiveness	%	80.0	85.0	87.0	85.0	90.0
OEE	%	61.2	65.0	66.6	61.4	77.0
Total Hours	hours	8,760	8,760	8,760	8,760	8,760
Scheduled Hours	hours	8,520	8,520	8,520	8,520	8,520
Down Hours	hours	1,278	1,278	1,278	1,278	852
Delay Hours	hours	1,304	978	847	923	728
Standby Hours	hours	724	724	724	1,086	383
Operating Hours	hours	6,518	6,518	6,518	6,156	7,285
Ready Hours	hours	5,214	5,540	5,670	5,232	6,556

Table 16.19: Major Equipment Purchase Schedule

Major Equipment	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Production Drill (4.5”-8)	4	1	-	1	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Auxiliary Pre-split Drill (4.5-8)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (15 m^3)	2	-	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Wheel Loader (23 m^3)	2	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Haul Truck (140 t)	18	-	-	6	7	-	-	-	-	-	-	-	3	2	-	-	-	-	-	-	-	-	-	-
Track Dozer (600 HP)	6	1	-	2	-	-	-	-	-	-	-	2	1	-	-	-	-	-	-	-	-	-	-	-
Motor Grader (18 ft)	5	1	-	-	-	-	1	-	1	-	-	-	1	-	-	1	-	-	-	-	-	-	-	-
Water Truck	4	-	-	1	-	-	-	-	1	-	-	-	-	1	-	-	-	-	1	-	-	-	-	-
Wheel Dozer (530 hp)	3	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.20: Major Equipment Requirement

Major Equipment	Max	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Production Drill (4.5”-8)	3	-	1	2	3	3	3	3	3	3	3	3	3	3	2	2	1	1	1	1	1	-	-	-
Auxiliary Pre-split Drill (4.5-8)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-
Diesel Hydraulic Shovel (15 m^3)	2	-	-	1	1	1	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	-	-	-
Wheel Loader (23 m^3)	2	-	-	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	-
Mining Haul Truck (140 t)	13	-	-	6	13	13	13	13	13	13	12	12	12	11	8	8	5	5	5	4	3	2	2	-
Track Dozer (600 HP)	3	-	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	-
Motor Grader (18 ft)	2	-	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	-
Water Truck	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Wheel Dozer (530 hp)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.21: Support Equipment Purchase Schedule

Support Equipment	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Blaster Pick-up Truck (2 t)	9	-	1	-	1	-	1	-	1	-	1	-	1	-	1	-	1	-	1	-	-	-	-	-
Wheel Loader Stemming (150 hp)	3	-	-	1	-	-	-	1	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-
Excavator (49 t)	4	1	-	1	-	-	-	-	-	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-
Excavator (90 t)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Hammers for Excavator 49 t	4	1	-	-	-	-	1	-	-	-	-	1	-	-	-	-	1	-	-	-	-	-	-	-
Wheel Loader (271 hp)	2	-	-	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
Telehandler (5 t)	2	1	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mechanic Service Truck (6 wheel)	8	1	-	1	-	-	-	1	1	-	-	-	1	1	-	-	-	1	1	-	-	-	-	-
Wheel Loader (350 hp)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuel & Lube Truck (17 kL) ADT	10	1	-	1	-	-	1	1	-	-	1	1	-	-	1	1	-	1	1	-	-	-	-	-
Truck Tractor for Trailers (560 hp)	2	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Trailer Lowboy (100 t)	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pick up Truck (1/2 t)	52	9	4	-	-	-	-	13	-	-	-	-	13	-	-	-	-	13	-	-	-	-	-	-
Pit Bus	3	1	-	-	-	-	-	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Mobile Air Compressor (185 CFM)	10	2	-	-	-	-	2	-	-	2	-	-	-	2	-	-	2	-	-	-	-	-	-	-
Welding Machine Electric (350 A)	6	-	-	2	-	-	-	-	-	-	2	-	-	-	-	-	-	2	-	-	-	-	-	-
Welding Machine Diesel (400 A)	6	2	-	-	-	-	-	-	-	2	-	-	-	-	-	-	2	-	-	-	-	-	-	-
Light Plant	32	8	-	-	-	-	-	8	-	-	-	-	8	-	-	-	-	8	-	-	-	-	-	-
Genset (60 kW)	12	3	-	-	-	-	-	3	-	-	-	3	-	-	-	3	-	-	-	-	-	-	-	-
Water Pump 3" - Gasoline	9	2	-	1	-	-	-	-	-	1	2	-	-	-	-	-	1	2	-	-	-	-	-	-
Diesel Heaters	4	2	-	-	-	-	-	-	-	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-
Snow Blower	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Soil Compactor (56”)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Pump HH 10 in – Diesel	9	-	1	-	1	-	1	-	1	-	1	-	1	-	1	-	1	-	1	-	-	-	-	-

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.22: Support Equipment Requirement

Support Equipment	Max	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Blaster Pick-up Truck (2 t)	1	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Wheel Loader Stemming (150 hp)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Excavator (49 t)	2	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Excavator (90 t)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Hydraulic Hammers for Excavator 49 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Wheel Loader (271 hp)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Telehandler (5 t)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Mechanic Service Truck (6 wheel)	2	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Wheel Loader (350 hp)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Fuel & Lube Truck (17 kL) ADT	2	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Truck Tractor for Trailers (560 hp)	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Trailer Lowboy (100 t)	1	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Pickup Truck (1/2 t)	13	9	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	5	5	-
Pit Bus	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Mobile Air Compressor (185 CFM)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Welding Machine Electric (350 A)	2	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Welding Machine Diesel (400 A)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Light Plant	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	-	-	-
Genset (60 kW)	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-	-	-
Water Pump 3" - Gasoline	3	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-	-	-
Diesel Heaters	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Snow Blower	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Soil Compactor (56”)	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Water Pump HH 10 in – Diesel	5	-	1	3	3	3	3	3	3	3	3	3	3	3	3	5	5	5	5	5	5	-	-	-

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

16.4.10 Mobile Crushing Plant

The production of crushed material will be necessary for blasthole stemming and road maintenance. It is assumed that the required aggregate material will be produced internally using a mobile crusher on site. Waste rock to supply the aggregate crushing plant will come from both pre-production and production phases of the pit.

16.4.11 Mine Maintenance

Maintenance will be performed by the owner's personnel supported by OEM personnel directly on site. The maintenance department and personnel requirements have been structured to fully manage this function, including maintenance planning and employee training. Tire monitoring, rotation and replacement will also be performed internally. For production shovels, maintenance will be performed in the field when possible or the shovels will move on their tracks to an assembly pad outside the production area when major repairs are required. Other mine maintenance will be carried in a temporary truck shop during the first years of operation and in a permanent facility thereafter. Further details about the surface mine garage are provided in Section 18.

16.4.12 Pit Slope Monitoring

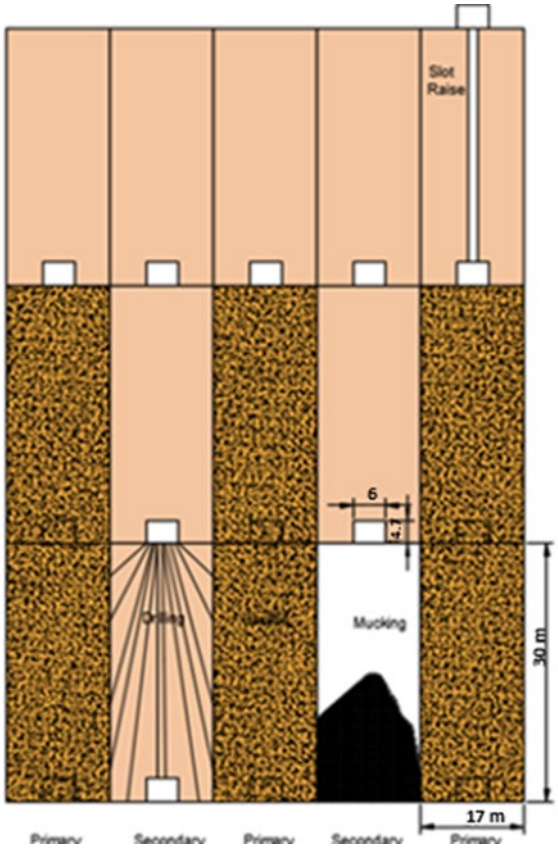
A pit slope monitoring system will be used to gather information on both micro- and macro-movements of the pit walls. It will consist of strategically placed prisms that are surveyed under a controlled environment (windless, rainless, and stationary).

16.5 Underground

16.5.1 Underground Mining Method

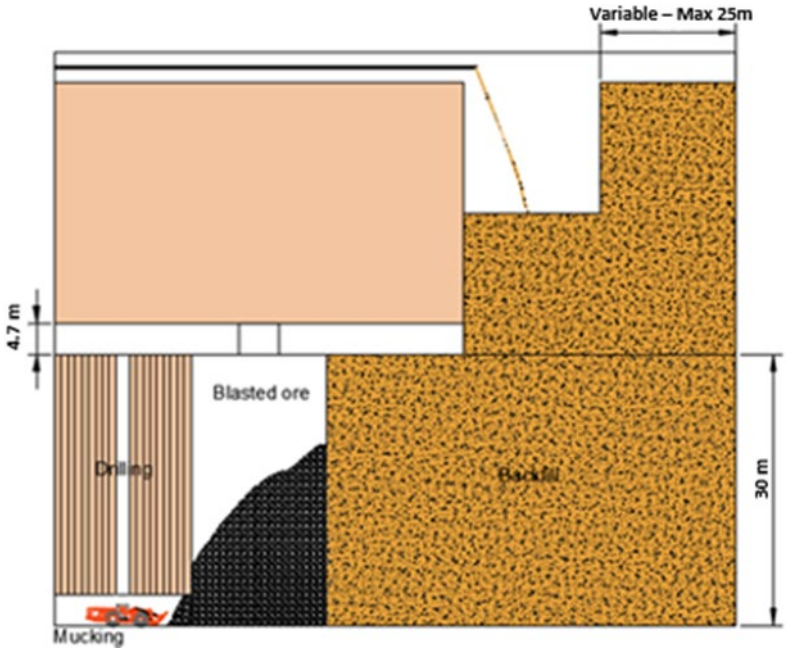
The selected underground mining method is long-hole open stoping (LHOS), employing transverse, longitudinal, or up-hole stoping configurations. The stoping sequence will be ascending from an initial undercut. Generally, stopes will be drilled from an upper access and mucked from a lower access. Stopes within sill pillars will require the upper access to be redeveloped through backfill to restore drilling access. Figure 16.29 illustrates the transverse LHOS mining method and Figure 16.30, the longitudinal LHOS mining method.

Figure 16.29: Representation of Transverse LHOS Mining Method



Source: GMS, August 2025.
**Note: Not to scale.*

Figure 16.30: Representation of Longitudinal Mining Method



Source: GMS, August 2025.
**Note: Not to scale.*

Long-hole open stoping (LHOS) is a widely used underground mining method for competent, hard-rock orebodies. The first phase of mining is the development phase, which includes excavating the decline, level access, haulage drifts, draw points, and associated support infrastructure. These openings provide access to the stoping area and facilitate production activities. The development phase also encompasses the construction of both an overcut drift and an undercut drift.

The overcut drift is developed to enable production drilling in the stoping area, while the undercut drift provides access for ore extraction. Once both the overcut (drilling) and undercut (extraction) drifts are completed, a slot raise is performed between these levels to create an initial void for production drilling and blasting activities.

The production drilling phase involves drilling long vertical or inclined holes, at regular intervals along the length and width of the stoping area. Once drilling is completed, the holes are charged with bulk explosives and stemming prior to blasting. In the case of the Shaakichiuwaanaan Project, only bulk or prepacked emulsion is planned to be used for both development and production. Emulsion is water resistant and, compared to ammonium nitrate fuel oil (ANFO), generates minimal dissolved ammonium nitrate in the environment when used in accordance with manufacturer's instructions.

During the blasting phase, the controlled use of explosives fractures the rock surrounding the slot raise. Once the rock is blasted and the blasting gases have cleared from the mine, the fragmented material is removed from the stope through the undercut drift with a load-haul-dump (LHD) unit. Depending on the size and geometry of the stopes, it can take up to 3 or 4 blasting and mucking cycles to extract all the material from the stope.

The broken material from the stopes is loaded into haul trucks and then transported to surface. LHOS is a non-entry mining method, as the stoping area is not accessible to personnel once production begins. The use of remote-controlled LHD units is required to completely remove blasted material from the stope.

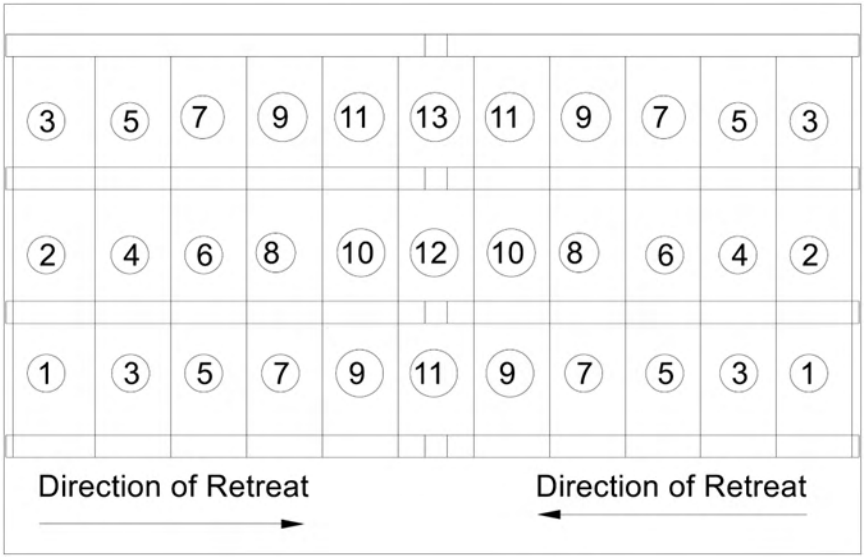
The final phase of LHOS is the backfilling phase. Depending on multiple factors, such as LHOS variant used and the mining sequence, stopes can be filled with cemented material, uncemented material or a combination of both. Cemented backfill can include cemented paste backfill (CPB), hydraulic fill and cemented rockfill (CRF) while uncemented backfill is typically rockfill (RF). The selected underground cemented backfill method for the Shaakichiuwaanaan Project is cemented paste backfill. This non-entry mining method offers several advantages including, but not limited to high productivity, high operational flexibility, low operating costs, efficient ore recovery, and improved worker safety.

The longitudinal variant of LHOS is used for the following situations:

- When it is economically non-viable to develop using transversal mining method.
- In non-prioritized mining areas.

The longitudinal mining areas are accessed by developing overcut and undercut ore drifts inside the stoping area along the strike of the orebody. Once the development is completed to the extremity of the longitudinal mining area, the production cycle of the initial stope can begin. Subsequent stopes are mined in the same cycle while retreating towards the main access located either at an extremity or near the middle of the orebody. Figure 16.31 illustrates the typical longitudinal LHOS mining sequence.

Figure 16.31: Typical Longitudinal Stoping Sequence



Source: GMS, June 2025.
**Note: Not to scale.*

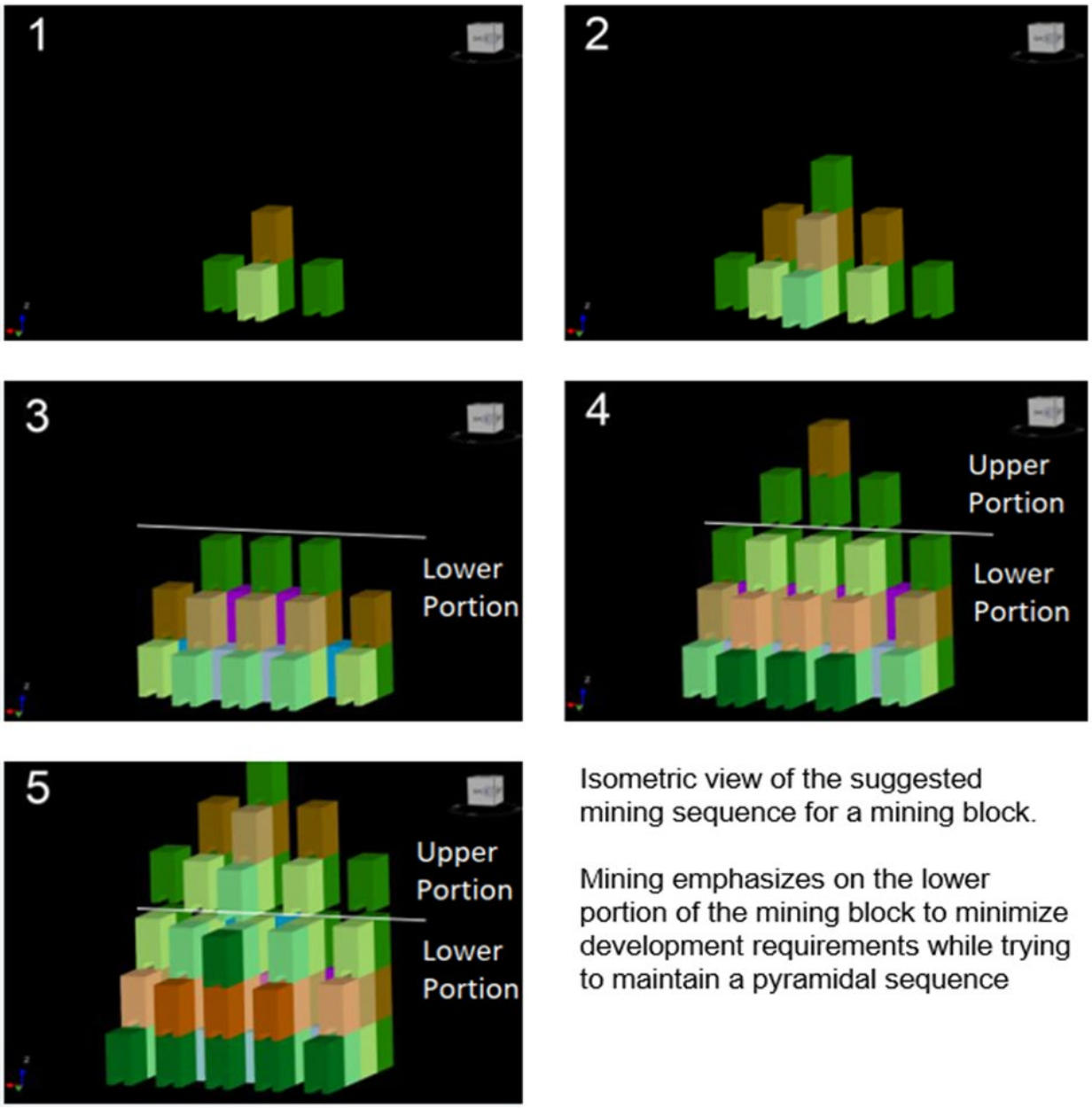
The transverse variant of LHOS is used for the following situations:

- Stope widths are wider and economically viable to pay off its development.
- Mining areas are prioritized to allow for an efficient primary–secondary mining sequence.

The transverse mining area is accessed by developing a haulage drift parallel to the strike of the orebody. This haulage drift is maintained at a sufficient distance from the stoping area to preserve its integrity during the production phase. Perpendicular to the haulage drift, a series of evenly spaced draw points are developed to access both the undercut and overcut ore drifts, extending across the full width of the orebody. Once the haulage drift is established and the draw points and ore drifts are developed, the production cycle of the initial primary stope can begin. Subsequent primary stopes are mined in the same cycle while retreating towards the extremities of the orebody.

Stopes will be sequenced in an overhand approach so that when two (2) lifts of primary stopes are mined, the secondary stope between these two (2) primaries can be mined on the first lift. This primary-secondary transverse mining approach is advantageous in terms of production and operational flexibility as multiple stopes can be in operation simultaneously. One disadvantage of the method is that a haulage drift must be excavated along the entire length of the production zone. However, the production cycle can still begin even if the haulage drift is not completely excavated to the extremities of the orebody. Figure 16.32 illustrates the typical sequence of transverse stoping.

Figure 16.32: Typical Transverse Stoping Sequence



Source: Alius, June 2025.
**Note: Not to scale.*

16.5.1.1 Mining Recovery

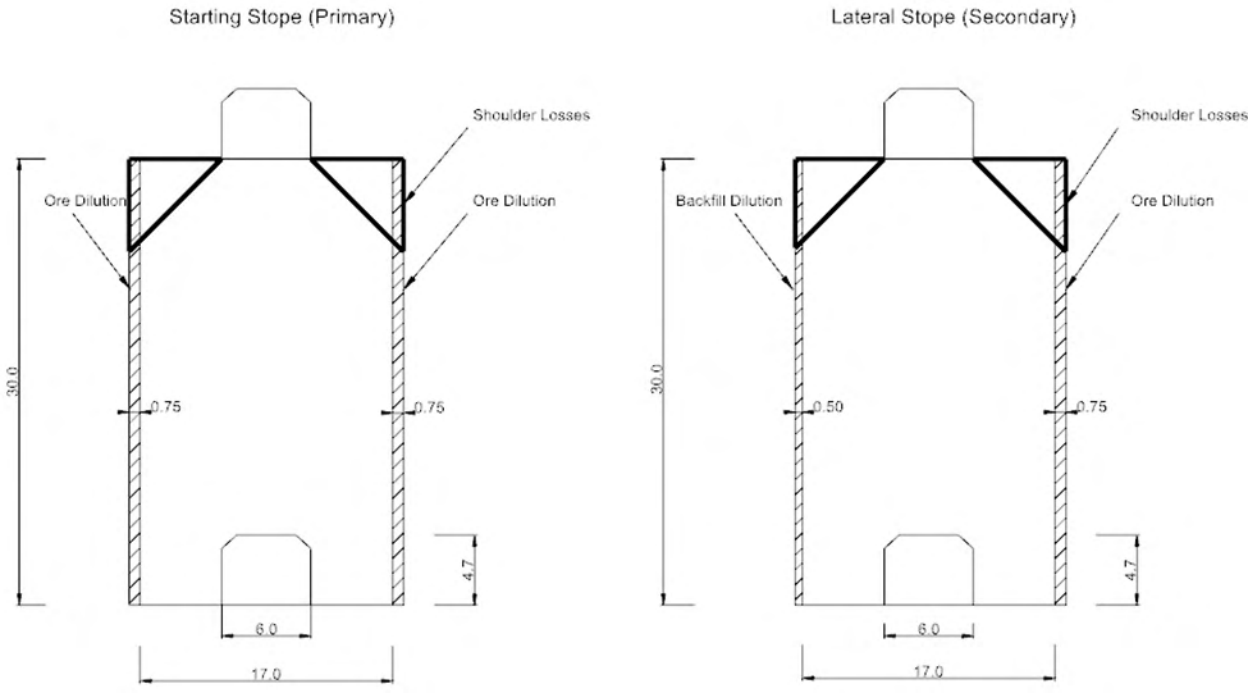
The mining recovery factor represents the ore that remains unrecovered due to typical operational constraints in the mining process. This factor has been determined based on the limitations of the planned mining method, stope dimensions, equipment limitations, production drilling design and practical operating conditions.

Long-hole mining methods are subject to drilling and blasting inefficiencies, which leave some ore unrecovered. Remote stope mucking losses occur due to difficult recovery of stope corners and edges, where equipment cannot easily reach.

- Stope and ore losses arise from operational problems, including misfires, hang-ups, poor blasting control, or challenges associated with difficult ground conditions.
- Dilution control measures, where ore is intentionally left behind to minimize the mixing of waste rock with ore, can reduce mill feed quality issues.

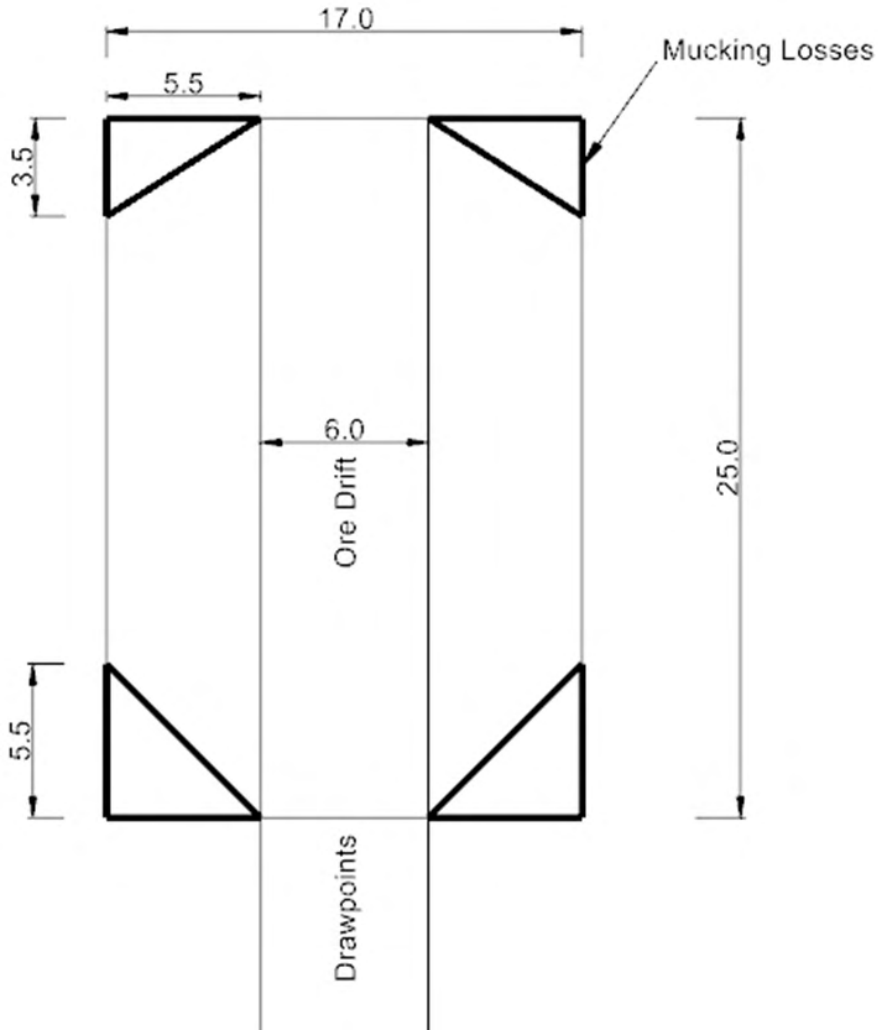
A mining recovery factor of 90% was used for all mining blocks. Figure 16.33 shows the shoulder losses caused by drilling and blasting, while Figure 16.34 illustrates the losses related to mucking.

Figure 16.33: Ore Losses During Drilling & Blasting Operations - Section View



Source: GMS, August 2025.
**Note: Not to scale.*

Figure 16.34: Ore Losses During Mucking Operations - Plan View



Source: GMS, August 2025.
**Note: Not to scale.*

16.5.2 Underground Mine Design

16.5.2.1 Development Design

The Shaakichiuwaanaan underground mine will be accessed by a single decline, with a portal located north of the open pit and near the ROM pad.

Each production level will be accessed by a level access and a haulage drift that leads to the crosscuts, driven either perpendicular to the orebody for transverse stoping or parallel to the direction of the orebody for longitudinal stoping. For a typical level access, the following infrastructure must be included: sump, electrical bay and fresh air access. For a typical haulage drift, the following infrastructure are included: loading bay, backfill bay, access to auxiliary ventilation raise, and safety egress.

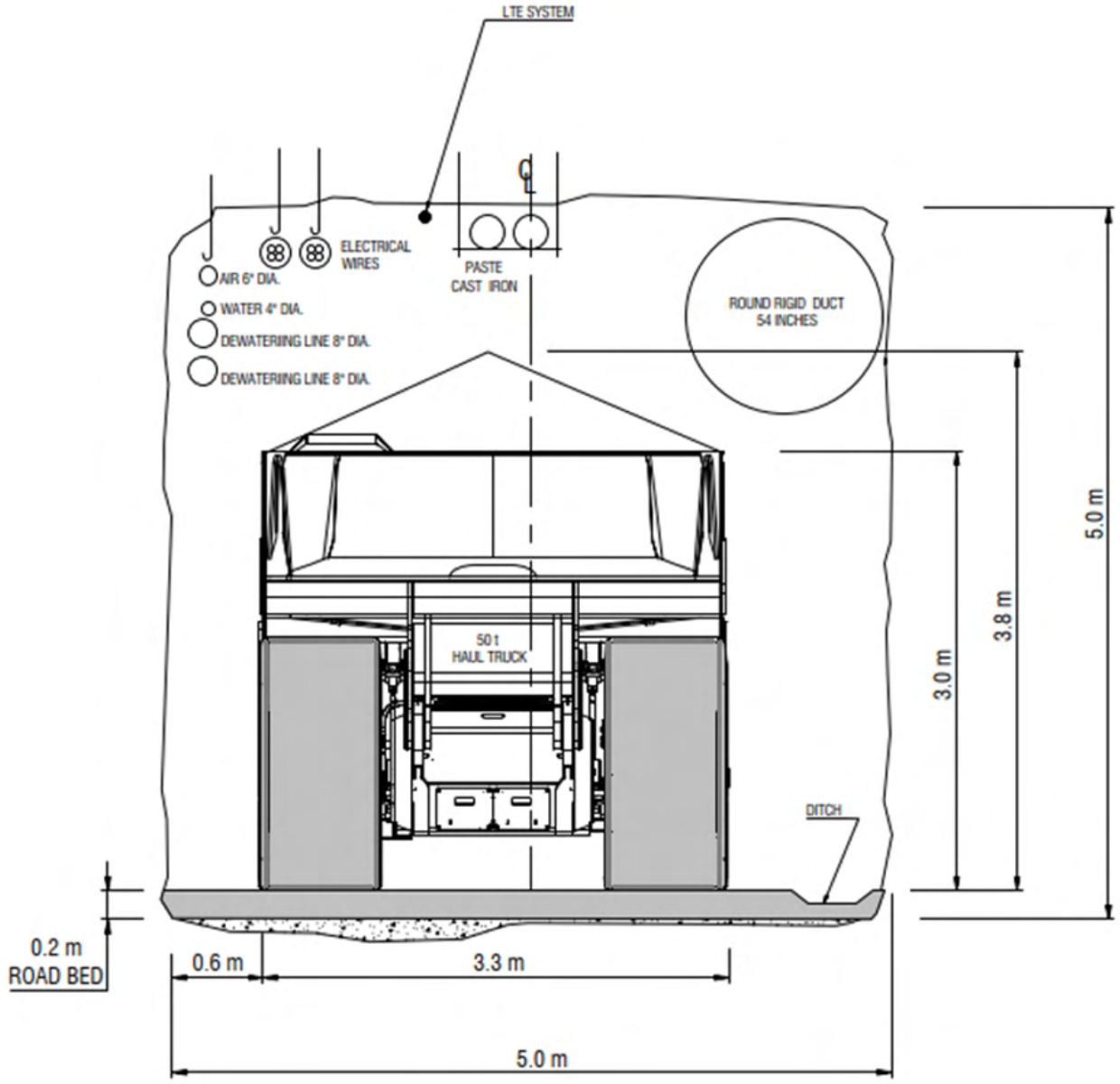
Underground development has been sized to ensure safe operation of the selected equipment along with enough clearance to account for the required underground mine services.

Decline is designed at 5.0 m (W) x 5.0 m (H) and level accesses at 5.5 m (W) x 6.0m (H) allowing additional space to install more mine services and ventilation ducting as typically required in level entry. Figure 16.35 presents a typical section of decline.

Hauling drifts are planned to be 5.0 m (W) x 05.0 m (H) and draw points are planned to be 6.0 m (W) x 4.7 m (H) allowing clearance for the selected LHDs and an optimal drilling pattern for the selected drilling rigs Figure 16.36 presents a typical section of the draw points.

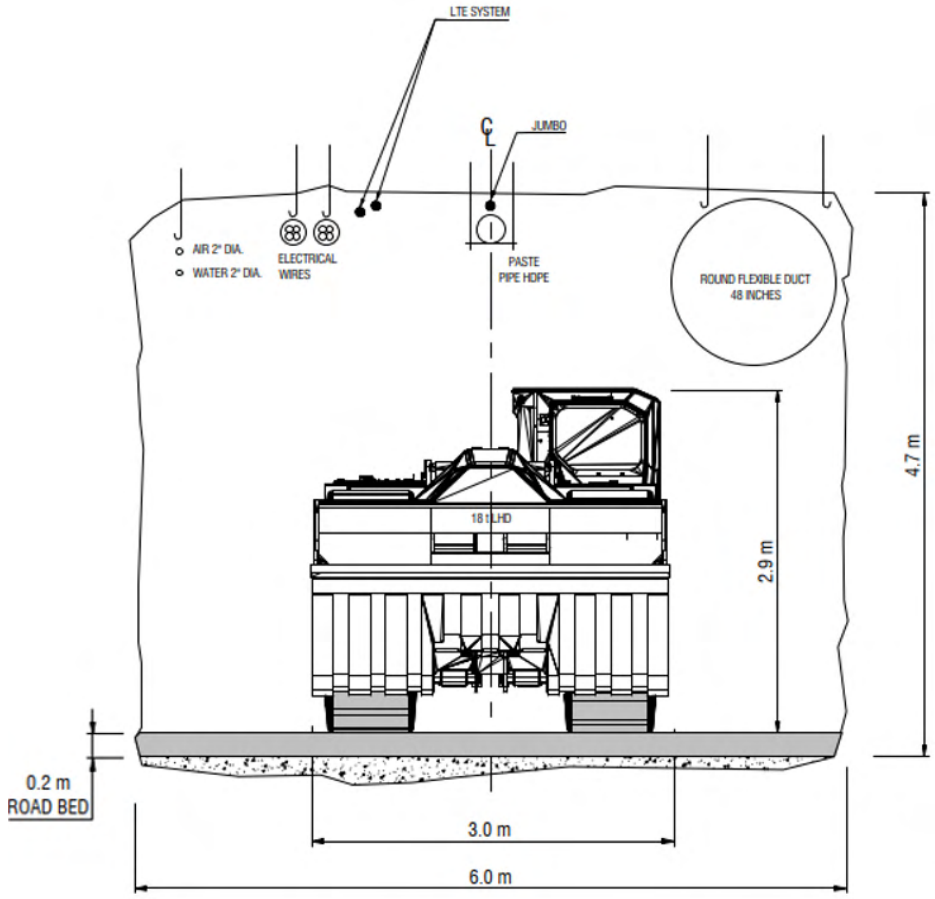
When possible, the ramp is designed in a straight line to optimize truck hauling. In the upper portion of the ramp, three (3) passing bays are included in the design, allowing two (2) vehicles to safely pass without reversing into another drift, thereby improving traffic management.

Figure 16.35: Typical Development Section of the Decline



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.36: Typical Development Section of a Draw Point



Source: GMS, June 2025.
**Note: Not to scale.*

To account for unplanned ground issues and blasting effects on the rock mass, a constant 10% overbreak factor was applied to all waste tonnes. An additional average 2.5% factor was added to waste tonnes to account for operational add-ons such as turning radius, take-down back, safety bays, transitional sections between different sizes, etc. No overbreak factor was applied to ore development. Figure 16.37 and Figure 16.38 show plan views of typical underground production levels, while Figure 16.39 and Figure 16.40 show a longitudinal view and a section view of the lateral and vertical development of the mine.

The underground portion of the Shaakichiuwaanaan Project is relatively shallow but extends up to 1,850 metres in the East-West direction. Constrained by the crown pillar at the upper level of mine, all haulage drifts and draw points were designed with a consistent 3.0% grade to better assess the impact on resource recovery and mine design. For the span of a level, a 3.0% grade implies an elevation gain of up to 37.1 metres on the same level for the upper levels of the mine. To maintain the upper level within the limit of the defined crown pillar, a “W” design was implemented as seen in Figure 16.40. The haulage drift

of the upper level is split into sections of about 200 m alternating positive and negative grades. To ease water management, sumps are planned at each low point.

Another point influenced by the significant strike length of the mine is the offset of many levels access in order to achieve an efficient decline design. Because of these offsets, the haulage drifts either pinch or deviate with the upper and lower levels, significantly altering the stope heights. This point will be discussed in the Section 16.5.2.2.

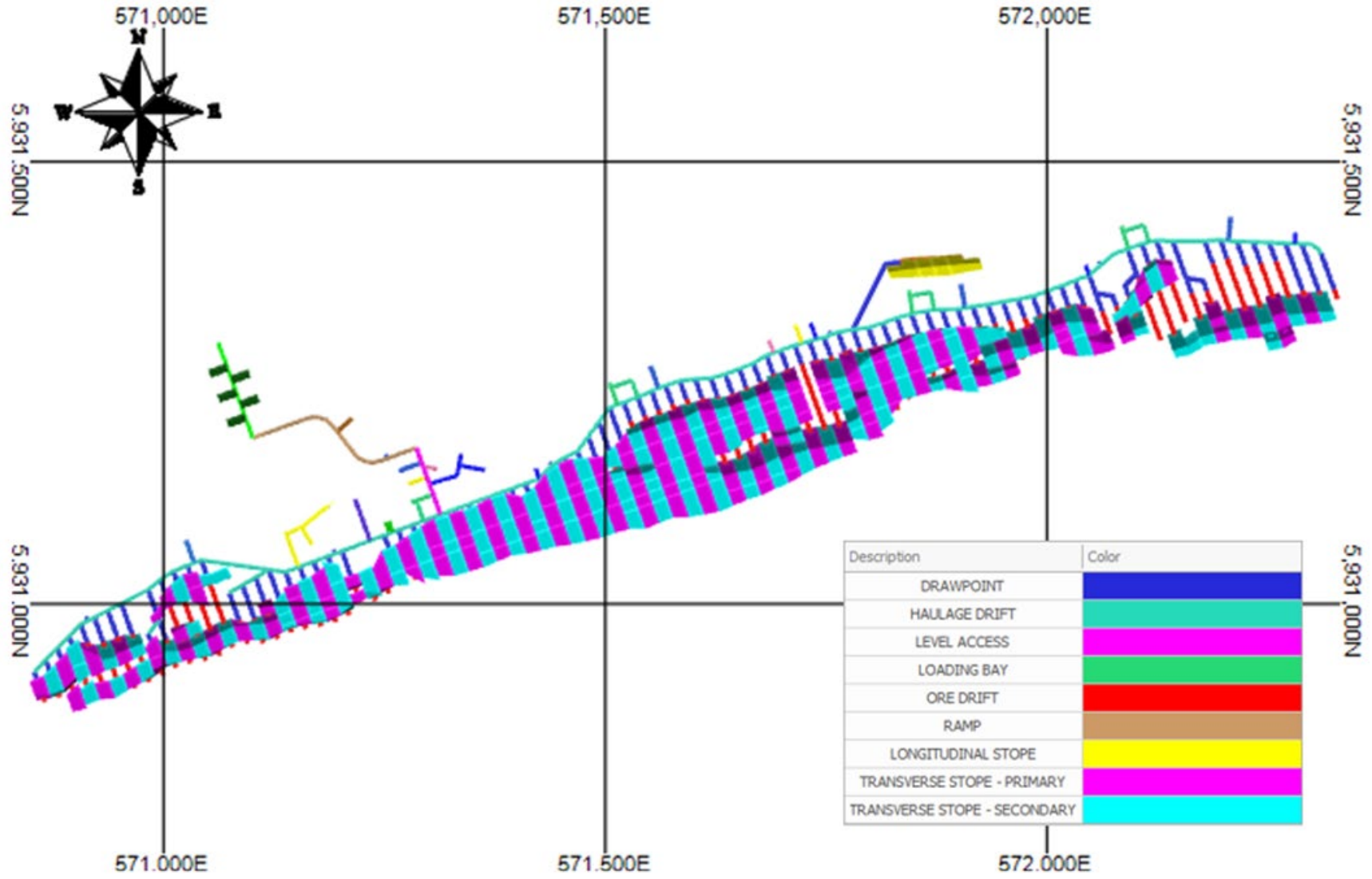
Figure 16.41 shows Lake 001 along with a schematization of the water management infrastructure in dark orange, overlaid on the underground mine and the eastern portion of the open pit. The original area covered by lakes in the region is represented in light blue, whereas the area unaffected by mining after the construction of the second dam is shown in dark blue.

The available surface area directly above the underground mine limits access for any breakthrough excavations as well as service boreholes.

The underground mine has two ventilation shafts. The first one is excavated along the decline to optimize the pre-production ventilation circuit as an intake and is later switched to an exhaust shaft. The location of this first ventilation shaft is a compromise between the proximity of Lake 001, the eastern open pit, and the needs of the underground mine.

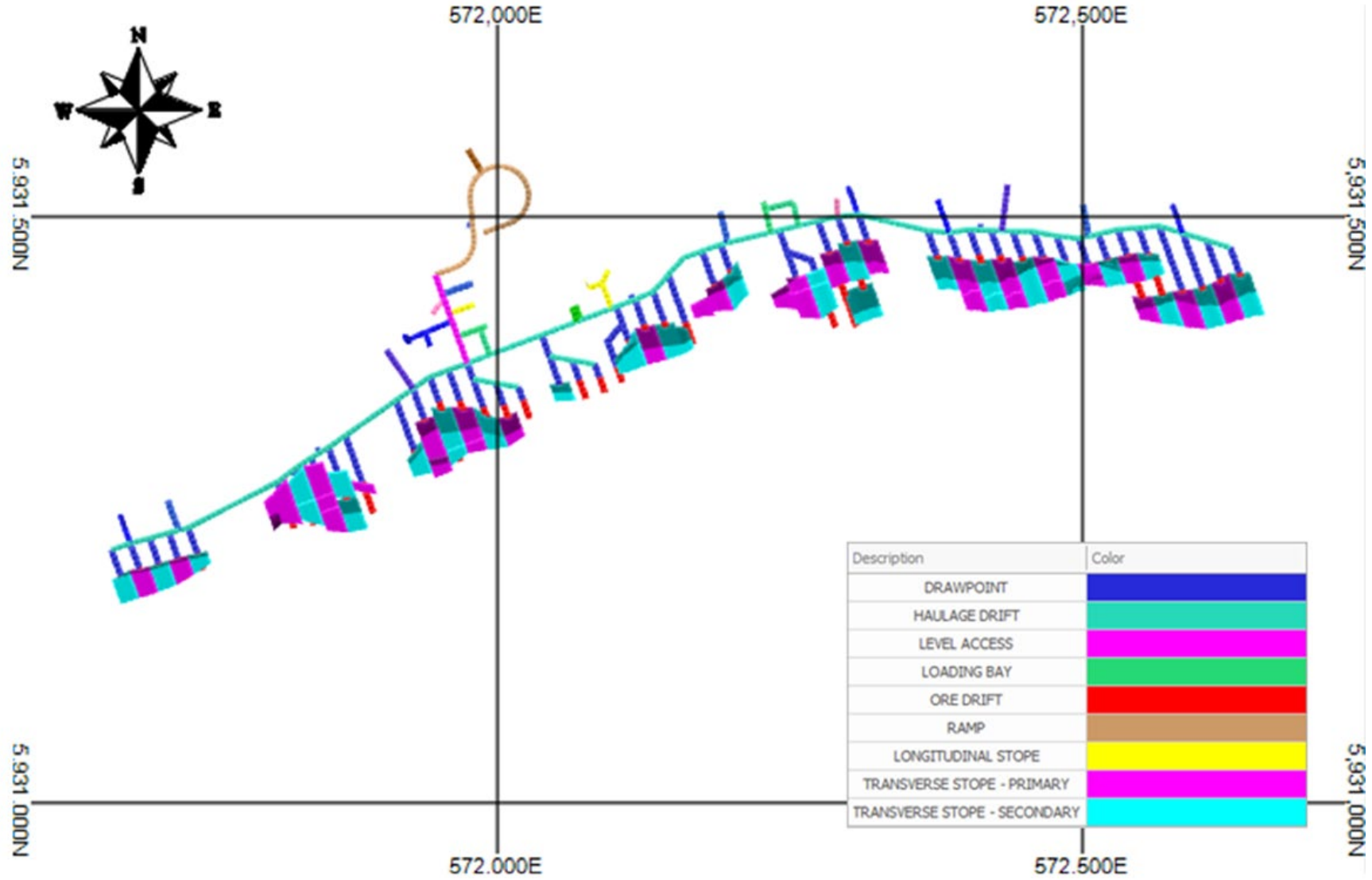
The second ventilation shaft, which serves as the intake of the underground mine, is located near the industrial area of the Project at the end of a ventilation drift. This drift is also used to bring most underground mine services, such as paste backfill, power, and fresh water, as well as to provide a mine safety egress.

Figure 16.37: Typical Larger Level - Plan View



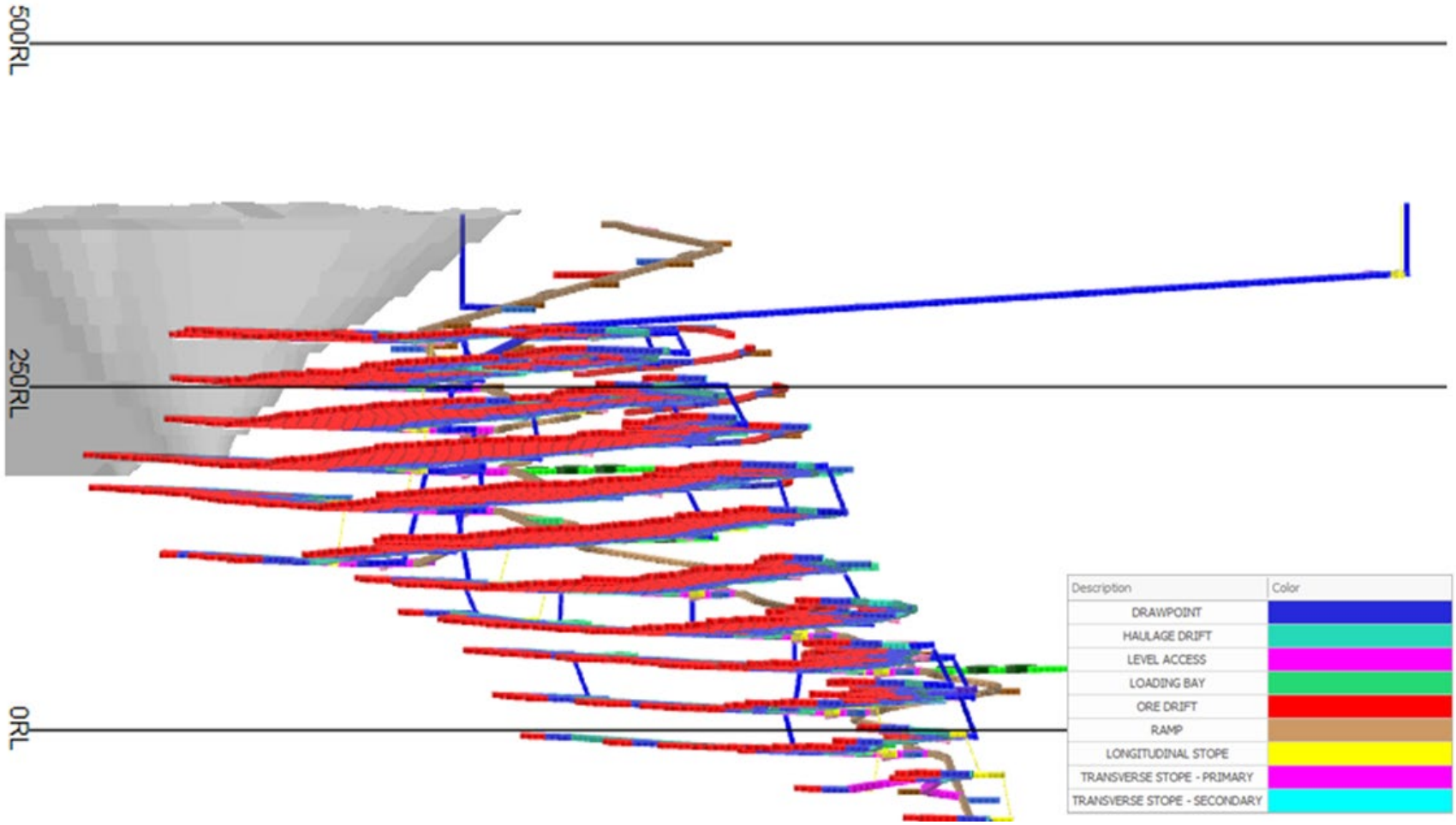
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.38: Typical Smaller Level - Plan View



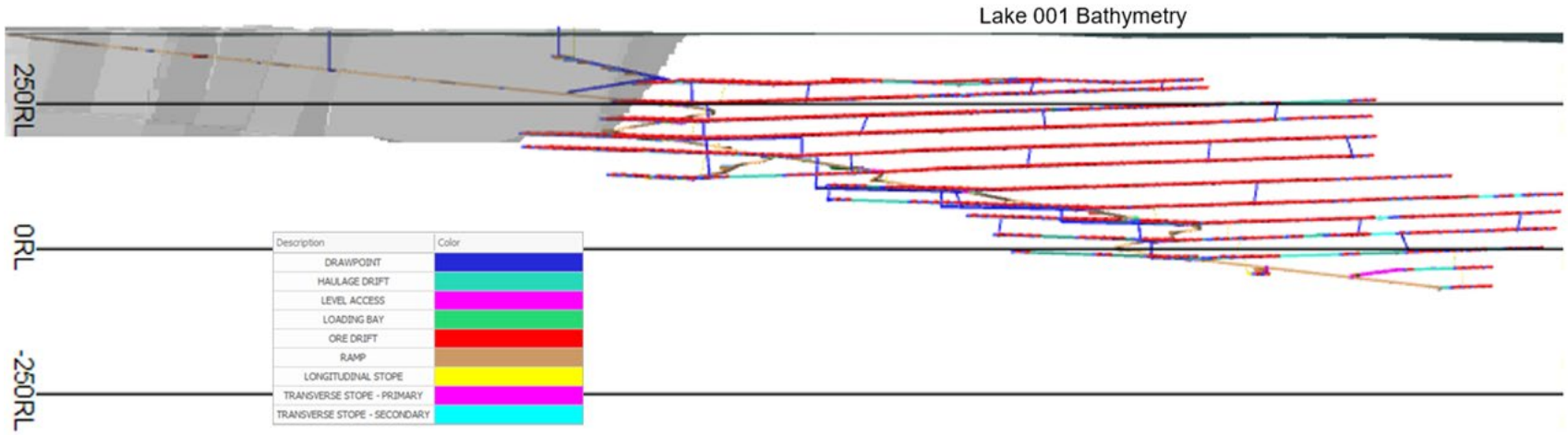
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.39: Mine Development Longitudinal View - Looking West



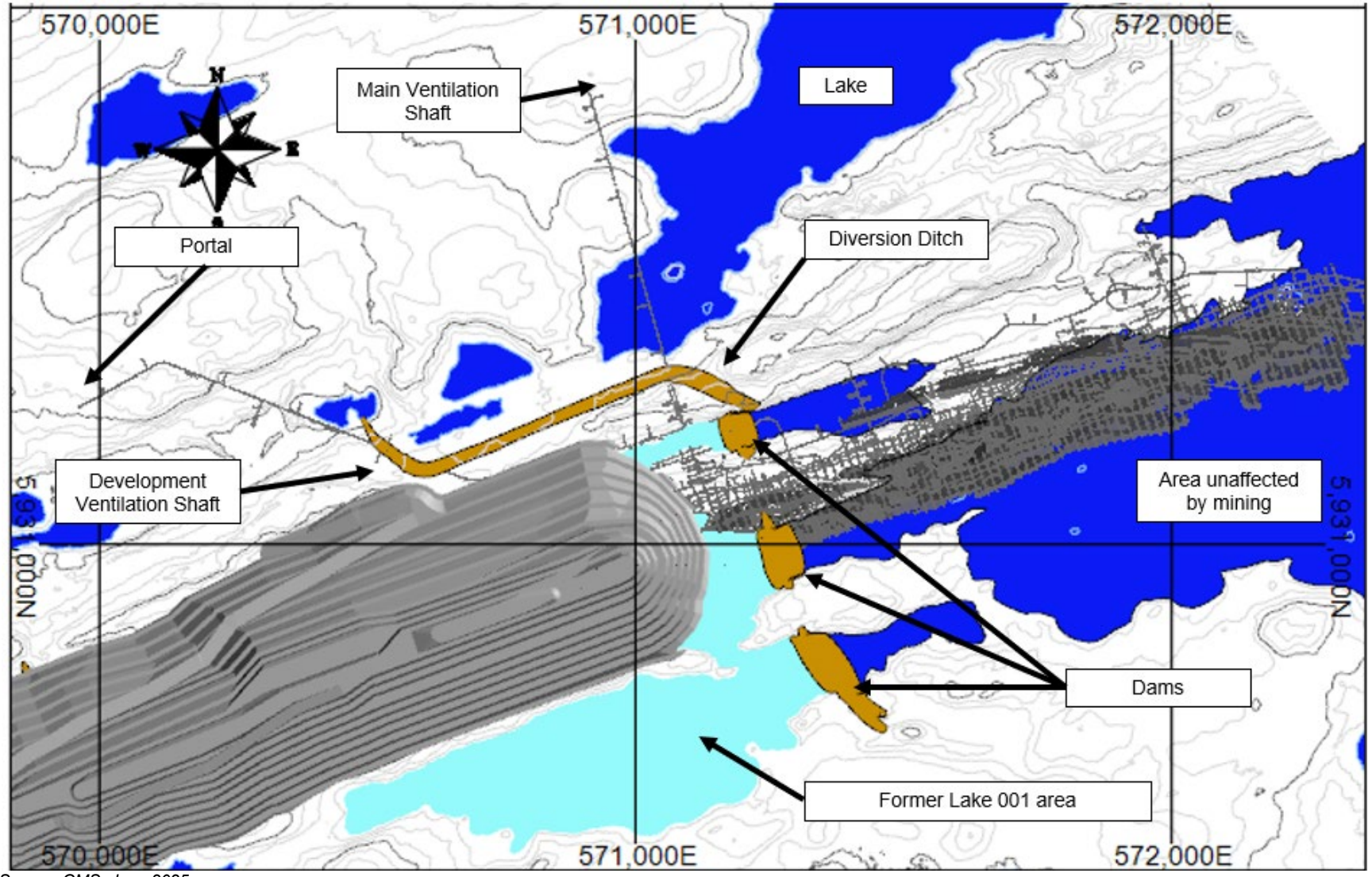
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.40: Underground Mine Development Longitudinal View - Looking North



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.41: Underground Mine and Surface Constraints – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

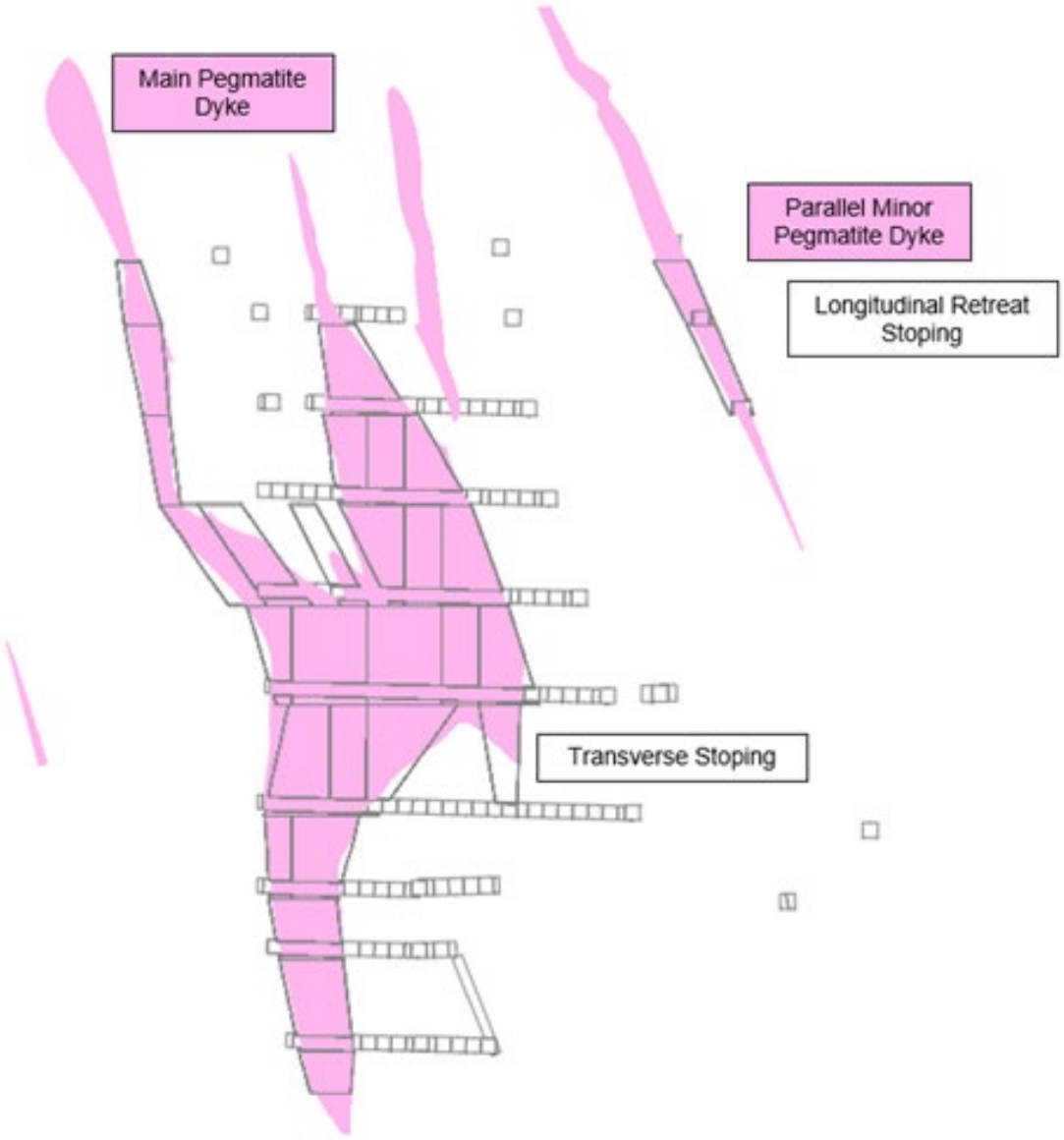
16.5.2.2 Stope Design

The Shaakichiuwaanaan underground mine follows the West striking CV5 pegmatite orebody extending from the Open pit mine at the east for nearly 1,900 m. Most of the underground mine is located under Lake 001.

As shown in Figure 16.42, most of the stopes are located within the thicker main dyke where transverse stoping mining method is used, whereas a minority of stopes are mined using longitudinal retreat stoping method in parallel smaller dykes.

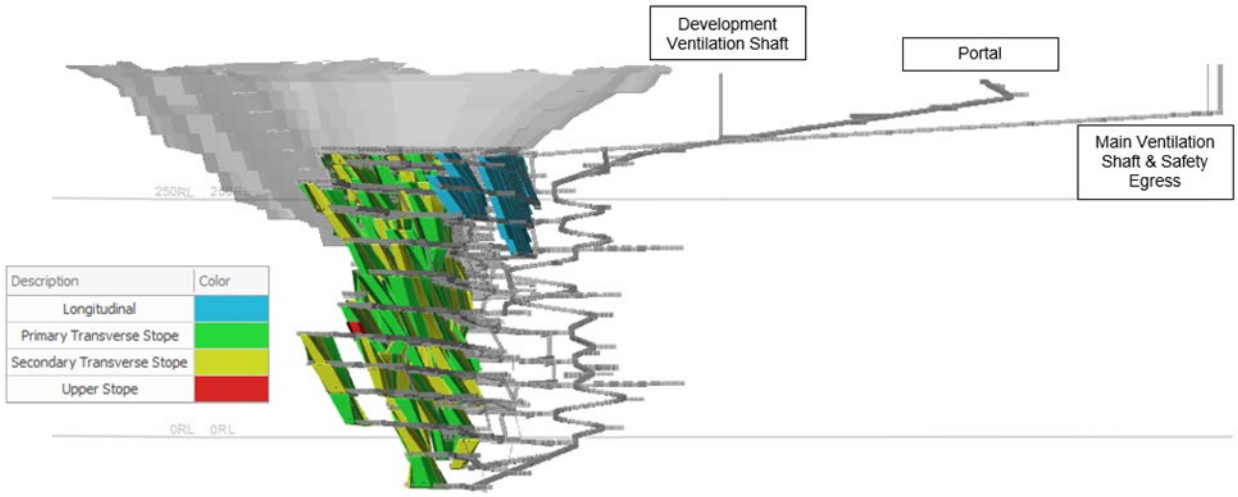
From the total of 1,548 stopes defined as economic, 72 are longitudinal retreat (5%), one (1) is up-hole stoping (<1%) and the remaining 1,475 (95%) are transverse stoping. To reflect more realistic operational constraints and optimization, 23 half-stopes have been joined to adjacent stopes, resulting in 1,525 stopes in the life of mine. Figure 16.43 and Figure 16.44 both show the distribution of the mining methods in the mine.

Figure 16.42: Stoping and Pegmatites - Looking West - Section at Easting 571,647



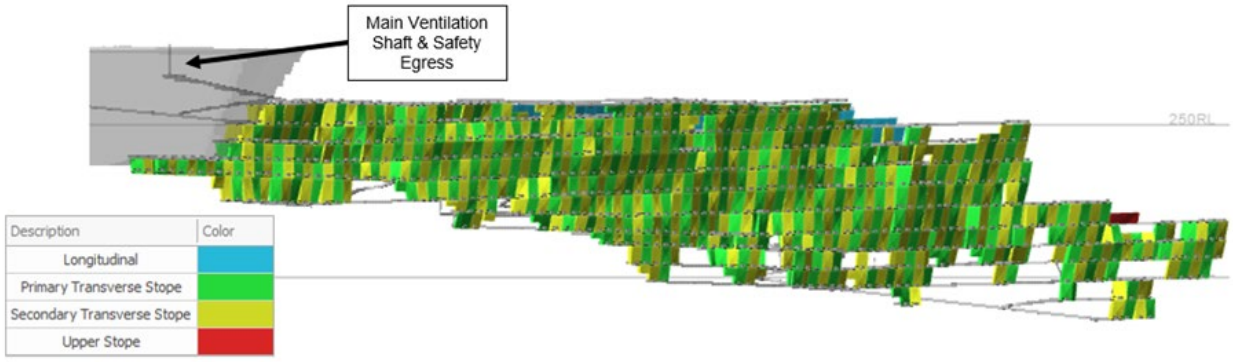
Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.43: Underground Mine Longitudinal View by Mining Method - Looking 289° W



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.44: Underground Mine Longitudinal View by Mining Method - Looking North



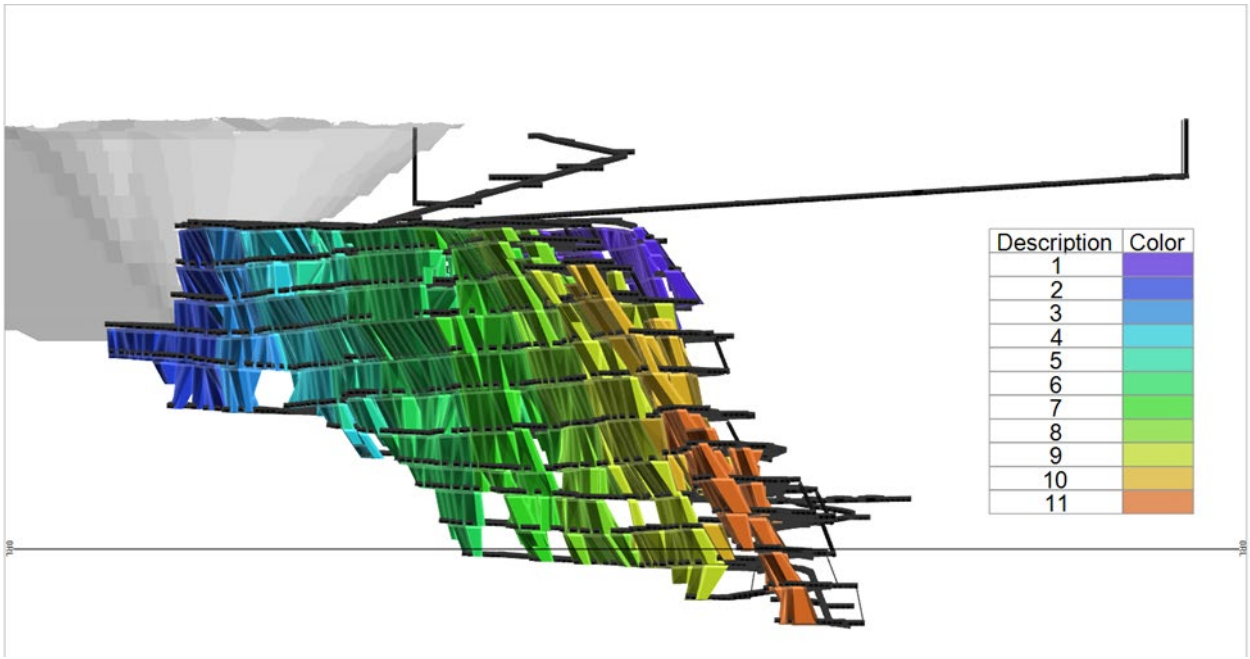
Source: GMS, June 2025.
**Note: Not to scale.*

The underground mine has a total of 12 mucking levels. To increase operational flexibility and achieve the production target, the underground mine was divided into several mining blocks. Figure 16.45 and Figure 16.46 show the 11 different mining blocks / pyramid of the underground mine.

Development and production activities will occur simultaneously in several mining blocks to ensure that multiple development headings and stopes are available.

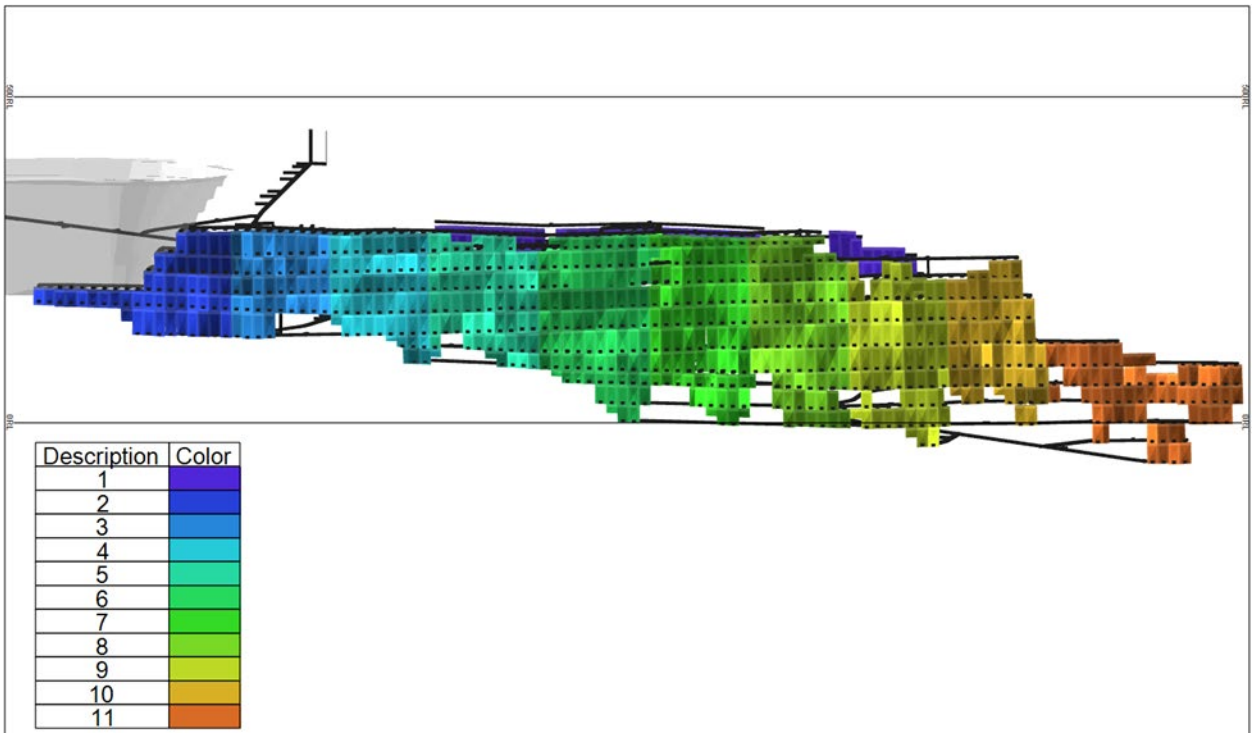
The orebody is shallow and extends along strike. To optimize development efficiency, mine stoping block sequencing will follow a West-to-East sequence, which helps reduce pressure on the development teams. The average stope thickness is 16.3 m, while the diluted stope tonnage averages 23,884 t.

Figure 16.45: Underground Mine Longitudinal View by Mining Blocks - Looking West



Source: GMS, June 2025.
**Note: Not to scale.*

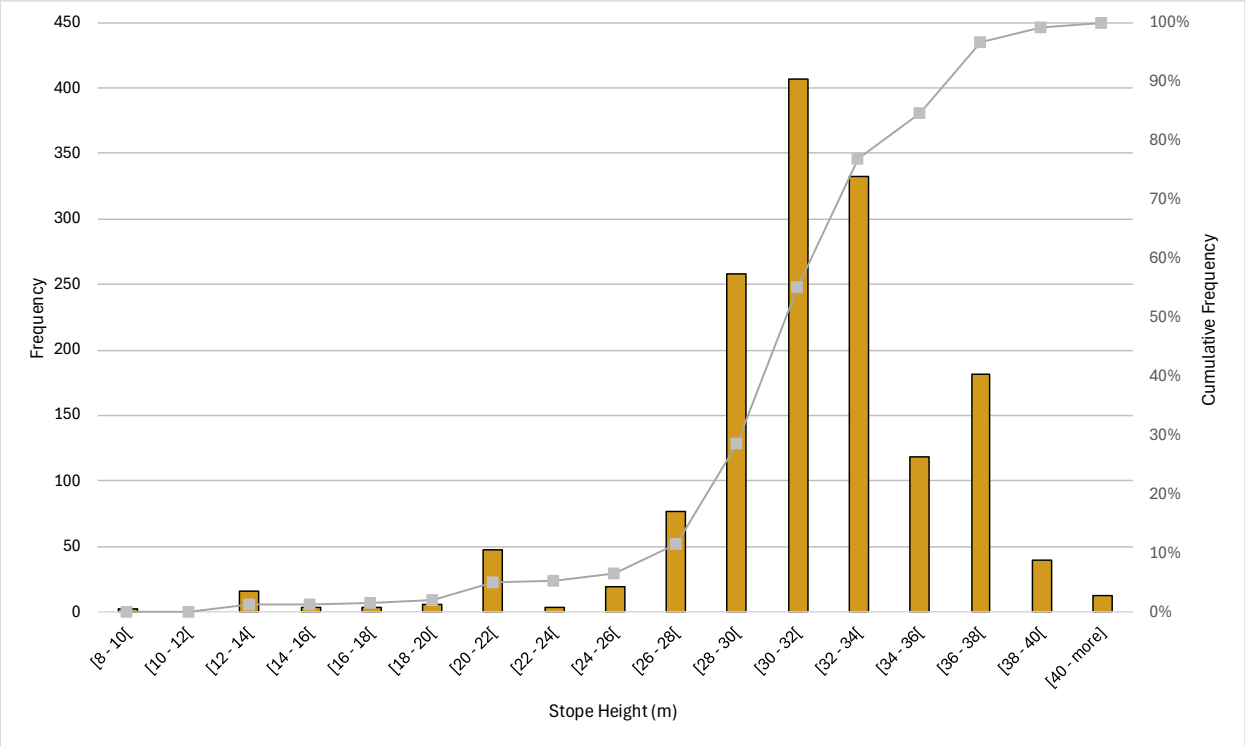
Figure 16.46: Underground Mine Longitudinal View by Mining Blocks - Looking North



Source: GMS, June 2025.
**Note: Not to scale.*

As described in Section 16.5.2.1, haulage drifts are designed with a typical gradient to fully capture the impacts on both the crown pillar and the stope height. Figure 16.47 illustrates the distribution of the stope height from the lower-lever floor to the upper-level floor across the underground mine.

Figure 16.47: Stope Height Histogram

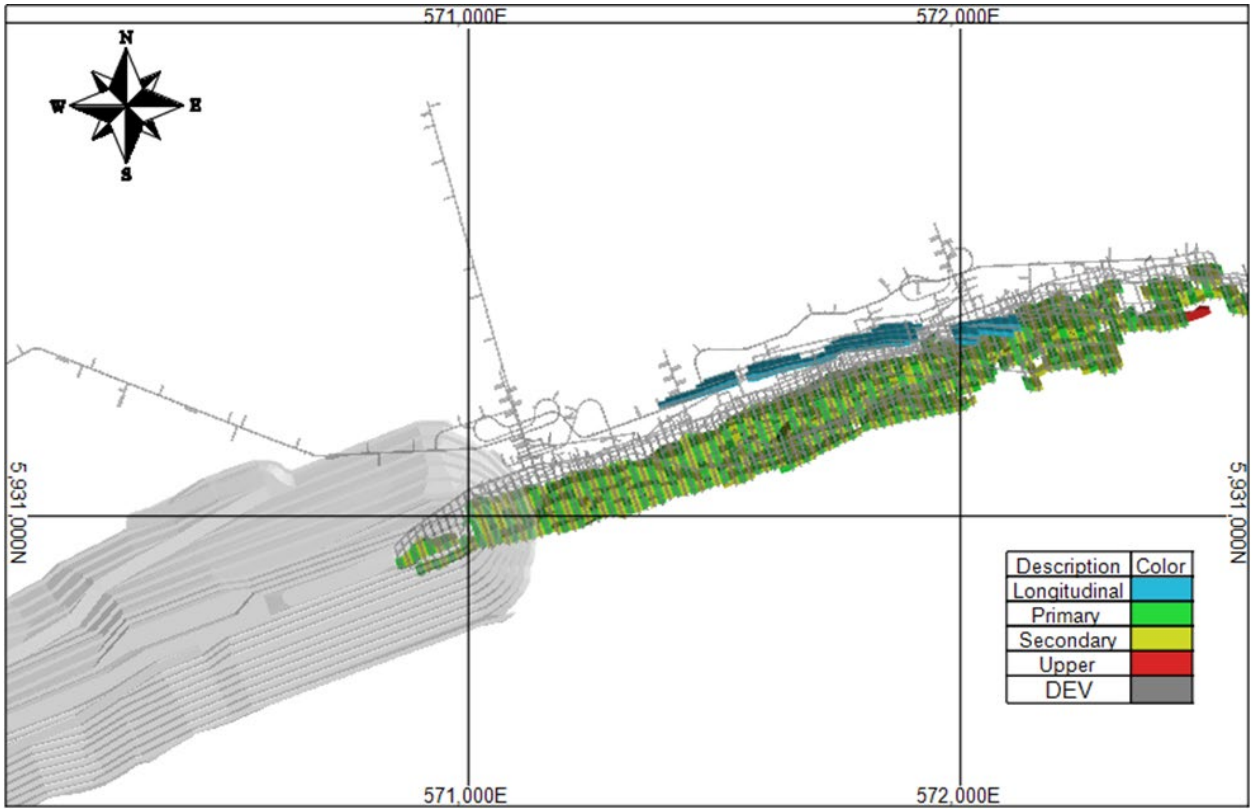


Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.48 and Figure 16.49 show the boundaries between the open pit mine and the underground mine. By mining the underground mine from the West-to-East and starting open pit mining in the smaller West pit, it is planned to complete mining of the western portion of the underground mine before the larger eastern pit reaches the elevations of the underground mine.

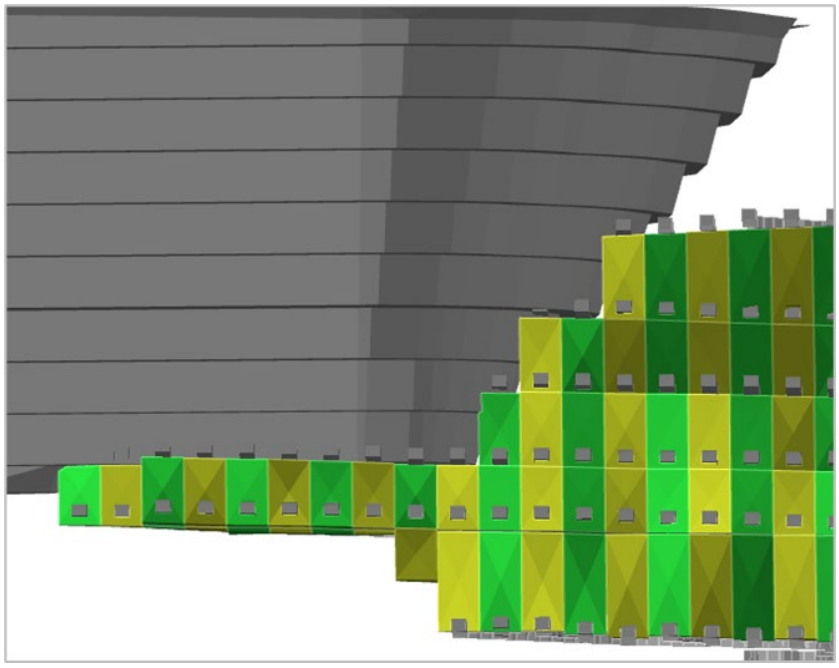
Thus, no pillar is planned between the two mines, allowing an optimization of the ore recovery. By the end of the eastern pit, cemented paste fill is expected to daylighting in the eastern wall. This interaction and geomechanical implications have been described in the Section 16.2.5.5.1.

Figure 16.48: Underground Mine with Open Pit – Plan View



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.49: Open Pit and Underground Mine Interaction Close-Up – Looking North



Source: GMS, June 2025.
**Note: Not to scale.*

16.5.2.2.1 Slot Raise

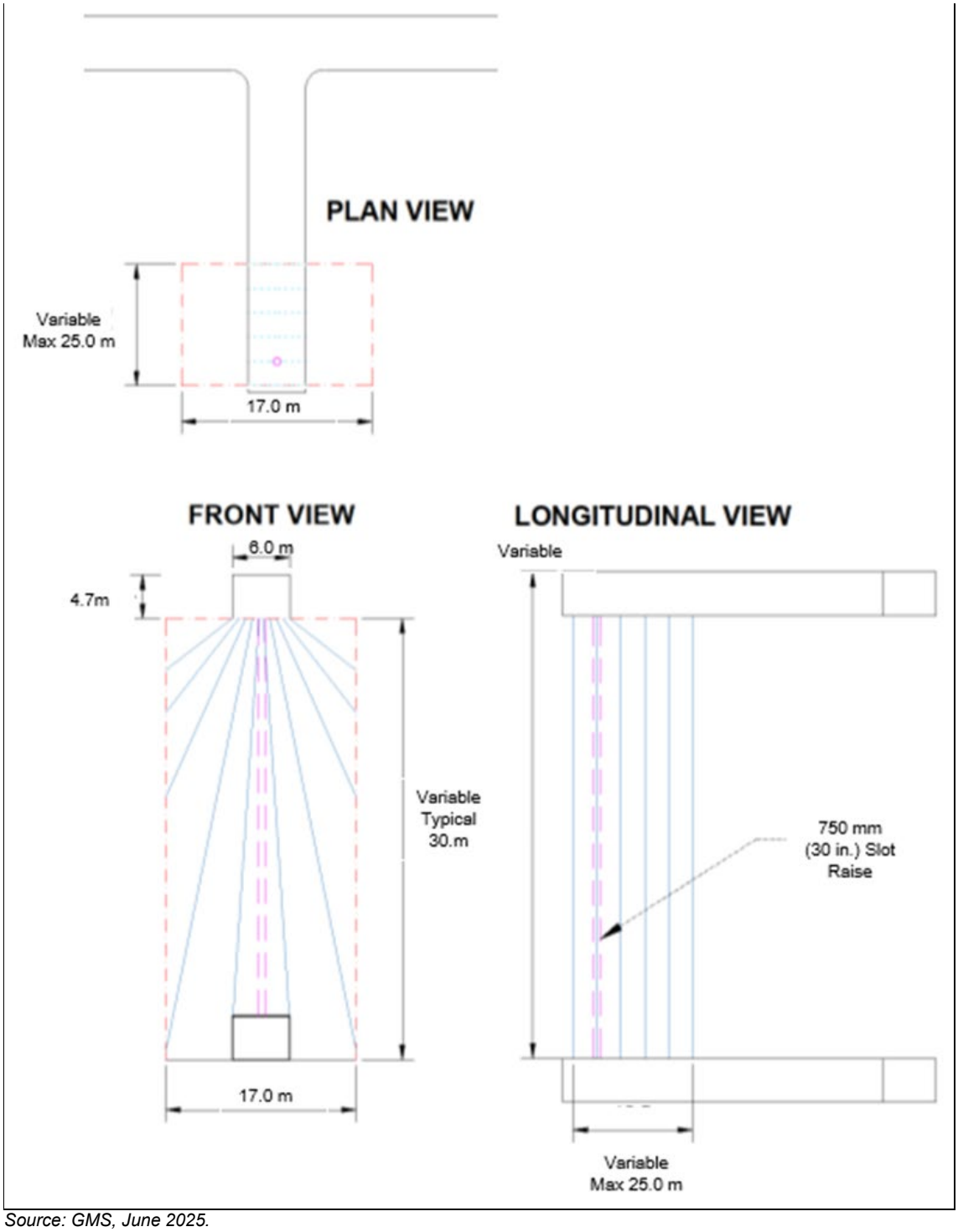
Primary stope opening (slot raises) will be created with the mobile raise boring machine, which does not require a concrete pad or any other specialized infrastructure. A 229 mm pilot hole will be drilled and enlarged to 750 mm (30 in.) with a boring head. This rig will also be used to drill the 1.20 m (42 in.) safety egress raise. Articulated carrier in-the-hole (ITH) production drill rigs, equipped with a 750 mm raising head, will also support the mobile raise boring machine when needed.

16.5.2.2.2 Production Drilling and Blasting

Production drilling operations will primarily be carried out using top-hammer drill rigs, designed to drill holes with a diameter of 102 mm (4 in.). Additionally, an in-the-hole (ITH) drill will be available to assist with production drilling when it is not otherwise occupied with the excavation of service holes or slot raises. Surface remote drilling has been factored into the drilling performance for the stopes.

The drilling pattern will follow a standardized configuration with a burden of 2.5 m and spacing of 2.8 m to ensure effective fragmentation and maximize ore recovery. Figure 16.50 illustrates a typical stope drilling section.

Figure 16.50: Typical Drilling Section for Underground Stoping

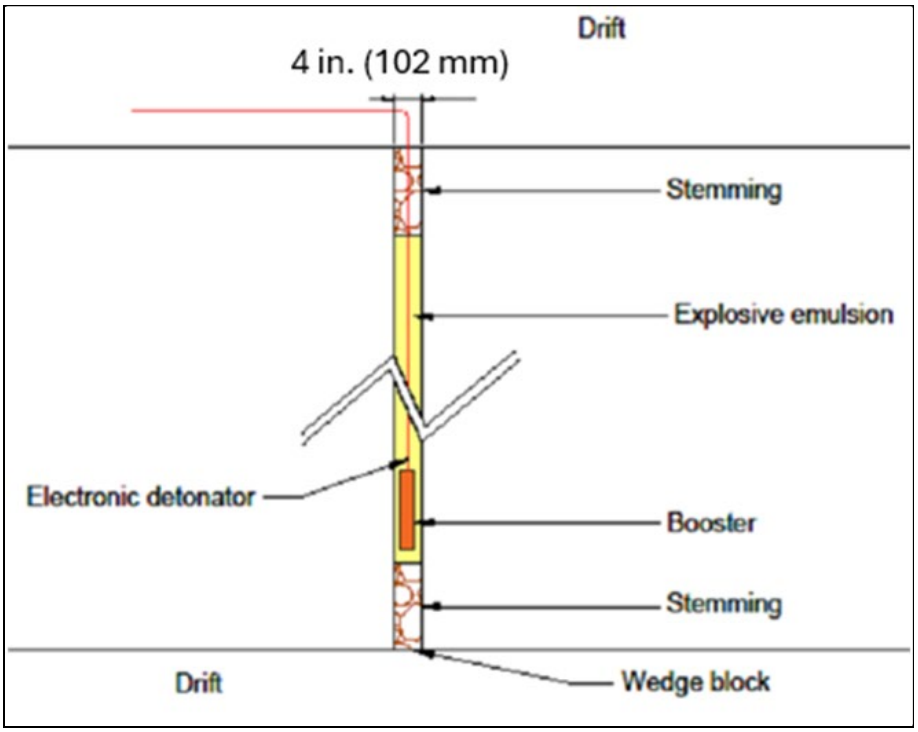


Source: GMS, June 2025.
**Note: Not to scale.*

Emulsion explosives will be used in production stopes, selected for their high energy output, water resistance, and safety advantages in underground applications. Blasts will be initiated using LTE connected electronic detonators, providing precise control over timing sequences to optimize fragmentation, minimize

vibration, and enhance overall blast performance. Figure 16.51 illustrates a typical production hole loading section.

Figure 16.51: Typical Production Hole Loading Specifications



Source: GMS, June 2025.
**Note: Not to scale.*

Each production stope will be blasted in multiple phases, typically ranging from two (2) to four (4) individual blast events. The exact number of blasts will depend on the stope design parameters and the geometry of the orebody. Table 16.23 summarizes the production drilling parameters considered in the evaluation for the evaluation.

Table 16.23: Production drilling Parameters

Parameter	Unit	Factor
Transverse Stope		
Drilling Factor	t/m	10.68
Drilling Productivity – Top Hammer	m/d	180
Drilling Productivity – ITH	m/d	120
Powder Factor	kg/t	0.76

Parameter	Unit	Factor
Longitudinal Stope		
Drilling Factor	t/m	8.43
Drilling Productivity – Top Hammer	m/d	150
Drilling Productivity – ITH	m/d	100
Powder Factor	kg/t	0.92
Upper Stope		
Drilling Factor	t/m	9.64
Drilling Productivity – Top Hammer	m/d	150
Drilling Productivity – ITH	m/d	100
Powder Factor	kg/t	0.69
Drop Raise - FAR		
Dimension	m	5.0 x 5.0
Drilling Factor	t/m	4.2
Drilling Productivity – Top Hammer	m/d	80
Powder Factor	kg/t	1.37
Drop Raise – RAR		
Dimension	m	3.0 x 3.0
Drilling Factor	t/m	2.4
Drilling Productivity – Top Hammer	m/d	80
Powder Factor	kg/t	1.52
Slot Raise		
Dimension	m	0.76
Drilling Productivity – ITH	eq. m/d	60
Drilling Productivity – Raise Bore	eq. m/d	240
Emergency Egress		
Dimension	m	1.2
Drilling Productivity – Raise Bore	m/d	15

16.5.2.3 Physicals Summary

Table 16.24 summarizes the total development statistics of the underground mine. The stoping and development tonnages and grades are presented in Table 16.25, including both ore and waste development quantities.

Table 16.24: Underground Mine Development Metres Summary

Parameters	Unit	Value
Lateral Development		
Main Decline	m	4,042
Level Access	m	1,172
Haulage Drift	m	16,203
Infrastructures	m	9,788
Sub-total CAPEX Development	m	31,205
OPEX Development	m	65,425
Total Lateral Development	m	96,631
Vertical Development		
Ventilation Drop Raise	m	818
Supported Surface Ventilation Shafts	m	100
Emergency Egress	m	441
Total Vertical Development	m	1,359

Table 16.25: Underground Mine Physical Quantities Summary

Parameters	Unit	Value
Development		
Development Waste	kt	5,165
Development Ore	kt	2,320
	%Li_2O	1.41
	Li_2O (kt)	32.6
Production		
Stoping Ore	kt	32,781
	%Li_2O	1.45
	Li_2O (kt)	475.9

Parameters	Unit	Value
Total		
Total Underground Ore	kt	35,101
	%Li_2O	1.45
	Li_2O (kt)	508.6

16.5.2.4 Development and Production Rates

The underground mine is expected to achieve an average production rate of 5,475 tpd of ore, with 5,200 tpd derived from stope production and an average of 275 tpd from lateral development. The production rate varies slightly since the quantity of development ore produced is not constant over the LOM. Multiple mining blocks will be mined simultaneously to maintain the targeted underground mine production rate. The production rate for underground mine of the Shaakichiuwaanaan Project was calculated using the Deswik™ mining sequence, considering the different rates shown in Table 16.26.

Table 16.26: Underground Mine Scheduler Rates

Parameters	Unit	Value
Single Face Development Rate	m/d/unit	5.0
Multi Face Development Rate	m/d/unit	10.0
Stope Preparation	d	5
Stope Cables	m/d	120
Slot Raise Drilling Rate	m/d	5
Production Drilling Rate – Transverse – Top Hammer	m/d	200
Production Drilling Rate – Transverse - ITH	m/d	150
Production Drilling Rate – Longitudinal & Drop Raise	m/d	150
Production Drilling Factor - Transverse	t/m drilled	10.0
Production Drilling Factor - Longitudinal	t/m drilled	8.0
Blasting Delay	d	3
Mucking Rate	t/d	1,100
Cemented Pastefill Rate	t/d	4,500
Short Cure Time	d	7
Long Cure Time	d	28
Maximum Stoping tonnage	t/d	5,200

16.5.3 Development and Production Sequencing

Following the completion of the portal excavation and construction, the Project will transition into the development phase, beginning with the advancement of the main decline toward the main zone. It is planned to use a mining contractor for the portal ground support and the pre-production development period which is estimated to be two (2) years. Following this phase, the owner will assume responsibility for all underground activities.

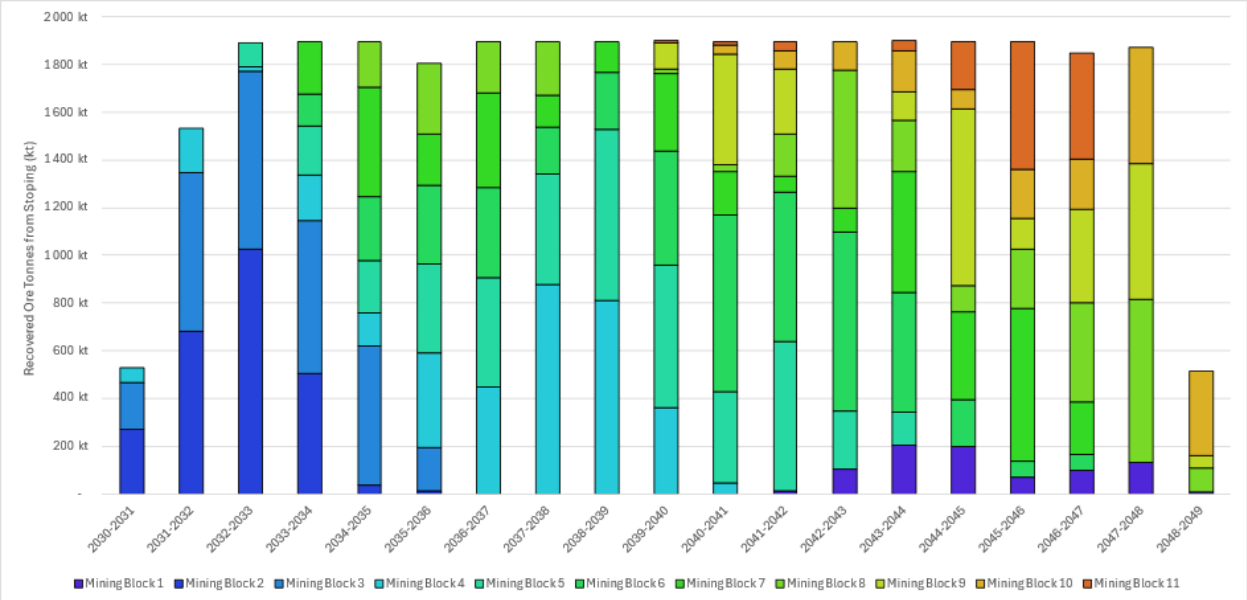
This decision is mainly based on the choice to use Load-Haul-Dump (LHD) BEV equipment. Using such equipment from the beginning is not deemed efficient, considering the initial CAPEX requirements and operational ramp-up needs (battery management, maintenance ramp-up, UG and OP site maturity, power grid establishment and availability during mine development phase, etc.). A development ramp-up was implemented during the first two (2) months of the Project. This ramp-up accounts for typically more challenging ground conditions within the first vertical 20 m in the James Bay area and operational ramp-up of workforce and maintenance.

Afterwards, the development rate is fixed at 150 linear metres per month, including first 20 m of all perpendicular excavations along the way (remucks, sumps, refuge stations, electrical substations, etc.). The underground mine will be developed by a single face team until the intersection leading to the ventilation drift. At this stage, two (2) development teams and enough mine headings will be available to ensure a multi-face approach, allowing additional advance in metres per month.

The development of the primary ventilation and safety egress networks is a priority as they are essential to allow stoping to begin. The main priority remains the development of the ramp to access the first mining horizon which is the nearest to the Eastern Pit. The goal is the completion of the stopes near the pit as quickly as possible in order not to interfere with pit operations.

Once the first mining front is mature enough to sustain production, ramp development will resume to develop the next mining blocks and access the high-grade Nova Zone, which is located in mining block number 7 (Figure 16.46). The underground mine ore from stoping, split by mining block is presented in Figure 16.52. Table 16.27 presents the LOM yearly production schedules, while Table 16.28 and Table 16.29 present the LOM yearly development schedules. The production and development schedule are reported according to a production year that begins on June 1 and ends on July 31 of each year.

Figure 16.52: Underground Mine Recovered Ore Tonnes from Stoping Split by Mining Block



Source: GMS, June 2025.
**Note: Column 2030-2031 is from July 2030 to June 2031.*

Table 16.27: Ore Production Schedule

Descriptions	Units	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Development Ore	kt	2,320	-	-	-	93	209	239	106	237	210	186	149	71	88	91	128	145	72	70	101	63	42	19	-
	Li_2O (%)	1.41	-	-	-	1.32	1.60	1.33	1.58	1.70	1.64	1.62	1.59	1.25	1.38	1.33	1.22	1.14	1.04	0.96	1.14	0.99	0.93	0.99	-
	Li_2O (kt)	32.6	-	-	-	1.2	3.4	3.2	1.7	4.0	3.5	3.0	2.4	0.9	1.2	1.2	1.6	1.6	0.7	0.7	1.2	0.6	0.4	0.2	-
Stoping Ore	kt	32,781	-	-	-	160	1,004	1,829	1,898	1,898	1,806	1,898	1,898	1,898	1,903	1,898	1,898	1,898	1,903	1,898	1,898	1,850	1,903	1,441	-
	Li_2O (%)	1.45	-	-	-	1.09	1.21	1.43	1.48	1.42	1.58	1.76	1.76	1.55	1.57	1.54	1.49	1.52	1.31	1.30	1.32	1.32	1.27	1.16	-
	Li_2O (kt)	475.9	-	-	-	1.7	12.1	26.2	28.1	27.0	28.5	33.5	33.4	29.5	29.8	29.3	28.3	28.8	24.9	24.6	25.0	24.5	24.1	16.8	-
Total Ore Underground	kt	35,101	-	-	-	253	1,214	2,068	2,004	2,135	2,017	2,084	2,047	1,969	1,991	1,989	2,026	2,043	1,975	1,968	1,999	1,913	1,945	1,461	-
	Li_2O (%)	1.45	-	-	-	1.17	1.27	1.42	1.48	1.45	1.58	1.75	1.75	1.54	1.56	1.53	1.47	1.49	1.30	1.29	1.31	1.31	1.26	1.16	-
	Li_2O (kt)	508.6	-	-	-	3.0	15.5	29.4	29.7	31.1	31.9	36.5	35.8	30.4	31.1	30.5	29.8	30.4	25.6	25.3	26.2	25.1	24.5	17.0	-

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.28: Lateral Development Schedule

Descriptions	Units	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Contractor Phase																									
Main Decline	m	2,251	-	922	1,054	275	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Level Access	m	492	-	-	335	157	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Haulage Drift	m	1,646	-	-	691	956	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructures	m	3,794	-	414	2,476	904	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-Total Capex Development	**m**	**8,184**	**-**	**1,336**	**4,556**	**2,291**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Opex Development	m	2,133	-	-	613	1,520	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Lateral Development	**m**	**10,317**	**-**	**1,336**	**5,170**	**3,811**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Owner Phase																									
Main Decline	m	1,791	-	-	-	-	680	11	189	-	-	-	-	232	-	-	-	322	358	-	-	-	-	-	-
Level Access	m	680	-	-	-	-	216	33	80	-	-	-	-	80	-	-	-	80	192	-	-	-	-	-	-
Haulage Drift	m	14,557	-	-	-	1,125	927	1,002	2,054	1,259	1,141	556	504	850	788	812	623	325	639	957	880	21	94	-	-
Infrastructures	m	5,994	-	-	-	159	845	792	1,343	439	169	153	28	337	319	288	206	372	261	219	62	-	-	-	-
Sub-Total Capex Development	**m**	**23,022**	**-**	**-**	**-**	**1,284**	**2,668**	**1,838**	**3,666**	**1,698**	**1,310**	**709**	**531**	**1,499**	**1,107**	**1,100**	**830**	**1,099**	**1,449**	**1,176**	**942**	**21**	**94**	**-**	**-**
Opex Development	m	63,292	-	-	-	2,517	5,274	5,937	4,226	5,844	5,415	3,989	3,268	2,322	2,817	2,884	3,173	2,720	2,594	2,862	2,925	2,412	1,497	616	-
Total Lateral Development	**m**	**86,314**	**-**	**-**	**-**	**3,801**	**7,943**	**7,774**	**7,891**	**7,542**	**6,724**	**4,699**	**3,799**	**3,821**	**3,924**	**3,985**	**4,003**	**3,820**	**4,044**	**4,039**	**3,866**	**2,433**	**1,591**	**616**	**-**

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.29: Vertical Development Perform by Contractor Schedule

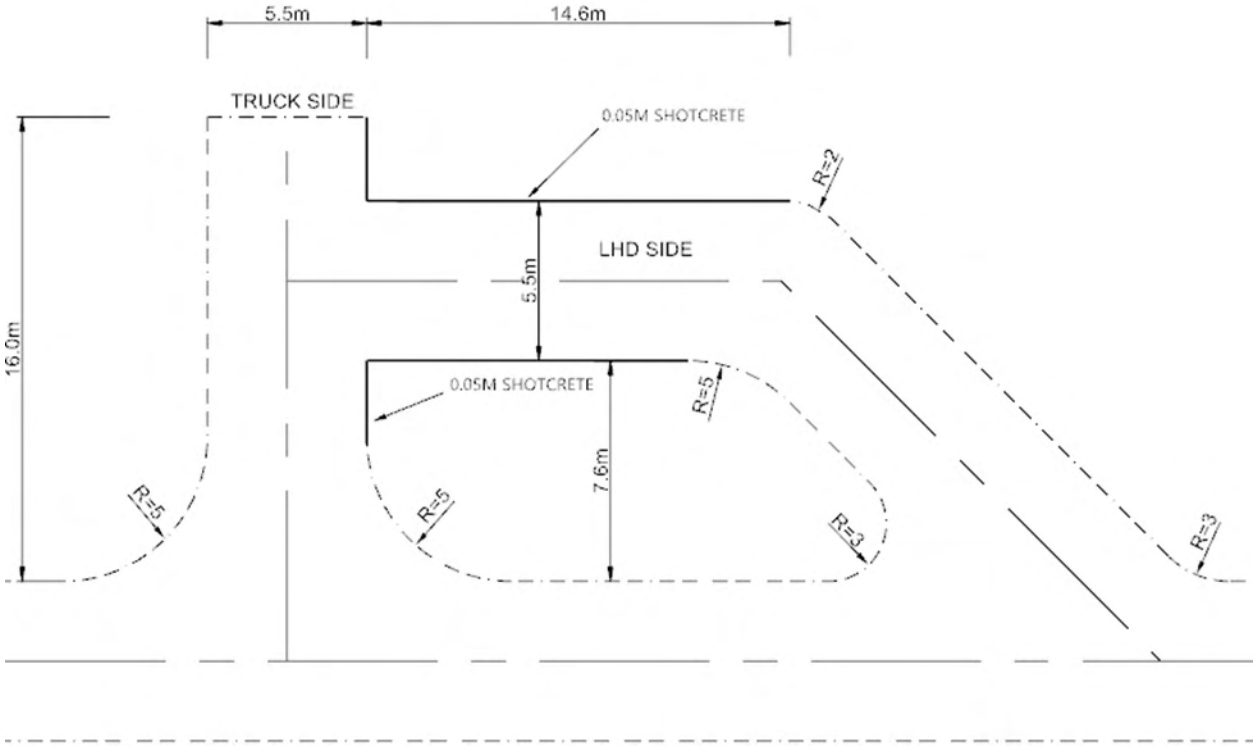
Descriptions	Units	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Contractor Phase																									
Drop Raise - FAR	m	175	-	-	88	87	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Raise Bore - FAR	m	39	-	-	-	39	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Drop Raise - RAR	m	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Raise Bore - RAR	m	61	-	61	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Emergency Egress	m	219	-	-	99	120	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-Total Raise Development	**m**	**495**	**-**	**61**	**187**	**247**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Services Holes	m	1,961	-	222	1,258	481	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Vertical Development	**m**	**2,456**	**-**	**283**	**1,445**	**728**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Owner Phase																									
Drop Raise - FAR	m	161	-	-	-	-	-	49	53	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Raise Bore - FAR	m	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Drop Raise - RAR	m	481	-	-	-	-	33	26	84	28	29	-	20	-	30	110	-	-	59	34	27	-	-	-	-
Raise Bore - RAR	m	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Emergency Egress	m	222	-	-	-	-	-	53	26	-	-	-	-	-	26	-	-	28	29	59	-	-	-	-	-
Sub-Total Raise Development	**m**	**864**	**-**	**-**	**-**	**-**	**33**	**128**	**164**	**28**	**29**	**-**	**20**	**30**	**57**	**110**	**-**	**58**	**88**	**93**	**27**	**-**	**-**	**-**	**-**
Services Holes	m	4,403	-	-	-	-	592	222	518	407	444	37	-	444	444	37	370	222	407	222	37	-	-	-	-
Total Vertical Development	**m**	**5,267**	**-**	**-**	**-**	**-**	**625**	**350**	**682**	**435**	**473**	**37**	**20**	**474**	**501**	**147**	**370**	**280**	**495**	**315**	**64**	**-**	**-**	**-**	**-**

16.5.4 Material Handling

For the Shaakichiuwaanaan Mine, 50 t underground battery electric vehicles (BEVs) haul trucks on the ramp have been selected for material handling to surface. It was determined that ramp haulage coupled with loading bays (Figure 16.53) was economically preferable to other means of ore handling, such as conveyors or dedicated haulage ramp.

To ensure efficient production, loading bays were included in the mine design at the level access and at distances of up to 350 m on production levels. In this design, most drawpoint entrances are within 175 m of a loading bay. They are designed with two (2) drifts, one (1) of them used to position the truck for loading and the second one, excavated perpendicularly and higher, to allow the LHD to load the trucks easily. It is planned to use adjacent drawpoints as temporary remuck areas to optimize loading cycles.

Figure 16.53: Typical Loading Bay - Plan View



Source: GMS, August 2025.
**Note: Not to scale.*

Trucks will travel to the surface where broken ore or waste will be unloaded on surface transfer pads near the portal. For the ore, the pad is located next to the primary crusher and surface front end loaders (FEL) will rehandle the daily production into the crusher to ensure that the correct grade blending with OP materials occurs, prior to introduction into the DMS Plant. The primary crusher grizzlies are not sized to

allow direct dumping from underground trucks. After being rehandled by a front-end loader (FEL), the waste will be transported by surface mining trucks to the appropriate waste dump.

The estimated truck fleet size has been calculated using haulage cycle times evaluated for each mining blocks and level based on the development and production plans. The mining sequence for each zone built in Deswik provides the timing and location of tonnage to be hauled with an adequate level of precision. The truck fleet size was then confirmed by an OEM independent assessment using a proprietary software.

A standard work schedule was established to determine the number of operating hours per year. It considers major unplanned events and scheduled delays during a shift (travelling, workplace and equipment inspection, supervision, lunch). Typically, a total of 4,020 net operating hours (NOH) per year are available for effective trucking. Table 16.30 presents the parameters used in the hauling study. Speeds when going downhill and on flat ground have been limited to 15 km/h for safety purposes.

Table 16.30: Truck Parameters for Hauling Studies

Equipment Capacity	Truck - 50 T - BEV
Box Volume (m^3)	28
Fill Capacity (%)	90
Material Density (t/m^3)	1.96
Truck Capacity (t)	44.6
Loading (s)	517
Unloading (s)	30
Delays (s)	180
Speed Up & Loaded (km/h)	12.4
Speed Down & Empty (km/h)	15
Speed on Flat Ground (km/h)	15
Mechanical Availability (%)	85
Yearly Hour Manual Mode	4,020

16.5.5 Underground Mine Equipment

The underground equipment requirements were established based on the estimated number of operating hours necessary to achieve the production and development targets defined in the mine plan.

During the production phase, haulage cycle analyses accounted for the travel distances from the loading points located in the footwall drifts to the level accesses, followed by ramp ascent and surface hauling to the surface stockpile. Mucking and hauling cycles were evaluated based on the average distance between the stopes and the designated truck loading locations. Underground haul trucks and the 18-t LHDs were selected as battery electric vehicles (BEVs). Benchmarks have been done among Canadian operations to validate operation readiness of BEV fleets. The review established that BEVs are now becoming commonplace in UG mines and implementing such technologies is no longer experimental. This selection was based on an economic analysis showing that the reduced impact on OPEX costs outweighs the higher CAPEX expenses only for the load and haul equipment. To reduce the environmental footprint, the grader was also selected as BEV equipment.

The required quantities of auxiliary equipment were determined according to the overall scale of the operation and through analysis of primary equipment demands. Table 16.31 summarizes the resulting maximum equipment requirements for the contractor phase, while Table 16.32 and Table 16.33 present the owner's equipment purchase schedule and the requirement schedule, respectively.

Table 16.31: Underground Contractor Mine Mobile Equipment Fleet Requirements

Equipment Type	Maximum
Jumbo – 2 Boom	2
Bolter	5
Production Drill – (In-The-Hole)	1
LHD – 15 T	2
Truck – 45 T	3
Excavator	1
Telehandler	1
Jeep Supervision	5
Jeep Maintenance	2
Surface Loader	1
Grader (contractor)	1
Total UG Mobile Equipment	**24**

Table 16.32: UG Mobile Equipment Purchase Schedule

Equipment Type Purchase	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Jumbo – 2 Boom	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Bolter	5	-	-	-	4	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Drill – Top Hammer	6	-	-	-	1	1	1	-	-	-	-	-	-	-	-	-	-	-	2	1	-	-	-	-
Production Drill - In-The-Hole	2	-	-	-	1	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
Cable Bolter	2	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-
LHD – 18 T BEV	4	-	-	-	2	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LHD – 10 T	3	-	-	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-
Truck – 50 T BEV	6	-	-	-	4	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Explosive Truck - Development	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Explosive Truck - Production	2	-	-	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Scissor Lift - Development	4	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	2	-	-	-	-	-	-	-
Scissor Lift – Construction + Paste	6	-	-	-	2	-	1	-	-	-	-	-	-	-	-	-	2	-	1	-	-	-	-	-
Boom Truck	6	-	-	-	1	-	1	-	-	-	-	1	1	-	-	-	-	1	1	-	-	-	-	-
Fuel & Lube Truck	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Truck	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Block Holer	2	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-
Grader	3	-	-	-	1	-	-	-	-	-	-	-	1	-	-	-	-	-	-	1	-	-	-	-
Light Vehicle	68	-	-	-	6	-	12	-	2	-	6	12	-	2	-	6	12	-	2	-	6	-	-	-
Light Vehicle - Mine Rescue	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Light Vehicle - Mechanics	4	-	-	-	2	-	-	-	-	-	-	-	-	-	-	2	-	-	-	-	-	-	-	-
Tractor - Electricians	4	-	-	-	2	-	-	-	-	-	-	-	-	-	-	2	-	-	-	-	-	-	-	-
Telehandler	4	-	-	-	1	-	1	-	-	-	-	-	-	-	-	1	1	-	-	-	-	-	-	-
Mobile Air Compressor	2	-	-	-	1	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
Backhoe Loader	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Raise Bore (include head 1.2 m head reamer)	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Equipment Type Purchase	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Reamer 30 inch	4	-	-	-	1	-	1	-	-	-	-	-	-	-	-	1	1	-	-	-	-	-	-	-
Jumbo Bolter	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total UG Mobile Equipment Requirement	**44**	**-**	**2**	**1**	**44**	**5**	**20**	**-**	**2**	**-**	**6**	**13**	**2**	**4**	**-**	**12**	**19**	**1**	**7**	**3**	**6**	**-**	**-**	**-**

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.33: UG Mobile Equipment Requirement Schedule

Equipment Type Requirements	Max	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Jumbo – 2 Boom	2	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	-
Bolter	6	-	-	-	4	6	6	5	5	5	4	4	4	4	4	4	4	4	4	4	3	3	3	-
Production Drill – Top Hammer	3	-	-	-	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	-
Production Drill - In-The-Hole	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Cable Bolter	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
LHD – 18 T BEV	5	-	-	-	2	4	5	5	5	5	5	4	4	4	4	4	4	4	4	4	4	4	3	-
LHD – 10 T	2	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Truck – 50 T BEV	9	-	-	-	4	8	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	6	-
Explosive Truck - Development	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Explosive Truck - Production	2	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Scissor Lift - Development	2	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Scissor Lift – Construction + Paste	3	-	-	-	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-
Boom Truck	2	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Fuel & Lube Truck	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Water Truck	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Block Holer	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Grader	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Light Vehicle	20	-	2	2	8	8	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	-
Light Vehicle - Mine Rescue	1	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-

Equipment Type Requirements	Max	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Light Vehicle - Mechanics	2	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Tractor - Electricians	2	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Telehandler	2	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Mobile Air Compressor	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Backhoe Loader	2	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Raise Bore (include head 1.2 m head reamer)	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Reamer 30 inch	2	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Jumbo Bolter	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

**Notes: Column 2027-2028 is from July 2027 to June 2028.*

16.5.5.1 Battery Electric Vehicles (BEVs) Infrastructure and Operating Conditions

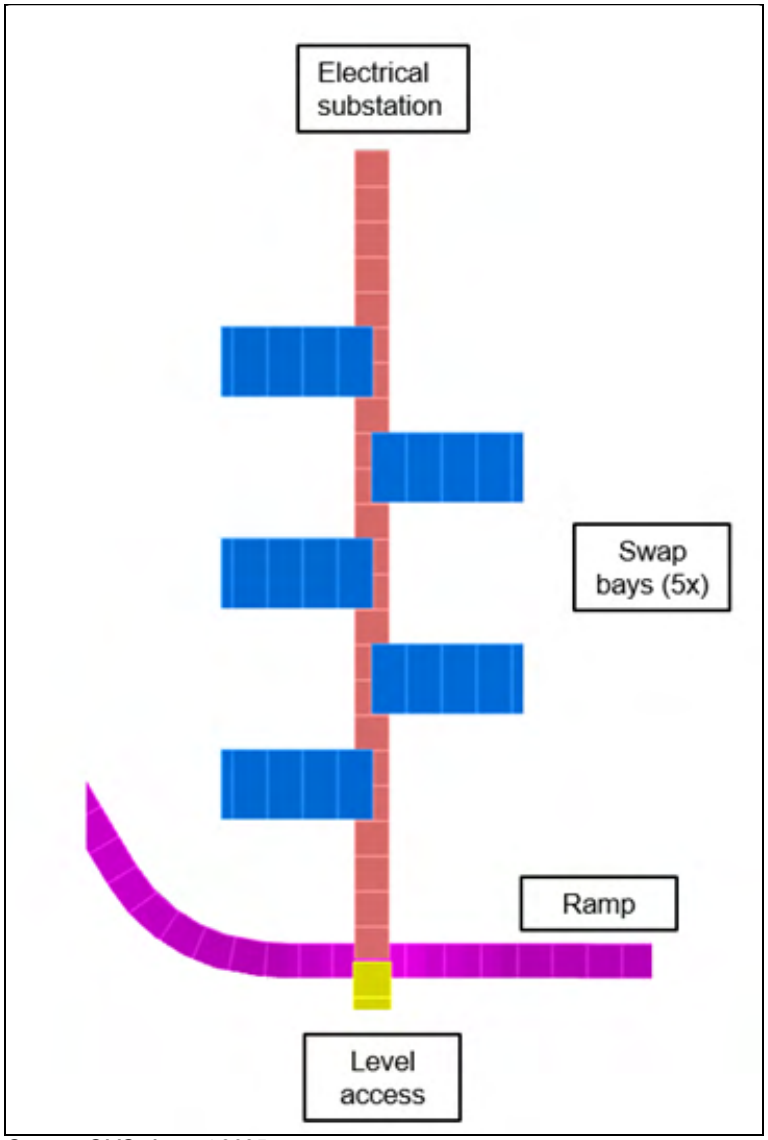
This section follows OEM recommendations. The machine operates normally when the battery's internal temperature remains within the standard range of +5°C to +50°C. If the internal temperature moves outside these limits, battery power output will be reduced to protect the cells from unsafe operating conditions and premature wear, causing the machine to run at reduced power.

To ensure maximum lifetime and performance for the BEV, charging stations have been planned underground where the ambient temperature is much more constant year-round. Selected BEV haul trucks and LHDs are based on a swapping principle where the depleted battery is unhooked from the equipment, placed next to a charger and replaced with a fully charged unit. The bay is designed to ensure that there is sufficient space to park the machine when idle and to carry out battery swaps. Adequate ventilation is needed in all bays. Cooling units can generate up to 65 kW of heat during operation, so ventilation will help maintain ambient temperatures within specification for operating equipment. Two (2) charging units and one (1) cooling unit are recommended per equipment.

Underground charging bays require a strong LTE signal, levelled concrete floors, adequate turning and working space, and a cable management plan. Figure 16.54 illustrates a typical layout for the BEV haul trucks charging bays. This infrastructure includes five (5) independent swap bays along with an electrical substation to handle the dedicated electrical power. Within the mine design, three (3) strategically located charging points, totalling 15 swap bays, have been planned to accommodate the truck fleet size and to optimize charging opportunities.

The charging points are located in front of the production level and directly connected to the ramp. This serves a dual purpose, acting as both the focal point of the truck's haulage path and the mine ventilation exhaust helping eliminate as quickly as possible the generated heat from the chargers. For the LHDs, 11 charging points are located directly in the production-level haulage drift near the level access. This layout minimizes travel time for charging while keeping the units close to the main level electrical substation.

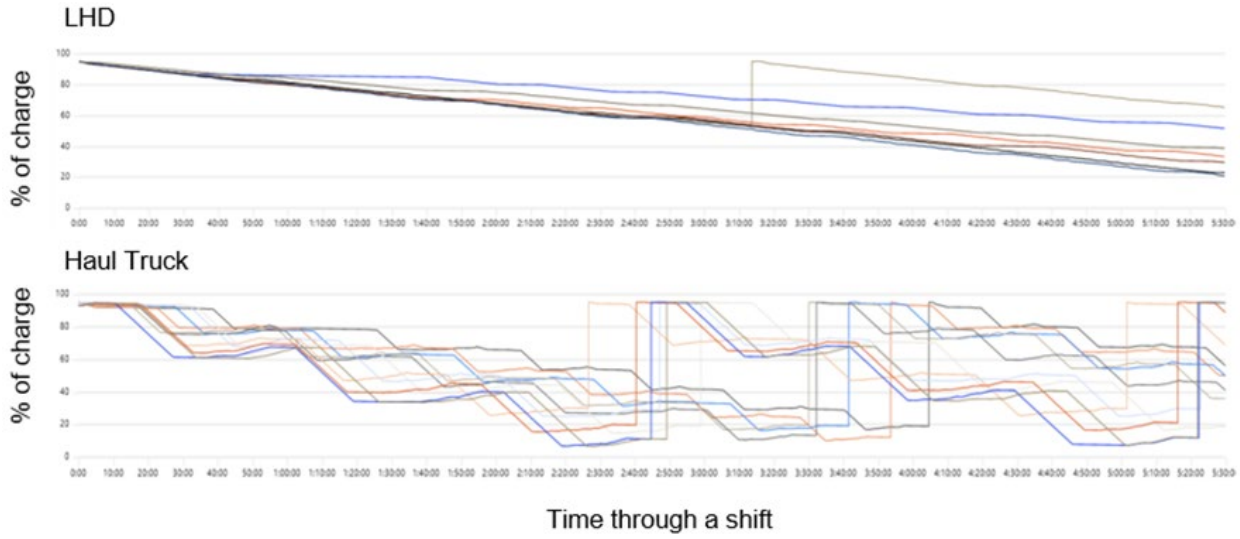
Figure 16.54: Charging Bays Layout



Source: GMS, August 2025.
**Note: Not to scale.*

Figure 16.55 shows an OEM model of the BEV haul trucks and LHDs, illustrating the battery charging cycles during an 8-hour shift. For a representative production month over the life of mine, the model suggests only one (1) or no battery swap per LHD per shift, and between two (2) and three (3) battery swaps per truck to sustain development and production needs. Battery regeneration while driving downhill on the ramp is also clearly visible in the truck section. Charging time from a 20% to a 100% state of charge is estimated at between 2.5 and 5.0 hours, depending on local conditions.

Figure 16.55: BEVs Haul Trucks and LHDs Modelled Battery Charging Cycles Through Shift



Source: Sandvik, August 2025.

To provide support for underground infrastructure, one (1) bay in the surface maintenance garage will be dedicated to battery maintenance. This bay will be equipped with a 15-t overhead crane, with sufficient lift clearance for battery packs weighing up to 11,000 kg. Battery management has been considered under a "battery-as-a-service" model**,** where the OEM provides batteries, maintenance, spare parts and waste management for the entire life of mine.

16.5.5.2 Automation

There are no automation-ready 50-t battery electric vehicles (BEVs) haul trucks currently available on the market. Two (2) different stages of automation were evaluated for LHDs:

- Fully manual operation. No operation within blast windows.
- Semi-automated (tele-operated from surface) operation within blast windows only for the development fleet.

It has been estimated that the utilization of surface tele-remote during blast windows, when fully installed and operational, do not fully translate into higher productivity. Without the ability to haul material out of the mine and remucks adjacent to loading bays having limited storage capacity that would rapidly be filled within blast windows, only the first several trucks of the shift would see the benefit of that fully loaded remuck.

Lack of available surface teleoperation for trucks and the shallow underground mine design (resulting in short transportation times) do not justify the additional investments required by surface tele-remote for the

production LHDs at this stage. However, surface tele-remote technology should be implemented on high-priority headings to increase achieved monthly advance rates.

The following different drilling modes have also been investigated:

- Normal with an allocated operator on drilling site.
- Surface tele-remote without the possibility of moving the equipment from ring to ring.
- Fully Automated, surface tele-remote + Ring-to-Ring / short tramming automation.

Based on the analysis, adding the ability to move the drill from ring to ring does not bring significant additional value to the Project. On the operational point of view, the probability that a UG drill finishes the last hole of a ring within the blast window and needs to move is quite low. It can reasonably be assumed that this specific condition would not happen very often and does not represent significant operational delays. However, drilling during blast windows represents a significant improvement and has been included in the equipment selection.

16.5.6 Underground Mine Ventilation

CGM Expert Inc. conducted an evaluation of the ventilation requirements for the proposed underground mine. This section provides a summary of their findings.

The ventilation requirements for the underground mine were established based on three (3) key factors. For BEVs, a minimum air speed of 0.25 m/s at the workplace was applied as a requirement to meet all regulations. In addition, the equipment manufacturer provided minimum air flow to dissipate the heat generated by the equipment while charging. For diesel-powered equipment, Canmet dilution factors were used to determine the required airflow per equipment type, in alignment with established Canadian best practices and Quebec Regulation respecting occupational health and safety in mines. Finally, in cases where the engine in the equipment is not Canmet approved, a minimum dilution rate of 0.06 m^3/kW was adopted to ensure adequate ventilation coverage. Finally, the mine's ventilation was adjusted to optimize the gas clearance time after blasting.

To further refine airflow requirements, an equipment utilization factor was applied, reflecting the expected duration and intensity of equipment operation throughout the mine's production cycle. This factor was derived from operational assumptions and typical duty cycles for each equipment category, and it was used to calculate the required airflow in CFM (Cubic Feet per Minute) to maintain compliance with occupational exposure limits. The air requirement has been established considering equipment requirements on an

annual basis. The worst-case air quantity established for a given year has been used to establish the fresh air requirement for the Project (Table 16.34).

A contingency of 10% was applied on the total estimated fresh air requirements to account for additional equipment that may be added during the life of mine, additional personnel and any potential system leaks. Ventilation network models were developed using Ventsim™, and simulations were performed to evaluate system performance under the maximum production scenario. These simulations validated that the proposed ventilation system can deliver sufficient airflow to all active working areas while maintaining regulatory compliance with respect to air quality. Heat stress was discounted from the design constraints as the mine is shallow.

Table 16.34: Underground Peak Production Mine Fresh Air Requirements per Equipment

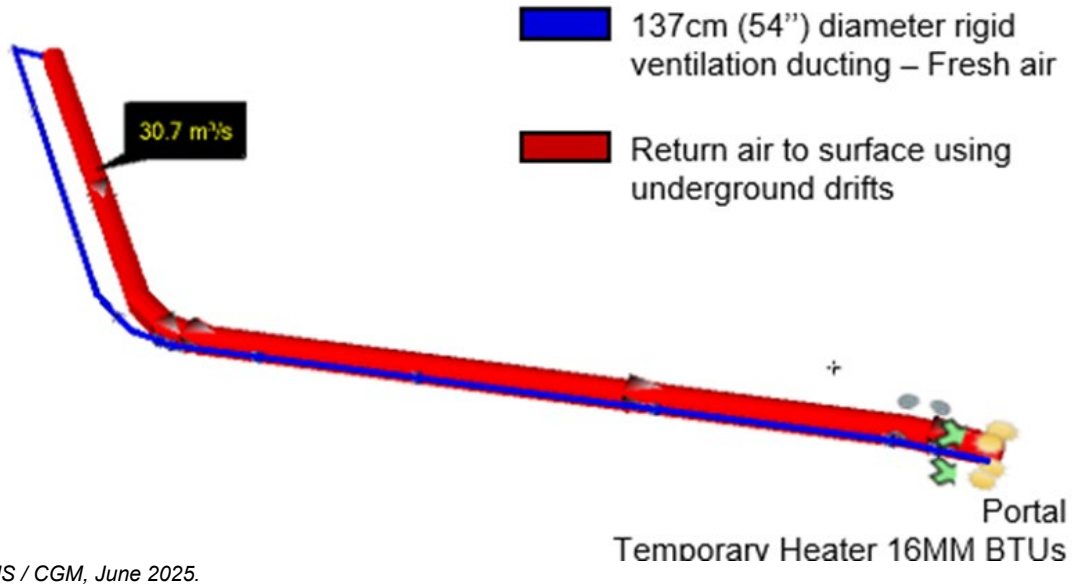
Equipment List	Recommended Airflow per Unit		Qty	Vent Requirements (fan sizing)		
	(m^3/s)	(kcfm)		(%use)	(m^3/s)	(kcfm)
Jumbo	3,3	7,0	4	35%	4,6	9,8
Bolter Unit (Scissor bolter)	4,3	9,2	4	35%	6,1	12,8
LHD (electric)	7,6	16,0	8	60%	36,5	76,8
Production Drill	4,3	9,2	5	35%	7,6	16,0
DEV - Emulsion Loader	4,3	9,2	3	35%	4,5	9,7
55-t HTR (electric)	7,6	16,0	7	60%	31,8	67,3
Boom Truck	4,3	9,2	6	50%	13,0	27,5
Grader (electric)	7,6	16,0	1	30%	2,3	4,8
Service Scoop 3 yd	5,7	12,0	3	50%	8,5	18,0
Tractor	2,3	4,8	6	50%	6,8	14,4
Scissor Lift	4,3	9,2	5	50%	10,8	22,9
Landcruiser	3,7	7,9	15	50%	28,0	59,3
Allowance for Leakage	25,0	53,0	1	100%	25,0	53,0
Sub-Total					185,2	392,3
Contingency (10%)					18,5	39,2
TOTAL					204,0	432,0

A phased approach will be employed for the implementation of the mine ventilation system. Initially, a temporary ventilation setup will be installed to supply fresh air and support the development of the main decline until the permanent ventilation network is constructed and commissioned.

There will be two (2) stages of temporary ventilation prior to the commissioning of the permanent system. In both stages, ventilation ducting will be extended from the temporary fan(s) to the working faces along the decline, with exhaust air returning via the decline.

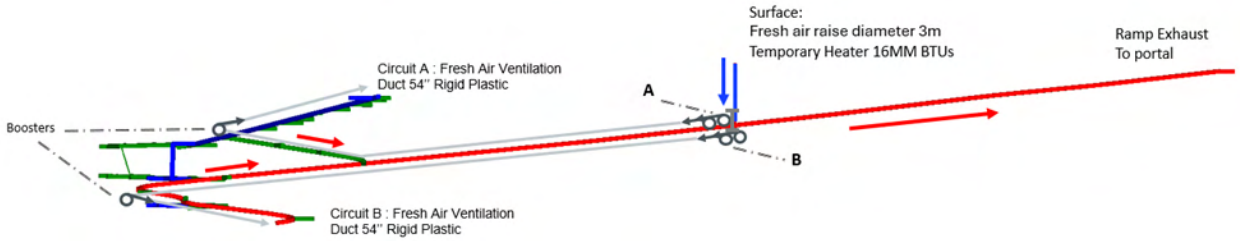
- Stage 1 involves installing a temporary ventilation system at the portal to support the initial development of the decline (Figure 16.56).
- Stage 2 will be implemented after approximately 620 metres of underground advancement, at which point a 3.0 m fresh air raise (FAR) will be developed. A box cut will be excavated to allow the FAR to be developed directly into competent fresh rock. This will allow the transition of fresh airflow from the portal intake to the FAR, thereby ensuring sufficient ventilation to continue development at greater depth while the main mine ventilation infrastructure is being developed. Two (2) fresh air ducting circuits will deliver fresh air on the upper and lower portions of the mine (Figure 16.57). The development FAR will be excavated using the Alimak raise excavation method with dimensions of 3.0 m x 3.0 m.

Figure 16.56: Underground Mine Ventilation Network – Stage 1 – Temporary Ventilation Set-up – Isometric View from North-West



Source: GMS / CGM, June 2025.
**Note: Not to scale.*

Figure 16.57: Underground Mine Ventilation Network – Stage 2 – Temporary Ventilation Set-up – Isometric View from North-West



Source: GMS / CGM, June 2025.
**Note: Not to scale.*

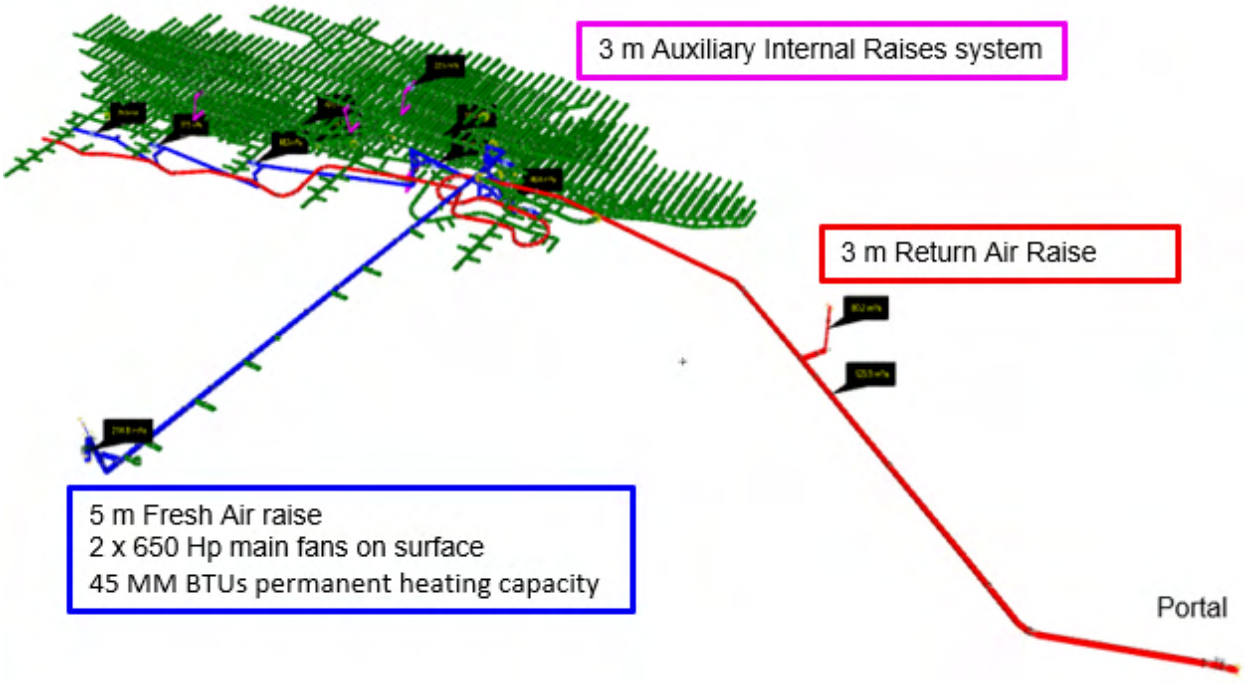
The permanent ventilation system (Figure 16.58) is designed to meet the airflow demands of the ramp-up period and the full production as well as providing an efficient blast clearance time. The proposed permanent system is a push-type ventilation system that will utilize one (1) 5.0 m fresh air raise (FAR) intake, located further north of the orebody but near main infrastructures such as the cemented pastefill plant and the process plant.

A box cut will be excavated to allow the FAR to be developed directly into competent fresh rock. The FAR will be developed using a first pass Alimak raise of 3 m allowing the utilization of a lowering platform to increase the diameter to 5 metres while installing ground support.

Two (2) primary fans of 650 Hp each will be installed in parallel on surface along with a 45 MM BTUs permanent heating capacity, supplying airflow into the mine via the FAR. A transfer ventilation drift, running from the FAR toward the internal ventilation raises and drifts, will distribute air to the lower levels of mine.

While development headings will have either rigid plastic or flexible ducting, each level will have several internal auxiliary ventilations raises allowing natural airflow without air duct as mining progresses from East to West. When the main ventilation system is commissioned, the development FAR will be converted and used alongside the portal as the mine exhausts.

Figure 16.58: Underground Mine Ventilation Network – Permanent Set-up – Isometric View from North-West



Source: GMS / CGM, June 2025.
**Note: Not to scale.*

The permanent ventilation system will operate under varying pressures and airflow rates to accommodate the different operating conditions within the mine. Ventilation louvers will be installed at the fresh air raise (FAR) access points on each level to ensure that an adequate volume of fresh air is distributed to the appropriate working areas. Table 16.35 summarizes the underground ventilation details. Gas monitoring probes and flowmeters will be installed along the fresh air and exhaust systems ensuring real time ventilation monitoring of the gas evacuation following the daily underground blasts. This system will ensure the safety of the workers while optimizing the downtime in between shifts. Real-time data monitoring will be used in an iterative process, optimizing the system performance.

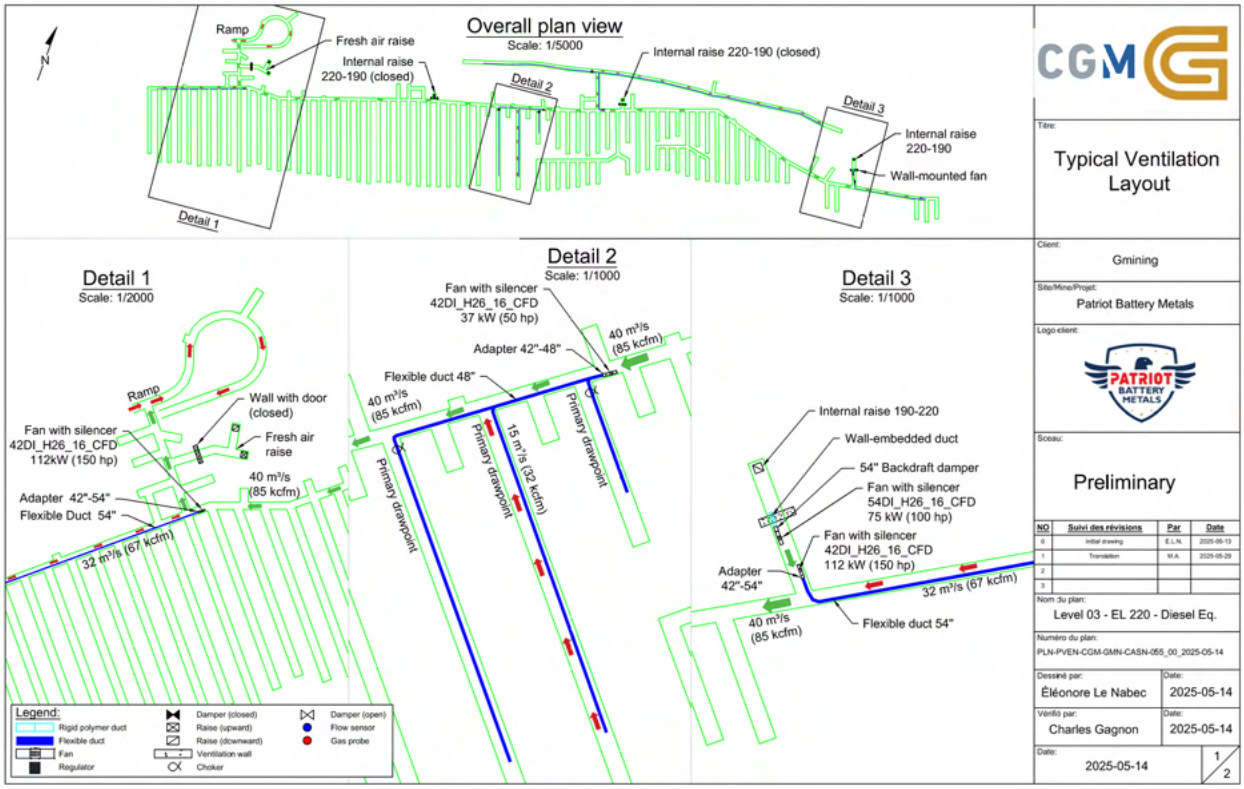
Table 16.35: Underground Mine Ventilation System Details

Parameters	Units	Value
Stage 1: Portal		
Fans	HP	150
Pressure	In. wg.	8.7
Airflow	cfm	65,000
Stage 2: Temporary Fresh Air Raise		
Fans	HP	2 x 150
Pressure	In. wg.	6.6
Airflow	cfm	65,000
Permanent Fresh Air Raise System		
Fans	HP	2 x 650
Pressure	In. wg.	6.2
Airflow	cfm	454,000

When needed on a production level, a 150-hp auxiliary fan will be installed to provide the required airflow, while a 50-hp fan will be installed at drawpoints.

Although 40-hp fans would be sufficient to ventilate the drawpoints, considering the required CFM, 50-hp fans were planned to reduce delays associated with gas clearance after blasting. Based on software simulations, the time required to clear the gases would be 90 minutes with a 40-hp fan. This time decreases to 58 minutes when using 50-hp fans. Main 150-hp level fans will be equipped with variable frequency drives (VFDs) and remote-control features, allowing adjustment from the surface control room if needed. Permanent ventilation walls and louvers will be simple, constructed with shotcrete and wooden planks, and will be manually adjusted when needed. Only two (2) pneumatic doors for equipment, constituting an airlock system (SAS doors), are planned to isolate the ventilation drift from the main system. No VFDs are planned for small auxiliary fans such as those installed at drawpoints and other crosscuts. Figure 16.59 presents a production level typical ventilation layout.

Figure 16.59: Typical Ventilation Layout



16.5.6.1 Underground Mine Heating

The mine is located in northern Quebec, Canada, where winter temperatures can average as low as -20°C (Table 16.36). A heating system supplied by natural compressed gas is planned to maintain an average air temperature of approximately 4°C within the underground workings. This will help prevent water from freezing and provide a more comfortable working environment for personnel. The system is designed with an average peak heating capacity of approximately 7.42 MJ/s during the coldest periods.

Table 16.36: Temperature Numbers Recorded at La Grande 4 Station

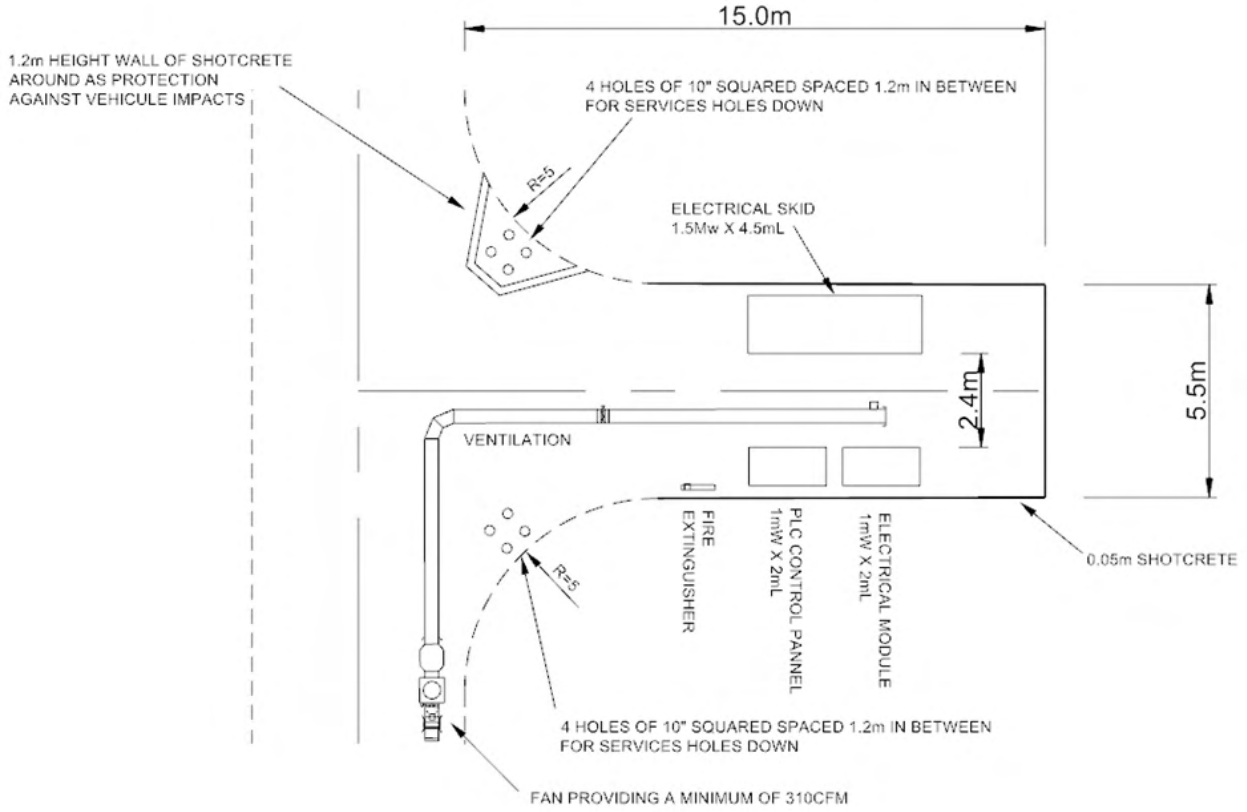
Month	Average Low (°C)	Average High (°C)	Average Temp. (°C)
January	-24.8	-14.2	-19.6
February	-27.2	-13.5	-20.4
March	-20.6	-6.2	-13.5
April	-9.5	2.7	-3.4
May	-1.5	11	4.8
June	5.7	19	12.4
July	9.4	21	15.2
August	8.5	19.7	14.1
September	4.6	14.6	9.6
October	0	7.8	3.9
November	-8.2	-2.1	-5.1
December	-17.4	-8.9	-13.2

16.5.7 Underground Mine Services

16.5.7.1 Underground Electrical Distribution

Development of the underground mine will start before the connection of Hydro-Québec's 315 kV overhead transmission system originating from the Tilly substation, located near the LG-4 Hydro-Québec Dam. Until the connection, diesel gensets will be used to generate power. Once the overhead transmission line is completed, the electric power will come from the main substation near the processing plant and will supply the underground mine via a 13.8 kV, 60 Hz distribution network, stepped down to 600 V for underground operations in electrical bays (Figure 16.60). Power will be delivered through the decline via electrical cables, and the first permanent electrical substation is planned approximately 275 metres into the decline development.

Figure 16.60: Typical Plan View of Electric Bay



Source: GMS, June 2025.
**Note: Not to scale.*

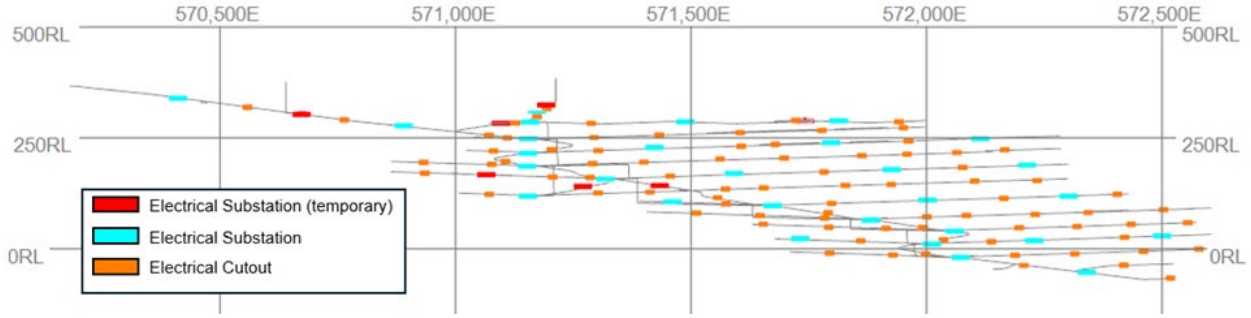
As development progresses, an electrical cutout will be installed 200 metres beyond the first substation to house power take-off (PTO) for supplying electrical power to nearby mobile equipment. This will be followed by a temporary substation located 150 metres after the cutout. Once the fresh air raise (FAR) is commissioned, a secondary electrical line will be routed through it. This redundant line will serve as a backup in the event of a failure or planned maintenance on the primary line.

As the decline advances toward the main zone, an electrical substation will be installed at each level access, which will also host other key infrastructure such as pumping stations and battery swap bays. When possible, as in the upper portion of the ramp development, electrical skid will be temporarily positioned in electric bays to sustain development and production and then moved to another location when the area is no longer active.

Most of the production levels will remain active for the majority of the life of mine, and electrical skids will need to remain in place. In the main production area, an electrical bay will be developed at each of the three (3) production levels. Each substation will be configured to supply power to its respective level as well as to the adjacent upper and lower levels. Electrical cutouts will also be excavated along haulage drifts. A

schematic view of the underground electrical distribution network is presented at Figure 16.61. Electrical cables supplying these levels will be routed through dedicated service holes. More details about the underground power distribution are presented in Section 18.

Figure 16.61: Underground Electrical Distribution Network – Looking North



Source: GMS, June 2025.
**Note: Not to scale.*

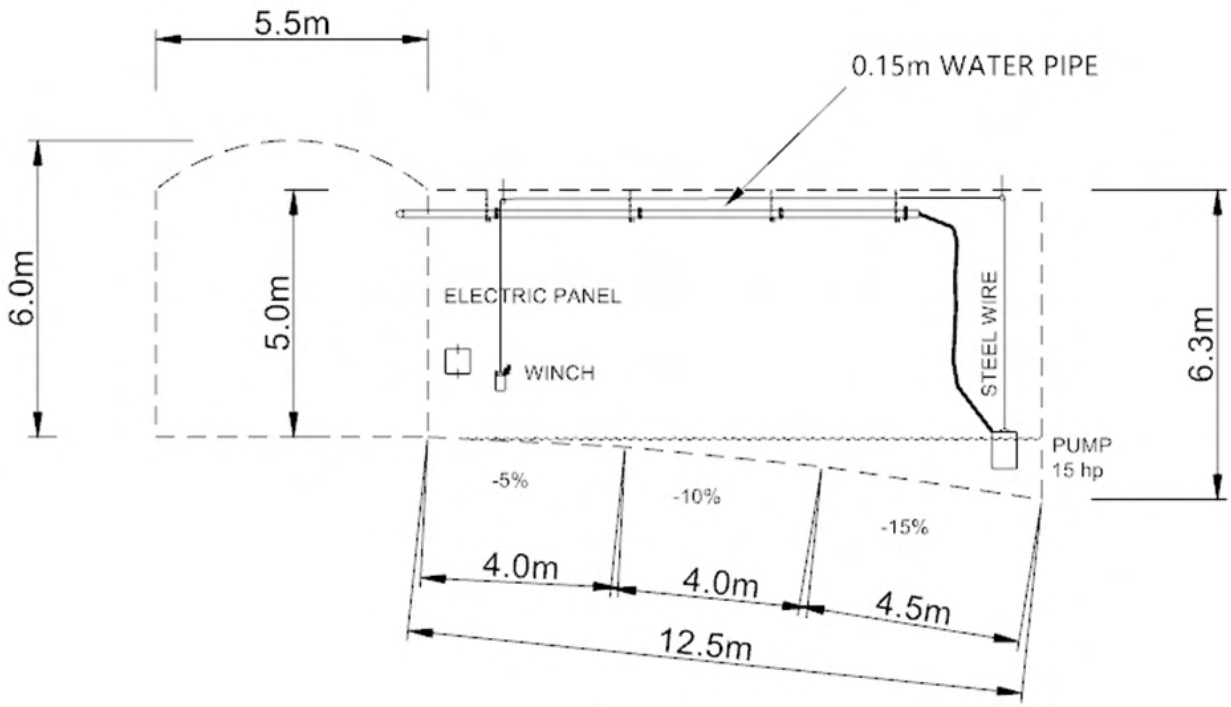
16.5.7.2 Industrial Water

Two (2) sedimentation ponds are planned as part of the Project to remove suspended solids from water collected from all mining activities, which will be transferred using pumping systems. The sedimentation ponds will discharge into the polishing pond by gravity. The polishing pond constitutes the final stage of the treatment process and acts as a freshwater reservoir for the underground mine, the process plant, and the paste plant. More details about industrial water are available in Section 18.

16.5.7.3 Dewatering Underground

A mine water balance was completed for the entire site, combining water generated from operations as well as natural groundwater inflow assumptions. Mine operations water consumption was estimated using the equipment list and the associated water usage per unit. For most of the mine blocks, water from all production levels will flow by gravity to the lowest point of each level the sump (Figure 16.62) located near the entry and then will be pumped to the decantation bay located at mid-mine elevation.

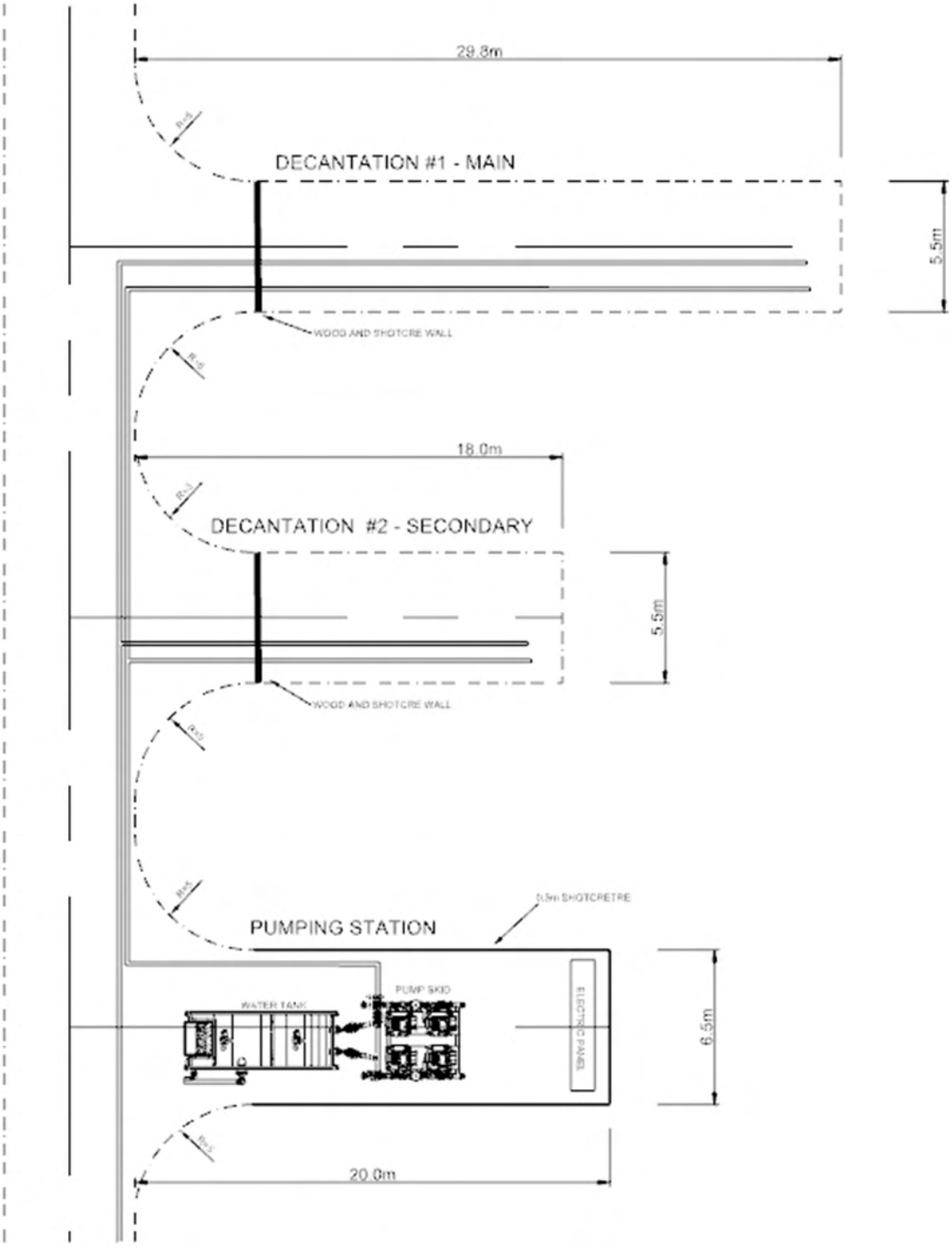
Figure 16.62: Typical Sump Located in Level Access



Source: GMS, June 2025.
**Note: Not to scale.*

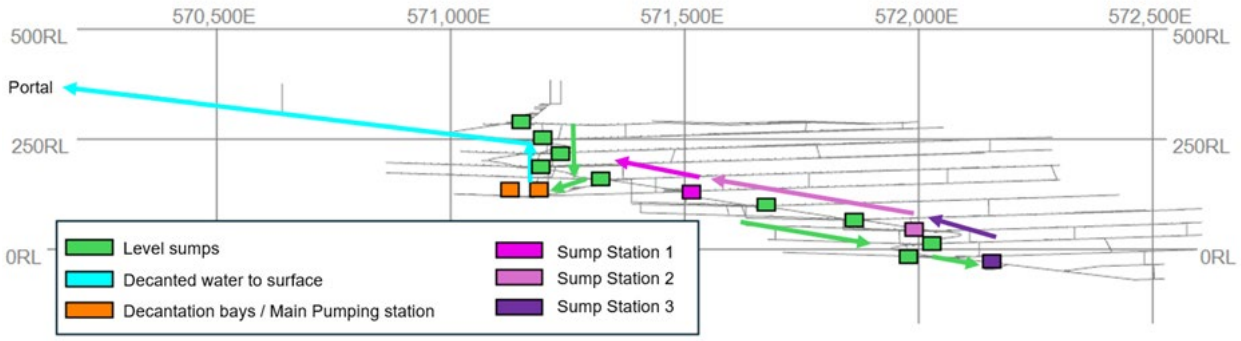
Sedimentation pond Water stored in the surface pond will then be reused as process supply for mine operations. Table 16.37 and Table 16.38 show the pumping requirements and assumptions.

Figure 16.63: Typical Underground Decantation and Pumping Bays



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.64: Underground Pumping Network – Looking North



Source: GMS, June 2025.
**Note: Not to scale.*

Table 16.37: Underground Mine Dewatering Assumption

Parameters	Unit	Value
Mine Operations Water	USGPM	840
	(l/min)	3,178
Natural Ground Water	USGPM	899
	(l/min)	3,403
Total Dewatering	USGPM	1,738
	(l/min)	6,581

Table 16.38: Underground Mine Pumping Requirements Details

Parameters	Units	Value
Main Pumping Station		
Pipe Length	m	2,650
Dewatering Flow Capacity	USGPM	1,738
Total Head	m	312
Pump Efficiency	%	75%
Pump Power	hp	1,000
Sump Station 1		
Pipe Length	m	471
Dewatering Flow Capacity	USGPM	1,738
Total Head	m	83
Pump Efficiency	%	0.75
Pump Power	hp	288

Parameters	Units	Value
Sump Station 2		
Pipe Length	m	437
Dewatering Flow Capacity	USGPM	869
Total Head	m	63
Pump Efficiency	%	1
Pump Power	hp	150
Sump Station 3		
Pipe Length	m	673
Dewatering Flow Capacity	USGPM	435
Total Head	m	91
Pump Efficiency	%	1
Pump Power	hp	108

16.5.7.4 Cemented Paste Backfill

Cemented paste backfill (CPB) has been selected as the primary backfilling method for the underground mine. The CPB mixture consists of filtered tailings, water, and binder. The cured CPB is required to support the mining cycle and to ensure local structural integrity to mined-out areas.

Backfilling requires confinement of backfill in stopes, with a containment structure required at drawpoint(s) to retain backfill until strength gain occurs after which point the paste becomes self-supporting. Most paste backfill operations use either a shotcrete arched barricade or a waste rock berm to be able to safely contain the CPB material. Barricades are designed to support the full fluid load induced by the “plug” pour. Following the cure of the plug pour, the body pour can be placed.

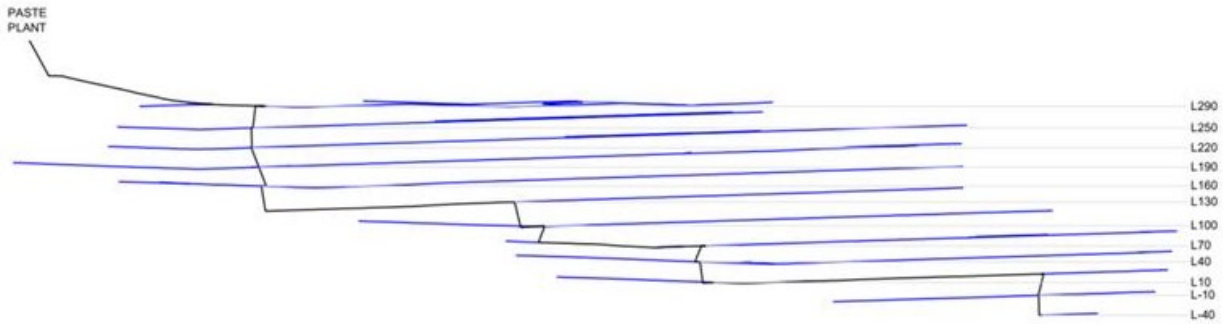
The CPB is produced at the surface pastefill plant and delivered underground directly to the stopes via a dedicated DN200 underground distribution system (UDS). For detailed information on the paste backfill plant and its equipment, refer to Section 18. The main distribution line will pass next to the Fresh Air Raise (FAR), where the piping will be branched and directed toward the active mining areas. Distribution from level to level will be facilitated by vertical boreholes. The initial focus of paste distribution will be on the first mining block. As mining advances, the system will be progressively expanded to reach the western mining blocks. This phased approach allows for efficient allocation of fill and reduces the need for extensive reconfiguration of the distribution system during early production phases.

The UDS consists of borehole dump systems (dump valves and emergency dump assemblies), pressure relief systems (rupture discs), paste monitoring instrumentation (pressure transmitters), and remote visual monitoring (cameras). The borehole dump systems serve to clear the line in emergency blockage scenarios. The pressure monitoring systems serve to monitor the rheology of the CPB during operation and report pressure values to the surface plant SCADA to allow for continuous monitoring and quicker identification of blockage events. Cameras are installed at the main borehole breakthrough, the pour point, and key pipe changeover points.

To optimize the binder consumption of the operation, the binder content in the CPB mix varies depending on the geotechnical / strength requirements for each pour and each paste pour location. Higher strength structural pours, including stopes adjacent to the pit shell, require higher concentration cement pours. Different paste strength requirements have been detailed in Section 16.2.5.3. During operations, flow modelling will be completed to each stope such that the densest paste can be delivered to each stope ultimately reducing the binder demands.

This approach ensures adequate strength gain in critical areas, enhancing ground support and stability throughout the LOM.

Figure 16.65: Underground Backfill Distribution Network – Looking North



Source: GMS, August 2025.
**Note: Not to scale.*

16.5.7.5 Compressed Air

During the initial development phase, prior to the commissioning of the permanent compressed air system, a temporary compressor unit will be deployed near the portal to supply air to active work zones. This interim setup will ensure reliable air delivery to support early mine development operations. The compressed air supply for the underground mine will then be provided by a series of electric compressors installed at the surface near the main ventilation fans. A network of compressed air piping will be routed next to the main

Fresh Air Raise (FAR) through services holes and extended along the main ramp to supply underground areas.

Compressed air will be used to power various underground systems, including:

- Dewatering pumps in development headings.
- Handheld pneumatic drills.
- Air-locked doors.
- Air-powered actuators.
- Other air-driven equipment, as required.

In addition, the compressors will supply emergency air to designated refuge stations.

Four (4) air-cooled air compressors will be installed to fulfill all underground activities. A combined power of 1,250 Hp will be installed to provide 5,000 CFM of compressed air to a maximal pressure of 125 PSI. The compressed air network will be equipped with systems to eliminate moisture and condensation within the circuit.

16.5.7.6 Communication

The underground communication system will be based on a Long-Term Evolution (LTE) network, which will be installed on site and progressively expanded throughout the life of mine (LOM). All mobile equipment operators, light vehicle drivers, and supervisory personnel will be equipped with LTE-enabled communication devices, enabling real-time voice and data communication with surface personnel. A redundant fibre-optic backbone will also be installed to provide high-capacity data transfer to surface.

In addition to supporting operational communication, the LTE network will also facilitate:

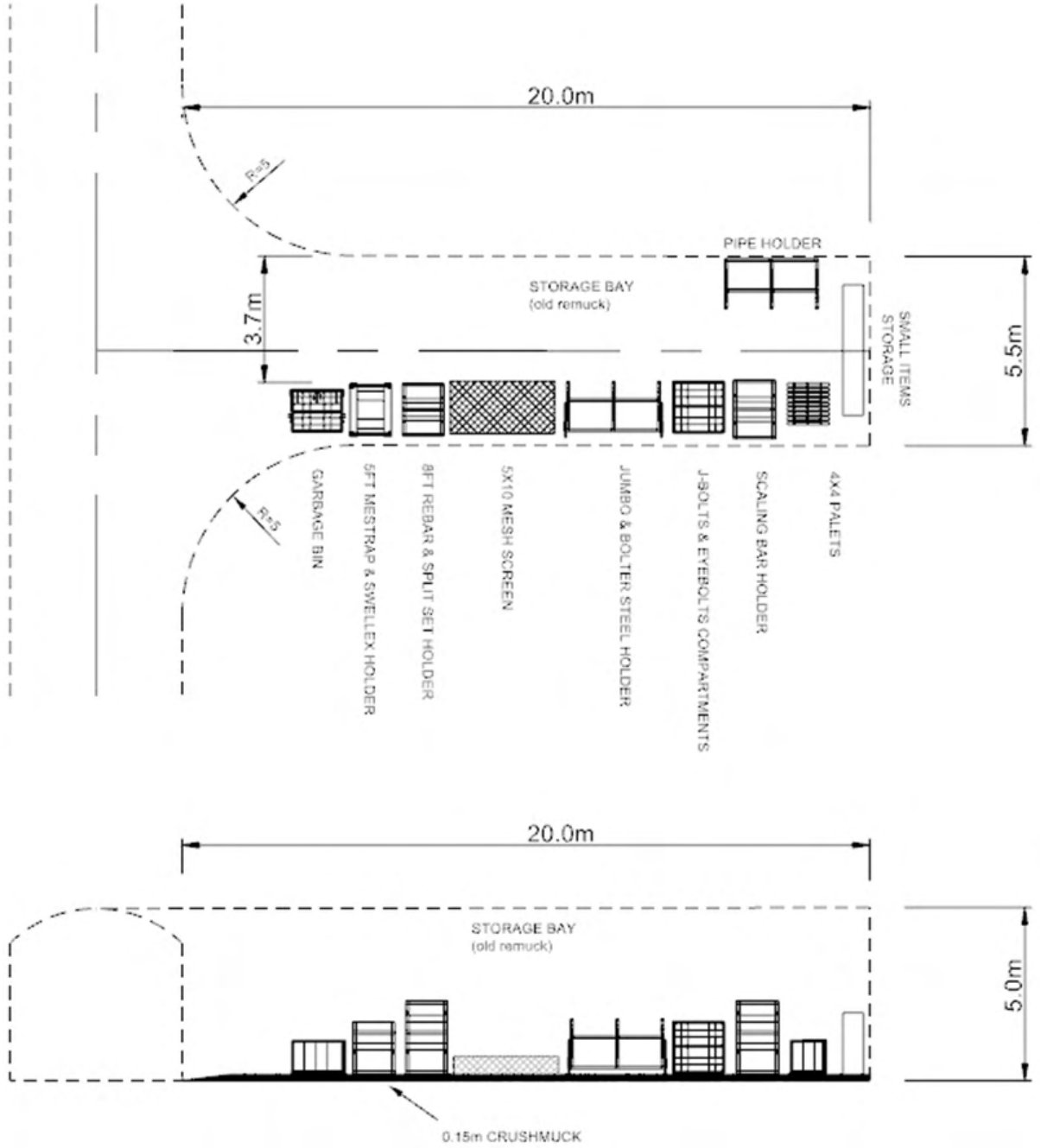
- Remote control and monitoring of the underground ventilation system when required.
- Integration of automation features for selected underground equipment.

This robust and scalable communication infrastructure will enhance operational efficiency, worker safety, and equipment management across the mine.

16.5.7.7 Storage and Warehouse

Underground storages area (Figure 16.66) will be in recycled excavations such as remucks with levelled concrete floors. Racking, garbage bins, pipe holders and other storage facilities will be installed on the concrete floor.

Figure 16.66: Typical Layout of Underground Storage (utilizing an existing excavation)



Source: GMS, June 2025.
**Note: Not to scale.*

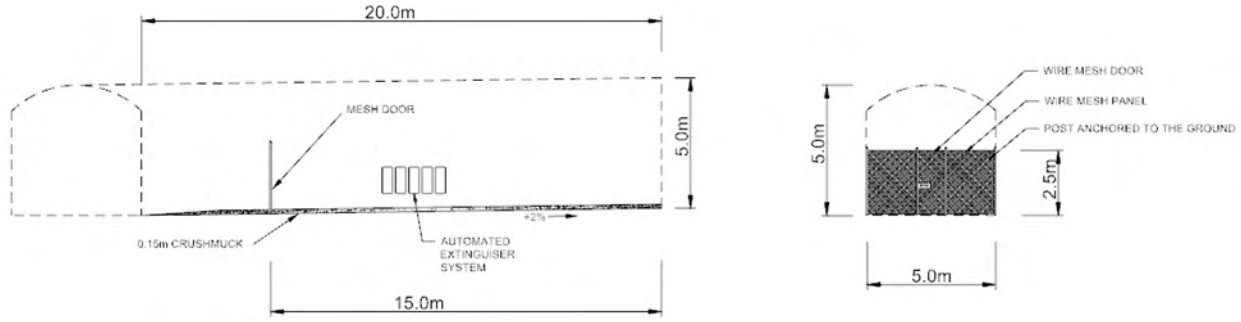
16.5.7.8 Fuel Storage and Distribution

Fuel will be primarily stored at the surface. However, two (2) underground maintenance bays dedicated to light maintenance work will be constructed to provide lubrication services for equipment that seldom returns to the surface. A dedicated fuel and lube truck will be included in the mobile equipment fleet to distribute fuel and lubricants to underground units that are unable to travel efficiently to the stationary refueling stations. In addition to servicing mobile equipment directly, the truck will also be used to replenish lubricant tanks located at underground maintenance facilities. This approach ensures continuous equipment availability and minimizes downtime associated with refueling and services cycles.

16.5.7.9 Explosives Storage and Handling

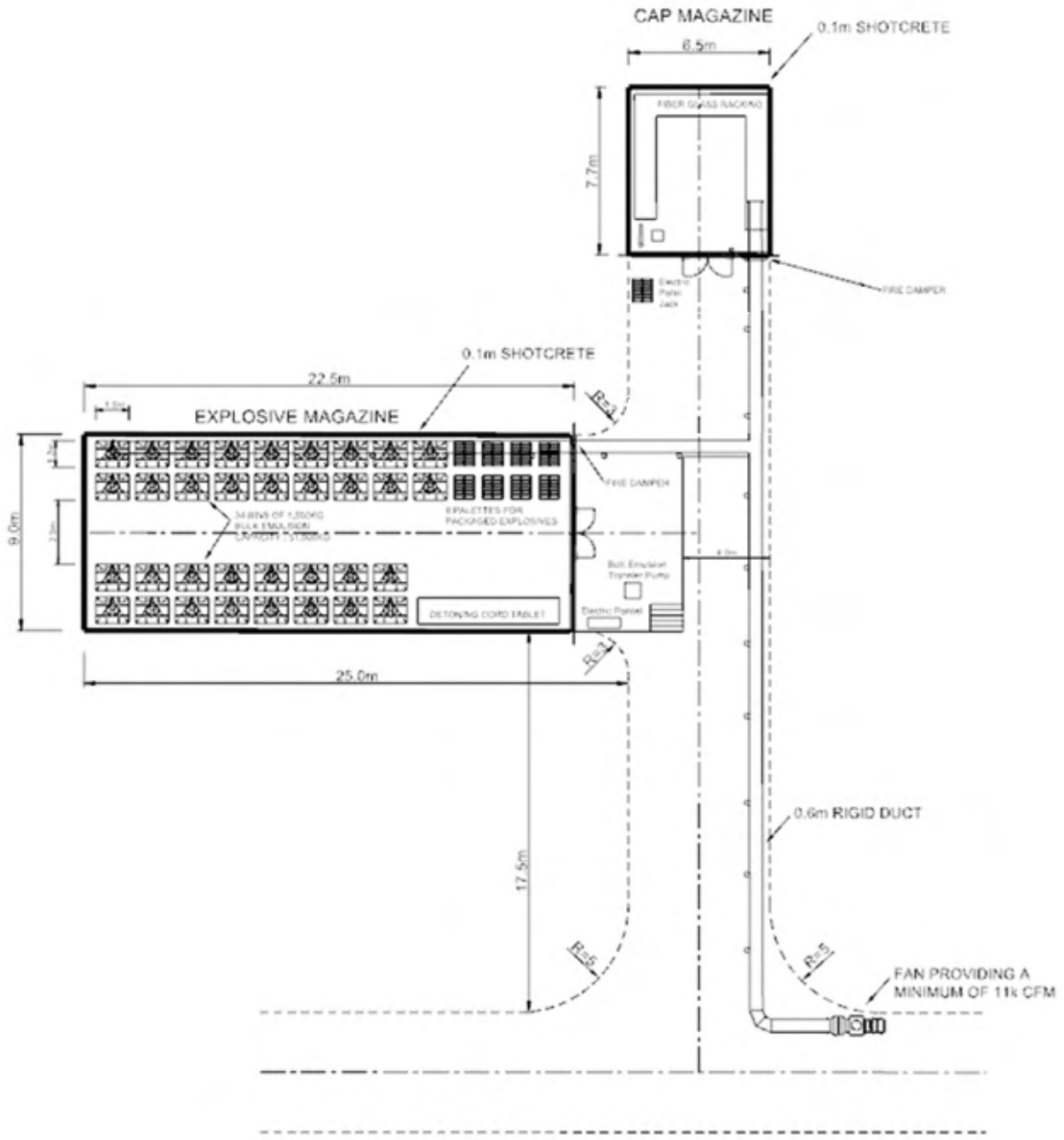
Two (2) underground explosive and detonator magazines will be installed in designated locations: one (1) near the surface and one (1) in the mid-level of the main zone. Parking for bulk emulsion trucks (Figure 16.67) as well as underground explosive and detonator magazines has been designed to meet the requirements of the Regulation respecting occupational health and safety in mines. Explosives will be delivered to the portal by the selected explosives supplier and then transported to the underground magazines by a flatbed service truck for later use in production and development activities. Sufficient space will be available to store bulk emulsion bins, packaged explosives, and detonators (Figure 16.68).

Figure 16.67: Typical Cross-section of Bulk Emulsion Truck Parking



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.68: Typical Explosives Storage and Handling Facilities



Source: GMS, June 2025.
**Note: Not to scale.*

16.5.7.10 Personnel and Underground Material Transportation

Access to the underground mine for supplies and personnel will be provided via the main ramp. Personnel carriers, such as Land Cruisers, will be used to transport workers from the surface to the underground work areas. Supervisors and technical services staff will also utilize light vehicles for underground transportation. The maintenance team, including mechanical and electrical personnel, will operate maintenance tractors

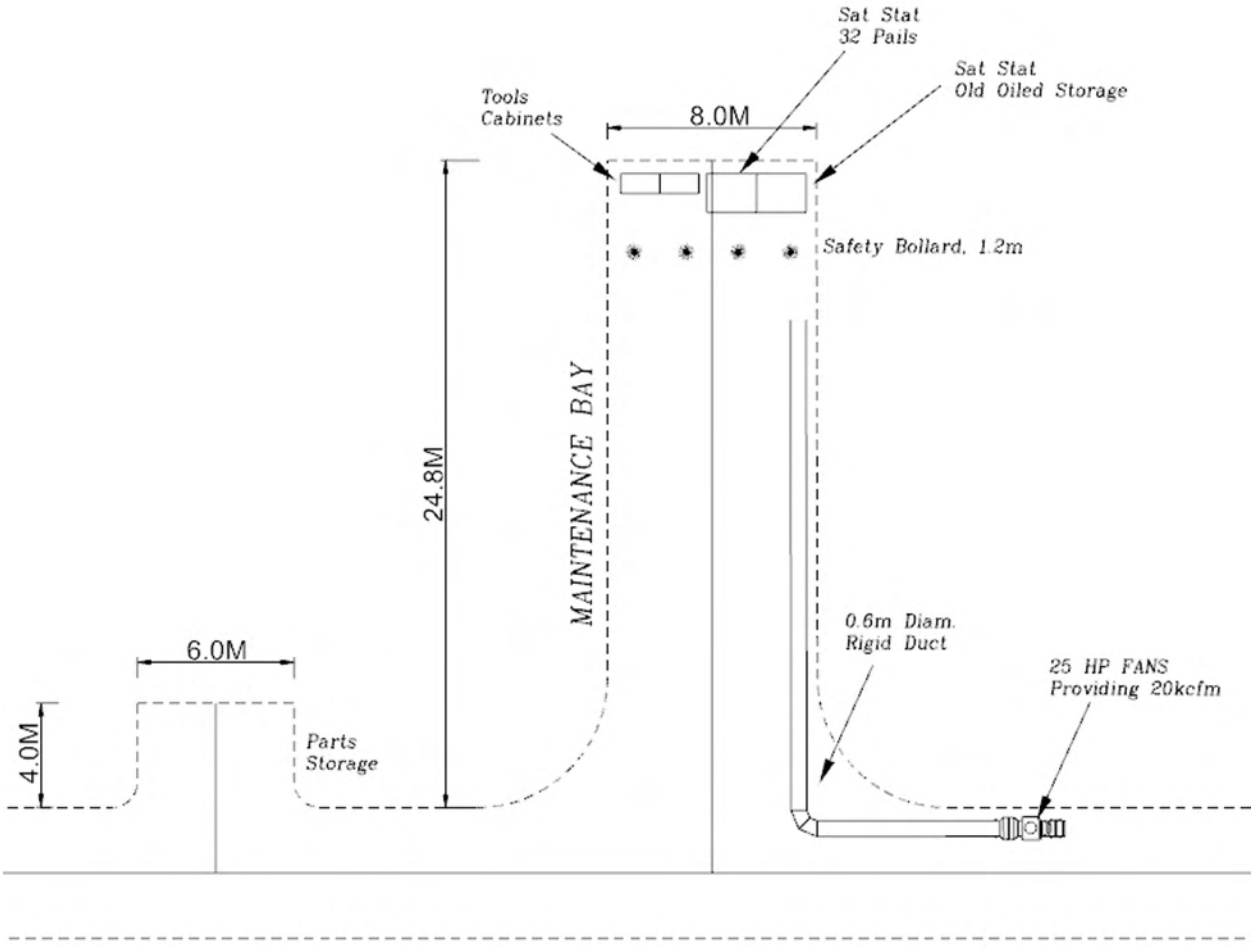
equipped with various attachments—such as forks, buckets, and booms to assist with repairs and support tasks. Two (2) boom trucks equipped with service booms will be used to transport supplies from surface to active underground headings, stopes, and material storage areas.

16.5.7.11 Equipment Maintenance

Maintenance for all equipment will be performed in accordance with supplier recommendations. To maximize equipment lifespan and reliability, rebuilds will be scheduled based on the manufacturer's recommended rebuild cycles. Maintenance will be performed by the owner’s personnel supported by OEM personnel directly on site. The maintenance department and staff requirements have been structured to fully manage this function, performing maintenance planning and employee training.

Most major mechanical maintenance activities will be conducted at the surface workshop, which is equipped to handle heavy repairs and overhauls. A temporary maintenance shop for the first years of mining will be available, and a permanent building will be constructed later. More details about the surface mine garage are provided in Section 18. Minor maintenance tasks and emergency interventions will be carried out underground in two (2) dedicated maintenance bays, specifically excavated and equipped for this purpose (Figure 16.69).

Figure 16.69: Typical Maintenance Bay



Source: GMS, June 2025.
**Note: Not to scale.*

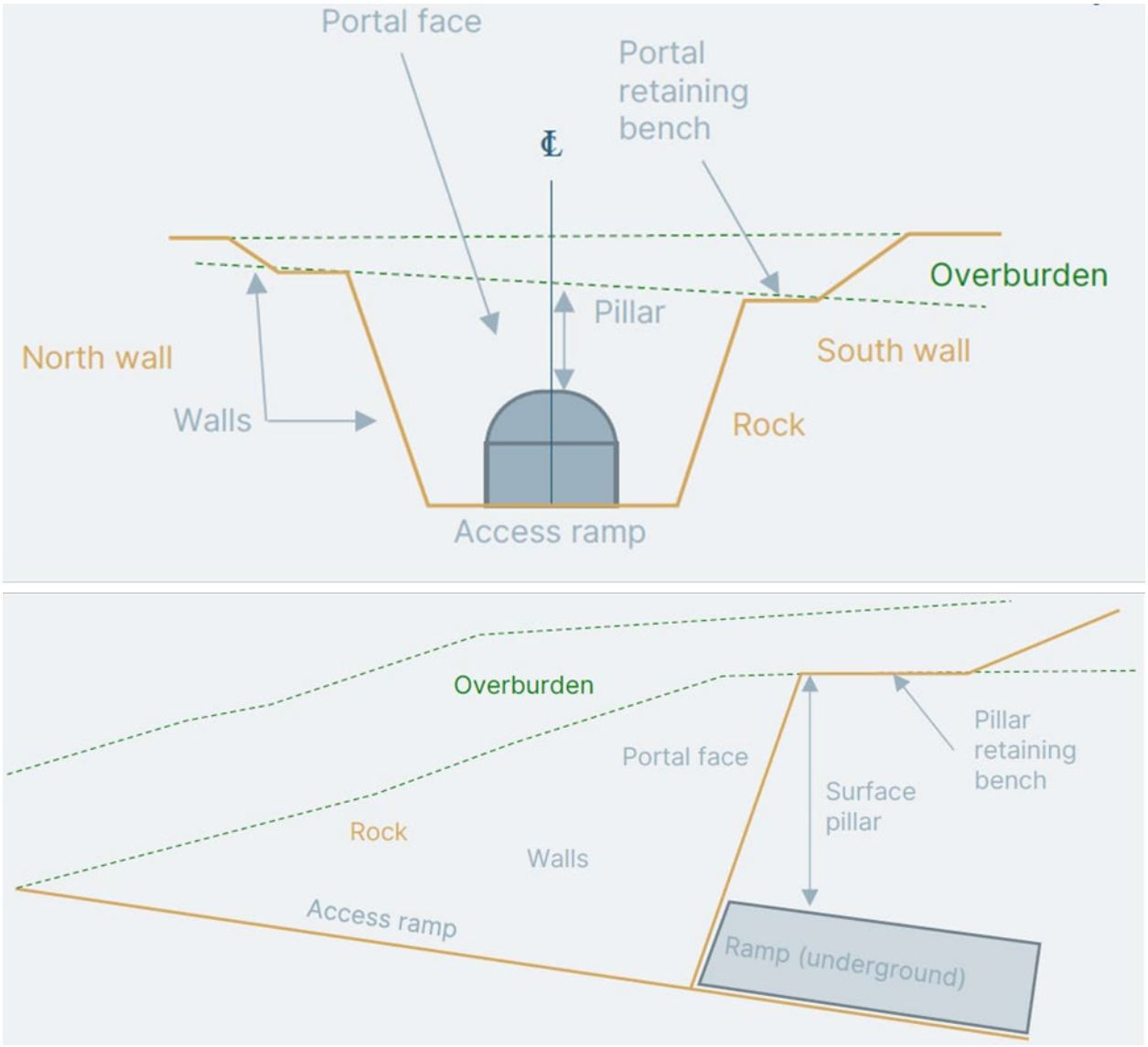
16.5.7.12 Mobile Crushing Plant

The same strategy as Open Pit will be implemented for underground needs.

16.5.7.13 Surface Portal

Access to the underground mine will be developed through a dedicated mining portal. To reach competent rock, a boxcut will first be excavated from surface down to the portal elevation. This excavation will provide the required entry profile for subsequent underground development. Figure 16.70 illustrates the cross-sectional and longitudinal views of the planned portal and boxcut configuration.

Figure 16.70: Schematic Section View of UG Mine Portal



Source: Alius, June 2025.
**Note: Not to scale.*

To mitigate the risk of material ingress and ensure long-term stability of the mine entrance, a culvert structure will be installed at the portal. This culvert will also serve as a protective measure against winter snow and ice, increasing safety and operational effectiveness. Figure 16.71 illustrates the type of culvert to be implemented.

Figure 16.71: Example of a Mine Portal Culvert



Source: Atlantic Industrial Limited Website, August 2025.

16.5.8 Underground Mine Safety Measures

16.5.8.1 Tag Boards

Tag boards will be used to control and identify people present in the underground mine. A tag board system is an essential safety tool in underground mining operations. Each worker will be issued a personal identification tag, which they will place on the tag board before entering the mine and remove upon exiting. This provides a clear, real-time record of who is underground at any given time.

The system improves accountability and enhances emergency response, as it allows supervisors and rescue teams to quickly identify personnel underground in the event of an incident.

16.5.8.2 Emergency Exits

The main decline will serve as the primary egress route from the underground workings. For secondary escape, a series of egress raises will be excavated between production levels, with one (1) key raise connecting the transfer ventilation drift to the surface. These independent emergency egress raises will be equipped with prefabricated modular Safescape Laddertube™ systems, providing safe and efficient vertical access for personnel in the event of an emergency.

16.5.8.3 Refuge Stations

Refuge stations will be strategically positioned to ensure a maximum distance of 1,000 metres from any working area, in accordance with safety regulations. To meet these requirements, some production levels will require two (2) refuge stations.

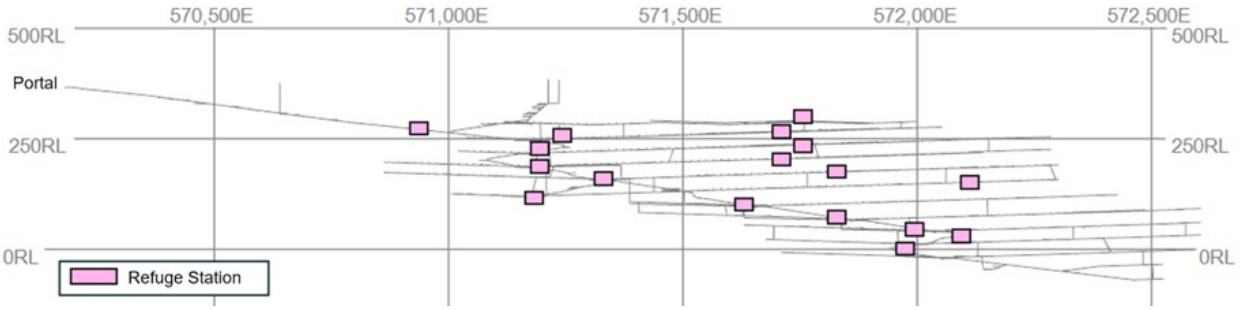
Two (2) types of refuge stations will be used:

- Portable, self-contained units: These will be deployed during lateral development phases to provide immediate protection without the delays associated with constructing permanent installations.
- Fixed installations: A total of 17 permanent refuge stations (Figure 16.72 and Figure 16.73) are planned as part of the mine's infrastructure to support ongoing production activities.

Each refuge station will be equipped with:

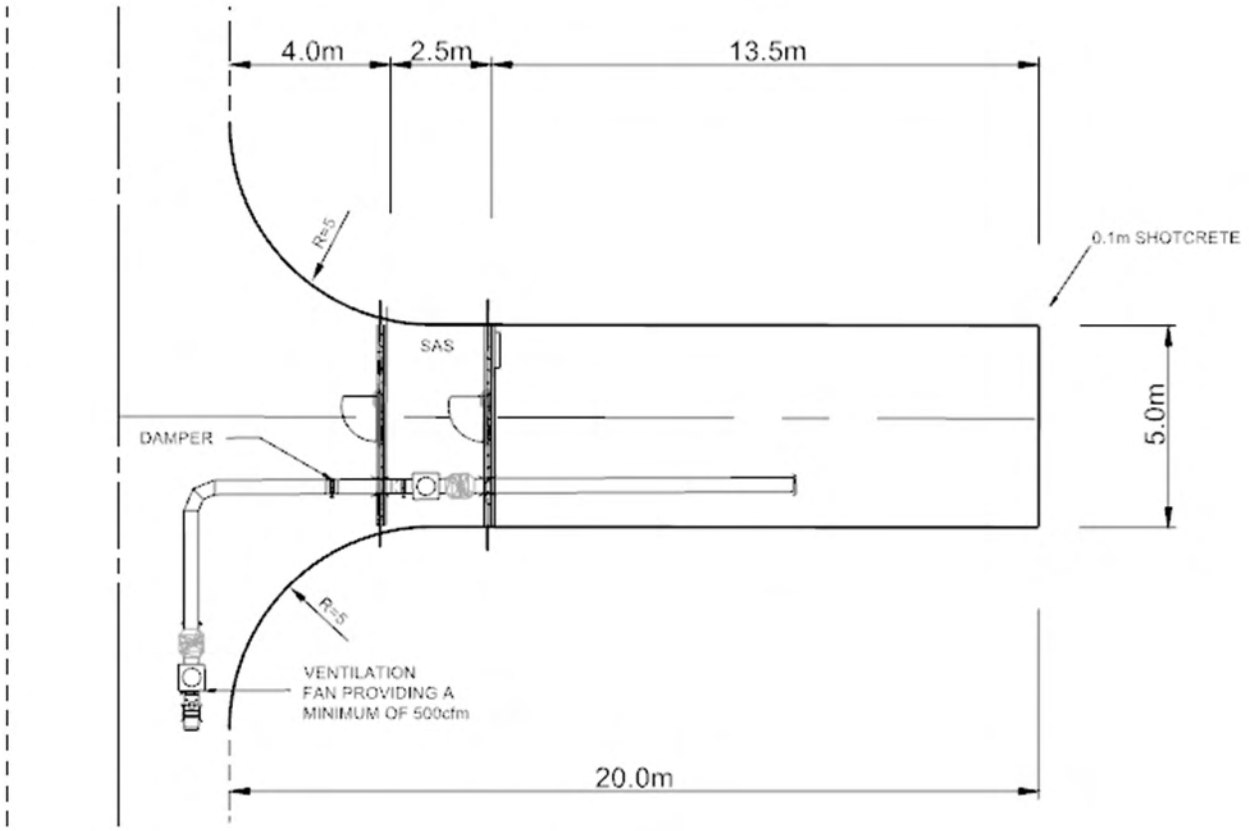
- A telephone or radio system for communication with surface, independent of mine's main power supply.
- Compressed air and water lines, along with a potable water supply.
- Emergency lighting.
- Hand tools and sealing materials.
- An updated underground mine plan showing all exits and ventilation layouts.
- A fire protection system.
- All other safety provisions required by applicable regulations.

Figure 16.72: Underground Refuge Stations Location – Section Looking North



Source: GMS, June 2025.
**Note: Not to scale.*

Figure 16.73: Typical Permanent Refuge Station I



Source: GMS, June 2025.

16.5.8.4 Mobile Equipment Safety

All underground mobile vehicles will be equipped with automatic fire suppression systems in accordance with industry best practices and safety standards. Fire extinguishers will be installed and maintained at all locations where fire hazards may exist, including electrical installations, pump stations, fuel storage areas, service garages, and other critical infrastructure, in full compliance with applicable regulations. Each underground vehicle will be equipped with at least one (1) portable fire extinguisher of the appropriate type and capacity, ensuring rapid response in the event of a fire.

16.5.8.5 Mine Rescue

Fully trained and equipped mine rescue teams will be established in compliance with all applicable regulations. A full complement of mine rescue equipment will be available on-site, including a dedicated underground emergency vehicle and a foam generator for fire suppression.

Rescue teams will be trained to respond to both surface and underground emergencies. A comprehensive Emergency Response Plan (ERP) will be developed and regularly reviewed and updated to reflect changes in mine operations and evolving regulatory requirements.

16.5.8.6 Emergency Stench System

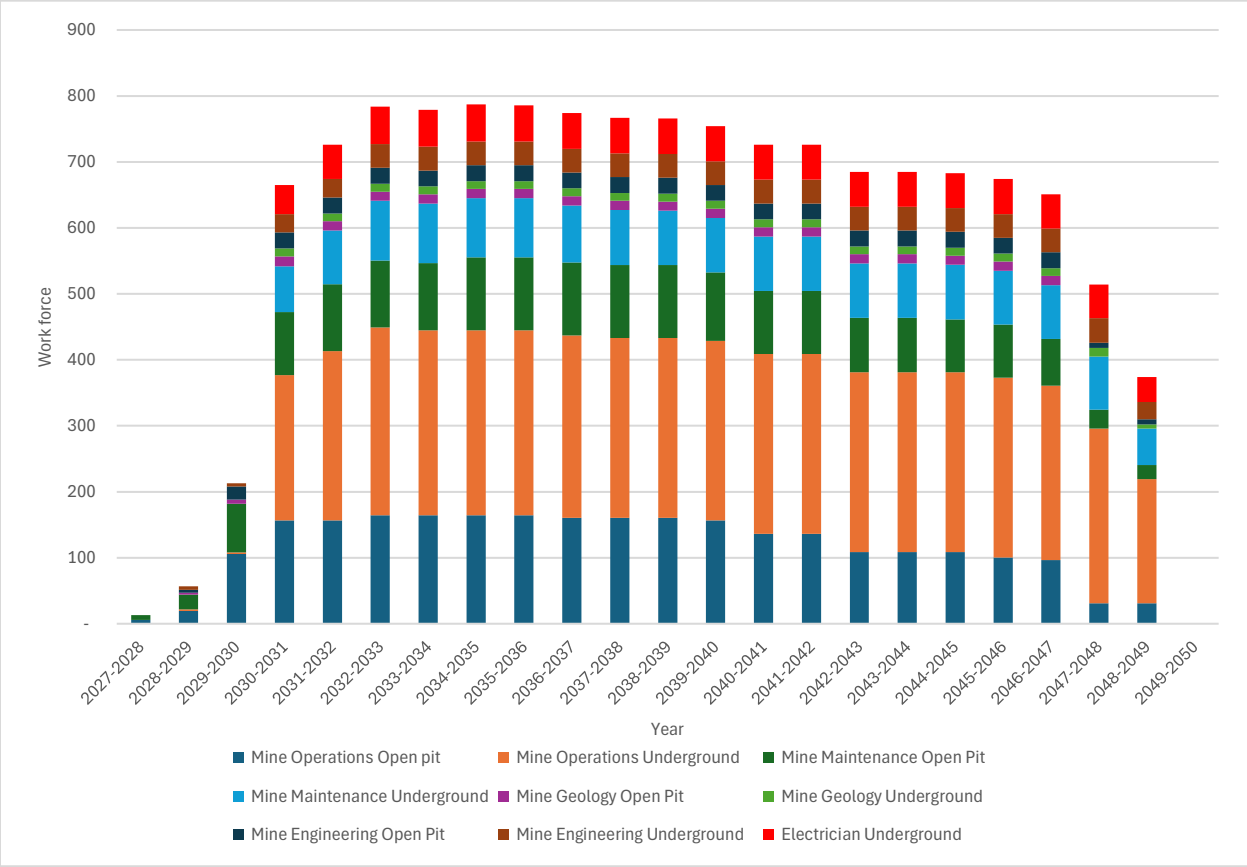
A stench gas warning system will be installed in the main surface ventilation system, beginning with the temporary system and later integrated into the permanent ventilation network. The system is designed to inject a calibrated dosage of stench gas, based on the measured airflow volume, to ensure effective distribution throughout the mine. This gas, characterized by a strong and distinctive odor, serves as an emergency alert mechanism, immediately notifying underground workers of a hazardous situation upon detection by their sense of smell. As a secondary alert method, an additional stench gas injection system will be installed in the compressed air network, providing redundancy and enhancing overall emergency response capabilities.

16.6 Mine Workforce

Mine personnel were divided into hourly and staff positions and divided between mine operations, mine maintenance, mine electrical, engineering and geology. Hourly positions were mostly associated with a shift roster of 14-days-on and 14-days-off, and as such, each unit of equipment requires four (4) operators hired in hourly positions.

Staff positions in management, supervision or technical services roles that require a continuous 7 days a week presence will also be on a rotational schedule. The combined UG and OP mine workforce peaks at 787 individuals in Year 2034-2035. Figure 16.74 shows the workforce requirements over the LOM. A total of 20% of all the mine workforce in the operation phase is assumed to be fulfilled by indigenous people.

Figure 16.74: OP and UG Mine Workforce



Source: GMS, June 2025.

16.6.1 Open Pit Mine Workforce Requirements

Table 16.39 to Table 16.43 show the estimated OP workforce requirements over the LOM. The OP workforce peaks at 313 individuals in 2034-2035. Some positions are shared with the UG mine workforce.

Table 16.39: Open Pit Mine Operations Workforce

Mine Operations	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Mine Manager*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Mine Superintendent*	Staff	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Mine Ops. General Supervisor	Staff	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Clerk*	Hourly	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Supervisor	Hourly	-	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	-
Mine D&B Supervisor	Hourly	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Trainer	Hourly	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Driller	Hourly	-	2	8	12	12	12	12	12	12	12	12	12	12	8	8	4	4	4	4	4	-	-	-
Auxiliary Drill Operator	Hourly	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-	-	-
Blaster	Hourly	-	1	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	-	-	-
Blaster Helper	Hourly	-	1	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	-	-	-
Utility Equip. Operator - Small Stemming Loader (95 HP)	Hourly	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	-	-	-
Laborer	Hourly	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	-	-	-
Shovel / Excavator Operator	Hourly	-	-	4	4	4	8	8	8	8	8	8	8	8	4	4	4	4	4	4	4	-	-	-
Loader Operator	Hourly	-	-	4	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	4	4	4	4	-
Haul Truck Operator	Hourly	-	-	24	52	52	52	52	52	52	48	48	48	44	32	32	20	20	20	16	12	8	8	-
Dewatering Labour	Hourly	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	-	-	-
Track Dozer 1	Hourly	-	2	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	-	-	-
Track Dozer 2	Hourly	-	-	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Grader Operator	Hourly	-	4	4	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	-
Water Truck Operator – Water / Sand Truck	Hourly	-	-	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Wheel Dozer	Hourly	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Shovel / Excavator Operator 49-t	Hourly	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Shovel / Excavator Operator 90-t	Hourly	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Ancillary Equipment Operator	Hourly	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Labourer	Hourly	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Sub. Total Mine Operations	**-**	**6**	**20**	**106**	**157**	**157**	**165**	**165**	**165**	**165**	**161**	**161**	**161**	**157**	**137**	**137**	**109**	**109**	**109**	**101**	**97**	**31**	**31**	**-**

**Note: These positions are shared with the underground mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the underground mine.

Table 16.40: Open Pit Maintenance Workforce

Mine Maintenance Workforce	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Superintendent*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
General Supervisor	Staff	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Supervisor	Staff	-	2	3	5	5	5	5	6	6	6	6	6	5	5	5	4	4	4	4	3	-	-	-
Senior Planner*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Planner*	Staff	-	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	-
Mechanical Engineer	Staff	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Trainer	Staff	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Clerk *	Hourly	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Mobile Mechanic	Hourly	2	7	26	38	41	41	41	46	46	46	46	46	43	38	38	31	31	29	29	24	10	7	-
Electrician	Hourly	-	1	4	6	7	7	7	8	8	8	8	8	7	6	6	5	5	5	5	4	2	1	-
Welder / Machinist	Hourly	-	1	4	6	7	7	7	8	8	8	8	8	7	6	6	5	5	5	5	4	2	1	-
Fuel & Lube Technician	Hourly	2	2	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	-	-	-
Tire Technician	Hourly	-	-	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	-	-	-
Tool Crib Attendant	Hourly	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Helper	Hourly	1	2	9	13	14	14	14	15	15	15	15	15	14	13	13	10	10	10	10	8	3	2	-
Sub. Total Mine Maintenance	**-**	**7**	**22**	**74**	**96**	**102**	**102**	**102**	**111**	**111**	**111**	**111**	**111**	**104**	**96**	**96**	**83**	**83**	**81**	**81**	**71**	**29**	**22**	**-**

**Note: These positions are shared with the underground mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the underground mine.

Table 16.41: Open Pit Mine Geology Workforce

Mine Geology Workforce	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Chief Geologist*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Production Geologist	Staff	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Grade Control Technician	Hourly	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Grade Control Labourers / Samplers	Hourly	-	-	1	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	-	-	-
Senior Geologist*	Staff	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-

Mine Geology Workforce	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Resource Geologist*	Staff	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Sub. Total Geology	**-**	**-**	**-**	**3**	**6**	**15**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**-**	**-**	**-**

**Note: These positions are shared with the underground mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the underground mine.

Table 16.42: Open Pit Mine Engineering Workforce

Mine Engineering Workforce	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Chief Mine Engineer*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Long-Term Planning Engineer*	Staff	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Short-Term Planning Engineer	Staff	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Drill & Blast Engineer	Staff	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Junior Mine Engineer	Staff	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Senior Geotechnical Engineer*	Staff	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Geotechnical Engineer	Staff	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Dispatch System Coordinator	Hourly	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Assistant Chief Mining Engineer	Hourly	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Drill & Blast Technician	Hourly	-	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Geotech. Technician	Hourly	-	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Senior Surveyor*	Hourly	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Surveyor	Hourly	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Clerk*	Hourly	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Sub Total Mine Engineering	**-**	**-**	**5**	**20**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**8**	**8**	**-**

**Note: These positions are shared with the underground mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the underground mine.

Table 16.43: Open Pit Mine Total Workforce

Total Work force Open Pit Mine	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Mine Operations	6	20	106	157	157	165	165	165	165	161	161	161	157	137	137	109	109	109	101	97	31	31	-
Mine Maintenance	7	22	74	96	102	102	102	111	111	111	111	111	104	96	96	83	83	81	81	71	29	22	-
Mine Geology	-	3	6	15	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	-	-	-
Mine Engineering	-	5	20	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24	8	8	-
Total Workforce	**13**	**49**	**206**	**292**	**296**	**304**	**304**	**313**	**313**	**309**	**309**	**309**	**298**	**270**	**270**	**229**	**229**	**227**	**219**	**205**	**68**	**61**	**-**

**Note: Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the underground mine.*

16.6.2 Underground Mine Workforce Requirements

All site-based positions will be on a rotational schedule. A workforce of 480 employees is expected to be employed for the Shaakichiuwaanaan underground mine. Some positions are shared with the open pit mine workforce. The underground mine workforce is described in Table 16.44 through Table 16.48, while Table 16.49 presents a summary of the workforce by underground department. Finally, Table 16.50 provides a summary of both the OP and UG mines workforce.

Table 16.44: UG Mine Operations Workforce

UG Mine Operations	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Mine Manager*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Mine Superintendent*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Ops. General Foreman	Staff	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Clerk*	Hourly	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Supervisor	Staff	-	-	-	8	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	8	-
Mine D&B Supervisor	Staff	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Trainer	Staff	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Miner 1	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Miner 2	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Miner 3	Hourly	-	-	-	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	24	-
Long-Hole Driller	Hourly	-	-	-	10	14	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	14	-
Blasters	Hourly	-	-	-	12	12	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	8	-
Scoop Operators	Hourly	-	-	-	12	20	28	28	28	28	28	24	24	24	24	24	24	24	24	24	24	24	20	-
Truck Operators	Hourly	-	-	-	16	32	36	36	36	36	36	36	36	36	36	36	36	36	36	36	36	36	24	-
Jumbo Operator	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	-
Rock Bolter	Hourly	-	-	-	20	24	24	20	20	20	16	16	16	16	16	16	16	16	16	16	12	12	12	-
Cable Bolter	Hourly	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Scissor lift Operator	Hourly	-	-	-	16	16	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	-
Level services	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Paste Services	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Grader Operator	Hourly	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
U/G Constructions	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Sumps and Services Labour	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Boom Truck Operator	Hourly	-	-	-	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Fuel Truck Operator	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-

UG Mine Operations	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Lamps-Dry	Hourly	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Labour - Lunchroom, UG Tool Crib ,etc.	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Total UG Mine Operation Workforce		**-**	**3**	**3**	**221**	**257**	**285**	**281**	**281**	**281**	**277**	**273**	**273**	**273**	**273**	**273**	**273**	**273**	**273**	**273**	**265**	**265**	**188**	**-**

**Note: These positions are shared with the open pit mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the open pit mine.

Table 16.45: UG Mine Maintenance Workforce

UG Mine Maintenance	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Superintendent*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
General Foreman	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Reliability Engineer	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mechanical Engineer	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Trainer	Staff	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-
Senior Planner*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Maintenance Planner	Staff	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	-
Clerk*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Leader Mechanics - Mobile Equipment	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Mechanics - Mobile Equipment	Hourly	-	-	-	28	40	49	49	48	48	45	42	41	41	41	41	41	41	40	40	39	33	-	-
Electro-mechanics	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-	-
Welders - Mobile Equipment	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-
Mechanics - Fixed Equipment	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-	-
Welder - Fixed Equipment	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Total UG Mine Maintenance Workforce		**-**	**-**	**-**	**70**	**82**	**91**	**91**	**90**	**90**	**87**	**84**	**83**	**83**	**83**	**83**	**83**	**83**	**82**	**82**	**81**	**56**	**-**	**-**

**Note: These positions are shared with the open pit mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the open pit mine.

Table 16.46: UG Geology Workforce

UG Geology	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Chief Geologist*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Senior Geologist*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	-
Resource Geologist*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Production Geologist	Staff	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	-
Grade Control Technician	Hourly	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Grade Control Labourers / Samplers	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-
Total UG Geology Workforce		**-**	**-**	**-**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**13**	**6**	**-**

**Note: These positions are shared with the open pit mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the open pit mine.

Table 16.47: UG Engineering Workforce

UG Engineering	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Chief Mine Engineer*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Assistant Chief Mining Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Long-Term Planning Engineer*	Staff	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Short-Term Planning Engineer	Staff	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Drill & Blast Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Junior Mine Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Senior Geotechnical Engineer*	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	-
Geotechnical Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Backfill Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Construction Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Backfill Technician	Hourly	-	-	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Drill & Blast Technician	Hourly	-	-	-	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Geotech. Technician	Hourly	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Ventilation Technician	Hourly	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-

UG Engineering	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Senior Surveyor*	Hourly	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	1	-
Surveyor	Hourly	-	-	-	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Draftsman	Hourly	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Total UG Engineering Workforce		**-**	**5**	**5**	**28**	**28**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**36**	**37**	**26**	**-**

**Note: These positions are shared with the open pit mine. Thus, positions presented here are whole, for presentation purposes, but are not double counted in the total manpower count.*
Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the open pit mine.

Table 16.48: UG Electrical Workforce

UG Mine Electrical	Type	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
General Foreman	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Supervisor	Staff	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Senior Planner	Staff	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Planner	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	-
Electrical Engineer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Trainer	Staff	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-
Electricians - Mobile Equipment	Hourly	-	-	-	19	27	32	31	31	30	29	29	29	28	28	28	28	28	28	28	27	26	22	-
Electricians - Instr. Technicians	Hourly	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	-
Electricians - Fixed Equipment	Hourly	-	-	-	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	-
Total UG Mine Maintenance Workforce		**-**	**-**	**-**	**44**	**52**	**57**	**56**	**56**	**55**	**54**	**54**	**54**	**53**	**53**	**53**	**53**	**53**	**53**	**53**	**52**	**51**	**38**	**-**

**Note: Column 2027-2028 is from July 2027 to June 2028.*

Table 16.49: UG Summary Workforce

Summary	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
UG Engineering	-	3	3	221	257	285	281	281	281	277	273	273	273	273	273	273	273	273	273	265	265	188	-
UG Geology	-	-	-	70	82	91	91	90	90	87	84	83	83	83	83	83	83	83	82	82	81	56	-
UG Mine Operations	-	-	-	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	13	6	-
UG Mine Maintenance	-	5	5	28	28	36	36	36	36	36	36	36	36	36	36	36	36	36	36	36	37	26	-
UG Mine Electrical	-	-	-	44	52	57	56	56	55	54	54	54	53	53	53	53	53	53	53	52	51	38	-
TOTAL	**-**	**8**	**8**	**374**	**430**	**480**	**475**	**474**	**473**	**465**	**458**	**457**	**456**	**456**	**456**	**456**	**456**	**456**	**455**	**446**	**447**	**314**	**-**

**Note: Column 2027-2028 is from July 2027 to June 2028. Totals might not add up due to shared position with the open pit mine.*

Table 16.50: Total Mining Workforce

Summary	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049	2049-2050
Total Underground	-	8	8	374	430	480	475	474	473	465	458	457	456	456	456	456	456	456	455	446	447	314	-
Total Open Pit	13	49	206	292	296	304	304	313	313	309	309	309	298	270	270	229	229	227	219	205	68	61	-
TOTAL	**13**	**57**	**213**	**666**	**726**	**784**	**779**	**787**	**786**	**774**	**767**	**766**	**754**	**726**	**726**	**685**	**685**	**683**	**674**	**651**	**514**	**374**	**-**

**Note: Column 2027-2028 is from July 2027 to June 2028.*

16.7 Combined Production

The open pit mine is scheduled for a pre-production period lasting 24 months. During this time, the mine will produce a total of 10.1 Mt of waste and overburden, along with 1.4 Mt of ore material. About 5.0 Mt of the mined material will be used for construction pads, roads, stockpile pads, and related infrastructure. The focus during this phase will be to collect non acid generating rock for construction needs, removing non-valuable material to access the mineralized zones and preparing the site for full-scale operations.

The commercial production of the process plant Stage 1 will happen four (4) months after the start of the pre-production period. The processing plant will then gradually ramp up to reach the process plant Stage 1 nominal production capacity (2.5 Mt). According to the ramp up curve associated with a DMS process plant, a full Stage 1 productivity will be reached seven (7) months after the commercial production of that same Stage.

The second stage of the process plant will start at the end of Year 2 of the production period. A second line of dense media separation will be added, resulting in a doubling of the overall capacity of the process plant. The ramp up of that line follows the same as the first line, resulting to a full process plant production capacity of 5.1 Mt in Year 3 of production.

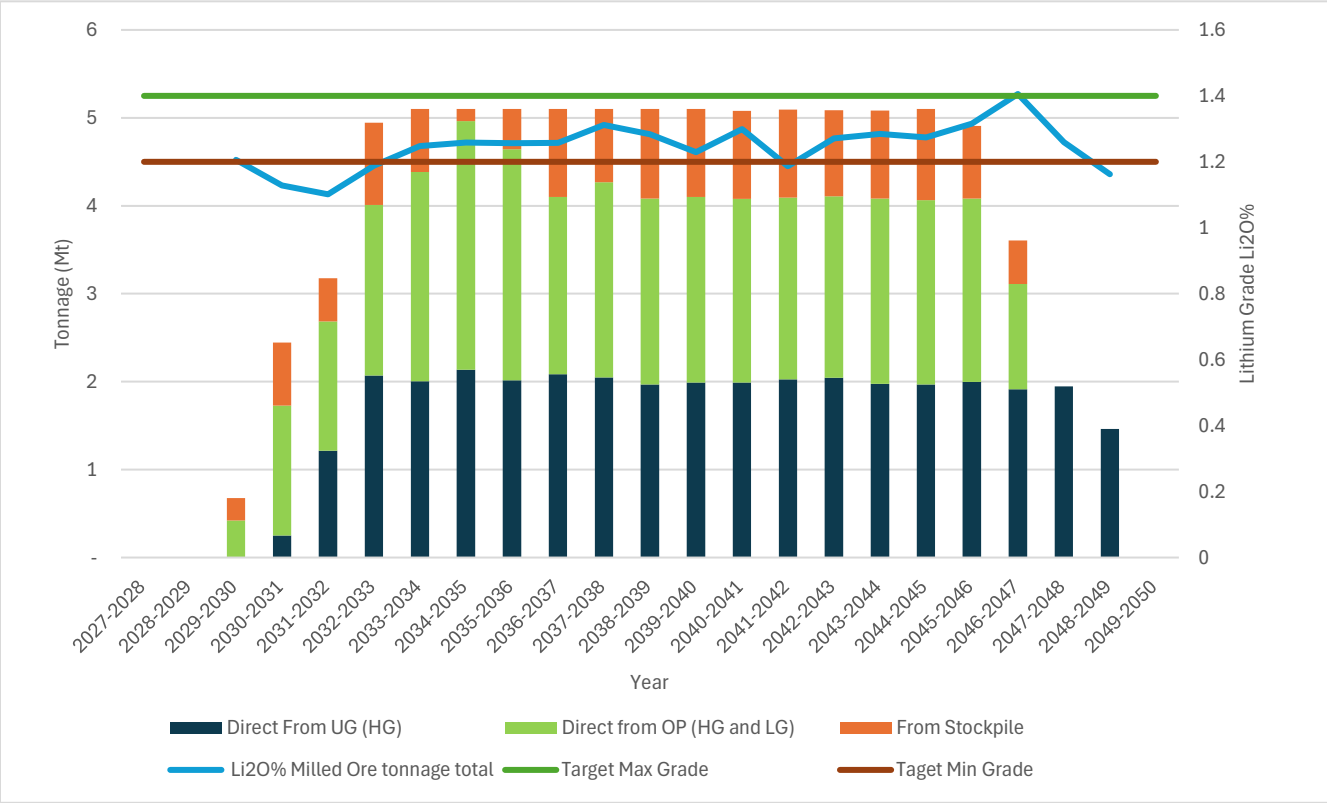
Table 16.51 shows annual production data for the Life-of-Mine (LOM). Additionally, Figure 16.75 shows mine production by source (UG,OP, or Stockpile), while Figure 16.76 shows total lithium concentrate production.

Table 16.51: LOM Production Details per Production Year

Description	Unit	Total	2027-2028	2028-2029	2029-2030	2030-2031	2031-2032	2032-2033	2033-2034	2034-2035	2035-2036	2036-2037	2037-2038	2038-2039	2039-2040	2040-2041	2041-2042	2042-2043	2043-2044	2044-2045	2045-2046	2046-2047	2047-2048	2048-2049
Mined																								
Open Pit Ore Tonnage	kt	49,196	-	-	1,412	2,238	1,886	2,826	3,188	3,030	3,161	3,059	2,649	3,184	3,170	3,149	3,136	3,091	3,138	3,118	2,563	1,197	-	-
Open Pit Ore Grade	%Li_2O	1.12	-	-	1.08	1.07	0.97	1.00	1.07	1.10	1.03	1.11	1.02	1.07	1.06	1.03	1.00	1.15	1.29	1.36	1.41	1.47	-	-
UG Ore Tonnage	kt	35,101	-	-	-	253	1,214	2,068	2,004	2,135	2,017	2,084	2,047	1,969	1,991	1,989	2,026	2,043	1,975	1,968	1,999	1,913	1,945	1,461
UG Ore Grade	%Li_2O	1.45	-	-	-	1.17	1.27	1.42	1.48	1.45	1.58	1.75	1.75	1.54	1.56	1.53	1.47	1.49	1.30	1.29	1.31	1.31	1.26	1.16
Sent Directly to Process																								
Open Pit Ore Tonnage	kt	35,303	-	-	286	1,607	1,471	1,939	2,382	2,827	2,626	2,016	2,219	2,115	2,109	2,090	2,067	2,065	2,109	2,096	2,083	1,197	-	-
Open Pit Ore Grade	%Li_2O	1.14	-	-	1.30	1.19	1.04	1.13	1.20	1.14	1.11	1.03	0.98	0.99	0.97	0.98	1.10	1.22	1.32	1.25	1.36	1.47	-	-
UG Ore Tonnage	kt	35,101	-	-	-	253	1,214	2,068	2,004	2,135	2,017	2,084	2,047	1,969	1,991	1,989	2,026	2,043	1,975	1,968	1,999	1,913	1,945	1,461
UG Ore Grade	%Li_2O	1.45	-	-	-	1.17	1.27	1.42	1.48	1.45	1.58	1.75	1.75	1.54	1.56	1.53	1.47	1.49	1.30	1.29	1.31	1.31	1.26	1.16
Stockpile to Process																								
Ore Tonnage	kt	13,893	0	0	209	763	491	935	715	107	440	976	510	893	1 ,000	840	1, 006	881	862	948	729	1, 405	182	0
Ore Grade	%Li_2O	1.06	-	-	1.05	1.00	0.87	0.80	0.76	0.61	0.65	0.67	1.14	1.41	1.10	1.48	0.78	0.93	1.22	1.32	1.19	1.41	0.55	-
Total																								
Total UG & OP Processed Ore	**kt**	**84,297**	**-**	**0**	**495**	**2,623**	**3,175**	**4,943**	**5,100**	**5 069**	**5,082**	**5,076**	**4,776**	**4,977**	**5,100**	**4,919**	**5,100**	**4,989**	**4,946**	**5,012**	**4,810**	**4,515**	**2,128**	**1,461**
Total UG & OP Processed Grade	**%Li_2O**	**1.26**	**-**	**-**	**1.19**	**1.14**	**1.10**	**1.19**	**1.25**	**1.26**	**1.26**	**1.26**	**1.33**	**1.28**	**1.23**	**1.29**	**1.19**	**1.28**	**1.29**	**1.27**	**1.32**	**1.39**	**1.20**	**1.16**
Total Produced Spodumene Concentrate - Dry	**kt**	**13,255**	**0**	**0**	**73**	**364**	**424**	**726**	**796**	**802**	**802**	**802**	**802**	**802**	**781**	**802**	**748**	**802**	**802**	**802**	**802**	**802**	**316**	**209**

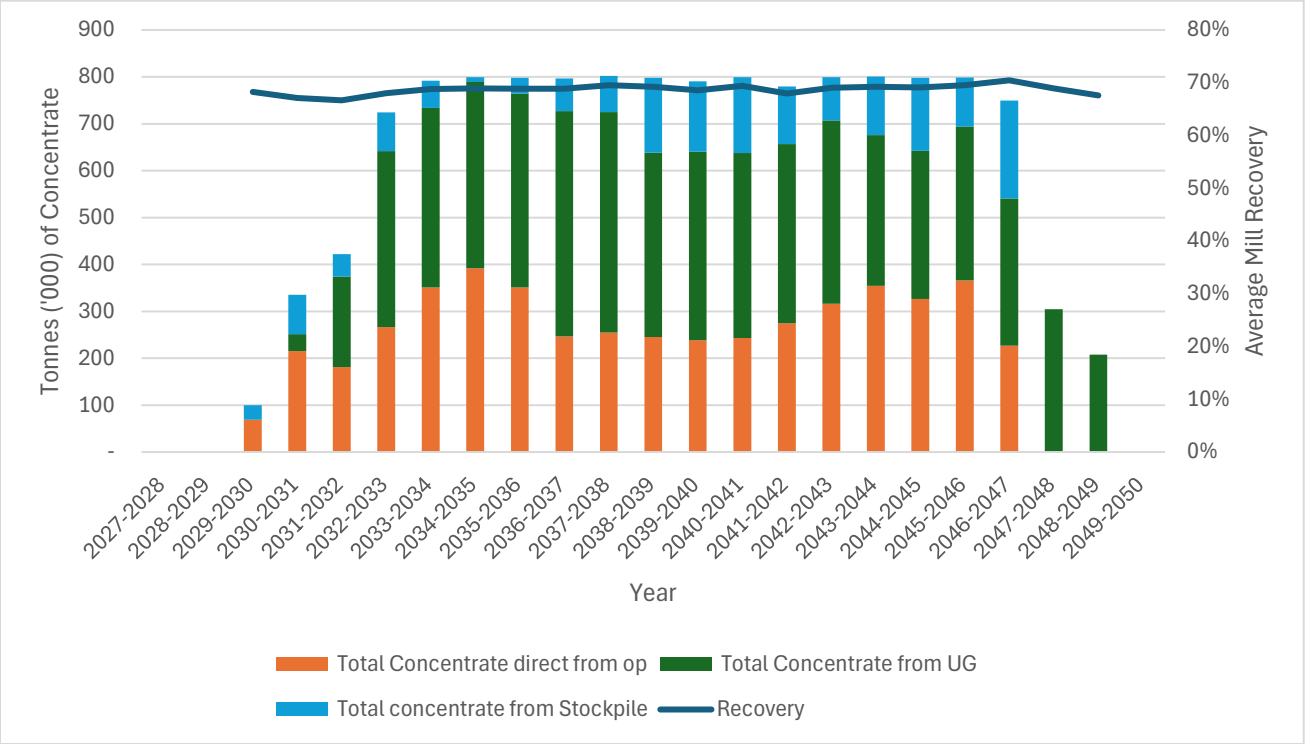
**Notes: Column 2027-2028 is from July 2027 to June 2028.*
Some numbers might not add up due to round up errors.

Figure 16.75: Ore Processing by Source Material



**Source: GMS, June 2025.*

Figure 16.76: Spodumene (5.5%Li_2O) Concentrate Production



**Source: GMS, June 2025.*

17. RECOVERY METHODS

17.1 Mineral Processing Facility Design

The mineral processing facility is designed to produce spodumene concentrate from the Run-of-Mine (ROM). The facility will be in James Bay, Québec, Canada. The facility will include ROM stockpiling, crushing, beneficiation, dewatering, and load-out areas. Crushing, beneficiation and dewatering will be performed using two (2) identical parallel process trains that could be operated independently of one another. Each process train will account for half (50%) of the crushing, beneficiation and dewatering capacity.

Each process train will be inside three (3) main buildings: the primary crushing building, the secondary and tertiary crushing building, and the main process plant. The crushed mineralized material will be stored under domes located on a concrete pad. The process trains may have shared or separate buildings.

The mineral processing facility is designed to produce spodumene concentrate at 5.50% lithium oxide grade (% Li_2O) from mineralized material containing 1.31% Li_2O. The ROM will be transported by truck either to the crushing area or to the ROM stockpiles. The comminution and beneficiation processes include crushing, dense media separation (DMS), magnetic separation, and dewatering. The facility will also perform thickening, filtration, product load-out, and tailings handling.

17.2 Design Criteria

The mineral processing facility is designed to nominally process 5,000,000 dry tonnes per year (tpa) of mineralized material with a grade of 1.31% Li_2O, producing 756,087 tpa of spodumene concentrate with a grade of 5.63 wt.% Li_2O achieving a minimum of 65.0% Li_2O recovery (these values are based on the years of full production, i.e., Years 4 to 18). These figures are based on current and historical test work results and may change depending on the composition of the mineralized material. All unit-process performances have been compared with benchmarked data from similar projects and mineralized material.

The crushing plant will have an overall availability and utilization of 68%, equivalent to 5,957 h/y of operation. The concentrator has an overall availability and utilization of 85%, equivalent to 7,446 h/y of operation. Bins have been planned to handle the differences in availability and utilization between the different areas.

Crushing, beneficiation and dewatering will be performed via two (2) identical parallel mineral process trains that could be operated either individually or simultaneously. Each process train will account for half (50%) of crushing, beneficiation, and dewatering capacity.

The mineral processing facility design criteria are summarized in Table 17.1. The design criteria include both process trains.

Table 17.1: Mineral Processing Facility Design Criteria

Parameter	Unit	Nominal
Crushing Plant		
Run of Mine (ROM)		
Specific Gravity (solids)	-	2.75
ROM Moisture Content	w/w %	5.0
Crushing Work Index	kWh/t	14.0
Bond Abrasion Index	g	0.6
Uniaxial Compression Tests	MPa	60
Li_2O Composition	%	1.31
Fe_2O_3 Composition	%	0.29-1.15
Ta_2O_5 Composition	ppm	0-300
F_{100} Sizing	mm	1,000
F_{80} Sizing	mm	202
Throughput		
ROM Processed Annually	Mtpa	5.1
Overall Availability and Utilization	%	68
Annual Operating Hours	h	5,957
ROM Processed Hourly (during operation)	tph	839.4
Crushed Mineralized Material		
P_{100} Sizing	mm	9.5
P_{85} Sizing	mm	6.3

Parameter	Unit	Nominal
DMS Plant		
Throughput		
Annually Plant Tonnage	Mtpa	5.1
Overall Availability and Utilization	%	85
Annual Operating Hours	h	7,446
Hourly Feed Tonnage (during operating hours)	tph	671.5
Coarse DMS Fraction	wt.% of plant feed	56.4
Fine DMS Fraction	wt.% of plant feed	17.0
Ultrafine DMS Fraction	wt.% of plant feed	8.3
DMS Bypass Fraction	wt.% of plant feed	18.2
Spodumene Concentrate Produced Annually(1)	tpa	756,087
Spodumene Concentrate Produced Hourly (during operating hours)	tph	101.5
Spodumene Concentrate		
Specific Gravity (solids)	-	2.96
Moisture Content	w/w %	5
Li_2O Composition	%	5.50
Fe_2O_3 Composition	%	<2.0
Minimum Li_2O Recovery (with 1.31% feed grade) (1)	% of Li_2O in plant feed	65.0%

**Note: (1) Production value and average feed grade are based on the years of full production, i.e., Years 4 to 18.*

17.2.1 Recovery

The concentrator has a recovery that is a function of the feed lithia grade (i.e., % Li_2O). The recovery can be estimated with the following function:

$$Recovery\ \% = Max\ Recovery\ \% \times \left(1 - e^{-C(Li_2O\ Feed\ Grade\ \%)}\right)$$

$$Recovery\ \% = 75\ \% \times \left(1 - e^{-1.995(Li_2O\ Feed\ Grade\ \%)}\right)$$

Where the Recovery is the recovery of lithium, Max Recovery is the theoretical maximum recovery attainable set to 75%, and C is a curve shape factor constant (it was found that in this case, C is equal to 1.995).

The lithium recovery expected from a three-size range, DMS concentrator treating material 9.5 mm to 0.65 mm, is shown in Figure 13.20. The recovery is deemed to be a relationship to the concentrators

Li_2O feed grade. Expected concentrator recoveries are lower than test work results based on scale-up factors that are driven by the effects of both larger diameter cyclones and the crowding effect seen in the DMS sinks. This variation between laboratory test work results and those achieved in operating plants has, to date, been observed within the industry with respect to operating spodumene DMS concentrators. For reference, lithium recoveries achieved by other DMS-only concentrators are shown for reference (Industrial DMS Only Performance). The Project's higher expected recovery is due both to the quality of the material (large spodumene grains with a narrow grain size distribution) and the three (3) size range DMS plant, which lessens the impact of particle size effect in the DMS process.

17.2.2 Mass Balance

A simplified mass balance for the mineral processing facility at its nominal capacity (i.e., 5.1 Mtpa at 68% and 85% utilization and availability for the crushing plant and concentrator, respectively) is shown in Figure 17.1.

17.2.3 Water Balance

A simplified water balance for the mineral processing facility at its nominal capacity (i.e., 5.1 Mtpa at 68% and 85% utilization and availability for the crushing plant and concentrator, respectively) is shown in Figure 17.2. The balance shows the raw water requirements for items such as gland seal water, filter belt wash and flocculant preparation, as well as the excess of process water from the process.

Figure 17.1: Mineral Processing Facility Simplified Mass Balance

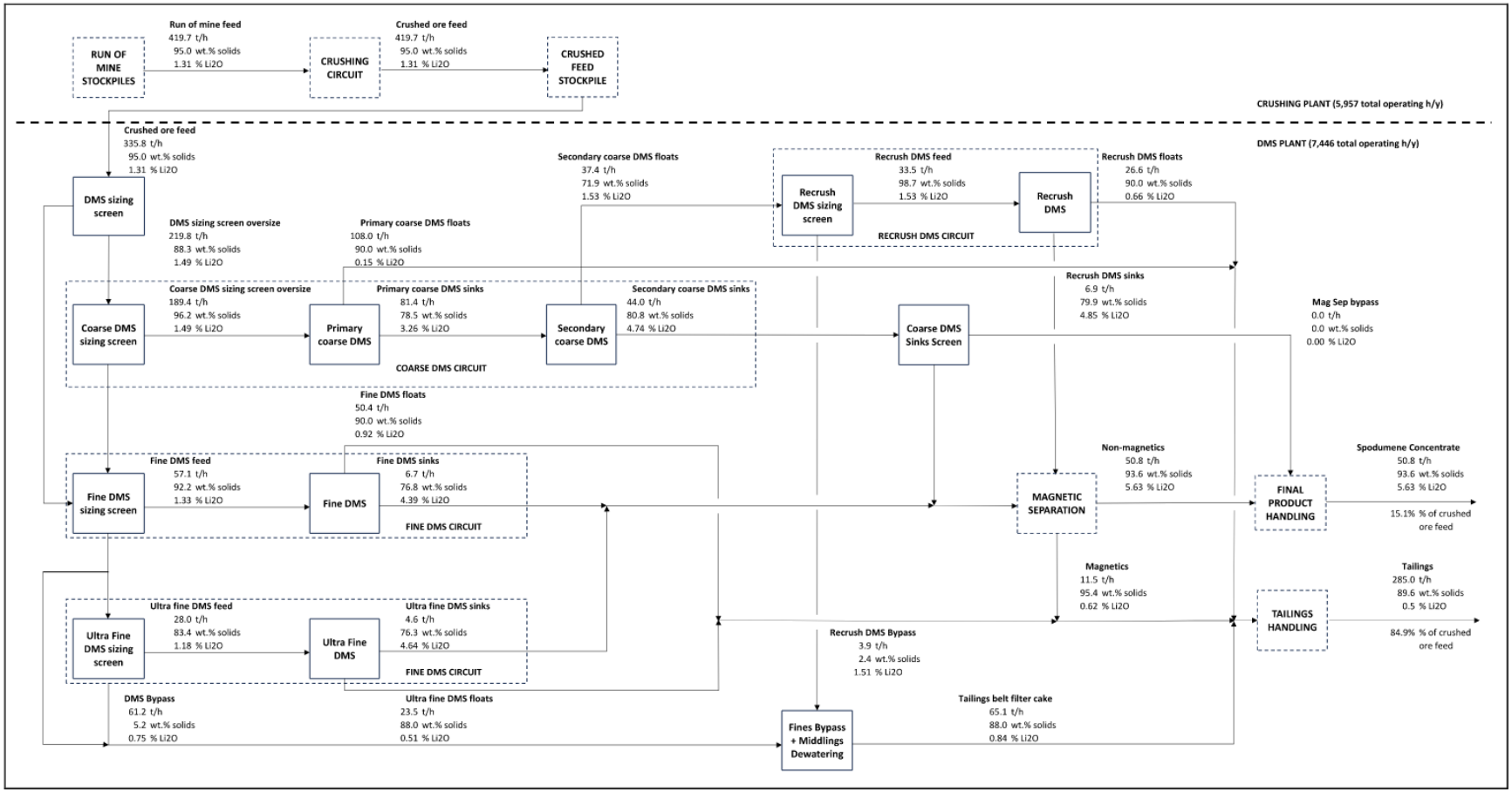
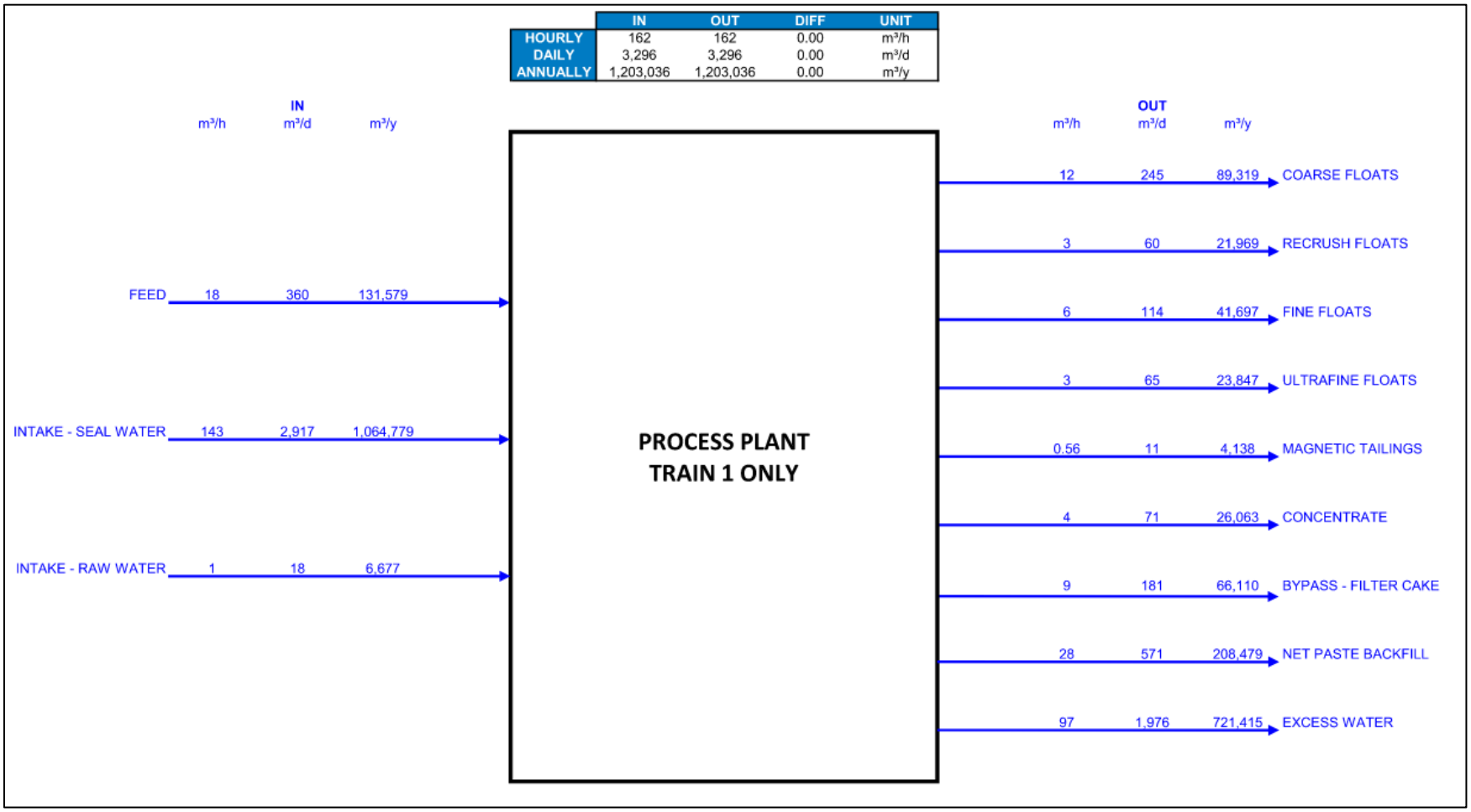


Figure 17.2: Mineral Processing Facility Simplified Water Balance



17.2.4 Reagents

The reagents required for the mineral processing facility are in Table 17.2.

Table 17.2: Mineral Processing Facility Reagent Requirements

Operation	Consumable	Usage	Delivery Form	Distribution Method
DMS	Ferrosilicon 270D	Dense media	Bulk bags	Bag breaker and mixing tank
Thickening and Dewatering	Magnafloc 10	Anionic flocculant	25-kg bags	Dilute with raw and process water

Fresh ferrosilicon (FeSi) will be added to a mixing tank, which will then be pumped to each DMS's respective correct medium tank. Flocculant mixing will be completed in a designated area within the plant with a dedicated sump pump which recycles spillage to the thickener feed tank.

17.2.5 Utilities

17.2.5.1 Raw Water

Within the concentrator, the raw water system will include a raw water storage tank and distribution pumps to deliver the water as needed. Raw water will be used for process makeup, gland seal and vacuum pump seal water, belt filter cloth wash, and fire protection.

17.2.5.2 Process Water

The process water circuits provide process water to the DMS circuit. Water will be recovered from thickener overflows with raw water as makeup, if necessary.

17.2.5.3 Water Treatment

The concentrator will have a dedicated bleed / purge stream that transfers excess process water to a site-wide multi-stage water treatment facility for contaminant removal.

17.3 Mineral Processing Facility Description

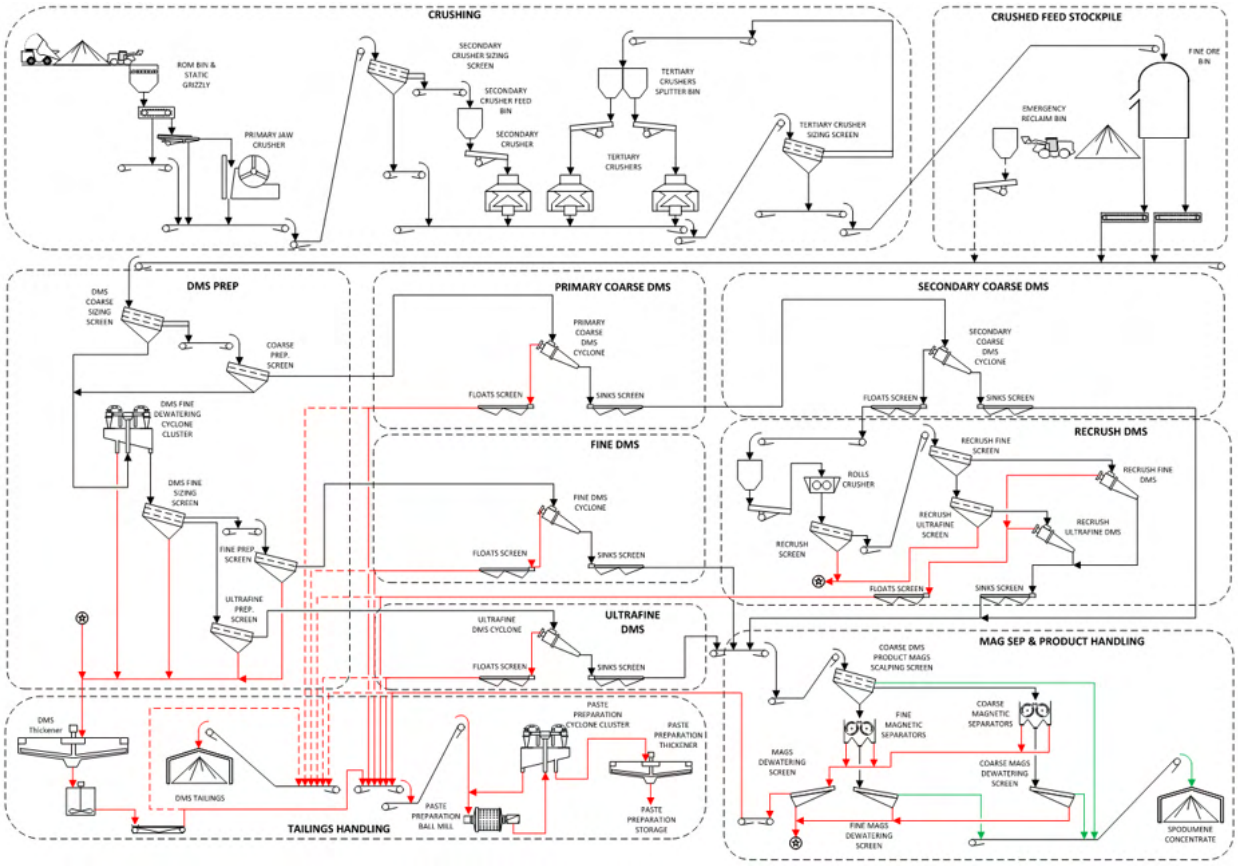
The key process areas of the mineral processing facility are:

- ROM stockpiles.

- Crushing circuit (with primary, secondary, and tertiary crushing).
- Crushed feed stockpile.
- Coarse DMS circuit.
- Fine DMS circuit.
- Ultrafine DMS circuit.
- Recrush DMS circuit.
- Magnetic separation and final product handling.
- Fines bypass + middling dewatering and handling.
- Final tailings handling.

The mineral processing facility's simplified process flow diagram is in Figure 17.3. This figure summarizes the process flows among the major sections of the mineral processing facility. Figure 17.3 shows only one (1) of the two (2) identical process trains. Similarly, the process descriptions below describe only one (1) of the two (2) identical process trains.

Figure 17.3: Mineral Processing Facility Simplified Process Flow Diagram



17.3.1 Run-of-Mine Stockpiles

The ROM will be transported from the mine to the mineral processing facility using mine trucks. There is the ability to dump directly into the dump pocket in front of the primary crusher or transport it to the ROM stockpiles.

The ROM stockpile will be located near the primary crusher and will have a minimum capacity of 288,000-t, equivalent to 21 days of residence time. The ROM will be rehandled from the ROM stockpiles to the ROM bins in the crushing circuit using a front-end loader (FEL).

17.3.2 Crushing Circuit

ROM feed with a top size of 100 mm and an 80% passing of 200 mm will be crushed by a single primary jaw crusher, producing a product with an 80% passing size of 150 mm.

The primary crusher product will be conveyed to the double-deck vibrating secondary crushing screen with a top deck opening of 30 mm and a bottom deck opening of 20 mm. The screen oversize from both decks will be directed to the secondary cone crusher with a product 80% passing size of 40 mm.

The screen undersize and secondary crusher product will be conveyed to a double-deck vibrating tertiary crushing screen with a top deck opening of 19 mm and a bottom deck opening of 9.5 mm. The tertiary crushing screen will be in a closed circuit with two (2) tertiary crushers. The screen oversize from both decks will be directed to the tertiary cone crusher. The tertiary crusher will have a product size of 80% passing 16 mm. The tertiary crusher product will be redirected to the tertiary crushing screen. The screen undersize will be conveyed to the fine ore stockpile.

17.3.3 Fine Ore Stockpile

The crushed mineralized material with a top size of 9.5 mm will be stored in the fine ore stockpile dome located on a concrete pad. The pile will be reclaimed via two (2) vibrating feeders located in a concrete reclaim tunnel under the pile. The tunnel will be equipped with two (2) exits and proper sumps as per the code. Feeders will move the material from the storage pile to a conveyor that will feed the dense media separation preparation circuit. This fine ore stockpile will act as a buffer between the crushing circuit and the processing plant.

17.3.4 Dense Media Separation Preparation Circuit

Material is conveyed from the fine ore stockpile to the concentrator building, where it is screened by the DMS coarse sizing screen. The bottom screen deck has a cut size of 3.2 mm. The oversize from both decks reports to the DMS coarse preparation screen. The screen undersize reports to the DMS fine feed pump box. The -9.5 mm to +3.2 mm fraction reporting to the coarse preparation screen is screened on a single deck screen at a cut size of 3.4 mm. The screen oversize (i.e., -9.5 mm to +3.4 mm) feeds the coarse DMS circuit. The screen undersize (i.e., -3.4 mm) reports to the DMS fine feed pump box, where it is combined with the coarse sizing screen undersize and pumped to a cyclone cluster for dewatering. The dewatering cyclone cluster underflow feeds the DMS fine sizing screen. The double-deck fine sizing screen has a top and bottom deck aperture of 1.6 and 0.65 mm, respectively. The top deck oversize (i.e., +1.6 mm) reports to the DMS fine preparation screen and the bottom deck oversize (i.e., -1.6 mm to +0.65 mm) reports to the DMS ultrafine preparation screen. The fine preparation screen has a single screen deck with a 1.6 mm cut size. The oversize (i.e., -3.4 mm to +1.6 mm) reports to the fine DMS circuit. The ultrafine preparation screen cut size is 0.65 mm. The screen oversize (i.e., -1.6 mm to +0.65 mm) reports to the ultrafine DMS circuit. The dewatering cyclone overflow, fine sizing screen undersize, fine prep screen undersize, and ultrafine preparation screen undersize collectively form the DMS “bypass fraction”, the

material from the plant's feed that is too fine for DMS processing. This material reports to the tailings dewatering thickener.

17.3.5 Coarse Dense Media Separation Circuit

The -9.5 mm to +3.4 mm size fraction from the preparation circuit is processed in a two-stage DMS circuit (referred to as primary and secondary coarse DMS). The primary coarse DMS stage will have a density cut point of approximately 2.65, which will generate a feed for the second stage (the denser fraction known as the sinks) and a reject stream sent to 'final reject handling' (the lower density material, known as the floats). The secondary coarse DMS stage will have a density cut point of approximately 2.90. The denser material (the sinks) will be sent to the magnetic separation and product handling circuit, while the less dense fraction (the floats) will report to the recrush DMS circuit for further processing.

17.3.6 Fine Dense Media Separation Circuit

The fine DMS circuit is fed with a -3.4 mm to +1.6 mm material. Unlike the coarse circuit, the fine DMS circuit has a single DMS stage at a specific gravity cut point at approximately 2.88. The floats are a final reject stream, while the sinks are a concentrate. The concentrate reports to the magnetic separation and product handling circuit. The rejects are conveyed to a tailings' storage pile.

17.3.7 Ultrafine Dense Media Separation Circuit

The ultrafine DMS circuit is fed with material having a size fraction of -1.6 mm to +0.65 mm. In the same fashion as the fine DMS circuit, the ultrafine DMS circuit has a single DMS stage at a specific gravity cut point at approximately 2.88. The floats are a final reject stream, while the sinks are a concentrate. The concentrate reports to the magnetic separation and product handling circuit. The rejects are conveyed to a tailings' storage pile.

17.3.8 Recrush Dense Media Separation Circuit

The floats material from the second stage of the coarse DMS circuit (material with an SG of -2.9 to +2.65) will be directed to a High-Pressure Grinding Rolls (HPGR). The crushed product will have an 80% passing of 3.30 mm. The material will be fed to a vibrating screen with a cut size of 0.65 mm. The screen oversize will be sent to the recrush DMS circuit. The circuit will have one stage with a density cut point of 2.88. The sinks will be combined with the final product from the fine DMS circuit and fed to the magnetic separation circuit. The floats from the process will be treated as final tailings and combined with the rejects from the

fine DMS circuit. The screen undersize (i.e., the 0.65 mm material) will be sent to the bypass dewatering circuit.

17.3.9 Tailings (Bypass) Dewatering

The screen undersize (i.e., -0.65 mm) from the DMS preparation circuit and the recrush DMS circuit reports to a thickener. The thickener settles and thickens the solids. The thickener underflow is pumped to a belt filter. The thickener overflow will report to the facility's process water tank. The belt filter's cake will be discharged onto a conveyor belt. The cake is conveyed to a covered stockpile in the tailings handling area.

17.3.10 Tailings Handling

The filtered bypass fraction (i.e., -0.65 mm) and the rejects fractions from the three (3) DMS size fractions (i.e., the -2.65 SG from the primary coarse DMS circuit and the -2.88 SG from the fine, ultrafine and recrush DMS circuits) report to the tailings handling area. Once Phase 2 is operational, a proportion of the tailings will report to the paste backfill plant.

17.3.11 Magnetic Separation and Final Product Handling

The magnetic separation circuit removes the minerals with iron contaminants by using high-intensity magnetic fields to ensure that the final concentrate specification does not exceed the final iron impurity value (i.e., Fe_2O_3 < 2 wt.%). The +2.9 SG material from all four (4) DMS circuits (i.e., coarse, fine, ultrafine and recrush) is conveyed to the magnetic separation circuit. The combined concentrate is fed to a screen with a 5 mm cut size. The screen oversize (+9.5 mm to +5 mm) is directed to the coarse magnetic separators. The screen undersize is fed to the fine magnetic separators. The magnetic fractions from the units are dewatered via a screen. The dewatered screen oversize is conveyed to the tailings handling piles. The non-magnetic fractions, considered final concentrates, are dewatered with a dewatering screen. The dewatered concentrate is conveyed to a covered storage pile and will be loaded via a front-end loader into the product transport trucks.

17.4 Recommendations

The following opportunities exist for advancing the Project:

- The iron specification of <2% Fe_2O_3 was based on typical values within the spodumene market as of the time of the study. Current design expects to make a concentrate consistently below the iron specification. Additionally, there seems to be very little sensitivity to +/-0.1% Fe_2O_3 to the selling price of the concentrate. Future implications of a +/-0.1% Fe_2O_3 on the sales price of the concentrate

will determine the feasibility of certain project options post start-up of the Project. For example, depending on the sensitivity of iron contamination on spodumene, the sales price will determine whether the Project should install ore sorting or increased magnetic separation units.

- Further optimization of the concentrate magnetic separation circuit is recommended. Additional testwork should be undertaken to evaluate alternative technologies and operating conditions. In particular, further testing by the Changsha Mining and Metallurgy Research Institute (CRIMMS) is warranted to assess the unit's performance at elevated throughputs. Demonstrating the unit's ability to operate under overloaded conditions could enable simplification of the circuit by allowing all concentrate to be treated without bypassing separation on the coarser material, potentially improving both efficiency and product consistency.
- With the current understanding of the geological body (i.e., its substantial width and its nearly vertical orientation), an integrated ore sorting plant within the crushing circuit is not required. However, there is an opportunity to target blocks that contact the host rock to be directed to a distinct storage pile. Spodumene can be recovered from this pile via a modular ore sorting plant. This would allow for more recovery of spodumene from the deposit without feeding high amounts of external dilution (the main source of final iron contamination) to the plant.
- To increase revenue generation, there is potential to incorporate by-product recovery circuits into the overall processing strategy. Additional testwork and trade-off studies are recommended to evaluate the feasibility of producing tantalum, caesium and rubidium products. In particular, caesium recovery could be integrated with the modular ore sorting plant concept. While the ore sorter is primarily being considered for intermittent use in processing high-dilution material, it could also be repurposed, during low dilution periods, for producing a saleable caesium product from a separate deposit. This dual application would enhance the plant's flexibility and strengthen the justification for investment in the ore sorting unit.

18. PROJECT INFRASTRUCTURE

18.1 Project Overview

The Shaakichiuwaanaan Project infrastructure is designed to support the operation of an open pit (OP) mine in Phase 1 and an underground (UG) mine in Phase 2, feeding a processing plant with a nominal throughput of 2.5 Mt per annum in Phase 1 and an additional 2.5 Mt per annum in Phase 2, operating 24 hours a day, 7 days a week. It is designed in consideration of local conditions and topography at site.

18.2 Site Infrastructure

The site infrastructure plan is presented in Figure 18.1. It has been outlined to minimize environmental impacts on surrounding water bodies, improve vehicle traffic safety and distances, optimize construction costs and maximize operational efficiency and flexibility.

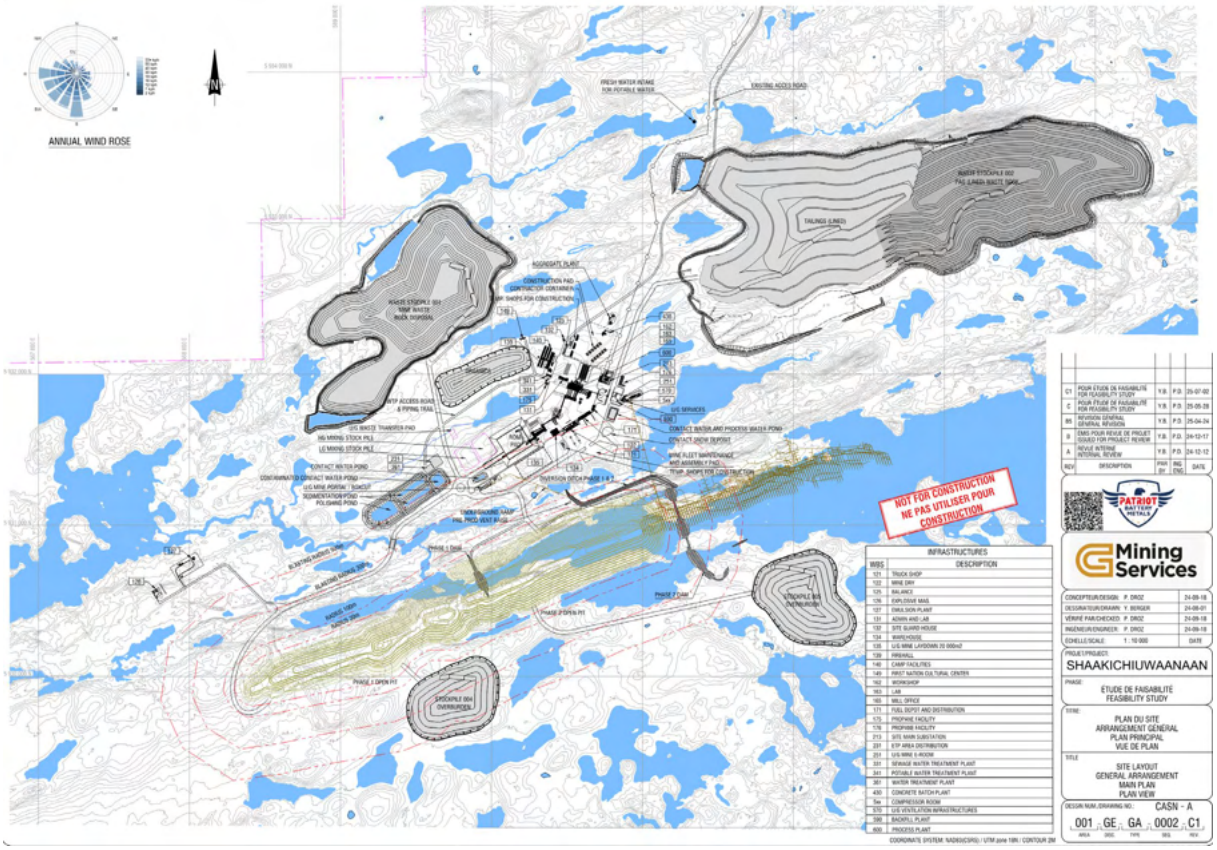
Where possible considering the building size, buildings will be made of prefabricated modules.

The main site infrastructure includes the following:

- Site main access road.
- Open-pit mine.
- Underground mine and portal to surface.
- Surface infrastructure for underground mine as mine ventilation and heating, UG raises to surface.
- Mine laydown area.
- Process plant (crusher and screening, crushed ore silos, DMS concentrators, concentrate and tailings loadouts).
- Paste preparation mill and backfill plant.
- Vehicle maintenance garage.
- Administrative offices, dry rooms, warehouses, laboratory and auxiliary buildings for the concentrator and the mine areas.
- Waste rock and tailings management piles with their associated ditching and basin systems for water management.
- Overburden piles storage with their associated ditching and basin systems.

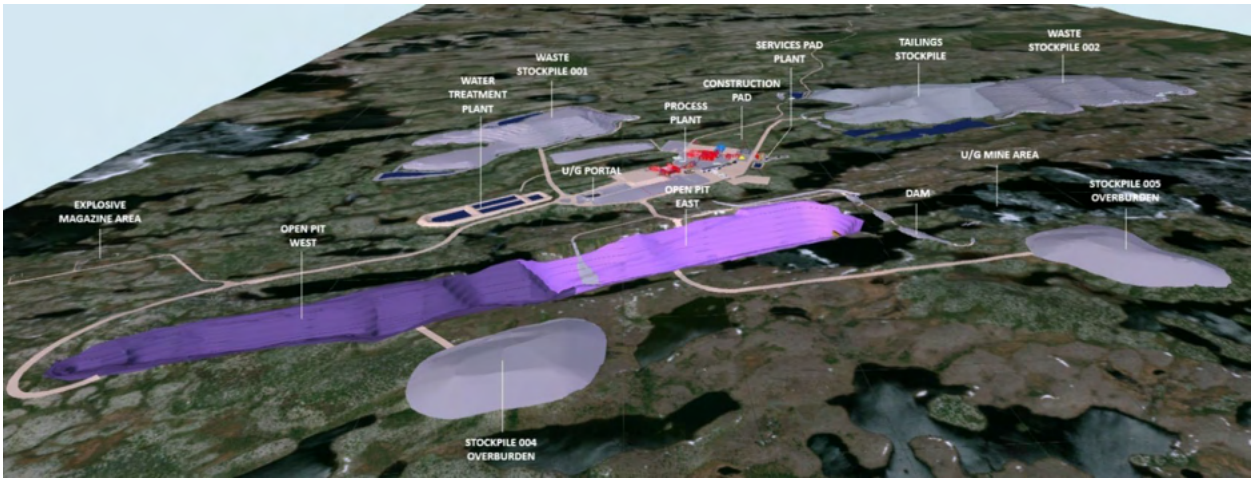
- Fresh / raw water lake intake and water treatment plants.
- Electrical substation and overhead electrical powerlines.
- Site roads and pads with their associated ditching and culvert, and bridge systems for drainage.
- Aggregate crushing plant area.
- Emulsion plant and explosive storage magazines buildings.
- Laydown area.
- Fuel storage pad and refuelling stations.
- Run-Of-Mine (ROM) pad.
- Water diversion dam and diversion channel for Lake 001.
- Permanent workers camp for construction and operational needs.
- First Nation cultural centre.
- Temporary construction facilities.

Figure 18.1: Main Site Infrastructure – Plan View



Source: GMS, July 2025.

Figure 18.2: Main Site Infrastructure – Perspective View



Source: GMS, July 2025.
**Note: Not to scale.*

18.2.1 Layout

The primary buildings have been strategically positioned to facilitate efficient construction access and to use the existing topography, thereby minimizing the volume of bulk earthworks. This placement also adheres to geotechnical recommendations, ensuring structural stability and safety and is based on environmental and exploration studies. The layout was oriented to minimize disruption to natural habitats and avoid sterilization of resources. All buildings are within the Property's limits.

All infrastructure will comply with the Canadian Building Code and the required North American standard codes to meet the Nordic climate. The main fire protection system will be an independent fire hydrant loop fed from clear process water. Where possible, infrastructure buildings have been designed as modular prefabricated structures combined with steel and cladding. Buildings will be equipped with smoke, carbon monoxide and heat detectors, as well as appropriate fire extinguishers with one (1) main fire alarm panel.

The camp area will be divided into several units, also referred to as dormitories, each consisting of rooms on a single level. The camp buildings, such as the Kitchen, Administration Office, Laundry, Recreational Centre, Gym and Ablution Units, will be placed within walking distance. This will optimize the electrical and piping network and create a PPE-free zone, away from the industrial area.

An access gate including the guard facility will be installed at the site entrance to control access to site and concentrate shipments. Preliminary screening of all traffic entering and leaving the property will be conducted at this location. Nearby truck weights will be managed from the main gate for concentrate shipment.

Permanent buildings are located outside the 500 m mine blast radius of the open pit mine to minimize flying rock and vibration risks, thus limiting downtime risks. The Emulsion Plant and explosive storage are located at a minimum of 860 m from other infrastructure to minimize risks.

The Run-Of-Mine pad is located on a natural hill near the open pit access ramp and adjacent to underground portals for direct access to the crushers and to minimize ore haulage distances. The ROM pad storage capacity covers the 1 Mt required during the LOM. When possible, material from the pit would be recycled to be used as construction material.

The camp is located near the processing plant and within walking distance, which reduces both transit time and transportation needs. Personnel assigned to the mine area will travel by bus or car to the mine garage, offices, and dry house. Noise mitigation measures have been considered around the camp due to its proximity to potential sources. Although haul roads, waste piles, and underground ventilation raise fans are located nearby, they are oriented in such a way as to minimize sound impact on the camp. Tree clearing and topsoil removal are expected to be required for all traffic areas and future building footprints. A mobile crusher will produce granular materials and aggregates for road and pad structures during construction using on-property borrow source material.

18.2.2 Access Roads and On-Site Roads

An existing exploration road provides a direct connection between the mine site and the Trans-Taiga Road (at KM-270). This road has an average running width of approximately 10 m, which is generally adequate to support early-stage and ongoing operational needs. Only localized improvements and targeted widenings will be required to bring it to the necessary operational standard. At present, the road facilitates the movement of personnel, equipment, and supplies between KM-270 of the Trans-Taiga Road, and the exploration camp (Camp Shaakichiuwaanaan) situated immediately south of KM-270, to the mine site. From Camp Shaakichiuwaanaan, the road continues to the location of the future main operations camp, providing a reliable transportation link during the construction and pre-production phases.

Within the mine site boundaries, a planned network of gravel-surfaced roads will be constructed to accommodate both light-duty vehicles and heavy mobile equipment. These internal roads will connect all major surface infrastructure facilities, including the process plant, workshops, fuel storage areas, material stockpiles, explosives facilities, and waste rock management areas. Their design will ensure safe, efficient, and continuous access for day-to-day mining and support activities. Road widths, alignments, and structural layers will be adapted to the intended traffic type and frequency. The overall road network layout has been planned to minimize operational interference and ensure seamless integration with the main site access road.

18.2.3 Haul Roads

Haul roads are designed to accommodate 140-tonne class rigid-frame haul trucks, ensuring safe and efficient transport of ore and waste rock between the open pit mining areas and key surface infrastructure. These roads will serve as the primary heavy equipment transportation corridors, linking the pits to the primary crusher, maintenance shop, fuel bay, tailings management facility, and waste rock and overburden stockpiles. Their alignment has been planned to minimize gradients, reduce travel distances, and avoid bottlenecks, while maintaining clear separation from light-vehicle traffic wherever possible to improve safety and productivity.

Haul roads will be 30 m wide and constructed with a granular base suitable for heavy-duty mine traffic. Drainage ditches will be constructed along both sides of the roads to control surface water, prevent erosion and maintain road integrity throughout the year. Where the elevation difference between the road surface and adjacent natural ground exceeds 3 m, safety rock berms will be installed. Two (2) stream crossings are required along the planned haulage routes. These will be constructed as permanent, built-in-place bridges with clear spans of approximately 30 m. Bridge structures will be engineered to accommodate both the static and dynamic loading of fully loaded 140-tonne haul trucks, with appropriate allowances for impact, braking, and environmental conditions. The bridge approaches will be integrated into the haul road design to provide smooth, continuous transitions and maintain optimal haul truck operating speeds.

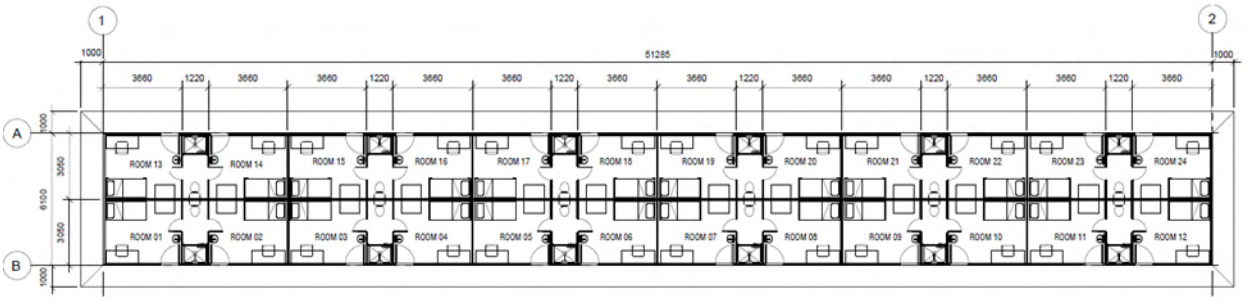
18.3 Camp Infrastructure

18.3.1 Dormitories

The permanent camp is designed to accommodate a maximum of 480 workers at site during Phase 1. During Phase 2, an additional 192 beds will be installed to house the workers. Additionally, the 150-bed existing Camp Shaakichiuwaanaan will be available at the time of construction when the total workforce needs will exceed the capacity of the operations camp.

The buildings will be divided into several units, also referred to as dormitories, each consisting of rooms on two (2) levels. Dormitories of type B will be installed, each with a capacity of 96 beds and consisting of one (1) bathroom for every two (2) rooms, plus one (1) janitor room and one (1) E-room. Figure 18.3 illustrates the typical camp dorms on one (1) of the two (2) levels for the type B camp.

Figure 18.3: Typical Camp Dorm – Type B



Source: GMS, July 2025.

18.3.2 Kitchen and Lunchroom

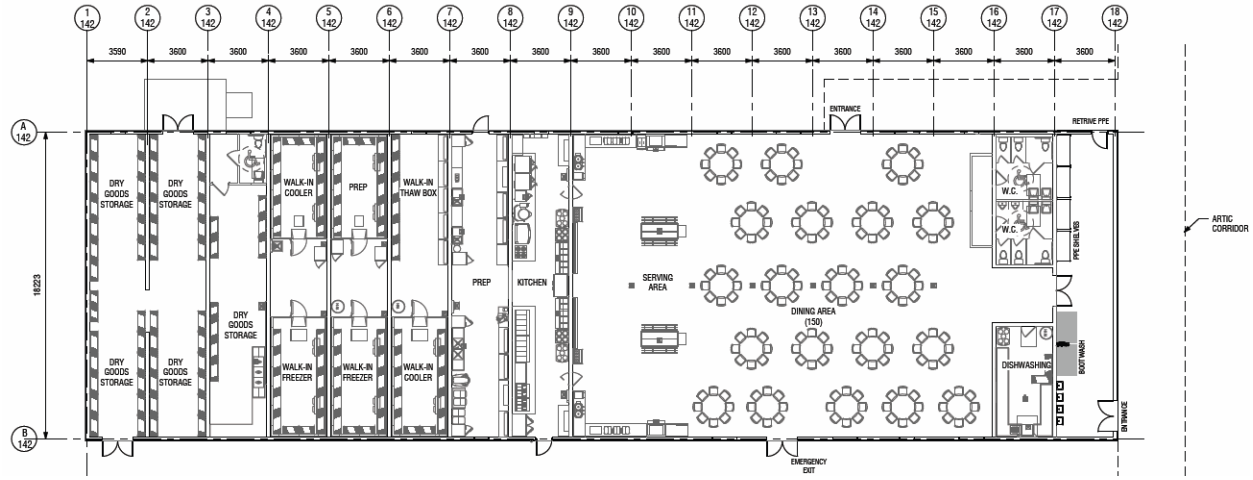
The kitchen and lunchroom facilities are designed to accommodate food service for approximately 150 individuals at a time. The layout includes dedicated cooking zones, serving areas, hygienic processing sections, administrative offices, freezer rooms, and storage spaces. Each cooking station will be equipped with commercial-grade appliances and compliant ventilation systems to support high-volume meal preparation.

Designated areas for meal preparation and distribution are optimized for operational efficiency, while cleaning zones are configured to meet all relevant health and safety standards. The facility also provides office space for management and staff meetings, along with specialized freezer rooms to maintain food safety. A centralized and structured storage area will facilitate streamlined inventory management.

To minimize heat and noise impact, HVAC equipment will be installed externally and acoustically insulated. A dedicated maintenance road will encircle the complex to provide discrete service access and prevent interference with pedestrian activity at the main entrance.

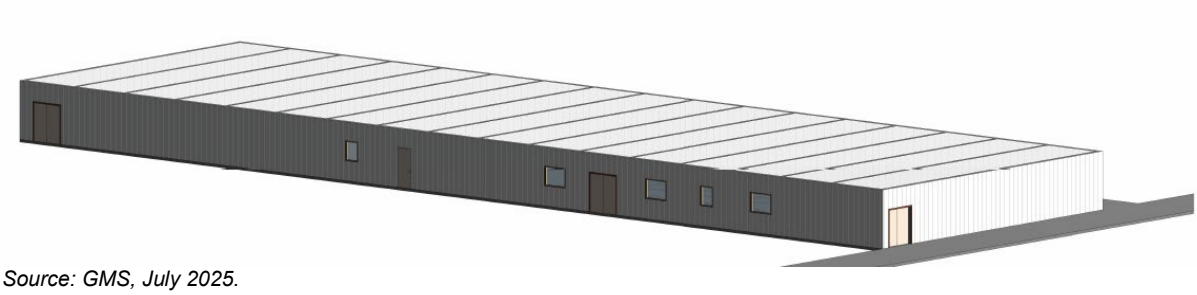
This configuration ensures a safe, efficient, and comfortable environment, with circulation and spatial planning carefully engineered to support optimal workflow. The planned kitchen and lunchroom layouts are illustrated in Figure 18.4 and Figure 18.5.

Figure 18.4: Kitchen Plan View



Source: GMS, July 2025.

Figure 18.5: Kitchen 3D View



Source: GMS, July 2025.
**Note: Not to scale.*

18.3.3 Camp Office and Welcome Centre

The camp office is a single-story structure with an approximate area of 750 m², situated adjacent to both the laundry facility and the recreational room. It will accommodate offices and open-plan spaces for the Transportation Supervisor and Camp Supervisor, along with meeting rooms, a janitorial area, a storage room, and a washroom. Figure 18.6 and Figure 18.7 provide visual representations of the camp office and the adjacent laundry area.

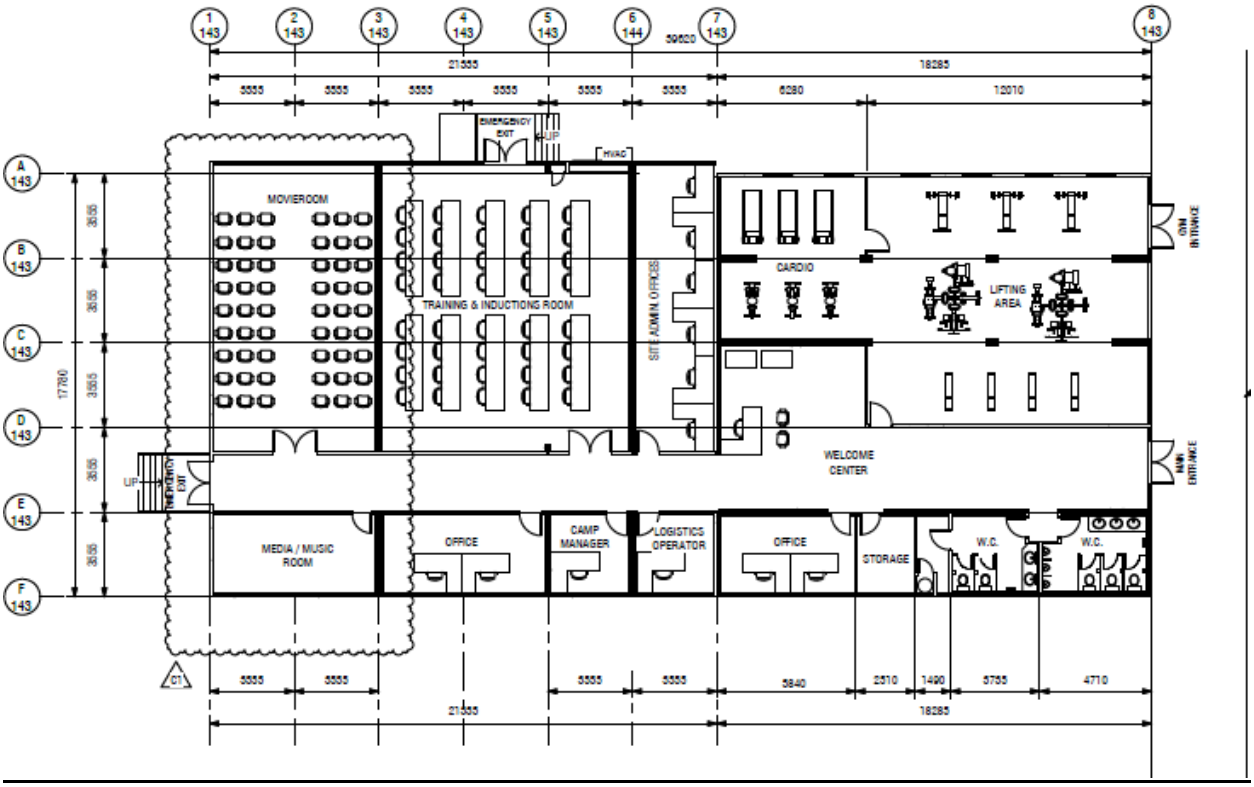
The gymnasium will include ablution facilities, dedicated zones for exercise equipment, spaces for aerobic activities, and additional areas to support a range of fitness programs. Recreational amenities will also feature sports fields, offering ample space for various outdoor activities.

The laundry facility will be outfitted with industrial-grade electric dryers and washing machines. Airflow within the space will be balanced by introducing fresh air to offset the volume extracted by the dryers, ensuring proper ventilation and operational efficiency.

Planned future expansions of the Recreational Centre include the addition of a game area, internet access points with shared computer terminals, a television room, lecture spaces, a convenience store, and other complementary amenities.

All facilities are strategically positioned near the main accommodation area, ensuring convenience while maintaining a sufficient buffer to reduce noise and minimize disruptions to pedestrian circulation.

Figure 18.6: Camp Office and Welcome Centre, Plan View



Source: GMS, July 2025.

Figure 18.7: Camp Office and Welcome Centre, 3D View



Source: GMS, July 2025.
**Note: Not to scale.*

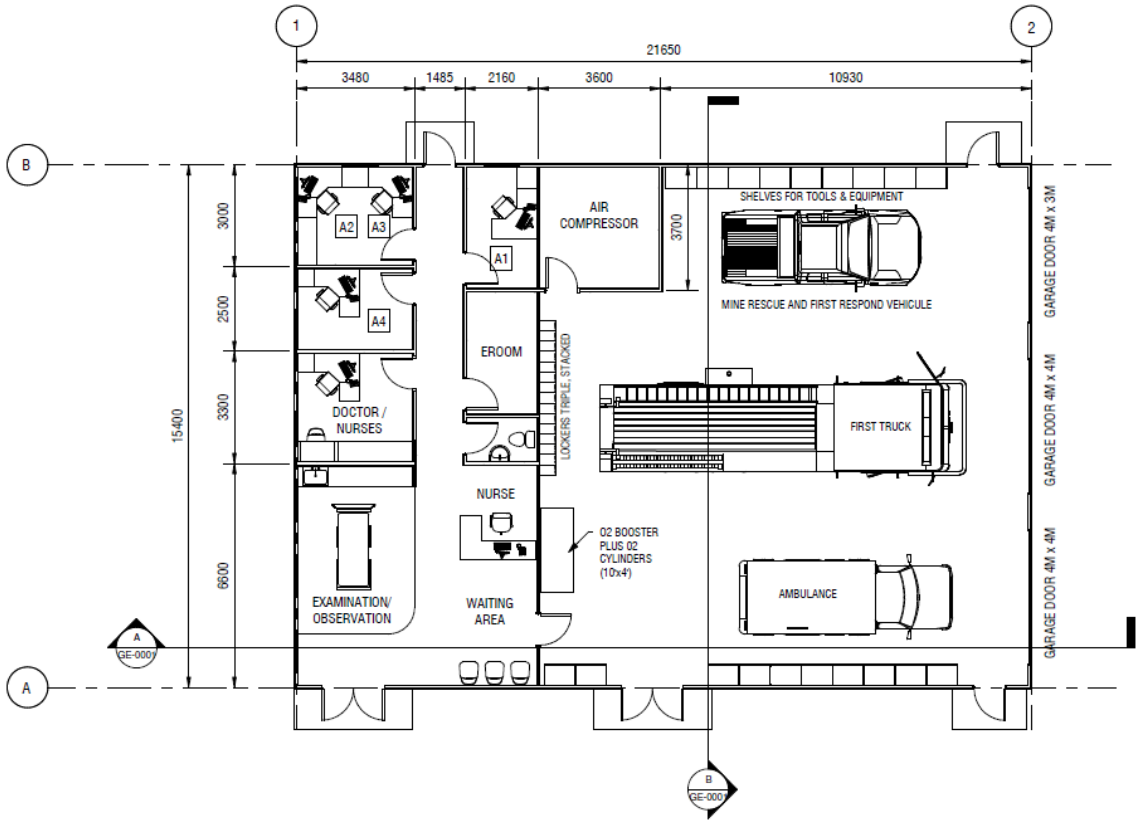
18.3.4 Fire Hall and Clinic

The Fire Hall and Clinic building is a single-story facility of approximately 330 m². It has been strategically designed to support both emergency response and occupational health services within the site.

The clinic section includes an examination room, a nurse station, ablution units, and a dedicated doctor's office. These spaces are configured to facilitate efficient medical evaluations, treatment, and first-aid response. In addition to clinical functions, the building will also house offices for the Health and Safety staff, ensuring close operational oversight and coordination of safety initiatives.

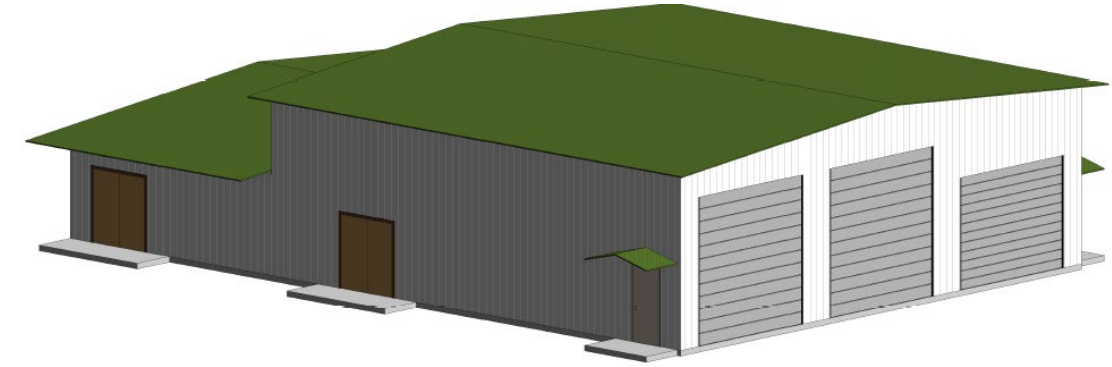
The facility further includes an air compressor room to support essential mechanical systems, as well as a dedicated space for critical emergency vehicles. This includes accommodations for a mine rescue and first response unit, a fire truck, and an ambulance, all of which will be housed to allow rapid deployment in the event of an incident.

Figure 18.8: Fire Hall & Clinic, Plan View



Source: GMS, July 2025.

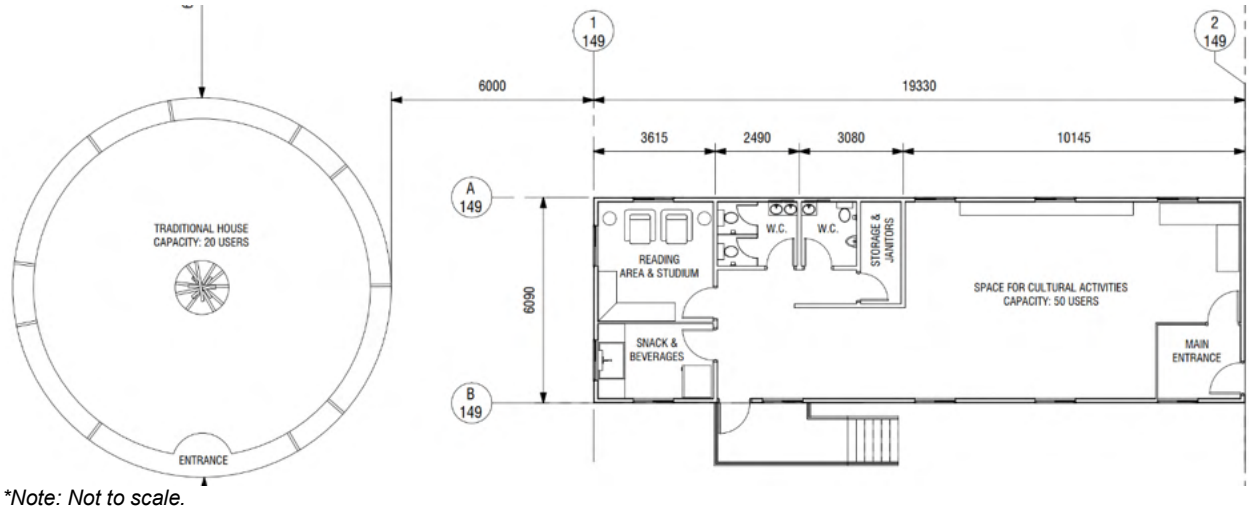
Figure 18.9: Fire Hall & Clinic, 3D View



Source: GMS, July 2025.
**Note: Not to scale.*

18.3.5 First Nations Cultural Centre

The First Nations Cultural Center will be in front of Lake 219, with spaces reserved for meetings and areas for traditional activities. This facility includes restrooms, a cold kitchen and wood stove, and a library. Installed adjacent to this building, a traditional house will be positioned, covered in a waterproof canvas, with a combination of traditional and modern materials.



**Note: Not to scale.*

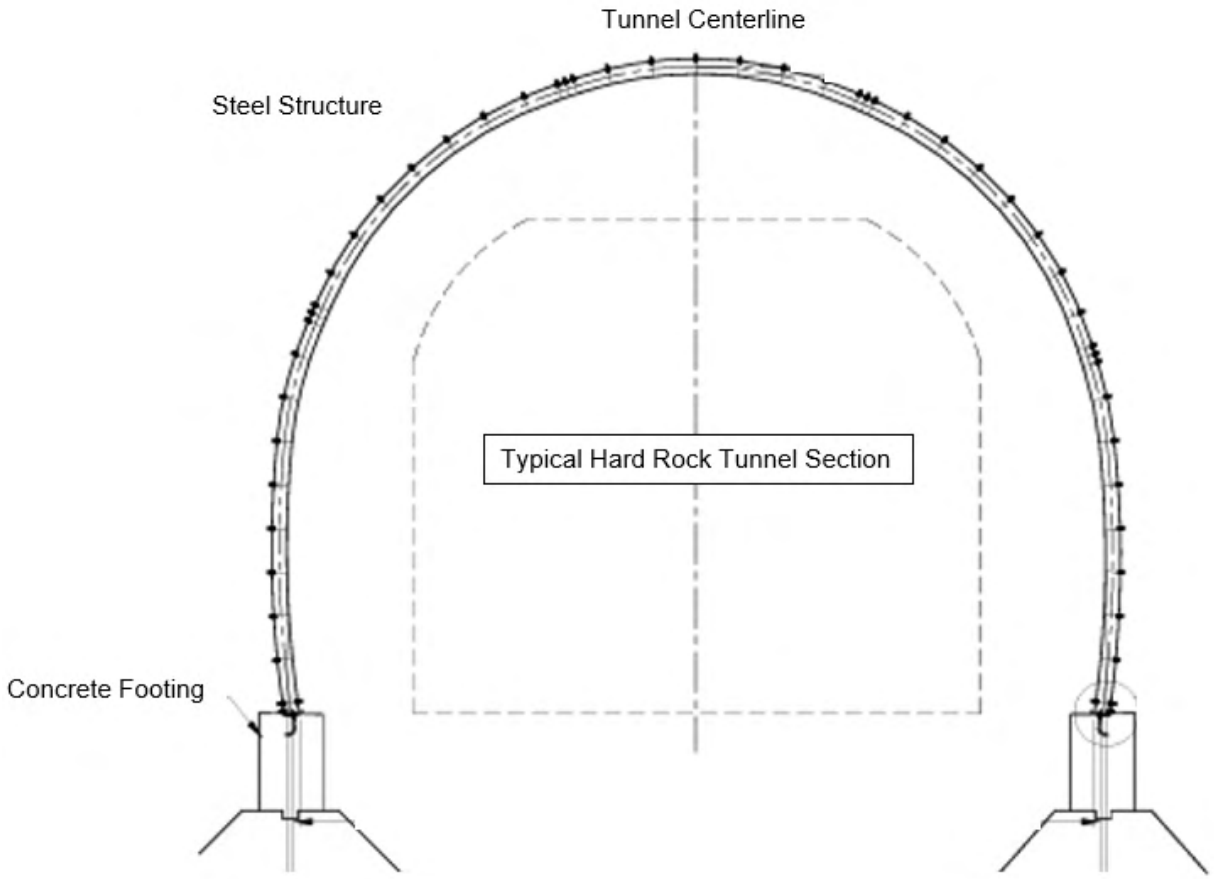
18.4 Mine Infrastructure

18.4.1 Mine Portal Area

The underground access, designed as the portal, will be established through a single main decline located on the south side of the ROM pad. The outdoor portion of the decline will be covered by a bolt-on lightweight

deep-corrugated galvanized steel plate over a distance of 102 m (Figure 18.10). This structure helps keep cold air, snow and rain out of the underground mine, increasing operation safety and productivity in a northern environment. A reinforced concrete collar will be constructed at the rock junction to provide long-term stability and allow for the installation of the steel plates. The portal has been positioned to minimize initial waste development, ensure geotechnical stability, and reduce haul distances to the process plant and surface infrastructure. Surface facilities at the portal area will include a laydown yard and temporary equipment for portal and ramp early development, as a compressor station, mine water management sumps and a temporary ventilation system. Further details about the underground development and portal are provided in Section 16.

Figure 18.10: Typical Deep-Corrugated Galvanized Steel Plates Portal



**Note: AIL quotation. Not to scale.*

18.4.2 Ventilation Air Raises

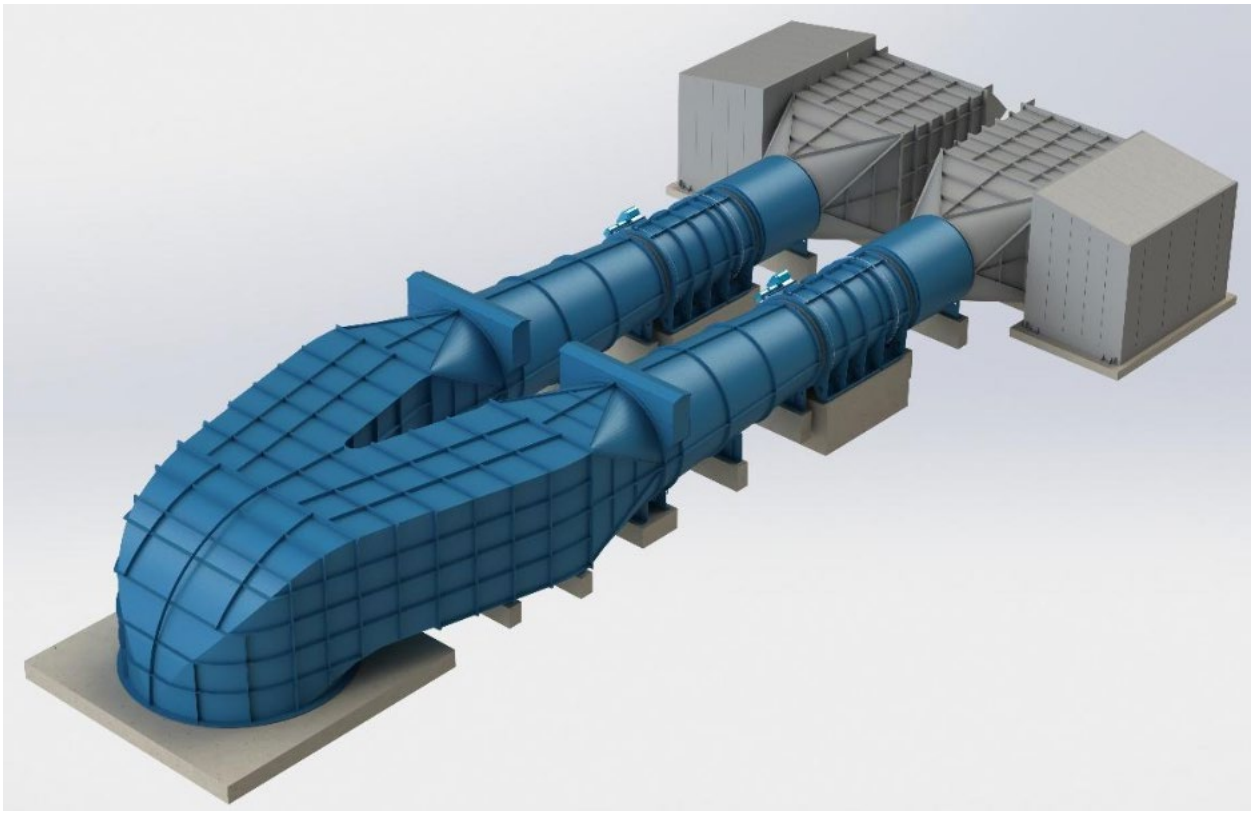
The underground ventilation system includes a dedicated vertical raise constructed to provide fresh air intake to the mine workings, two (2) fans and two (2) heating units (Figure 18.11). The fans will be housed within an insulated building to protect equipment from severe winter conditions and to facilitate routine

maintenance. The fresh air raise is located 80 metres from dust and air contaminant sources to prevent ingress to the system. Overburden will be removed in the vicinity of the ventilation raise to access the rock mass. The ventilation raise will be excavated to a final diameter of 5.5 m. Then, a concrete and steel collar will be erected before filling the hole with the stored overburden. Then, the main elbow linking the fans to the collar will be installed.

Given the northern climate, the intake raise will be equipped with a heating system to ensure a minimum supply air temperature during the winter months. Heating will be provided through direct-fired units using compressed natural gas (CNG), stored in surface mobile tanks, and decompressed prior to combustion. The system is designed with redundancy to maintain continuous heating capacity under peak winter conditions. Heated air will mitigate risks of ice accumulation within the raise and underground workings and provide above-freezing-point temperature for underground operations, supporting safe and efficient operations year-round.

A separate exhaust raise will be built during ramp development and used as an air intake at the initial phase. Once the permanent fresh air raise will be built in Phase 2, the first raise will switch to exhaust to complete the mine’s primary ventilation circuit. Further details about the underground mine ventilation system are provided in Section 16.

Figure 18.11: Underground Mine Main Ventilation Set-up



Source: Zitron Offer, 2025..
**Note: Not to scale.*

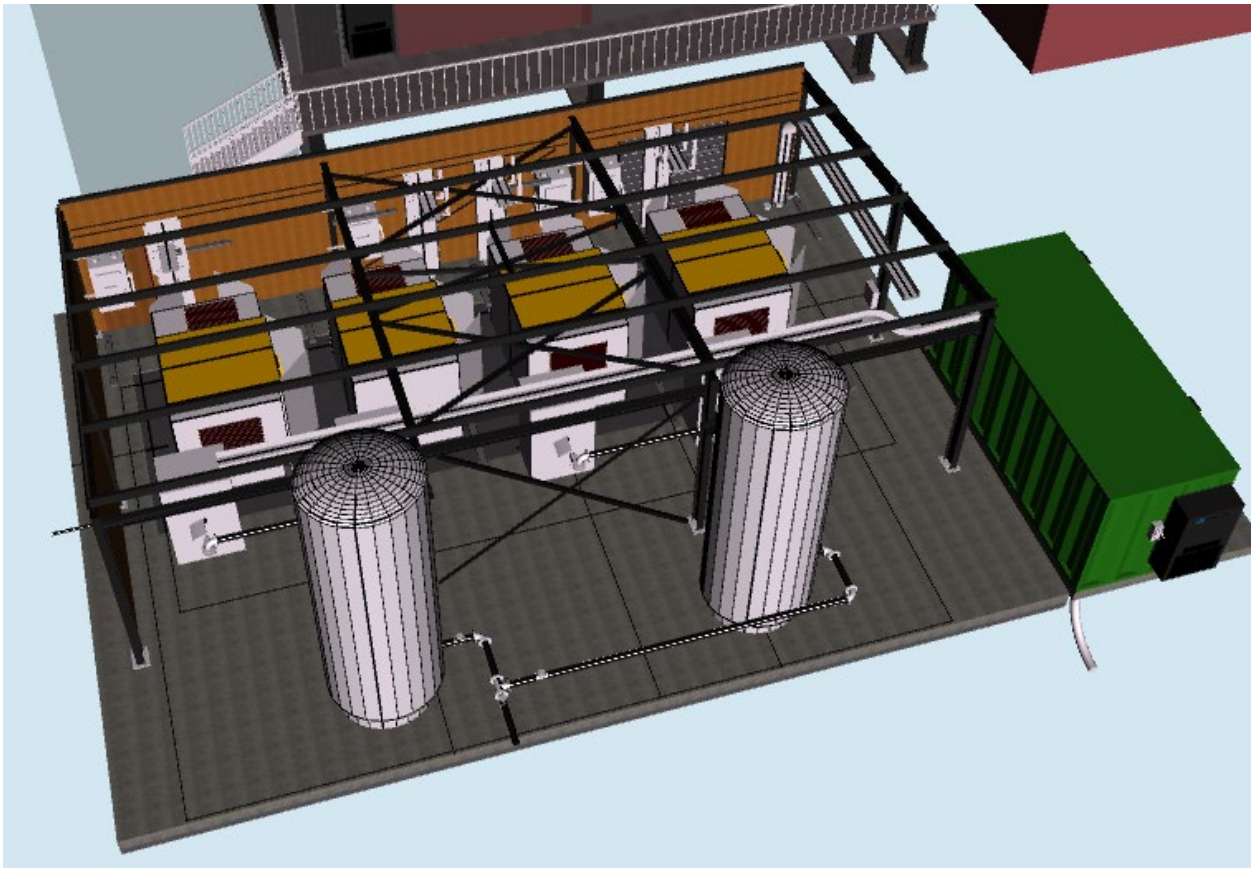
18.4.3 Underground Services

The services needed in the underground mine are compressed air and industrial water. Pipes will source these services via the paste plant building and using nearby boreholes and will reach the underground ventilation drift linking the fresh air raise to the rest of the mining development.

Clean water is provided from the process plant and transferred to the paste plant into a retention tank. From that tank, water will be pumped underground through a set of pumps.

The compressed air building is adjacent to the paste plant building. The system network will be made of four (4) compressors to provide the required air volume and pressure for the underground fleet.

Figure 18.12: Compressor Building



Source: GMS Database.
**Note: Not to scale.*

Once decanted underground, underground water will be pumped through pipes to exit through the portal. Nearby ponds will capture and further decant water prior to being sent to the water treatment plant. Contaminated water generated by underground activities will be primarily treated for TSS and hydrocarbons. Further details about mine services are provided in Section 16.

18.4.4 Mine Garage and Wash Bay

The Truck Shop is a purpose-built facility designed to support vehicle maintenance and industrial operations. It includes a combination of heavy-duty service bays, a truck wash bay, administrative functions, and technical support areas for mine development, all organized to ensure safety, efficiency, and optimal workflow.

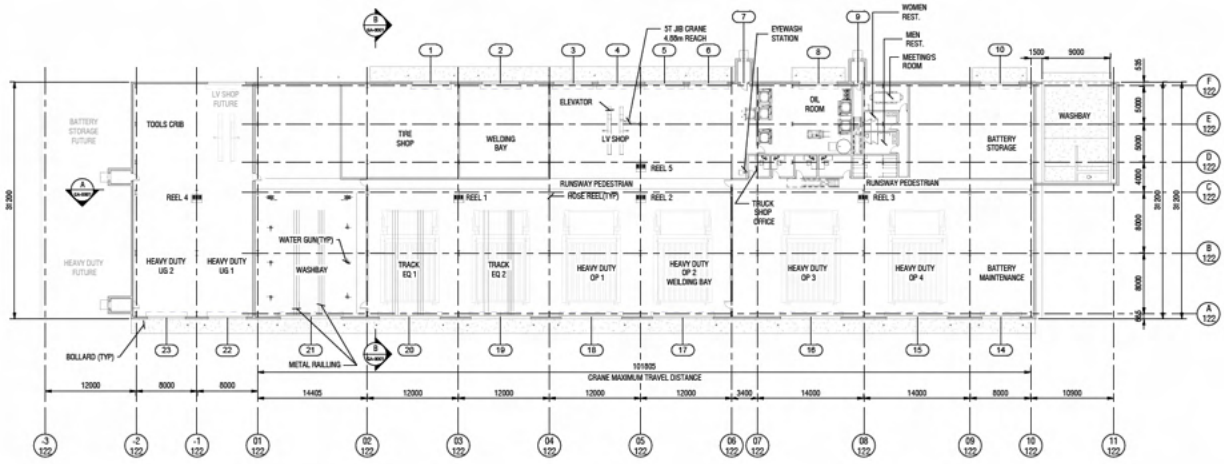
The ground floor layout will accommodate four (4) heavy-duty OP equipment operating bays, two (2) UG equipment bays, two (2) tracked equipment bays, a welding bay, a tire shop, battery storage, battery maintenance, a wash bay, and a dedicated light vehicle shop. These workspaces will be equipped for

high-load mechanical activities and support infrastructure, including hose reels, water guns, and designated crane zones. A 25-t overhead crane spans the workshop area, with a 5-t jib crane offering extended reach for localized lifting tasks.

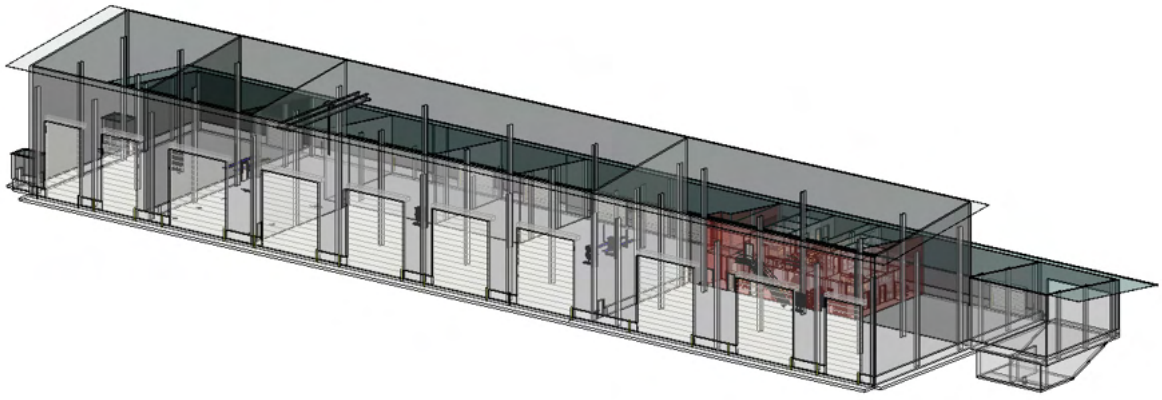
Administrative and support spaces will be located within the mezzanine level and include mechanical and electrical rooms, a meeting room, and a coffee station. The mezzanine will also include a series of open-plan offices or workstations for technical personnel and supervision staff.

Additional features include men's and women's restrooms, a janitorial room, and truck shop offices strategically placed for oversight. The inclusion of safety infrastructure such as eyewash stations and mechanical ventilation underscores the building's compliance with occupational health standards.

Figure 18.13: Truck Shop Layout



Source: GMS, July 2025.



Source: GMS, July 2025.
**Note: Not to scale.*

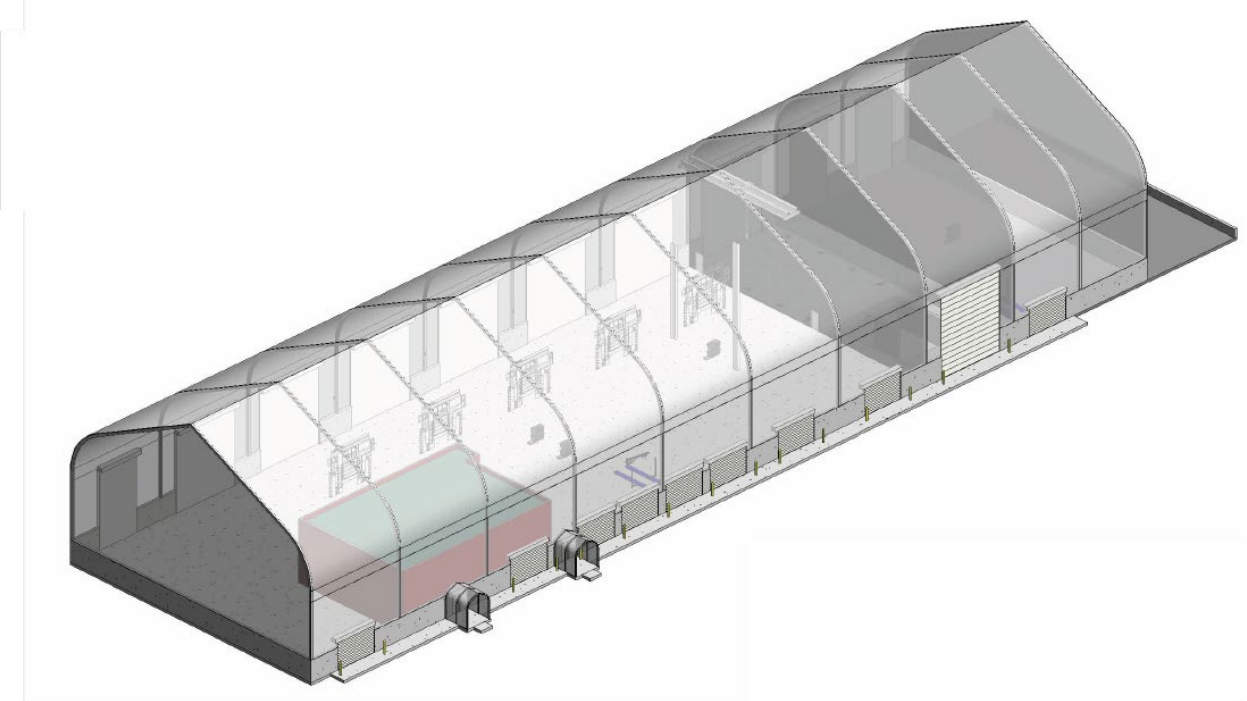
18.4.4.1 Temporary Truck Shop

To support early mine activities, a temporary truck shop will be installed. The layout includes multiple service bays for light vehicle maintenance, supported by specialized work areas such as a welding bay, tire shop, oil room, and battery storage. A wash bay is integrated for vehicle cleaning, with appropriate drainage and water management systems. Safety infrastructure is incorporated through eyewash stations, hose reels, and water gun stations placed at strategic points.

The facility will also include three (3) heavy-duty vehicle bays and two (2) tracked equipment bays. For large maintenance tasks when lifting will be required, the mobile crane will have sufficient room to be placed inside or on the external pad.

The combination of an insulated dome structure, specialized work areas, and integrated support facilities makes the Temporary Truck Shop a robust and adaptable maintenance hub tailored for demanding northern conditions.

Figure 18.14: Temporary Truck Shop – 3D View



Source: GMS, July 2025.
**Note: Not to scale.*

18.4.5 Fuel Storage and Distribution

The fuel bay will include a section for light-duty vehicles and a separate section for heavy-duty vehicles. The site layout has been designed in such a way that the circulation paths for light and heavy vehicles are fully segregated, ensuring that both types of traffic can operate independently without intersecting. This spatial separation enhances safety, reduces congestion, and improves the overall efficiency of fueling operations.

The fuel bay will consist of three (3) horizontal steel diesel tanks, each with a capacity of 50,000 litres, as well as a 20,000-litre tank for clear diesel for Phase 1, and an additional three (3) tanks of 50,000 litres in Year 3. This storage is designed to ensure five (5) days of autonomy at peak consumption without requiring delivery. The work also includes safety access walkways, grounding systems for lightning protection, and perimeter fencing for security and access control around the depot area.

The fuel distribution network consists of two (2) independent circuits:

- The heavy vehicle section is equipped with a complete piping network, including all supports and accessories necessary to transfer diesel from the storage tanks to the high-flow dispensing nozzles.
- The light vehicle section is equipped with a similar, lower-flow network, compatible with both gasoline and clear diesel, tailored to smaller vehicle refuelling.

A dedicated loading / unloading area is integrated into the fuel depot. It includes:

- A complete piping system with transfer pumps and accessories.
- A heated 20-ft insulated container housing diesel and gasoline transfer equipment.
- Safety mechanisms such as emergency shut-off, spill protection, and overfill alarms.

A hydrocarbon separator unit is installed downstream of the tanker unloading zone. The separator is housed in a concrete sump equipped with a steel grate, allowing it to capture and treat any accidental fuel discharge before it enters the drainage system.

The fuel management system includes RFID card access and real-time monitoring of fuel usage. It also includes a leak detection unit designed to trigger alarms and automatically shut down pumps in the event of a leak or fire.

18.4.6 Compressed Natural Gas (CNG)

Heating for the concentrator, underground mine, and auxiliary buildings will use 100% compressed natural gas (CNG). CNG will be delivered to site by a third-party provider using high-pressure trailer systems, which serve as both delivery and mobile storage units.

A dedicated decompression skid will be installed onsite to safely reduce gas pressure from trailer levels (up to 4,300 psi) down to building distribution pressure (5–100 psi). This skid includes heat exchangers, safety valves, and monitoring systems.

The system will be designed with enough CNG trailer capacity to ensure 4 days of peak heating autonomy. Trailer rotation will be managed by the supplier, ensuring a continuous and reliable fuel supply throughout the winter heating season.

18.4.7 Mine Dry and Mine Office

The Mine Dry and Administrative Building is a multi-functional facility designed to support both operational readiness and administrative coordination for site activities. The layout integrates personnel changing areas, hygiene facilities, and office spaces into a cohesive structure that promotes efficiency, safety, and comfort.

The mine dry section includes dedicated men’s and women’s locker rooms, equipped with individual lockers, wash areas, and associated ablution facilities. A boot wash area is positioned near the entrances to maintain cleanliness and control the spread of debris within the building. The design also accommodates future expansion space for additional IT and support functions.

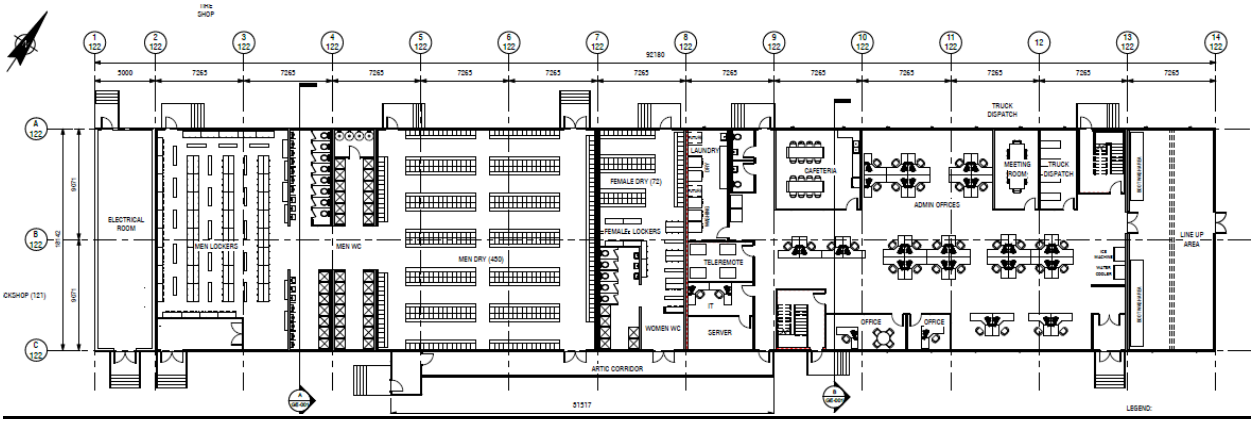
Administrative areas are organized to provide oversight and logistical coordination. These include offices for supervisors and dispatches, a meeting room, a server room, and dedicated truck dispatch control areas, as well as a tele remote operations office. Connectivity between the mine dry and administrative zones is maintained via an articulated corridor, allowing efficient movement while keeping work zones clearly delineated.

Supporting amenities include a cafeteria for personnel, an electrical room, and a laundry area for workwear maintenance. Circulation flows are optimized to manage peak shift-change traffic, with direct links between locker areas, boot wash, and the main access points.

The facility's configuration ensures that operational, safety, and comfort requirements are met, providing a central hub for both workforce preparation and administrative control within the mining complex.

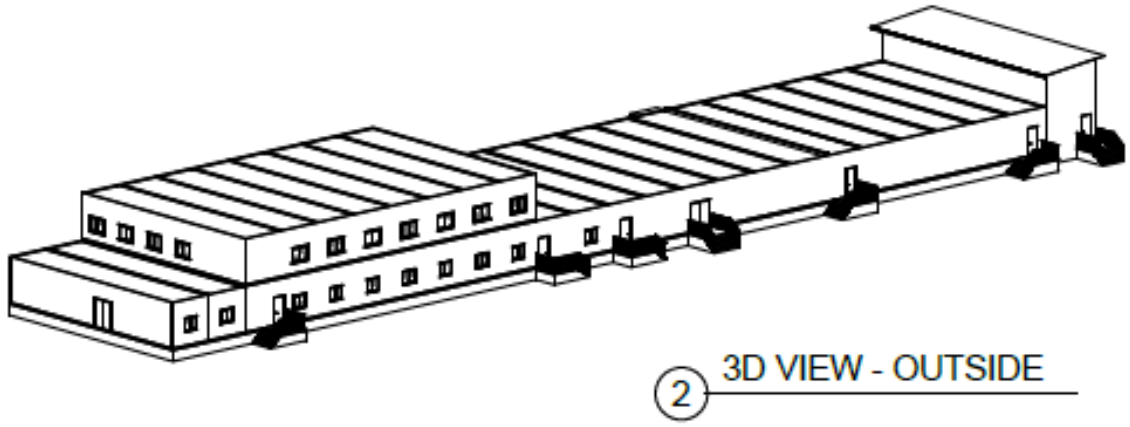
The general location of the dry and mine administration is beside the truck shop, considering the traffic segregation implemented in the mine for operations with security.

Figure 18.15: Dry & Mine Administrative Building - Plan View



Source: GMS, July 2025.

Figure 18.16: Dry & Mine Administrative Building - 3D View



Source: GMS, July 2025.
**Note: Not to scale.*

18.4.8 Warehouse

The Warehouse is a thermally insulated, steel-framed structure designed to accommodate storage, receiving, and distribution operations. Its envelope incorporates thermoacoustic panel cladding and a fibre-insulated dome with a footprint of 1,390 m^2.

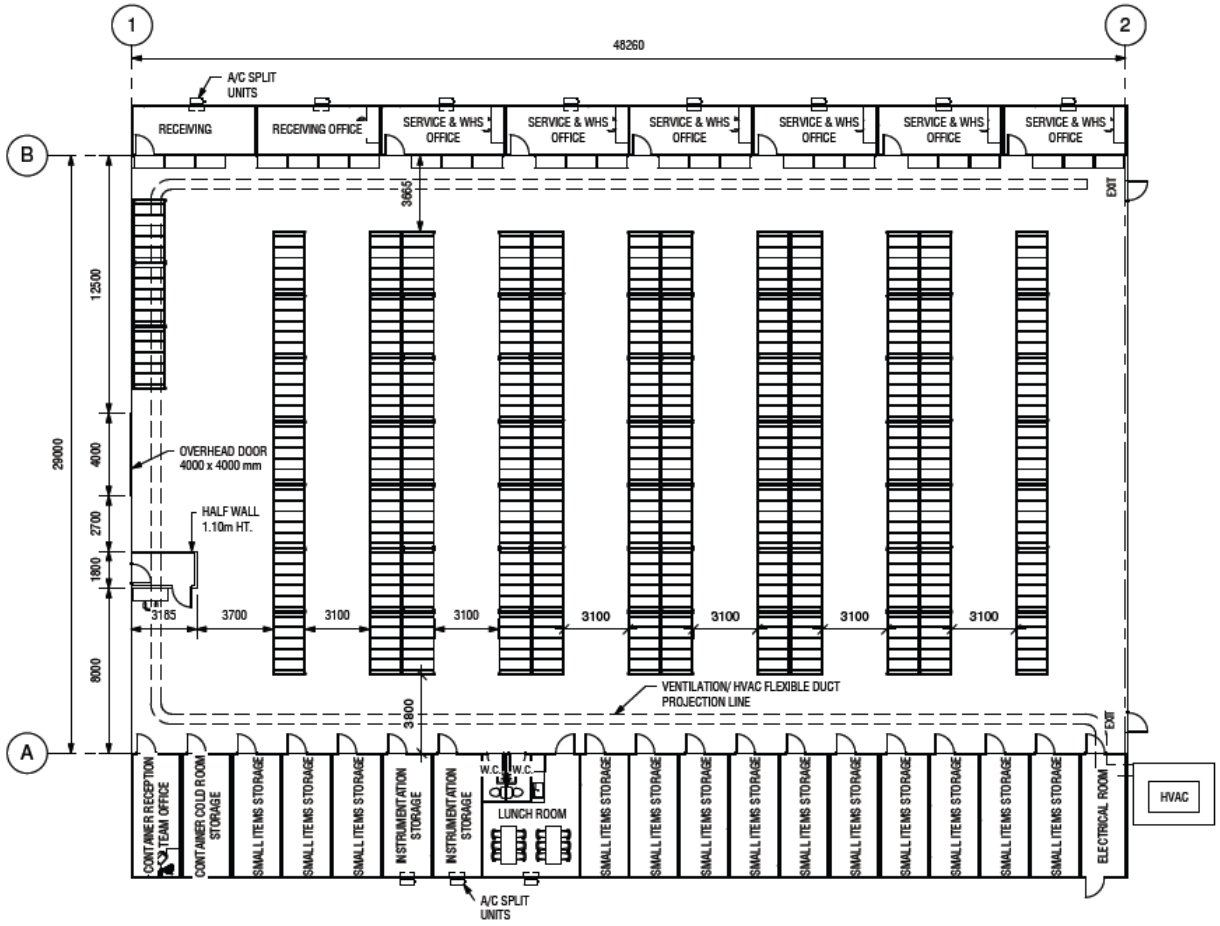
The internal layout is organized to optimize material handling and operational flow. Along the northern façade, the building houses the receiving area, a receiving office, and multiple service and warehouse offices. These spaces support administrative oversight, coordination, and logistics management. The receiving zone is serviced by a 4 m x 4 m overhead door for efficient vehicle access, with a half-height partition wall defining operational zones.

The core of the warehouse is dedicated to organized storage, including container reception, cold storage rooms for temperature-sensitive goods, and instrumentation storage. Extensive small-items storage is distributed along the southern and eastern perimeters, allowing for systematic inventory control. An electrical room and a centrally located lunchroom provide essential building services and amenities for staff.

Ventilation is managed through HVAC equipment and flexible ducting, complemented by A/C split units for localized climate control. Circulation within the building is designed for both pedestrian safety and material flow efficiency, with clear divisions between storage, office, and loading functions.

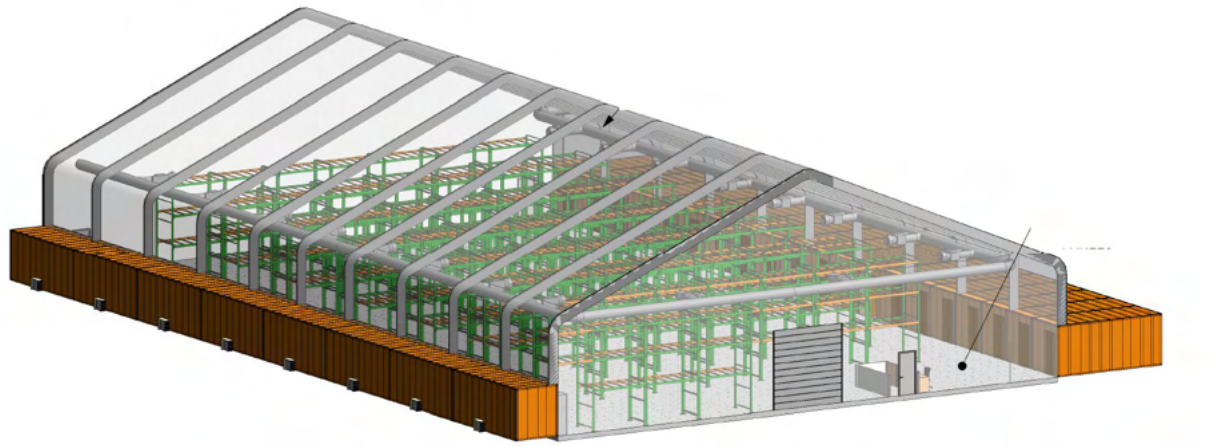
The warehouse’s structural design, climate control features, and functional zoning support high-capacity storage operations while ensuring worker comfort, safety, and operational reliability.

Figure 18.17: Warehouse Plan View



Source: GMS, July 2025.

Figure 18.18: Warehouse – 3D View



Source: GMS, July 2025.
**Note: Not to scale.*

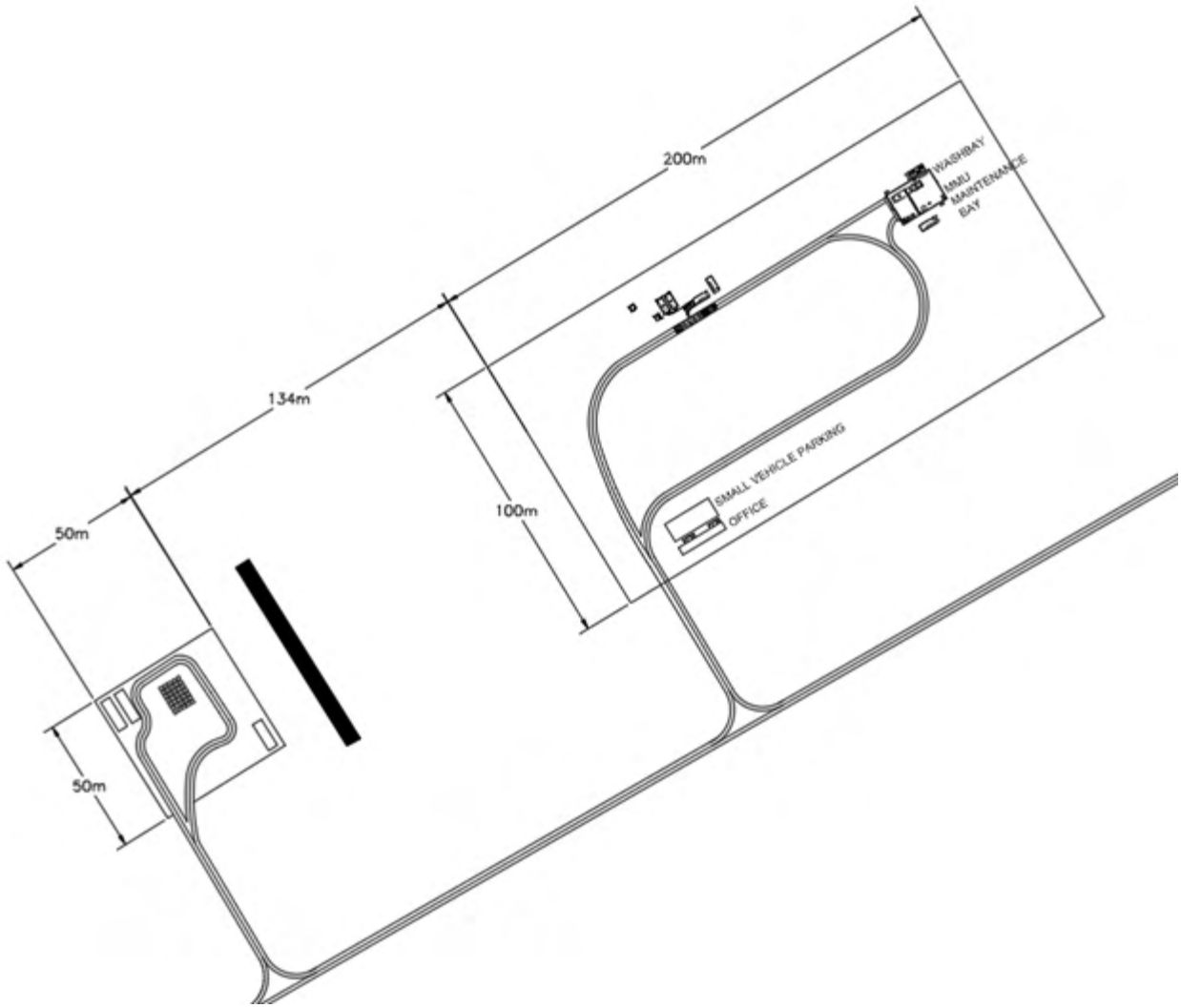
18.4.9 Explosive Storage

The explosive storage facility (Figure 18.19) will comprise two (2) primary components: a bulk emulsion depot with a storage plant and designated explosive magazines, co-located at a secured area on the mine site. This infrastructure is designed to support the full spectrum of blasting activities during mine operations.

A fleet of two (2) MMUs (Mobile Manufacturing Units) will be dedicated at site. The second MMU will be added to the fleet in a timely manner as a spare unit to accommodate scheduled maintenance and breakdowns.

The explosive requirement will be 63,000 kg of 100% emulsion per week. The bulk depot will include two (2) 25 mt ANE iso-containers, a garage / wash-bay for one (1) MMU and containers for mixing gasser products, a water evaporator for contaminated water treatment and other supplies. Moreover, a 12 m explosives magazine and a 3.6 m detonator magazine will be included. The spare MMU will be kept outside or can be kept in other garage / facilities if the MMU is decontaminated. The bulk depot sizing will provide 5.5 days of capacity without delivery.

Figure 18.19: Explosive Storage Area



Source: GMS, July 2025.

No infrastructure will be required for the underground operations, other than underground magazines. Emulsion will be delivered in 1,500 kg bins from the emulsion plant, and 65 bins total will be required.

The explosive storage area will be enclosed within a secured and fenced perimeter, with controlled access gates and regular security patrols.

18.5 Process Infrastructure

18.5.1 Mill Office and Laboratory

The Administrative and Laboratory Building is a purpose-built facility designed to consolidate office, meeting, and technical laboratory functions under one roof, supporting both operational management and specialized analytical work.

The administrative section includes ten (10) enclosed offices, a work area with 30 dedicated workstations, two (2) meeting rooms, and a lunchroom. Additional administrative support spaces include a copy / printing room and storage rooms. Circulation is organized to maintain efficient access between work areas while minimizing disruption to laboratory operations.

The laboratory section is equipped for a range of analytical and preparation activities, including:

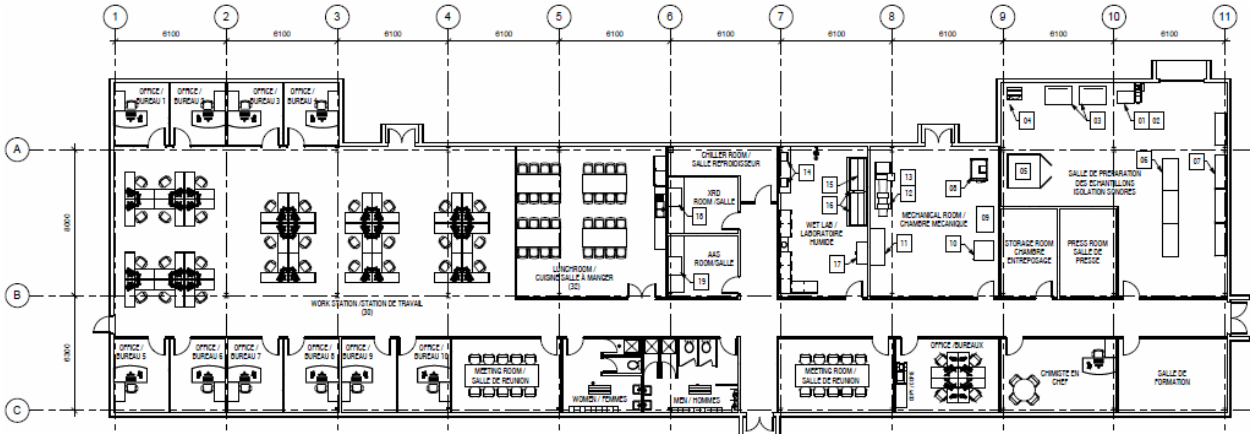
- Wet laboratory.
- Sample preparation room.
- X-Ray Diffraction (XRD) room, Atomic Absorption Spectrometry (AAS) room.
- Chiller room.
- Press room.
- Storage of instrumentation and chemicals.
- Dedicated acid cabinet.

Laboratory equipment, such as a jaw crusher, rotary splitter, multiple-head pulverisers, sample drying oven, fume hoods, dust collectors, and other analytical apparatus, is integrated into the design to support precise and reliable testing.

Support areas include mechanical rooms, a chemist’s office, and a training room, ensuring the facility can host both day-to-day operations and specialized staff development activities. Acoustic insulation measures are incorporated in select laboratory spaces to enhance safety and working comfort.

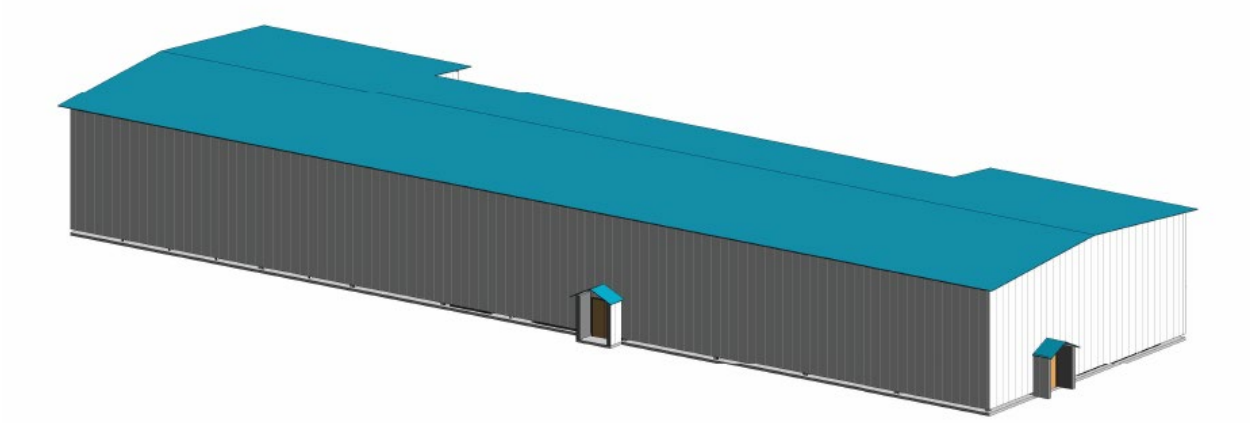
The layout ensures clear separation between office and laboratory functions, with dedicated access points and internal circulation paths that promote safety, cleanliness, and workflow efficiency. This integration of administrative and laboratory functions within a single facility allows for streamlined operations and direct communication between management, technical and field teams.

Figure 18.20: Mill Office Plan View



Source: GMS, July 2025.

Figure 18.21: Mill Office 3D View



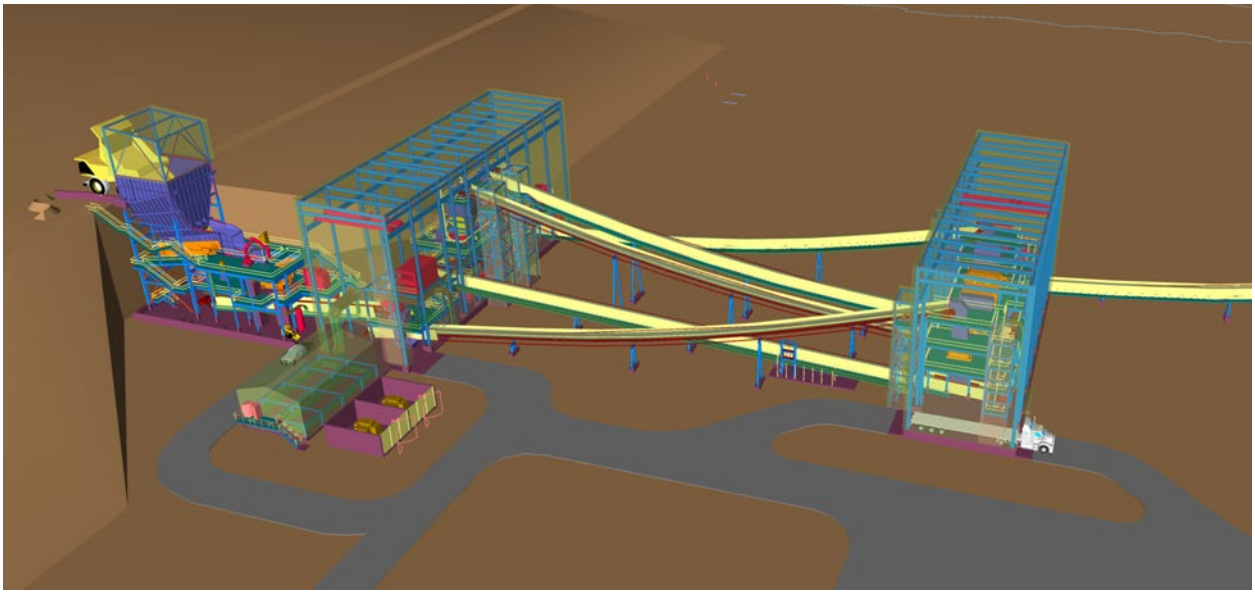
Source: GMS, July 2025
**Note: Not to scale*

18.5.2 Process Buildings

The secondary and tertiary crushing, screening, DMS sizing, and DMS buildings are all stick-built with a concrete foundation construction type. These buildings are enclosed, insulated and heated to prevent freezing. The majority of the process plant equipment is located indoors with the exception of large water storage tanks and thickeners, which are designed with necessary winterization auxiliary equipment.

The Primary Crusher area is left open to the environment with no enclosure or heating.

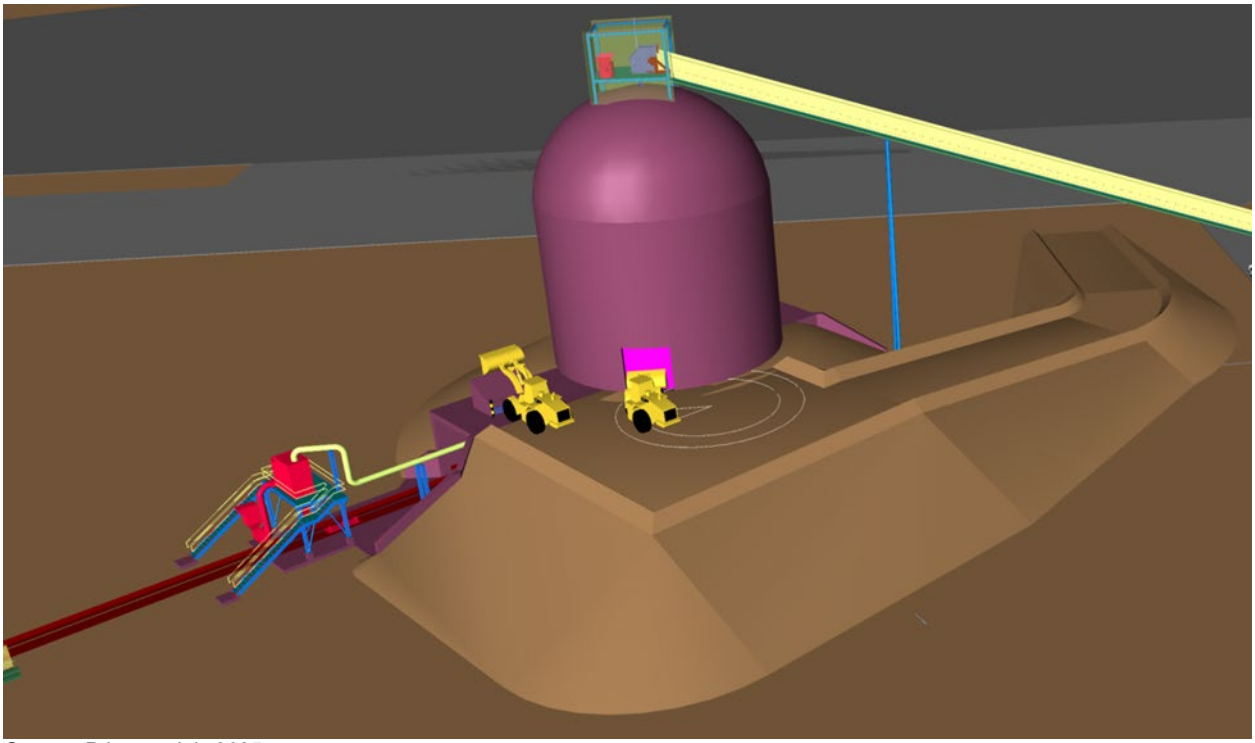
Figure 18.22: Primary Crushing, Secondary and Tertiary Crushing, Screening Buildings



Source: Primero, July 2025.
**Note: Not to scale.*

Crushed ore is stored in an insulated silo capped with an insulated concrete dome. The dome is to be erected on site with a steel frame and shotcrete construction. The crushed ore is discharged via a reclaim tunnel underneath. There is one (1) doorway suitable for a front-end loader for emergency reclaim on the side of the storage dome.

Figure 18.23: Crushed Ore Storage Dome

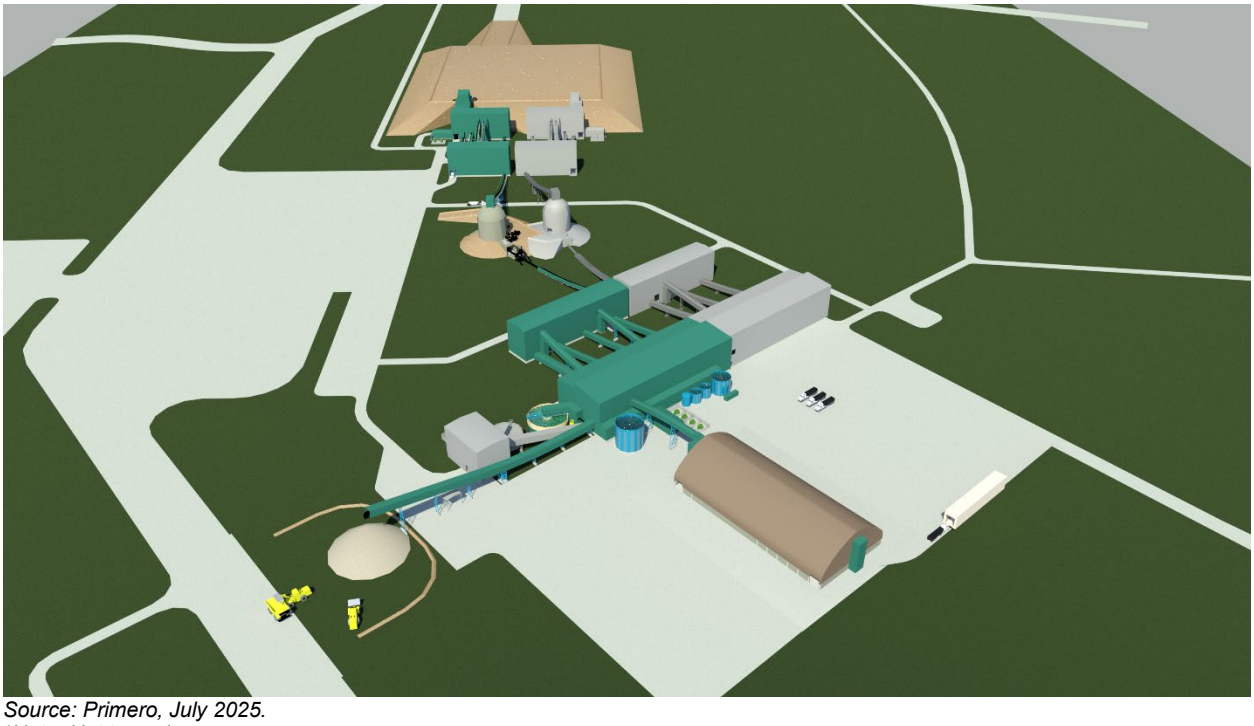


Source: Primero, July 2025.
**Note: Not to scale.*

All conveyors downstream of the DMS sizing building are to be enclosed, insulated, made of structural steel, and heated to prevent freezing.

For Phase 2, the secondary and tertiary crushing, screening and crushed ore storage buildings will be duplicated. The DMS sizing and DMS buildings will be extended in length to incorporate the additional equipment.

Figure 18.24: Phase 1 (Green) and Phase 2 (Grey) Structures



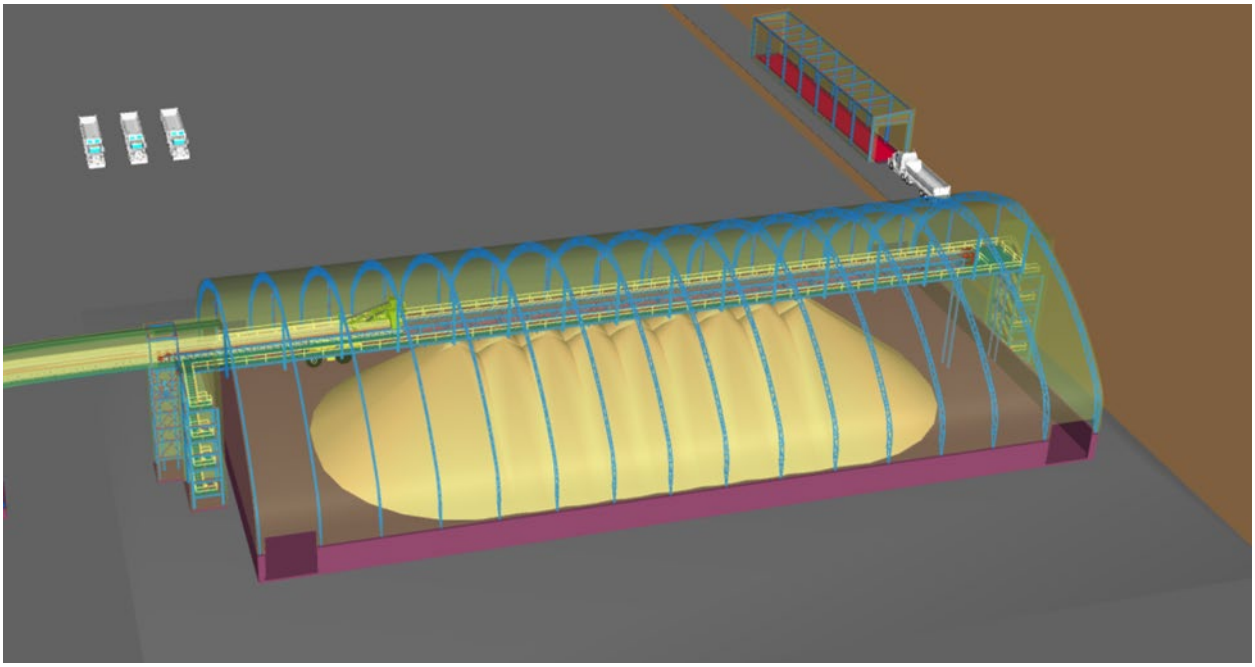
Source: Primero, July 2025.
**Note: Not to scale.*

18.5.3 Concentrate Load-Out

Spodumene concentrate exits the concentrator building into a dome and is stockpiled on a heated concrete slab. The storage capacity of the dome is 6 days. One set of two garage doors is positioned at each end of the structure to allow for transport trucks to access the building and be loaded with a front-end loader. The tripper cart will ensure equal concentrate distribution on the pile and allow the loader operator to circulate safely to both loading zones.

A spodumene concentrate truck scale will be installed near the site gatehouse and tared to control spodumene shipped. Control of the truck scale is to be provided from the main gatehouse before trucks leave the site.

Figure 18.25: Concentrate Load-Out



Source: Primero, July 2025.
**Note: Not to scale.*

18.5.4 Tailings Load-Out

Tailings product from the concentrator is stockpiled on a pad in the open air. The stockpile is designed to allow front-end loaders to reclaim the material and load trucks to transport the material to the tailings' disposal area.

18.5.5 Paste Preparation Plant

A portion of the concentrator tailings is required for paste backfill once underground mining begins in Phase 2. The discharge of the tailings' vacuum belt filter and the float screens are fitted with diverter chutes to send the tailings to either the paste preparation plant or the tailings load-out. The fines bypass and fine floats will primarily be sent to the paste preparation plant to reduce the milling power required. The paste preparation building contains a ball mill in a closed circuit with a cyclone cluster. The tailings are ground to the required particle size distribution, then sent to the paste preparation thickener. The thickened underflow is pumped to the paste plant. The paste preparation building housing the ball mill and cyclone is stick-built with concrete foundations. The building is enclosed, insulated and heated to prevent freezing. The paste preparation thickener is located outside, next to the paste preparation plant. The thickener is designed with winterization auxiliary equipment with access to the enclosed thickener bridge from inside the paste preparation plant.

18.5.6 Paste Backfill Plant

To achieve the underground mine backfilling requirements, only a portion of the DMS tailings stream is required. Nevertheless, to account for planned and unexpected downtimes associated with an underground backfill operation, the backfill system has been designed for a combined availability and utilization of 50%.

When the paste plant is operating, a portion of the DMS tailings stream is fed to the ball mill and cyclone circuit at the process plant. The fines bypass and recrush streams are used primarily with fine floats used to supplement tonnages, when required. The ball mill is sized to be able to produce a PSD with a fines and coarse portion suitable for paste fill. The milled DMS tailings are directed to a thickener at the process plant. The thickener underflow is then delivered to the paste plant via a ~310 m-long insulated and heat-traced overland pipeline.

At the paste plant, two (2) tanks store thickened milled DMS tailings and serve to buffer short-term paste plant outages. The thickened milled DMS tailings stream is filtered at the paste plant via vacuum disc filtration, reporting to a continuous mixer where trim water and binder (Type I/II cement) are added to create a homogenous cemented paste backfill (CPB).

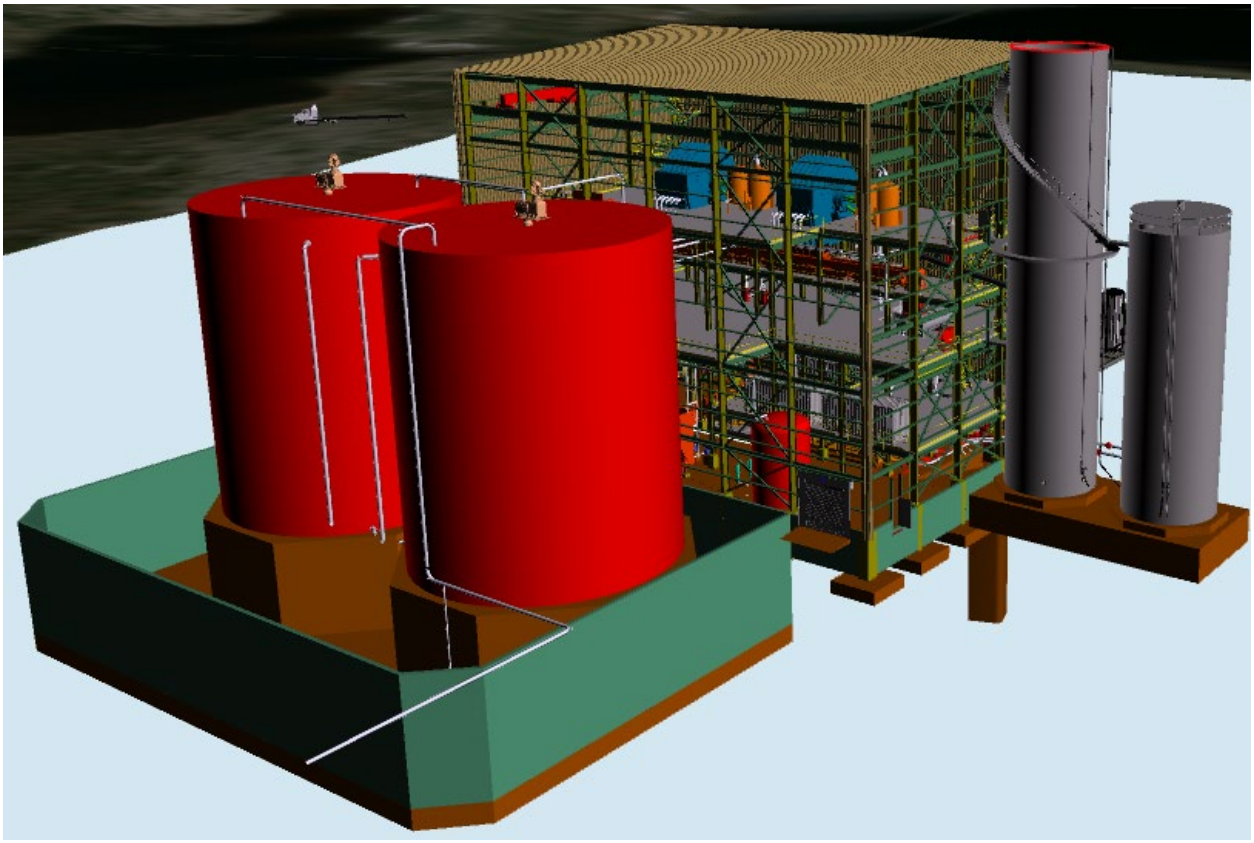
A binder system, consisting of two (2) 700-tonne silos, a weigh belt, and a screw conveyor, will be used to feed binder to the continuous mixer. The binder system is sized to support five (5) days of continuous operation. Binder will be delivered in bulk by truck to the two (2) silos located adjacent to the paste plant. Each silo has its own dedicated dust collector. Binder is weighed and screw fed continuously from the silos into the mixer feed chute, allowing for adjustments in throughput to achieve the desired CPB recipe.

The paste in the continuous mixer discharges by overflowing into a paste hopper. The hopper facilitates a continuous flow of paste to the underground. Paste from the hopper reports to a 150-bar-rated hydraulic piston pump, which distributes the paste through a DN 200 paste fill reticulation system to the mine stopes.

The CPB is pumped from the paste plant, located at ~381 m a.s.l., down one (1) of two (2) 60 m-long surface boreholes (~60-70° from horizontal). The two (2) surface-to-underground boreholes will be installed in Year 0 to limit risk and minimize the downtime caused by a line or borehole blockage. Adequate space has been made for a third borehole to be drilled and cased at a later stage of the mining life. The surface boreholes will be drilled down to 320-level, where dedicated underground cut-outs will keep the breakthroughs out of the main travel areas. A single DN200 paste reticulation system will run from the breakthroughs to service all the underground mine stopes.

Figure 18.26 shows a view of the paste plant area.

Figure 18.26: Paste Plant Area



Source: Paterson & Cooke, June 2025.
**Note: Not to scale.*

18.6 Tailing Storage Facility

18.6.1 Tailings Management

Tailings produced during process operations will exit the plant as a single stream. The tailings will either be disposed of on surface or used as paste backfill for the underground mine. The tailings for surface disposal will be dewatered by dense media separation (DMS) or with a belt filter to approximately 88% solids content (Ws/Wt) in the processing plant. The dry tailings will be transported to Stockpile 002, laid down by the hauling equipment, spread with a dozer, and compacted by a vibratory soil compactor. The anticipated total tailings production during the 22-year LOM is evaluated at 71.1 Mt, with approximately 54.0 Mt disposed on surface, and the remaining 17.1 Mt will be used for paste backfill.

Directive 019 on the Mining Industry (MDDEP, 2025) considers a dry stack to be an impoundment area without water retention. This defines the design criteria for the drainage system and earthquake resistance. Directive 019 refers to the Quebec guide (MRNF, 2024) for the minimum factors of safety for overburden stockpiles, which serve as the design criteria for this study. According to the Global Industry Standard on

Tailings Management (GISTM, 2020), new tailings storage facilities (including dry stacks) should meet the design criteria corresponding to the “Extreme” consequence classification for post-closure conditions. This classification was used to determine the return interval of the seismic event that was used in the stability analysis. The CDA Technical Bulletin on the Application of Dam Safety Guidelines to Mining Dams (CDA, 2019) clarifies that a tailings facility without ponded water, such as a dry stack, can be designed according to guidelines for dumps and overburden stockpiles, provided that the dry stack materials are neither liquefiable nor able to flow. A geotechnical site investigation of the Stockpile 002 footprint has also been completed. The principal references for the design of the tailings management facility are: BBA Factual Report (BBA Engineering Ltd., "Shaakichiuwaanaan Project, Technical Report, Geotechnical Campaign for Phase 2 (factual), Final," April 2025.); Vision Geochemistry Report “Geochemical Characterization of Mine Waste Materials & Modelling of Waste Rock Stockpiles for the Shaakichiuwaanaan Project (CV5 Pegmatite): Feasibility Study Update. Ref. 2405007-TR1), 2025”; Process Design Criteria (Primero, 2024); The AtkinsRéalis Technical Note on tailings laboratory testing, “Geotechnical Laboratory Analysis of Tailings”, June 2025.

- G Mining Services’ Mine Plan, “CASN-Final_Mining_Mass_Balance”.
- Available baseline data such as environmental constraints, sensitive and unique ecosystems, surface water bodies, property limits, Mineral Resources, and mining claims. All this information was integrated into a geographic information system.
- Area LiDAR provided by PMET Resources and extracted from government websites.

The design of the tailings’ storage facility is based on the following considerations:

- Tailings produced at the plant are conditioned to a sufficiently dry state that allows for placement, spreading and compaction with conventional earth-moving equipment.

The tailings will be transported from the plant to Stockpile 002 with mining trucks. Tailings for backfill will be pumped to the paste backfill plant.

- The tailings stockpile will be constructed utilizing a bottom-up methodology.
- A starter berm will be provided at the toe of the tailings’ facility.
- The tailings are considered non-acid generating but have been considered as metal leaching for purposes of this study.
- The base of the tailings portion of Stockpile 002 will be lined with a 1.5 mm double-textured linear low-density polyethylene (LLDPE) geomembrane.
- Runoff from the stockpile's footprint is managed by lined perimeter ditches and collection ponds.

- Excess water from the collection ponds will be pumped to the Industrial Area for treatment of both Total Suspended Solids (TSS) and Trace Metals prior to discharge to the environment or re-use.

The external slopes of the tailings' facility will be graded for closure as part of the normal operations. At closure, the tailings storage surfaces will be covered with layers of compacted clay, overburden and topsoil before hydroseeding.

18.6.2 Material Production and Characteristics

The tailings production increases from 1.9 Mt in Year 1 to 5.2 Mt in Year 3. From Year 4 to Year 16, the production is in the range of 4.1 to 4.3 Mt per annum. Production decreases in Year 17 to 2.9 Mt, 1.6 Mt in Year 18, and ceases in Year 19 with 1.25 Mt.

The following information was used for the volumetric analysis of the facility:

- The total tailings production for the life of mine is 71.1 Mt.
  - 17.1 Mt of tailings will be used as backfill.
  - The remaining 54.0 Mt will be disposed of in Stockpile 002.
- The laboratory testing of the tailings sample indicated a standard Proctor maximum dry density = 1,958 kg/m^3 (ASTM, D698) and optimum moisture content of 11.7% (Ww/Ws as per soil mechanics).
- The average in-situ dry density of the compacted DMS tailings within the facility for volumetric calculation is assumed to be 90% of the maximum standard Proctor dry density. This value is approximately 1.7 t/m^3, and the required storage volume in Stockpile 002 is therefore 31.0 Mm^3.
- Tailings for surface disposal will be a mixture of Primary Coarse DMS floats and degrit, Secondary Coarse DMS degrit, Fine DMS floats and degrit, Ultrafine DMS floats and degrit, Recrush DMS floats, DMS Mags, and Belt Filter Cake. The particle size distribution of this mixture from laboratory test work is as follows:
  - D80 = 7.68 mm.
  - Finer than 75 microns = 2.9%.
- The specific gravity (ASTM, D854) of the combined tailings sample is 2.68.

Table 18.1 shows the expected annual production for the tailing streams, the expected underground backfill and the tailings for surface disposal at Stockpile 002.

Table 18.1: Tailings Production – Stockpiled and Backfill

Period	Produced Tailings	Annual Tailings to Paste Backfill Plant for Underground	Annual Tailings to Stockpile 002	Cumulative Tailings Tonnage to Stockpile 002
(year)	(Mt)	(Mt)	(Mt)	(Mt)
Y-1	0.28		0.28	0.28
Y1	1.86	0.03	1.83	2.11
Y2	2.32	0.38	1.94	4.05
Y3	5.20	1.12	4.08	8.13
Y4	4.30	0.97	3.33	11.46
Y5	4.30	0.98	3.32	14.78
Y6	4.30	0.93	3.37	18.15
Y7	4.30	0.97	3.33	21.48
Y8	4.30	0.98	3.32	24.8
Y9	4.30	0.99	3.31	28.11
Y10	4.31	0.99	3.32	31.43
Y11	4.28	0.99	3.29	34.72
Y12	4.31	0.99	3.32	38.04
Y13	4.29	1.00	3.29	41.33
Y14	4.30	1.01	3.29	44.62
Y15	4.30	1.01	3.29	47.91
Y16	4.10	0.99	3.11	51.02
Y17	2.85	0.98	1.87	52.89
Y18	1.64	1.02	0.62	53.51
Y19	1.25	0.77	0.48	53.99
Total	**71.10**	**17.11**	**53.99**	**53.99**

18.6.3 Site Description of Tailings Storage

A site selection process was completed for the PEA study. Two (2) sites were identified for tailings and waste rock disposal. Stockpile 001 is located northwest of the plant site and will be used exclusively for waste rock disposal due to its proximity to the open pit. Stockpile 002 is located northeast of the plant and will be used for both waste rock and tailings storage. The total footprint area for Stockpile 002 is 218 ha. The stockpile footprint has been offset a minimum of 60 m from natural streams, lakes, or inferred fish

habitats. The site topography varies between elevations 386 m a.s.l. and 444 m a.s.l. The topography allows for runoff drainage by gravity to perimeter ditches. The footprint Stockpile 002 is fully lined.

Stockpile 002 will also hold potentially acid-generating waste rock. The tailings will be placed separately from the waste rock stockpile. Condemnation drilling has been completed within the Stockpile 002 footprint. A geotechnical site investigation and factual report were completed for Stockpile 002 (BBA, 2024b), which included 12 boreholes, sampling and laboratory testing. The soil conditions encountered in the Stockpile 002 footprint (as classified by the Unified Soil Classification System, USCS) are variable and are as follows:

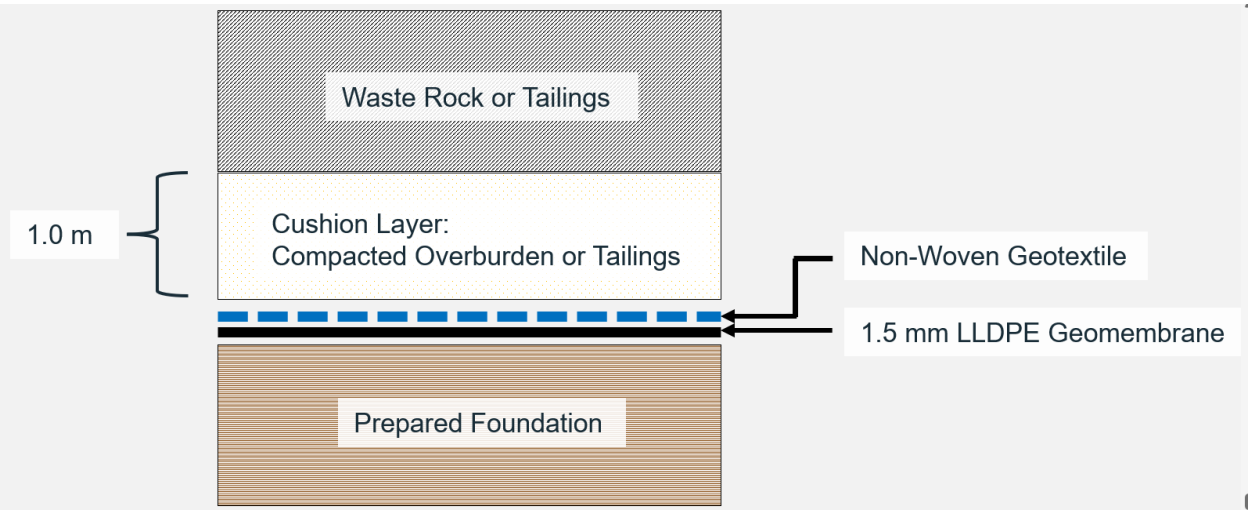
- Organics are encountered across the whole footprint, ranging from 0.1 to 1.5 m thick (average 0.4 m). Depth varies across the footprint with a localized low at the southwestern corner of the footprint of Collection Pond 04.
- ML (a soil unit of Silt / Sandy Silt / Silt with Gravel) was encountered only at the northeastern corner of the footprint to a depth of 1.8 m below ground surface (bgs), overlaying SM (a soil unit of Silty Sand / Silty Sand with Gravel), GM (a soil unit of Silty Gravel with Sand / Gravel with Silt and Sand), and cobbles / gravel.
- SM was encountered across the whole footprint overlaying GM, and/or cobbles / gravel, and bedrock. This soil unit is found to depths ranging from 0.5 to 13.4 m bgs, varying with bedrock surface, with the greatest depth at the centre of the stockpile footprint.
- GM was encountered in select areas near the centre of the footprint and at its eastern edge. It is overlain by SM and overlies cobbles / gravel and bedrock. The depth and thickness of this soil unit vary across the footprint.
- Cobbles / gravel are encountered in select areas near the centre of the stockpile footprint and at its eastern edge. Depth and thickness of the soil unit vary across the footprint from a minimum upper depth of 1.2 m bgs and a minimum thickness of 0.5 m to a maximum upper depth of 13.4 m and a thickness of 5.4 m.
- Depth to bedrock ranges across the stockpile footprint from 0.3 to 18.8 m bgs; elevation of the bedrock surface ranges from 374 to 440 m a.s.l. Bedrock is at its highest elevation at the centre of the eastern half of the footprint and is at its lowest elevation at the centre of the western half of the footprint.
- Groundwater depths range from 2.08 to 3.07 m bgs. With the variations in ground surface, groundwater elevations range from 393 to 418 m a.s.l.

18.6.4 Tailings Storage Description and Related Infrastructure

The footprint of the tailings portion of Stockpile 002 is approximately 92 ha (42%). The footprint for the stockpile will require tree clearing and removal of topsoil / organics. The stripped topsoil / organics will be stockpiled for use in closure. Overburden material within the footprint found to be loose / saturated will also be removed, following inspection by a geotechnical engineer, and stockpiled in Overburden Piles 004/005. The natural terrain within the footprint of the stockpile generally allows for drainage to the perimeter of the stockpile. Minimal re-grading of the footprint is expected. The basin will then be lined with a 1.5 mm double-textured LLDPE geomembrane. A cushion geotextile will be provided on top of the geomembrane, and the tailings can be placed directly onto the geotextile provided that care is taken to avoid equipment traffic onto the lined basin (see Figure 18.27). A starter embankment will be provided at the downstream toe of the tailings' facility. The starter embankment intends to provide vehicular access to the toe of the tailings facility to facilitate inspections. The starter embankment will be constructed with Low-Risk waste rock to a height of 3 m and a crest width of 5 m, with 3H:1V side slopes. A filter / bedding sand cushion layer will be constructed 300 mm thick below the starter dam. On the upstream slope of the dam, a transition sand and gravel layer will be placed 300 mm thick and will be overlain by a 300 mm thick layer of filter / bedding sand; this will provide a cushion between the waste rock dam and the cushion geotextile. The tailings stack will be constructed to facilitate closure with a global external slope of 3.25 H:1 V and intermediate benches 7.5 m wide every 30 m of elevation (see Figure 18.28).

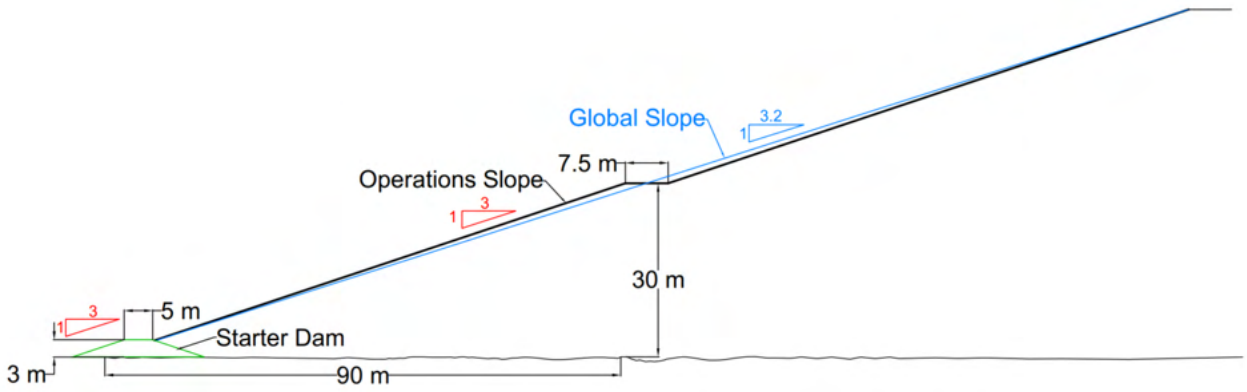
CDA does not address FoS applicable to dry stacks deemed to be non-liquefiable (CDA, 2019). FoS values were therefore determined in accordance with the Quebec Regulations (MRNF, 2024). The site is classified as Class D, and for a 1 in 10,000-year event (as required for a classification of Extreme), the ground acceleration is estimated as 0.1 g. This value was used in pseudo-static analysis and was extrapolated from the National Building Code of Canada (NBCC, 2020). A site-specific seismic hazard assessment is recommended in the next phase of the study.

Figure 18.27: Typical Lined Base Stockpile 002



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

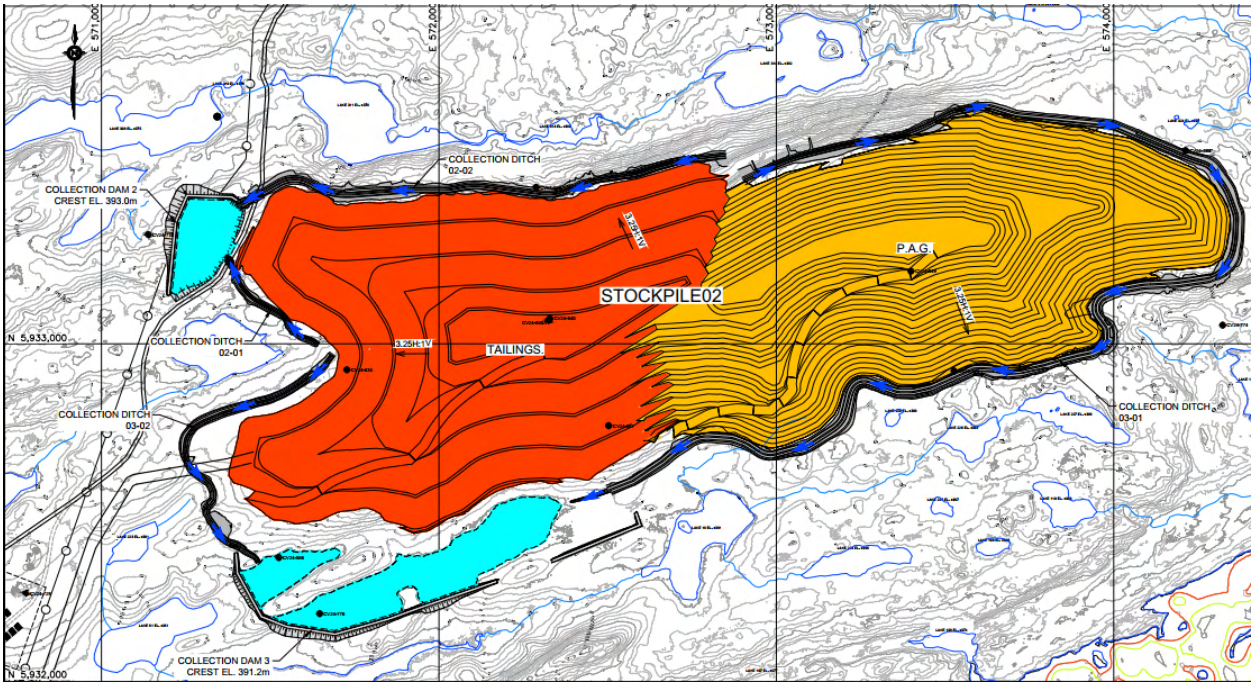
Figure 18.28: Typical Section for Tailings in Stockpile 002 Showing Intermediate Benches for Both Operation and Closure Stages



Source: AtkinsRéalis, July 2025.

Water management for Stockpile 002 includes a series of perimeter ditches that drain under gravity to two (2) collection ponds. Runoff from the tailings area will discharge via Ditch 02-01 and Ditch 02-02 to Pond 02 and via Ditch 03-01 and Ditch 03-02 to Pond 03. The ditches and ponds will be lined with a 1.5 mm LLDPE geomembrane, covered with a cushion geotextile and a layer of riprap. An access road is provided parallel to, and on the outside of, the ditches. The access road facilitates vehicular access for inspections and repairs and also diverts runoff from outside the stockpile footprint from discharging into the ditches. Water collected in Pond 02 will be pumped to Pond 03, and water from Pond 03 will be pumped to the Water Treatment Plant (WTP) for treatment prior to discharge to the polishing pond. Figure 18.29 presents the resulting arrangement of the proposed infrastructure. More detailed descriptions of water management infrastructures are provided in Section 18.7.3.

Figure 18.29: Stockpile 002 (Tailings and Waste Rock) and Related Infrastructure



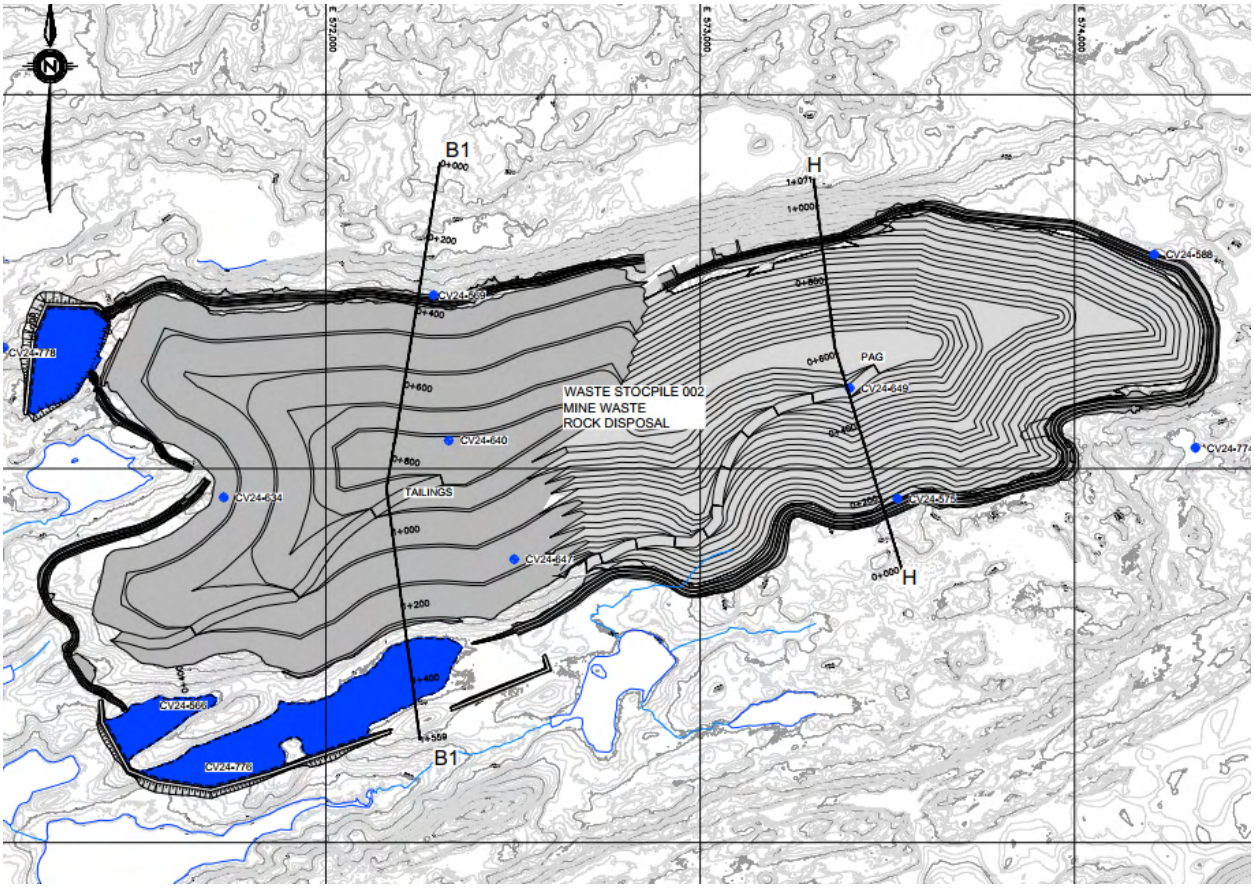
Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

Material take-offs (MTOs) are estimated assuming that soil and waste rock used for construction are non-metal leaching and non-acid generating, and materials are sourced from an on-site quarry / borrow pit or the open pit. Crushing of stone will be required on-site to produce the required materials.

18.6.5 Stability Analysis for Tailings Storage

Slope stability analyses were conducted for the tailings stockpile (on the tailings storage area of Stockpile 002) using SLOPE/W Version 2023.1.2, a two-dimensional limit equilibrium software program developed by Geo-Slope International Limited. The Morgenstern-Price method was employed to identify potential failure surfaces within the slopes of the stockpiles and their underlying foundations. The minimum required Factors of Safety (FoS) for the tailings stockpile are presented in Table 18.3. Stability analysis has been performed for the critical section (Section B1-B1) at the tailings' storage area (Figure 18.30). Cross-sections used for the stability analyses are based on the maximum height (120 m) and overall slope (3.25H:1V). The detailed model geometry of the stockpile has been discussed in Section 18.6.4. Intermediate benches are incorporated in the geometry for analysis.

Figure 18.30: Stockpile 002 Critical Sections for Stability Analysis



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

As outlined in Section 18.6.4, the basin of the tailings stockpile will be lined with a geomembrane, which will have a cushion geotextile placed on top. The interface between the geotextile and the geomembrane is critical for slope stability. The analysis revealed that the combination of a 1.5 mm (60 mil) double-textured LLDPE geomembrane and a non-woven geotextile provided the required FoS, and this combination has been selected for the design.

The material density and strength parameters used in the stability analyses were derived from data collected during geotechnical investigations, the laboratory test work and from AtkinsRéalis' experience with similar materials. The tailings materials are coarse-grained based on the particle size distribution from lab analysis (AtkinsRéalis, 2025) and can be considered free-draining materials. Therefore, slope stability analyses are performed using effective stress (drained) shear strengths. Table 18.2 provides a summary of the material parameters and materials model used for this study. To determine the necessary stable configurations of the tailings stockpile, both static and pseudo-static conditions were tested. Soil conditions modelled in the stability analyses are as outlined in Section 18.6.3. A conservative phreatic surface has

been assigned based on a steady-state model where the water table lowers to the level of the peripheral drains below the natural terrain.

Post-earthquake analyses, which typically involve evaluating the residual shear strengths of foundational or fill materials that may have liquefied, were not conducted. This was based on borehole investigations indicating that the foundation materials are not expected to experience liquefaction. In addition, the tailings will be compacted to more than 90% of the standard Proctor maximum dry density, making them considered non-liquefiable. Consequently, the residual strength is not considered in this study. A pseudo-static analysis has been performed to evaluate dynamic stability. For closure conditions, both a geosynthetic cover and a compacted clay cover were considered. The geosynthetic cover was more critical in terms of stability, and therefore, the results provided are for the stability analysis with a geosynthetic cover.

Table 18.2: Material Geotechnical Parameters – Tailings Stockpile[1]

Description	γ (kN/m³)	C' (kPa)	Φ' (˚)	Slope Stability Material Model
Organics	14	0	18	Mohr-Coulomb
ML – Silt / Sandy Silt / Silt with Gravel	18.5	0	31	Mohr-Coulomb
SM – Silty Sand / Silty Sand with Gravel	19.0	0	34	Mohr-Coulomb
GM – Silty Gravel with Sand / Gravel with Silt and Sand	20.0	0	38	Mohr-Coulomb
GW – Gravel	20.5	0	39	Mohr-Coulomb
Tailings (Saturated)	18	0	37	Mohr-Coulomb
Waste Rock	20	0	38	Mohr-Coulomb
1.5 mm Double-Textured LLDPE Geomembrane (Base Layer)	9	3.15	30.1	Mohr-Coulomb
Non-Woven Geotextile (Base Layer)	1	3.15	30.1	Mohr-Coulomb
1.5 mm HDPE Geomembrane (Cover)	9	0.65	17.8	Mohr-Coulomb
Geonet (Cover)	9	0.65	17.8	Mohr-Coulomb
Protective Layer of Geomembrane / Geotextile	19	0	34	Mohr-Coulomb
Crushed Stone (at Closure)	19	0	38	Mohr-Coulomb
Bedrock	Impenetrable			

**Note: (1) These materials properties are also used in the waste rock stockpiles.*

The results of the slope stability analysis are presented in Table 18.3. With the proposed configurations and optimizations, the FoS obtained shows that the stability of the tailings stack meets the minimum required values from the Quebec Regulations (MRNF, 2024).

Table 18.3: Safety Factor of Slope Stability Analysis – Tailings Stockpiles

Stockpile	Phase	Section	Loading Conditions	FoS[1] Minimum Requirement	FoS	Failure Type[2]
002	Operational	B1-B1	Static (Drained)	1.5	2.13	Global
002	Operational	B1-B1	Pseudo-static	1.3	1.60	Global
002	Closure	B1-B1	Static (Drained)	1.5	2.13	Global
002	Closure	B1-B1	Pseudo-static	1.3	1.60	Global
002	Operational	B1-B1	Static (Drained)	1.2	2.36	Local (Bench 1)
002	Operational	B1-B1	Static (Drained)	1.2	2.28	Local (Bench 2)
002	Operational	B1-B1	Static (Drained)	1.2	2.28	Local (Bench 3)
002	Operational	B1-B1	Static (Drained)	1.2	1.84	Local (Bench 4)
002	Closure	B1-B1	Static (Drained)	1.2	2.33	Local (Bench 1)
002	Closure	B1-B1	Static (Drained)	1.2	2.36	Local (Bench 2)
002	Closure	B1-B1	Static (Drained)	1.2	2.29	Local (Bench 3)
002	Closure	B1-B1	Static (Drained)	1.2	1.90	Local (Bench 4)

**Note(s): (1) (MRNF, 2024); (BCMWRPRC, 1991b) and (Hawley & Cunning, 2017).*
(2) The first bench is the upper one, with subsequent benches arranged chronologically down towards the ground surface.

18.6.6 Mine Waste Rock and Overburden Management

Among all sub-products of mining operations, waste rock and overburden materials will be generated over the life of mine. Two (2) waste rock stockpiles, two (2) overburden piles, and two (2) organics piles have been conceptualized. Stockpile and pile construction will be continuous over the LOM. Material stored in the overburden piles and organics piles will be utilized during the closure phase.

The anticipated material production over the 22-year LOM is evaluated as follows:

- Waste rock: 155.2 Mt.
  - Non-leaching (Low Risk) waste rock: 74.2 Mt.
  - PAG/ML waste rock: 81.0 Mt.
  - Overburden: 14.0 Mt.
- Organics: 3.2 Mt.

A portion of the non-leaching waste rock will be stored in Stockpile 001 (43.5 Mt), and the remainder will be dumped in the open pit (29.6 Mt). All of the PAG waste rock produced will be stored in Stockpile 002, and all the overburden and organics will be stored in their respective piles.

18.6.7 General Design Considerations

The principal references for the design of waste rock and overburden management facilities are:

- Technical note – Environmental, social, and hydrological surveys (NIIGAAN, 2022).
- Geochemical characterization of tailings and waste rock (Vision Geochemistry (2025) Geochemical Characterization of Mine Waste Materials & Modelling of Waste Rock Stockpiles for the Shaakichiuwaanaan Project (CV5 Pegmatite): Feasibility Study Update. Ref. 2405007-TR1).
- G Mining Services' Mine Plan, "CASN-Final_Mining_Mass_Balance".
- Available baseline data such as environmental constraints, sensitive and unique ecosystems, surface water bodies, property limits, Mineral Resources, and mining claims. This information was integrated into a geographic information system.
- Area LiDAR provided by PMET Resources.

The design for waste rock and overburden is based on the following:

- Overburden and waste rock extracted from the different mining phases are considered dried material; therefore, materials will have the time to drain between excavation and placement at the stockpile. Mining equipment will transport material from the pit to the final disposal area.
- Stockpiles will be built in ascending lifts, and adequate bench heights and widths will be maintained to achieve the overall design slope.
- Runoff from the Low Risk waste rock in Stockpile 001 is to be managed with perimeter ditches (Ditch 01-01, Ditch 01-02, Ditch 01A-01 and Ditch 01A-2) and Collection Pond 01 and Pond 01A. Water collected in Pond 01 and Pond 01A will be pumped to the Sedimentation Ponds at the Industrial Area for TSS management prior to discharge to the environment or re-use in the process plant.
- Runoff from the PAG waste rock within Stockpile 002 will be managed with lined ditches (Ditch 02-02 and Ditch 03-01) and Collection Ponds 02 and 03, which are both lined. This water will be pumped to the Industrial Area for treatment prior to discharge to the environment or re-use in the process plant.

- Runoff from the overburden stockpiles will be managed with a series of catchment paddocks located at the toe of the stockpile.
- Closure of the Low-Risk waste rock surfaces in Stockpile 001 will require the external slopes to be dozed down to a slope of 3H:1V, placement of a layer of overburden and topsoil and hydroseeding.

Closure of the PAG waste rock surfaces in Stockpile 002 will require the external slopes to be dozed down to a slope of 3H:1V, placement of a cushion layer, installation of a compacted clay liner, covering the clay with a layer of overburden and topsoil and hydroseeding.

- Material stored in Overburden Piles 004 and 005 and the Organics Piles will be utilized during the closure phase as clean fill / cover material.
- Material stored in the Organics Piles will be utilized during the closure phase.

18.6.8 Material Production and Characteristics

The mining plan indicates a ramp-up period for waste rock production from Year 1 to Year 2, a decrease in Year 3 and Year 4 and then another ramp-up period in Year 5, following which it remains relatively stable until Year 10, when production starts to decrease until Year 19. Overburden is expected only at early stages of the Project, with a peak of 3.9 Mt in Year -1 and Year 1; the total expected tonnage is about 14.0 Mt.

The volumetric analysis of the waste rock and overburden facilities is based on the following:

- The waste rock stockpiles will have a global external slope of 3.25H:1V.
- The grain size distribution for the waste rock is expected to be 0-1,000 mm. The material is considered self-draining with no relevant moisture content.
- The estimated in-place dry density is 2.0 t/m³. The resulting required storage capacity is therefore 77.2 Mm³.
- Waste rock will be stored in both Stockpile 001 and Stockpile 002, with Low-Risk waste rock stockpiled in Stockpile 001 and potentially acid-generating waste rock stockpiled in Stockpile 002.
- Low-Risk waste rock will be stored in the Open Pit once the capacity of Stockpile 001 has been reached in Year 9.
- For the Overburden Piles, the estimated in-situ dry density is 1.75 t/m³. The resulting required capacity is 8.01 Mm³.
- For the Organics Pile, the estimated in-situ dry density is 1.43 t/m^3. The resulting required capacity is 2.2 Mm^3.

Table 18.4 presents the yearly production to be stored in the stockpiles and overburden piles, as well as the cumulative produced tonnes and deposition location for waste rock and overburden.

Table 18.4: Waste Rock and Overburden Production and Storage Areas

Period	Yearly Low-Risk Waste Rock Production	Cumulative Waste Low-Risk Rock to Stockpile 001	Cumulative Waste Low-Risk Rock to Open Pit	Yearly PAG Waste Rock Production	Cumulative PAG Waste Rock, to Stockpile 002	Yearly Overburden Production	Cumulative Overburden to Pile 004	Cumulative Overburden to Pile 005	Yearly Organics Production	Cumulative Organics to Pile
(Year)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	Mt	Mt
Y-2				0.10	0.10	0.89	0.89			
Y-1	0.77	0.77		1.59	1.69	3.92	4.81			
Y1	1.76	2.53	-	3.83	5.52	3.91	7.00	1.72	0.89	0.89
Y2	6.27	8.79	-	7.08	12.60	0.89		2.61	0.89	1.77
Y3	4.63	13.42	-	5.00	17.60	1.04		3.65	1.39	3.16
Y4	4.37	17.80	-	3.83	21.43	1.07		4.72	-	-
Y5	9.03	26.82	-	7.15	28.60	2.08	-	6.80	-	-
Y6	10.88	37.71	-	8.33	36.91	0.23	-	7.03	-	-
Y7	7.48	45.18	-	6.37	43.27		-		-	-
Y8	8.21		8.21-	7.24	50.52		-		-	-
Y9	5.76		13.97	9.59	60.11		-		-	-
Y10	4.35		18.32	8.20	68.32	-	-	-	-	-
Y11	2.40	-	20.72	4.08	72.40	-	-	-	-	-
Y12	3.42	-	24.14	4.66	77.06	-	-	-	-	-
Y13	1.83	-	25.97	1.37	78.43	-	-	-	-	-
Y14	1.65	-	27.62	1.21	79.65	-	-	-	-	-
Y15	1.03	-	28.65	0.55	80.20	-	-	-	-	-
Y16	0.3	-	28.95	0.49	80.69	-	-	-	-	-

Period	Yearly Low-Risk Waste Rock Production	Cumulative Waste Low-Risk Rock to Stockpile 001	Cumulative Waste Low-Risk Rock to Open Pit	Yearly PAG Waste Rock Production	Cumulative PAG Waste Rock, to Stockpile 002	Yearly Overburden Production	Cumulative Overburden to Pile 004	Cumulative Overburden to Pile 005	Yearly Organics Production	Cumulative Organics to Pile
Y17	0.03	-	28.98	0.23	80.92	-	-	-	-	-
Y18		-		0.06	80.98	-	-	-	-	-
Y19		-		0.03	81.01	-	-	-	-	-
Total	**74.17**	**45.18**	**28.98**	**81.01**	**81.01**	**14.03**	**7.00**	**7.03**	**3.16**	**3.16**

18.6.9 Site Description for Waste Rock and Overburden Storage

A site selection process was completed for the PEA study. Stockpiles 001 and 002 were identified for waste rock disposal, and Overburden Piles 004 and 005 for overburden storage. An Organics Pile was identified to store organics (topsoil, peat, etc.). Stockpile 001 is located northwest of the plant site and will be used exclusively for Low-Risk waste rock disposal. Stockpile 002 is located northeast of the plant and will be used for PAG waste rock and tailings storage. The total footprint area for Stockpile 001 is 80 ha. The total footprint of Stockpile 002 is 218 ha. The portion of Stockpile 002 for PAG waste rock storage is 126 ha (58%). The stockpile footprints have been offset a minimum of 60 m from natural streams, lakes, or inferred fish habitats. The site topography for Stockpile 001 varies from elevation 374 m a.s.l. to elevation 400 m a.s.l. The site topography for Stockpile 002 varies from elevation 386 m a.s.l. to elevation 444 m a.s.l. The topography of both stockpiles allows for runoff drainage by gravity to perimeter ditches.

Condemnation drilling has been completed within the Stockpile 001 and 002 footprints. A geotechnical site investigation and factual report were completed for both stockpiles (BBA, 2024b), which included 19 boreholes, sampling and laboratory testing. The soil conditions encountered in the Stockpile 001 footprint are variable. The soil profile for Stockpile 001 (as classified by the USCS) is summarized as follows:

Organics are encountered across the whole footprint, ranging from 0.1 to 0.9 m thick (average 0.4 m), with the greatest depth at the centre of the footprint. Organics directly overlay bedrock at the eastern edge of the footprint. ML (a soil unit of Silt / Sandy Silt / Silt with Gravel) is encountered only on the western edge of the footprint to depths ranging from 0.8 to 1.4 m bgs, overlaying GM and cobbles / gravel.

SM (a soil unit of Silty Sand / Silty Sand with Gravel) is encountered only on the northern and southern portions of the footprint (not in the centre), overlaying cobbles / gravel or bedrock. This soil unit is found to depths ranging from 0.9 to 3.1 m bgs, with the greatest depth occurring at the northeastern corner of the footprint. GM is encountered in select areas at the western edge and centre of the footprint (generally 1 m thick). It is overlain by ML at the western edge and organics at the centre and is underlain by cobbles / gravel.

- Cobbles / gravel are generally found across the whole footprint, excepting at the eastern edge, directly overlaying bedrock.
- Depth to bedrock ranges across the footprint from 0.6 to 5.7 m bgs; elevation of the bedrock surface ranges from 360 to 392 m a.s.l. It generally dips northwest; its greatest depth is at the western edge, and its shallowest is at the eastern edge.

- Groundwater depths range from 0.13 to 0.51 m bgs. With the variations in ground surface, groundwater elevations range from 365 to 389 m a.s.l.

Overburden piles identified in the general layout are as follows:

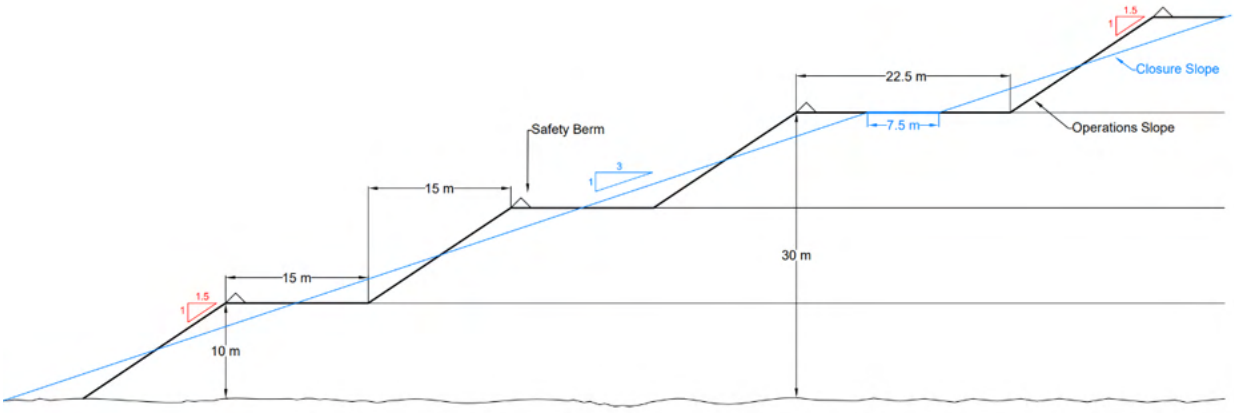
- Overburden Pile 004: Area located southwest of the Pit boundary – Years 1 to 4.
- Overburden Pile 005: Area located southeast of the East Pit boundary – Years 5 to 8.
- Organics Pile: Area located south of Stockpile 001 and north of the Industrial Area – Years 1 to 3.

18.6.10 Waste Rock Stockpiles and Related Infrastructure

The footprint of Stockpile 001 is 80 ha, and the maximum height is about 95 m. The PAG rock portion of Stockpile 002 is approximately 126 ha (58%) with a maximum height of about 120 m. The footprints for both stockpiles will require tree clearing and removal of topsoil / organics. The stripped topsoil / organics will be stockpiled in the Organics Pile for use in closure. Overburden material within the footprint found to be loose / saturated will also be removed, following inspection by a geotechnical engineer, and stockpiled in Overburden Piles 004/005. The basin of the PAG rock portion of Stockpile 002 will be lined with a 1.5 mm double-textured LLDPE geomembrane. Preparation for the geomembrane installation will include proof rolling of the surface and provision of a sand cushion layer in locations where the geomembrane can be damaged by the foundation, e.g. waste rock outcrops. A cushion geotextile will be provided on top of the geomembrane, and a cushion layer 1 m thick will be provided on top of the geotextile prior to the placement of waste rock. The cushion layer for the initial footprint development will consist of overburden stripped from the pit development or sourced from local borrow pits. Once the process plant is operational and tailings are produced, the tailings can be used as a cushion layer for the remainder of the footprint (see Figure 18.27).

The waste rock will be placed in 10 m high lifts with local slopes of about 1.5H:1V. A 15 m wide step-in is provided for each bench. Every third bench will have a step-in of 22.5 m. The benches will be dozed down to provide a local slope of 3H:1V at closure. The wider bench will be maintained, in part, when the intermediate benches are flattened. This is to facilitate the management of water at closure. This geometry provides a global external slope of 3.25H:1V (see Figure 18.31).

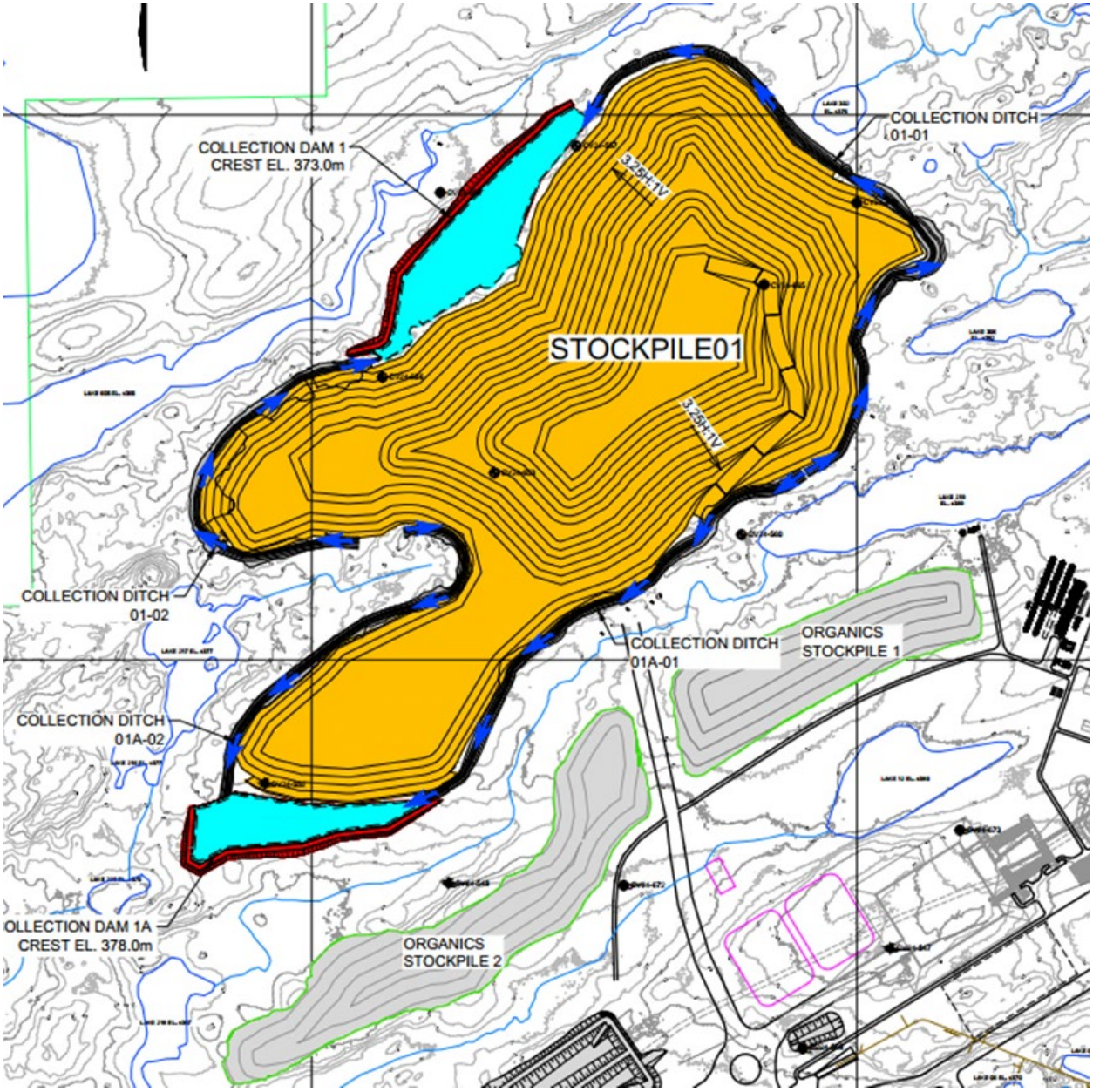
Figure 18.31: Typical Section for Waste Rock in Stockpiles 001 and 002 Showing Intermediate Benches for Both Operation and Closure Stages



Source: AtkinsRéalis, July 2025.

Water management for both stockpiles is achieved by enclosing the stockpile footprints with perimeter ditches and directing the water into water retention ponds. Excess water is pumped from the retention ponds to either the Sedimentation Ponds or the Treatment Plant at the Industrial Area for treatment prior to discharge to the environment or re-use in the plant. Access roads are provided next to and parallel to the drainage ditches. The water pumping lines are shown in the General Arrangement layout. The water treatment is described in Section 18.7.5. A layout of the water management infrastructure for Stockpile 01 is shown in Figure 18.32 and for Stockpile 02 in Figure 18.27. More detailed descriptions of water management infrastructures are provided in Section 18.7.

Figure 18.32: Stockpile 01 and Associated Water Management Infrastructure



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

MTOs are estimated assuming that the soil and rock used for backfilling are non-metal-leaching and non-acid-generating, and that quantities are sufficient either from the quarry source or the open-pit waste rock.

Overburden will be stored in four (4) separate areas:

- Overburden Pile 004 is located south of the pit, has a 23-ha footprint. Preparation of this area should start at early works. The related water management consists of a series of catchment paddocks

located around the stockpile perimeter. The catchment paddocks will be constructed with Low-Risk rock fill. The paddocks will temporarily store runoff, allow for sediments to settle and then seep to the environment or evaporate.

- Overburden Pile 005 is located to the southeast of the West Pit and has a 27-ha footprint; preparation of this area should be completed before the development of the East Pit. As for Pile 004, the related water management consists of a series of catchment paddocks located around the stockpile perimeter. These paddocks will temporarily store runoff, allow for sediments to settle and then seep to the environment or evaporate.
- Two (2) Organics Piles are located south of Stockpile 001 and north of the Industrial Area. They have footprints of approximately 11 ha. Preparation of this area should start at early works. Related water management consists of a series of catchment paddocks located around the stockpile perimeter. These paddocks will temporarily store runoff, allow for sediments to settle and then seep to the environment or evaporate.

18.6.11 Stability Analysis

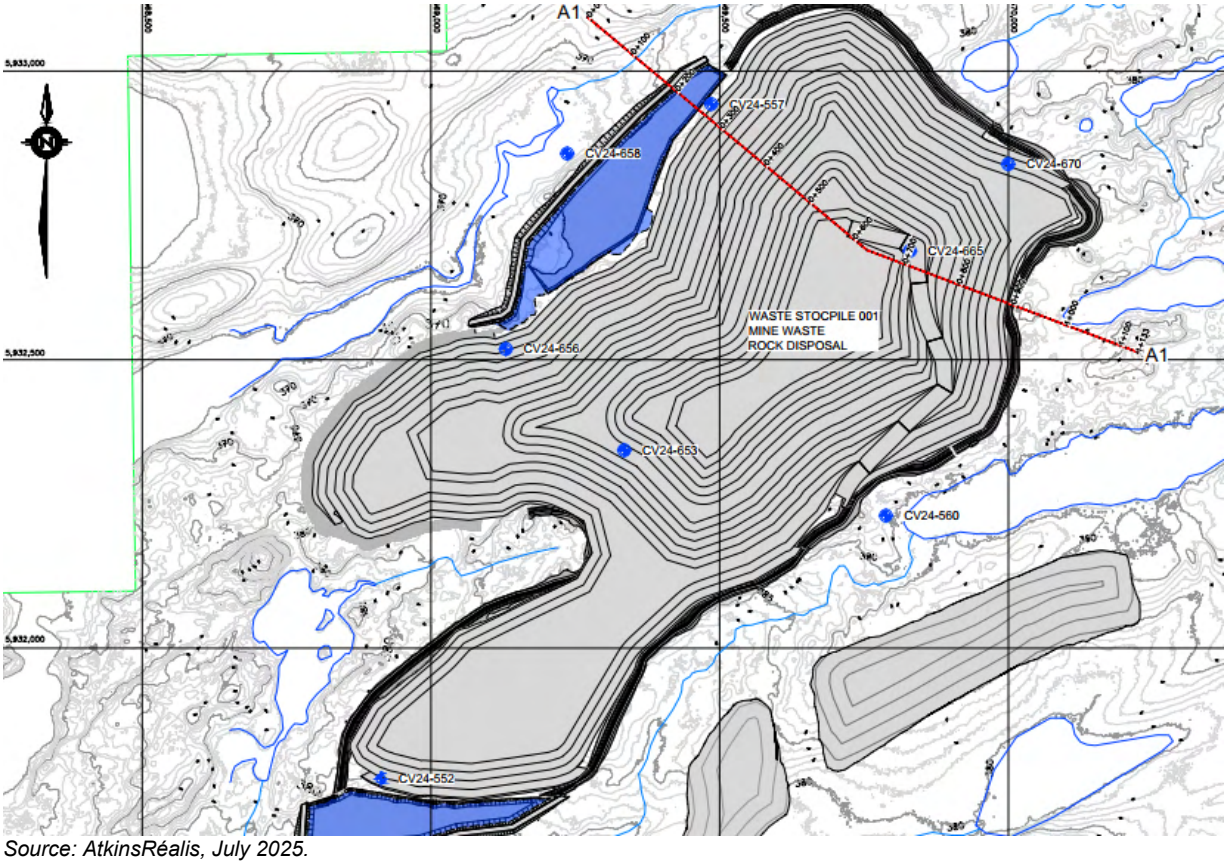
Slope stability analyses were carried out for both Stockpile 001 and 002 using SLOPE/W Version 2023.1.2, a two-dimensional limit equilibrium software program developed by Geo-Slope International Limited. The Morgenstern-Price method was employed to identify potential failure surfaces within the slopes of the stockpiles and their underlying foundations. Stability analyses were performed under both static and pseudo-static conditions for the critical sections: section A1-A1 for Stockpile 001 and section H-H for Stockpile 002 at the PAG portion, as shown in Figure 18.33 and Figure 18.27, respectively.

The minimum required FoS for the stockpiles is based on the Quebec Regulations (MRNF, 2024) and summarized in Table 18.5. The site is classified as Class D, with a design ground acceleration of 0.1 g (used in pseudo-static analysis) for a 1 in 10,000-year event (Extreme), extrapolated from NBC 2020 and used in the model as a conservative approach. This is higher than the requirements for the waste rock facilities, yet it has been chosen for consistency to maintain a single stockpile.

The material density and strength parameters used in the stability analyses were derived from data collected during geotechnical investigations and from AtkinsRéalis' experience with similar materials. The waste rock materials are coarse-grained and are free-draining materials. Therefore, slope stability analyses are performed using effective stress (drained) shear strengths. A summary of the material parameters and materials model used in this study is provided in Table 18.2. Both static and pseudo-static loading were tested to establish the necessary stable configurations of the waste rock stockpile. The soil conditions used in the stability analyses are outlined in Section 18.6.9. A conservative phreatic surface was assigned,

utilizing a steady-state model where the water table lowers to the level of the peripheral drains below the natural terrain.

Figure 18.33: Layout of Critical Sections for Slope Stability Analysis – Waste Rock Stockpile 001 Cross Section A1-A1



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

As outlined in Section 18.6.10, the basin of Stockpile 002 will be lined with a geomembrane, which will have a cushion geotextile placed on top to control seepage (see Figure 18.27). The interface between the geotextile and the geomembrane is critical for slope stability. The analysis revealed that the combination of a 1.5 mm double-textured LLDPE geomembrane and a non-woven geotextile provided the required factor of safety, and this combination has been selected for the design.

Post-earthquake analyses, which typically involve evaluating the residual shear strengths of foundational materials that may have liquefied, were not conducted. This was based on borehole investigations indicating that the foundation materials are not expected to experience liquefaction. Further geotechnical investigations are recommended in the next phase to verify this conclusion. A pseudo-static analysis has been performed to evaluate dynamic stability.

The results of the slope stability analysis are presented in Table 18.5. With the proposed configurations and optimizations, the FoS obtained shows that the stability of both structures meets the minimum required FoS for the operational and closure phases. The FoS in Stockpile 001 is slightly higher than in Stockpile 002, primarily because Stockpile 002 has a greater height. As mentioned earlier, two (2) distinct intermediate bench configurations have been designed for both the operational and closure stages. The FoS during the operational stage is the same as during the closure stage. A slight difference in geometry between the operational and closure phases does not affect the overall stability.

Table 18.5: Factor of Safety of the Slope Stability Analysis – Waste Rock Stockpiles

Stockpile	Phase	Section	Loading Conditions	[1]Minimum Requirement of FoS	FoS	[2]Failure Type
001	Operational	A1-A1	Static (Drained)	1.5	2.53	Global
002	Operational	H-H	Static (Drained)	1.5	2.31	Global
001	Operational	A1-A1	Pseudo-static	1.3	1.91	Global
002	Operational	H-H	Pseudo-static	1.3	1.76	Global
002	Operational	B1-B1	Pseudo-static	1.3	1.60	Global
001	Closure	A1-A1	Static (Drained)	1.5	2.41	Global
002	Closure	H-H	Static (Drained)	1.5	2.32	Global
001	Closure	A1-A1	Pseudo-static	1.3	1.84	Global
002	Closure	H-H	Pseudo-static	1.3	1.75	Global
001	Operational	A1-A1	Static (Drained)	1.2	1.80	Local (Bench 1)
001	Operational	A1-A1	Static (Drained)	1.2	1.30	Local (Bench 2)
001	Operational	A1-A1	Static (Drained)	1.2	1.22	Local (Bench 3)
001	Operational	A1-A1	Static (Drained)	1.2	1.27	Local (Bench 4)
001	Operational	A1-A1	Static (Drained)	1.2	1.26	Local (Bench 5)
001	Operational	A1-A1	Static (Drained)	1.2	1.25	Local (Bench 6)
001	Operational	A1-A1	Static (Drained)	1.2	1.25	Local (Bench 7)
001	Operational	A1-A1	Static (Drained)	1.2	1.38	Local (Bench 8)
001	Operational	A1-A1	Static (Drained)	1.2	1.24	Local (Bench 9)
001	Closure	A1-A1	Static (Drained)	1.2	2.41	Local (Bench 1)
001	Closure	A1-A1	Static (Drained)	1.2	2.33	Local (Bench 2)
001	Closure	A1-A1	Static (Drained)	1.2	2.21	Local (Bench 3)
002	Operational	H-H	Static (Drained)	1.2	1.63	Local (Bench 1)

Stockpile	Phase	Section	Loading Conditions	[1]Minimum Requirement of FoS	FoS	[2]Failure Type
002	Operational	H-H	Static (Drained)	1.2	1.49	Local (Bench 2)
002	Operational	H-H	Static (Drained)	1.2	1.31	Local (Bench 3)
002	Operational	H-H	Static (Drained)	1.2	1.37	Local (Bench 4)
002	Operational	H-H	Static (Drained)	1.2	1.37	Local (Bench 5)
002	Operational	H-H	Static (Drained)	1.2	1.43	Local (Bench 6)
002	Operational	H-H	Static (Drained)	1.2	1.37	Local (Bench 7)
002	Operational	H-H	Static (Drained)	1.2	1.42	Local (Bench 8)
002	Operational	H-H	Static (Drained)	1.2	1.46	Local (Bench 9)
002	Operational	H-H	Static (Drained)	1.2	1.37	Local (Bench 10)
002	Operational	H-H	Static (Drained)	1.2	1.35	Local (Bench 11)
002	Closure	H-H	Static (Drained)	1.2	2.43	Local (Bench 1)
002	Closure	H-H	Static (Drained)	1.2	2.37	Local (Bench 2)
002	Closure	H-H	Static (Drained)	1.2	2.44	Local (Bench 3)
002	Closure	H-H	Static (Drained)	1.2	2.21	Local (Bench 4)

Note: (1) (MRNF, 2024); (BCMWRPRC, 1991b) and (Hawley & Cunning, 2017).
(2) The first bench is the upper one, with subsequent benches arranged chronologically down towards the ground surface.

18.7 Water Management Infrastructure

18.7.1 Contact Water Management

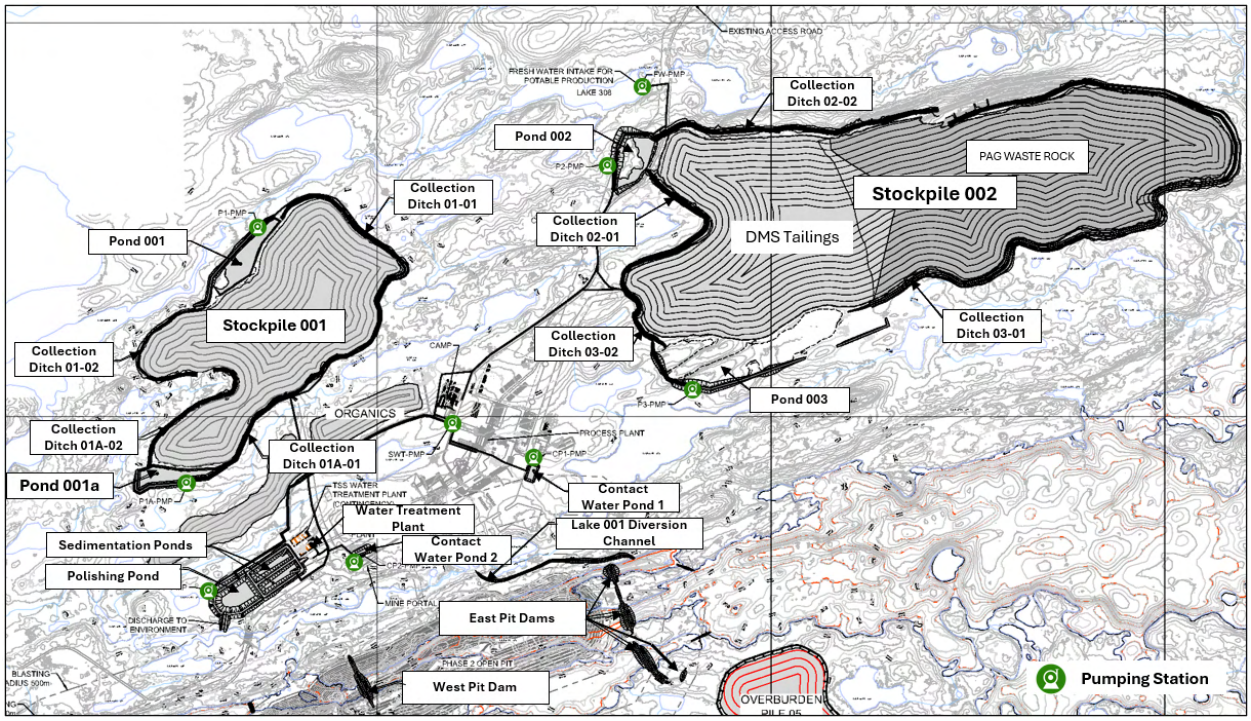
18.7.1.1 General

The water management plan was designed to effectively segregate contact water from non-contact water, in accordance with the requirements of Directive 019 on the mining industry (MDDEP, 2025). Contact water will be managed through a network of ditches and retention basins designed to collect runoff from site infrastructure, including Stockpiles 001 and 002, industrial facilities, and mine dewatering activities (both open-pit and underground). From there, the water is to be discharged into the treatment infrastructure, to be properly treated prior to being either discharged into the environment or reused in the Process Plant. Excess treated water will be discharged into the CE15 Stream (between Lake 001 to Lake 027), ensuring compliance with regulatory standards and maintaining a single, controlled discharge point.

Figure 18.34 illustrates the proposed water management infrastructures for the mine site, which are categorized into four (4) main components:

- Contact Water Collection Ditches: These ditches are designed to collect runoff from Stockpiles 001 and 002 and convey it to the designated contact water collection ponds.
- Contact Water Collection Ponds: These ponds serve as detention storage facilities to collect surface runoff from Stockpiles 001 and 002, and from the Industrial Area. Once the contact water is collected in these ponds, it will be transferred, by pumping systems, to the treatment infrastructure for treatment (shown in Figure 18.34). Moreover, emergency spillways are integrated into the design of all pond embankments to safely convey extreme floods.
- Pumping Stations: Strategically distributed across the site, these stations are to transfer contact water between the collection system and the treatment facilities. The pumping system was also designed to provide operational support to supply water for various mining activities.
- Industrial Area Treatment Infrastructure: This system is the final stage in the water management process. It includes a water treatment plant (described in Section 18.7.5), sedimentation ponds, and a polishing pond that serves as the final treatment before discharge to the environment or re-use in the Process Plant.

Figure 18.34: Site-Wide Projected Water Management Infrastructures



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

18.7.1.2 Contact Water Collection Ditches

Perimeter ditches will be constructed around Stockpiles 001 and 002. These ditches are designed as flood-control channels to direct runoff into designated collection ponds. Following the requirements outlined in Directive 019 (MDDEP, 2025), the ditches are engineered to safely manage runoff from a 1 in 100-year rainfall event, with a freeboard of 0.6 metres. The design also ensures that the ditch embankments will not be overtopped during a Probable Maximum Precipitation (PMP) event. Riprap will be used to provide erosion protection. Table 18.6 summarizes the specifications of the ditches.

The main characteristics of the contact water collection ditches are described as follows:

- Stockpile 001 (Low-Risk Waste Rock): Ditch 01-01, 01-02, 01A-01, and 01A-02 are proposed to be non-geomembrane-protected structures. These ditches were designed to be trapezoidal with 2(H):1(V) side slopes, base widths varying from 1.5 m to 2 m, and depths from 1.0 to 1.2 m. The base of the ditches includes a separation geotextile covered with riprap for erosion protection.
- Stockpile 002: Ditches 02-01, 02-02, 03-01, and 03-02 are proposed to be lined with a double-textured 1.5 mm LLDPE geomembrane due to the presence of PAG materials. The ditches were designed to be trapezoidal with 3H:1V side slopes, base widths varying from 1.5 m to 5.0 m and depths from 1.0 to 2.2 m. The geomembrane is covered with a cushion geotextile and riprap for erosion protection.
- A 4 m service road is included adjacent to and on the downstream side of all ditches.

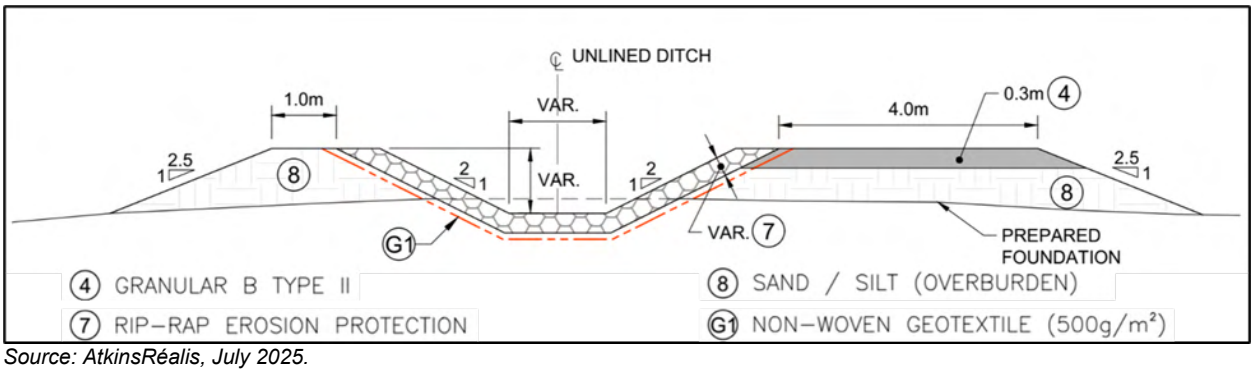
The typical cross-sections for ditches are presented in Figure 18.35 and Figure 18.36. Material take-offs (MTOs) are estimated assuming that soil and waste rock used for construction are non-metal leaching and non-acid generating, and materials are sourced from an on-site quarry / borrow pit or the open pit. Crushing of stone will be required on-site to produce the required materials.

Table 18.6: Water Collection Ditches – Stockpiles 001 and 002

Ditch	Associated Stockpile and Pond	Geometry	Base Width (m)	Side Slope (H:V)	Minimum Depth (m)
Ditch 01-01	Stockpile 001 – Pond 01	Trapezoidal	1.5 to 2.0	2:1	1.0 to 1.2
Ditch 01-02			1.5		1.0
Ditch 01A-01	Stockpile 001 – Pond 01A		2.0		1.0
Ditch 01A-02			1.5		1.0
Ditch 02-01	Stockpile 002 – Pond 02		2.5	3:1	1.2
Ditch 02-02			1.5 to 3		1.1 to 1.3

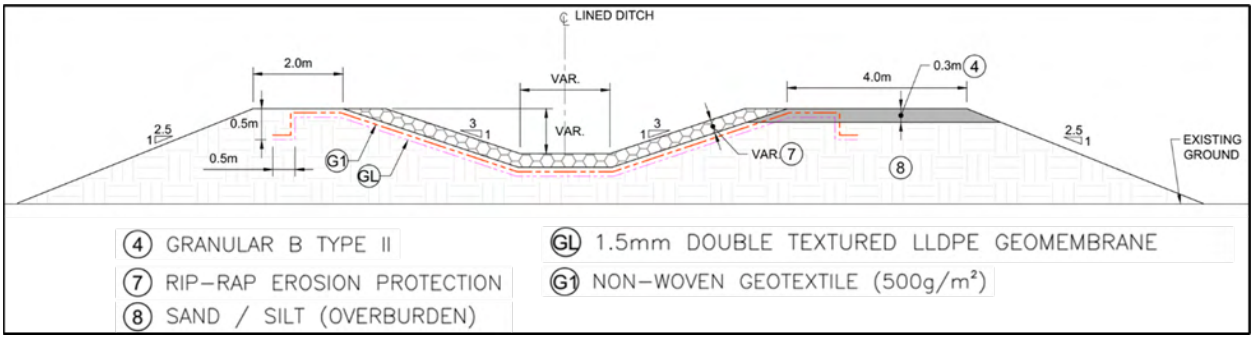
Ditch	Associated Stockpile and Pond	Geometry	Base Width (m)	Side Slope (H:V)	Minimum Depth (m)
Ditch 03-01	Stockpile 001 – Pond 02		2.0 to 5.0		1.2 to 2.2
Ditch 03-02			1.0		1.0

Figure 18.35: Typical Section for Stockpile 001 Ditch with Adjacent Road Inspection



Source: AtkinsRéalis, July 2025.

Figure 18.36: Typical Section for Stockpile 002 Lined Ditch with Adjacent Road Inspection



Source: AtkinsRéalis, July 2025.

18.7.1.3 Contact Water Collection Ponds

As outlined in the previous sections, collection ponds will be constructed to collect contact water. The collected water will be transferred to the treatment infrastructure, which includes sedimentation ponds and a polishing pond located in the industrial area. At this location, the water will undergo treatment before being either discharged into the environment or reused in the process plant.

Runoff from Stockpile 001 is to be collected through the perimeter ditches that will discharge into Pond 01 and Pond 01A. These ponds are designed to provide sufficient retention time and are strategically designed with suction lines to effectively facilitate the treatment of total suspended solids (TSS). Following this treatment, the water will be pumped to the polishing pond located in the industrial area, where additional TSS treatment will be performed to further enhance water quality.

Similarly, runoff from Stockpile 002, which contains PAG and metal leaching (ML) waste rock and tailings, is collected using perimeter ditches that will discharge into Pond 02 and Pond 03. These ponds serve as initial containment before the water is transferred by the pumping operations to the Industrial Area for trace metal treatment prior to discharge to the environment or reuse in the process plants.

Additionally, contact water from Industrial Area, such as roads, ROM pads, parking areas, and mine dewatering (both open-pit and underground), will be collected and transferred to sedimentation ponds to ensure the removal of TSS.

The design of the Contact Water Ponds complies with the recommendations provided in Directive 019 (MELCCFP, 2025), which outlines that ponds must be capable of storing and managing contact water, including the flood volume generated during the environmental design flood (EDF). The flood management strategy includes active pumping of snowmelt during the spring freshet and contingency storage capacity equivalent to two (2) days of pumping in case of pump failure.

The adopted EDF Design Criteria for collection ponds are as follows:

- Stockpile 001 (Pond 01 and Pond 01A): Combination of the 100-year rainfall event in 24-hr and the 100-year snowmelt in 13 days, with a required freeboard of 1.0 m above the peak EDF water level.
- Stockpile 002 (Pond 02 and Pond 03): Combination of the 2,000-year rainfall event in 24-hr and the 100-year snowmelt in 13 days, with a required freeboard of 1.5 m above the peak EDF water level.
- Industrial area collection ponds (CP1 and CP2): Combination of the 100-year rainfall event in 24-hr and the 100-year snowmelt in 13 days.

Moreover, to ensure safe operation during extreme precipitation events, emergency spillways are incorporated into the design of all ponds that require dams, including ponds 01, 01A, 02 and 03. These structures were designed to safely convey extreme flood events, preventing overtopping. In accordance with Directive 019 (MDDEP, 2025), all emergency spillways were designed based on the inflow design flood (IDF), which corresponds to the probable maximum flood (PMF), and include a freeboard of 0.6 m above the peak PMF water level. Erosion protection measures were also provided to ensure long term stability.

Table 18.7 summarizes the specifications of each contact water pond.

Table 18.7: Contact Water Collection Ponds

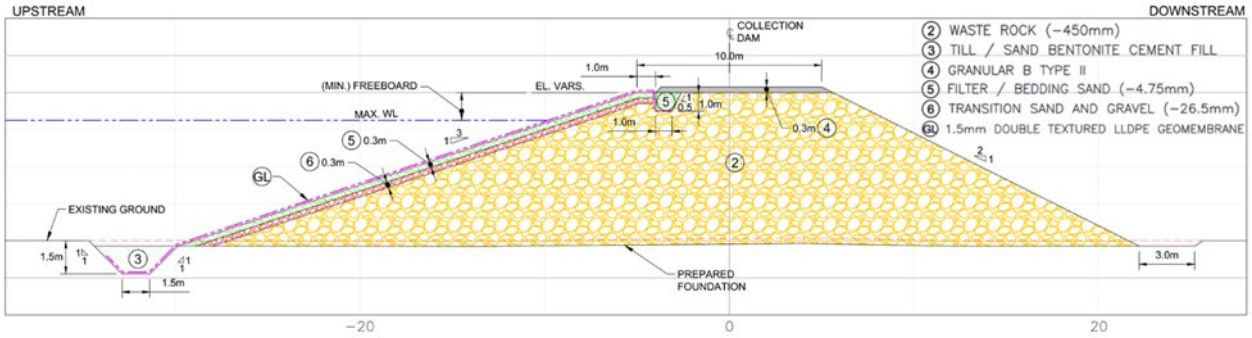
Pond	Description	Catchment Area (m^2)	Required Storage Capacity (m^3)	Associated Collection Ditches
Pond 01	Collect runoff from Stockpile 001	621,400	80,000	Ditch 01-01, Ditch 01-02
Pond 01A	Collect runoff from Stockpile 001	378,500	52,000	Ditch 01A-01, Ditch 01A-02
Pond 02	Collect runoff from Stockpile 002	935,300	137,000	Ditch 02-01, Ditch 02-02
Pond 03	Collect runoff from Stockpile 002	1,933,200	293,000	Ditch 03-01, Ditch 03-02
CP1	Collect runoff from industrial area infrastructure	435,000	10,000	NA
CP2	Collect runoff from industrial area infrastructure	230,000	6,000	NA

The main characteristics of ponds are described as follows:

- Ponds 01, 01A, 02 and 03: The embankments for the ponds will be constructed primarily from waste rock with a transition stone layer and sand cushion layer on the upstream slope. A crest width of 10 m was provided with an upstream slope of 3H:1V and a downstream slope of 2.5H:1V. The embankments will be lined with a 1.5 mm textured geomembrane. For Ponds 02 and 03, the pond basin will be lined with a 1.5 mm textured LLDPE geomembrane. The geomembrane will be exposed within the basin and on the upstream slope. The elevation of the pond embankments is determined by applying freeboard criteria that satisfy both EDF and IDF conditions as discussed above.
- CP1 and CP2: CP1 and CP2 are excavated ponds with no embankment dams. The excavated side slopes are proposed to be 3H:1V with a 1.5 mm textured LLDPE geomembrane on the prepared native foundation. The base of the pond is a 0.3 m thick ballast layer over a protection geotextile, while the lined slopes will be exposed. A 1 m freeboard is allowed above the maximum operating water level based on 1 in 100-year rainfall plus snowmelt EDF. Since the ponds are entirely within excavated ground with no containment dams, no spillway is required. An underdrainage system including a pump-out manhole is provided at each pond to prevent hydraulic uplift of the liner.

Typical cross-sections for the contact water collection ponds embankments are presented in Figure 18.37.

Figure 18.37: Waste Rock Stockpile Water Retention Dam Section



Source: AtkinsRéalis, July 2025.

18.7.1.4 Industrial Area Water Treatment Infrastructure

As outlined in the previous sections, two (2) sedimentation ponds were planned and proposed as part of this project to remove suspended solids from water collected from various sectors, which will be transferred using pumping systems. The sedimentation ponds will be discharging into the polishing pond by gravity.

The main pumping inflows to the sedimentation ponds are as follows:

- Dewatering of the open pit.
- Dewatering of the underground mine.
- Runoff from the industrial area (initially collected in CP1 and CP2).

During the winter months, the ponds CP1 and CP2 can be emptied into the polishing pond using a diesel pump. This operation enables the complete removal of accumulated sediments from these facilities. In the following winter, dewatering flows can be redirected to the cleaned pond, allowing for maintenance and cleaning of the other pond in turn.

As indicated, the polishing pond constitutes the final stage of the treatment process, serving three (3) main purposes:

- It consolidates all mine-related inflows, providing a single discharge point.
- It offers final treatment of the water before discharge.
- It will act as a reserve of freshwater for the process plant and the paste plant.

For the feasibility study, the sizing of the ponds was based on hydraulic retention time (HRT). To ensure adequate water management during the EDF, the ponds are required to provide a minimum HRT of

16 hours under EDF conditions. For the sedimentation ponds, the HRT has been increased to 18 hours to enhance performance.

Table 18.8 summarizes the design specifications of these ponds.

Table 18.8: Industrial Area Ponds

Pond	Inflows	Geometry (H:V)	Volume (m^3)	Hydraulic Retention Time (During Normal Operation) (Hr)	Hydraulic Retention Time (During EDF) (Hr)
Sedimentation Pond 01	Dewatering of the open pit Dewatering of the underground mine	4:1	42,000	>37	>18
Sedimentation Pond 02	Industrial area (CP1) Industrial area (CP2)	4:1	42,000	>44	>18
Polishing Pond	All	1:1	140,000	>36	>16

The main characteristics of ponds are described as follows:

- The embankments for the ponds will be constructed primarily from waste rock with a transition stone layer and sand cushion layer on the upstream slope. A crest width of 10 m has been provided with an upstream slope of 3H:1V and a downstream slope of 2H:1V. The embankments are sized to allow for vehicular access and accommodate the anchor trench and safety berms. These requirements can be reviewed as part of the next phase of the project. The upstream embankment slopes and pond basin will be lined with a 1.5 mm textured LLDPE geomembrane. The base of the ponds is provided with a 0.3 m thick ballast layer over a protection geotextile, while the lined slopes will be exposed. The elevation of the pond embankments is determined by applying freeboard criteria that satisfy both EDF and IDF conditions above the required storage volumes provided. Specifically, the required freeboard is the greater of 1.0 m above the peak EDF water level or 0.6 m above the peak IDF water level.
- Each sedimentation pond contains an access ramp to facilitate the periodic removal of accumulated sediments.

- Each sedimentation pond is provided with a rectangular concrete spillway to convey the EDF and IDF flows to the polishing pond to prevent overtopping of the sedimentation ponds.
- The polishing pond was designed to discharge normal flows via three (3) 36-inch diameter culverts through the dam. An emergency spillway is also provided in the design for safe discharge of inflows exceeding the EDF. The emergency spillway has a trapezoidal configuration with a 10 m wide concrete sill in the dam. The spillway channel configuration (downstream of the weir / control point) was determined based on the terrain and spillway crest, which resulted in a longitudinal slope of 18% at the proposed location. The discharge channel includes a separation geotextile with riprap provided as erosion protection.

18.7.1.5 Pumping Stations

Pumping stations are required to transfer contact water across the site and to supply water for mining activities. The objectives of the pumping stations are as follows:

- To pump contact water from runoff ponds to either active or passive water treatment systems. As mentioned, this includes actively pumping the 1 in 100-year snowpack melt during environmental flood events.
- To pump treated water from the polishing pond to the process plant, paste plant, and underground mine.

Depending on the collection pond's depth (more or less than 6 m), centrifugal or vertical pumps are employed to transfer contact water. Purge systems are designed for each pipeline: either using vacuum breakers and drain valves for lines carrying non-contaminated water, or pigging systems with compressors for contaminated water.

Each pumping station includes one (1) standby pump to ensure redundancy. A sufficient inventory of spare parts is maintained to allow for quick repairs in case of failure. All pumps are equipped with variable frequency drives (VFD) to adjust flow rates based on weather conditions. Pipes are sized based on total dynamic head and are made of HDPE to minimize head loss. Each pumping station is served by a dedicated pipeline. All pipelines are installed above ground on pipe benches along the roadside. The pipeline carrying treated water to the process facilities is fully insulated and heat-traced to prevent freezing.

18.7.1.6 Annual Contact Water Balance

A GoldSim model was developed to assess the annual contact water balance at the mine site. This model estimates the volume of water that must be managed, treated, and discharged into the environment

throughout the mine's operational life. It also confirms the availability of process water in the different scenarios.

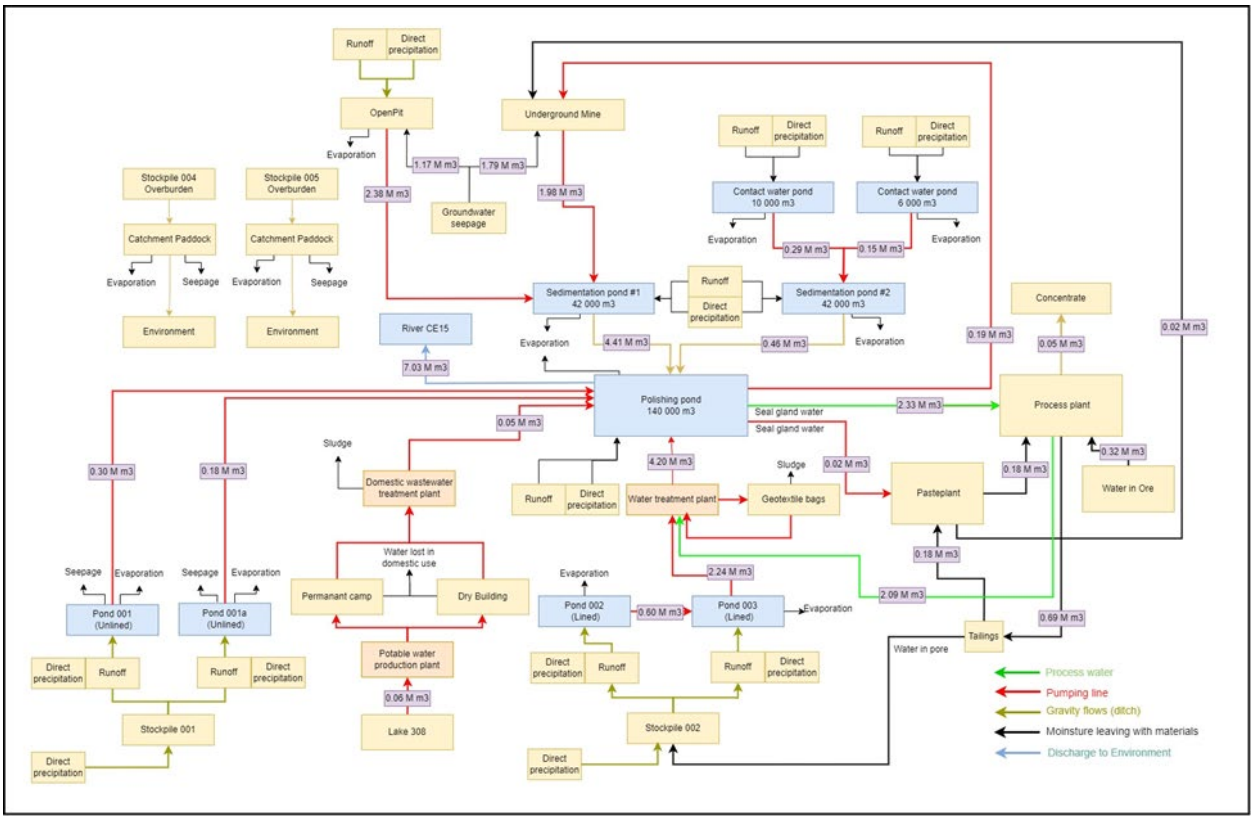
Three (3) hydrological scenarios were considered in the evaluation: average year, wet year, and dry year. To account for climate change, an 8% increase in total precipitation was applied to both the average and wet year scenarios. Additionally, extreme events, such as the EDF defined in Directive 019, were simulated at both the beginning and end of the mine's life. This ensures that water retention structures and pump stations are appropriately sized.

Figure 18.38 illustrates the flow diagram that was developed for the final site configuration and includes the annual flow rates under average conditions.

Key findings from the water balance analysis include:

- The fresh water supply varies from 4,424 m^3/month to 4,898 m^3/month.
- Under average conditions, when the mine is fully developed, the volume of excess water discharged to the environment ranges from 286,000 m^3 (February) to 989,000 m^3 (May) per month.
- On an annual basis, the excess water discharged to the environment is estimated at 4,950,000 m^3 in dry conditions, 7,030,000 m^3 in average conditions, and 8,410,000 m^3 in wet conditions.
- Treated water stored in the polishing pond will be sufficient to supply the process plant and other operational needs throughout the life of mine, under various climate scenarios.
- The runoff ponds located at the base of the stockpiles are adequately sized to manage the EDF.
- The pumping stations, designed to handle the Directive 019 snowmelt event, ensure efficient and reliable transfer of water to both passive and active treatment systems.
- The hydraulic retention time (HRT) in the passive treatment infrastructures consistently remains high enough to allow for effective particle settling.
- Under normal conditions, the ponds operate between the lowest operating water level (LOWL) and maximum operating water level (MOWL), utilizing only 40% of the pump capacity.

**Figure 18.38: Site Water Management Flow Diagram
(including annual flow rates during average climate condition)**



Source: AtkinsRéalis, July 2025.

18.7.2 Non-Contact Water Management

18.7.2.1 Lake 001 Water Management

18.7.2.1.1 Lake 001 Dewatering

The dewatering of Lake 001, required for the phased development of the open pit mine, will follow the same staged approach as the pit development. Dewatering strategies were developed for both West and East Pit phases, based on the lake's bathymetry, surrounding topography, and the progressive expansion of the pit.

In the West Pit phase, dewatering targeted the western portion of Lake 001 after the construction of the West Pit cut-off dams. Submersible pumps powered by external generators are suggested to lower the water level. The dewatering process will be supported by the early construction of sedimentation ponds, which will allow for the treatment of contact water generated during this phase.

The dewatering duration for the West Pit phase was estimated to be approximately one (1) month, based on preliminary engineering assessments of pumping estimates in 24 hours. During this phase, the volumes of collected water were calculated to be pumped out to the sedimentation ponds located in the industrial area for treatment before discharge.

The East Pit phase involved the extension of the open-pit footprint further into the former lake area, which also required dewatering strategies. To support this, submersible pumps and external generators were again recommended. The dewatering operation was scheduled to begin after the spring freshet to avoid overloading the sedimentation ponds. The estimated duration for this phase was a minimum of three (3) months, based on preliminary assessments.

To manage water in deeper sections of the pit, diesel pumps were proposed to supplement the submersible systems and ensure complete drainage of the area.

A berm will be constructed around the perimeter of the open pit to prevent surface runoff from entering the pit. Any water that accumulates within the pit is to be classified as contact water and managed using a permanent pump station. This pumping system is designed to transfer water to the sedimentation ponds, and it was sized to manage rainfall events with a return period of 1 to 5 years.

For more extreme precipitation events, it is assumed that excess water will temporarily accumulate in the pit and be pumped back to the sedimentation ponds using the dewatering system.

18.7.2.1.2 Fish Relocation

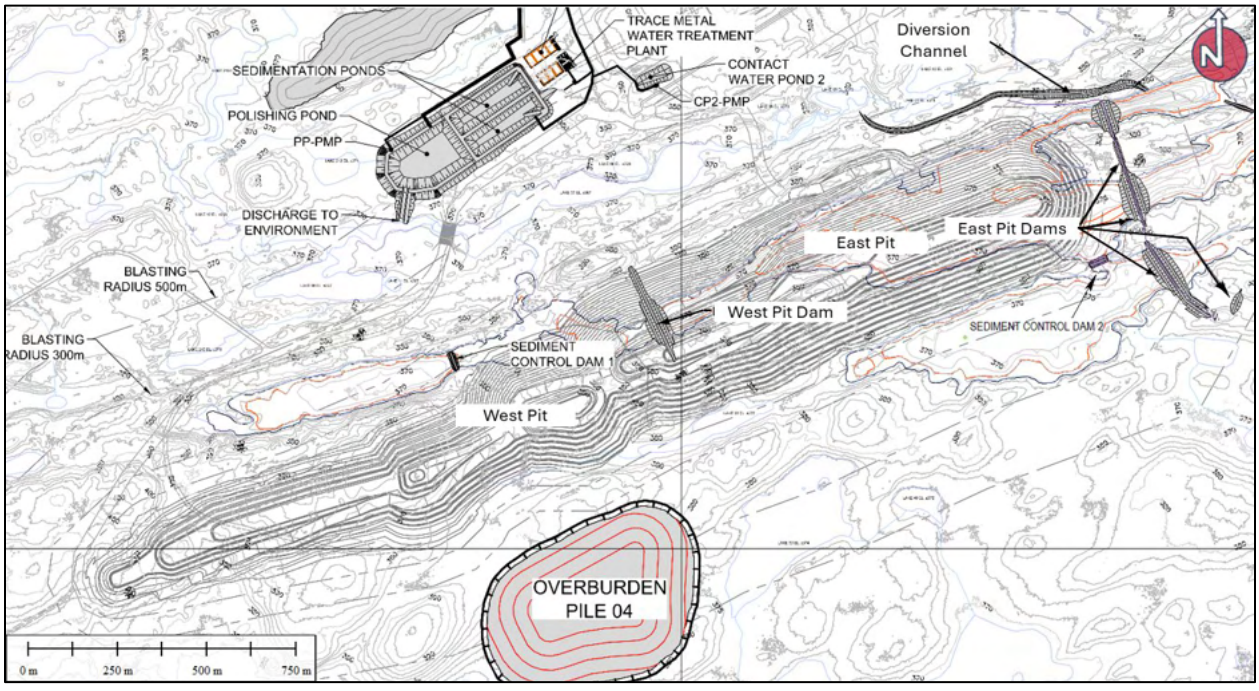
Before the commissioning of dewatering strategies for the open pit, and following the construction of the cut-off dam, a fish relocation program will be implemented to ensure compliance with applicable environmental regulations and aquatic habitat protection guidelines. This program will focus on the eastern portion of the lake, where fish may become trapped due to the modified hydrology. The program is established to capture the fish habitat within this isolated area and relocate it as safely and appropriately as possible to suitable nearby aquatic habitats, in coordination with regulatory authorities and following best practices for fish handling and habitat disturbance. The relocation will be conducted before any significant drawdown or disturbance of the water body to minimize ecological impacts and ensure the protection of aquatic life.

18.7.2.1.3 Diversion Channel

The open pit will be developed in two (2) phases, the West Pit and the East Pit, which will incorporate a portion of Lake 001. A diversion of the lake, around the pit, is therefore required. The required infrastructure includes a set of cut-off dams within the lake and a diversion channel. The diversion channel will be constructed in a single stage in Year 2. The cut-off dams will be staged and are described in more detail in Section 18.7.2.1.4. The combination of the diversion and cut-off dams will allow the eastern portion of the pit development to encroach on a portion of the dewatered footprint of Lake 001. The total watershed of Lake 001 is 106 km^2. The total drainage area impacted by the diversion is approximately 1.9 km^2, which represents 1.7% of the total Lake 001 watershed.

The diversion channel will divert flows from Lake 001 to Lake 05. It will have a total length of approximately 870 m, a 5 m wide base and an average gradient of about 0.3%. The bulk of the channel will be blasted through bedrock, and hence the side slopes are at 1 Horizontal: 1 Vertical. The alignment of the channel, as shown in Figure 18.39, was selected to have a minimum 45 m offset from the open pit, a minimum 60 m offset from Lake 02 and Lake 03 and a 10 m offset from Lake 04.

Figure 18.39: Lake 001 Diversion Channel and Cut-off Dams



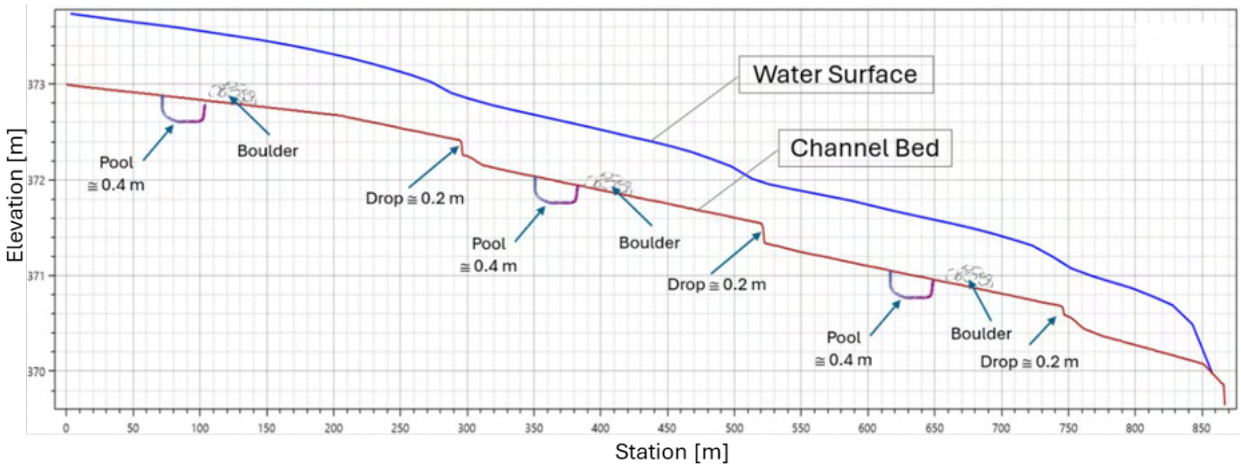
Source: AtkinsRéalis, July 2025.

The diversion channel and the cut-off dams will maintain the normal flow of the lake and safely convey the Probable Maximum Flood (PMF) with adequate freeboard. The PMF discharge was calculated to be approximately 94.9 m^3/s. A minimum freeboard of 1.5 m is required during flooding. The invert elevation of

the channel is at elevation 373 m a.s.l. (similar to the existing outlet) and the required cut-off dam crest elevation is 378.4 m a.s.l.

The diversion channel will replace the existing natural stream discharging from Lake 001 to the CE 15 stream. The design of the channel should therefore replicate the hydraulic conditions of the natural stream for fish habitat and fish passage. This will be achieved by providing a series of drops and pools along the channel bed to create varying hydraulic conditions. These hydraulic conditions will facilitate rapid flows and water depths, both upstream and downstream of the drops. Boulders will be placed between sections of the channel where high velocities are present to provide shelter for fish during their migration from downstream to upstream. Figure 18.40 shows a schematic representation of the proposed fishway channel. Comparison between the results of the hydraulic modelling and field measurements[3] showed that the velocities at the depths of the diversion channel are within a similar range of values to those measured in the natural stream.

Figure 18.40: Profile of Diversion Channel with Proposed Fish Habitat / Passage



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

18.7.2.1.4 Cut-Off Dams

The cut-off dams will be constructed on the east side of the open pit to direct flows from Lake 001 to the north through a diversion channel (as described in Section 18.7.1.3) and to keep the open pit area dry. The dams will be constructed in two (2) phases to match the pit development. For the West pit, a single dam is required within Lake 001. A portion of the dam extends onto dry land, next to Lake 001, and this is to ensure

[3] PMET Report – Technical Note Diversion Channel and Fish Habitat Requirements, Feb 11, 2025.

that the flood water does bypass the dam. For the East pit development, a total of four (4) dams is required within Lake 001. These dams are located to the east of the West pit dam.

The dams have been classified by AtkinsRéalis in terms of both the CDA Dam Safety Guidelines (CDA, 2013) and the Quebec Dam Safety Act. The CDA dam hazard classification is "Extreme", and the Quebec Dam Safety Act classification is "Class C", with a P value of 58.

The dam classification determines the return period for the seismic event and the Inflow Design Flood (IDF) used in the dam design. For a dam classified as "Extreme," a return period of 1 in 10,000 or the Maximum Credible Earthquake is required according to CDA guidelines from 2019 and MELCCFP 2025. The IDF is the Probable Maximum Flood (PMF), which considers Probable Maximum Precipitation (PMP), according to CDA (2019); MELCCFP (2025). As described in Section 18.7.2.1.3, to ensure a minimum freeboard of 1.5 m during flooding, the dam crest elevations are required to be at 378.4 m a.s.l.

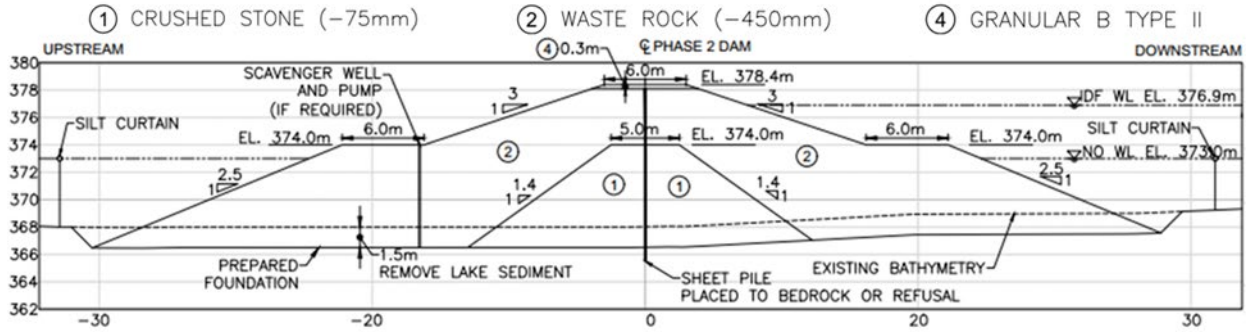
The Lake 001 cut-off dam is designed with two (2) main cross-sections: one for the dam within the lake (the critical section), and the other for the dam on dry land. The following features are included in the dam design (see Figure 18.41):

- Silt curtains will be required during construction on both the upstream and downstream sides to reduce sediment transport.
- Lake sediments will be removed from the dam foundation, either by displacement of the dumped rock or by dredging.
- The dams are constructed with geochemically inert rockfill. A finer crushed rock (referred to as Zone 1) is provided centrally, and the smaller particles facilitate the installation of sheet piles. The sheet piles are provided for seepage management. The Zone 2 rockfill provides the necessary stability for the dam. The berms will be constructed by dumping and dozing the rockfill, starting at the edge of the lake.

As shown in Figure 18.41, the normal operating water level of Lake 001 is at elevation 373 m a.s.l. Therefore, the upstream berm crest elevation is designed to be at elevation 374 m a.s.l, maintaining a height of approximately 1 metre above the water level. The crest width of the dam has been designed to be 6.0 metres. Both the upstream and downstream slopes of the cut-off dams will have slopes of 3 horizontal to 1 vertical (3H:1V). The 6-metre-wide toe berms on both the upstream and downstream sides have a slope of 2.5 horizontal to 1 vertical (2.5H:1V). The top of the sheet pile wall will be trimmed to an elevation of 378.1 m a.s.l. An additional 0.3 metres of road subbase material will be placed over it, resulting in a final dam crest elevation of 378.4 m a.s.l.

At closure, these dams will be breached or their height reduced to allow for flow into the open pit.

Figure 18.41: Typical Cross-Section for Cut-Off Dam (critical section)



Source: AtkinsRéalis, July 2025.

18.7.2.1.5 Seepage and Stability Analysis for Cut-Off Dams

Seepage and stability analyses were completed for the cut-off dams. Table 18.9 summarizes the parameters applied in this study. Material density and strength parameters used in the stability analyses were determined from data collected during geotechnical investigations, along with AtkinsRéalis' experience with similar materials. The hydraulic conductivity parameters are based on AtkinsRéalis' experience with similar materials.

Table 18.9: Hydraulic and Mechanical Properties for the Seepage & Slope Stability Analyses

Material Description	γ (kN/m^3)	C' (kPa)	Φ' (˚)	k_{sat} (m/s)	Model Type
Waste Rock (-450 mm)	20	0	38	5×10^{-4}	Saturated / Unsaturated
Crushed Stone (-75 mm)	19	0	32	5×10^{-5}	Saturated / Unsaturated
Granular B Type II	19	0	34	2×10^{-5}	Saturated / Unsaturated
Bedrock	Impenetrable			7×10^{-7}	Saturated
Gravel (GW)	19	0	36	5×10^{-4}	Saturated
Sandy Gravel (GP-GM)	19	0	35	5×10^{-5}	Saturated
Silty SAND (SM)	17.8	0	31	1×10^{-6}	Saturated
Sandy Silt (ML)	16.5	0	29	1×10^{-7}	Saturated

Steady-state, two-dimensional seepage analyses were conducted for the Cut-off Dams in Lake 001 to evaluate the seepage flow through the dams. The results were used in planning the dewatering of the open pit. The upstream phreatic surface and hydraulic boundary of the dam are assumed to be at the normal

water level (NWL) for Lake 001. The seepage analysis estimated the flow into the open pit to vary from 360 m^3/day for the West Pit phase to 1,000 m^3/day for the East Pit phase.

Stability analyses were conducted using the 2-dimensional software SLOPE/W 2022 Version, developed by GEO-SLOPE International Ltd. A general limit equilibrium model employing the Morgenstern-Price method was utilized to calculate the minimum Factor of Safety (FoS) for potential failure surfaces. The minimum FoS values follow the guidelines of MDDEP (MDDEP, 2012)**;** MRNF (MRNF, 2024) and the Canadian Dam Association (CDA, 2019).

The loose silty sand / sandy silt materials within the dam foundation will be completely removed during construction, making the foundation non-liquefiable, both statically and seismically. Consequently, a post-liquefaction analysis was not conducted. Instead, a pseudo-static analysis was performed to assess the dynamic stability of the dam. The site is classified as Class D, with a design ground acceleration of 0.1 g for a 1 in 10,000-year event (HPC: Extreme), as based on NBC, 2020 and extrapolated.

For the long-term loading condition, water is stored upstream of the dam at the normal water level (NWL) of the existing lake. During the IDF condition, the upstream phreatic surface of the dam is assumed to be at the IDF level of the upstream lake. The downstream phreatic surface was modelled both at the ground surface and, more conservatively, at an elevated level within the dam.

Multiple loading conditions were conducted to establish the necessary stable configurations of the dam. Table 18.10 summarizes the results of the stability analyses, and the dams meet the required factors of safety. Additional geotechnical investigations in the lake are planned for the next phase of work, and the dam's geometry can be further optimized.

Table 18.10: Slope Stability Analysis Results, Diversion Dams

Zone	Loading Condition	FoS	Minimum FoS *
Upstream	End of each construction phase (short-term)	2.45	1.3 to 1.5
Upstream	In the presence of the project flood (short-term)	2.31	1.3
Upstream	Stationary conditions (long-term)	2.39	1.5
Upstream	Pseudo-static	1.67	1.1
Downstream	End of each construction phase (short-term)	2.43	1.3 to 1.5
Downstream	In the presence of the project flood (short-term)	2.34	1.3
Downstream	Stationary conditions (long-term)	2.63	1.5
Downstream	Pseudo-static	1.99	1.1

**Source: MDDEP (MDDEP, 2012); MRNF (MRNF, 2024).*

18.7.2.1.6 Sediment Management Strategy

The dewatering of Lake 001 as part of the phased development of the Open Pit requires a staged sediment and water management strategy to mitigate environmental risks, ensure operational efficiency, and long-term stability. Strategies have been developed for both phases, the West phase and the East phase of the development of the open pit, based on the evolving pit footprint and the associated drainage area and topography / bathymetry of the sector that included the existing boundary of the lake.

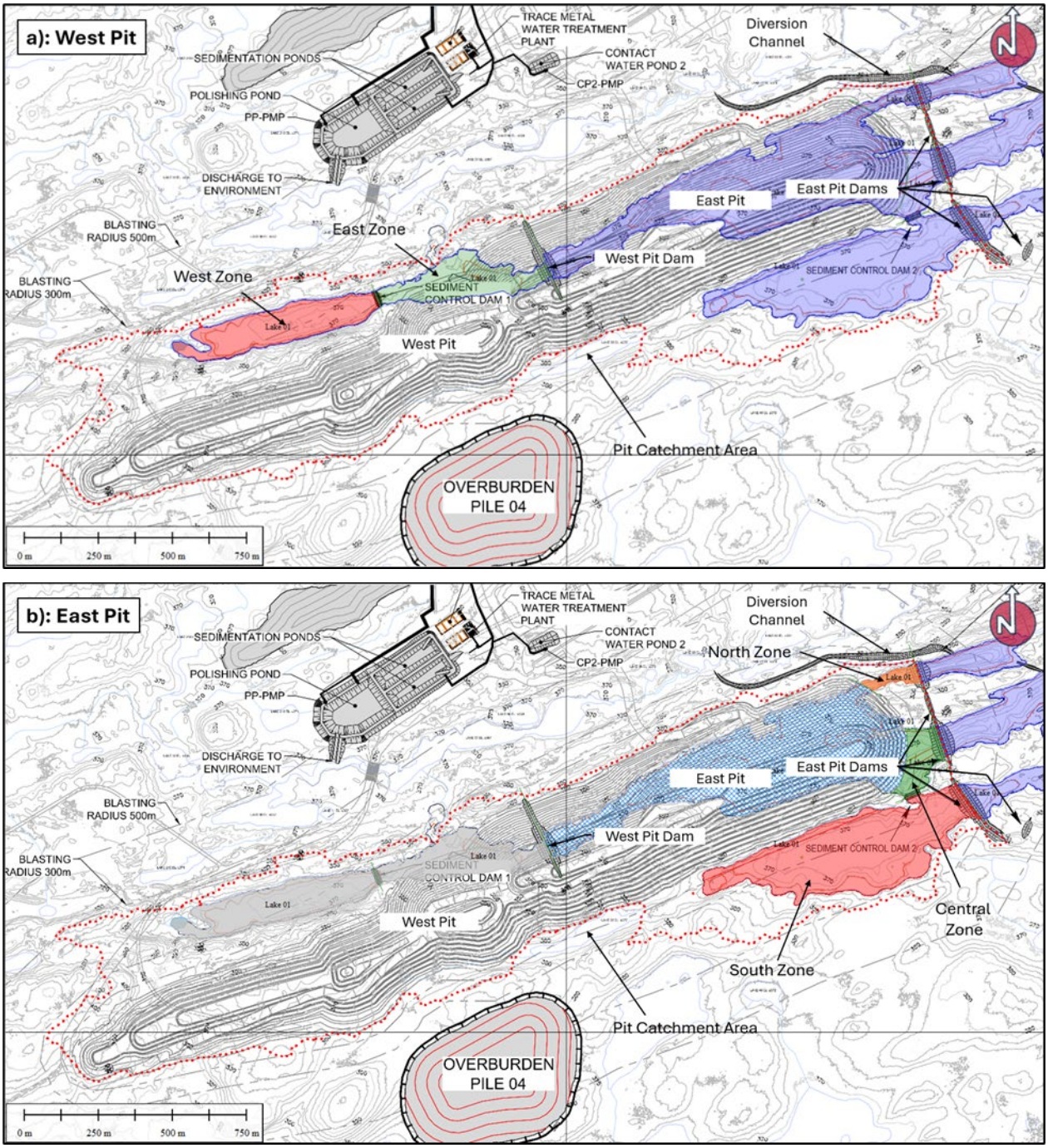
For the West Pit phase, the western portion of Lake 001 is to be dewatered following the construction of the West Pit cut-off dams. The sediment and water management approach involved dividing the dewatered area into two (2) zones, west and east, by constructing a berm (sediment control dam 1) at a narrow and shallow point within the former lakebed (Figure 18.42). In the east zone, sediments are to be removed to prevent mobilization into the pit. Runoff from this area will flow directly into the open pit, where it is classified as contact water and subsequently transferred to the Industrial Area ponds for treatment of sediment. In the west zone, sediments are proposed to be left in place. To prevent sediment-laden runoff from entering the pit, a permeable berm along with a pumping system is proposed to be set at this location that will allow to operation of the runoff volumes up to the 10-Year flood volume. The runoff will be pumped to the sedimentation pond in the industrial area before discharging to the environment.

For the East Pit phase, the open pit footprint is proposed to be extended further into the former lake area, requiring additional sediment and water management measures across three (3) zones: north, central, and south (Figure 18.42). In the North and Central zones, sediments are proposed to be removed to eliminate the risk of mobilization. The runoff from this area will be diverted into an open pit and then pumped out to the industrial area for treatment. The South zone will be separated by a berm (sediment control dam 2) to prevent sediment from reaching the pit. The berm, located at the highest surface of the natural topography, was designed to contain flood volumes up to a 10-year return period. In the South Zone, sediments are to be left in place. A pumping system is proposed to be installed to manage runoff by pumping to the sedimentation pond for TSS control prior to environmental discharge.

The main characteristics of the West and South berms are as follows:

- All lake sediments will be removed from the footprint of the berms.
- The berms will be constructed primarily from waste rock.
- A crest width of 5.0 m has been provided with an upstream and downstream slope of 1.5H:1V.
- The crest elevation of both West and South berms is proposed to be at El. 372.5 m.

Figure 18.42: Open Pit (West Pit and East Pit) Sediment and Water Management Layouts



Source: AtkinsRéalis, July 2025.

18.7.2.2 Non-Contact Water Diversion Ditches for Stockpile 002

Temporary non-contact water diversion ditches will be implemented to redirect clean surface runoff away from the footprint of Stockpile 002 during the initial four (4) years of the mine development. These ditches were designed to minimize the volume of contact water by intercepting and rerouting upstream flows from

the stockpile area into the natural drainage system. The design parameters are based on a 25-year rainfall event and include a minimum freeboard of 0.3 m to prevent overtopping.

The main characteristics of the non-contact water collection ditches, namely NC-01, NC-02, and NC-03, are described as follows:

- Trapezoidal shape with 2H:1V side slopes and a 2.0 m base width.
- The ditches will be lined with a separation geotextile and riprap for erosion protection.
- The ditches are temporary and will be decommissioned as the stockpile footprint expands.

The typical cross-sections of the ditches are presented in Figure 18.35 and Figure 18.36.

18.7.3 Fresh Water Intake and Potable Water Production

Fresh water for potable use will be sourced from Lake 308. There will be only one (1) intake and one (1) treatment plant for the entire site. Water will be pumped from a barge using two (2) submersible pumps to ensure full redundancy. An above-ground, insulated, and heat-traced pipe will run between the pumps and the treatment plant.

The raw water will be treated using ultrafiltration and nanofiltration to remove total suspended solids (TSS) and total organic carbon (TOC) from the surface water year-round. Disinfection will be achieved through ultraviolet (UV) treatment and chlorination. Potable water storage tanks will ensure an adequate supply, even during peak demand periods such as shift changes.

18.7.4 Sewage Water Treatment

Sewage generated from various buildings will be collected and directed to an equalization tank to buffer flow variations. The wastewater will then undergo biological treatment using a moving bed biofilm reactor (MBBR) system. To ensure high-quality effluent, phosphorus removal will be achieved through chemical coagulation, followed by disinfection using UV lamps. All sludge produced during the domestic wastewater treatment process will be extracted, stored in a dedicated tank, and transported by a specialized contractor to an authorized disposal facility. The potential for sludge stabilization through liming and subsequent disposal on the overburden will be assessed in future studies.

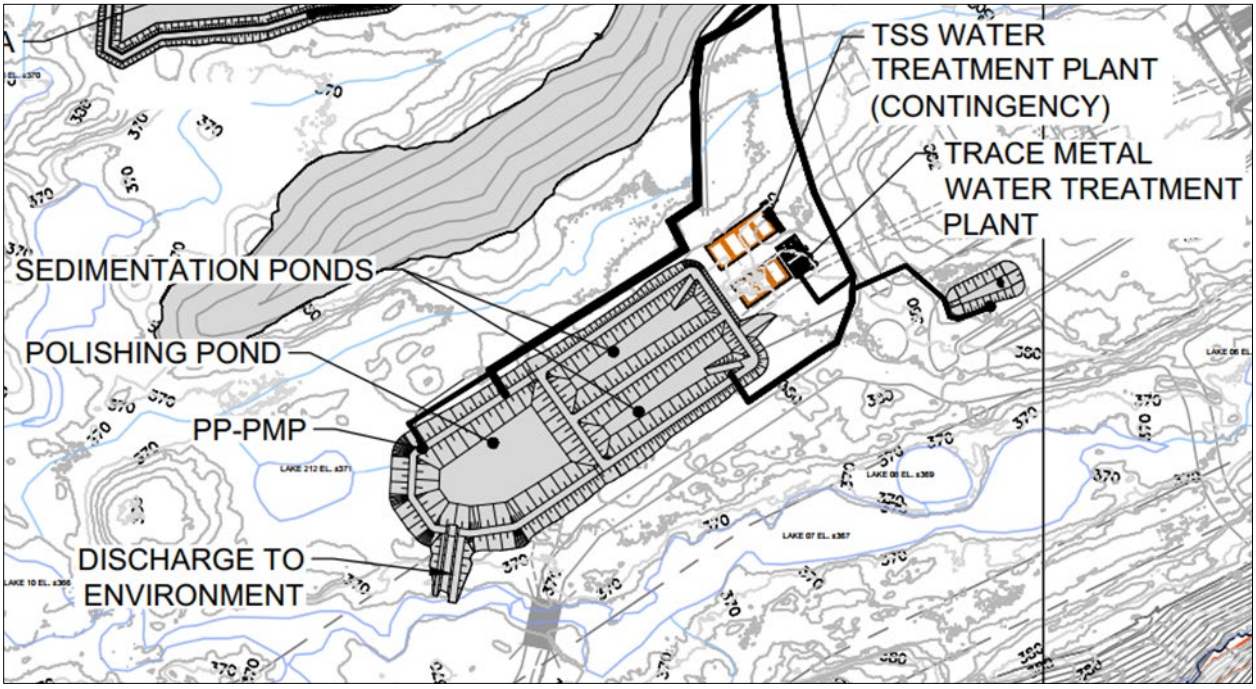
The treated effluent will be discharged into a polishing pond via an above-ground pipe, ensuring a single discharge point during operations.

18.7.5 Water Treatment

18.7.5.1 General

A water treatment system is planned as part of this project to ensure compliance with the discharge criteria outlined in Directive 019 (MELCCFP, 2025) and the federal *Metal and Diamond Mining Effluent Regulations* (SOR/2002-222). In addition to the sedimentation ponds designed to remove total suspended solids (TSS) from dewatering and contact water from the industrial area, a physico-chemical treatment system is proposed to address trace metals present in water from the process plant and runoff from Stockpile 002. This treatment system will be located near the industrial area ponds. Figure 18.43 illustrates the location of the water treatment plant (WTP) within the general site layout.

Figure 18.43: Water Treatment Pond and Plants



Source: AtkinsRéalis, July 2025.
**Note: Not to scale.*

18.7.5.2 Water Treatment Plant Process – Contact Water with Trace Metals

The WTP is designed to operate using a physico-chemical process that includes:

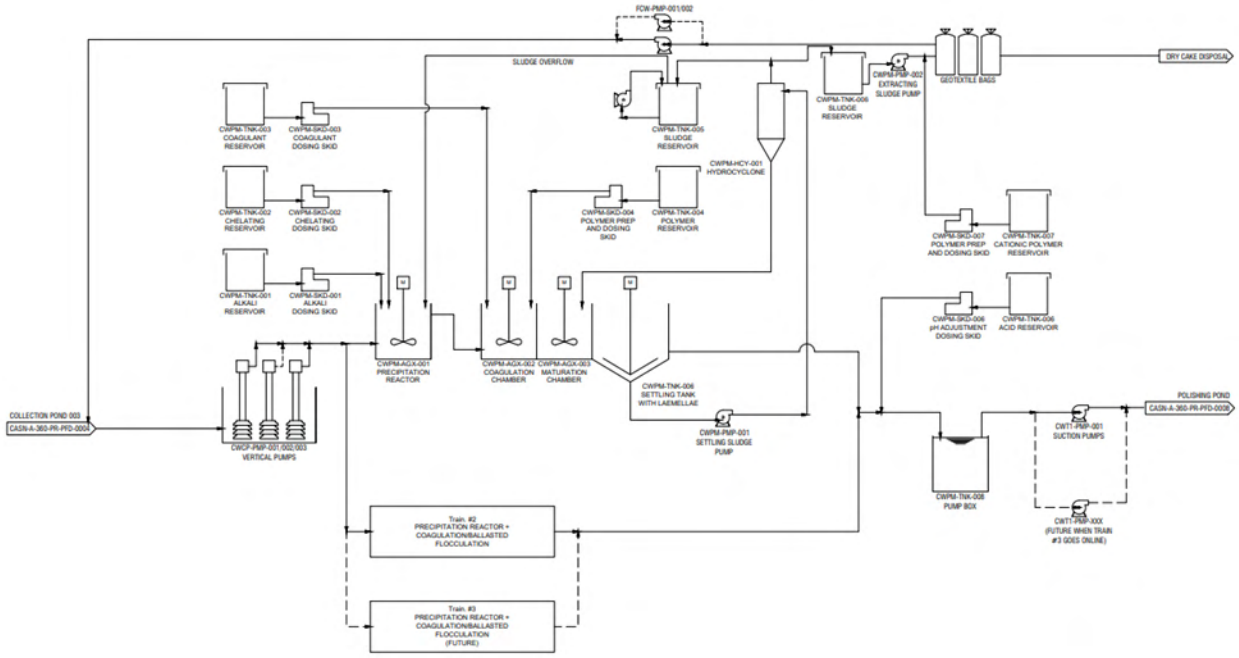
- Metal precipitation through pH adjustment and the use of chelating agents.
- Coagulation and ballast-assisted flocculation.
- High-rate clarification.

The treated water is discharged to a pump-box, which then feeds the polishing pond to complete the treatment process and ensure compliance with effluent regulations. Sludge generated during treatment is stored and dewatered using geosynthetic filtration bags. Water extracted from the geosynthetic bags is pumped back to the treatment.

The system is preassembled in fully mounted containers, which are shipped to the site. The concrete pad housing the geosynthetic filtration bags is divided into multiple bays, each containing one (1) filtration bag. At any given time, one (1) bay remains on standby while the others are active. When the bags in the active bay are full, clarified water is redirected to the standby bay. The collected sludge is further dewatered, and the resulting dry sludge is transferred to Stockpile 002.

Figure 18.44 illustrates the general flow diagram of the active treatment system for contact water contaminated with trace metals.

Figure 18.44: General Process Flow Diagram



Source: AtkinsRéalis, July 2025.

Based on the water balance, mine development, and deposition plan, the construction of the water treatment process will be phased over the life of mine. Table 18.11 outlines the flow rates for the two (2) identified periods. A total of three (3) treatment trains will be implemented, two (2) during Phase A and one (1) more during Phase B.

Table 18.11: WTP Design Capacity Through the Years

Years	WTP Flowrate Normal Condition (m^3/hr)	WTP Design Capacity (m^3/hr)
Phase A 2028-2030	800	2,000
Phase B 2030-2049	1,120	2,800

To treat the process water, one (1) treatment train is expected to operate continuously throughout the year, running 24 hours a day, 7 days a week. During periods of high flow, such as the spring freshet or intense summer rainfall, up to three (3) treatment trains may operate simultaneously. The treated effluent is discharged into the polishing pond before being released into the environment via stream CE15.

18.7.5.3 Contingency TSS Treatment

Water quality data were estimated through geochemical analysis of small rock samples. Additionally, some contact water, particularly from the former Lake 001 after dewatering, may contain high levels of TSS. To address this, a second water treatment system was designed as a contingency measure.

This secondary system is intended for implementation only once the mine is fully developed. Its inclusion serves two (2) purposes: to reserve space in the GA for potential future installation and to account for its cost in a future CAPEX plan.

This water treatment is designed to operate using a physico-chemical treatment process that includes:

- Coagulation and ballast-assisted flocculation.
- High-rate clarification.
- Sludge dewatering using geotubes.

Similar to the trace metal water treatment system, the treated water will be directed to the Polishing Pond using a pump box. The design and operational philosophy mirror that of the trace metal treatment system, utilizing pre-mounted containerized units. As the sludge produced consists exclusively of flocculated TSS, it is planned to be disposed of at the overburden pile.

18.7.5.4 Water Treatment Plant – Opportunity

Preliminary results from the geochemical characterization of the ore indicate that arsenic could leach out into the process water. Since this is the only source of contaminated water that would be produced on a year-round basis, it necessitates a fully winterized water treatment system. However, there is a potential to

eliminate the need for winter treatment through further geochemical analysis and by implementing source control measures. This would simplify operations and allow for the removal of most infrastructure required to prevent freezing, such as lightweight buildings or heated enclosures.

Furthermore, the current location of the water treatment plant does not permit gravity-based discharge from the treatment facility to the polishing pond. However, further optimization of the overall layout of the water management and treatment infrastructure could enable gravity discharge to the polishing pond, thereby eliminating the need for a pump station.

18.8 Electrical Distribution and Telecom

18.8.1 Off-Site Transmission System

The process plant and supporting infrastructure will be powered by Hydro-Québec's 315 kV overhead transmission system originating from the Tilly substation, located near the LG-4 Hydro-Québec Dam. A dedicated 315 kV interconnection point has been established to provide electrical service to the main Project site.

An off-site 315/120 kV substation, rated at 50 MVA, will be constructed approximately 2 km from the existing 735/315 kV Tilly substation. This facility will step down the transmission voltage and supply 120 kV power to the mine site via a new 54 km, 120 kV overhead transmission line. Wherever feasible, the transmission line routing will follow existing roadways to reduce environmental impact and simplify construction logistics. The transmission route is illustrated in Figure 18.45.

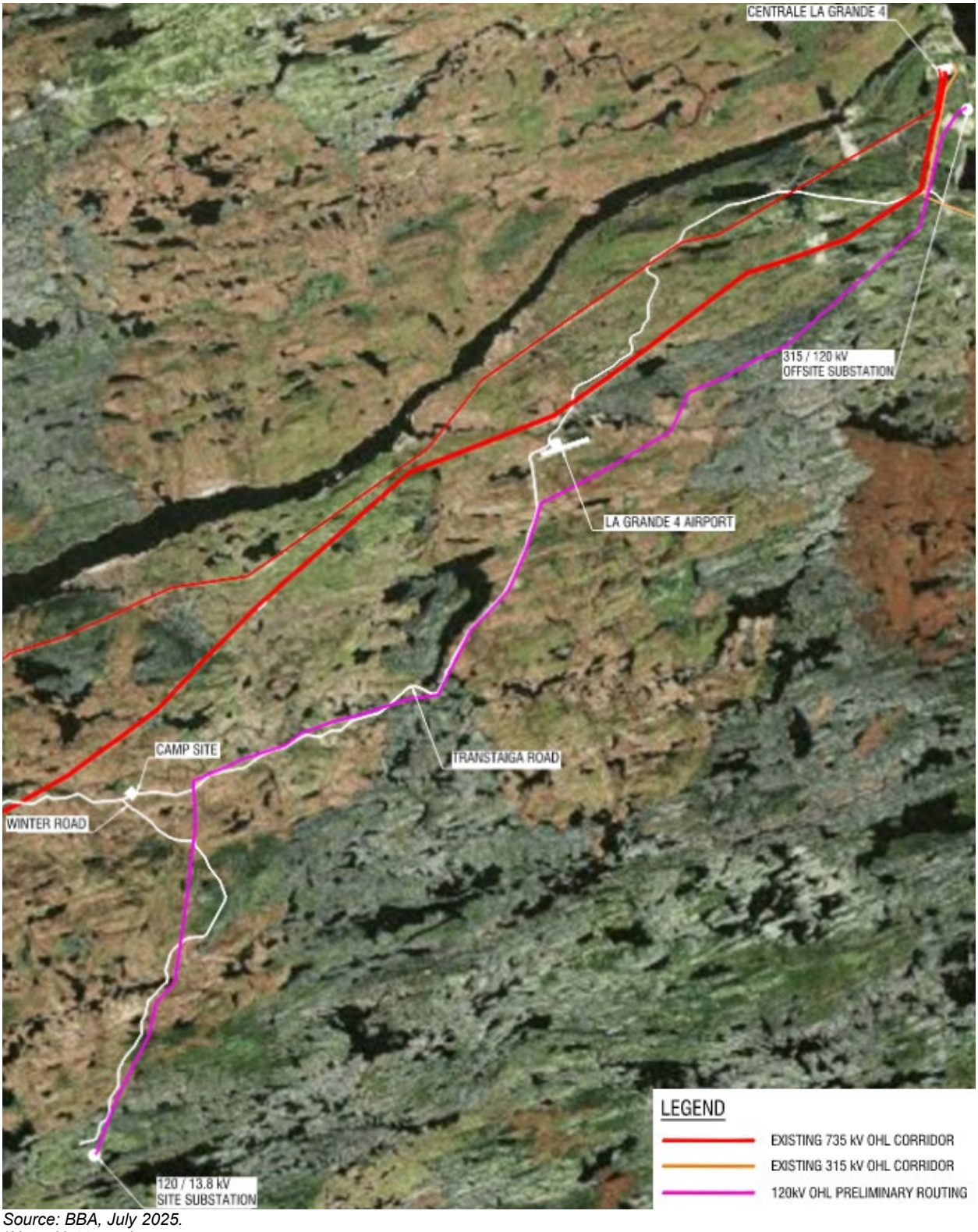
The 120 kV line will be designed with a minimum right-of-way (ROW) width of 38 m. Near-angle structures, or points of intersection (P.I.), larger ROW widths will be considered depending on the structure type and line angle. Preliminary P.I. positions have been identified, with these locations using guyed structures, some of which will serve as anti-cascade points. These positions require particular attention to factors such as positioning, soil type, and the presence of wetlands. Preliminary spotting of intermediate structures has also been completed, with validation required for all spans between P.I. positions. Structures for road, river, and line crossings will require the same level of attention as P.I. structures due to permitting and constructability considerations.

The Project's power allocation request with Hydro-Québec is in the approval process with the Ministry of Economy, Innovation, and Energy. Upon confirmation of the allocation, the power supply arrangement will be validated with Hydro-Québec to ensure compatibility with the Project's electrical design. The site's final

power demand will be confirmed in the next Project phase to match the available capacity of the Hydro-Québec transmission system.

Environmental impact assessments, detailed routing studies, and land acquisition planning will be advanced in parallel with the technical design to support permitting and construction readiness.

Figure 18.45: Transmission Route



Source: BBA, July 2025.
**Note: Not to scale.*

18.8.2 On-Site Distribution

18.8.2.1 Site Main Substation

The Site Main Substation, shown in Figure 18.46, receives the 120 kV incoming transmission line and steps it down to 13.8 kV for plant distribution. It is enclosed by a security fence and served by a maintenance access road. The facility includes high-voltage switchgear, step-down transformers, grounding equipment, reactive power compensation, and an elevated indoor electrical room for control and auxiliary systems.

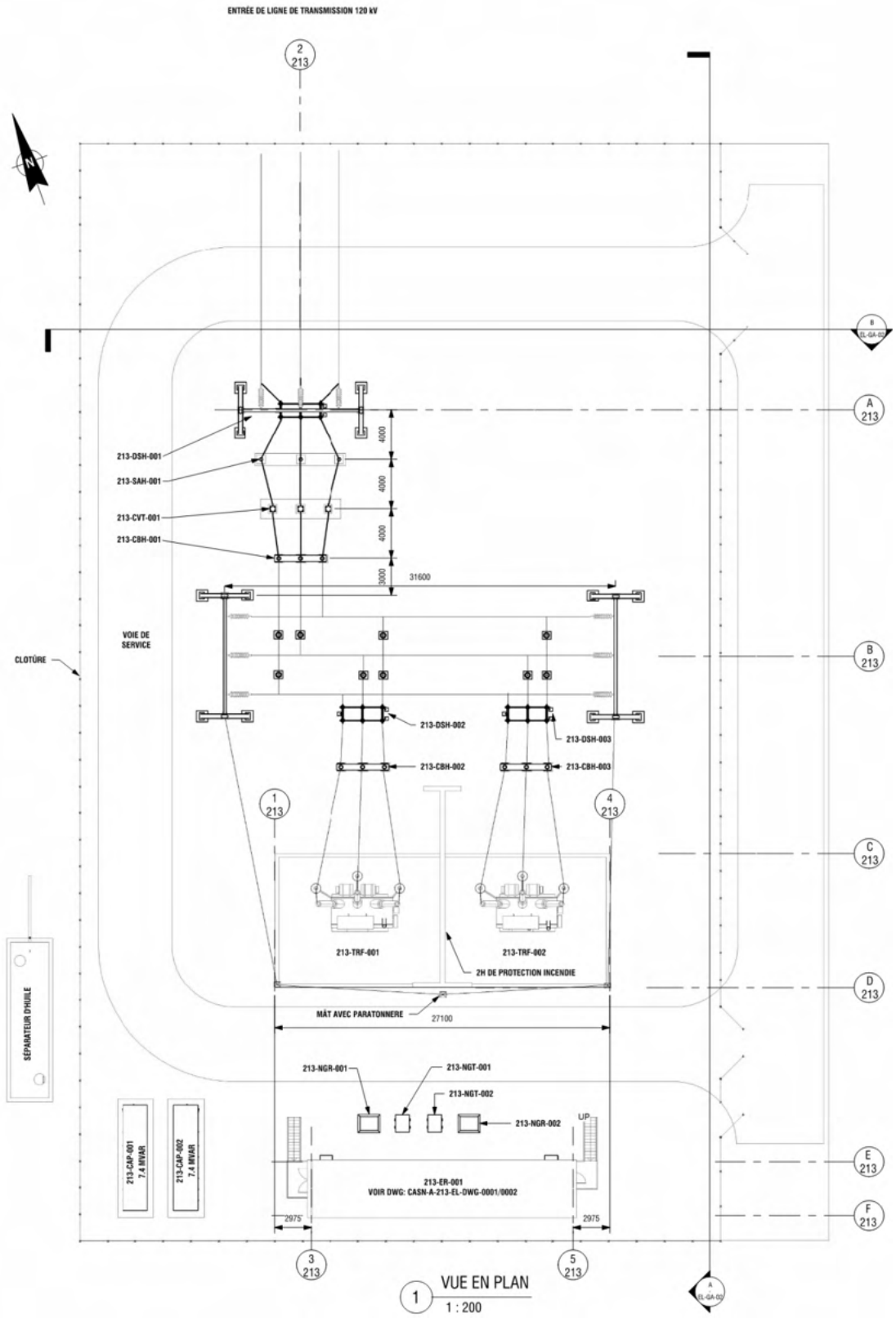
The high-voltage section on the north side receives the 120 kV line through disconnect switches, instrument transformers, and circuit breakers. Two (2) parallel 50/60 MVA step-down transformers provide full redundancy, with each unit capable of carrying the site load independently. The transformers are separated by a two-hour fire-rated wall and equipped with a dedicated oil separator for environmental protection.

Reactive power compensation is supplied by two (2) 7.4 MVAR capacitor banks positioned near the southern perimeter. Neutral grounding transformers and resistors limit fault current and maintain a stable system neutral, improving voltage balance and enhancing protection performance.

The electrical room houses the main control, protection, and distribution systems, including 13.8 kV switchgear with arc-flash venting, control and protection panels, and telecommunications cabinets for communication with the utility and plant systems.

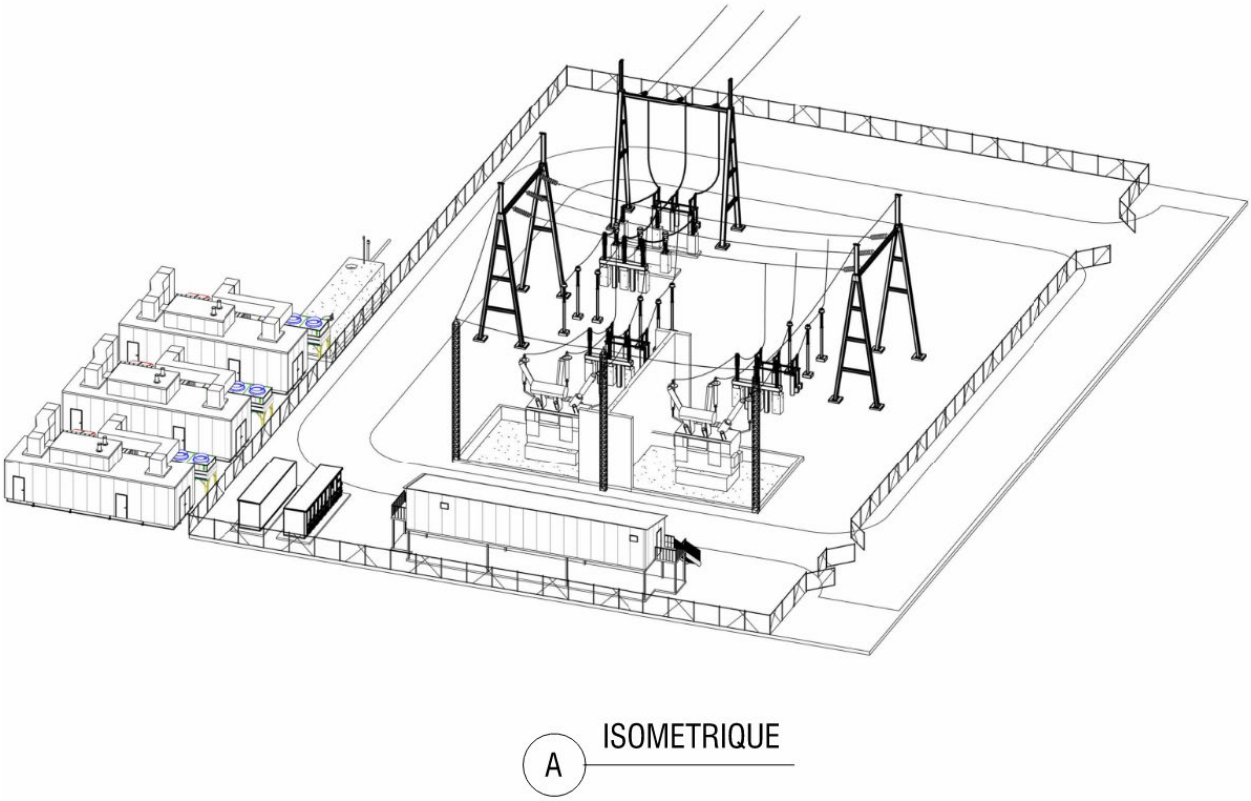
In the event of a grid power failure, a load-shedding scheme will be implemented to maintain power to essential systems. These critical loads will be supplied by two (2) standby diesel generators rated at 2 MW each, operating at 13.8 kV. To further improve reliability, a third 2 MW backup generator will also be installed, providing additional redundancy. This standby generation system ensures that all essential systems remain energized during an outage. The generators will be installed adjacent to the main substation area and directly connected to the main electrical room, as shown in Figure 18.47.

Figure 18.46: Site Main Substation



Source: GMS Feasibility

Figure 18.47: Main Electrical Room



Source: GMS Feasibility
**Note: Not to scale.*

18.8.2.2 Site Power Demand and Load Profile

The site's electrical demand progresses in distinct phases, aligned with the staged development of infrastructure, utilities, mining operations, and processing facilities, as illustrated in the load profile in Figure 18.48. In early 2028, initial demand is minimal — below 1 MW — reflecting early works and limited operational activity. Over the following months, demand rises steadily as mining support facilities, infrastructure systems, and utilities are energized, surpassing 5 MW in late 2029 and exceeding 10 MW in early 2030.

From 2030 to 2032, load growth accelerates sharply with the full deployment of large mining equipment and the commissioning of process plant areas, crushing facilities, and supporting utilities. This period marks the fastest expansion, with demand reaching the full operational plateau of approximately 34.55 MW in mid-2032. During the steady-production phase (2032–2048), process facilities account for a substantial portion of total site demand, alongside mining operations and plant utilities. A detailed sector-by-sector load breakdown is provided in Table 18.12.

As production begins to wind down after 2048, the site's power demand tapers to around 27 MW by 2050, followed by a sharp drop during decommissioning. By 2052, power requirements approach zero as major equipment and facilities are taken out of service.

Figure 18.48: Load Profile

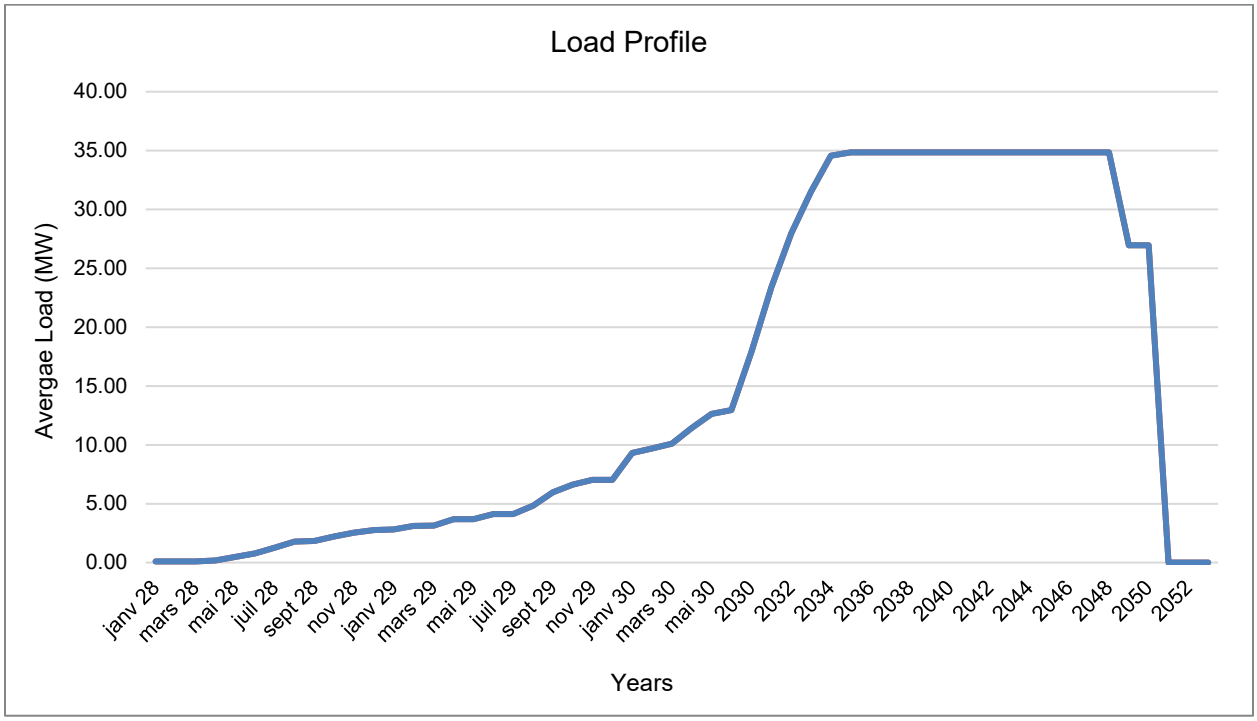


Table 18.12: Electrical Loads by Sector

WBS	Description	Annual Power Demand at Peak Production (MW)
100	Infrastructure	2.78
200	Power & Electrical	0.38
300	Water & Tailings Management	2.13
400	Surface Operation	0.00
520	Mining - U/G Mine Surface Services Infrastructure	6.02
540	Mining - U/G Mine Services	4.27
550	Mining - U/G Mine Equipment	0.34
580	Mining - OP Mine Equipment	0.27
590	Paste Preparation Plant	3.49
600	Process Plant - Process (Phase 1)	7.93
600	Process Plant - Process (Phase 2)	5.29

WBS	Description	Annual Power Demand at Peak Production (MW)
680	Process Plant - Paste Material Preparation	1.65
700	Construction Indirect	0.67
	Total Average Power (MW)	34.55

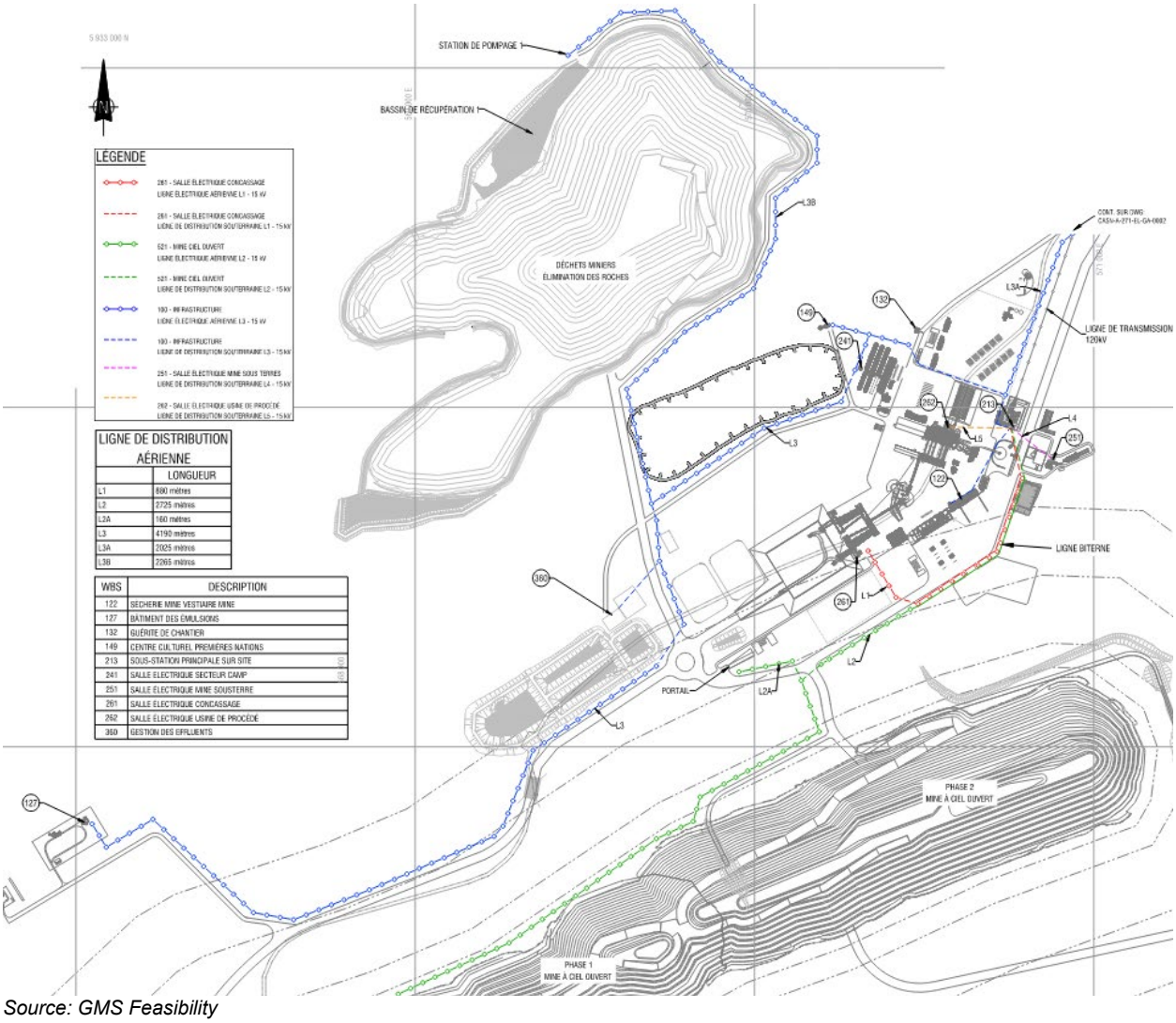
18.8.2.3 13.8 kV Site Power Distribution and Electrical Rooms

On-site medium-voltage distribution is provided at 13.8 kV through a combination of overhead and buried lines. The overhead network delivers power from the main substation to various operational areas across the site, following the routes indicated in Figure 18.49. Where required for physical constraints or operational safety, the network transitions to buried 13.8 kV cables. This mixed configuration ensures reliability in high-traffic or sensitive zones.

At each point of use, voltage is stepped down from 13.8 kV to 4.16 kV, 600 V, or low voltage, using oil-filled distribution transformers, matching the supply to the requirements of each facility — from large process equipment to general building services.

The electrical infrastructure includes several prefabricated electrical rooms strategically located to serve key load centers — including the main substation, crushing area, Dense Media Separation (DMS) plant, paste material preparation area, paste plant, fuel bay, and camp areas. Additional electrical rooms are integrated into the structural layouts of other site buildings as required. Each electrical room houses switchgear, control and protection equipment, and low-voltage distribution systems necessary to distribute power within its respective service zone.

Figure 18.49: Overhead Network



Source: GMS Feasibility
**Note: Not to scale.*

18.8.2.4 Underground Power Distribution

The underground mine power distribution system is supplied at 13.8 kV from the surface via two (2) independent feeders: one routed through the main ramp and the other through the ventilation shaft. This dual-feed arrangement provides operational flexibility and redundancy, ensuring reliable power to underground operations.

From these entry points, the 13.8 kV feeders supply a network of mobile substations positioned throughout the workings. Each mobile substation is equipped with a 2 MVA, 13.8/0.6 kV dry-type transformer, primary switchgear rated at 600 A, and a 600 V distribution panel for local load supply.

The mobility of these substations allows them to be relocated as mining activities advance, keeping power distribution points close to active work areas. This reduces cable runs, minimizes voltage drop, and improves operational efficiency. The network employs a combination of radial and loop configurations, allowing for sectionalizing and reconfiguration in case of maintenance or fault isolation.

At the point of use, power is stepped down to the required voltage levels — including 600 V for mining equipment and lower voltages for lighting, controls, and communications. The underground network also incorporates feed points for battery-electric vehicle (BEV) charging stations, paste fill systems, pumping installations, and ventilation fans.

18.8.3 Telecommunication

The telecommunication and industrial IT infrastructure part of this study is aligned with the design of a modern mine with all the services required for an Industry 4.0 style operation, such as short interval control, predictive maintenance, Ventilation-On-Demand (VOD), on-site teleoperation, and a local integrated operation centre.

The equipment, accessories, and installation services to supply the following systems / services throughout the site facilities have been considered in the study estimate, including redundant fibre-optic backbone, a private 4G/5G LTE system covering the mine site, redundant satellite WAN links (Starlink), wired and wireless networking, physical security system (video surveillance and access control), cybersecurity systems, high-performance hyperconverged servers, office / operation workstations, laptops and tablets, as well as PoC (PTT-over-Cellular, and Wi-Fi) as a more modern type of two-way radio using rugged smartphones.

18.9 Spodumene Concentrate Transportation

18.9.1 Road Transportation

The mine site is located along the Trans-Taiga Highway, approximately 844 km from the Matagami transfer yard. This route includes 290 km on the east–west Trans-Taiga Road and 554 km on the north–south Billy-Diamond Highway. Under optimal conditions, a one-way trip takes roughly 11 hours. The speed limit is set to 80 km/h on the unpaved Trans-Taiga Road and 100 km/h on the Billy-Diamond Highway.

Figure 18.50: Road Transportation



Source: BBA PEA Report 2024.

Transportation of concentrate will be contracted to a local service provider. At nominal plant capacity, daily output will total 2,191 tonnes of concentrate to be hauled. The Société de Développement de la Baie-James (SDBJ) imposes a Gross Vehicle Weight Rating (GVWR) limit of 68,750 kg on the Billy-Diamond Highway, allowing a net ore load of 45-47 tonnes per trip. However, the Project assumes the

usage of trucks with a 75-t capacity, requiring special annual permits from the Authorities Having Jurisdiction. Special conditions will also have to be respected during thaw periods.

These 75-t vehicles will consist of a four-axle tractor and a five-axle bi-train trailer, equipped with a trailer cover and side-discharge system. Accounting for 2,191 tonnes per day plus an additional 5% capacity margin, approximately 30 truck trips will be required daily between the mine and Matagami from 2032 to 2048.

Operations will run 24 hours a day, 7 days a week, year-round. A fleet of 80 trucks will be maintained to provide a 25% buffer for maintenance, road restrictions, or environmental disruptions.

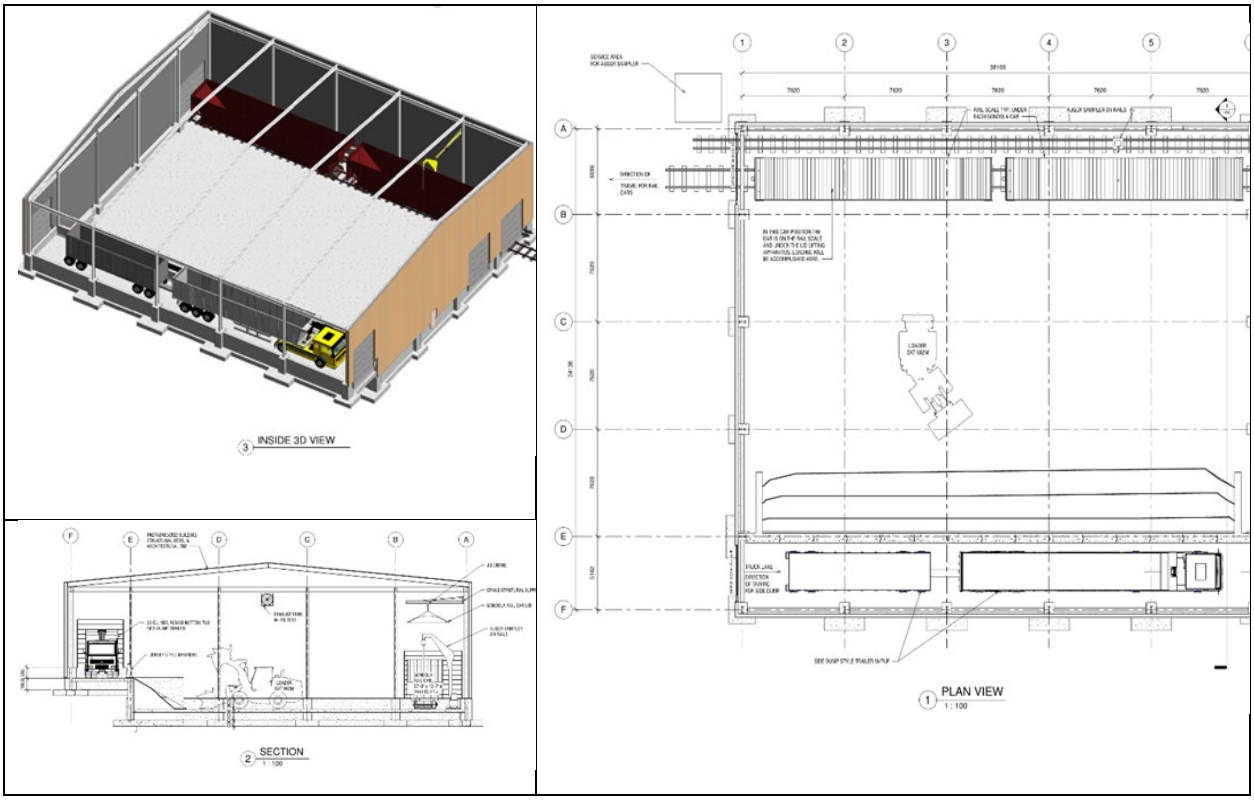
18.9.2 Matagami Transshipment Centre (MTC)

The Canadian rail network offers access to deep-sea ports for international lithium refining facilities. At the MTC, concentrate will arrive in side-dump haul trucks, be unloaded onto a concrete floor within a new prefabricated building and exit on the opposite side. From there, front-end loaders will transfer the material into railcars.

A new spur line will be constructed to accommodate covered railcars, with Canadian National Railway (CN) responsible for delivery and pick-up. The Matagami Transshipment Centre will construct, operate, and maintain the facility.

Conceptual design of the building is shown in Figure 18.51.

Figure 18.51: MTC Conceptual Design



Source: GMS database,
**Note: Not to scale.*

18.9.3 Railroad and Railcars

From Matagami, trains will transport spodumene concentrate to the maritime terminal at Grande-Anse, La Baie, Québec, for vessel loading. Railcars will be 93 tonnes capacity gondolas, 52’ in length, weighing 31.4 tonnes each (including covers), with a maximum payload of 98,430 kg. The rail corridor’s capacity from Matagami to Grande-Anse is 130 tonnes.

Annual shipments will total approximately 8,602 railcars—about 172 cars per week, assuming a 50-week operational year. The rail distance is 1,075 km, and a full round trip, including loading, CN transport, transit time, and unloading, is estimated at 18 days. To sustain operations, 620 leased railcars will be required.

18.9.4 Train Unloading Station – Grande-Anse Maritime Terminal, La Baie

At Grande-Anse, railcars will be unloaded using a straddle excavator. Up to 15,000 tonnes of spodumene concentrate will be stored in a warehouse before being transferred via front-end loader onto a conveyor system and moved to the dock area. Vessels will be loaded using a telescopic conveyor.

To avoid cross-contamination, the warehouse and conveyor will be cleaned prior to use for spodumene concentrate. Terminal handling and stevedoring will be provided by a third-party operator.

Under PMET Resources' agreement with the client, product ownership transfers once loaded onto the vessel.

19. MARKET STUDIES AND CONTRACTS

The Company commissioned Benchmark Intelligence (Benchmark Minerals, 2025) to assess the lithium spodumene market for the Shaakichiuwaanaan Project's Feasibility Study (FS). The following summary reviews market conditions and projections, including global supply-demand trends, pricing, technology, and applications. It covers global supply-demand dynamics, price trends, technological developments, and downstream applications for lithium.

NI 43-101 technical reports from other Projects, Benchmark Intelligence guidance and Consensus forecast are used to support the yearly price assumptions for 5.5% Li_2O spodumene concentrate FOB Australia.

All figures in Chapter 19 were provided by Benchmark Intelligence.

19.1 Lithium Market Overview

Lithium has firmly established itself as a critical enabler of the global energy transition, underpinned by the accelerating adoption of electric vehicles (EVs) and the rapid build-out of battery energy storage systems (BESS). Demand for lithium has grown at double-digit rates over the past decade and is forecast to continue expanding strongly well into the 2030s. This demand growth is both broad-based and resilient, supported by government policies, OEM electrification mandates, and the structural integration of batteries into transport and energy infrastructure.

On the supply side, the industry has achieved significant growth, with new hard rock and brine Projects entering the market across Australia, South America, and Africa. However, development timelines remain long, capital requirements substantial, and execution risks high. These factors constrain the ability of supply to scale at the unprecedented pace required to meet demand.

The result is a lithium market characterized by persistent tightness. Surpluses are expected to remain minimal in the near term, leaving the market highly vulnerable to disruption, before shifting into sustained deficits after 2030. Together, these dynamics underpin a constructive long-term outlook for lithium pricing, reinforcing the commodity’s strategic importance and value capture potential.

19.2 Lithium Supply / Demand

19.2.1 Supply

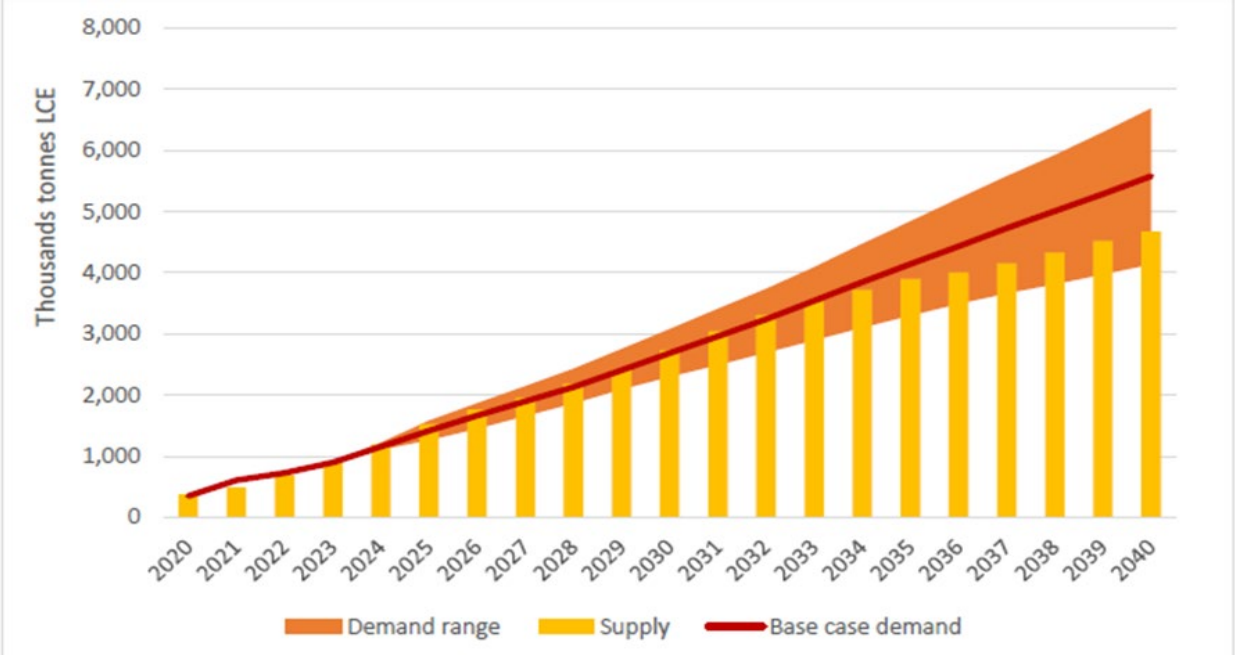
Global lithium supply has grown significantly in recent years, but the market remains highly concentrated and increasingly strained by structural bottlenecks. In 2024, global mine production reached approximately 1.2 million tonnes LCE, a sharp rise from just 310,000 tonnes in 2015. Supply forecast is 2.7 Mt LCE by 2030, 3.9 Mt LCE by 2035 and 4.7 Mt LCE by 2040. Demand trajectories imply the industry will need to deliver close to 2.7 Mt LCE by 2030 — representing nearly a two-and-a-half-fold increase in required output within only six (6) years, equivalent to a 15% compound annual growth rate (CAGR) from 2024 to 2030. Looking further out, production would need to climb to ~4.1 million tonnes by 2035 (a 12% CAGR from 2024, ~3.6 x higher) and ~5.6 million tonnes by 2040 (a 10% CAGR from 2024, ~4.8 x higher). See Table 19.1 and Figure 19.1 for details.

These figures highlight the extraordinary challenge the sector faces: sustaining double-digit annualized supply growth over decades while navigating permitting, financing, ESG scrutiny, and technical hurdles. Few other commodities have ever been required to expand at such a pace, underscoring the likelihood of persistent supply bottlenecks.

Table 19.1: Lithium Supply Growth Requirements

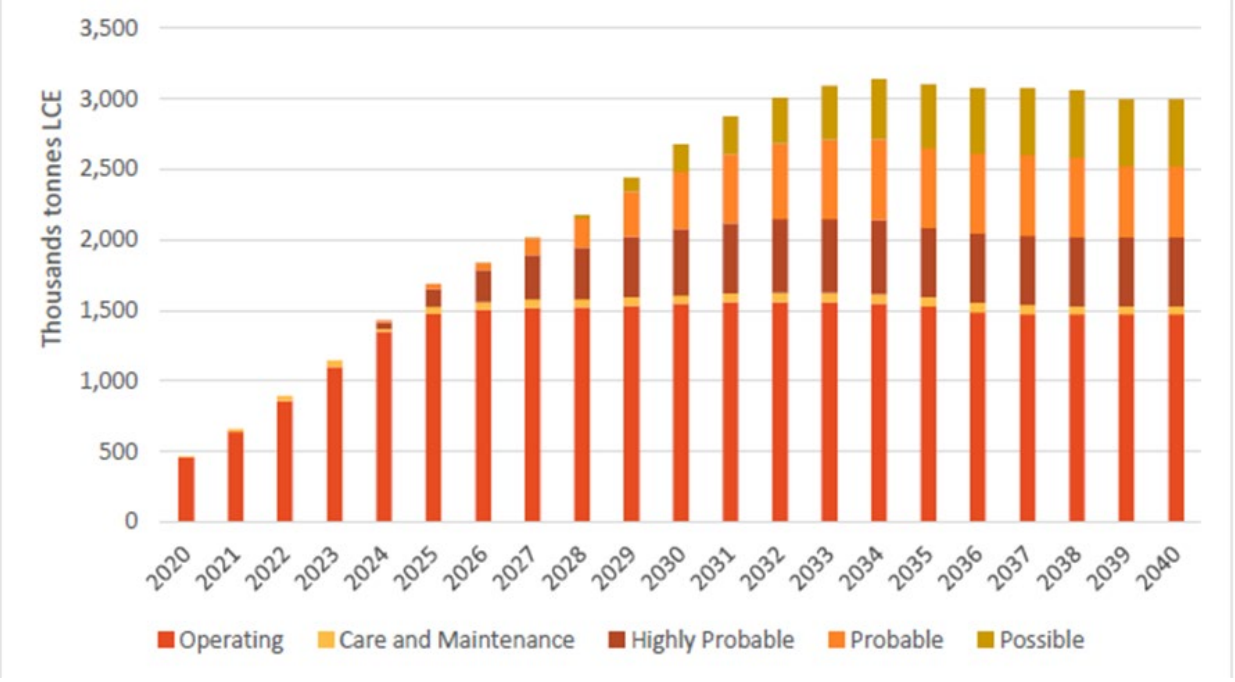
Year	Supply (Mt LCE)	Demand (Mt LCE)	Demand CAGR vs 2024	Demand Multiple vs 2024
2024	~1.2	~1.1	–	1.0x
2030	~2.7	~2.7	~16%	~2.4x
2035	~3.9	~4.2	~13%	~3.7x
2040	~4.7	~5.6	~10%	~4.9x

Figure 19.1: Lithium Total Non-Risked Supply and Demand Scenarios



The new supply is expected to come mainly from new mining Projects and partly from recycling. Omitting "possible" (Figure 19.2) supply would result in supply peaking at approximately 2.7 Mt LCE in 2033, based on the forecast shown.

Figure 19.2: Risk-Weighted Lithium Mine Supply by Operating Status



19.2.2 Demand

Lithium demand is powered by one of the most compelling growth stories of the century: the global electrification of transport and energy systems. Sales of electric vehicles (EVs) are expected to surpass 21 million units in 2025, representing nearly a quarter of total vehicle sales, and are forecast to rise to 42% by 2030 and 70% by 2040 (Figure 19.3). Each EV requires a significant amount of lithium — often 30-60 kg LCE per vehicle, depending on battery chemistry — cementing lithium's role as the cornerstone of mobility's future.

Beyond mobility, the battery energy storage system (BESS) sector is accelerating even faster. According to Benchmark Intelligence / Rho Motion, annual BESS installations grew more than 60% year-on-year in 2024, growing at more than twice the pace of EVs in terms of percentage and are forecast to maintain strong growth through the 2030s. Energy storage is emerging as a critical enabler of renewable integration, with multi-GWh deployments now common across North America, Europe, and Asia. This represents an entirely new demand vertical for lithium, expanding the market beyond transport and into grid-scale infrastructure.

Recent updates show that electric vehicle (EV) sales continue to provide strong structural demand for lithium. According to Benchmark Intelligence / Rho Motion's July 2025 update, global EV sales reached 8.7 million units year-to-date, representing a 26% year-on-year increase. This robust growth trajectory suggests that EV adoption is running slightly ahead of Benchmark Intelligence's forecast for lithium demand in 2025, highlighting that the fundamental demand story for lithium remains both resilient and accelerating.

Battery Energy Storage Systems (BESS) are also a rapidly expanding source of lithium demand. Benchmark Intelligence / Rho Motion's August 2025 assessment shows that 107 GWh of new grid BESS capacity was installed globally in the first seven (7) months of the year, 38% higher than the same period in 2024 (Figure 19.4). Meanwhile, Project announcements for the year-to-date have reached 475 GWh, reflecting a 45% increase compared to last year. About 91 GWh entered the Project pipeline in July 2025 only, with the 91 GWh representing almost 43% of total BESS capacity in the full year of 2024.

With China leading grid-scale deployments and the US also commissioning multi-gigawatt Projects, the momentum in the BESS sector reinforces the expectation that energy storage will remain a critical growth vector for lithium demand over the coming decade.

By 2030, global lithium demand is projected to more than double from current levels, with Benchmark Intelligence forecasting an inflection point where consumption overtakes feasible supply. Crucially, demand

growth is not only strong but also resilient: OEM electrification mandates, government policy support, and cost-down trajectories in batteries create powerful structural drivers that are unlikely to reverse.

Even considering ongoing efforts to optimize chemistries (such as LFP adoption and recycling), the scale of growth ensures that lithium demand will continue to rise steeply. Recycling, while important in the long term, is not expected to materially offset virgin lithium requirements until well into the 2040s due to the lag in EV fleet turnover.

Figure 19.3: Lithium Demand by End-Use Applications

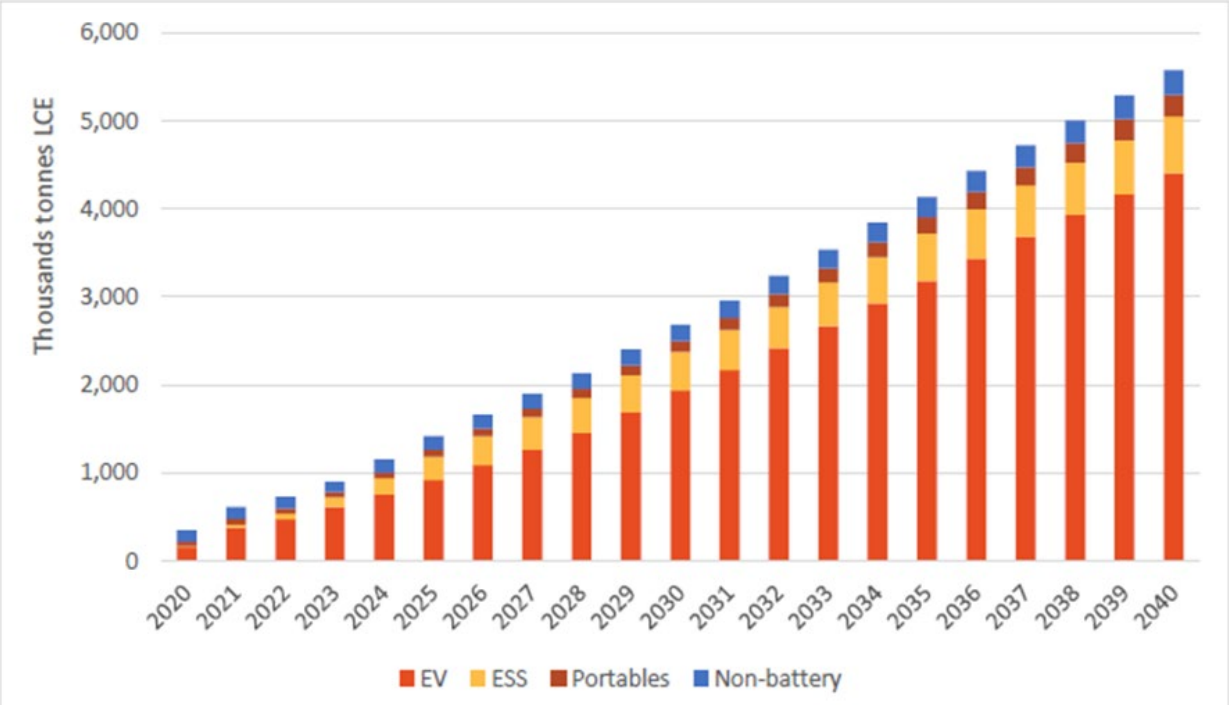
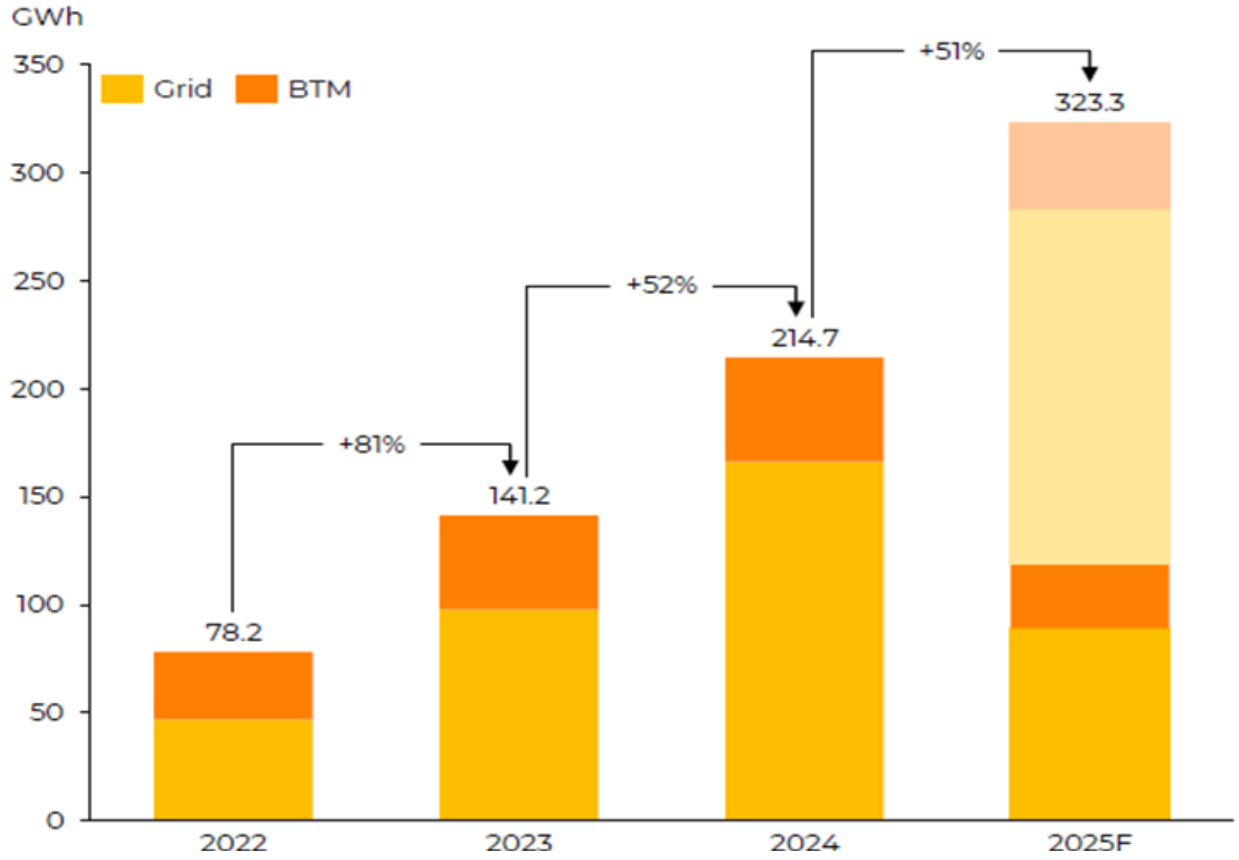


Figure 19.4: BESS Installed Capacity Outlook by Storage Type, New Additions



19.3 Market Balance

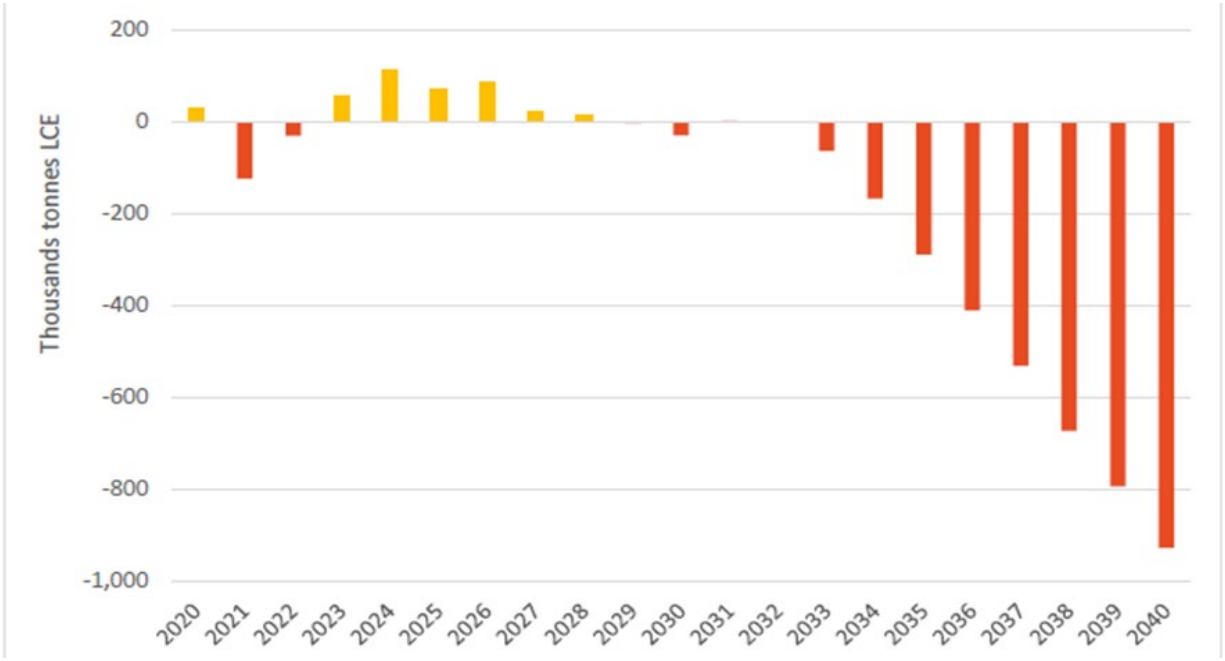
The lithium market is poised to remain exceptionally tight across the remainder of this decade and into the 2030s. Benchmark Intelligence forecasts indicate that from now until 2030, global supply is expected to keep pace with demand only marginally, with average annual surpluses of less than 5% of total demand. In a market exceeding one (1) million tonnes per year in 2024 and reaching approximately 2.7 Mt in 2030, this margin is effectively negligible and within the normal variance of Project performance. Such a narrow “cushion” means that any delay in commissioning, ramp-up underperformance, or disruption to existing production could quickly tip the balance into deficit, driving higher prices in response.

By 2030, the balance is projected to turn negative, with an annual shortfall of approximately 290,000 tonnes LCE by 2035 (Figure 19.5). This marks a structural turning point: even with the significant wave of Projects currently in development, the industry is not expected to deliver sufficient capacity on time to meet rapidly expanding demand from both electric vehicles and stationary storage. Historically, lithium Project development has been characterized by frequent delays due to permitting challenges, financing

hurdles, ESG requirements, and the technical complexity of scaling new extraction technologies. These risks make the already narrow forecast surpluses, in the years preceding 2030, even less secure.

Looking beyond 2030, the deficit becomes embedded and more pronounced. Between 2030 and 2035, Benchmark Intelligence Projects average annual deficits of around 91,000 tonnes LCE, and between 2035 and 2040, Benchmark Intelligence Projects average annual deficits of around 600,000 tonnes LCE, highlighting the structural inability of the industry to scale at the rate required by global electrification. This gap is not only significant in absolute tonnage but also represents a persistent drag on market stability, with the potential to lock in higher average pricing as buyers compete for constrained volumes.

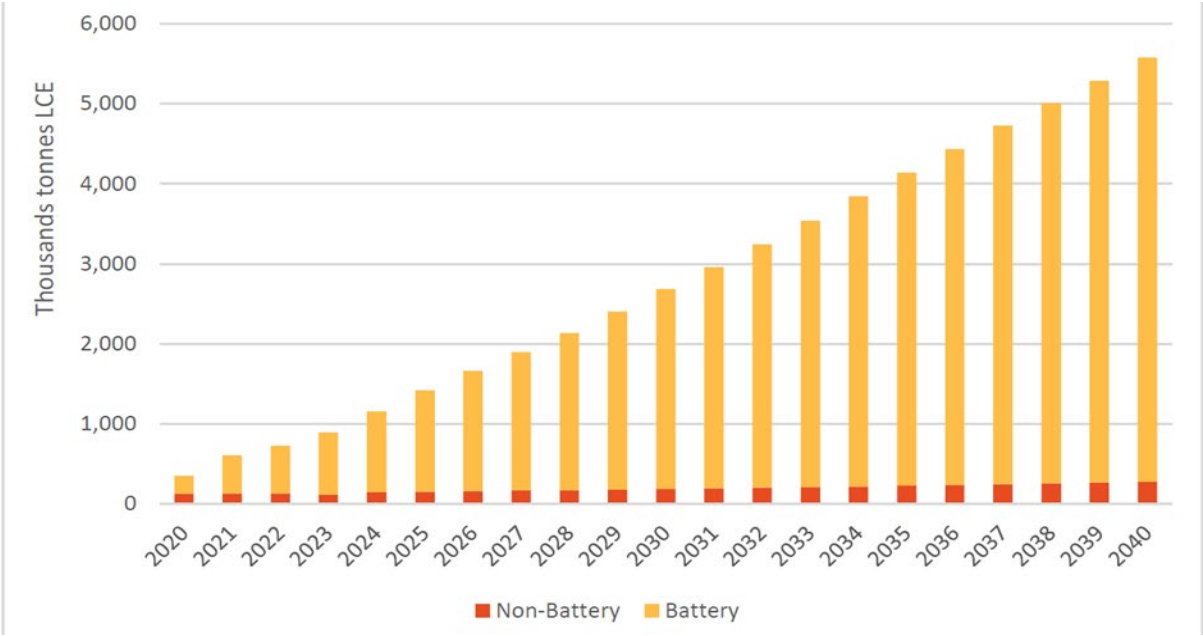
Figure 19.5: Global Market Balance



19.4 Lithium Demand Outlook

Refined lithium is primarily consumed in the form of lithium carbonate and is expected to account for around 72% over the forecast period, with the remaining share going to lithium hydroxide. The growth driver is the battery application (Figure 19.6).

Figure 19.6: Global Demand for Refined Lithium



19.5 Lithium Spodumene Concentrate Price Assumptions

19.5.1 Recent Spot Prices

Fastmarkets' assessment for spodumene concentrate (SC6%) CIF China was USD 800 – USD 950/t on August 13, 2025, a significant decrease from the peak in early 2023. This decline is attributed to softening downstream chemical prices and ample lithium raw material supply (Fastmarkets, 2024). The Shanghai Metals Market reported price for SC6% on August 13, 2025, is USD 926/t.

Table 19.2: Current Market Spot Price

	Source	Spodumene Concentrate (%)	Spodumene Price (USD/t)	Year	FOB / CIF China
Current Market Spot Price	Shanghai Metals Market (SMM), Spodumene Concentrate Index (CIF China)	6.0	926 (August 13, 2025)	2025	CIF China

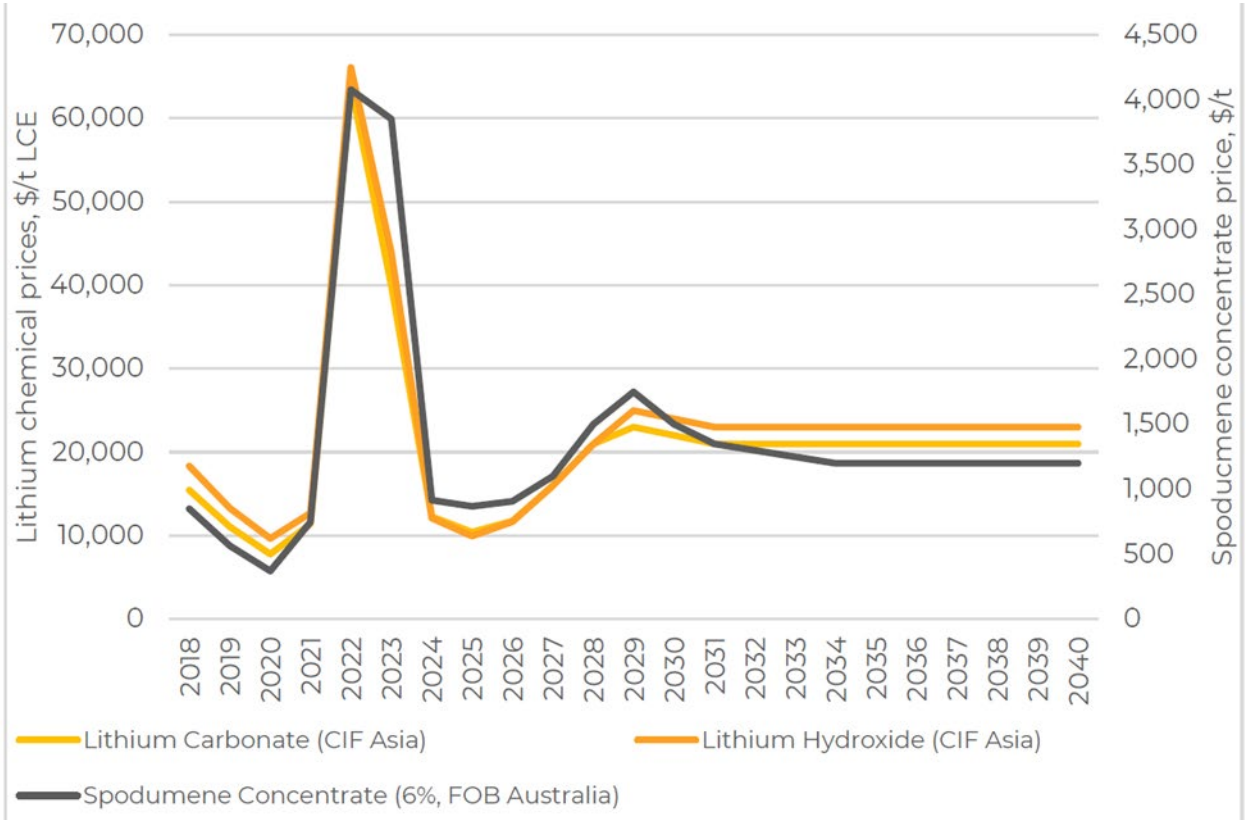
Source: Shanghai Metals Market (SMM). "Spodumene Concentrate Index (CIF China)." metal.com. Accessed 13 Aug. 2025. https://www.metal.com/Lithium/201906260003

19.5.2 Benchmark Intelligence

Benchmark Intelligence forecasts that the lithium carbonate price in the long term is expected to be at USD 21,000/t LCE. Considering the direct correlation between spodumene concentrate and lithium

carbonate, along with historical pricing for LCE and spodumene concentrate, an acceptable ratio between LCE price and spodumene concentrate price would be in the range of 15 to 18. Based on these ratios, an acceptable spodumene concentrate price would sit in the range of USD 1,167 to USD 1,400 for 6% Li_2O spodumene concentrate, FOB Australia. The forecast price assumption based on Benchmark Intelligence for the long term is USD 1,200 (Figure 19.7).

Figure 19.7: Lithium Price Forecasts, $/tonne, Real 2025



19.5.3 Recent NI 43-101 Reports & Press Releases

Several mining companies have recently published technical reports and press releases using a range of long-term spodumene price assumptions. Table 19.3 presents spodumene price references (in US dollars per metric tonne) in the jurisdiction of Canada, obtained from recent NI 43-101 and JORC Code (2012) technical reports, company announcements, and banking commodity reports.

Table 19.3: Jurisdiction of Canada Spodumene Price

Project / Company Name	Report Type / Source	Spodumene Concentrate (%)	Spodumene Price (USD/t)	Reference	Year	FOB / CIF China
Moblan Lithium Project	Definitive Feasibility Study	6%	1,190	NI 43-101 Feasibility Study Report for the Moblan Lithium Project, Eeyou Istchee James Bay Territory, Québec, Canada	2024	FOB (Québec)
Lake Superior Lithium	Preliminary Economic Assessment	6%	1,360	Avalon Advanced Materials, "Avalon Completes PEA: Post-Tax C$ 4.1 Billion NPV (8%) and 48% IRR at its Thunder Bay Lithium Processing Facility, ON," Sept 3, 2024	2024	Purchase Price
Adina Project	Scoping Study	5.5%	1,375	Winsome Resources: Scoping Study delivers a capital-efficient solution for North American lithium production, Sept 17, 2024	2024	FOB (Québec)
PAK Lithium Project	Feasibility Study	6%	1,475	Frontier Lithium Inc., News Release: "FRONTIER LITHIUM'S FEASIBILITY STUDY CONFIRMS...", May 28, 2025	2025	FOB (Thunder Bay)
NAL Expansion Project	Scoping Study	6%	1,392	Sayona News Release: “NAL EXPANSION SCOPING STUDY CONFIRMS LOWER COSTS AND STRONG RETURNS”, September 15, 2025	2025	
Lithium Universe (Bécancour Refinery)	Definitive Feasibility Study	6%	1,170	Lithium Universe Limited, "Becancour Lithium Refinery Definitive Feasibility Study," Feb 17, 2025	2025	CIF (Bécancour Port)
CIBC Commodities Consensus Forecast Summary (Long-Term)	Analyst Report	6%	1,264	CIBC Global Mining Group Analyst Consensus Commodity Price Forecasts Aug 1, 2025	2025	CIF China

Source: Avalon Advanced Materials. "Avalon Completes PEA: Post-Tax C$4.1 Billion NPV (8%) and 48% IRR at its Thunder Bay Lithium Processing Facility, ON." News Release, 3 Sept. 2024. https://avalonadvancedmaterials.com/wp-content/uploads/2024/09/Avalon-Advanced-Materials-Inc_PEA.pdf; Frontier Lithium Inc. "FRONTIER LITHIUM'S FEASIBILITY STUDY CONFIRMS CA$932M NET PRESENT VALUE AND LOW-COST CONCENTRATE SUPPLY FOR PAK LITHIUM PROJECT." News Release, 28 May 2025. https://www.newswire.ca/news-releases/frontier-lithium-s-feasibility-study-confirms-ca-932m-net-present-value-and-low-cost-concentrate-supply-for-pak-lithium-Project-828623138.html; Winsome Resources. "Adina Project." Winsome Resources Limited. Accessed 13 Aug. 2025. https://winsomeresources.com.au/our-assets/adina/; Lithium Universe Limited. "Becancour Lithium Refinery Definitive Feasibility Study." News Release, 17 Feb. 2025. https://www.listcorp.com/asx/lu7/lithium-universe-limited/news/becancour-lithium-refinery-definitive-feasibility-study-3152006.html.

**Notes:*

1) Variability in Reporting: Not all companies explicitly disclose the exact spodumene price assumptions used in their studies. Some might use a range, a long-term average, or focus on different lithium products (e.g., lithium hydroxide).

2) Spodumene concentrate prices vary depending on the location reference. CIF China prices, which include shipping and insurance to China, are typically higher than FOB prices from other locations. Buyers should factor in these additional costs when comparing and making purchasing decisions.

3) Market Volatility: Spodumene prices are subject to significant volatility due to market forces, including supply and demand dynamics, geopolitical events, and technological advancements. The prices listed here are snapshots in time and may not reflect current market conditions.

4) Concentrate Grade: The price of spodumene concentrate can vary depending on the lithium oxide (Li_2O) content. Table 19.1 above indicates the Li_2O grade for each reference.

19.5.4 Commodities Consensus

PMET has compiled its own commodities consensus that includes the CIBC commodities consensus, with the addition of eight (8) other analysts from banks. The forecast for long-term price includes 15 data ranging from USD 1,050/t to USD 1,750/t for 6% Li_2O spodumene concentrate FOB Australia. When excluding the highest forecast long-term price and the lowest forecast long-term price, the range is between USD 1,100/t and USD 1,650/t, with an average of USD 1,332/t for 6% Li_2O spodumene concentrate FOB Australia.

19.6 Conclusion

The lithium spodumene market is characterized by dynamic pricing, shaped by a confluence of multifaceted factors. While price volatility is expected to persist, a price in the range of USD 1,100–USD 1,350 per tonne for 5.5% spodumene concentrate FOB Australia is reasonable given recent technical reports, consensus forecast, Benchmark Intelligence report, and the backdrop of the Q2-Q3 2025 market conditions. While there is a reasonable expectation to be able to extract the non-Li metals in the mining of CV5, namely tantalum and caesium, these metals are not included in the financial analysis of the FS for CV5, and as such, the FS report does not consider them at this stage.

Based on the spodumene concentrate price assessment, it is recommended that PMET use a fixed long-term price of USD 1,221/tonne (SC5.5%, FOB Australia basis) (Table 19.4), equivalent to USD 1,332 for 6% Li_2O spodumene concentrate.

Table 19.4: Spodumene Concentrate Price Recommendation

Product	Price (USD/t)
Spodumene concentrate at 5.5% Li_2O	1,221

19.7 Contracts

In January 2025, the Company entered into a binding offtake term sheet with Volkswagen's 100%-owned vertically integrated battery manufacturer, PowerCo SE (PowerCo), to supply 100,000 tonnes of spodumene concentrate (SC5.5 target) per year over a 10-year term (PMET, 2024) (PMET, 2025).

The offtake is expected to supply PowerCo's cell production activities in Europe and North America, including its battery cell factory in St. Thomas, Canada. St. Thomas is intended to become PowerCo's largest cell factory with a production capacity of up to 90 GWh, enough to produce over

one (1) million electric vehicles a year. Building a North American EV supply chain, from raw materials supply to the construction and operation of Gigafactories, is part of PowerCo's mission.

Concurrently, the Company also entered into a non-binding Memorandum of Understanding (the MoU) to establish an ongoing strategic relationship between PowerCo and the Company to jointly explore and collaborate on shared strategic objectives, including opportunities for the future development of the Shaakichiuwaanaan Project centred around establishing a cost-competitive, sustainable and ESG-compliant battery supply chain that will attract government support and incentives and the potential development of a chemical conversion facility.

As of the effective date of this Technical Report, no contracts have been executed with vendors to purchase equipment or contractors to construct the Project.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Setting

This chapter describes the main components of the current physical, biological, and human environments in the Project area. Data in this section is based on existing inventories and public databases. When necessary, further studies have been undertaken or are underway to provide the level of information required for the Environmental and Social Impact Assessment (ESIA).

20.1.1 Physical Environment

20.1.1.1 Physiography

Natural provinces are broad territories whose recognition is based on physiographic contrasts expressed by the nature and configuration of bedrock, relief, hydrography, and surface deposits. According to Québec's ecological reference framework, the Shaakichiuwaanaan Project area is in the natural province of the Grande Rivière Low Hills (Li, Ducruc, Côté, Bellavance, & Poisson, 2019). This natural province has low terrain with an undulating plain, which is succeeded inland by low hills. The geological bedrock consists mainly of tonalite and gneiss. Thin glacial deposits interspersed with rocky outcrops cover the hills (MELCCFP, 2023). The study area lies at an altitude between 260 m and 350 m above sea level (Séguin, Gagnon, Lepage, & Thomassin, 2022).

20.1.1.2 Climate

The climate of the Grande Rivière Low Hills natural province, in which the study area is located, is characterized by cool summers and very cold winters (Environment Canada, 2023). The mean annual temperature is approximately -4°C, with a summer mean of 8.5°C and a winter mean of -16.5°C. Average annual precipitation ranges from less than 600 mm to 800 mm.

20.1.1.3 Noise

Ambient noise studies were carried out by WSP in November 2024 at three (3) sensitive receptors (campsites) identified by local land users along the Trans-Taiga Road, as well as at four (4) additional measuring points surrounding the Project site. The noise measurements were undertaken to describe ambient noise before planned mining activities, and to determine the noise criteria for each sensitive receptor according to land uses and applicable regulations. Noise criteria depend on the Project phase

(construction vs mining activities) and on the receptor location (sensitive zone vs non-sensitive zone) (WSP, 2025).

Results indicate that the main noise contribution at Point KM283 comes from road traffic on the Trans-Taiga Road, with an average daytime noise level of 35.3 dBA. At Point KM258, a generator operating to power the camp during the night produced a relatively high noise level, with the lowest nighttime hourly noise level at 42.1 dBA.

Moving away from the Trans-Taiga Road, ambient sound levels are relatively very low, corresponding to those typically measured in early winter in remote areas far from dwellings and human activity, when wind speeds are low. There, the average noise levels ranged from 19.6 to 26.9 dBA.

The modelling of noise and vibration levels that could potentially result from planned construction and mining activities is ongoing. If required, site-specific mitigation measures will be proposed to comply with all applicable noise regulations.

20.1.1.4 Air Quality

Ambient air quality levels, defined as pre-existing concentrations of contaminants, must be considered in the development of an atmospheric model of Project activities. In order to verify compliance with standards and criteria, the sum of the initial concentration and the modelled concentrations must be demonstrated to be below the regulatory standard or criterion for each substance.

To determine ambient levels of contaminants, different approaches can be considered. Initial concentrations can be determined from:

- Measurements taken on site to obtain actual values.
- Measurements taken by Federal, Provincial or Municipal monitoring networks, such as Environment and Climate Change Canada's (ECCC) National Air Pollution Surveillance Network (NAPSN) or the MELCCFP's Quebec Air Quality Monitoring Network (RSQAQ).
- Generic initial concentrations defined by the MELCCFP.

In the case of the Shaakichiuwaanaan Project, the existing RNSPA station in Radisson is located at a latitude equivalent to that of the Project, approximately 250 km from the study area. This station samples total particulate matter, fine particulate matter PM2.5, ozone and certain metals in total particulate matter.

The concentration results available at this station will be analyzed and considered. However, some of the pollutants likely to be emitted by the Project are not sampled at this station. In this context, following the recommendations of the MELCCFP, the initial concentrations prescribed for northern projects (PN) in the document *Guide d'instructions – Préparation et réalisation d'une modélisation de la dispersion des émissions atmosphériques – Projets miniers* (MELCCFP, 2025) will also be considered. A comparison between the concentrations for PN and the data from the Radisson station will also be carried out.

Finally, for all other substances not available for northern projects, the initial concentrations used are generic initial concentrations taken from the document Quebec Air Quality Standards and Criteria [NCQQA v9] (MELCCFP, 2025).

Specifically for crystalline silica, no initial concentration is specified for northern projects. During the assessment of a similar mining project located in a remote area, the MELCCFP proposed the use of an initial annual concentration of crystalline silica of 0.01 µg/m^3 (COMEX, 2022) rather than the default value of 0.04 µg/m^3 in NCQQA v8. In this context, this initial concentration will be used for the analysis of the results.

20.1.1.5 Hydrology

The Project site is in the James and Hudson Bays drainage area (Region 09), the largest in Québec in terms of surface area.

Within this broad region that drains the inland waters of northwestern Québec to the west, the Project is in the heart of the Grande Rivière watershed (level 1). The Grande Rivière (or Rivière La Grande) runs for over 800 km, flowing from east to west, from its source on the Québec-Labrador border to its mouth in James Bay. Its watershed covers a vast area of over 200,000 km^2. In its natural state, this watershed covered an area of 97,643 km^2, but the construction of the La Grande Complex Hydroelectric Project has resulted in a major diversion of water into this watershed, more than doubling the area. Just south of the proposed Shaakichiuwaanaan Project area, the Pontois River watershed (level 2) drains an area of 19,142 km^2 westwards, and then joins the Grande Rivière watershed, some 50 km from the Project site. The boundary between these two (2) watersheds is located approximately 1 km south of the proposed open pit.

Locally, the Shaakichiuwaanaan Project site is scattered with numerous bodies of water, some of which conflict with the planned Project infrastructure. This is particularly true in the case of Lake 001, which will be impacted by the establishment of the CV5 Open Pit. Every effort has been made to eliminate the need

for waste rock deposition into waterways, and the proposed footprint has successfully avoided infringement on fish habitat by waste rock.

Hydrological inventories and studies for the Project were initiated in 2022 and are continuing in the 2025 field season to provide the level of information required for the ESIA. For the 11 delineated watersheds of the Shaakichiuwaanaan Project area, preliminary results indicate that the annual low flow rate of Q10.7 is estimated between 0.001 and 10.3 m^3/s, while the annual low flow rate of Q2.7 is estimated between 0.002 and 10.7 m^3/s. The high-flow rate for a 2-year return period varies from 0.35 to 194 m^3/s (WSP Canada Inc., 2025).

20.1.1.6 Surface Water and Sediment Quality

Surface water quality and sediment quality sampling campaigns were conducted in 2022, 2023, 2024 and 2025 to characterize the quality of waterbodies and water courses within the study area. Throughout the field program, 27 waterbodies and two (2) water courses were sampled for surface water, and 13 waterbodies and two (2) water courses were sampled for sediments. Because the 2025 campaign is still ongoing, the latest results presented in this section date from 2024 (Niigaan, 2025).

All the lakes studied had similar characteristics, i.e. oligotrophic lakes, with slightly acidic pH (average 6), clear, nutrient-poor waters and low biological productivity. Lake substrates are predominantly composed of fine sediments such as silt and clay, sometimes mixed with sand. The two (2) streams presented the same characteristics. In terms of depth, most of the lakes studied are shallow, with depths varying between 0.5 and 6 metres, except for the deeper Lake 001 (approx. 18 m). At the -twenty-seven (27) stations sampled for surface water, exceedances were mainly observed for pH, dissolved oxygen, and metals such as aluminum, iron, copper, zinc, lead and to a lesser extent silver. At the 12 stations sampled for sediments, exceedances were observed for metals such as arsenic, cadmium, chromium, copper, mercury, lead and zinc. The natural occurrence of these metals in sediments is primarily linked to the regional geology, characterized by pegmatitic deposits and the associated mineralogical assemblages. Their mobilization can be influenced by local geochemical conditions, including pH, oxygen, redox potential, and the presence of iron and manganese oxides, which can absorb and later release certain metals. Mercury, though often present in low concentrations, can accumulate in sediments through natural processes, including organic matter interactions and microbial activity. The concentrations obtained are also generally within the geochemical background of the site studied (Choinière & Beaumier, 1997).

During the operational phase, the Project will have a surplus of water (mine dewatering, process water and runoff), and management will include discharge into the environment. As described in Section 18.2.3,

mitigation measures, including a water treatment plant, will be implemented to maintain the surface water quality.

20.1.1.7 Hydrogeology

The hydrogeological baseline was developed using publicly available information as well as field investigations carried out in 2023 and 2024 (Bouazza, Mahdi, & Fournier, 2024) (BBA, 2025). All data was compiled to determine the different hydrostratigraphic units, assess the overall hydraulic properties of the units, the groundwater flow and groundwater quality.

The site is surrounded by water bodies such as lakes and marshy areas, and the topography is generally flat with small hills, such as drumlins and rock outcrops.

Two (2) hydrostratigraphic units are present at the site: the overburden unit and the bedrock unit. The overburden unit at the study area is discontinuous and varies in thickness from 0 m to 48 m, but generally around 8 m when present, mainly comprised of undifferentiated till or drumlins. The overburden is water-bearing, but due to its discontinuous nature, it is not considered a large-scale aquifer. The second hydrostratigraphic unit is the bedrock.

The groundwater level in the overburden is shallow, generally between 0.1 and 2.7 m below ground surface (bgs), while in the bedrock, it ranges from 0.1 to 3.1 m bgs, with two (2) locations reaching more than 7 m bgs. Groundwater flow in the overburden and shallow bedrock follows topography, with groundwater generally flowing from topographic highs, which act as recharge areas, towards the lakes and wetlands. In general, groundwater flows towards Lake 001 and Lake 027.

The background groundwater quality was assessed in samples collected during the two (2) sampling campaigns in the monitoring wells located at the site. The results show background concentrations lower than the EDC and RES criteria of the MELCCFP for the majority of the parameters analyzed, except for arsenic, manganese, molybdenum, antimony, ammoniacal nitrogen and sulphides exceeding EDC and copper exceeding RES.

The geological and hydrogeological data available for the study area were used to develop three-dimensional groundwater numerical models. One of the models was used to assess the dewatering rates for the OP and UG operation pits and to estimate the extent of drawdown over the study area. The second model was developed to assess the efficacy of the seepage control measures proposed to prevent any significant degradation of the groundwater quality due to any potential seepage from Stockpile 002, as

the material stored in this stockpile was assessed as being a metal leaching risk, particularly arsenic, and potentially acid-generating (Vision Geochemistry, 2025).

The mitigation measures evaluated during this study were the installation of a geomembrane below Stockpile 002 during the mine operation, followed by the installation of an impermeable cover following mine closure. The results of the solute transport model suggest that the mitigation system is effective, with solute concentrations reaching the environmental receptors being below the applicable criteria after 200 years.

20.1.2 Biological Components

20.1.2.1 Flora

The Project area is in the boreal vegetation zone, more specifically in the open boreal forest subzone, located between latitudes 52 N and 55 N (Gouvernement du Québec, 2023a) This subzone is characterized by low-density forests of black spruce with lichen beds. This area is also located in the Western subdomain of the spruce-lichen stands bioclimatic domain. Compared with the Eastern subdomain, the Western subdomain is characterized by a dry continental climate, where fire is more frequent, and relief is less pronounced.

The study area is in the Eastmain and Sakami rivers ecological region (No. 7d). The regional landscape unit is the *Lac de la Corvette* (No. 720), and the ecological district is Lac Nochet Low Hills (No. 720-008). According to the ecoforestry map available online (Gouvernement du Québec, 2023a) vegetation in the study area generally consists of lichen and moss spruce stands, with areas of lichen barrens. Since fires have been burning in the area for the past 20 years, several burned areas are present. Wetlands are mainly represented by minerotrophic and ombrotrophic bogs.

Surveys were completed by WSP in August 2023, June 2024 and August 2024 to characterize the vegetation and wetlands and validate the presence of special-status species (WSP Canada Inc., 2025) (see Section 20.1.2.9). A total of 428 characterization plots or validation points have been completed throughout a 114.25 km² inventory area. The inventory area is mainly occupied by terrestrial environments, equivalent to 74% of its total surface area, while wetlands occupy 10%. Other environments cover 16.5% of this area and are largely dominated by bodies of water. The vast majority of wooded stands in the study area are aged or intermediate-aged coniferous stands. Wooded and open ombrotrophic peatlands and treed swamps are the most abundant wetland types in the inventory area. Overall, the area has undergone very little anthropogenic disturbance, and natural disturbances are associated with the forest fires of recent decades.

20.1.2.2 Fish and Fish Habitat

The James Bay region is characterized by countless bodies of water and watercourses that are home to a variety of aquatic life (CRNTBJ, 2010). Generally, fish habitat in Northern Quebec is of very high quality due to low levels of human disturbance. Fish populations in this region generally grow more slowly, live longer, and have a lower density associated with lower ecosystem productivity compared to southern regions.

Fishing activities were carried out by Niigaan in 2022, 2023, 2024 and 2025 in the study area (Niigaan, 2025). Various fishing gear was used in addition to the environmental DNA collection method. A total of 10 species were captured or detected. These were northern pike, burbot (*Lota lota*), lake chub (*Couesius plumbus*), round whitefish (*Prosopium cylindraceum*), white sucker (*Catostomus commersonii*), longnose sucker (*Catostomus catostomus*), pearl dace (*Margariscus margarita*), eastern blacknose dace (*Rhinichthys atratulus*), brook trout, and lake trout. None of these species has any special status (see Section 20.1.2.9).

During surveys, particular attention was paid to the delineation of fish habitat in all permanent and intermittent water bodies and streams. Under Canada's *Fisheries Act* and Québec's *Act Respecting the Conservation and Development of Wildlife*, any infrastructure encroaching on fish habitat and resulting in habitat loss must be compensated.

As mentioned in Section 20.1.2.2, the current development plan encroaches on fish habitat in certain areas, especially at pit CV5, where diking of Lake 001 will result in direct fish habitat loss. Subsequent design phases of the Project will aim to minimize infrastructure encroachments into fish habitat.

20.1.2.3 Herpetofauna

According to the literature consulted, the study area is likely to be frequented by 10 species of herpetofauna (6 anurans, 3 urodeles, and 1 squamate), presented in Table 20.1; None of these species has any special status.

Surveys were carried out in the spring and summer of 2023 and 2024 to describe the herpetofauna communities present in the study area and provide the level of information required for the ESIA (Niigaan, 2025). A total of five (5) anuran species were detected in both years: northern spring peeper, American toad, mink frog, wood frog and northern leopard frog (in order of relative importance). No salamanders or reptiles were detected.

Table 20.1: List of Herpetofauna Species Likely to Frequent the Study Area

Order	English Name	Scientific Name
Anurans	American toad	Anaxyrus americanus
	Wood frog	Lithobates sylvaticus
	Mink frog	Lithobates septentrionalis
	Northern leopard frog	Lithobates pipiens
	Green frog	Lithobates clamitans
	Northern spring peeper	Pseudacris crucifer
Urodeles	Northern two-lined salamander	Eurycea bislineata
	Blue-spotted salamander	Ambystoma laterale
	Yellow-spotted salamander	Ambystoma maculatum
Squamates	Common garter snake	Thamnophis sirtalis

Source: (AARQ, 2023) & (CRNTBJ, 2010).

20.1.2.4 Avian Fauna

According to CRNTBJ (2010), 238 bird species have been recorded in the Project region. Of these, many are potentially present in the study area. Bird inventories were carried out in the study area using a variety of techniques during the breeding, spring migration, autumn and winter periods between 2023 and 2025, including 18 sound recorders, 522 listening point stations and 24 crepuscular stations. Data from 2025 have not been analyzed yet; therefore, the results presented in this section date from 2024 (Niigaan, 2025).

Throughout the field surveys, various inventory methods enabled recording of 109 bird species, including 9 species at risk (see Section 20.1.2.9) and 94 species of migratory birds as defined by the Migratory Birds Convention Act of 1994.

During the breeding season, the average breeding pair density for the entire study site is 35.25 pairs/10 ha. Wooded peat bogs and mixed coniferous forests have above-average pair densities. Regenerating habitats show the lowest density of all habitat groups. In addition, wooded peat bogs show the greatest diversity of species observed, while regenerating habitats and mixed forests show the lowest diversity.

The vast majority of species observed during spring and autumn migration were also detected during the breeding season, suggesting that the study area's habitats play a minor staging role for more northerly breeding birds. Indeed, most of the species passing through the study area on migration also stay to breed.

20.1.2.5 Bats

Acoustic surveys (16 automated recording stations) were carried out in the study area in 2023, 2024 and 2025 in order to cover the entire active season for chiropterans, i.e. spring dispersal / migration (April 1 to June 14), summer breeding (half-period 1: June 15 to July 19, half-period 2: July 20 to August 14) and autumn migration / swarming (half-period 1: August 15 to September 4, half-period 2: September 5 to 25). Recordings from 2025 have not been analyzed yet; therefore, the results presented in this section date from 2024 (WSP, 2025).

The inventories carried out confirmed the presence of five (5) species and two (2) groups of species: migratory and resident. Bats of the genus *Myotis*, the northern myotis (*Myotis septentrionalis*) and the big brown bat (*Eptesicus fuscus*) are considered resident species. The silver-haired bat (*Lasionycteris noctivagans*), the hoary bat (*Lasiurus cinereus*), and the eastern red bat (*Lasiurus borealis*) are considered migratory species. The majority of recordings collected during the various surveys came from hoary bats. Of these species, only the big brown bat does not have special status. The provincial and federal status of the various bat species is presented in Section 20.1.2.9.

The results of the acoustic surveys did not reveal any areas of concentrated chiropteran activity. Activity levels were low at all survey periods, and the results do not suggest the presence of maternity sites or migration corridors in the vicinity of any of the survey stations.

No critical habitat (maternity or hibernacula) was observed during the specific field investigations or suggested by the acoustic inventory results. However, the presence of maternity habitat in the survey area remains a possibility, given the snag densities observed on the site (CRNTBJ, 2010).

20.1.2.6 Small Mammals

Based on their ranges, the study area is likely to be frequented by 15 species of small mammals (Table 20.2). These include two (2) special status species, which are the rock vole (*Microtus chrotorrhinus*) and the southern bog lemming (*Synaptomys cooperi*). Both of these species are designated as threatened or vulnerable (Gouvernement du Québec, 2023c).

Field surveys for small mammals were undertaken in 2023, and 747 specimens were captured. The presence of eight (8) species was confirmed in the study area: southern red-backed vole, meadow vole, southern bog lemming, northern bog lemming, meadow jumping mouse, cinereous shrew, eastern heather vole (*Phenacomys ungava*) and mouse of the *Peromyscus* genus. Since some of the specimens have not

been identified based on dental or cranial criteria, the rock vole is still considered potentially present in the study area, although it was not surveyed as part of the 2023 inventory.

Table 20.2: List of Small Mammals Likely to Frequent the Study Area

Order	English Name	Scientific Name
Rodents	Southern red-backed vole	Myodes gapperi
	Meadow vole	Microtus pennsylvanicus
	Rock vole (1)	Microtus chrotorrhinus
	Western heather vole	Phenacomys intermedius
	Deer mouse	Peromyscus maniculatus
	Southern bog lemming (1)	Synaptomys cooperi
	Northern bog lemming	Synaptomys borealis
	Woodland jumping mouse (1)	Napaeozapus insignis
	Meadow jumping mouse	Zapus hudsonius
	Ungava lemming	Dicrostonyx hudsonius
Insectivores	Cinereous shrew	Sorex cinereus
	American water shrew	Sorex palustris
	Arctic shrew	Sorex arcticus
	Hoy's pigmy shrew	Sorex hoyi
	Star-nosed mole	Condylura cristata

Source: (Gouvernement du Québec, 2023c).
**Note: 1) The study area is located north of the known distribution range of these species. Species in bold have a conservation status (see Section 20.1.2.9).*

20.1.2.7 Small Fauna and Fur-bearing Animals

According to CRNTBJ, 25 species of small fauna and fur-bearing animals frequent the James Bay region (CRNTBJ, 2010). During surveys undertaken in March 2023, the presence of 10 species was confirmed. These are the red squirrel (*Tamiasciurus hudsonicus*), the northern flying squirrel (*Glaucomys sabrinus*), the snowshoe hare (*Lepus americanus*), the river otter (*Lontra canadensis*), the Canadian lynx (*Lynx canadensis*), the American marten (*Martes americana*), the black bear (*Ursus americanus*), the porcupine (*Erethizon dorsatum*), the red fox (*Vulpes vulpes*), and the American mink (*Neovison vison*). Tracks of weasel species and mustelid species were also observed. To date, no species of small fauna or furbearer with a special status, or evidence of the presence of such species, has been observed in the study area.

20.1.2.8 Large Fauna

The study area is likely to be frequented by moose (*Alces alces*), as well as two (2) caribou ecotypes, namely the migratory caribou (*Rangifer tarandus caribou*) and the boreal caribou (*Rangifer tarandus caribou*). To validate the presence of these species, a literature review and field survey were conducted.

According to information provided by the MELCCFP's *Direction de la gestion de la faune du Nord-du-Québec* office on caribou telemetry data for the period 2004 to 2024, no woodland caribou were found in the study area, which covers a 50-km radius applied to the Project centroid. This data corroborates the results of the survey conducted by Szor and Gingras (2020) and by Niigaan in 2023 for the purpose of the Shaakichiuwaanaan Project, during which no caribou were located within the study area. These results are also consistent with information gathered from tallymen who frequent the territory. Note that wildlife observations have been recorded and documented by the exploration team since May 2024. Throughout the period, 5 caribou sightings have been recorded, the majority of which were observed in groups of 5-30 individuals travelling along the road in winter or spring. One (1) individual caribou was observed travelling the shoreline of Lake 027 in June 2025.

Interpretation of Szor and Gingras (2020) habitat quality for caribou in the study area shows that, overall, the area is unsuitable for woodland caribou. Most of the study area around the proposed mine site is of average to poor quality. The large-scale forest fires that occurred in the area in 2023 impacted a significant portion of the available caribou habitat, particularly in one of the main areas frequented by migratory caribou in the study area, west of Lac de la Corvette.

For moose, a total of 27 individuals in 14 groups were counted during the January 2023 survey by Niigaan, corresponding to a very low abundance of 0.18 moose/10 km^2. Females accounted for 37% of the total, fawns for 22% and males for 41%. The presence of the species was also confirmed during a survey of trail networks undertaken in March 2023.

20.1.2.9 Species at Risk

Several species with a special status are likely to frequent the study area. Local wildlife species with status in Canada, as defined by the *Species at Risk Act* (SARA), and in Québec, as defined by the *Act Respecting Threatened or Vulnerable Species*, are presented in Table 20.3. According to the database of the *Centre de données sur le patrimoine naturel du Québec* (CDPNQ), available via the interactive online map, no such plant or wildlife species are present within a 50 km radius of the Project (CDPNQ, 2023). However, the presence of 14 species has been confirmed in the Project area throughout specific field inventories.

Regarding the special status for plant species, the “Potentiel” tool (Gouvernement du Québec, 2023d) was used to develop a preliminary list of threatened plant species potentially present in the Nord-du-Québec administrative region. This list contains 55 vascular plants potentially present in the study area, none of which were observed during the vegetation and wetland surveys carried out by WSP.

Table 20.3: Special Status Wildlife Species Potentially Present in the Project Area

Class	English Name	Scientific Name	Status		
			LEMVQ(1)	COSEWIC(2)	SARA(3)
Avian Fauna	Golden eagle	Aquila chrysaetos	V	–	–
	Harlequin duck	Histrionicus	V	SC	SC
	Red crossbill percna subspecies	Loxia curvirostra percna	–	T	T
	Common nighthawk	Chordeiles minor	LDTV	SC	SC
	Peregrine falcon anatum / tundrius	Falco peregrinus	V	–	–
	Barrow’s goldeneye	Bucephala islandica	V	SC	SC
	Short-eared owl	Asio flammeus	LDTV	SC	T
	Bank swallow	Riparia	–	T	T
	Olive-sided flycatcher	Contopus cooperi	LDTV	SC	SC
	Bald eagle	Haliaeetus leucocephalus	V	–	–
	Rusty blackbird	Euphagus carolinus	LDTV	SC	SC
	Yellow rail	Coturnicops noveboracensis	T	SC	SC
Bats	Little brown myotis	Myotis lucifugus	T	E	E
	Northern myotis	Myotis septentrionalis	T	E	E
	Hoary bat	Lasiurus cinereus	LDTV	E	–
	Eastern red bat	Lasiurus borealis	V	E	–
Small Mammals	Rock vale	Microtus chrotorrhinus	LDTV	–	–
	Southern bog lemming	Synaptomys cooperi	LDTV	–	–
Small Fauna and Fur Animals	Least weasel	Mustela nivalis	LDTV	–	–
	Wolverine	Gulo	T	SC	SC
Large Fauna	Boreal caribou	Rangifer tarandus caribou	T	T	T

Note: Status: **E: Endangered; **LDTV:** Likely to be designated threatened or vulnerable specie; **SC:** Special concern; **T:** Threatened; **V:** Vulnerable.*

(1) Liste des espèces désignées menacées ou vulnérables au Québec (LEMVQ). (Gouvernement du Québec, 2023c).

(2) Committee on the Status of Endangered Wildlife in Canada (COSEWIC, 2023).

(3) Species at Risk Act (SARA). Government of Canada (2023).

Regarding special status plant species, the “Potentiel” tool (Gouvernement du Québec, 2023d) was used to develop a preliminary list of threatened plant species potentially present in the Nord-du-Québec administrative region. This list contains 95 vascular plants potentially present in the study area but has been refined to 10 species following habitat characterization (Table 20.4). Indeed, the boreal forest dominated by spruce-lichen woodland in the inventory zone follows a dynamic associated with the fire regime and has a generally low diversity of vascular flora. Habitats are fairly homogeneous, and the relative paucity of remarkable floristic elements characterizes the area as a whole. No special status plant species were observed during the surveys carried out by WSP in August 2023, June 2024 and August 2024.

Table 20.4: Special Status Plant Species Potentially Present in the Project Area

Class	English Name	Scientific Name	Status		
			LEMVQ (1)	COSEWIC (2)	SARA (3)
Plant	Brown-edged pussytoes	Antennaria rosea subsp. confinis	LDTV	–	–
	Great northern aster	Canadanthus modestus	LDTV	–	–
	Spatulate moonwort	Botrychium spathulatum	LDTV	–	–
	Upswept moonwort	Botrychium ascendens	LDTV	–	–
	Ojibway waterwort	Elatine ojibwayensis	LDTV	–	–
	Rocky Mountain willowherb	Epilobium saximontanum	LDTV	–	–
	Limestone swamp bedstraw	Galium brevipes	LDTV	–	–
	Nahanni oak fern	Gymnocarpium continentale	LDTV	–	–
	Purple meadow-rue	Thalictrum dasycarpum	LDTV	–	–
	McCalla’s willow	Salix maccalliana	LDTV	–	–

20.1.3 Social Components

20.1.3.1 Administrative Context

James Bay Northern Quebec Agreement (JBNQA)

The Shaakichiwuaanaan mining Project is located in the territory covered by the JBNQA, on Category III lands. The environmental and social protection regime applicable in the James Bay region is established under Chapter 22 of the JBNQA and is governed by the provisions of Title II of the Environment Quality Act (LQE).

The land use regime introduced by the JBNQA is a determining factor in land use and guarantees the participation of the Cree in forestry, mining and hydroelectric development on the territory. It divides the territory into Category I, II and III lands. Under the applicable land regime, Category II and III lands are public lands of the Government of Quebec. The Government may assign land rights for various projects while ensuring diversity of use, compliance with applicable laws and regulations, and land use planning. The study area is located on Category III lands.

Category III lands represent all lands in the agreement area that are not included in Categories I and II. On these lands, the Cree have exclusive trapping rights for fur-bearing animals and certain advantages in the outfitter industry without exclusive rights. They may establish any camp necessary for hunting, fishing and trapping without a title from the Quebec Government. On Category III lands, hunting and fishing are also permitted to non-Cree. Mining rights belong to the Provincial Government.

Category II lands are also part of the Quebec public domain. These are lands where the Cree have exclusive rights to hunt, trap, fish and engage in commercial fishing, and the right to establish camps. Since the formation of the GREIBJ, the Cree Nation Government (CNG) has jurisdiction over Category II lands.

Category I lands are reserved for the exclusive use of the Cree. They may be used for residential, community, commercial, industrial or other purposes. The Cree have exclusive hunting, fishing and trapping rights on these lands.

The hunting, fishing and trapping regime established by the JBNQA applies to land mammals, freshwater and marine fish, migratory birds and marine mammals, and grants Cree and Inuit beneficiaries the right to harvest any wildlife species at any time and in any place on the territory to meet their subsistence needs. Beneficiaries also have exclusive rights to harvest certain species such as beaver, bear, polar bear and sturgeon (CNG, 2025).

Gouvernement Régional d'Eeyou Istchee Baie-James (GREIBJ)

In addition to the JBNQA, the legislative and legal context of Northern Quebec is also governed by the *Agreement Concerning a New Relationship Between the Government of Quebec and the Cree of Quebec*, also known as the *Paix des Braves*. Based on the values of trust and mutual respect, the agreement aims to promote respect for the traditional way of life of the Cree and to foster greater autonomy for the Cree in their own development. Following this agreement, *the Agreement on Governance in the Eeyou Istchee James Bay Territory* was signed by the Cree of Eeyou Istchee and the Government of Quebec in July 2012, with a view to modernizing the governance regime and creating a joint Cree-James Bay regional government (GREIBJ).

The GREIBJ is governed by Quebec law and exercises the same powers, functions and authorities over Category III lands located in the Eeyou Istchee James Bay territory as those that were, until 2014, assigned to the Municipality of James Bay. The regional government has the option of declaring its jurisdiction as a regional county municipality (RCM). It also acts as a regional conference of elected officials (CRÉ) with respect to the territory and resources (GREIBJ, 2025a).

The GREIBJ has adopted urban planning by-laws applicable to its territory, i.e., the territory outside Categories I and II lands as designated by the Act respecting the land regime in the James Bay and New Québec territories (chapter R-13.1). These regulations designate the preferred uses for the various parts of the Category III territory, such as housing, resort development, commerce and services, leisure and recreation, public and institutional uses, agriculture, forestry, resources and conservation. With its implementation, the preferred uses for each of these zones are defined, along with the standards to be considered for these uses (GREIBJ, 2020). The Project is located in zone 53-12-R, with resource development (R) as its dominant use.

It should be noted that the issuance of resort leases has been suspended throughout the Eeyou Istchee James Bay territory since the signing of the new governance agreement for this territory in 2012 (MRNF, 2023).

20.1.3.2 Local Communities

The Eeyou Istchee James Bay Region is made up of the traditional territory of Eeyou Istchee, with the Indigenous nation of the Eastern Cree, as well as the Jamésie, a non-Indigenous territory equivalent to a regional county municipality (RCM). Divided into 16 communities, the Jamésiens and the Crees live side by side. This section presents a brief portrait of Radisson, Matagami and the Cree communities of Chisasibi, Wemindji and Mistissini, which are closest to the Shaakichiuwaanaan Project (Table 20.5).

Table 20.5: Cree Communities and Non-Indigenous Towns Near the Shaakichiuwaanaan Project

First Nation / Non-Indigenous Town	Land Status	Name of the Community	Affiliated Tribal Council	Approximate Distance from the Project	Population
Cree Communities					
Cree Nation of Chisasibi	JBNQA Territory	Chisasibi	Grand Council of the Crees	330 km west	5,000
Cree Nation of Wemindji	JBNQA Territory	Wemindji	Grand Council of the Crees	330 km southwest	1,562
Cree Nation of Mistissini	JBNQA Territory	Mistissini	Grand Council of the Crees	350 km south	3,190
Jamésie					
Radisson	JBNQA Territory	N/A	N/A	250 km west	200
Matagami	JBNQA Territory	N/A	N/A	815 km southwest	1,400

**Note: N/A - Not applicable.*

Note that Makivvik, as the administrative representative of the Inuit under the JBNQA, has also expressed an interest in the Shaakichiuwaanaan Project. The Inuit interest is tied to the Kiggaluk Category I lands reserved for the Inuit population living in the Cree community of Chisasibi. The Inuit land in question is managed by the Kigaluk Land Holding Corporation, and Makivvik acts as their representative under the JBNQA. Further engagement is planned with Makivvik to better understand their interest in the Project. (Makivvik, 2025) (Gouvernement du Québec - Ministère des Affaires municipales et de l'Habitation, 2025)

20.1.3.2.1 Cree Communities

Eeyou Istchee, the traditional territory of the Cree, covers an area of over 400,000 km^2, and includes nine (9) communities located on the shores of James Bay (Waskaganish, Eastmain, Wemindji and Chisasibi), Hudson Bay (Whapmagoostui) and inland (Nemaska, Waswanipi, Mistissini and Oujé-Bougoumou) (CNG, 2024). Washaw Sibi was added as the tenth community of the Cree Nation in 2003. The GNC has also recognized MoCreebec Eeyoud, located on the Ontario shore of James Bay, as the eleventh community.

The traditional territory of the Cree, Eeyou Istchee, includes more than 300 traplines, which can be described as traditional family hunting and trapping grounds (GNCC, 2023). Each Cree community is administered by a local government (band council), independent of which the elected chief sits on the board of directors of the Grand Council of the Crees (Eeyou Istchee) (GCC) and on the GNC council.

The Crees are among the most populous Indigenous nations in the province and make up more than one-third (35%) of the population of Northern Quebec. The total population of the nine (9) communities included in the 2021 census was 18,225. Chisasibi and Mistissini are the two (2) most populous communities, together accounting for nearly half of the population of the nine (9) communities of Eeyou Istchee. The largest community is Chisasibi, with a population of 4,985 (Statistics Canada, 2023a). In contrast, the communities of Nemaska and Oujé-Bougoumou are the least populated, with populations of less than 850 (Statistics Canada, 2023a).

Between 2016 and 2021, the population of Cree communities grew at a faster rate than that of Quebec (6.1% versus 4.1%). Population growth in these communities has been impressive for several years. As an indication, the Cree population more than tripled between 1971 and 2011, while in Quebec, the increase was around 30% (Statistics Canada, 2022). In fact, nearly 50% of the Cree population living in these communities was under the age of 25 in 2021.

The following provides a brief summary of the communities of Chisasibi, Wemindji and Mistissini, as these three (3) communities are most likely to be affected by the Project. According to the Cree Nation Government, Chisasibi remains the community most affected by the Shaakichiuwaanaan Project, as all planned Project infrastructure is located on its traditional territory.

Chisasibi

Chisasibi, which means “Great River” in Cree, is the northernmost community in Eeyou Istchee accessible by road, connected to the Billy Diamond Road by a 90-km paved east-west road starting at kilometre 600 of the road. In February 2025, the First Nation had 4,822 registered members, the vast majority (4,510) of whom lived in Chisasibi. The remaining members lived off reserve (203 members) or on another reserve (109 members) (RCAANC, 2025).

Formerly located on Fort George Island, the village of Chisasibi was relocated to the mainland at the mouth of the Grande Rivière on the east coast of James Bay in 1981. This new location was designated following negotiations with the provincial government and Hydro-Québec, when the village was threatened by hydroelectric development planned for the river (Commission de toponymie Québec, 2016). Archaeological excavations have uncovered signs of Indigenous habitation dating back more than 5,000 years.

The community's territory consists of 37 traplines spread between the mouth of the Grande Rivière and the Caniapiscau Reservoir. The community covers an area of 1,305 km^2, comprising 825 km^2 of Category 1A land and 480 km^2 of Category 1B land. The traditional territory, including Category II and III lands, covers 81,199.56 km^2. It is the largest Cree community in terms of population (4,985 inhabitants in 2021) and area.

Wemindji

Wemindji, which means “painted hills” or “red ochre mountain“ (depending on the source), is located at the mouth of the Maquatua River on the coast of James Bay. In February 2025, the First Nation had 1,665 registered members, the vast majority (1,467) of whom lived in Wemindji (AADNC, 2025).

The Wemindji territory consists of 21 traplines on both sides of the Maquatua River. It encompasses Boyd and Sakami lakes, which were affected by hydroelectric development in 1980 (Phase 1 of the La Grande complex). The community can be reached from Billy Diamond Road via an access road built in the early 1990s. The nearest town by road is Radisson, 203 km away.

Mistissini

Mistissini, which means “big rock” in Cree, is one of the Cree communities located inland, approximately 350 km south of the Shaakichiuwaanaan Mining Project, on the shores of Lake Mistassini. In April 2025, the First Nation had 4,256 registered members, the vast majority of whom (3,857 members) lived in Mistissini (AADNC, 2025).

The Mistissini territory consists of 77 traplines totalling 117,844 km^2, spread between the south of Mistissini Lake and the north of the Caniapiscau Reservoir. It is the largest territory and has the largest number of trapline ranges among the Cree communities of Eeyou Istchee.

The community is accessible by a 15-km paved access road that was completed in the early 1970s and connects to Route 167 at PK 304. The nearest Cree village is Oujé-Bougoumou, located 150 km by road from Mistissini. Mistissini is less than 90 km by road from Chibougamau, where the nearest airport is located. It should be noted that the construction of an airport in Mistissini is being studied as part of the La Grande Alliance feasibility studies.

20.1.3.2.2 Jamesian

Jamésie has four (4) municipalities (Chapais, Chibougamau, Lebel-sur-Quévillon and Matagami) and three (3) localities (Radisson, Valcanton and Villebois) (Table 6.1). Each municipality and locality is administered by a municipal council. The population of Jamésie totalled 13,448 in 2023 (Institut de la Statistique du Québec, 2024b). The following section focuses on the locality of Radisson and the municipality of Matagami. It should be noted that Radisson is the Jamésian community closest to the Project (350 km by road) and that Matagami is where the ore is planned to be delivered for further transport by rail. Therefore, the sections below discuss certain characteristics of these two (2) communities, since Radisson and Matagami are the Jamésian localities and municipalities most likely to be affected by the Project.

Radisson

Located 250 km west of the Project, the community of Radisson is the only non-Indigenous community in Quebec beyond the 53rd parallel, at the northern end of the Billy-Diamond Road.

The village of Radisson was founded in 1974 to serve as the regional development centre for Hydro-Québec's James Bay hydroelectric project. The project consisted of a series of hydroelectric developments, mainly in the Grande Rivière watershed. Throughout the various phases of hydroelectric development, the village saw its population grow and acquired facilities and infrastructure capable of serving a community of several thousand residents. Formerly administered by the SDBJ, the village obtained the status of a locality in 1994 (Localité de Radisson, 2024).

In the late 1970s, Radisson had a population of about 2,500, consisting of those working on the construction of the Hydro-Québec dam and power plant and their families. Today, Radisson is home mainly to the families of Hydro-Québec workers who maintain the hydroelectric complexes in James Bay. In addition to Hydro-Québec, the Director General of Radisson mentions Transport Taiga, Air Inuit, CBJ Mécanique, Construction Paradox Inc. and Distribu-Nord among the main employers in the community. Part of Radisson's population also works in the growing tourism industry, which represents an avenue for diversifying the local economy (Grand Québec, 2025).

Municipality of Matagami

The municipality of Matagami, located on the 50th parallel in the southwestern section of the Eeyou Istchee James Bay territory, is one of the four (4) municipalities that make up Jamésie.

Established at the confluence of the Bell, Allard and Waswanipi rivers, Matagami (which means ‘meeting of the waters’ in Cree) was founded in 1963 following the arrival of mining explorers in the region and the establishment of their mining camp, Camp Matagami. The Quebec government's idea to build a modern mining town took shape a few months later. At the turn of the 1970s, Matagami experienced a period of economic growth linked to hydroelectric development in the James Bay region. The population grew rapidly, and businesses developed. Today, Matagami is a town with modern infrastructure and a local economy based on mining and forestry (Matagami, 2024).

Matagami is the seat of the regional government of Eeyou Istchee James Bay and is the starting point of the Billy-Diamond Road and the junction with the railway. The municipality has a transshipment centre, a strategic point for intermodal freight transport between Eeyou Istchee James Bay and the rest of the province.

According to the town, the closure of the Matagami Mine in June 2022, after 59 years of mining that led to the founding of Matagami, will lead to local economic diversification and a community takeover to secure its future (Matagami, 2025). Much of Matagami's economic activity currently relies on the development of mining projects located near and north of Matagami. In this regard, the Chief Executive Officer of Matagami, who was interviewed as part of the Project, hopes to see the town position itself as a transportation hub in the region, particularly in the development of the critical and strategic minerals sector. These are expected to generate significant traffic at the Matagami transshipment yard for decades to come.

20.1.3.3 Local Land Use and Dwellings

Land use and dwellings in the area local to the Project are mostly temporary and seasonal in nature.

The Project site is located in hunting zone number 22 north and overlaps fur management units (UGAF) numbers 91 and 94 (Québec, Gouvernement du, 2022b). In addition, lake trout and walleye fishing are activities of significant interest. The Project is located on Category III lands, which are accessible to the public, but the Cree retain exclusive hunting and trapping rights for fur-bearing species, as well as fishing rights for certain aquatic species, such as lake whitefish, yellow sturgeon, burbot and brown trout.

There are no active outfitting operations in the local study area. The former Mirage Adventure outfitting operation, located approximately 75 km east of the Project at kilometre 358 of the Trans-Taiga Road, was transferred to the Cree Health Board in 2024. It should also be noted that some of the infrastructure of the former Nouchimi outfitter at kilometre 286 of the Trans-Taiga Road and some of its satellite camps were dismantled in 2023 (COMEV, 2025). An exploration company has been authorized to take possession of certain facilities and materials that were initially to be disposed of. In fact, since the ban on sport hunting of caribou in 2018 for the Rivière aux Feuilles herd and in 2011 for the Rivière Georges herd, several outfitters have had to cease operations (Lecavalier, 2017).

According to the Executive Director of Tourisme Baie-James, the local study area may be frequented for recreational tourism by adventure tourism enthusiasts and van life travellers. However, no recreational tourism projects are currently under development in the local study area.

20.1.3.4 Economic Activities

In 2022, the gross domestic product (GDP) of the Nord-du-Québec administrative region was $5.844 billion, representing 1% of Quebec's total GDP of $505.16 billion. Between 2014 and 2022, the administrative region posted an average annual GDP growth rate of 6.7%, the highest among Quebec's administrative regions during this period (Institut de la Statistique du Québec, 2024b). The region's main economic sectors,

based on GDP in 2022, were mining, quarrying and oil and gas extraction (45.9%); construction (13.3%); public services (8.9%); public administration (7.3%); and health care and social assistance (6.3%) (Institut de la Statistique du Québec, 2024b).

The mineral resource extraction sector accounted for 18.1% of the GDP of Northern Quebec. Investment spending by mining companies in the administrative region represented 1.8% of total investment spending for this sector at the provincial level. The Nord-du-Québec administrative region had 4,075 direct jobs related to the mining sector, representing 26.1% of jobs in this sector province-wide, the highest proportion of mining jobs after Abitibi-Témiscamingue among all administrative regions in Quebec (EcoTec Consultants, 2024).

It should be noted that in 2022, the mining sector had an average annual salary of $117,916, more than twice the median employment income in Quebec, which was $54,328 (EcoTec Consultants, 2024).

20.1.3.5 Regional Infrastructure

The Project is located south of the Trans-Taiga Road, approximately 20 km by road from kilometre 270. The Trans-Taiga Road was built in 1979. It provides access to Caniapiscau and several facilities at the La Grande hydroelectric complex. It is a gravel road administered by Hydro-Québec that extends 666 km, generally in an east-west direction (James Bay Road, undated).

The main road in the regional study area is the Billy-Diamond Highway, formerly known as the James Bay Road. It crosses the region on a north-south axis. The 620-km Billy-Diamond Highway was built in the 1970s to provide access to the James Bay hydroelectric construction sites. The Trans-Taiga Road begins at kilometre 544 of this road. The Billy-Diamond Highway, which connects Matagami to Radisson, has a rest area at kilometre 381 (the only place along the road with a service station) and six (6) emergency telephones. This road has special status, allowing the transport of heavy and oversized loads. It is administered by the *Société de développement de la Baie-James* (SDBJ), which is responsible for maintaining this road. The SDBJ is currently mandated by the Quebec government to maintain the Billy-Diamond Highway.

The Cree communities are also served by access roads from the Billy-Diamond Road (for Waskaganish, Eastmain, Wemindji and Chisasibi), the North Road (Nemaska), Route 167 (Mistissini) and Route 113 (Waswanipi). It should also be noted that most Cree communities are served by an airport owned by Transport Canada and operated by the communities themselves. In the Cree community of Chisasibi, the Robert Kanatewat Airport welcomes Air Creebec flights and also provides medical transportation services for the James Bay Cree Health and Social Services Council (CCSSSBJ) (Chisasibi Cree Nation, 2024).

The region is also served by other airports and aerodromes. La Grande 4 aerodrome is located approximately 30 km northeast of the Project. This facility serves the La Grande 4 hydroelectric facilities and is owned by Hydro-Québec (operations have been gradually transferred to the SDBJ). In addition, La Grande-Rivière Regional Airport is located in the municipality of Radisson (see Section 6.4). It is operated by the SDBJ (SDBJ, 2024). According to the Director General of the Baie-James Regional Administration (ARBJ), who was interviewed as part of this Project, the region is working to improve airport infrastructure in Jamésie, in particular to enable air shuttle service for workers in the region. In addition, an airstrip is available at Camp Mirage Aventure, about 40 kilometres east of the mining Project site. This airfield is used by workers at the PMET exploration camp, and two (2) weekly flights serve this location, carrying 10 to 20 passengers.

The regional study area includes various hydroelectric developments, particularly related to the La Grande and Eastmain-Sarcelle-Rupert hydroelectric complexes. It is therefore crossed by several power transmission lines and includes various substations and power plants.

20.1.3.5.1 Prospective Future Regional Development

Launched in February 2020, the “La Grande Alliance” (LGA) (meaning “The Great Alliance” in English) was formed to ensure the sustainable development of infrastructure in the Eeyou Istchee James Bay region. The transportation infrastructure examined as part of LGA’s feasibility studies is designed to meet specific needs or seize opportunities, with the aim of fully integrating the economy of the Eeyou Istchee James Bay region, and Cree communities specifically, with a view to sustainable resource development.

The main infrastructure studied by *La Grande Alliance* (LGA, 2023a) included the following:

- Rehabilitation and paving of access roads to the communities of Waskaganish, Eastmain, Wemindji, Nemaska and Mistissini.
- Rehabilitation and paving of the North Road.
- The rail link between Matagami and Rupert River.
- Reactivation of the Grevet-Chapais rail line.
- The rail link between Rupert River and La Grande River.
- The extension of the road to Whapmagoostui.
- The rehabilitation and extension of Road 167.
- The rail link between La Grande River and Whapmagoostui.
- Development of a seasonal port at Whapmagoostui.

For the Shaakichiuwaanaan Project, PMET plans to transport the spodumene concentrate to Matagami by truck, via the existing Billy-Diamond Road. In the context of LGA's development, PMET continues to engage with LGA and supports future development, which would improve regional connections. LGA's proposal for the possible extension of Road 167 to the Trans-Taiga Road, creating a second north-south transportation corridor to serve the eastern part of the territory, could significantly reduce travel time between Mistissini / Chibougamau and Chisasibi, linking the two (2) most populous regions in the area, facilitating interregional connectivity and providing access to currently isolated areas (LGA, 2023b). Although PMET remains very interested in the LGA projects, the FS does not assume any of them will go forward.

20.1.3.6 Traditional Land Use

Among the most important values that emerged from the 2017 report of the Eeeyou Planning Commission of Chisasibi (EPC Chisasibi, 2017, cited in WSP, 2024) are the importance of the territory, being together on the territory, and the transmission of Cree culture, knowledge and values.

20.1.3.6.1 Traditional Land Use in the Community of Chisasibi

The Great River (La Grande) is a big part of who the people of Chisasibi are. It has played a huge role in the community's history as a way to get around, a place to gather, and a spot for food and cultural activities, before being seriously impacted by the hydroelectric development mentioned earlier. In addition to the construction of four (4) reservoirs along its course, the current flow of the river in front of the Chisasibi community is approximately three (3) times greater than it was under natural conditions, as a result of numerous watersheds being diverted into the La Grande complex. It remains a very significant place in the eyes of community members (EPC Chisasibi, 2017, cited in WSP, 2024).

In addition, the old camps and cultural sites are highly valued, and their protection is essential to maintaining the connection to the land and identity. In fact, the Cree way of life has undergone dramatic changes in recent decades that have made the preservation of cultural identity a challenge that the Chisasibi Cree are determined to meet for future generations (EPC Chisasibi, 2017, cited in WSP, 2024).

In their vision for the future, the Chisasibi Cree who participated in the 2017 CFP survey intend to do everything possible to protect the territory that has not yet been altered and that which is valued as activity areas, including sites of cultural significance. They want to be involved in decisions about the territory so that development is carried out in accordance with the values and wishes of the community. They also believe that programs and other opportunities could facilitate travel and the presence of members in the territory. Finally, the trapline system should be reviewed to ensure that its implementation respects Cree values (EPC Chisasibi, 2017, cited in WSP, 2024).

Among the Cree communities, Chisasibi had the highest participation in the Economic Security Program (ESP) in 2022-2023. Seventeen per cent (17%) of residents were enrolled in the program to ensure their livelihood, sometimes in combination with other casual jobs. This represents 519 family units, or 664 adults and 223 children. The previous year, this number was slightly higher (574 family units). Overall, the number of EBP registrants has been declining over the years in Cree communities.

In total, 101,290 days spent in the woods were paid to land users for that same year, for an average of $18,597 per family unit. More than ten (10) years earlier (2011–2012), the number of people registered was slightly higher (976), representing 24% of the population, but the number of registered family units was lower (496) (OSECC, 2012, 2022, 2023). The fluctuation in WIP participation can be explained by several factors, such as the economic climate, employment opportunities and a readjustment of eligibility.

In addition, many other Chisasibi members who do not participate in the EIP stay on the territory when possible. For example, more than 100 users of the territory were identified on a single trapline near James Bay (VEI-WSP, 2024). Other less accessible traplines are accessed by helicopter or snowmobile for goose, moose and caribou hunting and fishing. Some users also trap beavers and wolves.

However, the Chisasibi ATA administrator interviewed for this Project indicated that changes in ice cover have restricted snowmobile use in some areas. In particular, community members no longer use snowmobiles for subsistence activities in the LG2 reservoir area as they did in the past because there is no longer any ice cover.

20.1.3.6.2 Traditional Land Use in the Project Study Area

The Project is located on trapline CH39, which is frequented by many Cree people. This land is also crossed by the Trans-Taiga Road, which will be used as part of the Project. This section describes the camps used on this trapline, the movements and activities of Chisasibi community members as shared during a series of meetings and interviews.

20.1.3.6.3 Camps

Several camps are used by the extended family of the trapline holder, which includes many siblings. These camps are mainly located in the northern section of the trapping grounds, along the Trans-Taiga Road or La Grande Rivière. Two (2) camps are located at kilometre 258 of the Trans-Taiga Road, and three (3) other camps are located on two (2) sites in the Katatipawasakakamaw Lake area (also known as Cladonia Lake), on the northern edge of the Trans-Taiga. It should be noted that the main infrastructure of a former outfitter's camp (Nouchimi) was dismantled in 2022 in this sector at kilometre 286 of the

Trans-Taiga (COMEV, 2025). Four (4) other camps are located along the Trans-Taiga Road, east of this area. Thus, nine (9) camps have been identified along the Trans-Taiga Road, including the trapline holder's camp at kilometre 341. Other camps are located inland near lakes, such as Camoy and Mihiiwaayaapaakw Lakes, east of the trapping grounds, and closer to the proposed mine site, in the Lake Nochet area, approximately 10 km east of the proposed site. A camp, built in 2024, is located 11 km from the proposed mine site, near a goose hunting area.

20.1.3.6.4 Water Supply

The trapline holder’s extended family obtains water from various locations, depending on the location of their camp, and also generally brings water with them from the community. Two (2) sites for collecting drinking water have been identified.

20.1.3.6.5 Access and Means of Transport

Before the construction of the Trans-Taiga Road (in 1979), users of the territory travelled to their camps by seaplane. Landing sites have been identified along the Grande Rivière, Lake Magin and Lake Mihiiwaayaapaakw. The Trans-Taiga Road is now the main access route to the land and is particularly busy during the traditional spring “Goose Break”. Snowmobile trails have also been identified by users, particularly for accessing goose hunting sites.

20.1.3.6.6 Fishing Activities

Various species are fished on the territory, including sturgeon, brook trout and brown trout. Fishing is practised in the Grande Rivière and in certain lakes on the trapping grounds. They prefer not to fish in reservoirs, as they believe the fish taste worse there. They are also wary of the quality of the fish in Mihiiwaayaapaakw Lake, despite the presence of large trout, due to mercury contamination and the activities of seaplanes from the former Nouchimi outfitter. Furthermore, the 2023 forest fire may have had an impact on the fish population, as according to one of the family elders, it can take about six (6) years for fish to repopulate the lakes at the heart of the fires. It should be noted that fishing activities also take place on Shaakichiuwaanaan Lake (identified as Lake 027). Shaakichiuwaanaan Anaadohyatch Lake (Lake 001) has long been known to have few fish.

20.1.3.6.7 Goose Hunting

This traditional and culturally significant activity for the Cree takes place in the spring, mainly between late April and late May, during the northward migration of geese. It is an opportunity for family gatherings on the

land, sharing and cultural transmission. It is an activity that excites all generations in Cree communities. On the CH39 site, families also gather for the annual Goose Break.

20.1.3.6.8 Moose and Caribou Hunting

Moose hunting is practised in various locations on the trapping grounds, particularly in the autumn by the trapline holder. The latter indicates that he does not need to hunt every year due to the amount of meat that a moose provides. However, he is concerned about overhunting by users from other trapping grounds via the Trans-Taiga Road.

As for caribou, users are seeing fewer and fewer in the area, but community members continue to hunt them. Some believe that caribou meat has changed in taste, which could be an indicator of poorer animal health.

20.1.3.6.9 Other Species

Ptarmigan are also hunted in the trapping area. Some users of the territory have reported that traffic is affecting the quality of the meat, as ptarmigan from disturbed areas do not taste the same as those from more remote areas.

Bears may also be hunted or trapped, and the meat is then shared with elders. It was noted that in autumn 2023, bears were starving due to the previous summer's forest fires. Beavers, porcupines (which have medicinal properties) and hares are also harvested in the area.

Users report that the construction and presence of infrastructure in the area (hydroelectric facilities, power lines, the Trans-Taiga road, borrow pits, etc.) have had an impact on the environment, to the extent that there are almost no small mammals left. Users have also observed bird species from the south that they were not used to seeing in the trapping area. Other species are more prevalent, such as the Arctic fox.

Users also harvest berries in the area, including blueberries, blackberries and cranberries, away from the mining exploration site.

20.1.3.7 Heritage and Archeology

According to the information available on the *Inventaire des Sites Archéologiques du Québec* (ISAQ) of the *Ministère de la Culture et des Communications du Québec* (MCC), no areas of high archaeological potential have been identified near the Shaakichiuwaanaan Project.

Four (4) archaeological sites have been previously identified along the shore of Hammerhead Lake (Katatipawasakamaw Lake), 30 km northeast of the Project site, along the Trans-Taiga Road. No other registered archaeological sites were present in the area prior to field investigations (Archéoconsultant Inc., 2024).

A study of archaeological potential was conducted in 2024 in the Project study area by a specialized firm, Archéoconsultant Inc. (Archéoconsultant Inc., 2024). The study area, covering 74 km^2, was subdivided into 354 zones with indigenous archaeological potential, including 273 zones with moderate potential (3.42 km^2) and 81 zones with high potential (0.21 km^2). The rest of the study area has low to no potential.

The non-Indigenous archaeological potential appears to be very low due to the absence of historical settlements, but remains of former explorer, surveyor, or prospector camps may still be present. An overview of areas with archaeological potential, carried out in 2024, concluded that the infrastructure planned for the Project could potentially impact an area of approximately 0.179 km^2 with medium to high archaeological potential.

An archaeological field study was undertaken in July and August 2025 within areas of high and moderate potential within a 100-m buffer of the planned Project infrastructure area to confirm the presence of archaeological or cultural artifacts (Archéoconsultant Inc., 2025). The field study included conducting surveys at a minimum of 10 m intervals and a thorough visual inspection. The field survey led to the uncovering of four (4) sites of archaeological interest throughout the field survey: quartz flakes; a multi-purpose quartzite tool; a metal lid of a tea kettle (dated 1940-1970); and a group of flat stones, some of which are arranged vertically on a rocky ledge. The Cree community and the Ministry of Culture and Communications have been alerted to these discoveries. Some sites may be of sufficient interest to carry out further investigation and to implement protection measures as the Project advances.

20.2 Jurisdiction, Applicable Laws and Regulations

20.2.1 Provincial Laws and Regulations

Schedule 1 of Section 22 of the James Bay and Northern Québec Agreement contains a list of projects subject to the environmental and social impact assessment (ESIA) and review process described in Division III of Chapter II of Title II of the *Environment Quality Act* (EQA"; c. Q-2). Schedule A of the EQA repeats this list and refines it to make it operational. The Shaakichiuwaanaan Project, as a mining project on the territory of the JBNQA, is designated by paragraph a) of Schedule A of the EQA:

- All mining developments, including the additions to, alterations or modifications of existing mining developments.

The Preliminary Information Statement was submitted to the MELCCFP in November 2023. On April 5, 2024, the MELCCFP confirmed that the Project was subject to the environmental and social impact assessment procedure and issued a directive for the completion of the impact study.

20.2.2 Federal Laws and Regulations

The Schedule of the Physical Activities Regulations (SOR/2019-285) describing the Project in whole or in part is as follows:

- *18(c): The construction, operation, decommissioning and abandonment of a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5,000 t/day or more.*
- *18(d): The construction, operation, decommissioning and abandonment of a new metal mill, other than a uranium mill, with an ore input capacity of 5,000 t/day or more.*

An Initial Project Description was submitted to the Impact Assessment Agency of Canada (IAAC) in February 2025. Following a public consultation period and a series of questions, the Project was formally designated as being subject to the federal Impact Assessment process in May 2025. The federal impact assessment process is currently underway, with the Tailored Impact Assessment Guidelines published in August 2025.

20.3 Environmental Permitting

Table 20.6 presents the most significant acts, regulations, directives and guidelines that apply to the Project. This list is non-exhaustive and is based on information known so far. Their applicability will have to be reviewed as the Project components are further defined.

Table 20.6: Provincial and Federal List of Permits

Acts and Regulations
Provincial
***Environment Quality Act* (c. Q-2)**
Regulation respecting the application of section 32 of the *Environmental Quality Act* (Q-2, r. 2)
Regulation respecting the application of the *Environment Quality Act* (Q-2, r. 3)
Regulation respecting the regulatory scheme applying to activities on the basis of their environmental impact (Q-2, r. 17.1)
Design code of a storm water management system eligible for a declaration of compliance (Q-2, r.9.01)

Acts and Regulations
Clean Air Regulation (Q-2, r. 4.1)
Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec (Q-2, r. 25)
Regulation respecting the operation of industrial establishments (Q-2, r. 26.1)
Snow, road salt and abrasives management regulation (Q-2, r. 28.2)
Regulation respecting pits and quarries (Q-2, r. 7)
Regulation respecting the landfilling and incineration of residual materials (Q-2, r. 19)
Regulation respecting used tire storage (Q-2, r. 20)
Regulation respecting the declaration of water withdrawals (Q-2, r. 14)
Regulation respecting mandatory reporting of certain emissions of contaminants into the atmosphere (Q-2, r. 15)
Regulation respecting halocarbons (Q-2, r. 29)
Regulation respecting hazardous materials (Q-2, r. 32)
Regulation respecting the reclamation of residual materials (Q-2, r. 49)
Regulation respecting activities in wetlands, bodies of water and sensitive areas (Q-2, r. 0.1)
Regulation respecting compensation for adverse effects on wetlands and bodies of water (Q-2, r. 9.1)
Protection policy for lakeshores, riverbanks, littoral zones and floodplains (Q-2, r. 35)
Water withdrawal and protection regulation (Q-2, r. 35.2)
Land protection and rehabilitation regulation (Q-2, r. 37)
Regulation respecting the quality of the atmosphere (Q-2, r. 38)
Regulation respecting the charges payable for the use of water (Q-2, r. 42.1)
***Directive 019 sur l'industrie minière* (2012)**
Protection and rehabilitation of contaminated sites policy (1998)
***Mining Act* (c. M-13.1)**
Regulation respecting mineral substances other than petroleum, natural gas and brine (M-13.1, r. 2)
***Threatened or Vulnerable Species Act* (c. E-12.01)**
Regulation respecting threatened or vulnerable wildlife species and their habitats (E-12.01, r. 2)
Regulation respecting threatened or vulnerable plant species and their habitats (E-12.01, r. 3)
Compensation Measures for the Carrying out of Projects Affecting Wetlands or Bodies of Water Act (M-11.4)
***Act respecting the conservation of wetlands and bodies of water* (2017, chapter 14; Bill 132)**
***Watercourses Act* (c. R-13)**

Acts and Regulations
Regulation respecting the water property in the domain of the State (R-13, r. 1)
***Conservation and Development of Wildlife Act* (c. C-61.1)**
Regulation respecting wildlife habitats (C-61.1, r. 18)
***Act respecting the lands in the domain of the state* (chapter T-8.1)**
Regulation respecting the sale, lease and granting of immovable rights on lands in the domain of the State (chapter T-8.1, r. 7)
***Sustainable Forest Development Act* (chapter A-18.1)**
Regulation respecting the sustainable development of forests in the domain of the State (chapter A-18.1, r. 0.01)
Regulation respecting forestry permits (chapter A-18.1, r. 8.)
***Building Act* (c. B-1.1)**
Safety Code (B-1.1, r. 3)
Construction Code (B-1.1, r. 2)
***Explosives Act* (c. E-22)**
Regulation under the *Act respecting explosives* (E-22, r. 1)
***Cultural Heritage Act* (c. P-9.002)**
***Occupational Health and Safety Act* (c. S-2.1)**
Regulation respecting occupational health and safety in mines (S-2.1, r. 14)
Highway Safety Code (c. C-24.2)
Transportation of Dangerous Substances Regulation (C-24.2, r. 43)
Federal
***Impact Assessment Act* (S.C. 2019, c. 28, s. 1)**
Physical Activities Regulations (SOR/2019-285)
Designated Classes of Projects Order (SOR/2019-323)
Information and Management of Time Limits Regulations (SOR/2019-283)
***Fisheries Act* (R.S.C., 1985, c. F-14)**
Authorizations Concerning Fish and Fish Habitat Protection Regulations (SOR/2019-286);
Metal Mining Effluent Regulations (SOR/2002-222)
***Canadian Environmental Protection Act* (S.C. 1999, c. 33)**
PCB Regulations (SOR/2008-273)
Environmental Emergency Regulations, 2019 (SOR/2019-51)
Federal Halocarbon Regulations (SOR/2003-289)

Acts and Regulations
National Pollutant Release Inventory
***Species at Risk Act* (S.C. 2002, c. 29)**
***Canadian Wildlife Act* (R.S.C., 1985, c. W-9)**
Wildlife Area Regulations (C.R.C., c. 1609)
***Migratory Birds Convention Act*, 1994 (S.C. 1994, c. 22)**
Migratory Birds Regulations (C.R.C., c. 1035)
***Nuclear Safety and Control Act* (S.C., 1997, c. 9)**
General Nuclear Safety and Control Regulations (SOR/2000-202)
Nuclear Substances and Radiation Devices Regulations (SOR/2000-207)
***Hazardous Products Act* (R.S.C., 1985, c. H-3)**
***Explosives Act* (R.S.C., 1985, c. E-17)**
***Transportation of Dangerous Goods Act* (1992)**
Transportation of Dangerous Goods Regulations (SOR/2001-286)

Table 20.7 presents a non-exhaustive list of required approvals, authorizations, permits or licences based on the known components of the Project and typical activities related to mining projects.

Table 20.7: Preliminary and Non-exhaustive List of Permitting Requirements

Activities	Type of Request	Authority
Closure plan	Approval	MRNF
Mining operations	Lease	MRNF
Mine waste management facilities and processing plant location	Approval	MRNF
Mine waste management facilities	Lease	MRNF
Infrastructure implementation on public land	Lease	MRNF
Construction and operation of an industrial establishment, the use of an industrial process and an increase in the production of property or services	Authorization	MELCCFP
Withdrawal of water, including related work and works	Authorization	MELCCFP
Establishment of potable, wastewater and mine water management and treatment facilities	Authorization	MELCCFP
Work, structures or other interventions carried out in wetlands and bodies of water	Authorization	MELCCFP

Activities	Type of Request	Authority
Installation and operation of any other apparatus or equipment designed to treat water to prevent, abate or stop the release of contaminants into the environment	Authorization	MELCCFP
Installation and operation of an apparatus or equipment designed to prevent, abate or stop the release of contaminants into the atmosphere	Authorization	MELCCFP
Industrial depollution attestation	Attestation	MELCCFP
Carry out an activity likely to modify a wildlife habitat	Authorization	MELCCFP
Operation of a borrow pit	Authorization	MELCCFP
Harvest wood on public land where a mining right is exercised	Authorization	MRNF
Build or improve a multi-use road	Authorization	MRNF
Use of high-risk petroleum equipment	Permits	RBQ
Construction	Permits	RCM
Construct, place, alter, rebuild, remove or decommission a work in, on, over, under, through or across any navigable water	Approval	Transport Canada
Harmful alteration, disruption or destruction of fish habitat	Authorization	DFO
Explosives possession, magazine and transportation	Permit	SQ
Explosives transportation	Permit	NRCan
Use of nuclear substances and radiation devices	Licence	CNSC
Notice and Environmental Emergency Plan	-	ECCC

20.4 Ore, Waste Rock and Tailings Management

A description of minerals, waste rock and tailings management is provided in Section 18.2. The geochemistry of the various materials is provided in Section 20.7.

20.5 Site-Wide Water Management

Site-wide water management is provided in Section 18.2.

The Project's water management system is designed to segregate contact water from non-contact water, ensuring appropriate treatment based on contamination levels. Several types of contact water are anticipated:

- From industrial areas, with elevated levels of metals or total suspended solids (TSS).
- From inert waste rock (Stockpile 001), also with elevated levels of TSS.

- From dewatering activities (both open-pit and underground), with elevated levels of metals or TSS.
- From potentially acid-generating (PAG) or Metal Leaching (ML) waste rock (Stockpile 002), with elevated level of arsenic (As) (Vision Geochemistry, 2025).
- From tailings (Stockpile 002), with elevated level of arsenic (Vision Geochemistry, 2025).
- Bleed water from the process plant, with elevated level of arsenic (Vision Geochemistry, 2025).

Water with elevated levels of TSS will be directed to passive treatment systems, including sedimentation and polishing ponds. In contrast, water containing metals such as arsenic will undergo chemical treatment in a dedicated water treatment facility.

Comprehensive hydrological studies and a site-wide water balance have been conducted to determine the required treatment capacity. The treatment process includes pH adjustment, the use of chelating agents, coagulation and flocculation, and sludge recirculation to ensure effective contaminant removal. Treated water is discharged to the polishing pond for final polishing. This pond also serves as the source of process water for the plant. The sludge generated during treatment is directed to geotubes for dewatering, then transported by truck to Stockpile 002 for co-disposal. Further details on water management and treatment processes are provided in Sections 18.6 and 18.7, respectively.

20.6 Baseline Hydrogeology

The hydrogeological baseline was developed using publicly available information as well as fieldwork investigations carried out in 2023 and 2024. All data was compiled to determine the different hydrostratigraphic units, assess the overall hydraulic properties of the units, the groundwater flow and groundwater quality.

The site is surrounded by water bodies such as lakes and marshy areas, and the topography is generally flat with small hills, such as drumlins and rock outcrops.

Two (2) hydrostratigraphic units are present at the site: the overburden unit and the bedrock unit. The overburden unit at the study area is discontinuous and varies in thickness from 0 m to 48 m, but generally around 8 m when present. It is comprised of a thin layer of organic material over undifferentiated till or drumlins. The till is mainly composed of silty sand, silt or sand with occasional gravel and trace amounts of clay becoming gravelly with depth. The overburden is water bearing but due to its discontinuous nature, it is not considered a large-scale aquifer. The bedrock unit generally consists of alternating gray and green

amphibolite, metasediments and pegmatitic intrusions. The RQD shows that the rock is generally of good quality with low fracturing. The bedrock outcrops at several locations within the study area.

In 2024, the groundwater level in the overburden was generally encountered at depth between 0.1 and 2.7 m below ground surface (bgs). The groundwater levels in the bedrock range from 0.1 to 3.1 m bgs, with two (2) locations reaching more than 7 m bgs. Groundwater flow in the overburden and shallow bedrock follows topography, with groundwater generally flowing from topographic highs, which act as recharge areas, towards the lakes and wetlands. In general, groundwater flows towards Lake 001 and Lake 027. The vertical hydraulic gradient between the overburden and the bedrock units is downwards in topographic highs, where groundwater occurs, and upwards in other areas and below Lake 001. Preliminary seasonal variation was assessed in three (3) boreholes, showing that the highest groundwater elevation was generally in May and June during the spring freshet.

The overburden hydraulic conductivity was assessed through various hydraulic testing methods, such as infiltration test in the vadose zone and rising and falling head tests in the saturated zone. The unsaturated hydraulic conductivity of the vadose zone ranges between 1.6×10^{-6} m/s and 4.1×10^{-5} m/s, while the hydraulic conductivity in the saturated overburden ranges between 2.9×10^{-8} m/s and 3.9×10^{-6} m/s with a geometric mean of 1.1×10^{-6} m/s.

The bedrock hydraulic conductivity was assessed through Lugeon tests, where specific intervals were tested, as well as rising and falling head tests and pumping tests. The geomean hydraulic conductivity of the bedrock is 5.8×10^{-8} m/s, with a maximum of 2.0×10^{-6} m/s and a minimum of 8.6×10^{-10} m/s. No correlation could be found between RQD and hydraulic conductivity, thus suggesting that flow within the bedrock is dominated by discrete features that are not captured by the RQD measurements. Slug and packer test results suggest that hydraulic conductivity in the bedrock decreases with depth.

Two (2) groundwater sampling campaigns were carried out in six (6) observation wells in the proposed CV5 pit, and 15 observation wells within the footprint of the proposed stockpiles. The results show background concentrations lower than the EDC and RES criteria of the MELCCFP for the majority of the parameters analyzed, except for arsenic, manganese, molybdenum, antimony, ammoniacal nitrogen and sulphides exceeding EDC and copper exceeding RES.

The geological and hydrogeological data available for the study area were used to develop three-dimensional groundwater numerical models. One (1) of the models was used to assess the dewatering rates for the open and underground pits and to estimate the extent of drawdown over the study area. The second model was developed to assess the efficacy of the seepage control measures proposed to prevent any significant degradation of the groundwater quality due to any potential seepage from

Stockpile 002, as the material stored in this stockpile was assessed as being a metal leaching risk, particularly arsenic, and potentially acid-generating (Vision Geochemistry, 2025).

20.7 Geochemistry

20.7.1 Static Testing

Static testing was conducted on a total of 354 samples, including 316 waste rock samples, 25 spodumene pegmatite (ore) samples, and 9 tailings samples representing different tailings processing streams. The waste rock samples were selected based on estimated lithological proportions, spatial representativeness, including variability along strike and at depth, and geochemical variability based on identified constituents of potential concern (COPCs), particularly arsenic, antimony, and lithium. The number of samples for each waste rock lithology was generally selected based on their relative abundance within the pit shell, with 128 samples of amphibolite, 101 samples of metasediment, 33 samples of barren pegmatite, and 54 samples of ultramafic waste rock. All samples were tested for Acid-base Accounting (ABA), Synthetic Precipitation Leaching Procedure (SPLP) and CTEU-9 leaching tests were conducted on all samples, with some samples selected for additional testing with Toxicity Characteristic Leaching Procedure (TCLP).

A total of 56 waste rock samples (18%) and one ore sample (4%) were identified as potentially acid-generating (PAG) according to the Guidelines on Characterization of Ore and Mining Residues (MELCCFP, 2023; previously MELCC, 2020). None of the tailings' samples were classified as PAG. A majority of the PAG occurrences were identified in the metasediment samples, with 36% of them being PAG, while a smaller proportion of amphibolite samples (13%) were also classified as PAG.

Leaching tests conducted using the SPLP and CTEU-9 procedures indicated that a portion of the waste rock, ore, and tailings samples were potentially metal leaching (ML) with several COPCs according to the MELCCFP Guidelines, with concentrations compared against the Criteria for the Evaluation of Acid and Alkaline Leachates (CVAA).

For arsenic, 116 waste rock samples (37%) were identified as potentially leachable by CTEU-9 compared with 20 samples (6%) by SPLP, with ultramafic material showing the highest leaching potential (93% and 31%, respectively). A smaller number of samples were classified as leachable at higher thresholds, with 28 exceedances under CTEU-9 and only four (4) under SPLP. CTEU-9 testing determined antimony leaching was also noted in 67 waste rock samples (21%), most prominently in ultramafic rocks (67% of samples), with only two (2) exceedances in SPLP. Both arsenic and antimony were also found to be potentially leachable in ore (79% and 40%, respectively) and tailings (100% and 56%, respectively) under CTEU-9, but no exceedances were recorded under SPLP.

Under CTEU-9 testing, uranium exceeded CVAA guidelines in barren pegmatite (64%), ore (76%), and tailings (78%), while SPLP tests reported exceedances only in two (2) barren pegmatite samples (6%). Lithium exceedances were restricted to CTEU-9, with elevated concentrations in 51% of waste rock, 100% of ore, and 56% of tailings.

Aluminum and, to a lesser extent, other metals, including copper, iron, lead, manganese, nickel, and zinc, were identified as potentially exceeding the CVAA criteria, with aluminum being the most commonly elevated (70–79% of waste rock samples in both CTEU-9 and SPLP tests). These elevated concentrations are believed to be largely associated with the nature of the leaching tests, which involve prolonged agitation of samples in contact with water and tend to exaggerate colloidal release compared with what would be observed in kinetic tests or in the field. Moreover, isolated exceedances were observed for beryllium and vanadium in approximately 1% of waste rock samples, again more evident in CTEU-9 than in SPLP.

TCLP leach tests were run on Directive 019 criteria for high-risk mining waste, and both of the identified exceedances were from the ultramafic waste rock. Two (2) ultramafic samples, previously reported by BBA / Vision Geochemistry (2024a), displayed TCLP-arsenic results above the D019 criterion of 5 mg/L, with values of 5.80 and 7.58 mg/L, respectively. However, since these samples were classified as Non-PAG, they are not subject to ‘high risk’ classification under MELCCFP (2003) criteria.

20.7.2 Kinetic Testing

The kinetic testing program was implemented to quantify the rates of metal leaching and acid generation to better assess the risk for metal leaching and acid generation. Kinetic testing included the continuation of 16 humidity cell tests (HCTs) on waste rock and ore, including one (1) duplicate for data QA/QC. The HCTs were initiated in 2023, and the results from the first 40 weeks of kinetic testing were presented by BBA / Vision Geochemistry (2024a). Updated reporting of the HCT data, up to 100 weeks, was presented by Vision Geochemistry (2025).

Results from the HCTs indicated no clear net acid generation, although slight pH decreases were observed over time. Arsenic showed the most elevated values compared to the criteria, followed by antimony, with ultramafic samples accounting for the majority of the elevated values. Elevated metal concentrations (cadmium, copper, nickel and zinc) were limited, and primarily observed in metasediment, and to a lesser extent, ultramafic samples, while occasional elevated uranium values were observed in pegmatite and ore samples at the onset of the tests.

In early 2025, six (6) column tests with tailings material, including one (1) duplicate, were initiated to assess their risk for metal leaching and acid generation. Results of the column testing up to 20 weeks were reported

by Vision Geochemistry (2025). The tested tailings material included floats, middlings, magnetics, bypass and a master composite. Column tests confirmed that arsenic, antimony, and uranium are the primary COPCs associated with tailings. Arsenic presented elevated concentrations in all tailings' streams, while antimony and uranium values were found elevated in specific streams. Copper, nickel, lithium, and zinc showed only punctual exceedances at the onset of some column tests.

20.7.3 COPC Screening and Stockpile Modelling

A conservative COPC screening effort was conducted to assess whether the elevated concentrations measured via static and kinetic testing could translate into potential leaching risks at Stockpile 001, Stockpile 002, and the Ore Stockpile, in comparison with the CVAA and MDMER/D019 guidelines. The screening results indicate that antimony, arsenic, lithium, and uranium are likely to present leaching risks at Stockpile 002, with the tailings material playing a primary role in elevating these values, regardless of the development of acidic conditions. If acidic conditions develop in Stockpile 002, additional exceedances may occur for certain metals, notably aluminum, cadmium, copper, and nickel. For the Ore Stockpile, the COPC screening highlighted antimony, arsenic, and lithium as potentially exceeding the criteria.

Geochemical modelling of Stockpile 001 and Stockpile 002 was executed to better understand the evolution and COPC peak concentration estimates for arsenic, antimony and lithium, during mine operations and into mine closure (Vision Geochemistry, 2025). Modelling was performed using kinetic data scaled to field conditions and paired with a random distribution of waste rock characteristics drawn from the exploration drillhole database (DHDB), containing 4,678 waste rock samples. The models were designed to predict the evolution of COPC concentrations in waste rock and tailings contact water at Stockpile 001 and Stockpile 002, to support the design of mitigation measures and treatment systems.

Previously reported stockpile modelling results presented by BBA / Vision Geochemistry (2024b) showed that co-deposition of all waste rock in the same stockpiles results in contact water with arsenic concentrations exceeding the MELCCFP resurgence in surface water criteria, and therefore segregation of ultramafic waste rocks was recommended. However, recent modelling results by Vision Geochemistry (2025) incorporated the updated FS design dimensions of the stockpiles and determined that a more conservative segregation protocol was required to prevent arsenic leaching. The revised segregation protocol included the complete separation of ultramafic lithology, paired with the separation of all material with arsenic concentrations above 30 ppm, to maintain arsenic and antimony concentrations below the CVAA and MDMER/D019 criteria in Stockpile 001. Conversely, leaching conditions were identified at Stockpile 002, with arsenic concentrations exceeding the MDMER/D019 criteria by up to two (2) orders of magnitude, primarily driven by the contribution of the tailings. Similarly, antimony and, to a lesser extent, lithium are expected to leach from Stockpile 002, largely contributed from the tailings.

The outcome of the COPC screening effort, which integrates static testing, kinetic testing, and geochemical modelling in terms of peak concentration estimates for Stockpile 001, Stockpile 002 and the Ore Stockpile, is summarized in Table 20.8.

Table 20.8: Summary of COPC Peak Concentration Estimates in Stockpile-001, Stockpile-002, and Ore Stockpile

Parameter	Unit	CVAA / MDMER[1]	Stockpile-001		Stockpile-002				Ore Stockpile	
			Neutral Conditions		Neutral Conditions		Acidic Conditions		Neutral Conditions	
			Concentration	Ratio to Criterion	Concentration	Ratio to Criterion	Concentration	Ratio to Criterion	Concentration	Ratio to Criterion
Aluminum	mg/l	0.28	0.063	0.22	0.116	0.41	**0.790**	**2.8**	0.059	0.21
Antimony	mg/l	0.31	0.136	0.44	**3.04**	**9.8**	**3.04**	**9.8**	**3.40**	**11.0**
Arsenic	mg/l	0.1	0.085	0.85	**10.42**	**104**	**10.42**	**104**	**3.08**	**30.8**
Cadmium	mg/l	0.00021	0.0001	0.35	0.00005	0.26	**0.0007**	**3.5**	0.000005	0.02
Copper	mg/l	0.0016	0.001	0.75	0.0015	0.92	**0.015**	**10**	0.0012	0.75
Lithium	mg/l	0.91	0.077	0.085	**2.23**	**2.45**	**2.23**	**2.45**	**10.48**	**11.5**
Nickel	mg/l	0.067	0.007	0.11	0.0184	0.27	**0.29**	**4.3**	0.0005	0.01
Uranium	mg/l	0.017	0.002	0.09	**0.38**	**22**	**0.38**	**22**	0.0116	0.68

**Note: (1) The most conservative value was selected of the two criteria (CVAA for aluminum, cadmium, copper and nickel; MDMER for iron).*
Colour code: yellow = Static testing screening; green = kinetic testing screening; blue = stockpile modelling.
Bold text: concentrations exceed the CVAA/MDMER criteria.

20.8 Rehabilitation and Closure Planning

The Project Closure Phase is designed to ensure a responsible transition from active operations to post-mining land use, in collaboration with First Nation communities and in compliance with Quebec's regulations from the Ministry of Natural Resources and Forests (MRNF) and the Ministry of the Environment, and the Fight Against Climate Change, Wildlife and Parks (MELCCFP). The main goals of closure and rehabilitation activities will be the following:

- Eliminate unacceptable health hazards and ensure public safety.
- Limit the production and spread of contaminants that may affect the receiving environment and, in the long term, aim to eliminate any form of maintenance and monitoring.
- Ensure long-term geotechnical and geochemical stability.
- Allow progressive restoration.
- Ensuring social acceptability.
- Return the site to a condition that is, as much as possible, in line with the initial or surrounding ecosystem and is visually acceptable.
- Return infrastructure areas to a state that is compatible with future use.

The site closure plan is being developed as per the provincial guide (*Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec* (MRNF, 2024)).

For planning purposes, closure and reclamation strategies have been developed for each mine component. Following completion of mining, the mine will be closed, reclaimed, and monitored in accordance with its authorizations, regulations, and plans.

20.8.1 Closure Sequence

The closure sequence will be planned in three (3) stages, starting with:

- Progressive restoration works will be carried out during the mining operations in areas that are no longer active as a means of verifying the success of larger-scale efforts that will take place during the mine closure phase.
- Once operations are completed, the post-operation phase begins, which includes the dismantling of the industrial facilities and the execution of the closure works.

- Once closure works and water treatment infrastructures are dismantled, the post-closure site maintenance and monitoring will begin.

The infrastructures to be restored are presented in the next sections.

20.8.2 Industrial Zone Closure and Linear Infrastructure

All processing plants, power facilities, administrative buildings, and other industrial structures, infrastructures, and equipment will be dismantled and removed from the site for disposal at an authorized landfill. Materials generated during the dismantling process will be managed by applying the principles of reduction, reuse, recycling, and reclamation. Concrete slabs and foundations will be broken or cracked to facilitate drainage, then covered with 15 cm of topsoil-like material before being revegetated with hydroseeding. Access roads, haul roads and former storage or laydown areas on the mine site will be scarified to restore drainage and support revegetation.

Note that the 20-km all-season access road from the Trans-Taiga Road is considered a multi-user road and will remain open for use by the public and Cree community members throughout operations and after mine closure.

The Matagami transshipment site will be owned and operated by the Town of Matagami, and therefore, closure of these facilities will be their prerogative and may remain in place for other users in the future. No cost has been included for closure.

Discussions with shareholders and local communities are ongoing to determine the future of the planned First Nations Cultural Centre on site, as well as the electric power line. PMET intends to transfer ownership to the community; therefore, the cost of dismantling the electrical line and cultural centre is not included.

20.8.3 Mine Infrastructure: Pit, Waste Rock, Tailings and Overburden Storage Facilities

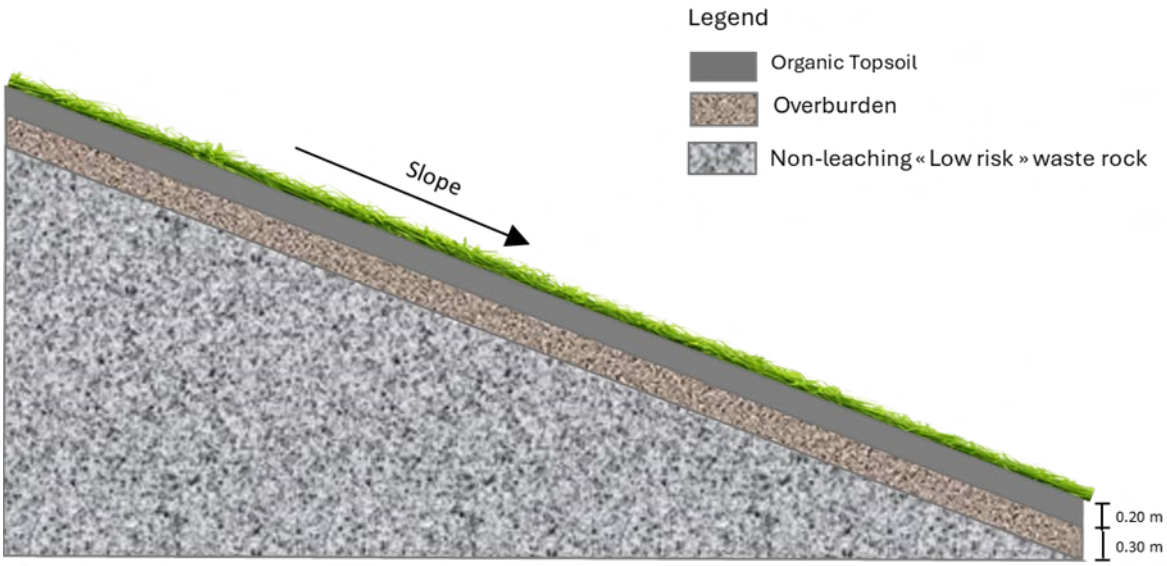
Closure and reclamation strategies developed to date are the following: comparative analysis sessions for infrastructure closure scenarios (for pit, Stockpiles 001 and 002) were carried out as part of the preparation of the site closure plan to determine the best concepts for each of these infrastructures. Various closure concepts have been retained. The closure activities for each of these infrastructures are presented below.

20.8.3.1 Stockpile 001

Stockpile 001 will be composed of a portion of the "low-risk" waste rock (Vision Geochemistry, 2025), while the remaining low-risk waste rock will be stored in the open pit. The Stockpile 001 closure activities include the following:

- The Stockpile 001 surface would be reprofiled to allow efficient drainage towards the collection water ditches.
- A 30-cm layer of overburden would be placed on the waste rock after reprofiling to prevent topsoil from flowing through the waste rock voids.
- Addition of a 20-cm layer of topsoil on the granular material. Then, the surface of Stockpile 001 would be vegetated to create a closed landscape and mitigate erosion.

Figure 20.1: Stockpile 001 Closure Configuration



It is important to note that this cover concept for Stockpile 001 applies only to waste rock which are classified as low risk (MELCCFP, 2025). Should this geochemical classification of the waste rock change with ongoing geochemistry studies (Vision Geochemistry), a new cover concept will have to be designed.

20.8.3.2 Stockpile 002

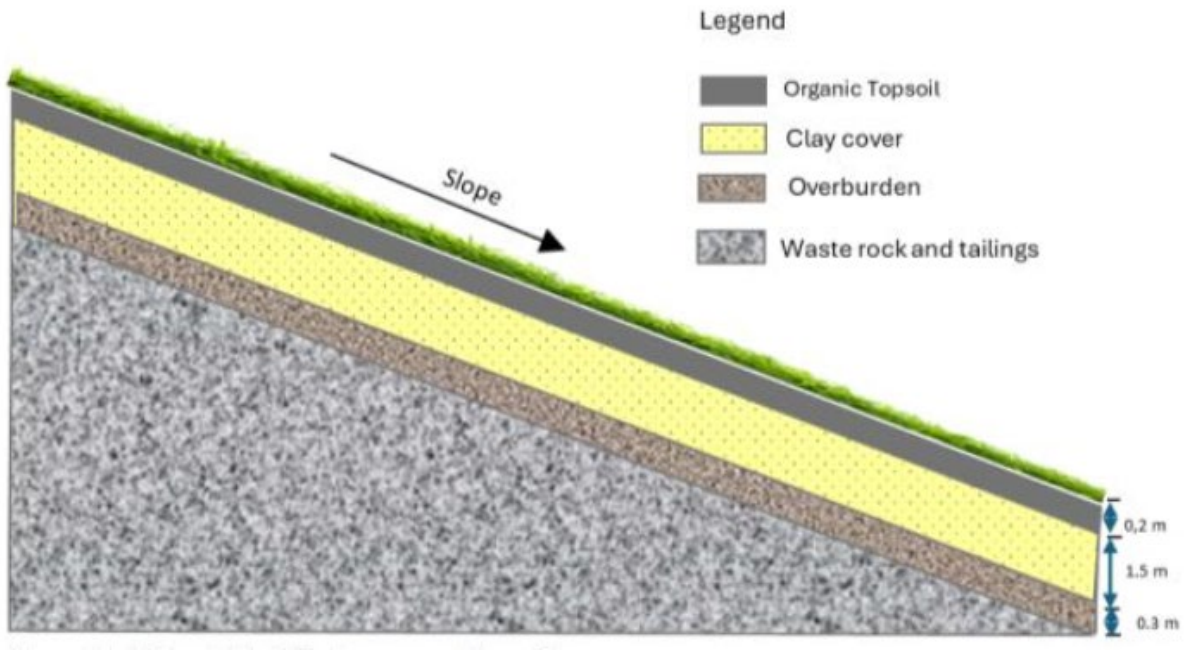
Stockpile 002 will receive all the potentially metal-leaching (ML), high-risk, or acid generating (PAG) waste rock. Due to the nature of the rocks to be stored in Stockpile 002, the closure concept chosen for Stockpile 002 is a clay cover. This concept consists of the installation of a barrier to water infiltration on

waste rock and tailings using natural materials. The role of this impermeable layer, made up of 1.5 m thick clay, is to limit water infiltration and oxygen diffusion to the underlying materials (PAG or PML waste rock) to avoid the generation of contaminants. Indeed, this layer, offering low hydraulic conductivity, acts as a water retention layer and limits oxygen diffusion to the underlying materials. No investigations have been completed to identify potential clay sources. However, it is therefore assumed that a borrow pit investigation will be carried out in the next phases of the Project. Otherwise, another concept will have to be developed.

The Stockpile 002 surface would be reprofiled to allow the cover system placement and the efficient drainage towards the water collection ditches. The general configuration of this concept includes, from bottom to top:

- A 30-cm layer of granular material (overburden) would be placed on the waste rock after reprofiling.
- A clay cover (1.5 m thick) will then be put in place.
- A 20-cm layer of topsoil as support for vegetation will be placed on the clay cover. Then, the surface of Stockpile 002 would be vegetated to create a closed landscape and mitigate erosion.
- A drainage system to evacuate percolation water through the layers of natural covering materials to networks of peripheral ditches.

Figure 20.2: Stockpile 002 Closure Configuration



20.8.3.3 Organics and Overburden Piles 001, 004 and 005

Topsoil and overburden piles (001, 004 and 005, respectively) will be used for closure activities. Once the material has been utilized, the remaining footprint of these stockpiles will be scarified to break up compacted surfaces and promote natural drainage. These areas will then be revegetated using hydroseeding, which facilitates rapid vegetation establishment and erosion control.

20.8.3.4 Open Pit

20.8.3.4.1 Flooding and Site Securing

Under this closure strategy, the open pit will be allowed to flood naturally, resulting in the formation of a stable pit lake that integrates with the local hydrogeological system. The pit closure activities are

- Passive pit flooding: Allow natural runoff and groundwater inflows to flood the pit up to the current water level in Lake 01 (El. 373 m).
- Securing the pit by constructing a safety berm around it to prevent animals and unauthorized persons from entering the flooded pit.
- Dismantle the small water retention berm.
- The pit lake will be connected to the existing natural channel that discharges to the north.
- Promote the establishment of aquatic habitat through features such as low-flow channels.

20.8.3.4.2 Access Roads

Closure and reclamation strategies developed are the following:

- Decommissioning and Removal: All access roads will be fully decommissioned, including:
  - Removal of culverts and drainage structures.
  - Restoration of natural hydrological flow.
- Rehabilitation Measures:
  - Recontouring of the roadbed to match the surrounding topography.
  - Revegetation using native plant species to prevent erosion.

20.8.3.4.3 Restoration of the Water Management System

Once the site has been restored and the water quality has been confirmed, the four (4) ponds located within the water treatment area, as well as the two (2) ponds in the industrial area, will be backfilled. The areas will be reprofiled and revegetated using hydroseeding. Before any earthwork begins, the sediments accumulated at the bottom of the ponds will be characterized, excavated, and either stored in Stockpile 002 or transported to an authorized disposal site. The geomembrane will be removed to allow drainage.

As for the four (4) ponds collecting water from Stockpiles 001 and 002, a breach in each of the dikes will be excavated to ensure there is no water accumulation at the toe of the stockpiles. Sediments collected at the bottom of these ponds will also be removed and managed appropriately.

20.8.4 Monitoring and Maintenance Programs

A site characterization study will be carried out at closure to identify the presence of contaminants exceeding regulatory thresholds and to implement necessary measures, in accordance with the provisions of the *Environment Quality Act* and the *Land Protection and Rehabilitation Regulation*. The closure cost estimate also includes a provision for the management of contaminated soils.

Long-term considerations include monitoring programs to assess geochemical stability and potential settlement risks, to ensure that restored areas do not pose long-term contamination or subsidence issues. The implementation of an environmental monitoring program will demonstrate that the restoration works have achieved their goals. Based on the 2025 edition of Directive 019 from the MELCCFP, the following is a summary of the post-operation and post-closure environmental monitoring requirements:

20.8.4.1 Post-Operation

20.8.4.1.1 Effluent Monitoring Requirements

Final effluent must be monitored weekly during the restoration phase. Monitoring includes:

- Water quality sampling at the discharge point.
- Flow rate and pH measurements, preferably with continuous monitoring equipment.
- Data transmission to MELCCFP as per prescribed formats and frequency.

20.8.4.1.2 Groundwater and Runoff Management

- Groundwater must be sampled twice per year during restoration and post-restoration phases.
- Runoff from stockpiles must be collected and treated at the water treatment facility until water quality objectives are met.

20.8.4.1.3 Post-Restoration Monitoring

- Once the site is restored and water quality confirmed:
  - Breaches will be created in the stockpile collection ponds.
  - This will result in four (4) effluents to monitor.
- Monthly monitoring (during open water season) of these effluents is required for 20 years post-restoration.
- Groundwater monitoring continues twice per year during this period.
- Mine water and final effluent monitoring continues eight (8) times per year during this period.
- The effectiveness of revegetation will be monitored.

20.8.4.1.4 Rehabilitation and Closure Planning

A closure plan, in accordance with the Guide for Preparing Mine Site Rehabilitation and Restoration Plans in Quebec (MRNF, 2024), is currently underway by GCM Consulting and will include a detailed schedule of closure work. The mine closure plan outlines the steps that PMET will take to safely decommission the mine once operations have ceased, ensuring all environmental impacts are mitigated and the site is restored to safe and stable conditions.

20.8.4.1.5 Transshipment Station

At the Matagami transshipment site, installations (rails, yards and storage buildings) dedicated to the Project will be owned and operated by the Town of Matagami and therefore, closure of these facilities will be their prerogative and may remain in place for other users in the future. No cost has been included for closure.

20.8.5 Cost Estimation and Financial Guarantee

The total cost of reclamation (and the guarantee) is estimated at $248.4M. This cost includes the direct and indirect costs of site rehabilitation as well as post-closure monitoring, engineering costs (30%) and the mandatory 15% contingency. A detailed cost estimate will be developed throughout the closure planning process in accordance with the Guidelines for Preparing Mine Closure Plans in Québec.

A financial bond corresponding to the total anticipated cost of completing all the work set forth in the rehabilitation and restoration plan will be provided to the Minister of Finance of Québec as required by the regulations.

20.9 GHG Emissions

Greenhouse gas emissions for the Project were estimated on the basis of available preliminary technical data. At this stage of the Project, emissions were estimated for the construction and operations phases only. Total construction-related emissions are estimated at 95 kt CO_2-eq, mainly due to site clearing.

Annual direct emissions related to site operation are estimated to range from a minimum of 24 kt CO_2-eq in the first year of operation to a maximum of approximately 145 kt CO_2-eq in 2036. The annual average over the entire operation period is 129 kt CO_2-eq.

These estimates do not include off-site logistics transportation.

20.10 Community Relations

The following section is an overview of the engagement approach with Indigenous groups, local communities and authorities. Detailed information on community relations activities will be provided in the upcoming ESIA report.

20.10.1 Consultation Activities

As part of the design of the Project, PMET organized information sessions, beginning in 2022 and intensifying throughout 2023, 2024 and 2025. All stakeholder communication activities are tracked in PMET's stakeholder consultation log, which includes date, parties present, meeting objectives and key comments and questions. To identify stakeholders potentially affected by the Project, PMET undertook an initial scoping based on publicly available information and focused on stakeholders in the Eeyou Istchee James Bay Territory, as well as on officials of the various levels of government. PMET used the advice and

mechanisms set out by the local communities in the management of the territory and the existing inter-communities' network and structure. Stakeholder mapping is revisited as the Project evolves, new stakeholders are identified, and some of the previously identified stakeholders have a lesser or greater desire to be involved. The consultation approach was tailored based on the recommendations gathered from the key stakeholders.

20.10.1.1 Consultation Process

Recognizing the importance of involving Indigenous groups, local communities and authorities, interest groups, and land users in the design, planning and development of the Project, the main objective of these sessions was to contextualize the Project within its environment and gather preliminary concerns, recommendations, and interests from stakeholders.

It should be noted that in 2023, PMET started the ECOLOGO UL 2723 certification program for mining exploration companies. The purpose of this certification is to audit exploration companies and their service providers to ensure the application of recommended social, environmental, and economic practices. PMET completed the audit process in April 2025.

The consultation and mobilization program aims to meet the following objectives:

- Encourage transparent, proactive, and effective communication between PMET, host communities and all Project stakeholders.
- Increase the sharing of information about the Project and ensure adequate accountability for associated activities.
- Gather information related to the land use, culture, and traditions of local and Indigenous communities affected by the Project.
- Identify the concerns of stakeholders and the local realities, as well as potential challenges related to Project realization.
- Take a position on the concerns expressed, correct misperceptions when needed, and make the necessary commitments to answer the questions, comments, and issues about the Project.
- Develop a sustainable relationship of trust with the various Indigenous groups and other stakeholders.

Through its consultation and mobilization approach, PMET wishes to offer local communities the opportunity to participate proactively in the planning and monitoring of the Project. The information

gathered, especially the traditional knowledge of Indigenous groups, will thus be integrated into the design and impact analysis.

Various communication channels have been used to establish and maintain dialogue with authorities, stakeholders, and Indigenous groups since 2023. These include the following:

- Written communications (e-mails, letters, newsletters, fact sheets).
- Verbal communications (telephone interviews, videoconferencing).
- Video, website and social media posts.
- Face-to-face meetings.
- Public events.
- Radio broadcast.
- Community digital board.
- Site visits.
- Working group.
- Community Liaison Office.

The stakeholders targeted in this prior information process are presented in Table 20.9

Table 20.9: Stakeholders Targeted as Part of the Prior Information Process

Category	Stakeholders
Indigenous Communities and Regional Organizations	• Cree Nation Government • Band Council of Chisasibi • Cree Nation of Chisasibi • Cree Nation of Wemindji • Cree Nation of Mistissini • Main Land Users (trapline owners / families) • Cree Trappers' Association (CTA) • Chisasibi Business Development Group and Business Community • Chisasibi Eeyou Resource and Research Institute • Cree Health Board • Cree School Board

Category	Stakeholders
Provincial and Federal	• Ministère des Ressources naturelles et des Forêts • Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs • Ministère de l'Économie, de l'Innovation et de l'Énergie • Secrétariat aux relations avec les Premières Nations et les Inuits • Impact Assessment Agency of Canada • Environment and Climate Change Canada • Transport Canada • Fisheries and Oceans Canada
Municipal	• Eeyou Istchee James Bay Regional Government • Locality of Radisson • Ville de Matagami
Economic	• Hydro-Québec • Société du Plan Nord • Société de développement de la Baie-James • Carrefour jeunesse-emploi de la Jamésie • Local Entrepreneurs • Other Companies from the Mining Industry
Health and Education	• Centre d'études collégiales à Chibougamau • Centre régional de santé et des services sociaux de la Baie-James • Centre de santé de Radisson
Recreation and Tourism	• Leisure Lease Holders • Tourisme Baie-James • Mirage Aventure • Camping Radisson • Club de motoneige Radisson
Other Organizations	• Sûreté du Québec • Centre de femmes Uni-Vers-Elles • Centraide Abitibi-Témiscamingue et Nord-du-Québec • Solidarité alimentaire Matagami

With a particular focus on Indigenous groups, the Project was presented to initiate communication, explore points of interest, and begin the development of a clear understanding of the issues of concern of the land users. Questions were asked, concerns were expressed, and suggestions were made by land users. Detailed land-use documentation will be developed and integrated as part of the ESIA.

Land users are interested in partnering and collaborating with PMET on the various activities and work to be carried out on the land during the study phase. In general, the Indigenous parties insisted on the importance of their upstream involvement in planning activities to be done on the trapline as well as during the preparation of the Project's environmental and social assessment.

20.10.1.2 Main Concerns

Engagement activities identified issues and concerns related to the Project. The following information describes the initial thoughts and concerns expressed by the stakeholders should not be regarded as definitive. More detailed information on the concerns and PMET responses will be integrated into the upcoming ESIA report.

20.10.1.2.1 Main Concerns of Indigenous Groups

The meetings held so far have identified the preliminary concerns shared by Indigenous groups. The main concerns expressed at these meetings are outlined in Table 20.10.

Table 20.10: Main Comments and Concerns Expressed by Indigenous Groups during Engagement and Consultation Activities

Theme	Comment / Concern
Water and Fish Habitat	• Impact of mine effluent on the environment. • Mine surface water / runoff management. • Water requirements to supply the mine. • Water protection and how water will be treated by the proponent. • Cumulative impacts on fish habitat.
Traditional Use of Land and Resources	• Disruption of traditional activities (hunting, fishing, trapping, berry picking, etc.) throughout the mine life cycle (construction, operation and closure). • Impact on air, water, and soil quality, as well as on plants and animals.
Cumulative Impacts	• So far, the cumulative impact of disturbances on traplines is believed to have depleted the resource. • Increased activity on the Trans-Taiga Road is influencing the decline in moose and caribou numbers in the area. • Hydroelectric development has impacted water levels. • Land users have mentioned that due to the creation of Hydro-Québec reservoirs, fish quality has declined considerably. • The various projects in Eeyou Istchee James Bay are increasing pressure on the region's only road access, the Billy-Diamond Road. • Pressure on health services from various development projects.

Theme	Comment / Concern
Local and Regional Economy	• Indigenous communities want to benefit from the opportunities the Project offers and not only suffer the negative impacts. • Land users would be interested in partnering with the proponent in the various activities and works to be carried out on the land as the Project's environmental and social assessment progresses. • Comments were made on the importance of addressing future training and employment / contract opportunities within affected families and the community. It was also made clear that the current spirit of collaboration in these early stages of the Project does not translate into acceptance or approval of the Project. • Community representatives have questions about the Project schedule. • Stakeholders would like to know more about the economics of the lithium mining industry. • Suggestion that PMET hire an Indigenous liaison officer to facilitate participation of the Cree community members (information sharing, jobs, contracts, etc.). • The importance of drawing up a list of training needs and the jobs that will be available was emphasized. • Issues concerning certain hiring criteria deemed too high, particularly concerning the French language. • Establishment of education agreements.
Communication and Consultation Processes	• All the Eeyouch (James Bay Cree) we met felt it was important to establish a relationship of trust. • For consultation events, the Crees favour the World Café method as the most productive. • For the Crees, particularly the main users of the territory, the best communication tools for reaching communities would be local radio and television. • Reach out to the younger generation by expanding communication channels, including the web, social media, site visits and direct invitations to students to participate in information and consultation events. • It is important to translate documentation and include Cree words when possible.
Transport	• Risk of accidents / collisions caused by increased traffic. • Impact of increased traffic on large wildlife.

Theme	Comment / Concern
Health and Quality of Life	• Equity in employment and career development. • Systemic racism problems. • Cultural safety (way of life, language, spirituality, cultural sites, traditional food, etc.). • Greater openness of the territory comes with the risk of increased human trafficking (disappearance of Indigenous women). • Risk to workers' and land users' health and safety. • Forest fire measures and evacuation plans. • Difficulties in reconciling work and family life (rotating work schedules). • Competition with local services for labour. • Risks and failures related to site operations (exceptional events).
Regulation	• Many questions were raised to understand the authorization and consultation process, and the environmental protection regime included in the JBNQA.

**Note: Details on the consultations can be found in the initial Project description on the Government of Canada website: https://www.iaac-aeic.gc.ca/050/documents/p89271/160847F.pdf (Métaux de Batterie Patriot Inc. 2025. Projet minier Shaakichiuwaanaan – Description initiale de projet. Eeyou Istchee Baie-James, Nord-du-Québec, Québec, janvier 2025.)*

PMET respects the rights of Indigenous peoples and will continue to consider their interests, aspirations, and culture in the design, development, and operation of the Project. The Company's commitment is centred on four (4) priorities:

- Working to reduce impacts at the source, prevent them and avoid them when possible. This includes impacts set out in Table 20.10 and Table 20.11.
- Maximize positive spin-offs and benefits for parties affected by the Project.
- Co-define, with the community, the conditions to be put in place for the Project to integrate harmoniously with the environment.
- Execute an in-depth approach to address elements of concern or interest to stakeholders in a spirit of collaboration and take them into account in developing the Project.

20.10.1.2.2 Main Concerns of Non-Indigenous Groups

The main comments and concerns of non-Indigenous stakeholders expressed so far during the various consultation activities presented in the previous section are summarized in Table 20.11.

Table 20.11: Main Comments and Concerns Expressed during Consultation Activities with Non-indigenous Stakeholders

Theme	Comment / Concern
Local and Regional Economy	• Radisson and Matagami would like to take advantage of the opportunities offered by the Project to benefit from local and regional economic spin-offs. • Risks associated with fluctuating lithium prices. • Processing opportunities in Québec. • Project energy requirements.
Communication and Consultation Processes	• Initiate dialogue with the Radisson population with a Project presentation session. • Promote an inclusive approach to achieve greater social acceptability.
Transport	• The various projects in the Eeyou Istchee James Bay region are increasing pressure on the region's only road access, the Billy-Diamond Road. • Increased traffic may translate into more collisions on the Billy-Diamond Road.
Health and Quality of Life	• How to maximize the positive impact of the Project to attract new residents to Radisson. • Measures to reduce commuting (Fly-In / Fly-Out). • Forest fire measures and evacuation plan.
Regulations	• Compliance with laws and authorization processes at various levels of government, including municipal regulations.

**Note: Details on the consultations can be found in the initial Project description on the Government of Canada website: https://www.iaac-aeic.gc.ca/050/documents/p89271/160847F.pdf (Métaux de Batterie Patriot Inc. 2025. Projet minier Shaakichiuwaanaan – Description initiale de projet. Eeyou Istchee Baie-James, Nord-du-Québec, Québec, janvier 2025.)*

20.10.1.3 Future Engagement Plan

PMET has developed the engagement plan based on collaboration with the stakeholders potentially affected by the Project. The engagement approach is flexible to allow for adaptation according to feedback received. PMET will continue its outreach and consultation activities with the stakeholders.

20.10.1.3.1 Indigenous Groups

With a view to maintaining a strong and ongoing relationship with the Indigenous groups affected by the Project, PMET will continue to set up adapted, concerted information and consultation processes with Indigenous groups, and establish mutual collaboration and partnership agreements with them.

To this end, PMET is working with the communities to execute a consultation, communication and engagement plan that will include ongoing Project updates. This plan aims to continue gathering the concerns and interests of Indigenous groups, particularly those relating to environmental issues, land use, employment, training opportunities, service provision, and other potential collaborations.

Through this approach, PMET will continue to seek to understand the opinions and concerns of Indigenous groups, and to openly discuss and record these communication activities. The Company encourages open dialogue, both formally and informally, to give the involved communities the opportunity to express their opinions and concerns about the Project. The outcome of these discussions with Indigenous groups will enable the Project to address their concerns and interests and optimize its social acceptability.

It should be noted that the Project details have been presented to the members of the Chisasibi community, notably at meetings and a public event, including the tallyman's family, band council members, and the community at large throughout 2023, 2024 and 2025. A video summarizing the Project Description is available in Cree, English, and French on the PMET website.

Other key topics of discussion among Indigenous groups include:

- Closure planning.
- Baseline study results.
- Fish habitat compensation planning.
- Water management plan.

PMET will continue its outreach and consultation activities with the stakeholders. PMET’s strategy is to maintain its current approach.

- Regular meetings with Chisasibi leadership and occasional meetings with Mistissini and Wemindji leadership at Project milestones.
- Regular meetings with the CH39 tallyman family members and occasional meetings with VC26 and M02A tallymen and mainland users at Project milestones.
- Chisasibi community information sessions: presentations on Project progress.
- Presence of the Community Liaison Coordinator and leasing of an office in Chisasibi’s commercial center to facilitate consultation, communication, and recruitment.
- Continue its involvement in the Regional Transportation Working Group for Spodumene Concentrate.

- Organize Shaakichiuwaanaan site visits.
- Organize events to educate site workers on Cree culture.
- Maintain up-to-date information on PMET's website.
- Publication of press releases.
- Collaboration with CNC to identify and provide training, employment and business opportunities for community members.

20.10.2 Stakeholder Engagement Approach

The Company's stakeholder engagement approach aims to build meaningful relationships with the Indigenous and non-Indigenous groups and individuals that are likely to be impacted by the Project activities. The Company has taken a proactive approach, holding more than 385 communication activities since January 2022, mainly with the Cree Nation of Chisasibi. The objectives of this approach are to:

- Inform stakeholders likely to be affected of the purpose of the Project as well as its potential socio-economic and environmental effects.
- Consider and address issues raised by stakeholders.
- Document traditional knowledge and land use in the vicinity of the Project.
- Share results of field studies or other relevant studies.
- Gather feedback, information, concerns, questions, suggestions and comments to guide or adapt the design of certain Project components / activities.
- Improve the Project and its social acceptability by involving community members.

The Company is encouraging open discussion, formally and informally, through various ways:

- Regular meetings and calls with key stakeholders (tallyman and family, local leadership and organizations).
- Community information sessions and presentations.
- Using the services of a Cree translator.
- Participation of community members in field inventories.
- Distribution of written documentation on the Project.
- Sending information by letters, e-mails and texts.
- Posting information, videos and press releases on its website.

20.10.3 Agreements

To date, there have been no formal agreements signed with any stakeholders.

In February 2023, PMET, the Cree Nation of Chisasibi, and the Cree Nation Government jointly initiated the development of a communication protocol specific to exploration activities for the Shaakichiwaanaan Project. This protocol defines the framework for collaboration and the relationship between the Company and the Cree partners. It includes mutual commitments regarding communication, environmental monitoring, employment and training, negotiation of Impact and Benefits Agreement (IBA), and business opportunities. Discussions are ongoing regarding the communication protocol, and current Project activities are conducted in alignment with the anticipated conditions of this document.

A formal IBA is expected to be executed with the Grand Council of the Crees / Cree Nation Government (GNC/CNG) and the Cree Nation of Chisasibi community prior to mine operations.

20.11 Preliminary Environment Monitoring Plan

As part of the activities associated with the construction, operation and closure phases of the Shaakichiuwaanaan Project, PMET will develop and implement an environmental monitoring and follow-up program according to discussions with regulators.

An environmental monitoring plan will be designed to ensure compliance with PMET's environmental commitments and obligations, as well as with applicable laws and regulations. Due diligence will be carried out to ensure compliance with the specific clauses stipulated in the government decree for the Project, as well as any other contractual conditions included in the plans and specifications. In addition, the application of mitigation measures proposed in the ESIA Report.

An environmental follow-up plan will be implemented during the construction, operation and closure phases of the Project. The plan will respect the conditions of the Project authorizations. The purpose of this plan is to detect and document any change in the environment from baseline levels (whether related to the Project or not), to verify the accuracy of the impact assessment, and to evaluate the effectiveness of the mitigation measures. During the various stages of the Project, the plan will include components such as:

- Ambient air quality.
- Greenhouse gas emissions.
- Geochemistry of ore, waste rock and tailings.
- Mine effluent compliance.

- Surface water quality.
- Groundwater quality.
- Fish and benthic invertebrates.
- Wildlife and biodiversity.
- Revegetation of affected areas.
- Soil characterization.

A social monitoring plan will also be implemented if required to confirm the conclusions and anticipated social benefits from the Project. The results of both the environmental and social monitoring plans will be reviewed to allow for ongoing improvement of management and mitigation measures and to identify potential opportunities for the local population.

20.12 Preliminary Habitat Compensation Plan

Since encroachment of the Shaakichiuwaanaan Project on fish habitat and wetlands cannot be avoided, PMET has begun working with professional aquatic biologists and a variety of Project stakeholders to develop a fish habitat compensation plan. The following preliminary list of potential compensation projects is under consideration:

- Replacement of culverts on regional roads.
- Restoration of identified watercourses.
- Restoration of borrow pits.
- Restoration of orphan mine sites.
- Restoration of a fish pass that burned in the 2023 forest fires.
- Restoration of a wetland used as an illegal dumping ground.
- Creation of conservation areas to improve ecological connectivity and provide carbon sinks.
- Research project on carbon storage in peatlands affected by a mining project.
- Waterfowl habitat enhancement in wetlands.
- Research project on mercury in the food chain.
- Installation of boat washing stations to prevent the introduction of aquatic exotic species.

Ongoing planning is underway for the fish habitat compensation project. This planning includes finalizing the list of proposed projects, characterizing and confirming the habitat that requires compensation and calculating the compensation units each proposed project could provide. Outreach and communication with the various stakeholders, including the Department of Fisheries and Oceans, is planned to confirm project details, seek research project proposals and confirm land ownership and land use rights in potential project areas prior to finalizing the compensation plan.

21. CAPITAL AND OPERATING COSTS

The capital expenditures (CAPEX) and operating expenditures (OPEX) estimates for the Project cover the construction the pre-production, operation, and closure periods of the Shaakichiuwaanaan mine site. The capital costs are presented in two (2) phases of development: Phase 1 is the development of the open pit mine and the first stage of the process plant, taking the Project through to commercial production of 400 ktpa of concentrate per year. Phase 2 is the development of the underground mine and construction of the second stage of the process plant, increasing the Project's production capacity to 800 ktpa concentrate, and essentially represents the expansion capital. Sustaining capital costs include the open pit mine and underground mine sustaining costs. Costs were estimated using quotes from reputable vendors and contractors known and trusted by G Mining Services (GMS) and other consultants. All capital and operating cost estimates cited in this report are expressed in Canadian dollars as of Q3 2025, unless otherwise indicated.

Figure 21.1 shows the capital expenditure timeline according to Initial capital, Expansion and Sustaining capital, and Closure costs in parallel with site construction phases.

- Initial Capital Expenditure: This includes all expenditures for Phase 1 up to commercial production for the Phase 1 (June 1st, 2030), whereby development of the open-pit mine site and construction of the first stage of the process plant (designed to nominally treat 2.5 Mtpa of lithium ore that corresponds to 400,000 t of lithium concentrate per year) is also complete. Initial capital expenditure also includes 23% of the capital cost of Phase 2 and allocation for the full 800,000 tpy spodumene production. For instance, upstream connections to the Hydro-Québec network and portions of the water treatment system are fully considered in the initial capital required. Initial capital cost totals $1,497.7 million (including $1,218.7 million from initial CAPEX of Phase 1 and $133.3 million from Phase 2 initial CAPEX, plus a contingency of $145.7 million, and excluding the pre-production revenue of $101.7 million). The engineering, early works, construction, pre-production and commissioning periods will be carried out over 41 months (Q1 2027 to Q2 2030) due to certain long-lead items required in the early phase of the Project.
- Expansion and Sustaining Capital Costs: Sustaining capital costs are estimated to be $1,417.0 million, covering two main categories:
  - Site expansion expenditures (Phase 2): This includes expenditures related to the expansion Phase 2, which follows the commencement of commercial production from Phase 1. Phase 2 spans a 22-month period, from June 2030 through March 2032.
  - Life of mine sustaining capital: These costs involve the acquisition, replacement, or major overhaul of assets necessary to maintain ongoing operations. The complete sustaining costs period spans from June 2030 to June 2049.

- Closure Costs: Closure costs include all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs total $248.4 million, including a general 15% contingency plus an additional $5.7M cost for clay covering contingency. Note that closure costs are excluded from the Capital Cost grand total.
- Operating Costs: The operating cost estimate is broken down as follows:
  - Open pit and underground mining (drill and blast, load and haul, geology, power, fuel, maintenance, dewatering, electric cable handling, other).
  - Processing (crushing and screening, storage and reclaim, DMS, concentrate handling, tailings handling, ore feed, maintenance, power and others).
  - General & Administration (office, camp, health and safety, community, environment, other).

Capital and sustaining expenditures are summarized in Table 21.1 according to the Level 1 work breakdown structure (WBS).

Table 21.1: Initial and Expansion & Sustaining Capital Expenditures Summary

Capital Expenditure	Phase 1 OP Initial Capital Cost ($M)	Phase 2 UG Initial Capital Cost ($M)	Initial Capital Cost ($M)	Phase 2 UG Expansion Capital Cost ($M)	Total Devel. Capital Cost ($M)	LOM Sustaining Capital Cost ($M)	Total Capital Cost ($M)
100 - Infrastructure	124.9	-	**124.9**	24.8	**149.7**	30.8	**180.5**
200 - Power and Electrical	173.8	-	**173.8**	46.2	**220.0**	25.0	**245.1**
300 - Water Management	128.2	-	**128.2**	18.7	**146.9**	100.5	**247.4**
400 - Surface Operations	18.6	-	**18.6**	-	**18.6**	11.9	**30.5**
500 - Mining	120.0	99.1	**219.1**	36.4	**255.5**	550.5	**806.0**
600 - Process Plant	217.3	20.1	**237.4**	167.0	**404.4**	-	**404.4**
700 - Construction Indirect	262.8	0.1	**262.9**	123.8	**386.7**	-	**386.7**
800 - General Services / Owner's Cost	99.8	4.7	**104.5**	13.4	**117.9**	31.6	**149.6**
900 - Pre-production, Start-up, Comm.	73.3	9.3	**82.6**	1.5	**84.1**	186.1	**270.2**
Total Initial Capital Expenditures Excluding Contingency	**1,218.7**	**133.3**	**1,352.0**	**431.8**	**1,783.8**	**936.4**	**2,720.2**
990 - Contingency	130.7	15.0	**145.7**	48.7	**194.4**	-	**194.4**
Total Initial Capital Expenditures	**1,349.4**	**148.3**	**1,497.7**	**480.5**	**1,978.2**	**936.4**	**2,914.6**
Less: Pre-Prod. Credit Net of TC / RC & Royalties	**(101.7)**	-	**(101.7)**	-	**(101.7)**	-	**(101.7)**
Total Initial CAPEX Net of Pre-Production Crédit	**1,247.7**	**148.3**	**1,396.0**	**480.5**	**1,876.5**	**936.4**	**2,813.0**

Figure 21.1 shows the capital expenditure timeline according to Initial capital, Expansion and Sustaining capital, and Closure costs. Initial capital includes all pre-production expenditures up to June 1, 2030, covering plant construction, equipment, and infrastructure. Expansion capital represents the capital expenditure post June 1, 2030, to complete Phase 2 development works and increase production to 800 ktpa concentrate. Sustaining capital represents ongoing investments required during the 19-year mine life to maintain production capacity and operational efficiency.

Figure 21.1: Initial and Expansion & Sustaining Capital Expenditures Timeline

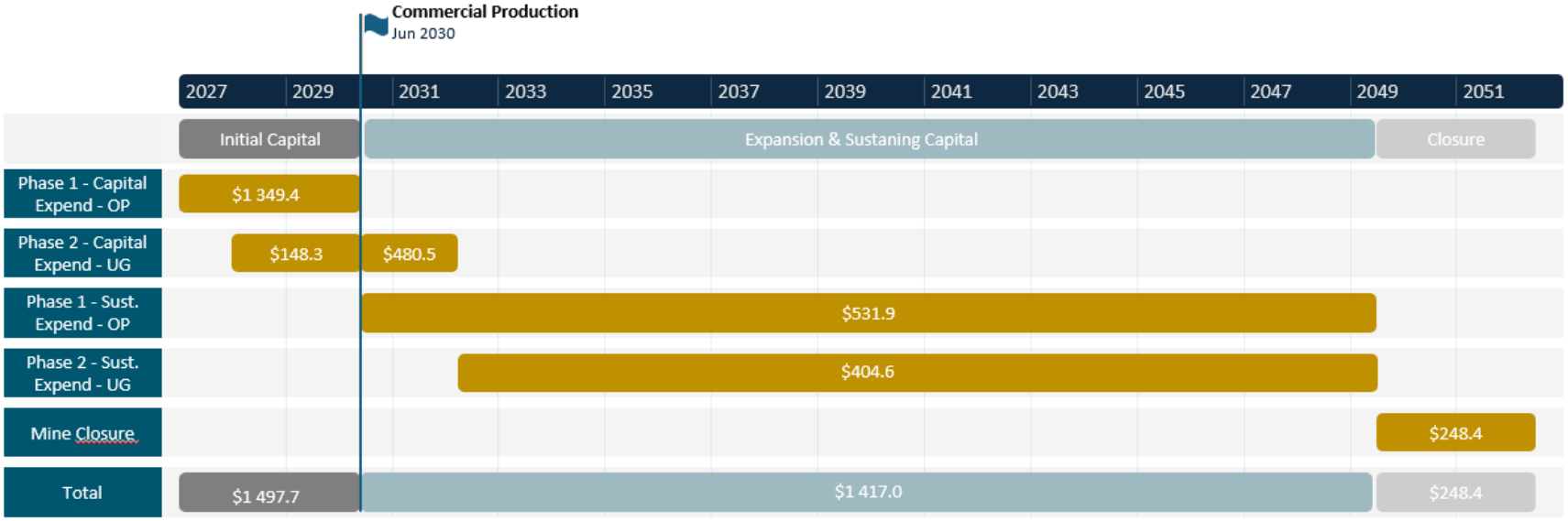
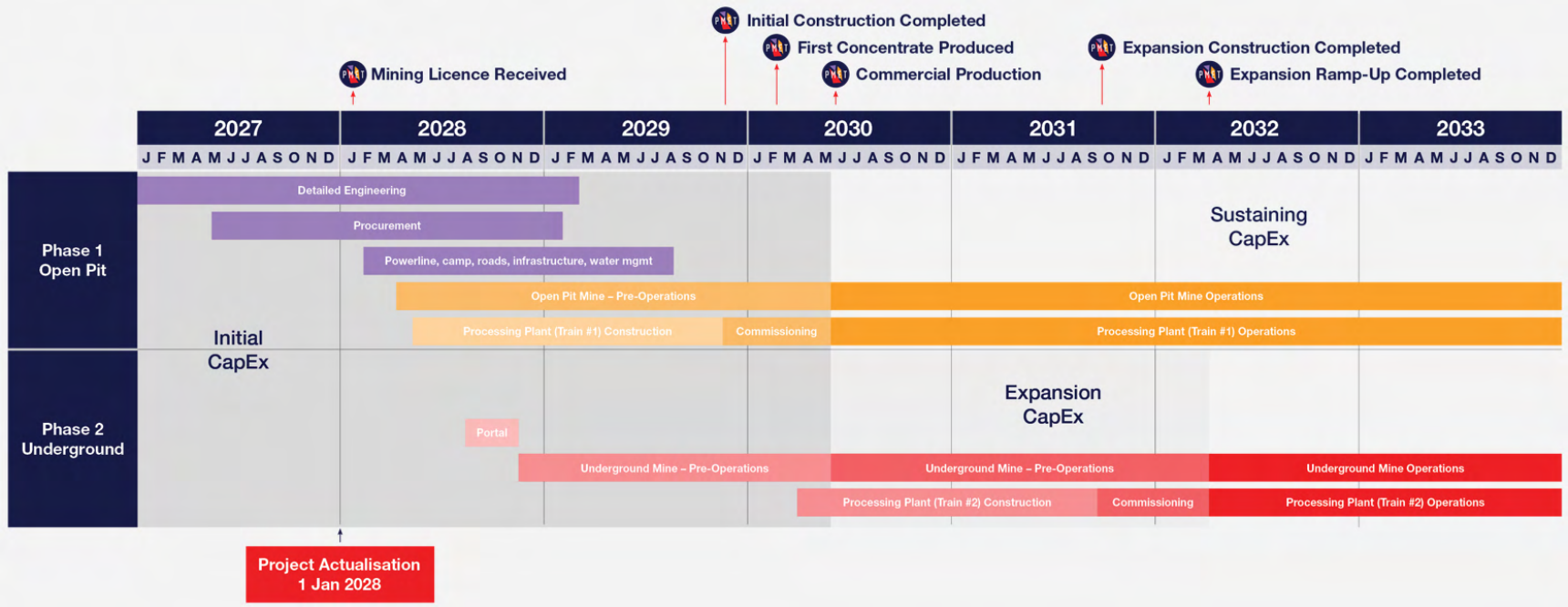


Figure 21.2 focuses on the construction period for Phase 1 (open pit) and Phase 2 (underground) and their relation to initial, expansion costs and the transition to early sustaining costs. It also defines the milestone dates for the Project.

Figure 21.2: Timeline for Construction Phases 1 and 2



The all-in sustaining costs (AISC) include open pit and underground mining costs, processing, general services and administration (G&A), concentrate transportation, and sustaining expenditures. The average OPEX is $729 per tonne of concentrate, including site costs and concentrate transport costs. The all-in sustaining costs (AISC), which include sustaining capital costs, average $800 per tonne of concentrate.

Table 21.2: Operating Costs Summary

Operating Costs	Unit Cost ($/t conc)	Operating Cost ($M)
Open Pit Mining Cost	104.9	1,390.4
Underground Mining Cost	215.2	2,852.2
Processing Cost	91.2	1,208.3
General & Administration Cost	100.7	1,334.7
Total Site Cost	511.9	6,785.5
Concentrate Transport Cost	217.2	2,878.8
Total Operating Costs (FOB Grande Anse)	729.1	9,664.3
Sustaining Capital	70.7	936.4
All-In Sustaining Costs (AISC)	799.8	10,600.8

21.1 Capital Expenditures

21.1.1 Basis of Estimate

The capital cost estimate follows the AACEI Class 3 standard and is accurate to a -10% / +20% range. The initial capital expenditure (CAPEX) duration is planned over a period of 41 months, from Q1 2027 to the end of May 2030. The base date of the CAPEX estimate is as of Q3 2025.

A work breakdown structure (WBS) was developed to organize the Project in a logical structure based on function and location. The first-level WBS structure includes the following areas:

- 100 - Infrastructure.
- 200 - Power & Electrical.
- 300 - Water & Tailings Management.
- 400 - Surface Operations.
- 500 - Mining (Open Pit and Underground).
- 600 - Process Plant.

- 700 - Construction Indirects.
- 800 - General Services - Owner's Costs.
- 900 - Pre-production, Start-up, Commissioning & Contingency.

The CAPEX estimate is aligned with a self-perform owner-managed Project delivery model. It is detailed by major facility and discipline based on GMS's in-house database of executed Projects and studies, as well as experience from similar operations. In certain cases, detailed cost breakdowns by cost type, including labour, materials, construction equipment, consumables, and services, were prepared based on material takeoffs from drawings, design concepts and consultant partners. In line with standards defined at the start of the Project, the pricing of equipment, materials, and labour was estimated according to the following guidelines.

The capital cost estimate followed the following principles:

- Equipment proposals for major, high-value and long-lead items were specified and quoted specifically for the Project by the consultants responsible for the area. Equipment prices for minor items were sourced from recent Projects or from GMS's or other consultants' current database.
- Earthworks quantities and deforestation were based on take-offs from the general arrangement drawings with benchmarked unit costs from executed Projects.
- Concrete and structural costs for these infrastructures were based on executed material take-offs (MTOs), using unit costs per cubic metre of concrete and per tonne of steel, respectively. The cost per cubic metre of concrete was updated using purchase order data and material quotations received in Québec. For structural steel, budgetary quotations were obtained from experienced companies with whom GMS has previously executed Projects. These quotations underwent commercial and technical evaluations to determine pricing by steel type.
- Reputable contractors were invited to bid on specific parts of the Project, such as overburden removal and pit grubbing.
- Labour rates are based on Québec's Construction Association (ACQ – Association de la construction du Québec*)* guidelines and applicable factors for Canadian Industrial construction rates.
- Specific material prices were based on quotations received from suppliers such as portal tunnel sheet plates.

The following assumptions apply to the capital cost estimate:

- All equipment and materials will be new.
- The labour rate build-up is based on statutory laws governing worker benefits.
- Fuel Cost: CAD 1.60/L.
- Electricity Cost: CAD 0.0529/kWh.
- Foreign exchange rate: CAD 1.34/USD.
- Workweek of seven (7) days @ 10 hours per day. It should be noted that the first weekly 50 hours are paid at regular time, with the remaining hours paid at double the base salary.
- Rotation schedule of 14 days of work followed by 14 days of rest. Single shift per day.
- Labour rates are fully burdened, i.e. inclusive of salaries, fringe benefits, fees, funds, premiums.
- A local employment rate of 15% has been factored into the construction period.
- Employers' contributions to various plans, as well as income tax and are based on the labour decree in effect in Québec.
- Source of aggregate, adequate for fill / backfill as well as for concrete mix, in sufficient quantity, is located at site. No aggregate will be imported.
- Waste rock from the mine pit and surrounding borrow pit or quarry will be adequate for fill requirements for the ROM pad.
- Transfer of tailings to the TSF will be via 140-ton haul trucks.
- No provision for rework or repair of equipment and material delivered to site.
- No rework to field-erected and installed equipment and material.
- The estimate assumes limited concrete work during cold / winter period requiring heating, i.e. work to occur between November to May.
- The estimate assumes no shortage of skilled trades workers throughout the entire construction phase.
- Mid-point industry salary rates.
- No provision for a potential increase in salaries necessary to attract skilled trades workers.
- Construction contractors’ facilities will be located within a maximum of five (5) minutes’ walking distance from any working point for the whole duration of the Project implementation.

- The construction site will be accessible 24 hours daily and seven (7) days weekly, with sufficient and adequate safety supervision.
- No allowance for time and material type construction contracts.
- Permanent administration offices will be made available in the early stages of the construction phase and used during construction.
- The estimate assumes transportation will be via chartered flights to LG4 Hydro-Québec airport will be obtained by the client.

21.1.2 General

The mining capital and operating cost estimates were developed by GMS to include the mine mobile equipment, i.e. primary, secondary, support, auxiliary and ancillary equipment, as well as pre-production mine development.

Mining infrastructures, including haul roads, mine facilities, as well as explosives storage and ventilation air raise, were developed by GMS. Costs pertaining to the paste plant were estimated by Paterson & Cooke and integrated by GMS as equipment cost and bulk quantities. The capital and operating cost estimates for the process plant were developed by Primero for equipment and bulk quantities and completed with GMS rates and historical values. The tailings, waste rock and overall site water management capital and operating cost estimates were developed by Atkins Realis and integrated by GMS. Costs pertaining to off-site infrastructures, such as the transmission power line and connection to Hydro-Québec's network, were developed by BBA Consultant.

21.1.3 Exclusions

The following items were excluded from the capital cost estimate:

- Risk-related costs.
- Residual value of equipment.
- Equipment financing costs.
- Taxes (included in the financial model).
- Hazardous waste issues, including battery equipment recycling.
- Allowance for escalation or exchange rate fluctuation.

21.1.4 Initial Capital

21.1.4.1 Phase 1 Capital Costs

Phase 1 capital costs are defined as all costs incurred for the open pit mine development, stage 1 of the process plant and all non-process infrastructure required to enable the Project to achieve commercial production. Phase 1 is completed over Years -3 to -1, and this includes all costs associated with building the camp, roads and infrastructure to start the open pit, stockpiles, open pit mining pre-production costs, the first phase of the processing plant and their related indirect costs and contingencies. It is important to highlight that part of the full 800,000 tpy processing capacity is already incorporated into the initial capital infrastructure. For instance, the 120 kV transmission line from Hydro-Québec's network is technically only needed to meet Phase 2 power requirements, but it cannot be implemented in stages. Likewise, the water treatment plant cannot be expanded in Phase 2 without incurring significant additional costs. As a result, several items are fully built during the first phase of construction.

The total Phase 1 capital cost for the Shaakichiuwaanaan Project is estimated to be $1,349.4M (including contingencies, indirect costs and pre-production costs, and excluding $101.7M for pre-production revenue). Phase 1 capital cost summary of the Project is outlined in Table 21.1.

According to the Level 1 Work Breakdown Structure (WBS), Areas 100 to 600 represent the Project's direct costs, while Areas 700 to 900 encompass indirect costs, owner's costs, and pre-production costs.

The cost includes a contingency of $130.7M, which is 11% of the total before contingency. Total labour hours for the initial Phase 1 capital phase are estimated at 1.40M hours.

21.1.4.1.1 100 - Infrastructure

The initial costs for Infrastructure are summarized in Table 21.3. The major contributors are notably the mine infrastructure, such as the truck shop, and the camp facilities. The detailed description of each infrastructure can be found in Chapter 18 of this report.

Table 21.3: Phase 1 Infrastructures Capital Expenditures by WBS

WBS	Items	$M
110	Roads, Bridges and Fencing	26.34
111	Deforestation	1.95
112	Site Roads	3.16
113	External Site Roads	0.22
114	Bridges	20.03
116	Fencing	0.99
120	Mine Infrastructure	28.18
121	Truck Shop (Truck shop, offices, Wash Bay)	14.49
122	Mine Dry (Dry, office)	11.29
125	Truck Scale	0.70
126	Explosive Magazine	0.73
127	Emulsion Building	0.98
130	Support Infrastructure	16.94
131	Administrative Building	8.74
132	Site Guard House	0.66
134	Warehouse	4.27
135	Laydown	0.68
139	Firehall and Clinic	2.60
140	Camp Facilities	44.26
140	Camp Facilities – Earthworks	0.67
141	Camp Dorms	33.45
142	Kitchen	5.61
143	Camp Office / Welcome Centre (Includes Rec Centre)	3.40
144	Laundry	0.56
146	Recycling / Sorting Facility	0.08
147	Domestic Waste	0.01
149	First Nations Cultural Centre	0.49

WBS	Items	$M
160	Process Plant Infrastructure	4.28
160	Process Plant Infrastructure – Earthworks	3.31
163	Metallurgical Lab	0.63
165	Process Plant Office	0.34
170	Fuel Systems Storage	4.90
171	Fuel Depot and Distribution	4.32
175	Gas Facility Camp / DMS	0.58
	100 – Total	**124.91**

21.1.4.1.2 200 - Power and Supply

The initial cost estimates for WBS Area 200 – Power Supply and Electrical are summarized in Table 21.4. Major costs are for the connection to Hydro-Québec network and the transmission line at $117.62M, which accounts for more than 65% of the total power and telecom cost of WBS 200. This cost includes all the installation to reach the power demand of Phase 1 and Phase 2.

Table 21.4: Phase 1 Power Supply and Communications Capital Expenditures by WBS

WBS	Items	$M
210	Main Power Generation	137.17
211	Offsite Substation	38.12
212	Power Transmission Line	79.50
213	Site Main Substation	19.55
220	Secondary Power Generation	11.06
221	Emergency Power Generation	11.06
240	Service Electrical Room	0.70
241	Camp Area E-Room	0.61
243	Fuel Bay E-Room	0.09
260	Process Plant Electrical Rooms	12.68
261	Crushing Electrical Room ***Primero	5.07
262	DMS Circuit Electrical Room ***Primero	7.62
270	MV Distribution O/H Line	6.96
271	MV Distribution O/H Line	6.96

WBS	Items	$M
290	IT Network & Fire Detection	5.26
291	IT Network	2.79
293	Process Control Room ***Primero	0.10
295	Fire Detection Network	2.36
	200 – Total	**173.85**

21.1.4.1.3 300 - Water and Waste Rock Management

Water and waste rock management capital costs were jointly developed by AtkinsRéalis and GMS to capture all required expenses and were based on a material take-off approach, supported by AtkinsRéalis. Capital expenditures summary for water and rock waste management is presented in Table 21.5.

Table 21.5: Phase 1 Water and Waste Rock Management Capital Expenditures by WBS

WBS	Items	$M
310	Fresh Water Intake / Wells ***AtkinsRéalis	4.43
311	Fresh Water Intake in Lake 308 ***AtkinsRéalis	1.88
313	Pumping and Distribution Process Water ***AtkinsRéalis	2.55
320	Water Ponds and Water Management ***AtkinsRéalis	17.45
324	Contact Water Ponds ***AtkinsRéalis	1.12
325	Sedimentation and Polishing Pond ***AtkinsRéalis	8.43
327	Dyke Lake 001 Phase 1 ***AtkinsRéalis	3.04
329	Diversion Channel for Lake 001 (Short) (spillway and ditch) ***AtkinsRéalis	4.87
330	Potable Water	3.69
331	Potable Water Treatment Plant ***AtkinsRéalis	1.84
334	Potable Water Distribution	1.85
340	Sewage	5.97
341	Sewage Treatment Plant ***AtkinsRéalis	4.94
342	Sewage Pumping Discharge and Pipe Line ***AtkinsRéalis	0.23
344	Sewage Lifting Stations and Pipes	0.80
350	Fire Protection	1.58
351	Process Plant Fire Protection	1.58

WBS	Items	$M
360	Effluent Water Treatment ***AtkinsRéalis	21.76
361	Water Treatment Plant Phase 1 ***AtkinsRéalis	9.27
364	Pumps and Pipelines Phase 1 ***AtkinsRéalis	11.91
366	Discharge and Diffuser ***AtkinsRéalis	0.58
370	Waste Rock Stockpile ***AtkinsRéalis	68.03
371	Waste Stockpile 001 ***AtkinsRéalis	6.41
372	Waste Stockpile 001 Ponds and Ditches ***AtkinsRéalis	4.07
374	Waste Stockpile 002 ***AtkinsRéalis	34.76
375	Waste Stockpile 002 Ponds and Ditches ***AtkinsRéalis	22.80
390	Overburden and Peat Storage Facility (OPSF) ***AtkinsRéalis	5.26
391	Overburden and Peat Storage 004 ***AtkinsRéalis	5.26
	300 – Total	**128.17**

Given the complexity of potentially acid-generating and metal-leaching characteristics of the waste Stockpile 002 material, its construction accounts for 45% of the total water and waste rock management costs. As part of the initial CAPEX, only 35% of the full footprint of Stockpile 002 is built, with the remaining portion to be developed as sustaining capital throughout the life of mine.

21.1.4.1.4 400 – Surface Operations

A summary of the capital expenditures of the surface mobile equipment is presented in Table 21.6. Construction mobile equipment includes purchasing costs for lifting equipment, utility vehicles, and specialized construction equipment.

Table 21.6: Phase 1 Surface Equipment Capital Expenditures by WBS

WBS	Items	$M
410	Surface Operations Equipment	18.62
411	Construction Mobile Equipment	13.82
412	Process Plant Mobile Equipment	0.07
414	G&A Mobile Equipment	4.73
	400 – Total	**18.62**

21.1.4.1.5 500 - Mining

There are no leasing options for the open pit mining equipment. All necessary equipment during the pre-production period will be bought up front and according to the manufacturer's terms and conditions. The capital costs estimate for the mining areas is presented in Table 21.7. The costs are based on an owner-operated mining fleet, apart from the overburden and grubbing contract that was estimated at $34.97M in initial capital expenditure.

Table 21.7: Phase 1 Mining Capital Expenditures by WBS

WBS	Items	$M
510	Surface Mine Infrastructure	10.50
511	Haul Road	10.50
520	Surface Services infrastructure	0.95
522	Surface Mine Ventilation	0.01
524	Surface Compressor	0.94
570	Pit Surface Preparation	34.97
571	Overburden	34.97
580	OP Mine Equipment	73.55
581	Primary Mining Equipment	47.16
582	Secondary Mining Equipment	12.87
583	Auxiliary Mining Equipment	5.77
584	Other Mining Support Equipment	6.48
585	FMS / Dispatch / Comm Mining Equipment	0.12
586	Mining Software / Survey& Monitoring Equipment	1.16
	500 – Total	**119.97**

21.1.4.1.6 600 - Process Plant

The initial capital expenditure for Phase 1 includes the costs related to a 2.5 Mtpa throughput DMS as described in Table 21.8.

The capital costs of the processing plant have been jointly developed by the teams from Primero and GMS to ensure all required costs are fully captured.

Steel structure, piping, mechanical, instrumentation, HVAC, fire protection, earthworks, electric, concrete pad and architecture components are the main costs considered. Direct costs cover all labour, permanent equipment, and materials, excluding mobile equipment and freight, which are accounted for under indirect costs (700s) and G&A costs (800s), respectively.

The total capital cost of Phase 1 of the processing facility was estimated at $217.31M.

Table 21.8: Phase 1 Process Expenditures by WBS

WBS	Items	$M
600	Process Plant	12.28
601	Process Plant Phase 1 ***Primero	0.48
607	ROM Pad & MSA Wall	11.79
610	Crushing ***Primero	23.91
611	Crushing Phase 1 ***Primero	23.91
620	Screening ***Primero	18.95
621	Screening Phase 1 ***Primero	18.95
630	Crushed Ore Stockpile ***Primero	14.47
631	Crushed Ore Stockpile Phase 1 ***Primero	14.47
640	DMS (Dense Media Separation) ***Primero	96.02
641	Primary Coarse DMS Phase 1 ***Primero	96.02
650	Concentrate Handling ***Primero	24.58
650	Concentrate Handling – Instrumentation	0.05
651	Concentrate Handling Phase 1 ***Primero	24.53
660	Tails Dewatering ***Primero	5.49
661	Tails Dewatering Phase 1 ***Primero	5.49
670	Tails Handling ***Primero	3.17
671	Tails Handling Phase 1 ***Primero	3.17
690	Process Plant Services ***Primero	18.44
691	Process Plant Services Phase 1 ***Primero	18.44
	600 – Total	**217.31**

21.1.4.1.7 700 - Construction Indirect

Construction indirect costs for the Shaakichiuwaanaan Project are based on the quotation received on specific items, plus GMS's historical executed Project costs and MTOs. The indirect costs are listed in Table 21.9.

Table 21.9: Phase 1 Indirect Costs by WBS

WBS	Description	$M
710	Site Engineering, CM, PM	38.66
711	Site CM Staff and Consultants	30.02
713	Surveying	1.37
714	QA/QC	5.37
715	Induction	0.79
716	Project Controls	1.12
720	Construction Offices, Facilities & Services	18.85
721	Construction Offices / Trailers	3.45
722	Temporary Truckshop	0.35
723	Site Office Supplies & Expenses	2.16
724	Temporary Laydown Facilities	1.21
725	Camp Construction Temporary Facilities	2.63
726	Concrete Batch Plant	1.24
727	Site Toilets / Ablution Units	0.57
728	Construction Temp Power Distribution	6.65
729	Construction Temp Water and Piping Network	0.59
730	Shops	4.27
731	Fab Shop	1.26
732	Electric Shop	1.26
733	Carpenter	0.91
735	Piping	0.13
736	Electric and Instrument	0.13
737	Lifting Equipment	0.48
739	Tool Crib	0.11

WBS	Description	$M
740	Construction Equipment & Tools	55.99
741	Owned Equipment	10.15
742	Equipment Rentals	13.37
743	Operation and Maintenance	11.25
744	Major Construction Tools	2.92
745	Construction Tools and Consumables	12.50
746	LOTOTO	0.71
747	EPP for Construction	0.31
748	Scaffoldings	4.06
749	IT Equipment	0.71
750	Construction Staff Logistics	35.42
752	Transportation & Travelling Fees	31.23
753	Recycling / Sort Facility	1.00
754	Domestic Waste	3.20
760	Energy	52.16
761	Fuel	48.15
762	CNG	0.73
763	Electricity	3.28
780	Contractor Indirects	28.10
781	Construction Bonds, Insurances, etc.	0.13
782	Construction Engineering Charge	27.97
790	External Engineering	29.33
791	External Engineering	29.33
	700 – Total	**262.78**

21.1.4.1.8 800 - General Services and Owner's Costs

The owner's costs were provided by PMET alongside GMS's own estimation. The G&A departments included general administration of the site, procurement, accounting, HR & training, security, telecommunications and owner's salaries at site. Logistics includes all freight costs for international and in-country equipment and material transportation. The operating expenses of the site during construction

Phase 1 initial capital expenditures include all costs related to operating the camp, travelling of workers and surface transportation (buses and shuttles in and out of site).

Approximately 58 ha of fish habitat will need to be moved, replaced or compensated. This surface is divided into two (2) phases and sequenced with the open pit mine, on which the first area of 12.5 ha is included in the initial costs and corresponds to an estimated cost of $3.72M.

Table 21.10: Phase 1 General Services – Owner's Costs by WBS

WBS	Description	$M
810	G&A Departments ***PMET / GMS	33.67
811	General Administration ****PMET / GMS	6.70
815	Security ***PMET / GMS	2.94
817	IT & Telecommunications Service ***PMET / GMS	1.89
819	Owner Team at Site - Salaries ***PMET / GMS	22.14
820	Logistics / Taxes / Insurance	31.15
821	All Freight Cost - Out of Country and Inland - 8%	30.79
825	Customs, Taxes and Duties ***PMET / GMS	0.25
826	Freight Insurance (included in 860) ***PMET	0.10
830	Operating Expenses ***PMET / GMS	28.07
831	Camp OPEX ***PMET / GMS	20.12
832	Travel & Transportation ***PMET / GMS	7.27
833	Surface Support ***PMET / GMS	0.68
840	Environment, Community and Permitting ***PMET / GMS	3.97
848	Fees to Ministries ***PMET / GMS	0.25
849	Lake Fish Compensation and Relocation	3.72
850	Health and Safety	0.81
852	PPE - Construction – Owner's Team	0.30
855	Training ***PMET / GMS	0.05
856	Medical Expenses ***PMET / GMS	0.15
857	Health and Safety Equipment and Systems ***PMET / GMS	0.32
860	Health and Safety	2.14
861	Site Team (included in 819) ***PMET / GMS	2.14
	800 – Total	**99.80**

21.1.4.1.9 900 - Pre-Production Costs

Section 900 includes the operating costs before the commercial production of Phase 1 and the initial contingency. Pre-production costs include the costs for mining the early phase of the open pit, operational costs for commissioning the processing plant, first fills, spare parts and consumables bought in pre-production.

Pre-production processing costs include the period with full processing personnel for the training and commissioning team. The process plant commissioning period includes four (4) months of activity. Then, a 4-month ramp-up period prior to commercial production includes all processing costs required to hot commission the plant, assuming 65% of nameplate throughput for one (1) month before declaring commercial production.

An estimate of $0.9M was also included for vendor representatives and operating specialists.

Table 21.11 presents the preproduction costs and contingency for the processing plant and mining.

Table 21.11: Phase 1 - Pre-Production Costs

WBS	Description	$M
910	Mining Pre-Prod	49.12
913	Transfer OPEX / CAPEX	49.12
950	Process Plant Pre-Prod	17.51
951	Process Plant Management & Training ***Primero / GMS	12.22
953	Process Plant Commissioning ***Primero / GMS	1.07
954	Vendor Reps ***Primero / GMS	0.92
956	Consulting Services ***PMET	3.30
960	First Fill, Spares & Consumables	6.68
961	Spare Parts Capital ***Primero / GMS	4.09
962	Spare Parts Commissioning	1.63
965	First Fill (reagents, grease & oil) ***Primero	0.97
990	Contingency	130.67
991	Project Contingency ***GMS / PMET	130.67
	900 – Total	**203.98**

21.1.4.1.10 991 - Contingency

Contingency has been estimated on an area-by-area basis by evaluating the confidence level of both scope definition and cost estimates, followed by a Monte Carlo iteration analysis. The recommended contingency of $130.7M amounts to 11% of direct and indirect expenditures before revenue.

Contingency analysis does not consider Project risks, currency fluctuations, escalation beyond predicted rates, or costs due to potential scope changes or labour disruptions.

With the Project definition and lower exposure to typical execution risks due to the self-perform approach, a 11% contingency is considered appropriate, providing balanced risk coverage given the current level of Project certainty.

21.1.4.2 Phase 2 Capital Costs

Phase 2 CAPEX is defined as all costs incurred for the underground mine preparation and development, and the process plant throughput increase from 2.5 Mtpa to 5.1 Mtpa, representing 400,000 t to 800,000 t of concentrate per year. This phase has a duration of 3.6 years (from September 2028 to March 2032). This includes all costs associated with process plant expansion, buildings at the camp, roads and infrastructure related to the underground mine. The total Phase 2 capital cost for the Shaakichiuwaanaan Project is estimated to be $628.8M (including contingencies, indirect costs and pre-production costs).

As outlined in Table 21.12, Phase 2 capital costs are separated into two (2) phases, $148.3M (23% of total) during initial capital and $480.5M spent as expansion capital.

The CAPEX includes a contingency of $63.7M, which is 11% of the total cost. The total labour hours for the initial Phase 2 are estimated at 823,000 hours of construction.

Table 21.12: Project Phase 2 Capital Cost Summary

WBS	Items	Initial Capital ($M)	Expansion Capital ($M)	Total ($M)
100	Infrastructure	-	24.8	24.8
200	Power and Electrical	-	46.2	46.2
300	Water and Tailing Management	-	18.7	18.7
500	Mining	99.1	36.4	135.5
600	Process Plant	20.1	167.0	187.1
700	Construction Indirect	0.1	123.8	123.9
800	General Services / Owner's Cost	4.7	13.4	18.1
900	Pre-production, Start-up, Comm.	9.3	1.5	10.8
990	Contingency	15.0	48.7	63.7
Total Including Contingency		**148.3**	**480.5**	**628.8**

21.1.4.2.1 100 – Infrastructure

The CAPEX estimates for infrastructure are summarized in Table 21.13. The two (2) main expenses for the Phase 2 are the expansion of the mine garage associated with underground truck fleet maintenance and the increase of dorm modules at the camp for the additional workers at site.

Table 21.13: Phase 2 Infrastructures Capital Expenditures by WBS

WBS	Items	$M
110	Roads, Bridges and Fencing	0.24
111	Deforestation	0.24
120	Mine Infrastructure	8.18
121	Truckshop (truckshop, offices, washbay)	8.08
125	Truck Scale	0.10
130	Support Infrastructure	3.54
131	Administrative Building	0.20
134	Warehouse	3.25
139	Firehall and Clinic	0.09
140	Camp Facilities	12.03
141	Camp Dorms	12.03

WBS	Items	$M
160	Process Plant Infrastructure	0.64
162	Workshop - ***Included in DMS Office ***Primero	0.60
165	Process Plant Office	0.04
170	Fuel Systems Storage	0.21
176	Gas Facility Paste Pant and Underground	0.21
	100 - Total	**24.84**

21.1.4.2.2 200 - Power and Supply

The CAPEX estimates for Power Supply and Electrical are summarized in Table 21.14. The major cost for electrical distribution of the Phase 2 pertains to additional electrical room for the underground mine and paste plant distribution, and processing plant expansion.

Table 21.14: Phase 2 Power Supply and Communications Capital Expenditures by WBS

WBS	Items	$M
210	Main Power Generation	5.17
213	Site Main Substation	5.17
240	Service Electrical Room	0.02
241	Camp Area E-Room	0.02
250	Mine Electrical Room	19.62
251	U/G mine E-Room (Phase 2)	3.00
252	Underground Distribution (Phase 2)	12.91
259	Paste Plant Electrical Room (Phase 2)	3.71
260	Process Plant Electrical Rooms	15.00
261	Crushing Electrical Room ***Primero	4.44
262	DMS Circuit Electrical Room ***Primero	7.57
263	Crushing Electrical Room Phase 2 ***Primero	0.50
264	DMS Circuit Electrical Room Phase 2 ***Primero	0.67
268	Paste Production Electrical Room	1.83
270	MV Distribution O/H Line	0.37
271	MV Distribution O/H Line	0.37

WBS	Items	$M
290	IT Network & Fire Detection	6.06
291	IT Network	6.01
293	Process Control Room ***Primero	0.05
	200 - Total	**46.24**

21.1.4.2.3 300 - Water and Tailings Management

The CAPEX summary for water management is presented in Table 21.15. Water management CAPEX costs were jointly developed by AtkinsRéalis and GMS to capture all required expenses. The main costs in this section related to Phase 2 capital expenditures are the dyke lake 001 pumping and the water treatment plant augmentation.

Table 21.15: Phase 2 Water Management Capital Expenditures by WBS

WBS	Items	$M
320	Water Ponds and Water Management ***AtkinsRéalis	11.19
328	Dyke Lake 001 Phase 2 ***AtkinsRéalis	9.57
329	Diversion Channel for Lake 001 (Long) (spillway and ditch) ***AtkinsRéalis	1.61
360	Effluent Water Treatment ***AtkinsRéalis	7.48
362	Water Treatment Plant Phase 2 ***AtkinsRéalis	6.08
365	Pumps and Pipelines Phase 2 ***AtkinsRéalis	1.40
	300 - Total	**18.67**

21.1.4.2.4 500 - Mining

There are no leasing options for the underground mining equipment apart from the batteries themselves. Battery leasing is a common option taken by underground mines due to the complexity of handling batteries post useful life. All necessary equipment during the pre-production period will be bought up front and following the manufacturer's agreements. The CAPEX estimate for the mining areas is presented in Table 21.7. The costs are based on an owner-operated mining fleet, apart from the main ramp development contract that will last about 27 months (until December 2030). In addition to the lateral development, underground infrastructure will need to be put in place, including:

- Electrical stations.
- Pumping stations.

- Ore passes and ore chutes.
- Fans and ventilation system infrastructure.
- Truck trolley system.
- Electrical and communication wiring.
- Garage, fuels and gear bays.
- Explosives and detonators depot.
- Paste backfill infrastructure.

A paste plant will also need to be constructed next to the processing plant at an estimated cost of $39.2M, excluding the paste preparation plant into the process plant of $17.8M. This paste preparation plant is part of the area WBS 680. The total CAPEX of the underground infrastructure, including the paste plant, was estimated to be $135.5M.

Table 21.16: Phase 2 Mining CAPEX by WBS

WBS	Items	$M
520	Surface Services Infrastructure	2.46
522	Surface Mine Ventilation	1.65
523	Ventilation Heating	0.45
524	Surface Compressor	0.26
529	Surface Safety Egress Shelter	0.10
530	U/G Mine Excavation	73.02
531	Portal (boxcut)	4.36
532	Lateral Development	53.39
534	Ventilation Raise & Escapeways	1.47
536	Services Excavation (sub-station, remuck, etc.)	12.20
537	Muck Bay	1.33
539	Services Hole	0.28
540	U/G Mine Services	7.13
541	U/G Power Supply & Transmission	1.28
542	U/G Dewatering	2.18
543	U/G Ventilation	1.33
544	Misc. Construction	1.31

WBS	Items	$M
545	Refuge	1.02
550	U/G Mine Equipment	11.57
551	Development Equipment U/G	2.68
552	Production Drill U/G	1.85
553	LHD U/G	1.09
554	Truck U/G	1.93
555	Services Equipment U/G	0.64
556	Utility Equipment U/G	2.63
557	Misc. Equipment U/G	0.75
590	Paste Preparation	41.31
591	Backfill Plant	4.96
591	Slurry Storage	19.09
592	Filtration	4.77
593	Cake Repulping and Mixing	1.24
594	Binder Storage	2.41
595	Paste Pumping	6.11
597	Plant Services	2.72
	500 - Total	**135.50**

21.1.4.2.5 600 - Process Plant

The process plant CAPEX associated with Phase 2 includes the costs related to a second train of 2.5 Mtpa throughput Dense Media Separator (DMS) and reaches the total of 800 ktpa of concentrate as described in Table 21.17. Processing plant CAPEX were jointly developed by Primero and GMS to capture all required expenses. The total CAPEX of the second phase of the processing plant facility was estimated at $187.09M.

Table 21.17: Phase 2 Process Expenditures by WBS

WBS	Items	$M
600	Process Plant	1,45
601	Process Plant Phase 1 ***Primero	0,97
602	Process Plant Phase 2 ***Primero	0,48

WBS	Items	$M
610	Crushing *** Primero	23,73
612	Crushing Phase 2 ***Primero	23,73
620	Screening ***Primero	18,04
622	Screening Phase 2 ***Primero	18,04
630	Crushed Ore Stockpile ***Primero	14,64
632	Crushed Ore Stockpile Phase 2 ***Primero	14,64
640	DMS (Dense Media Separation) ***Primero	85,89
642	Primary Coarse DMS Phase 2 ***Primero	85,89
650	Concentrate Handling ***Primero	5,57
652	Concentrate Handling Phase 2 ***Primero	5,57
660	Tails Dewatering ***Primero	6,73
662	Tails Dewatering Phase 2 ***Primero	6,73
670	Tails Handling ***Primero	2,39
672	Tails Handling Phase 2 ***Primero	2,39
680	Paste Material Preparation – Phase 2 ***Primero	15,75
681	Process Plant ***Primero (Phase 2)	15,75
690	Process Plant Services ***Primero	12,89
692	Process Plant Services Phase 2 ***Primero	12,89
	600 – Total	**187,09**

21.1.4.2.6 700 - Construction Indirect

The main CAPEX in this category are Project management and contractual services, construction equipment, both owned and rented, logistics, fuel, construction engineering charges and external engineering. The construction indirects costs for Phase 2 are calculated using a 30% rate of the direct cost, for an amount of $123.92M.

The indirect costs listed in Table 21.18 concern the pre-production portion of the indirect costs.

Table 21.18: Phase 2 Indirect Costs by WBS

WBS	Description	$M
700	Construction Indirects	123,70
701	Indirects Phase 2	123,70
730	Shops	0,09
735	Piping	0,09
780	Contractor Indirects	0,13
781	Construction Bonds, Insurances, etc.	0,13
	700 - Total	**123,92**

21.1.4.2.7 800 - General Services and Owner's Costs

General Services and owner's costs for the Phase 2 include the logistics of material and equipment, as well as the construction insurance for Phase 2.

Table 21.19: Phase 2 General Services – Owner's Costs by WBS

WBS	Description	$M
820	Logistics / Taxes / Insurance	16.91
821	Out of Country & Inland Freight	16.81
826	Freight Insurance	0.10
860	Site Insurance: Construction. All Risk and Marine Cargo	1.21
861	Insurance	1.21
	800 - Total	**18.12**

21.1.4.2.8 900 - Pre-Production Costs and Contingency

The pre-production costs refer to mining costs incurred prior to the end of Phase 2 in March 2032. The pre-production costs include the costs for initializing the underground development. Table 21.20 shows the preproduction costs for mining.

Contingency has been estimated on an area-by-area basis by evaluating the confidence level of both scope definition and cost estimates, followed by a Monte Carlo iteration analysis. The recommended contingency of $63.69M amounts to 11% of direct expenditure before revenue.

The contingency analysis does not consider Project risks, currency fluctuations, escalation beyond predicted rates, or costs due to potential scope changes or labour disruptions.

Table 21.20: Phase 2 Pre-Production Costs

WBS	Description	$M
910	Mining Pre-Prod	10.77
913	Mining Pre-Production Cost	10.77
990	Contingency	63.69
991	Project Contingency ***GMS / PMET	63.69
	900 - Total	**74.46**

21.1.5 LOM Sustaining Capital Expenditures

The Project sustaining capital costs include all costs related to continuing and maintaining the operation through the life of mine. It includes stockpile expansion, purchasing mining equipment for both open pit and underground, development underground and their related and indirect costs and contingencies. It also covers the major repairs for the mine equipment. Sustaining costs start at Year 1 until the end of the mining operations. The LOM sustaining capital costs are estimated to be $936.4M.

Table 21.21: Sustaining Cost Summary

WBS	Items	$M
100	Infrastructure	30.8
200	Power and Electrical	25.0
300	Water and Tailing Management	100.5
400	Surface Operations	11.9
500	Mining	550.5
800	General Services / Owner's Cost	31.6
900	Pre-production, Start-up, Comm.	186.1
Total Including Contingency		**936.4**

Each mining Project phase, i.e. the open pit and underground, has its specific sustaining capital cost as presented in Table 21.22 and Table 21.23, respectively; $531.9M for Phase 1 and $404.6M for Phase 2.

Table 21.22: Phase 1 Sustaining Capital Expenditure – in $M

Sustaining Cost Summary – Phase 1	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
120 - Mine Infrastructure	**29.4**	8.8	17.7	2.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
160 - Process Plant Infrastructure	**0.5**	0.2	0.3	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
170 - Fuel Systems Storage	**0.8**	0.2	0.5	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
370 - Waste Rock Stockpile	**89.6**	25.6	38.4	25.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
390 - Overburden and Peat Storage Facility (OPSF)	**4.3**	1.1	1.1	1.1	0.0	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
410 - Surface Operations Equipment	**11.9**	0.0	0.4	0.1	0.3	0.4	0.8	0.1	1.7	1.2	0.3	2.2	0.7	0.4	1.8	0.6	0.6	0.0	0.0	0.0
510 - Surface Mine Infrastructure	**5.9**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.5	1.5	1.5	1.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0
570 - Pit Surface Preparation	**44.4**	24.2	0.0	4.7	4.9	9.5	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
580 - OP Mine Equipment	**133.8**	47.3	0.2	7.8	4.0	5.2	1.2	3.0	13.0	19.3	11.7	2.7	6.3	4.7	4.4	2.5	0.2	0.3	0.0	0.0
820 - Logistics / Taxes / Insurance	**14.7**	2.5	0.1	0.8	1.5	1.5	1.5	1.2	0.8	1.4	0.7	0.4	0.5	0.4	0.5	0.4	0.1	0.2	0.0	0.0
840 - Environment, Community and Permitting	**13.8**	0.0	0.0	3.5	3.5	6.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
910 - Mining Major Repairs	**182.7**	4.4	1.4	10.7	16.0	10.6	23.6	13.9	12.3	24.0	10.3	9.7	9.1	5.9	11.5	9.5	1.9	5.8	1.1	1.2
Total	**531.9**	**114.3**	**60.1**	**57.3**	**30.2**	**35.2**	**28.1**	**18.2**	**27.9**	**47.4**	**24.5**	**16.5**	**18.1**	**11.3**	**18.2**	**13.0**	**2.8**	**6.3**	**1.1**	**1.2**

Table 21.23: Phase 2 Sustaining Capital Expenditure – in $M

Sustaining Cost Summary – Phase 2	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
250 - Mine Electrical Rooms	**25.0**	5.7	2.4	1.5	3.2	1.4	1.1	0.6	0.5	1.5	1.0	1.0	0.7	1.1	1.4	1.0	0.9	0.0	0.0	0.0
360 - Effluent Water Treatment	**6.6**	1.5	0.6	0.4	0.8	0.4	0.3	0.2	0.1	0.4	0.3	0.3	0.2	0.3	0.4	0.3	0.2	0.0	0.0	0.0
520 - Surface Services infrastructure	**2.1**	2.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
530 - U/G Mine Excavation	**140.2**	32.1	13.2	8.6	17.7	7.9	6.1	3.5	2.8	8.3	5.8	5.7	4.0	5.9	7.6	5.7	4.6	0.1	0.6	0.0
540 - U/G Mine Services	**19.7**	3.8	2.3	1.7	3.5	1.3	0.8	0.7	0.5	0.8	0.9	0.6	0.2	0.6	0.6	0.7	0.3	0.2	0.2	0.0
550 - UG Mine Equipment	**178.2**	33.9	20.9	11.7	4.0	4.6	5.2	5.5	6.8	8.3	9.9	13.6	9.6	7.6	7.4	4.7	6.6	6.8	4.8	6.3
590 - Paste Preparation	**26.3**	7.7	1.2	2.5	2.1	1.7	1.0	0.6	1.2	1.2	1.1	1.2	1.0	1.0	0.6	0.2	1.4	0.3	0.3	0.0
820 - Logistics / Taxes / Insurance	**3.1**	0.6	0.3	0.2	0.2	0.1	0.1	0.1	0.1	0.2	0.1	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
910 - Mining Major Repairs	**3.4**	0.0	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
Total	**404.6**	**87.5**	**41.0**	**26.8**	**31.8**	**17.6**	**14.7**	**11.4**	**12.2**	**20.8**	**19.3**	**22.8**	**16.0**	**16.8**	**18.2**	**12.8**	**14.3**	**7.6**	**6.2**	**6.6**

21.2 Mine Closure Expenditures

The closure CAPEX is estimated at $248.4M. Engineering inputs (MTOs) had been performed by AtkinsRéalis, and Cost estimation was performed by GCM Consultant. A summary of all the closure costs is presented in Table 21.24. The mine closure cost involves the following:

Waste Storage Facility (WSF)

The closure scenarios of WSF are:

- Stockpile 001 will be designated for a portion of the "low-risk" waste rock. As part of its closure, the surface of Stockpile 001 will be reprofiled to ensure proper drainage toward water collection ditches. A 30 cm layer of overburden will then be placed over the waste rock to prevent topsoil infiltration into the voids, followed by a 20 cm layer of topsoil. Finally, the surface will be vegetated to stabilize the area, reduce erosion, and promote landscape integration.
- Stockpile 002 will receive all the potentially metal-leaching (ML), high-risk, or acid-generating (PAG) waste rock. Due to the nature of the rocks to be stored in Stockpile 002, an impermeable cover consisting of a 1.5 m thick clay layer has been selected to restore Stockpile 002. This impermeable cover is intended to limit water infiltration and oxygen diffusion to the underlying materials, thereby preventing the production of acid mine drainage and metal leaching. The benches will have already been re-sloped to a 3:1 ratio during the operation of the mine to ensure long-term stability. During restoration, the impermeable cover will be followed by the placement of a 20 cm layer of topsoil. Finally, the surface will be vegetated to stabilize the area, reduce erosion, and promote landscape integration.
- Organics and overburden piles (001, 004, and 005) will be used for closure activities. Once the material has been utilized, the remaining footprint of these stockpiles will be scarified to break up compacted surfaces and promote natural drainage. These areas will then be revegetated using hydroseeding, which facilitates rapid vegetation establishment and erosion control.
- The underground transfer pad, the high- and low-grade mixing stockpiles, and the ROM pad will be cleared of materials. Their surfaces will be scarified and revegetated using hydroseeding.

Underground Mine

- The closure scenario for the underground mine is to de-energize and decouple all infrastructure, remove all equipment, cap or plug surface openings and allow the water to naturally fill the voids. During the post-closure phase, the portal and the underground mine ventilation raise will be secured with waste rock or concrete.

Open Pit Flooding and Site Securing

- The closure scenario for the open pit is to allow the pit to flood naturally with both groundwater and surface water, to construct a safety berm around the pit perimeter and install signage.

Restoration of the Industrial Area and Access Roads

- All processing plants, power facilities, administrative buildings, and other industrial structures and equipment will be dismantled and removed from the site for disposal at an authorized landfill. Materials generated during the dismantling process will be managed by applying the principles of reduction, reuse, recycling, and reclamation. Concrete slabs and foundations will be broken or cracked to facilitate drainage, then covered with topsoil-like material before being revegetated with hydroseeding. Access roads and former storage or laydown areas on the mine site will be fully decommissioned and scarified to restore drainage and support revegetation.

Restoration of the Water Management System

- Once the site has been restored and the water quality has been confirmed, the four (4) ponds located within the water treatment area, as well as the two (2) ponds in the industrial area, will be backfilled. The areas will be reprofiled and revegetated using hydroseeding. Before any earthwork begins, the sediments accumulated at the bottom of the ponds will be characterized, excavated, and either stored in Stockpile 002 or transported to an authorized disposal site. The geomembrane will be removed to allow drainage.
- As for the four (4) ponds collecting water for Stockpiles 001 and 002, a breach in each of the dikes will be excavated to ensure there is no water accumulation at the toe of the stockpiles. Sediments collected at the bottom of these ponds will also be removed and managed appropriately. It should be noted that during the restoration of Stockpile 002, runoff from the restored sections will be collected in a newly constructed pond. This temporary pond will also be breached once the restoration work is completed.

Management of Contaminated Soils

- A site characterization will be conducted at closure to identify any soil contamination exceeding regulatory limits and to apply appropriate remediation measures, in compliance with the Environment Quality Act and the Land Protection and Rehabilitation Regulation. The closure cost estimate includes a contingency for the management of potentially contaminated soils.

Monitoring and Maintenance Programs

- Long-term consideration will include environmental monitoring programs to evaluate geochemical stability and potential settlement risks, ensuring that restored areas do not pose contamination or subsidence issues over time. The implementation of these programs will confirm the effectiveness of restoration efforts. A variety of maintenance and monitoring activities have been scheduled for the closure and post-closure periods, including cover / revegetation monitoring, biodiversity monitoring, groundwater and surface water monitoring, revegetation, cover maintenance, and WSF-specific monitoring and security measures.

Cost Estimation and Financial Guarantee

- The CAPEX of reclamation (and the guarantee) is estimated at $248.4M. This cost includes the direct and indirect costs of site rehabilitation as well as post-closure monitoring, engineering costs (30%) and the mandatory 15% contingency. A detailed cost estimate will be reviewed and submitted at least every five (5) years throughout the closure planning process, in accordance with the Guidelines for Preparing Mine Closure Plans in Québec.

Table 21.24: Mine Closure Cost Summary

Reclamation and Closure Costs	Cost ($M)
Site Security	1.1
Dismantling, Demolition, and Demobilization of Surface Infrastructure	10.7
Restoration of the Industrial Area and Access Roads	5.2
Restoration of Stockpiles	102.5
Restoration of the Water Management System	9.5
Management of Contaminated Soils	12.3
Monitoring Costs	15.0
Indirect Costs	54.7
Other Costs (contingency (15%))	37.3
Total Reclamation and Closure Costs	**248.4**

A financial bond corresponding to the total anticipated cost of completing all the work set forth in the rehabilitation and restoration plan will be provided to the Minister of Finance of Québec as required by the regulations.

21.3 Operating Cost

21.3.1 Summary

Operating costs are all the costs related to the daily operation of the Project that are not capitalizable. These include the costs associated with the operation, maintenance, and administration of the site and the concentrate transportation to Grande Anse. The total LOM operating cost for the Shaakichiuwaanaan Project was estimated at $9,664.3M, as detailed in Table 21.25.

Table 21.25: Total LOM Operating Cost including Concentrate Transportation

Description	Operating Cost ($M)	Unit Cost ($/t conc)
Open Pit Mining	1,390.4	104.9
Underground Mining	2,852.2	215.2
Processing	1,208.3	91.2
G&A	1,334.7	100.7
Concentrate Transportation	2,878.8	217.2
Total	**9,664.3**	**729.1**

A summary of the total operating costs, including mining, processing, G&A, and concentrate transportation, as well as total cost per tonne processed, is presented in Table 21.26.

Table 21.26: Total Operating Costs Summary

Description	Operating Costs ($M)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Open Pit Mining	1,390.4	84.1	87.2	83.7	82.8	96.5	106.5	94.3	96.9	98.0	90.5	75.1	79.4	63.0	62.0	59.8	55.4	49.3	13.5	12.5
Underground Mining	2,852.2	71.3	131.1	175.5	166.4	166.7	162.1	161.4	157.5	152.2	154.8	156.1	157.2	154.5	155.5	154.0	156.5	153.5	151.3	114.4
Processing	1,208.3	43.4	42.7	65.6	69.7	69.7	69.5	70.1	68.6	69.5	70.2	69.3	69.8	67.7	67.5	67.9	66.9	64.0	49.9	46.3
G&A	1,334.7	64.4	65.5	74.5	76.3	76.7	75.5	75.1	74.9	74.9	74.7	73.7	73.7	72.6	72.7	72.6	72.1	71.0	50.3	43.5
Concentrate Transportation	2,878.8	78.6	91.5	158.8	174.0	175.2	175.2	175.2	175.2	175.2	170.7	175.2	163.4	175.2	175.2	175.2	175.2	175.2	69.1	45.6
Total OPEX (incl. Concentrate Transportation)	**9,664.3**	**341.8**	**418.0**	**558.1**	**569.2**	**584.7**	**588.9**	**576.1**	**573.0**	**569.8**	**560.9**	**549.3**	**543.5**	**533.0**	**533.0**	**529.5**	**526.1**	**513.0**	**334.0**	**262.3**
Total Cost/t (Tonnes Processed)	114.6	130.3	131.6	112.9	111.6	115.3	115.9	113.5	120.0	114.5	110.0	111.7	106.6	106.8	107.8	105.6	109.4	113.6	157.0	179.6
Total Cost/t Concentrate (dry)	729.1	938.5	986.2	768.3	715.1	729.5	734.6	718.7	714.9	710.9	718.2	685.3	727.1	665.0	664.9	660.5	656.3	640.0	1,056.6	1,256.3

21.3.2 Open Pit Mining

The total operating costs over the 17 years of open pit mining are approximately $1,390M. The LOM average unit costs per tonne mined are summarized in Table 21.27. The mine will be operated by the owner, apart from the overburden and grubbing contracts and the explosive contracts.

Table 21.27: Mining Unit Cost Summary

Description	Unit	Cost
Cost per Tonne Mined	$/t	6.42
Cost per Tonne of Mineralized Material Mined	$/t	28.26
Cost per Tonne of Concentrate	$/t	104.9

21.3.2.1 Consumable Costs

To calculate the mine's operating costs, GMS conducted a detailed analysis of the base consumables (fuel, electricity, and emulsion price). Table 21.28 presents the consumable costs used in this Study.

Table 21.28: Consumables Costs

Consumables	Unit	Value
Clear Fuel (public road transportation)	$/L	1.60
Coloured Fuel (non-licensed vehicles)	$/L	1.43
Electrical Cost	$/kWh	0.0529
Emulsion Cost (Titan XL 1000) Open Pit	$/kg	1.46

21.3.2.2 Open Pit Operating Costs

Table 21.29 summarizes the operating costs for the open pit mine. The operational costs were estimated for labour costs, equipment maintenance and consumables. Some rehandling and mine maintenance activities will remain until the UG mine is depleted at Year 19.

A total of 217 Mt of material will be mined over the LOM, including 150.0 Mt of waste rock, 17.5 Mt of overburden and 49.2 Mt of mineralized material.

Table 21.29: Open Pit Mine Operating Costs Summary

Parameters	LOM Total ($M)	Average LOM ($/t mined)	% OPEX
Mine Operations	55.17	0.25	4%
Mine Geology	37.28	0.17	3%
Mine Maintenance Admin.	162.16	0.75	12%
Mine Engineering	43.15	0.20	3%
Drilling	82.56	0.38	6%
Pre-Split Drilling and Blasting	21.19	0.10	2%
Blasting	179.78	0.83	13%
Loading	70.05	0.32	5%
Hauling	288.41	1.33	21%
Dewatering	42.11	0.19	3%
Dump Maintenance	89.59	0.41	6%
Road Maintenance	150.21	0.69	11%
Grade Control	8.80	0.04	1%
Support Equipment	129.74	0.60	9%
Total Electrical Services	2.19	0.01	0%
Ore Feed	11.31	0.05	1%
Sub-Total In-Situ Mining	1,373.69	6.34	99%
Rehandling	16.70	0.08	1%
Total	1,390.39	6.42	100%

21.3.2.2.1 Mine Services

The mine services costs include the following:

- Mine Operation.
- Mine Geology.
- Mine Maintenance Administration.
- Mine Engineering.

These costs include:

- Salaries.
- Training.
- Travel Expenses.
- Safety Equipment.
- Office Supplies.
- Computer Equipment.
- Consulting Fees.
- Tools (when appropriate).
- Contractors (when appropriate).
- Mine Dispatch System.
- Business Readiness (when appropriate).
- Survey Instruments Maintenance and its Software (when appropriate).
- Radio and Communication Systems.

21.3.2.2.2 Drill & Blast

Drill and blast costs include the costs related to the drilling and blasting of the waste and ore rock. The costs include:

- Salaries.
- Training.
- Consumables (Drilling and Blasting).
- Maintenance Parts.

21.3.2.2.3 Loading & Hauling

Costs for the loading and hauling equipment include the following:

- Salaries.
- Training.

- Consumables, including Tires, Lubricants, and Fuel (Loading and Hauling).
- Maintenance Parts.

The costs exclude rehandling, where the appropriate costs are reported in the rehandling operating costs section. Mobile equipment costs were estimated based on the operating hours of each equipment per period. The operating costs per hour per equipment are based on data provided by the equipment manufacturers.

21.3.2.2.4 Dewatering

Costs for the open pit dewatering include:

- Salaries of the dewatering crew.
- Training.
- Consumables for the pumps and the crew vehicles.

21.3.2.2.5 Dump Maintenance

The dump maintenance includes the crew and the track dozers used. Salaries, training on equipment and the consumables are included in the operating costs.

21.3.2.2.6 Road Maintenance

Road maintenance is performed by graders, water and sand trucks and wheel dozers. Salaries, training on equipment and consumables are included in the operating costs. The aggregate production is also included in the operating costs of this category.

21.3.2.2.7 Grade Control

Grade control costs represent the assay charges from the laboratory.

21.3.2.2.8 Support Equipment

Support equipment operational costs represent the salary, training and consumables related to the support equipment below:

- Excavator 49 t.

- Excavator 90 t.
- Hydraulic Hammers.
- Wheel loader 271 HP.
- Telehandler 5 t.
- Mechanic trucks.
- Wheel Loader 350 HP.
- Fuel and lube truck.
- Truck tractors for trailers (560 HP) Type Kenworth C500.
- Trailer Lowboy 100 t.
- Pick up trucks.
- Pit buses.
- Compressors.
- Welding machines.
- And other smaller items.

21.3.2.2.9 Ore Feed

Ore feed operational costs are associated with the loading and rehandling at the ore pad. Salaries, training and consumables are included in this section. The underground ore rehandled costs are back-charged to underground operating costs.

21.3.2.2.10 Rehandling

Rehandling costs are all costs associated with open pit ore rehandling (about 14 Mt). They include salaries, training and consumables.

21.3.3 Underground Mine Operating Cost

21.3.3.1 Summary

Underground mine operations, for both production and development, will be operated by the owner. The underground mining OPEX was estimated based on quotations. Table 21.30 presents a summary of the OPEX.

Table 21.30: Underground OPEX Summary

UG Operating Costs	LOM Total Cost ($M)	Unit Cost ($/t mined)[1]	LOM Total Cost (%)
Total Diamond Drilling and Geology	56.74	1.62	2%
Total Stope Preparation (incl. OPEX dev.)	372.60	10.62	13%
Total Drilling and Blasting	261.73	7.46	9%
Total Mucking and Hauling	269.64	7.68	9%
Total Backfilling	403.79	11.50	14%
Total Supervision	92.37	2.63	3%
Total Mine Services	784.40	22.35	28%
Total Surface Rehandling	29.37	0.84	1%
Total Maintenance Services	164.19	4.68	6%
Total Electrical Services	271.46	7.73	10%
Total Technical Services	145.87	4.16	5%
Total UG Operating Costs	**2,852.16**	**81.26**	**100%**

**Note: 1) Unit costs from mineralized material from the UG mine only (35.1 Mt).*

21.3.4 Processing Facility Operating Costs

The annual operating costs for the process plant were estimated for a nominal year of processing 5,100,000 dry tonnes per year of mineralized material with an average grade of 1.26% Li_2O and producing 801,600 tpa of spodumene concentrate with a grade of 5.5% Li_2O, achieving 68.9% Li_2O recovery. These operating costs could be scaled up or down as a function of the mass of mineralized material processed using fixed and variable cost attribution.

21.3.4.1 Basis

The following cost centres were used for the operating cost estimation:

- Plant labour.
- Consumables.
- Reagents.
- Energy and utilities.
- Maintenance.
- Mobile equipment.
- Tailing Rehandling.
- Laboratory.

Select cost centres (transportation, water treatment, camp, and G&A spend) were out of Primero's scope for the cost estimation, as they are included in GMS' Project spend estimate. Also, power cost, mobile equipment cost and tailing rehandling are based on GMS' estimates.

21.3.4.2 Summary

A summary of the mineral processing facility operating cost estimate is provided in Table 21.31. All fixed costs were applied for each year the plant was in operation, and variable costs were adjusted based on the tonnages fed to the processing plant each year. The variable costs shown in Table 21.32 are based on a 5.1 Mtpa throughput.

Table 21.31: Processing Plant Cost Summary

Processing Plant OPEX	LOM Total Cost ($M)	Unit Cost ($/t Processed)	Unit Cost ($/t conc)
Plant Labour	496.73	5.89	37.48
Consumables	112.78	1.34	8.51
Reagents	88.83	1.05	6.70
Energy and Utilities	142.44	1.69	10.75
Maintenance	159.31	1.89	12.02
Mobile Equipment	57.61	0.68	4.35
Tailing Rehandling	93.49	1.11	7.05
Laboratory	57.07	0.68	4.31
Total	**1,208.27**	**14.33**	**91.16**

A summary of the LOM process costs is presented in Table 21.32.

Table 21.32: LOM Processing Plant Cost Summary

Processing Plant Activity	LOM Total Cost ($M)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Plant Labour	496.7	19.6	18.7	27.2	27.2	27.2	27.2	27.2	27.1	27.2	27.2	27.1	27.2	27.2	27.2	27.2	27.1	26.9	25.6	25.3
Consumables	112.8	2.8	3.3	6.7	6.9	6.9	6.9	6.9	6.5	6.8	6.9	6.7	6.9	6.8	6.7	6.8	6.6	6.2	3.1	2.3
Reagents	88.8	2.8	3.4	5.2	5.4	5.4	5.4	5.4	5.1	5.3	5.4	5.2	5.4	5.3	5.2	5.3	5.1	4.8	2.3	1.5
Energy and Utilities	142.4	4.3	5.0	6.5	7.6	7.8	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9	7.9
Maintenance	159.3	4.9	5.6	9.2	9.5	9.4	9.4	9.4	9.0	9.3	9.5	9.2	9.5	9.3	9.3	9.3	9.1	8.6	5.3	4.4
Mobile Equipment	57.6	3.3	3.1	3.1	3.1	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Tailing Rehandling	93.54	3.8	1.6	4.3	6.5	6.5	6.2	6.8	6.8	6.7	6.8	6.7	6.4	4.8	4.9	4.9	4.9	3.5	0.9	0.6
Laboratory	57.1	1.8	2.0	3.3	3.4	3.4	3.4	3.4	3.2	3.4	3.4	3.3	3.4	3.4	3.3	3.4	3.3	3.1	1.7	1.3
Total	**1,208.3**	**43.34**	**42.7**	**65.6**	**69.7**	**69.7**	**69.5**	**70.1**	**68.6**	**69.5**	**70.2**	**69.3**	**69.8**	**67.7**	**67.5**	**67.9**	**66.9**	**64.1**	**49.9**	**46.3**
Total Cost/t Tonnes Processed	14.3	16.5	13.4	13.3	13.7	13.7	13.7	13.8	14.4	14.0	13.8	14.1	13.7	13.6	13.7	13.5	13.9	14.2	23.4	31.7
Total Cost/t Concentrate (dry)	91.2	119.1	100.8	90.4	87.6	86.9	86.7	87.4	85.6	86.7	89.9	86.4	93.4	84.4	84.3	84.7	83.4	79.9	157.7	221.7

21.3.4.3 Methodology

The methodology used to estimate the mineral processing facility operating costs is described in the following subsections. Primero's lithium processing experience and data were used along with, when possible, vendor quotes.

21.3.4.3.1 Plant Labour

The plant labour cost was estimated to be $27.2M per year with 164 plant personnel. Operations and maintenance labour costs are based on a two-weeks-on, two-weeks-off rotations with 12-hour shifts. Staff are on a 5-days-on, 2-days-off with 8-hour shifts, and management is on a four-days-on, three-days-off with 12-hour shifts. Labour costs were estimated as the sum of GMS's supplied salaries and an additional labour cost burden, which includes overtime pay, performance pay, bonus, and benefits, but excludes travel spend and accommodation spend. Table 21.33 describes the number of employees by sector.

Table 21.33: Plant Personnel Summary

Department	Working Off-Site	Day Shift	Night Shift	Off Rotation	Total
Process Plant Management and Admin					
Process Manager		1			**1**
Process Superintendent		1		1	**2**
Maintenance Superintendent		1		1	**2**
Operation Personnel					
Plant Supervisor		1	1	2	**4**
Plant Lead Hand		2	2	4	**8**
Control Room Operator		2	2	4	**8**
Crushing Operator		2	2	4	**8**
DMS Operator		4	4	8	**16**
Dewatering & Reagent Operator		1	1	2	**4**
Concentrate / Tailings Loader		1	1	2	**4**
Process Trainer		1		1	**2**
WTP Lead Hand		1		1	**2**
WTP Operator		1		1	**2**
WTP Tech		1		1	**2**

Department	Working Off-Site	Day Shift	Night Shift	Off Rotation	Total
Paste Backfill Operator		1	1	2	**4**
Paste Backfill Helper		1	1	2	**4**
Maintenance Personnel					
Maintenance Supervisor		1		1	**2**
Maintenance Lead Hand		1	1	2	**4**
Maintenance Planner	1				**1**
Scheduler	1				**1**
Mechanical Reliability Engineer	1				**1**
Electrical Reliability Engineer	1				**1**
Mechanics		6	6	12	**24**
Welders		3	1	4	**8**
E&I Supervisor		1		1	**2**
Electrical Lead Hand		1		1	**2**
Electricians		3	3	6	**12**
Instrumentation Techs		1	1	2	**4**
E&I Planner	1				**1**
Scaffolders		2		2	**4**
Metallurgy and Laboratory					
Metallurgist		2		2	**4**
Junior Metallurgist		2		2	**4**
Metallurgical Technician		1	1	2	**4**
Chemist		1	1	2	**4**
Laboratory Technician		2	2	4	**8**
Total	**5**	**49**	**31**	**79**	**164**

21.3.4.3.2 Consumables

The cost of consumables was estimated to be $6.8M per year. Consumables consist of items such as screen panels, crusher liners, slurry pump parts, conveyor belts, chute liners and ball mill media.

21.3.4.3.3 Reagents

Reagents cost was estimated to be $5.3M per year. Reagents include FeSi and flocculent.

21.3.4.3.4 Energy and Utilities

The processing plant energy cost was estimated to be $7.9M per year with 131.2 MWh/y of energy consumption at a price of $0.0529/kWh. The processing plant power requirements were estimated on a unit-by-unit basis using vendor data and/or the Primero and GMS database.

21.3.4.3.5 Maintenance

Maintenance costs were estimated to be $9.3M per year. Maintenance spends included spend on operator tools, maintenance contractors for a shutdown, historian license, and additional third-party inspection and contractors for conveyor belts, overhead cranes, and fire detection and suppression.

21.3.4.3.6 Mobile Equipment

Mobile equipment cost was estimated to be $3.0M per year.

21.3.4.3.7 Tailing Rehandling

Tailings management involves loading, hauling and dozing of tailings from the plant to Stockpile only. The cost associated with this activity includes salaries, training and consumables.

21.3.4.3.8 Laboratory

Laboratory cost was estimated to be $3.4M per year. This cost includes slurry samples, solid samples, filter cake samples, solution samples, independent metallurgical testing, and other spend.

21.3.5 General and Administration Costs

General and administration costs (G&A) include all costs that are not directly linked to the daily operation of the mine. General Services include general management, accounting and finance, IT, environmental and social management, human resources, supply chain, camp, surface support, health and safety, security and operating cost of the various supply chain equipment. In most cases, these services represent fixed costs for the site as a whole.

The total LOM costs for the G&A were estimated to be $1,334.7M or approximately $100.69/t of concentrate produced. The average annual cost at peak production is approximately $76.7M.

A summary of the LOM G&A costs by sector is presented in Table 21.34.

Table 21.34: G&A Costs Summary

G&A OPEX Estimation	LOM Total Cost ($M)	Unit Cost ($/t processed)	Unit Cost ($/t conc)
Owner's Costs	112.79	1.34	8.51
General Management	14.76	0.18	1.11
Supply Chain	41.91	0.50	3.16
Human Resources	36.27	0.43	2.74
Security	15.12	0.18	1.14
Information Technology	33.40	0.40	2.52
Accounting / Finance	18.93	0.22	1.43
Freight	66.05	0.78	4.98
Customs, Taxes and Duties	54.71	0.65	4.13
Camp / Catering Costs	280.88	3.33	21.19
Employee Transportation Costs	381.06	4.52	28.75
Surface Support	126.90	1.51	9.57
G&A Mobile Equipment	27.31	0.32	2.06
Environment	52.92	0.63	3.99
Health & Safety	43.02	0.51	3.25
Insurance & Banking Fees	28.64	0.34	2.16
Total	**1,334.66**	**15.83**	**100.69**

From Year 18, given that the open pit mining operations will cease and only the UG and processing plant will be operating, the G&A costs will be reduced in some areas.

Table 21.35: General Services & Administration Cost Summary

G&A OPEX Estimation	LOM Total Cost ($M)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Owner's Costs	112.8	0.6	1.4	6.0	7.1	7.1	7.1	7.2	7.2	7.2	7.3	7.4	7.4	7.6	7.5	7.6	7.5	7.2	1.6	0.7
General Management	14.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.7	0.7
Supply Chain	41.9	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	1.1	1.1
Human Resources	36.3	2.2	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.2	0.6
Security	15.1	0.9	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8
Corporate	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Information Technology	33.4	2.0	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.3	1.0
Accounting / Finance	18.9	1.1	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.6	0.6
Freight	66.0	4.0	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	1.8	1.8
Customs, Taxes and Duties	54.7	2.7	2.5	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9
Camp / Catering Costs	280.9	13.4	14.6	16.4	16.6	16.7	16.2	16.0	15.9	15.9	15.8	15.4	15.4	14.8	14.9	14.8	14.7	14.3	10.5	8.6
Employee Transportation Costs	381.1	18.6	20.0	22.2	22.5	22.7	22.0	21.7	21.6	21.5	21.4	20.8	20.8	20.0	20.1	20.0	19.8	19.3	14.5	11.6
Surface Support	126.9	7.0	6.5	6.5	6.6	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	5.9	5.9
G&A Mobile Equipment	27.3	1.5	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4
Environment	52.9	3.1	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.3	2.2
Health & Safety	43.0	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.1	2.0
Insurance & Banking Fees	28.6	1.6	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Total	**1,334.7**	**64.4**	**65.5**	**74.5**	**76.3**	**76.7**	**75.5**	**75.1**	**74.9**	**74.9**	**74.7**	**73.7**	**73.7**	**72.6**	**72.7**	**72.6**	**72.1**	**71.0**	**50.3**	**43.5**
Total Cost/t Tonnes Processed	15.8	24.5	20.6	15.1	15.0	15.1	14.9	14.8	15.7	15.0	14.6	15.0	14.5	14.6	14.7	14.5	15.0	15.7	23.6	29.8
Total Cost/t Concentrate (dry)	100.7	176.8	154.5	102.5	95.9	95.7	94.2	93.7	93.4	93.4	95.7	92.0	98.6	90.6	90.7	90.6	90.0	88.6	159.0	208.3

21.3.6 Energy Costs

21.3.6.1 Electricity

The electrical consumption required for the overall operation is presented in Chapter 18. Based on the estimated consumption, the electrical costs were calculated at a rate of $0.0529/kWh. The electrical costs shown in Table 21.36 are the extract of all areas power cost, already considered previously in operating costs. It must not be double counted.

Table 21.36: Electrical Costs

WBS	Description	Total Cost by WBS (CAD)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21
			2030	2031	2032	2033	2034	2035	2036	2037	3038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050
100	Infrastructure	**30.88**	1.11	1.25	1.25	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.25	1.25
200	Power & Electrical	**3.13**	0.11	0.11	0.11	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15
300	Water & Tailings Management	**20.80**	0.75	0.75	0.75	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	0.90	0.90
520	Mining - U/G Mine Surface Services Infrastructure	**54.93**	0.98	1.82	2.10	2.24	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80	2.80
540	Mining - U/G Mine Services	**38.20**	0.79	0.89	1.09	1.48	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98	1.98
550	Mining - U/G Mine Equipment	**3.25**	0.04	0.15	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16
580	Mining - OP Mine Equipment	**2.67**	0.06	0.06	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12	0.12
590	Paste Preparation Plant	**30.99**	0.81	1.21	1.29	1.46	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	0.97	0.97
600	Process Plant - Process (Phase 1)	**78.63**	3.33	3.52	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71
600	Process Plant - Process (Phase 2)	**41.01**	-	0.37	1.48	2.22	2.35	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	-	-
680	Process Plant - Paste Material Preparation	**14.09**	-	0.38	0.58	0.69	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.46	0.46
700	Construction Indirect	**1.88**	0.32	0.32	0.32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Cost (CAD)		**318.57**	**8.31**	**10.84**	**12.96**	**14.61**	**16.03**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**16.15**	**12.49**	**12.49**

21.3.6.2 Compressed Natural Gas

The heating of the surface infrastructures, namely process plant building, paste plant, mine garage and camp module, and the underground mine will be guaranteed by compressed natural gas (CNG) heaters. Average yearly consumption of CNG is approximately 87,691 GJ or 2.5 Mm^3 per year, at a cost of $47.71/GJ. Therefore, the total yearly cost of CNG for heating is estimated at $7.1M annually. The transportation to site of CNG is provided by the gas supplier and is included in the supply cost. Those costs are included in the UG mining and processing plant operating costs and must not be double counted.

21.3.7 Concentrate Transportation Costs

All the spodumene concentrate will be transported by a reputable contractor with extensive experience on the Trans-Taiga and the Billy-Diamond roads. The estimated concentrate transportation cost from the mine site to Grande Anse is $217.19/t dry of concentrate ($206.85/t wet). This includes the material transfer from the trucks into railcars at the Matagami Transshipment Centre. The transportation price excludes the costs for the driver's accommodation and travel to the mine site. Those costs are captured under the G&A costs. The cost per tonne is considering a moisture content of 5%. Transshipment in Matagami will be serviced by another contractor and railway transportation will be serviced by Canadian National (CN). The estimated concentrate transportation costs are presented in Table 21.37.

Table 21.37: Concentrate Transportation Costs

Activities	Cost $/t Wet	Cost $/t Dry
Truck Transportation from Mine Site to Matagami	95.0	99.8
Transshipment in Matagami from Truck to Railcars	17.00	17.8
Rail Transportation from Matagami to Grande Anse	64.6	67.9
Port Operation[1]	30.20	31.7
Total	206.8	217.2

**Notes: 1) Port operation is estimated at $29.00/t when producing 400 kt per annum and $31.50/t when producing 800 kt per annum.*

22. ECONOMIC ANALYSES

22.1 Overview

The economic and financial evaluation presented in this Technical Report is based on a discounted cash flow (DCF) analysis prepared on both a pre-tax and after-tax basis, using spodumene concentrate grading 5.5% Li_2O as the sole sales product. The commodity price assumptions applied in the evaluation are detailed in Section 19. Key financial indicators generated by the model include Net Present Value (NPV), Internal Rate of Return (IRR), and payback period for the Shaakichiuwaanaan Project (lithium only at the CV5 Pegmatite). The analysis is presented in real terms, excluding the effects of inflation, and is expressed in Q3 2025 Canadian dollars, unless otherwise stated.

Project financing is not considered in this evaluation. The economic model projects annual cash flows over the Life of Mine (LOM), consistent with the level of engineering and design appropriate for a feasibility study. Cash flow estimates incorporate revenues from spodumene concentrate sales, operating expenditures (OPEX), capital expenditures (CAPEX), and other costs. CAPEX is categorized into four (4) components: initial capital, sustaining capital, closure and reclamation costs, and working capital. OPEX includes labour, consumables, maintenance supplies, services, fuel, and power. Additional costs, such as royalties, depreciation, and taxes, are based on the current project cashflow.

The economic results are calculated effective January 1, 2028. The tax impact was determined by an external tax expert, applying current Canadian tax regulations.

The internal rate of return (IRR) on total investment was calculated based on 100% equity financing. The net present value (NPV) was calculated from the cash flow generated by the Project, using a discount rate of 8%. The payback period, based on the undiscounted annual cash flow of the Project, is also reported as a financial measure. The payback period starts after the initial capital is spent to commence concentrator plant production. Figure 22.1 presents the different sensitivities on a tornado graph for the post-tax NPV at 8%. Table 22.1 presents the economic results on both a pre-tax and after-tax basis.

Furthermore, a sensitivity analysis was performed for the pre-tax base case and after-tax base case to assess the impact of variations in Li_2O grade, exchange rate, commodity prices, recoveries, operating costs, initial and sustaining capital costs on NPV and IRR.

Figure 22.1: Tornado Graph for the Sensitivity Analysis (+/-20%)

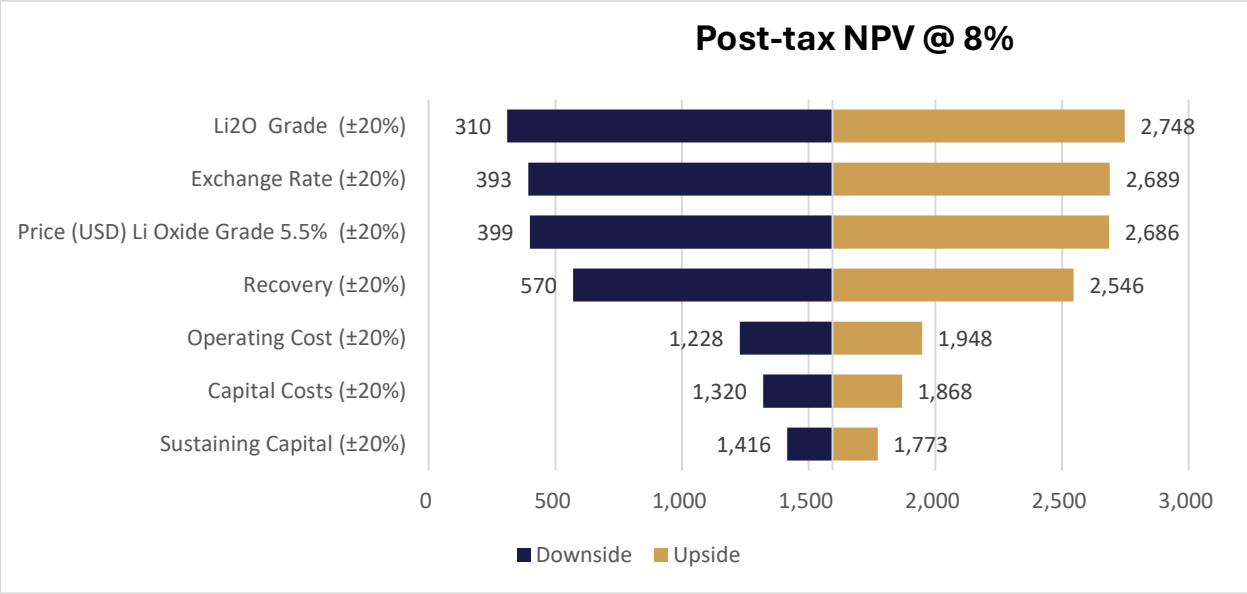


Table 22.1: Base Case Economic Result

Description	Unit	CA$	US$
Long-term Price Assumption 5.5% Li_2O (Grande Anse POL)	$/t	1,636	1,221
Pre-Tax NPV 0%	$M	8,358	6,237
Pre-Tax NPV 8%	$M	2,514	1,876
After-Tax NPV 0%	$M	5,418	4,043
After-Tax NPV 8%	$M	1,594	1,190
Pre-Tax IRR	%	19.87%	
After-Tax IRR	%	18.06%	
Pre-Tax Payback Period	year	4.9	
After-Tax Period	year	4.7	

22.2 Cautionary Statement

The results of the economic analyses presented in this section represent forward-looking information as defined under Canadian securities laws. Such information is inherently subject to known and unknown risks, uncertainties, and other factors that could cause actual outcomes to differ materially from those anticipated. Forward-looking information in this section includes, but is not limited to, the following assumptions and risks:

- Future prices of spodumene concentrate.

- CAD/USD Exchange rate.
- The availability of the CTM-ITC and the TCRR tax credits.
- The proposed mine production plan, including assumptions regarding mining dilution and recovery.
- Recovery rates at the processing plant.
- Proposed capital, sustaining capital, and operating cost estimates, including assumptions regarding cost inflation, and labour and material costs being approximately consistent with those set out in this Report.
- Availability of labour and materials.
- Changes to applicable tax rates.
- Assumptions regarding closure and reclamation costs.
- Environmental, social, and permitting risks, including the ability to obtain and maintain a social licence to operate.
- Variations in the quantity of mineralized material and ore grades relative to assumptions.
- Geotechnical or hydrogeological conditions encountered during mining that may differ from assumptions.
- Unanticipated environmental risks.
- Failure of plant, equipment, or processes to operate as anticipated.
- Significant disruptions impacting the development and operation of the Shaakichiuwaanaan Project.
- Time and cost required to develop the Project in accordance with the Feasibility Study mine design.
- General business and macroeconomic conditions, including spodumene concentrate market dynamics and the capacity to sell this concentrate.

22.3 Key Assumptions

22.3.1 Mineral Reserve

The Life-of-Mine (LOM) production schedule has been derived from the Project's Probable Mineral Reserves. The Probable Reserves for the Shaakichiuwaanaan Project, which includes the CV5 Pegmatite only, are estimated at 84.3 Mt at 1.26% Li_2O for 2.62 Mt of contained lithium carbonate equivalent (LCE), as summarized in Table 22.2.

Table 22.2: Shaakichiuwaanaan Project Mineral Reserve

Area	Category	Tonne (Mt)	Grade (%Li_2O)	Contained Li_2O (kt)	Contained Lithium (Li) (kt)	Contained LCE (kt)
Open Pit	Proven	-	-	-	-	-
	Probable	49.2	1.12	551.9	256.4	1,364.7
	Proven and Probable	**49.2**	**1.12**	551.9	**256.4**	1,364.7
Underground	Proven	-	-	-	-	-
	Probable	35.1	1.45	508.0	236.0	1,256.0
	Proven and Probable	**35.1**	**1.45**	508.0	**236.0**	1,256.0
Total (Open Pit + Underground)	Proven	-	-	-	-	-
	Probable	84.3	1.26	1,059.9	492.4	2,620.7
	Proven and Probable	**84.3**	**1.26**	1,059.9	**492.4**	2,620.7

**Notes on Mineral Reserves:*

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).*
2. *The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. As such, the Mineral Reserves are based on Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.*
3. *Mineral Reserves are estimated using a long-term lithium price of US$1,303/t of spodumene concentrate @ 5.5% Li_2O and an exchange rate CAD/USD of 1.32.*
4. *The Qualified Person for the estimate is Carl Michaud, Ing, MBA. The estimate has an Effective Date of September 11, 2025.*
5. *The Mineral Reserves for open pit are estimated using a cut-off grade of 0.40% Li_2O. Open-pit marginal material containing a grade above 0.37% Li_2O is also included within this statement. Mineral Reserves for underground stoping are estimated using a cut-off grade of 0.70%. Underground development tonnages containing material above 0.37% Li_2O are also included in the statement.*
6. *The following mill recovery equation was used in the cut-off grade recovery:*

$$Mill\ Recovery = \frac{75 * (1 - e^{-1.995 * Li2O\ feed\ Grade})}{100}$$

7. *The open pit strip ratio is 3.40, and the dilution factor is 2.0% based on the smallest mining unit (SMU). The OP mine mining recovery is 97%.*
8. *The underground mine average external dilution factor is 12.7% including 3.9% for backfill dilution and 8.8% for ELOS dilution.*
9. *For the underground Mineral Reserves, a minimum mining width of 5 m was applied with a mining recovery of 90% for all stopes, while 100% extraction was assumed for all development mining.*
10. *Contained Lithium Oxide (Li_2O), lithium, lithium carbonate equivalent (LCE) are reported without accounting for metallurgical recovery*
11. *Total may not sum due to rounding; rounding followed the recommendations of the NI 43-101.*

22.3.2 Spodumene Price

The price forecast for spodumene concentrate 6% Li_2O is based on projections from the 2025 lithium market study presented in Section 19 and is estimated at US$ 1,332/t (FOB/FCA Australia). A penalty of US$ 111/t has been applied to reflect the lower lithium oxide grade, with the final product containing 5.5% Li_2O instead. Table 22.3 presents the concentrate price.

Table 22.3: Li_2O Concentrate Price

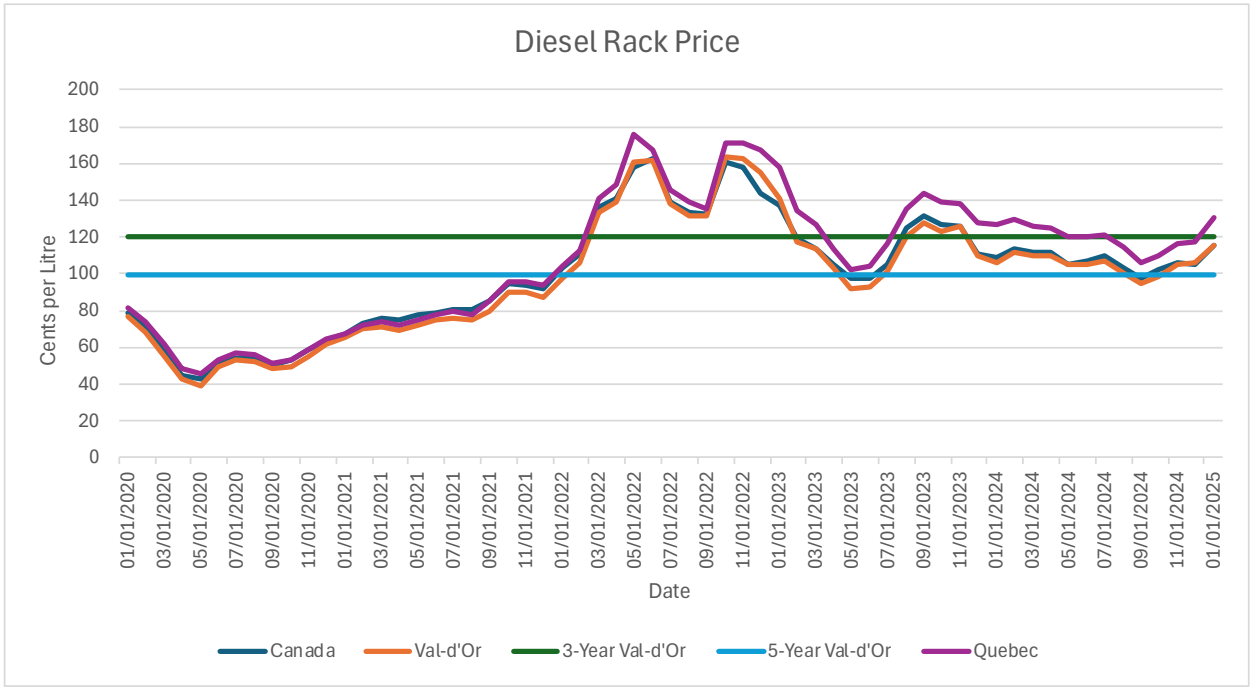
Commodity Prices	Unit	Price
6.0% Li_2O Concentrate	US$/t	1,332
5.5% Li_2O Concentrate	US$/t	1,221

22.3.3 Fuel Price

The reference diesel fuel price used for estimating operating costs is CA$1.43/L, which is an estimated delivered price to site for coloured diesel destined for off-road vehicles. It is exclusive of provincial road taxes and sales taxes, which are reimbursable, but includes the federal excise tax.

The reference price is benchmarked from the Val-d'Or and Québec price for ultra-low sulfur diesel no. 1. The price assumption is based on a flat average rate between the 2020 and 2025 rates. The average Val-d'Or rack price is 99.5 ¢/litre, on which the transportation cost and taxes were applied. The 5-year Montreal Fuel Rack Price is detailed in Figure 22.2.

Figure 22.2: Price Evolution of Fuel Rack Price Over the Last Five Years (per month)



Source: NRCAN website 2025.

However, the average cost includes the fuel cost drop that occurred in early 2020 due to the worldwide pandemic situation, as well as the post-pandemic economic rebound, which raised the prices for several months. As seen in the data analyzed, the fuel rack price is gradually coming back to a steadier price

starting at the end of 2023 for the whole of 2024. It can also be seen that Quebec's rack price has been higher than Val-d'Or's for the last months. Val-d'Or is used for the NRCan rack price; it is suggested to use this data as a benchmark for fuel costs. Considering current economic uncertainties, it is recommended to use the 3-year average instead of the 2-year average. The latter one is considered too optimistic. Averages wholesale price for diesel at the Val-d'Or rack are presented in Table 22.4.

Table 22.4: Monthly Average Wholesale (Rack) Prices (CA$/L) for Diesel in Val-d'Or (NRCan)

City	5-year Average	3-year Average	2-year Average
Val-d'Or	0.995	1.200	1.098

The cost of fuel to be used in this study is the cost of diesel fuel at the mine site. The transportation cost corresponds to the cost of transporting the fuel from Val-d'Or to the mine site fuel farm. Fuel transportation will be by rail up to Matagami, and then by truck to the mine site. It has been assumed that rack prices in Val-d'Or and Matagami are the same. No supplier discounts are included in the calculations; however, a discount is usually granted based on purchased quantities.

Table 22.5 presents the transportation cost and any applicable taxes that are added to the rack price. GMS selected the taxation rate associated with clear fuel and coloured fuel. The first one is dedicated to all vehicles going on public roads, and the second one can be typically used on non-licensed vehicles (vehicles that do not go on public roads), under which category all the mining equipment falls. In the case of the Shaakichiuwaanaan Project, it will be very important to identify the right fuel for each consumption, as several highway transports and other vehicles will replenish on site each day and circulate on the Trans-Taiga Road. GST and QST have been removed due to tax credits. Being used on site only, coloured fuel does not have a provincial road tax. Table 22.5 shows the fuel analysis.

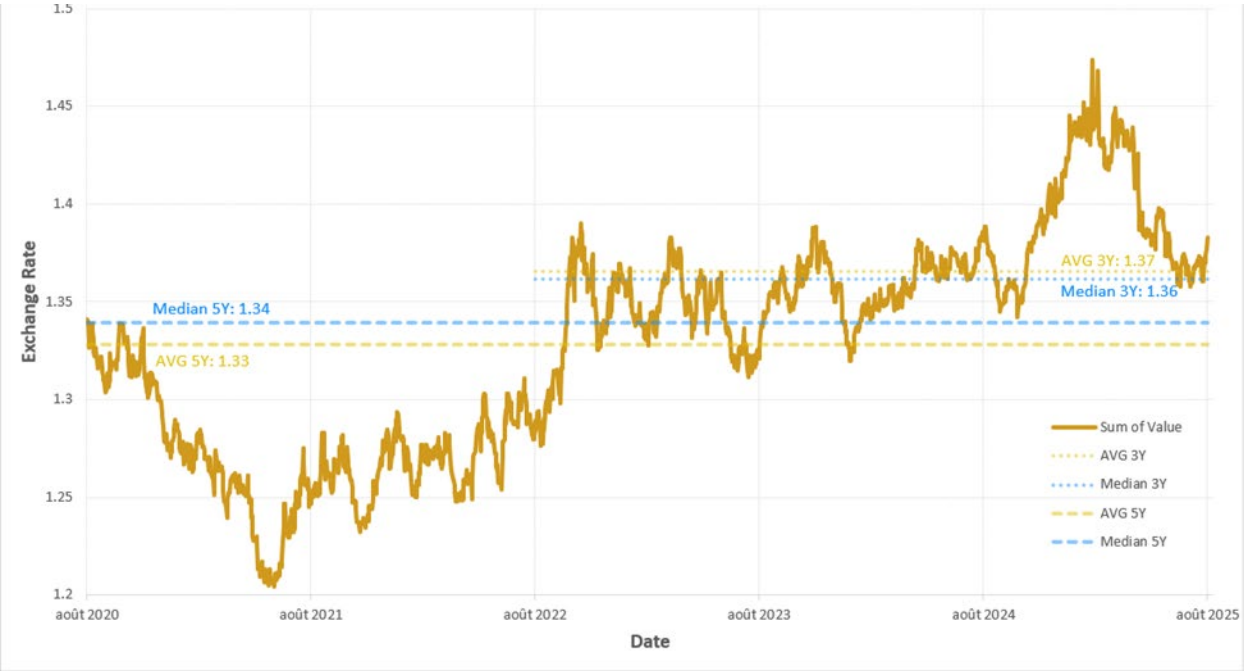
Table 22.5: Diesel Fuel Price Analysis

Fuel Type	Units	Clear Fuel	Coloured Fuel
Val-d'Or Rack Price Estimate	$/L	1.2000	1.2000
Transport Rack to Site	$/L	0.1936	0.1936
Excise tax	$/L	0.0400	0.0400
Provincial Road Tax	$/L	0.1638	-
GST	$/L	-	-
QST	$/L	-	-
Sum Cost (Cost On-site, Rounded)	$/L	1.60	1.43

22.3.4 Exchange Rate

The base case Canadian dollar exchange rate for economic evaluation is CAD/USD 1.34 based on the 5-year median exchange rate. Most operating costs are estimated in Canadian dollars, with the US dollar-denominated spodumene revenue converted to Canadian dollars. Figure 22.3 shows the CAD/USD exchange rate over the last 5 years.

Figure 22.3: Five-Year Exchange Rate



Source: Metal Price API, 2025.

22.3.5 Metal Recovery

Mineralogical, dense media separation (DMS), heavy liquid separation (HLS), flotation, filtration, and gravity separation test work was conducted by SGS Canada at its Lakefield, Ontario facility. The test program utilized drill core composite samples from the CV5 Pegmatite, CV5 host rock, and CV13 Pegmatite. Metallurgical recovery has been determined based on the results of the SGS test work and the process plant design, from which the following recovery formula has been derived. The detailed test results are presented in Section 13 of this Report.

$$Concentrate\ Recoveries = 0.75 * (1 - EXP(-1.995 * \%Li_20))$$

The design weighted average overall plant recovery is 68.9%.

22.3.6 Other Assumptions

The other key assumptions used in economic analysis are as follows:

- Discount rate 8%.
- All cost estimates are in constant H1 2025 United States dollars with no inflation or escalation factors considered.
- The economic results are calculated effective January 1, 2028. All costs incurred prior to this date are escalated / updated to this date.
- The existing sales contract has been applied to the entire production of the Project.

22.4 Spodumene Concentrate Production and Revenues

Spodumene concentrate production over the Project life is 13,255 dry-kt with an average annual spodumene concentrate production of 795 dry-kt. The spodumene concentrate gross revenue during operations is CA$21,686M. This study assumes an owner mining operation. The design weighted average overall plant recovery is 68.9%. The concentrate production is summarized in Figure 22.4. The annual mine and mill production is summarized in Figure 22.5 to Figure 22.7 and Table 22.6.

Figure 22.4: Annual Spodumene Concentrate (5.5% Li2O) Production

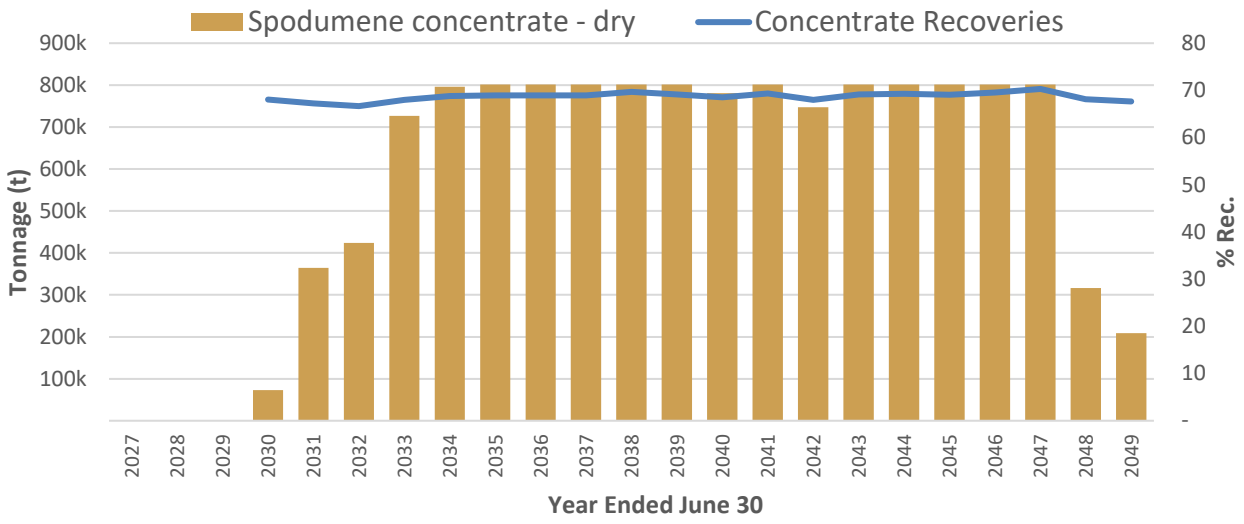


Figure 22.5: Open Pit Mine Production Profile

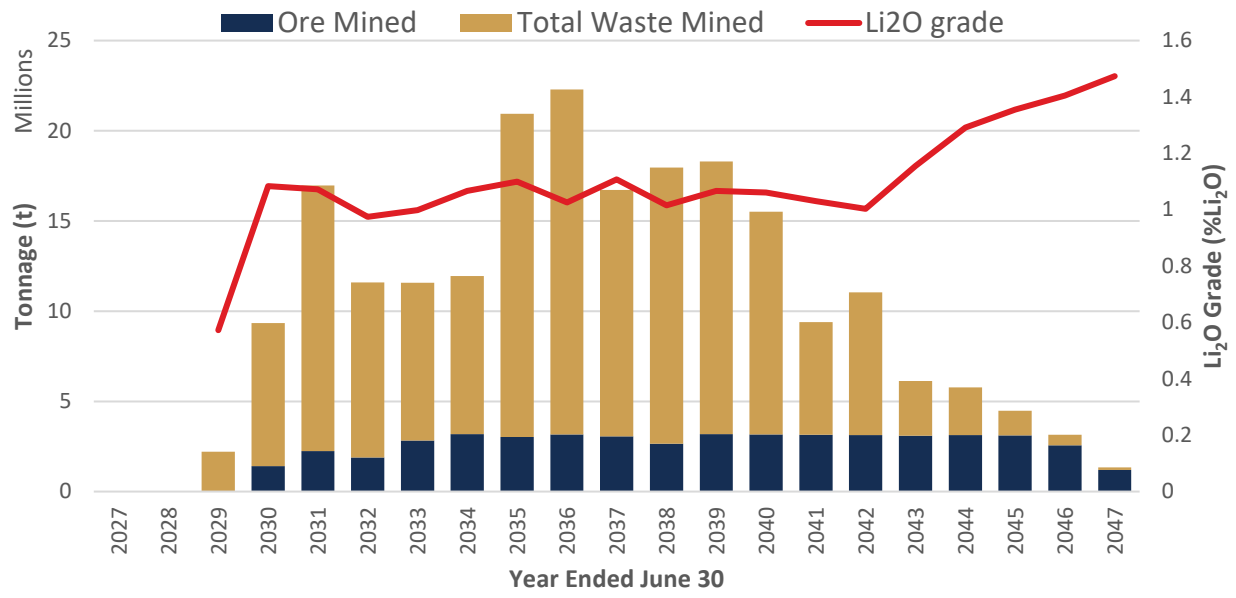


Figure 22.6: Underground Mine Production Profile

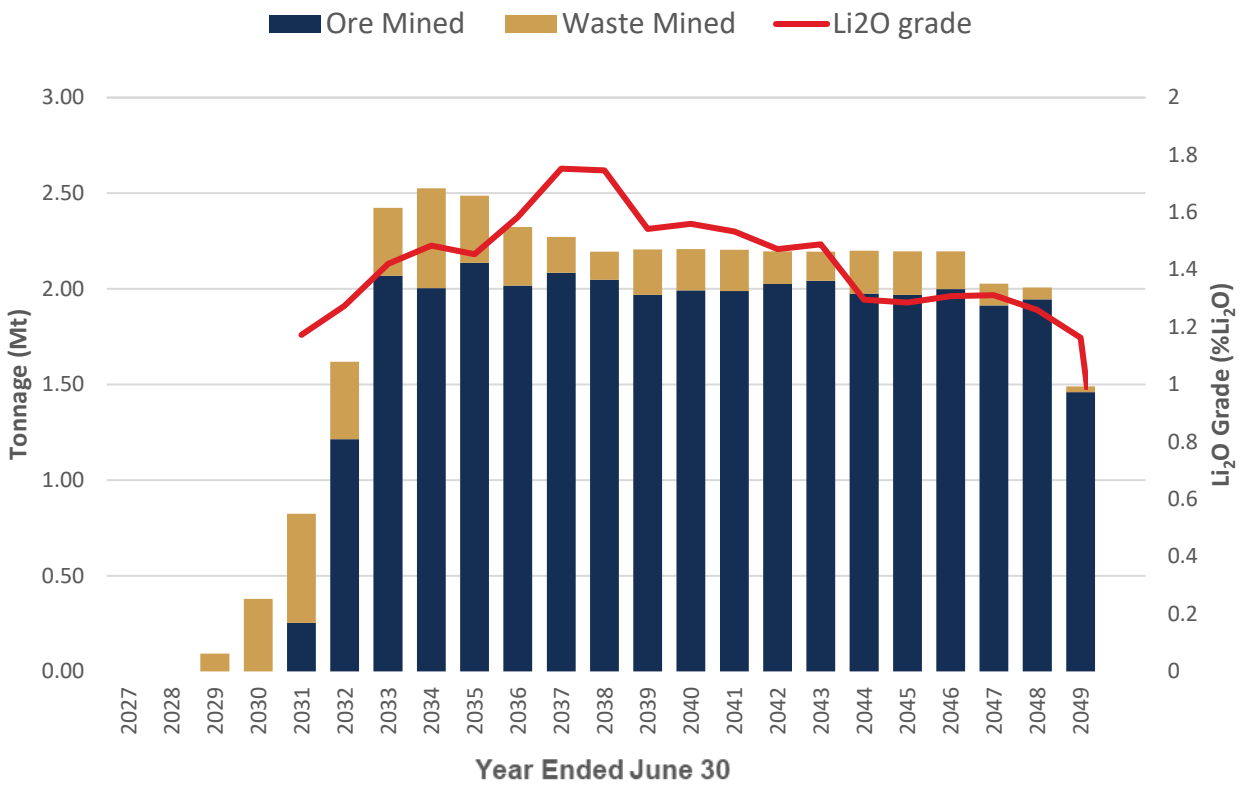


Figure 22.7: Process Production Profile

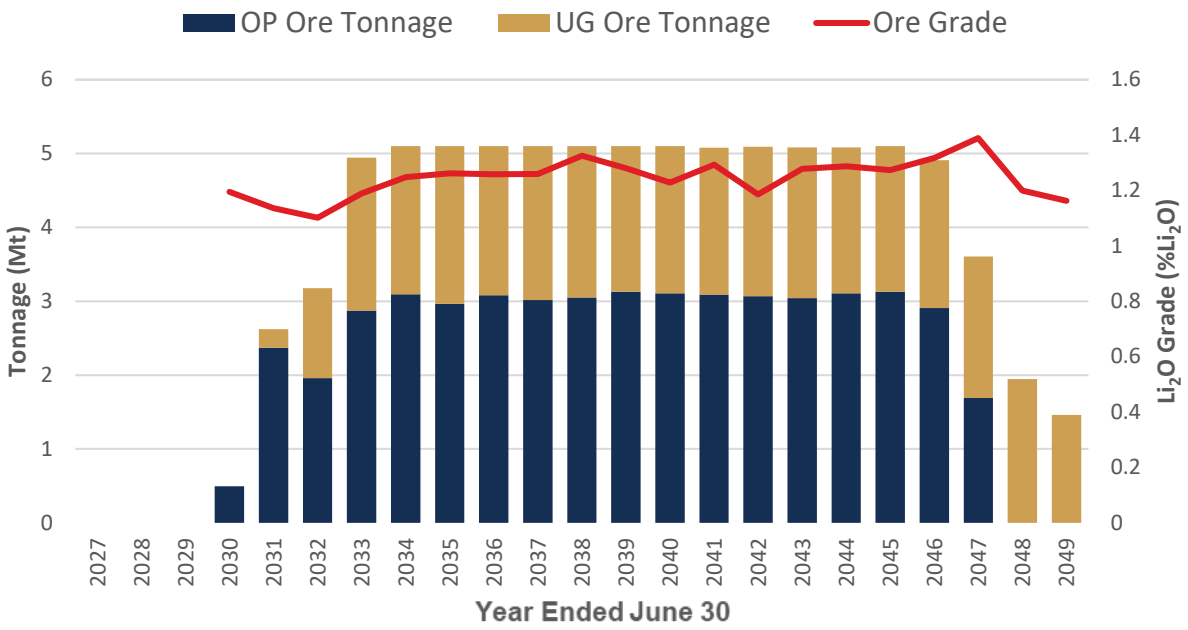


Table 22.6: Annual Mine and Mill Production Summary

Description		Total	2028 - 2029	2029 - 2030	2030 - 2031	2031 - 2032	2032 - 2033	2033 - 2034	2034 - 2035	2035 - 2036	2036 - 2037	2037 - 2038	2038 - 2039	2039 - 2040	2040 - 2041	2041 - 2042	2042 - 2043	2043 - 2044	2044 - 2045	2045 - 2046	2046 - 2047	2047 - 2048	2048 - 2049
Concentrate Production																							
Tonnage Processed	*Mt*	84.3	-	0.50	2.62	3.17	4.94	5.10	5.07	5.08	5.08	4.78	4.98	5.10	4.92	5.10	4.99	4.95	5.01	4.81	4.51	2.13	1.46
Head Grade	*% Li_2O*	1.26	-	1.19	1.14	1.10	1.19	1.25	1.26	1.26	1.26	1.33	1.28	1.23	1.29	1.19	1.28	1.29	1.27	1.32	1.39	1.20	1.16
Contains Li_2O	*kt Li_2O*	1,059	-	5.9	29.8	35.0	58.8	63.6	63.9	64.0	64.0	63.3	63.7	62.7	63.6	60.5	63.8	63.7	63.8	63.4	62.7	25.5	17.0
Contained Li	*kt Li*	491	-	2.7	13.8	16.2	27.3	29.5	29.7	29.7	29.7	29.4	29.6	29.1	29.5	28.1	29.6	29.5	29.6	29.4	29.1	11.8	7.9
Recovery	*%*	68.87	-	68.08	67.22	66.67	68.00	68.78	68.94	68.91	68.93	69.67	69.17	68.54	69.31	67.96	69.14	69.25	69.09	69.58	70.31	68.15	67.62
Recovered Li_2O	*kt Li_2O*	729	-	4	20	23	40	44	44	44	44	44	44	43	44	41	44	44	44	44	44	17	11
Recovered Li	*kt Li*	338	-	2	9	11	19	20	20	20	20	20	20	20	20	19	20	20	20	20	20	8	5
Concentrate Production @ 5.5% Lithium Oxide	*kt-Conc Dry*	13,255	-	73	364	424	726	796	802	802	802	802	802	781	802	748	802	802	802	802	802	316	209
Mine Production																							
OP - Overburden Mined	*Mt*	17.5	2.2	5.4	5.5	0.0	1.0	1.1	2.1	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-
OP - Waste Rock Mined	*Mt*	150.0	0.0	2.6	9.2	9.7	7.7	7.7	15.8	18.9	13.7	15.3	15.1	12.3	6.3	7.9	3.0	2.6	1.4	0.6	0.1	-	-
OP - Ore Mined	*Mt*	49.2	0.0	1.4	2.2	1.9	2.8	3.2	3.0	3.2	3.1	2.6	3.2	3.2	3.1	3.1	3.1	3.1	3.1	2.6	1.2	-	-
OP - Total Mined	*Mt*	216.7	2.2	9.3	17.0	11.6	11.6	11.9	20.9	22.3	16.7	18.0	18.3	15.5	9.4	11.0	6.1	5.8	4.5	3.1	1.3	-	-
OP - Strip Ratio	*W:O*	3.40	-	5.62	6.58	5.14	3.09	2.75	5.91	6.05	4.46	5.78	4.75	3.89	1.99	2.52	0.99	0.84	0.43	0.23	0.12	-	-
UG - Waste Rock Mined	*Mt*	5.2	0.1	0.4	0.6	0.4	0.4	0.5	0.4	0.3	0.2	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.0
UG - Ore Mined	*Mt*	35.1	-	-	0.3	1.2	2.1	2.0	2.1	2.0	2.1	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.9	1.9	1.5
UG - Total Mined	*Mt*	40.3	0.1	0.4	0.8	1.6	2.4	2.5	2.5	2.3	2.3	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.0	2.0	1.5

22.5 Royalties

The Project is subject to a 2% revenue-based royalty payable to DG Resource Management Ltd. and OR Royalties Inc. (formerly known as Osisko Gold Royalties Inc.). For this Technical Report, the royalty applicable to the CV5 Pegmatite of the Shaakichiuwaanaan Project has been modelled on a net smelter return (NSR)-like basis. The calculation is defined as gross sales less concentrate transport cost and transport losses.

An estimated $374.9 M in royalties will be paid over the LOM.

22.6 Transportation Loss

A transportation loss of 0.2% of lithium concentrate has been assumed to account for product losses during loading and unloading at the various transport points (mine site, Matagami, and Grande Anse). Over the Life of Mine (LOM), this transportation loss represents an estimated revenue reduction of $45.5 million.

22.7 Capital Expenditures

The capital expenditures include initial capital expenditures as well as the sustaining capital expenditures to be invested after the commencement of commercial operations.

Table 22.7: Initial and Expansion & Sustaining Capital Expenditures Summary

Capital Expenditure	Phase 1 OP Initial Capital Cost ($M)	Phase 2 UG Initial Capital Cost ($M)	Initial Capital Cost ($M)	Phase 2 UG Expansion Capital Cost ($M)	Total Devel. Capital Cost ($M)	LOM Sustaining Capital Cost ($M)	Total Capital Cost ($M)
100 - Infrastructure	124.9	-	**124.9**	24.8	**149.7**	30.8	**180.5**
200 - Power and Electrical	173.8	-	**173.8**	46.2	**220.0**	25.0	**245.1**
300 - Water Management	128.2	-	**128.2**	18.7	**146.9**	100.5	**247.4**
400 - Surface Operations	18.6	-	**18.6**	-	**18.6**	11.9	**30.5**
500 - Mining	120.0	99.1	**219.1**	36.4	**255.5**	550.5	**806.0**
600 - Process Plant	217.3	20.1	**237.4**	167.0	**404.4**	-	**404.4**
700 - Construction Indirect	262.8	0.1	**262.9**	123.8	**386.7**	-	**386.7**
800 - General Services / Owner's Cost	99.8	4.7	**104.5**	13.4	**117.9**	31.6	**149.6**
900 - Pre-production, Start-up, Comm.	73.3	9.3	**82.6**	1.5	**84.1**	186.1	**270.2**
Total Initial Capital Expenditures Excluding Contingency	**1,218.7**	**133.3**	**1,352.0**	**431.8**	**1,783.8**	**936.4**	**2,720.2**
990 - Contingency	130.7	15.0	**145.7**	48.7	**194.4**	-	**194.4**
Total Initial Capital Expenditures	**1,349.4**	**148.3**	**1,497.7**	**480.5**	**1,978.2**	**936.4**	**2,914.6**
Less: Pre-Prod. Credit Net of TC / RC & Royalties	**(101.7)**	-	**(101.7)**	-	**(101.7)**	-	**(101.7)**
Total Initial CAPEX Net of Pre-Production Crédit	**1,247.7**	**148.3**	**1,396.0**	**480.5**	**1,876.5**	**936.4**	**2,813.0**

22.7.1 Initial Capital

The initial CAPEX for Project construction, including processing facilities, mine equipment purchases, pre-production activities, infrastructure and other direct and indirect costs, is estimated to be $1,497.7M before pre-production concentrate sales.

The initial capital includes all expenditures for Phase 1 up to commercial production for the Phase 1 (June 1st, 2030) whereby development of the open-pit mine site and construction of first stage of the process plant (designed to nominally treat 2.5 Mtpa of lithium ore that corresponds to 400,000 t of lithium concentrate per year) is also complete. Initial capital expenditure also includes 23% of the capital cost of Phase 2 and allocation for the full 800,000 tpy spodumene production. For instance, upstream connection to Hydro-Québec's network and portions of water treatment system are fully considered in the initial capital required.

The initial capital cost totals CA$1,497.7M (including CA$1,218.7 million of Phase 1 initial CAPEX and CA$133.3 million of Phase 2 initial CAPEX, plus a contingency of CA$145.7M, and excluding pre-production revenue of CA$101.7M). The engineering, early works, construction, pre-production and commissioning periods will be carried out over 41 months due to certain long-lead items required in the early stages of the Project.

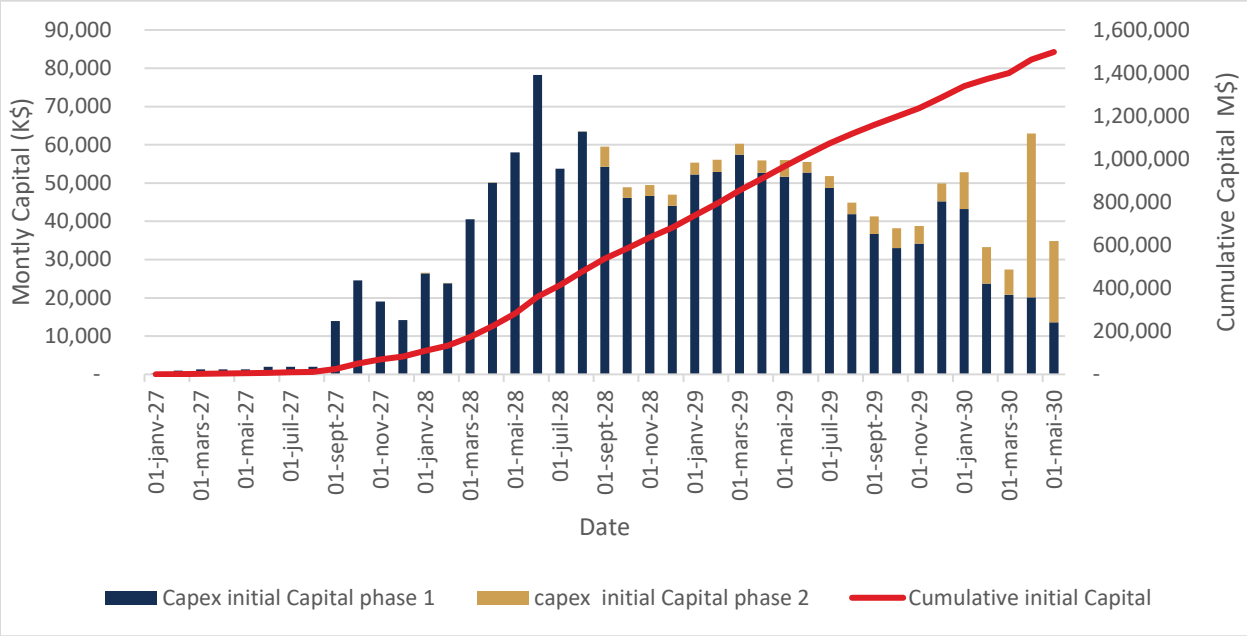
The total initial CAPEX includes a contingency of $145.7M, which is 11% of direct and indirect expenditures.

Table 22.8 summarizes the initial capital costs, while Figure 22.8 illustrates the timing of the expense of initial capital expenditures. The total initial capital cost net of pre-production credit is estimated at $1,396.0M.

Table 22.8: Initial Capital

Initial Capital Expenditure	Phase 1 OP Initial Capital Cost ($M)	Phase 2 UG Initial Capital Cost ($M)	Initial Capital Cost ($M)
100 – Infrastructure	124.9	-	124.9
200 – Power and Electrical	173.8	-	173.8
300 – Water Management	128.2	-	128.2
400 – Surface Operations	18.6	-	18.6
500 – Mining	120.0	99.1	219.1
600 – Process Plant	217.3	20.1	237.4
700 – Construction Indirect	262.8	0.1	262.9
800 – General Services / Owner's Cost	99.8	4.7	104.5
900 – Pre-production, Start-up, Comm.	73.3	9.3	82.6
Total Initial Capital Expenditures Excluding Contingency	**1,218.7**	**133.3**	**1,352.0**
990 – Contingency	130.7	15.0	145.7
Total Initial Capital Expenditures	**1,349.4**	**148.3**	**1,497.7**
Less: Pre-Prod. Net Credit of TC / RC & Royalties	(101.7)	-	(101.7)
Total Initial Net CAPEX of Pre-Production Credit	**1,247.7**	**148.3**	**1,396.0**

Figure 22.8: Initial CAPEX by Month



22.7.2 Expansion and Sustaining Capital

The sustaining capital requirements for the Project are divided into two (2) principal components:

- Site expansion expenditures (Phase 2): This includes expenditures related to the expansion phase 2, which follows the commencement of commercial production from Phase 1. Phase 2 spans a 22-month period, from June 2030 through March 2032, and
- Life of mine sustaining capital: These costs involve the acquisition, replacement, or major overhaul of assets necessary to maintain ongoing operations. The complete sustaining costs period span from June 2030 to June 2049.

The Phase 2 expansion Capital after the commercial production is estimated at $480.5M, and the LOM Sustaining Capital is estimated at $936.5M. The total sustaining capital for the Project is therefore estimated at $1,417.0M, as detailed in Table 22.9.

Table 22.9: Expansion and Sustaining Capital

Capital Expenditure	Phase 2 UG Expansion Capital Cost ($M)	LOM Sustaining Capital Cost ($M)	TOTAL Sustaining Capital Cost ($M)
100 – Infrastructure	24.8	30.8	55.6
200 – Power and Electrical	46.2	25.0	71.3
300 – Water Management	18.67	100.5	119.1
400 – Surface Operations	-	11.7	11.7
500 – Mining	36.4	550.5	586.9
600 – Process Plant	167.0	-	167.0
700 – Construction Indirect	123.8	-	123.8
800 – General Services / Owner's Cost	13.4	31.6	45.01
900 – Pre-production, Start-up, Comm.	1.45	186.1	187.6
Total Capital Expenditures Excluding Contingency	**431.8**	**936.4**	**1,368.3**
990 – Contingency	48.7	-	48.7
Total Capital Expenditures	**480.5**	**936.4**	**1,417.0**

22.7.3 Working Capital

Working capital is required to finance supplies and inventory. A 5-day inventory of fuels (HFO and LFO) is planned, while storage of other main consumables is estimated at one (1) month. This represents a maximum inventory value of $29.8M. In addition, operational suppliers are assumed to be paid within 30 days, and accounts payable are included in the working capital calculation.

For receivables, it is assumed that 90% of revenues will be payable upon delivery at the Port of Grande-Anse, with the remaining 10% payable on a net 60-day basis following receipt. For the purpose of this study, it has been assumed that the terms of the offtake term sheet described in Chapter 19 for 100,000 t of concentrate apply to the full life of mine.

22.8 Reclamation & Closure Cost

Reclamation and closure costs include the decommissioning of infrastructure, site contouring and revegetation, as well as maintenance and post-closure monitoring. The total reclamation and closure cost is estimated at $248.4M, as detailed in Section 21. In addition, an insurance bonding mechanism is

planned, with an estimated cost of $64.6M. In the financial model, three (3) payments (Québec regulation requirement) are scheduled for 2028, 2029 and 2030. Once the Project begins generating profits, the insurance mechanism may be implemented, at which point insurance fee payments would commence, and the withdrawal of the deposited funds could be executed.

22.9 Salvage Value

After nearly 20 years of operation, GMS assume that there is no residual value at the end of the mine life.

22.10 Operating Cost Summary

Operating costs are presented by year in Table 22.10. The operating costs include mining, processing, general services and administration (G&A) and concentrate transportation. The average LOM operating cost is $729/t-dry concentrate 5.5% Li_2O or $114.6/t processed.

Table 22.10: Operating Cost Summary

Description	Total Operating Cost	2030 2031	2031 2032	2032 2033	2033 2034	2034 2035	2035 2036	2036 2037	2037 2038	2038 2039	2039 2040	2040 2041	2041 2042	2042 2043	2043 2044	2044 2045	2045 2046	2046 2047	2047 2048	2048 2049
Total Cost ($M)																				
Open Pit Mining	1,390	84	87	84	83	97	107	94	97	98	90	75	79	63	62	60	55	49	14	12
Underground Mining	2,852	71	131	175	166	167	162	161	158	152	155	156	157	154	156	154	157	153	151	114
Processing	1,208	43	43	66	70	70	70	70	69	70	70	69	70	68	68	68	67	64	50	46
G&A	1,335	64	65	74	76	77	76	75	75	75	75	74	74	73	73	73	72	71	50	43
Concentrate Transportation	2,879	79	92	159	174	175	175	175	175	175	171	175	163	175	175	175	175	175	69	46
Total OPEX	**9,664**	**342**	**418**	**558**	**569**	**585**	**589**	**576**	**573**	**570**	**561**	**549**	**544**	**533**	**533**	**529**	**526**	**513**	**334**	**262**
Cost per Process Tonnes																				
Open Pit Mining	16.5	32.1	27.5	16.9	16.2	19.0	21.0	18.6	20.3	19.7	17.7	15.3	15.6	12.6	12.5	11.9	11.5	10.9	6.3	8.5
Underground Mining	33.8	27.2	41.3	35.5	32.6	32.9	31.9	31.8	33.0	30.6	30.4	31.7	30.8	31.0	31.5	30.7	32.5	34.0	71.1	78.3
Processing	14.3	16.5	13.4	13.3	13.7	13.7	13.7	13.8	14.4	14.0	13.8	14.1	13.7	13.6	13.7	13.5	13.9	14.2	23.4	31.7
G&A	15.8	24.5	20.6	15.1	15.0	15.1	14.9	14.8	15.7	15.0	14.6	15.0	14.5	14.6	14.7	14.5	15.0	15.7	23.6	29.8
Concentrate Transportation	34.2	30.0	28.8	32.1	34.1	34.6	34.5	34.5	36.7	35.2	33.5	35.6	32.0	35.1	35.4	35.0	36.4	38.8	32.5	31.2
Total OPEX	**114.6**	130.3	131.6	112.9	111.6	115.3	115.9	113.5	120.0	114.5	110.0	111.7	106.6	106.8	107.8	105.6	109.4	113.6	157.0	179.6
Cost per Concentrate Dry-t																				
Open Pit Mining	104.9	230.9	205.7	115.3	104.0	120.4	132.9	117.7	120.8	122.3	115.8	93.6	106.2	78.7	77.3	74.5	69.1	61.4	42.7	59.7
Underground Mining	215.2	195.7	309.3	241.6	209.1	207.9	202.2	201.3	196.5	189.9	198.3	194.7	210.4	192.7	194.0	192.2	195.3	191.5	478.6	548.0
Processing	91.2	119.1	100.8	90.4	87.6	86.9	86.7	87.4	85.6	86.7	89.9	86.4	93.4	84.4	84.3	84.7	83.4	79.9	157.7	221.7
G&A	100.7	176.8	154.5	102.5	95.9	95.7	94.2	93.7	93.4	93.4	95.7	92.0	98.6	90.6	90.7	90.6	90.0	88.6	159.0	208.3
Concentrate Transportation	217.2	215.9	215.9	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6	218.6
Total OPEX	**729.1**	938.5	986.2	768.3	715.1	729.5	734.6	718.7	714.9	710.9	718.2	685.3	727.1	665.0	664.9	660.5	656.3	640.0	1,056.6	1,256.3

22.11 Taxation

The Project is subject to three (3) levels of taxation: federal corporate income tax, provincial corporate income tax, and provincial mining taxes. The taxation model for the Project was prepared by third-party taxation specialists; however, this information has not been independently verified by GMS.

The Canadian tax system currently applicable to Mineral Resource income was used to assess the annual tax liabilities for the Project. This includes:

- Federal corporate income taxes (15.0% corporate taxable income).
- Provincial corporate income taxes (11.0% corporate taxable income).
- Provincial mining taxes (Marginal tax rates under Québec's Mining Tax Act are 16%, 22% and 28% of taxable income, depending on the profit margin).

In addition to the taxes described above, the tax model also includes the following elements:

- The Clean Technology Manufacturing Investment Tax Credit (CTM-ITC), enacted on June 20, 2024, provides a refundable tax credit of up to 30% of the cost of qualifying investments. The CTM-ITC has been applied in the tax model; however, eligibility throughout the Project execution period cannot be guaranteed.
- Carbon emission costs are included in the model and deducted in the tax calculations.
- Tax Credit Relating to Resources (TCRR) provides a refundable tax credit of up to 45% of the cost of eligible exploration and development expenses for "critical" and "strategic" minerals. The TCRR has been applied in the tax model; however, eligibility throughout the Project execution period cannot be guaranteed.

Considering the combined impact of these taxation levels and credits, the effective tax rate for the Project is estimated at 35.2%. Based on these assumptions, the Project is expected to pay approximately CA$2,940.5M in taxes over the Life of Mine (LOM), as summarized in Table 22.11.

Table 22.11: Total Amount of Tax

Tax	$M
Federal and Provincial Taxes	1,787.3
Québec Mining Tax	1,553.9
Québec Minimum Mining Taxes Paid in Prior Periods (Non-refundable Credit)	(51.8)
Total Income Tax	3,289.4
Clean Technology Manufacturing Investment Tax Credit	(359.8)
Tax Credit Relating to Resources	(57.1)
Total Cash Income Tax - Net of Incentives	2,872.5
GHG Tax	68.0
Total Taxes	2,940.5

22.12 Economic Results

The main economic metrics used to evaluate the Project consist of the net undiscounted after-tax cash flow, the net discounted after-tax cash flow or NPV, IRR, and payback period. An 8% discount rate is commonly used as the base. A summary of the Project economic results is presented in Table 22.12 and the annual Project cash flows are presented in Table 22.13.

The total pre-tax undiscounted cash flow over the Project life is $8,358M, and the pre-tax NPV 8% is $2,514M. The pre-tax Project cash flow results in a 4.9-year payback period from the commencement of commercial operations with a pre-tax IRR of 19.9%.

The total after-tax undiscounted cash flow over the Project life is $5,418M, and the after-tax NPV 8% is $1,594M. The after-tax Project cash flow results in a 4.7-year payback period from the commencement of commercial operations with an IRR of 18.1% after tax. Figure 22.9 presents the Life of Mine After-tax Cumulative Cash Flow Projection.

Table 22.12: Project Economic Results Summary

Parameters	Unit	Value
Physicals		
Phase 1 OP Construction and Ramp Up Phase	year	3.4
Phase 2 UG Expansion construction and Ramp up Phase	year	3.6
Life of Mine (LOM)	year	19
Open Pit		
Ore Mined	Mt	49.2
Waste Mined (Including Pre-strip)	Mt	167.5
Total Tonnes Mined	Mt	216.7
LOM Open Pit Strip Ratio (Waste Tonnes: Resource Tonnes)	w:o	3.4:1
Underground		
Ore Mined	Mt	35.1
Waste Mined	Mt	5.2
Total Tonnes Mined	Mt	40.3
Total		
Total Resource (Open Pit + Underground) Mined and Processed	Mt	84.3
Nominal Process Plant Feed Rate	Mtpa	5.1
Average DMS Process Plant Feed Rate (LOM)	Mtpa	4.4
Average Li_2O Recovery	%	68.9
Average Feed Grade	%	1.26
Spodumene Concentrate	Mt	13.3
Spodumene Concentrate Grade	%	5.5
Nominal Spodumene Concentrate Production Rate	ktpa	801.6
Average Annual Production Rate	ktpa	693.8
Exchange Rate		
Exchange	CAD/USD	1.34
Concentrate Grade		
Li_2O Grade	%	5.50
Commodity Prices		
Li_2O Grade 5.5%	USD/t	1,221

Parameters	Unit	Value
Recovery		
Plant Recovery	%	$= \frac{75 * (1 - e^{-1.995 * Li2O\ feed\ Grade})}{100}$
Operating Costs		
Open Pit Mining	$/t conc	104.9
Underground Mining	$/t conc	215.2
Processing	$/t conc	91.2
G&A	$/t conc	100.7
Cash Operating Cost at Site	**$/t conc**	**511.9**
Concentrate Transport	$/t conc	217.2
Total Cash Operating Cost (FOB Grande-Anse)	**$/t conc**	**729.1**
Sustaining Capital	$/t conc	70.6
All-in Sustaining Cost	**$/t conc**	**799.8**
Utilities / Energy Cost		
Power (Site and Mills)	$/kWh	0.053
Diesel Fuel	$/L	1.43
Salvage Value		
Salvage Value Factor	%	0.00
Royalty on Concentrate		
Royalty	%	2.00
Transport Losses		
Transport Losses (Mat. Movement)	%	0.20
Capital Costs		
Phase 1 OP Initial Capital Cost	$M	1,349.40
Phase 2 UG Capital Cost	$M	628.8
Sustaining CAPEX - OP	$M	531.9
Sustaining CAPEX - UG	$M	404.6
Salvage Value	$M	0
Closure & Rehabilitation Cost	$M	313.0
Total Capital Costs	$M	3,227.6

Parameters	Unit	Value
Financial Evaluation		
Free Cash flow FCF	$M	5,418
After-Tax NPV 8%	$M	1,594
After-Tax IRR	%	18.1
Payback Years	years	4.7

Table 22.13: Project Cash Flow Summary

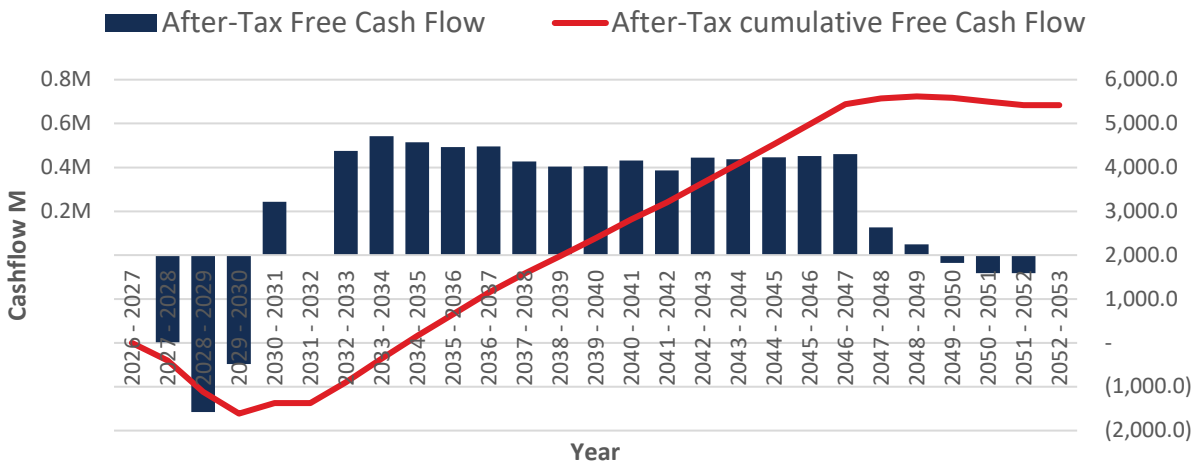
Description		Total	2026 2027	2027 2028	2028 2029	2029 2030	2030 2031	2031 2032	2032 2033	2033 2034	2034 2035	2035 2036	2036 2037	2037 2038	2038 2039	2039 2040	2040 2041	2041 2042	2042 2043	2043 2044	2044 2045	2045 2046	2046 2047	2047 2048	2048 2049	2049 2050	2050 2051	2051 2052
Spodumene Price	CA$	1,221	-	-	-	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	1,221	-	-	-
Total Ore Process	Kt	84,297	-	-	-	495	2,623	3,175	4,943	5,100	5,069	5,082	5,076	4,776	4,977	5,100	4,919	5,100	4,989	4,946	5,012	4,810	4,515	2,128	1,461	-	-	-
Ore Grade	%Li_2O	1.26	-	-	-	1.19	1.14	1.10	1.19	1.25	1.26	1.26	1.26	1.33	1.28	1.23	1.29	1.19	1.28	1.29	1.27	1.32	1.39	1.20	1.16	-	-	-
Contained Li2O	kt	1,059	-	-	-	6	30	35	59	64	64	64	64	63	64	63	64	60	64	64	64	63	63	26	17	-	-	-
Spodumene Concentrate - Dry	kt-Conc Dry	13,255	-	-	-	73	364	424	726	796	802	802	802	802	802	781	802	748	802	802	802	802	802	316	209	-	-	-
Spodumene Concentrate - Wet	kt-Conc Wet	13,917	-	-	-	77	382	445	763	836	842	842	842	842	842	820	842	785	842	842	842	842	842	332	219	-	-	-
Gross Revenue	CA$ M	21,686	-	-	-	120	596	693	1,189	1,302	1,312	1,312	1,312	1,312	1,312	1,278	1,312	1,223	1,312	1,312	1,312	1,312	1,312	517	342	-	-	-
Transport Cost	CA$ M	(2,895)	-	-	-	(16)	(79)	(92)	(159)	(174)	(175)	(175)	(175)	(175)	(175)	(171)	(175)	(163)	(175)	(175)	(175)	(175)	(175)	(69)	(46)	-	-	-
Concentrate Transport Losses	CA$ M	(46)	-	-	-	(0)	(1)	(1)	(2)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(1)	(1)	-	-	-
Royalties	CA$ M	(375)	-	-	-	(2)	(10)	(12)	(21)	(23)	(23)	(23)	(23)	(23)	(23)	(22)	(23)	(21)	(23)	(23)	(23)	(23)	(23)	(9)	(6)	-	-	-
Total NSR	CA$ M	18,371	-	-	-	102	506	588	1,007	1,103	1,111	1,111	1,111	1,111	1,111	1,082	1,111	1,036	1,111	1,111	1,111	1,111	1,111	438	289	-	-	-
Mining OP	CA$ M	(1,390)	-	-	-	-	(84)	(87)	(84)	(83)	(97)	(107)	(94)	(97)	(98)	(90)	(75)	(79)	(63)	(62)	(60)	(55)	(49)	(14)	(12)	-	-	-
Mining UG	CA$ M	(2,852)	-	-	-	-	(71)	(131)	(175)	(166)	(167)	(162)	(161)	(158)	(152)	(155)	(156)	(157)	(154)	(156)	(154)	(157)	(153)	(151)	(114)	-	-	-
Processing	CA$ M	(1,208)	-	-	-	-	(43)	(43)	(66)	(70)	(70)	(70)	(70)	(69)	(70)	(70)	(69)	(70)	(68)	(68)	(68)	(67)	(64)	(50)	(46)	-	-	-
General & Administration	CA$ M	(1,335)	-	-	-	-	(64)	(65)	(74)	(76)	(77)	(76)	(75)	(75)	(75)	(75)	(74)	(74)	(73)	(73)	(73)	(72)	(71)	(50)	(43)	-	-	-
Total OPEX	CA$ M	(6,785)	-	-	-	-	(263)	(326)	(399)	(395)	(410)	(414)	(401)	(398)	(395)	(390)	(374)	(380)	(358)	(358)	(354)	(351)	(338)	(265)	(217)	-	-	-
EBITDA	CA$ M	11,586	-	-	-	102	243	262	607	708	701	697	710	713	716	692	737	656	753	753	757	760	773	173	73	-	-	-
Phase 1 OP Initial Capital Cost	CA$ M	(1,349)	(5)	(277)	(653)	(414)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Phase 2 UG Expansion Capital Cost	CA$ M	(629)	-	(0)	(30)	(118)	(310)	(171)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX - OP	CA$ M	(532)	-	-	-	-	(114)	(60)	(57)	(30)	(35)	(28)	(18)	(28)	(47)	(25)	(17)	(18)	(11)	(18)	(13)	(3)	(6)	(1)	(1)	-	-	-
Sustaining CAPEX - UG	CA$ M	(405)	-	-	-	-	(87)	(41)	(27)	(32)	(18)	(15)	(11)	(12)	(21)	(19)	(23)	(16)	(17)	(18)	(13)	(14)	(8)	(6)	(7)	-	-	-
Salvage Value	CA$ M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure & Rehabilitation Cost	CA$ M	(313)	-	(124)	(62)	(62)	242	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(83)	(83)	(83)
Total Capital Costs	CA$ M	(3,228)	(5)	(401)	(746)	(594)	(269)	(275)	(87)	(65)	(56)	(46)	(33)	(43)	(71)	(47)	(43)	(37)	(31)	(40)	(29)	(20)	(17)	(11)	(11)	(83)	(83)	(83)

Description		Total	2026 2027	2027 2028	2028 2029	2029 2030	2030 2031	2031 2032	2032 2033	2033 2034	2034 2035	2035 2036	2036 2037	2037 2038	2038 2039	2039 2040	2040 2041	2041 2042	2042 2043	2043 2044	2044 2045	2045 2046	2046 2047	2047 2048	2048 2049	2049 2050	2050 2051	2051 2052
Change in Working Capital	CA$ M	0	-	-	-	(8)	(29)	(20)	(13)	(3)	(1)	(0)	1	0	0	1	0	1	(0)	0	0	0	1	17	5	47	(0)	-
Pre-Tax Cash Flow	CA$ M	8,358	(5)	(401)	(746)	(500)	(56)	(33)	507	640	644	651	678	670	645	646	695	619	721	714	728	740	757	180	67	(35)	(83)	(83)
Federal and Provincial Taxes	CA$ M	(1,787)	-	-	-	-	-	-	(2)	(62)	(65)	(66)	(79)	(135)	(136)	(135)	(147)	(131)	(154)	(153)	(155)	(158)	(161)	(35)	(13)	-	-	-
Québec Mining Tax	CA$ M	(1,554)	-	-	-	(0)	(9)	(12)	(30)	(65)	(84)	(89)	(100)	(104)	(102)	(103)	(113)	(99)	(121)	(120)	(123)	(127)	(132)	(17)	(5)	-	-	-
Québec Minimum Mining Taxes Paid in Prior Periods (Non-refundable Credit)	CA$ M	52	-	-	-	-	-	-	-	30	22	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Clean Technology Manufacturing Investment Tax Credit	CA$ M	360	-	-	-	-	311	44	3	1	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tax Credit Relating to Resources	CA$ M	57	2	4	32	5	-	1	1	2	1	1	1	0	1	1	1	1	1	1	1	1	0	0	0	-	-	-
GHG Tax	CA$ M	(68)	-	(0)	(1)	(2)	(3)	(3)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(3)	(3)	(3)	(3)	(3)	(1)	(1)	-	-	-
After-Tax Free Cash Flow	CA$ M	5,418	(3)	(398)	(715)	(497)	243	(4)	476	542	515	492	496	428	404	405	432	387	444	438	447	452	461	127	49	(35)	(83)	(83)

Notes:

1. *Periods are years from July to June*
2. *2026-2027 period is six (6) months only*

Figure 22.9: Life of Mine After-Tax Cumulative Cash Flow Projection



22.13 Sensitivity Analysis

The Project's financial performance is most sensitive to the spodumene price, exchange rate, mine grade and process recovery and much less to the operating costs and capital expenditures.

The results of the sensitivity analysis of the Project in terms of NPV, IRR and Payback are summarized in Table 22.14 to Table 22.19. Figure 22.10 to Figure 22.13 illustrate the sensitivity of the Project.

Table 22.14: OPEX Sensitivity

OPEX	Pre-Tax				After-Tax			
	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)
-20%	9,715	3,107	22%	4.4	6,191	1,948	20%	4.3
-10%	9,037	2,811	21%	4.7	5,806	1,772	19%	4.5
Base Case	8,358	2,514	20%	4.9	5,418	1,594	18%	4.7
+10%	7,680	2,217	19%	5.2	5,024	1,413	17%	4.9
+20%	7,001	1,920	17%	5.5	4,622	1,228	16%	5.3

Table 22.15: CAPEX Sensitivity

CAPEX	Pre-Tax				After-Tax			
	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)
-20%	8,658	2,788	23%	4.5	5,717	1,868	21%	4.1
-10%	8,508	2,651	21%	4.7	5,567	1,731	20%	4.4
Base Case	8,358	2,514	20%	4.9	5,418	1,594	18%	4.7
+10%	8,208	2,377	19%	5.2	5,268	1,457	17%	5.0
+20%	8,059	2,240	18%	5.4	5,118	1,320	15%	5.3

Table 22.16: Spodumene Price Sensitivity

Spodumene Price	Pre-Tax				After-Tax			
	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)
-20%	4,117	693	12%	7.4	2,764	399	11%	6.9
-10%	6,237	1,604	16%	5.9	4,142	1,023	15%	5.5
Base Case	8,358	2,514	20%	4.9	5,418	1,594	18%	4.7
+10%	10,479	3,424	23%	4.3	6,658	2,144	21%	4.1
+20%	12,600	4,334	27%	3.8	7,884	2,686	24%	3.7

Table 22.17: Lithium Grade Price Sensitivity

Grade	Pre-Tax				After-Tax			
	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)	FCF (CAD M)	NPV 8% (CAD M))	IRR	Payback Period (y)
-20%	3,836	568	11%	7.7	2,578	310	10%	7.2
-10%	6,104	1,543	16%	6.0	4,055	982	15%	5.6
Base Case	8,358	2,514	20%	4.9	5,418	1,594	18%	4.7
+10%	10,593	3,476	24%	4.2	6,732	2,179	21%	4.1
+20%	12,803	4,428	27%	3.8	8,018	2,748	24%	3.7

Table 22.18: Exchange Rate Sensitivity

Exchange Rate	Pre-Tax				After-Tax			
	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)
-20%	4,117	693	12%	7.4	2,753	393	11%	6.9
-10%	6,237	1,604	16%	5.9	4,138	1,020	15%	5.5
Base Case	8,358	2,514	20%	4.9	5,418	1,594	18%	4.7
+10%	10,479	3,424	23%	4.3	6,662	2,146	21%	4.1
+20%	12,600	4,334	27%	3.8	7,891	2,689	24%	3.7

Table 22.19: Process Recovery Price Sensitivity

Recovery	Pre-Tax				After-Tax			
	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)	FCF (CAD M)	NPV 8% (CAD M)	IRR	Payback Period (y)
-20%	4,684	937	13%	6.9	3,139	570	12%	6.5
-10%	6,521	1,725	17%	5.7	4,322	1,104	15%	5.4
Base Case	8,358	2,514	20%	4.9	5,418	1,594	18%	4.7
+10%	10,195	3,302	23%	4.4	6,498	2,073	21%	4.2
+20%	12,032	4,091	26%	3.9	7,570	2,546	23%	3.8

Figure 22.10: Free Cash flow Sensitivity

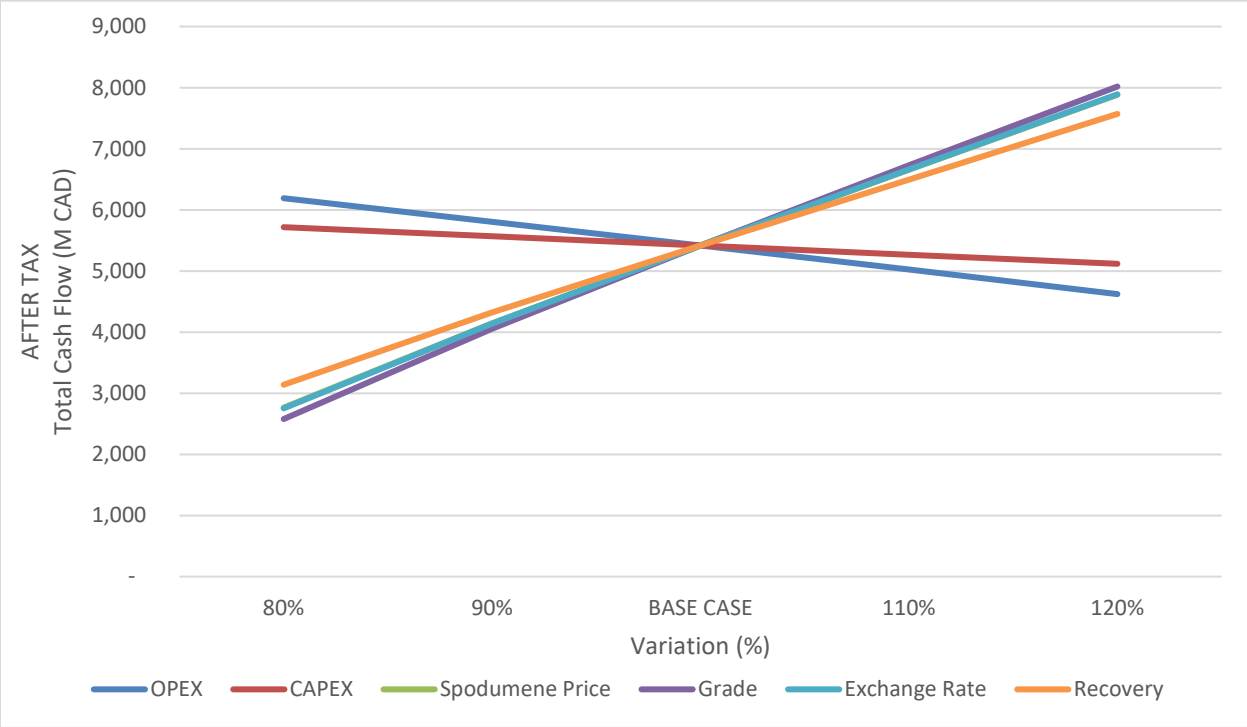


Figure 22.11: NPV 8% Sensitivity

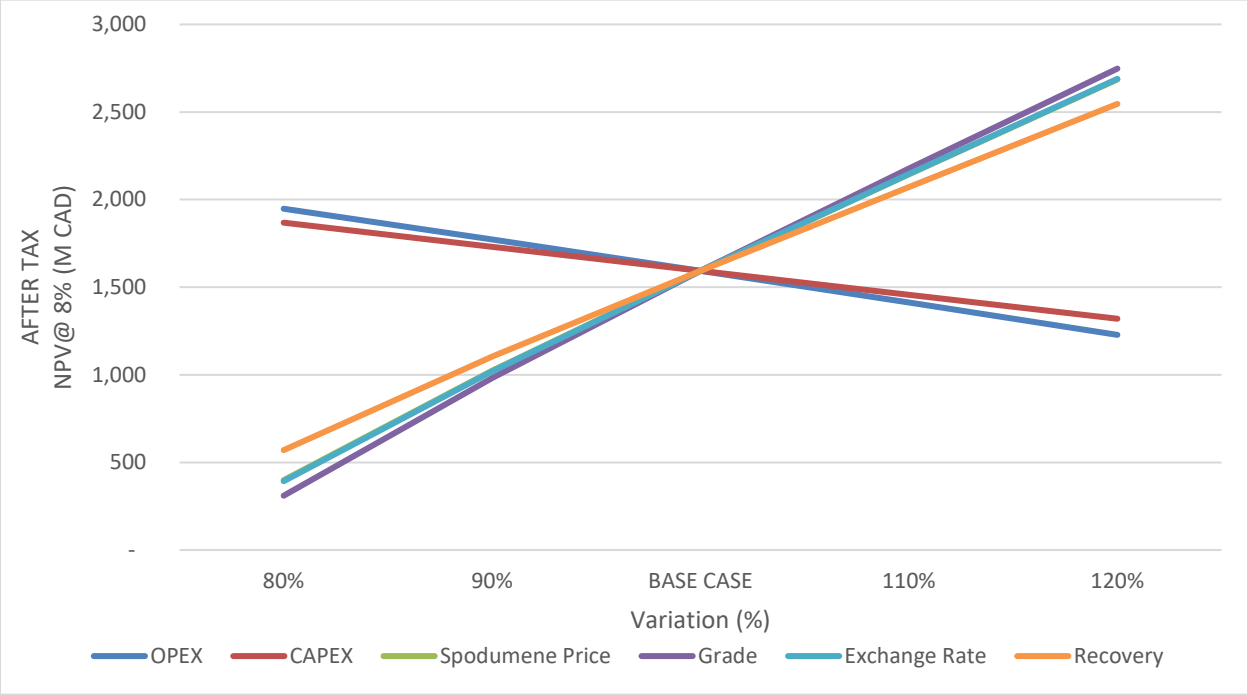


Figure 22.12: IRR Sensitivity

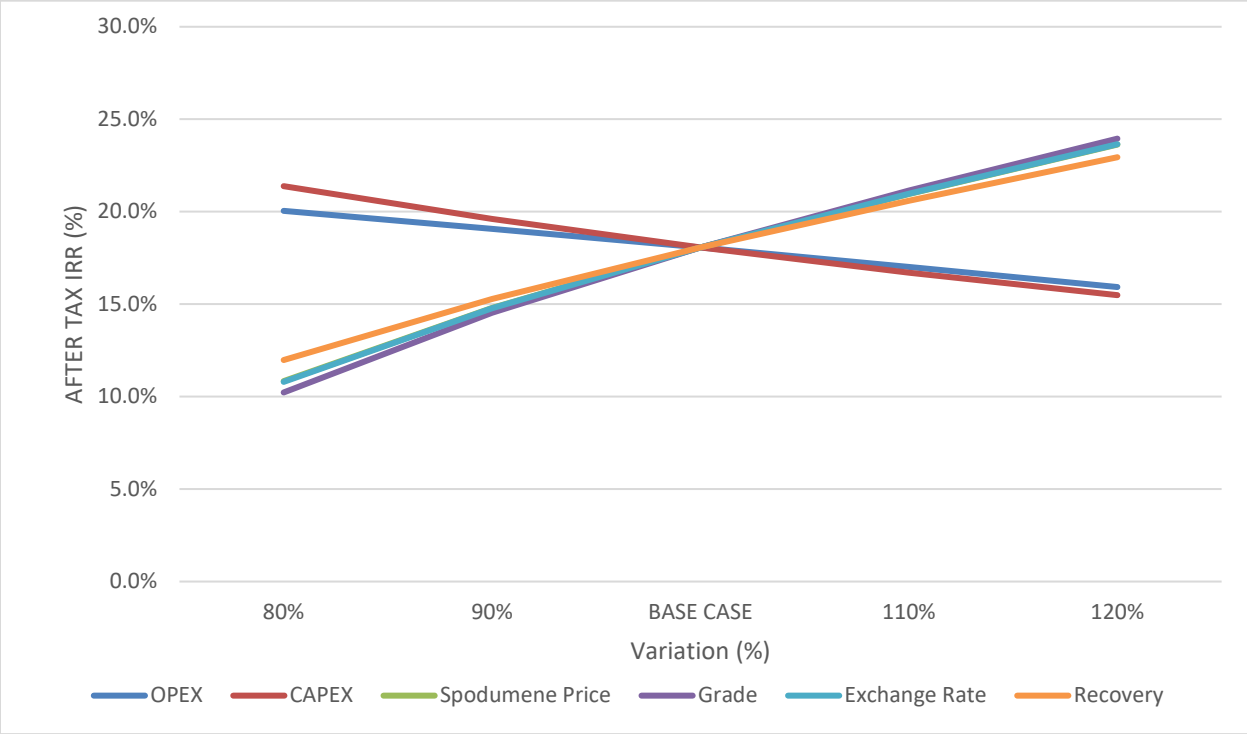
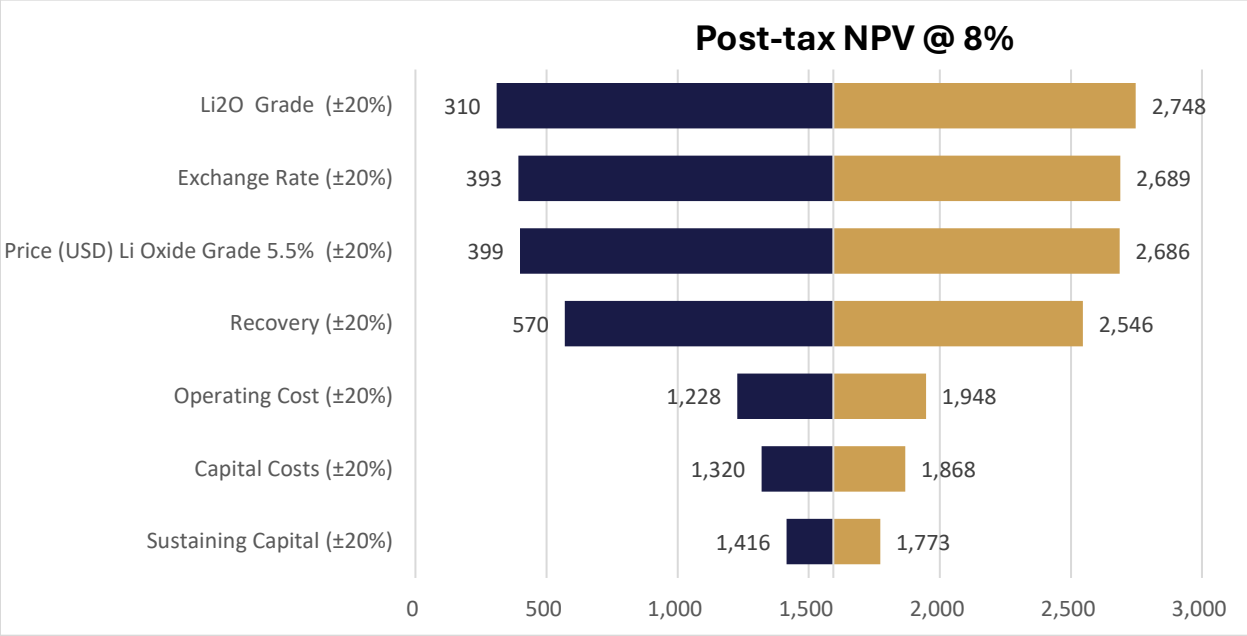


Figure 22.13: NPV 8% Sensitivity



23. ADJACENT PROPERTIES

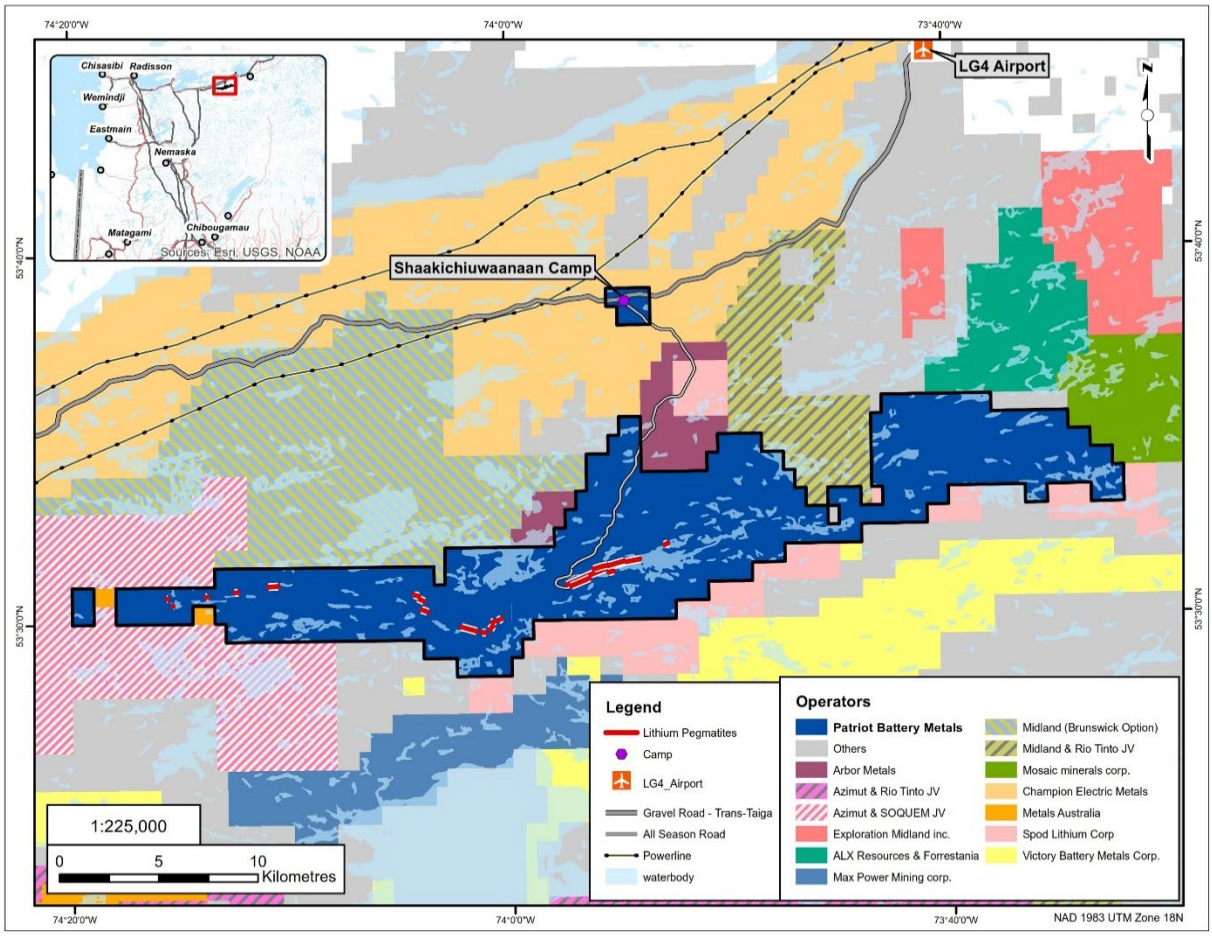
The Shaakichiuwaanaan Property is located in a region of active mineral exploration within the La Grande Greenstone Belt of James Bay. The geological setting is prospective for multiple commodities over several different deposit styles including orogenic gold (Au), volcanogenic massive sulphide (Cu, Au, Ag), komatiite-ultramafic (Au, Ag, PGE, Ni, Cu, Co), and LCT pegmatite (Li, Cs, Ta, Ga, Rb). In addition, a magmatic-hydrothermal Cu-Au-Ag-Mo deposit style (potential Archean porphyry) has been recognized in the immediate region.

The Company holds the dominant land position with respect to greenstone belt in the region; however, the Property is fully surrounded by other properties held by multiple mineral exploration companies. As of August 2025, mineral exploration companies with properties immediately adjacent to the Company's Shaakichiuwaanaan Property are noted below in Figure 23.1.

The closest spodumene pegmatite occurrence to the Property is the Western Prospect, held by Champion Electric Metals Inc. (+2% Li_2O in boulder, 10 m at 0.96% Li_2O in channel, and 3 m at 0.62% Li_2O in drill hole EIQ24-007) located approximately 23 km west of the Property (Champion Electric Metals, 2024a); (Champion Electric Metals Inc., 2024b)

The QP notes that he has not directly verified information related to mineralization on adjacent properties, and that it is not necessarily indicative of the mineralization present on the Shaakichiuwaanaan Property.

Figure 23.1: Adjacent Properties to the Shaakichiuwaanaan Property (as of August 2025)



24. OTHER RELEVANT DATA AND INFORMATION

This chapter presents other relevant data and information that are pertinent to the understanding and evaluation of the mineral Project but are not covered in the preceding sections. The information herein aims to provide a comprehensive overview of the Project execution and ensure that all relevant aspects are considered in the assessment of the Project.

24.1 Project Execution Plan

An integrated Project Management Team (IPMT) has been created to lead the execution of the Shaakichiuwaanaan Project using a self-perform approach, composed of qualified and experienced persons. The Project team will be supplemented by contractors working within the IPMT for both specialized needs and peak workforce requirements. The plan is for the IPMT to lead the Project execution and construction of all on-site infrastructure and the process plant. Mine development will also be self-performed by the owner's Project team.

The Project team will work in unison to achieve Project objectives through the effective use of the Owners' equipment, material and personnel, and to minimize difficulties common to commissioning and start-up. The IPMT will share the responsibility for the planning and execution of the Project.

The Project team will use a quality assurance / quality control (QA/QC) system in all phases of the Project (engineering, procurement, construction, commissioning and start-up). The system will include audits, certification, factory inspection and non-destructive testing during construction, as appropriate. It will also provide traceability from origin for each component of the Project. The detailed procedures and practices will be developed with the Project QA/QC team, integrating the Project document control process.

The Mine and Process operations team will be recruited during construction and will work within the IPMT during construction, pre-commissioning, commissioning, start-up, and handover of the facilities. Handover will be a structured and planned process that will require strong coordination between the Project personnel and operations team for a gradual transfer of responsibility.

All departments included in General Services will be staffed with the Owner's employees to service the construction efforts. These departments will become fully operational early in the construction period, with a trained workforce and established programs and service providers. As construction is demobilized, the service departments will seamlessly continue into operations

The Owner mining team will consist of operations, maintenance and technical services and will be a critical component of the overall Project team as it will execute a great deal of the mass earth movement. The Project team will be headed by a Project director and a general manager who will provide overall Project management during construction. The Owner's operations team will take full control of the operation after commercial production is achieved. Commercial production is defined as the first day of a period of 30 days when the ore tonnage processed averages at least 65% of the nameplate

24.2 Project Execution Schedule

Schedule development begins at the management level and drills down through the Project and control levels. The management level schedule is used to establish work goals and overall time frames for the scope of work. Project milestones are detailed in Figure 24.1. This Level 1 schedule contains the least amount of detail but provides a high-level tool for management to evaluate and track the main Project milestones. The Project schedule defines the detailed tasks and the duration of each task. The schedule for the Shaakichiuwaanaan Lithium Project spans from January 2027 to May 2030 for Phase 1, related to the open pit mine and covering over three (3) years of multidisciplinary Project development. It includes a comprehensive permitting phase, starting with early works authorizations and progressing to environmental, and construction permits through governmental agencies. Concurrently, activities and early site infrastructure (roads, camp facilities, and utilities) are launched in early 2028, after permits are granted. Major infrastructure components such as the process plant, power stations, water and sewage systems are gradually implemented. Construction peaks in summer 2029, followed by a commissioning phase and ramp-up starting in Q4 2029 to reach commercial production in May 2030.

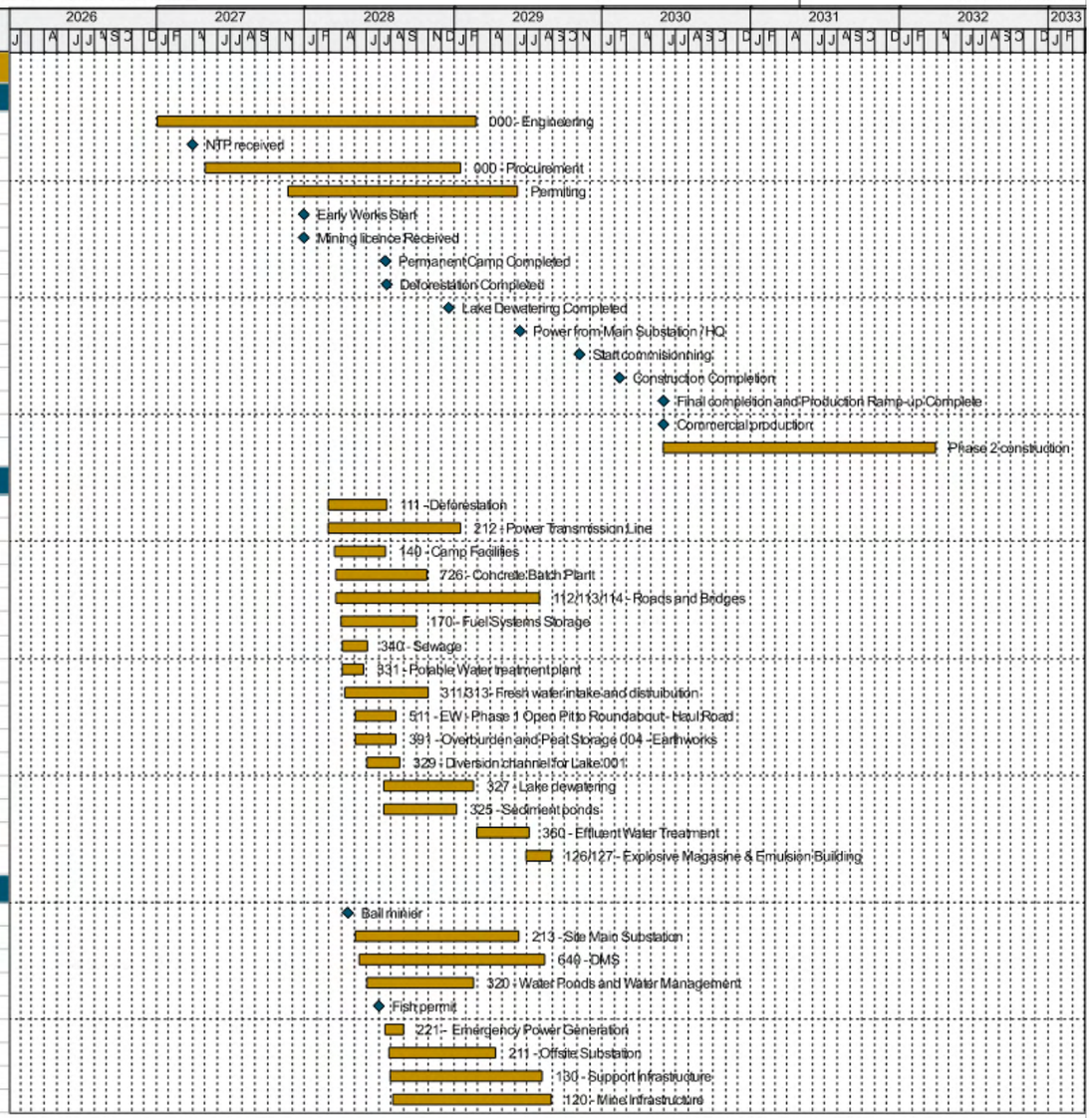
Phase 2 construction activities begin in parallel for the underground mine development, starting in Q3 2028. Process plant expansion is planned from June 2030 to March 2032. The schedule demonstrates structured execution per work packages, including procurement, mobilization of heavy mining equipment, installation of key utilities, and construction of permanent buildings such as assay lab, administrative offices, and truck shop. Mining equipment delivery and assembly are aligned with the completion of haul roads and mechanical workshops to support the progressive development of the open pit and underground operations. The final phase of the schedule focuses on commissioning and ramp-up. Commissioning is slated to begin in Q3 2029, followed by a full ramp-up in Q1 and Q2 2030, ensuring operational readiness and performance validation of all systems.

Figure 24.1: Project Schedule – Level 1

CASN - Shaakichuwaanaan - Construction Schedule - Feasibility
DRAFT SCHEDULE

Printed date: 29-Sep-25 10:49
Project ID: CASN-1
Data date:01-Jan-26

#	Activity ID	Activity Name	Original Duration	Start	Finish
1	**CASN - Shaakichiuwaanaa - Construction -In Progres**		1351	04-Jan-27	31-Mar-32
2	**General**		1351	04-Jan-27	31-Mar-32
3	C-000-LOE-1020	000 - Engineering	785	04-Jan-27	26-Feb-29
4	MM-1290	NTP received	0	01-Apr-27*	
5	C-001-LOE-1020	000 - Procurement	627	01-May-27	16-Jan-29
6	PM-LOE-1000	Permiting	395	22-Nov-27	08-Jun-29
7	MM-EW-1010	Early Works Start	0	31-Dec-27	
8	MM-1030	Mining licence Received	0	31-Dec-27*	
9	MM-EW-1050	Permanent Camp Completed	0		18-Jul-28
10	MM-EW-1060	Deforestation Completed	0		20-Jul-28
11	MM-EW-1080	Lake Dewatering Completed	0		19-Dec-28
12	MM-2010	Power from Main Substation / HQ	0	12-Jun-29	
13	MM-1330	Start commisionning	0	08-Nov-29	
14	MM-2020	Construction Completion	0		12-Feb-30
15	MM-1360	Final completion and Production Ramp-up Complete	0		31-May-30
16	MM-2000	Commercial production	0	01-Jun-30*	
17	C-000-2000	Phase 2 construction	670	01-Jun-30*	31-Mar-32
18	**Early works**		548	29-Feb-28	29-Aug-29
19	C-111-LOE-1020	111 - Deforestation	143	29-Feb-28	20-Jul-28
20	C-212-LOE-1010	212 - Power Transmission Line	323	29-Feb-28	16-Jan-29
21	C-140-LOE-1000	140 - Camp Facilities	126	15-Mar-28	18-Jul-28
22	C-726-LOE-1010	726 - Concrete Batch Plant	223	20-Mar-28	28-Oct-28
23	C-112-LOE-1020	112/113/114 - Roads and Bridges	500	20-Mar-28	01-Aug-29
24	C-170-LOE-1010	170 - Fuel Systems Storage	185	30-Mar-28	30-Sep-28
25	C-340-LOE-1010	340 - Sewage	61	03-Apr-28	02-Jun-28
26	C-331-LOE-1010	331 - Potable Water treatment plant	50	04-Apr-28	23-May-28
27	C-310-LOE-1010	311/313- Fresh water intake and distruibution	205	09-Apr-28	30-Oct-28
28	C-511-EW-1000	511 - EW - Phase 1 Open Pit to Roundabout - Haul Road	100	04-May-28	11-Aug-28
29	C-391-EW-1010	391 - Overburden and Peat Storage 004 - Earthworks	100	04-May-28	11-Aug-28
30	C-329-LOE-1010	329 - Diversion channel for Lake 001	80	03-Jun-28	21-Aug-28
31	C-327-LOE-1010	327 - Lake dewatering	220	13-Jul-28	17-Feb-29
32	C-325-LOE-1010	325 - Sediment ponds	180	13-Jul-28	08-Jan-29
33	C-360-LOE-1010	360 - Effluent Water Treatment	125	01-Mar-29	04-Jul-29
34	C-126-LOE-1000	126/127 - Explosive Magasine & Emulsion Building	63	28-Jun-29	29-Aug-29
35	**Construction**		555	17-Apr-28	31-May-30
36	PM-MI-1010	Bail minier	0	17-Apr-28	
37	C-213-LOE-1010	213 - Site Main Substation	403	05-May-28	11-Jun-29
38	C-640-LOE-1010	640 - DMS	455	14-May-28	11-Aug-29
39	C-320-LOE-1010	320 - Water Ponds and Water Management	260	03-Jun-28	17-Feb-29
40	PM-001-1180	Fish permit	0	03-Jul-28*	
41	C-221-LOE-1010	221 - Emergency Power Generation	45	19-Jul-28	01-Sep-28
42	C-211-LOE-1010	211 - Offsite Substation	260	27-Jul-28	12-Apr-29
43	C-130-LOE-1010	130 - Support Infrastructure	374	29-Jul-28	06-Aug-29
44	C-120-LOE-1010	120 - Mine Infrastructure	388	07-Aug-28	29-Aug-29



Level 2
Layout: CASN-1 A-Level 2
Filter: TASK filter: Level 2 .
Page 1 of 2

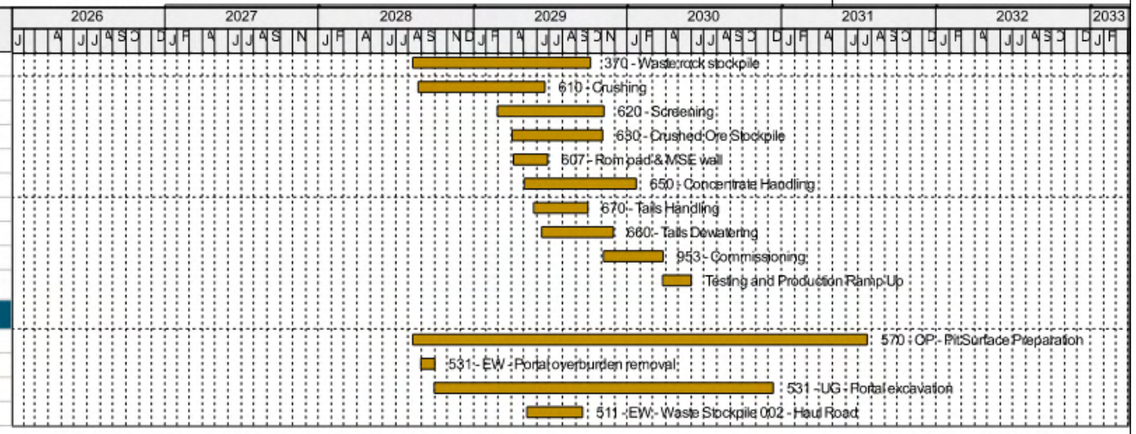
Mining Services

Date	Revision	Checked	Approv...
29-Sep-25	For review	CJ	

CASN - Shaakichuwaanaan - Construction Schedule - Feasibility
DRAFT SCHEDULE

Printed date: 29-Sep-25 10:49
Project ID: CASN-1
Data date:01-Jan-26

#	Activity ID	Activity Name	Original Duration	Start	Finish
45	C-370-LOE-1010	370 - Waste rock stockpile	420	13-Aug-28	07-Oct-29
46	C-610-LOE-1010	610 - Crushing	299	25-Aug-28	19-Jun-29
47	C-620-LOE-1010	620 - Screening	253	28-Feb-29	07-Nov-29
48	C-630-LOE-1010	630 - Crushed Ore Stockpile	213	04-Apr-29	02-Nov-29
49	C-607-LOE-1010	607 - Rom pad & MSE wall	80	09-Apr-29	27-Jun-29
50	C-650-LOE-1010	650 - Concentrate Handling	263	04-May-29	21-Jan-30
51	C-670-LOE-1010	670 - Tails Handling	128	24-May-29	28-Sep-29
52	C-660-LOE-1010	660 - Tails Dewatering	168	13-Jun-29	27-Nov-29
53	C-953-LOE-1010	953 - Commissioning	140	08-Nov-29	27-Mar-30
54	C-953-COM-1040	Testing and Production Ramp Up	65	28-Mar-30	31-May-30
55	**Mining**		1074	12-Aug-28	21-Jul-31
56	C-570-UG-1000	570 - OP - Pit Surface Preparation	1074	12-Aug-28	21-Jul-31
57	C-531-EW-1000	531 - EW - Portal overburden removal	30	01-Sep-28*	01-Oct-28
58	C-531-UG-1000	531 - UG - Portal excavation	800	02-Oct-28*	11-Dec-30
59	C-511-EW-1020	511 - EW - Waste Stockpile 002 - Haul Road	130	10-May-29	17-Sep-29



Level 2
Layout: CASN-1 A-Level 2
Filter: TASK filter: Level 2 .
Page 2 of 2

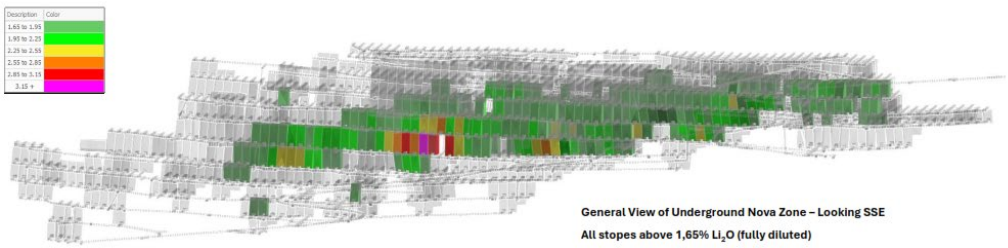
Date	Revision	Checked	Approv...
29-Sep-25	For review	CJ	

The critical path for this schedule involves the environmental technical studies, the provincial impact assessment process, and the process facility construction. The period for the provincial impact assessment process is estimated to be completed in Q4 2027.

24.3 High-Grade Nova Zone

The zone contains very high-grade stopes – with ten (10) stopes averaging >2.8% Li_2O and two (2) stopes >3.0% Li_2O – and is characterized by very large spodumene crystals, which makes the material easily recovered by DMS techniques at relatively large crush size. By developing the open pit mine and the underground Nova Zone concurrently, a blended high feed grade can be realized. The first years of underground mining aim to reduce capital costs by delaying lateral development and mining the Western zones first, ensuring that no ore is sterilized at the open pit and underground interface. Once the first mining blocks are well developed to sustain production, the ramp heading will continue towards the Nova Zone. The high-grade ore included in this area will be blended with lower-grade ore from the open pit, allowing more mineralization recovery and an enhancement of the global Project economics.

Figure 24.2: General View of Underground Nova Zone – Looking SSE



Top 10 Nova Zone Stopes

Tonnes	Grade (%Li_2O)
21,905	3.48
17,849	3.01
41,247	2.97
19,984	2.97
17,344	2.91
20,184	2.89
33,786	2.87
19,902	2.84
37,944	2.84
16,931	2.83

24.4 Shaakichiuwaanaan Exploration Target

On August 5, 2024, the Company released an Exploration Target for the Shaakichiuwaanaan Project (PMET Resources, 2024b). The Exploration Target has been presented as an approach to assess the potential endowment of the Project or the potential to host additional Mineral Resources of lithium pegmatite, subject to the success of future mineral exploration at the Property, and outside of that already defined.

The Exploration Target (also known as a “Target for Further Exploration”) for the Shaakichiuwaanaan Project is approximately:

- 146 Mt to 231 Mt at 1.0% to 1.5% Li_2O.

The potential quantity and grade of the Exploration Target are conceptual in nature. There has been insufficient exploration to define a Mineral Resource, and it is uncertain if further exploration will result in the target being delineated as a Mineral Resource. The Exploration Target has been determined based on the interpretation of a consolidated dataset of surface rock sample descriptions and assays, outcrop mapping and descriptions, drill hole logs and core sample assays, geophysical surveys, and remote sensing data (refer to “Methodology of Determination for the Exploration Target” below for more information about the base on which the disclosed potential quantity and grade has been determined).

The Exploration Target is in addition to (i.e., does not include) the current 2025 MRE and only considers the CV Lithium Trend and areas immediately proximal to the Shaakichiuwaanaan Property (Figure 7.5).

The Company intends to test the validity of the Exploration Target in future exploration programs at the Project, extending over several years. Systematic diamond drilling (NQ core size) of the known spodumene pegmatite clusters and corridors between and proximal to these clusters, which collectively form the basis of the Exploration Target, will be the primary method of exploration.

The Exploration Target was completed by BBA Engineering Ltd., a consultant independent of the Company, and reported in accordance with NI 43-101 and Clause 17 of the JORC Code on August 5, 2024.

25. INTERPRETATION AND CONCLUSIONS

25.1 Summary

This Feasibility Study (FS) Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) and Form 43-101F1. The objective of this FS Report is the evaluation of the technical feasibility and economic viability of the Project (lithium only for the CV5 Pegmatite), including the development of an open pit and underground mine with processing facilities and required infrastructure and lake dewatering.

The conclusion of the qualified persons (QPs) is that the Shaakichiuwaanaan Project, as summarized in this NI 43-101 Feasibility Study (FS) report, contains a sufficient level of detail and information to support the positive economic results presented, and confirms the technical feasibility and economic viability based on a combined open pit and underground mining operation. Table 25.1 shows the Feasibility Study highlights. The total life-of-mine (LOM) is 19 years. At peak capacity for 15 years, the Project will produce an average of 801,600 sold tonnes of spodumene concentrate per year, grading 5.5% Li_2O. With an additional two (2) years leading up to and the two (2) years following full capacity, average production is estimated at 400,000 tonnes of dry concentrate grading 5.5% Li_2O.

Based on the summary presented in Table 25.1, it is recommended to advance the Project to the Detailed Engineering, Construction, and Development phase.

Table 25.1: Technical Report Feasibility Study Update Life-of-Mine Results

Technical Report Feasibility Study Update Life-of-Mine Results		
Spodumene Concentrate Price 6.0%	*USD/Tonne (dry)*	1,332
Spodumene Concentrate Price 5.5%	*USD/Tonne (dry)*	1,221
Phase 1 OP Construction and Ramp Up Phase	*year*	3
Phase 2 UG Expansion Construction and Ramp Up Phase	*year*	4
Years of Operations	*year*	19.0
Open Pit		
Reserve Mined	*Mt*	49.2
Waste Mined (including pre-strip)	*Mt*	167.5
Total Tonnes Mined	*Mt*	216.7
LOM Open Pit Strip Ratio (waste tonnes: resource tonnes)	*w:o*	3.4:1

Technical Report Feasibility Study Update Life-of-Mine Results		
Underground		
Reserve Mined	*Mt*	35.1
Waste Mined	*Mt*	5.2
Total Tonnes Mined	*Mt*	40.3
Total Open Pit and Underground		
Total Reserve (Open Pit + Underground) Mined and Processed	*Mt*	84.3
Nominal Process Plant Feed Rate	*Mtpa*	5.1
Average LOM DMS Process Plant Feed	*Mtpa*	4.4
Average Li_2O Recovery	*%*	68.9
Average Feed Grade	*%*	1.26
Spodumene Concentrate	*Mt*	13.3
Spodumene Concentrate Grade	*%*	5.5
Annual Production Rate	*ktpa*	693.8
Operating Costs		
Open Pit Mining Cost	*CAD M*	1,390.4
Underground Mining Cost	*CAD M*	2,852.2
Processing	*CAD M*	1,208.3
General & Administration	*CAD M*	1,334.7
Total OPEX	***CAD M***	**6,787.5**
Operating Costs (LOM Average)		
Open Pit Mining Cost	*$/t mined*	6.42
Underground Mining Cost	*$/t mined*	81.26
Open Pit Mining Cost	*$/t processed*	16.49
Underground Mining Cost	*$/t processed*	33.83
Processing	*$/t processed*	14.33
General & Administration	*$/t processed*	15.83
Total OPEX	*$/t processed*	**80.50**
Capital Costs		
Phase OP Construction and Ramp Up Phase	*CAD M*	1,349.4
Phase 2 UG Expansion Construction and Ramp-up Phase	*CAD M*	628.8
Sustaining CAPEX - OP	*CAD M*	531.9

Technical Report Feasibility Study Update Life-of-Mine Results		
Sustaining CAPEX - UG	*CAD M*	404.6
Salvage Value	*CAD M*	0
Closure & Rehabilitation Cost	*CAD M*	313.0
Total Capital and Closure Costs	*CAD M*	3,227.6
Financial Evaluation		
After-Tax NPV 8%	*CAD M*	1,594.26
After-Tax IRR	*%*	18.1
Payback	*years*	4.7

25.2 Geology and Mineral Resources

The Shaakichiuwaanaan Property is an advanced-stage exploration property located within the La Grande Greenstone Belt in the James Bay region of Quebec. The geological setting is prospective for multiple commodities over several different deposit styles, including orogenic gold (Au), volcanogenic massive sulphide (Cu, Au, Ag), komatiite-ultramafic (Au, Ag, PGE, Ni, Cu, Co), and LCT pegmatite (Li, Cs, Ta, Ga, Rb). The primary exploration focus of the Company has been LCT pegmatite since acquiring the initial claim block of the Property in 2016.

The Shaakichiuwaanaan Property hosts a Consolidated MRE, which includes the CV5 and CV13 LCT pegmatites, of:

- 108.0 Mt at 1.40% Li_2O, 0.11% Cs_2O, 166 ppm Ta_2O_5, and 66 ppm Ga, Indicated, for a contained lithium carbonate equivalent (LCE) of 3.75 Mt.
- 33.4 Mt at 1.33% Li_2O, 0.21% Cs_2O, 155 ppm Ta_2O_5, and 65 ppm Ga, Inferred, for a contained LCE of 1.09 Mt.

The CV5 Pegmatite hosts 101.8 Mt at 1.38% Li_2O, 0.09% Cs_2O, 164 ppm Ta_2O_5, and 66 ppm Ga, Indicated, and 13.9 Mt at 1.21% Li_2O, 0.08% Cs_2O, 147 ppm Ta_2O_5, and 60 ppm Ga, Inferred. The CV13 Pegmatite hosts 6.2 Mt at 1.86% Li_2O, 0.59% Cs_2O, 199 ppm Ta_2O_5, and 76 ppm Ga, Indicated, and 19.5 Mt at 1.41% Li_2O, 0.30% Cs_2O, 161 ppm Ta_2O_5, and 69 ppm Ga, Inferred.

The Consolidated MRE includes the Rigel and Vega caesium zones hosted within the CV13 Pegmatite component. These Mineral Resources include:

- At Rigel: 163,000 t at 10.25% Cs_2O, 1.78% Li_2O, and 646 ppm Ta_2O_5, Indicated.

- At Vega: 530,000 t at 2.61% Cs_2O, 2.23% Li_2O, and 172 ppm Ta_2O_5, Indicated, and 1,698,000 t at 2.40% Cs_2O, 1.81% Li_2O, and 245 ppm Ta_2O_5, Inferred.

The cut-off grade of the Consolidated MRE is variable depending on the mining method and pegmatite (0.40% Li_2O open pit, 0.60% Li_2O underground CV5, and 0.70% Li_2O underground CV13). A grade constraint of 0.50% Cs_2O was used to constrain the Rigel and Vega caesium zones. The Effective Date is June 20, 2025 (through drill hole CV24-787). Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. Mineral Resources are inclusive of Mineral Reserves.

Both the CV5 and CV13 pegmatites remain open along strike at both ends and to depth. Coupled with the Exploration Target for the Shaakichiuwaanaan Property, significant potential is present for additional Mineral Resources to be defined.

25.3 Mineral Reserve Estimates

The Mineral Reserves are based solely on Indicated Mineral Resources. Inferred mineral Resources have not been included in the Mineral Reserve. Any Inferred Resources that fall within the mine design envelope have been treated as waste and assigned a grade of 0% Li_2O for the purposes of mine planning and economic evaluation.

The Probable Reserve for the Shaakichiuwaanaan Project, which includes only the CV5 Pegmatite, is estimated at 84.3 Mt at 1.26% Li_2O for 2.62 Mt contained LCE.

Changes in factors and assumptions may affect the Mineral Reserve Estimate. The Project is subject to various geological considerations, including geological complexity, geological interpretation, and Mineral Resource block modelling, rock quality, geotechnical constraints, dilution and mining recovery factors, hydrogeological assumptions, and metallurgical process recoveries. Economic assumptions such as commodity prices, market conditions and foreign exchange rate assumptions, long-term consumables price assumptions, operating cost assumptions, and sustaining capital costs may also impact the Mineral Reserve Estimate. Finally, the ability to meet and maintain permitting and environmental licence conditions, as well as the ability to maintain the social licence to operate, may also impact the Mineral Reserve Estimate.

There are no other environmental, legal, title, taxation, socioeconomic, refining, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in this Report. It is reasonably expected that all necessary government approvals will be issued for the Project to proceed.

25.4 Mining Method

The Project employs a hybrid approach, integrating both open pit and underground mining operations. The following criteria and limitations on both technical and non-technical parameters were considered in selecting the preferred scenario:

- Minimizing impacts on Lake 001, thereby reducing environmental impacts and fish habitat compensation.
- Maintaining water diversion within the same watershed.
- Economic and financial performance.

The material within the open pit will be extracted using the conventional truck and shovel method. For the underground mine, the long hole mining method, including both transverse and longitudinal techniques, will be utilized.

The open pit is located south of the process plant and is separated into six (6) different phases. The first two (2) phases, phases 1B and 1A, are focused on gathering waste for road construction and a small stockpile of ore to supply feed during the early months of the process plant ramp-up. Phase 2 is constrained by the location of the first dam. Focusing on the West Area of the open pit allows for more rapid ore production while delaying as much as possible the impacts on Lake 001. Mining the East pit will require the construction of a second dam, increasing the impacts on Lake 001.

Table 25.2 summarizes the material mined in each phase of the open pit development.

Table 25.2: Pit Phases Tonnage Summary

Mining Reserves by Phase	Units	West Area			East Area			TOTAL
		Phase 1A	Phase 1B	Phase 2	Phase 3A	Phase 3B	Phase 4	
Waste Tonnage	**kt**							
Overburden	kt	1,313	6,546	5,200	1,135	434	2,854	17,481
Rock	kt	1,718	4,643	32,855	4,360	4,832	101,566	149,975
Ore Tonnage	**kt**							
Ore[2]	kt	1,795	1,181	12,487	2,002	3,603	28,127	49,196
Total Tonnage	kt	4,826	12,371	50,542	7,497	8,834	132,547	216,652
Strip Ratio (W:O)	**W:O**	**1.69**	**9.47**	**3.05**	**2.74**	**1.48**	**3.71**	**3.40**

**Notes on Physical Quantities:*
1) Total may not sum due to rounding; rounding followed the recommendations of the NI 43-101.
2) Open pit marginal material containing grade above 0.37% Li_2O is also included within this statement.

Open pit (OP) mining will include drilling and blasting techniques in combination with large hydraulic shovels and front-end loader-type excavators for loading blasted material into haul trucks. The trucks then haul the material from the mine to the crusher, ore stockpiles, the run-of-mine (ROM) pad or mine waste rock stockpiles, depending on the material type. Support equipment includes dozers, graders, utility loaders, water trucks, and service vehicles. Removal of the overburden and generating material suitable for construction purposes will be the main pre-production activity. The open pit average mining rate is 14.6 Mtpa over the first five (5) years, with a peak mining rate of 23 Mt at year six (6). The Life-of-Mine (LOM) is 19 years, including the pre-production period. A total of 49.2 Mt at 1.12% Li_2O of ore will come from the open pit. A total of 216.7 Mt of material, including ore, waste rock and overburden, will be moved during the LOM, resulting in an average stripping ratio of 3.4. Variations in the quantity of open pit tonnage mined are attributable to multiple factors:

- The maximum production of dry spodumene concentrate is set to 801,600 t per year, based on both open pit and underground tonnage contributions.
- A targeted blended head grade between 1.2% Li_2O and 1.4% Li_2O is maintained to ensure optimal process recovery.

The contribution of the underground mine production to the process plant feed is variable, requiring the open pit schedule to adapt. Process plant throughput at peak capacity is capped at 5.1 Mtpa, considering both open pit and underground tonnage contributions.

The underground (UG) operation comprises a single ramp accessible through a portal located near the run-of-mine (ROM) pad. The selected mining method is long-hole open stoping (LHOS) with cemented

paste backfill (CPB), using both transverse and longitudinal stoping techniques. The LOM for the UG mine is expected to be 21 years, including construction, development, pre-production, and the full production period. Over this LOM, the UG mine is expected to be at full production for 16 years. A two-year pre-production period is planned to allow sufficient underground development to be completed to sustain full production. The UG mine is expected to achieve an average production rate of 5,475 tpd of ore, with 5,200 tpd from stope production and an average of 275 tpd from lateral development. A total of 35.1 Mt of ore is expected to be mined at an average diluted grade of 1.45% Li_2O. The primary production and development equipment fleet includes an 18-t Battery Electric Vehicle (BEV) load-haul-dump (LHD) coupled with 55-t BEV underground mining trucks. Both ore and waste will be rehandled by surface equipment.

The overall life of mine processed ore is 84.3 Mt at a diluted grade of 1.26% Li_2O and considers a mining recovery of 97% for surface operations and 90% for underground operations. Ore from the open pit is 49.2 Mt at a grade of 1.12% Li_2O, while the underground ore is 35.1 Mt at 1.45% Li_2O. See Table 25.3 for the summary.

Table 25.3: Mine Reserve Material Over the Life of Mine (CV5 Pegmatite)

Design	Ore Tonnage (Mt)	Rock Waste Tonnes (Mt)	OVB Tonnes (Mt)	Avg Li_2O Grade (%)	Total Material (Mt)
Open Pit	49.2	150.0	17.5	1.12%	216.7
Underground	35.1	5.2	0	1.45%	40.3
Total	**84.3**	**155.2**	**17.5**	**1.26%**	**257.0**

**Notes on Physical Quantities:*
1) Total may not sum due to rounding; rounding followed the recommendations of the NI 43-101.

25.5 Processing

The feasibility study uses a DMS-only process for resource beneficiation, selected for its processing simplicity and efficiency in commissioning and ramp-up. By adopting the DMS-only approach, the Project benefits from lower operating expenses due to the reduced complexity and energy requirements. This streamlined process is expected to enhance economic efficiency and align with the commitment to sustainable and responsible mining practices.

This design incorporates two (2) parallel production lines at a feed design capacity of 2.5 Mtpa each, a plant size that has been repeatedly and successfully built and operated in the spodumene industry. Additionally, the DMS process generates a minimal quantity of dry-stacked tailings, further enhancing the Project's efficiency and reducing waste management requirements.

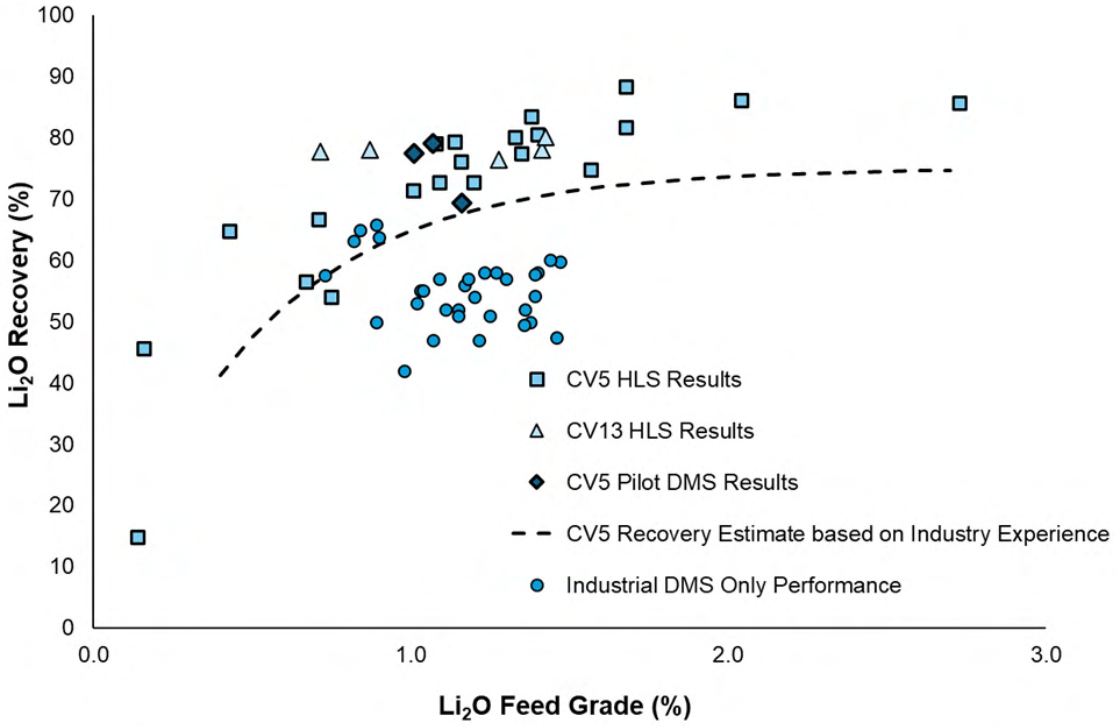
The DMS-only flowsheet has been validated by extensive metallurgical test work conducted by SGS Canada and supervised by Primero, both with extensive experience in lithium processing operations.

The test work to date (summarized in Figure 25.1) has confirmed that the coarse spodumene is the dominant lithium mineral, achieving concentrate grades of over 5.5% Li_2O with global lithium recoveries in HLS testing ranging from 70% to 85% (for feed grades in the range 1.0% to 1.5% Li_2O, respectively).

The Shaakichiuwaanaan pegmatites have repeatedly shown excellent processing performance, generating high recoveries at the target concentrate grade. This ease of processing is attributed to the consistently large spodumene crystals found in the CV5 Pegmatite. The robust recoveries exhibited across a range of feed lithium grades are a key differentiator for the Project.

The test work results from both HLS and DMS of the CV5 material, the expected recovery curve from a 3-size range DMS plant (processing Shaakichiuwaanaan pegmatites), and, for reference, recoveries from other operating DMS-only plants (as compiled by external consultants, Primero) are shown comparatively in Figure 25.1. The Project's higher expected recovery (from other DMS-only operations) is due to the wide size range being treated (9.5 mm to 0.65 mm), the quality of the material (large spodumene grains with a narrow grain size distribution), and the 3-size range DMS plant (which lessens the impact of particle size effect in the DMS process).

Figure 25.1: Metallurgical Test Work Recovery (Global) Results & Industry-Based Recovery Estimates for 3x Size Range DMS Process Plant



By achieving high recoveries with a simpler DMS process design, the Shaakichiuwaanaan Project positions itself with a competitive advantage in the lithium market. Table 25.1 shows the majority of other operating DMS-only plants (Industrial DMS Only Performance), achieving recovery rates well below the estimated Shaakichiuwaanaan Project DMS lithia recovery performance from test work to date.

During this phase of the study, several key recommendations from the previous stage were addressed and incorporated into the updated Project design. While the concentrator was always intended to start as a DMS-only plant, previous recommendations suggested evaluating the potential for a future flotation plant to treat the middlings stream (i.e., the -0.65 mm fraction). Testwork completed during this phase confirmed that the lithium recovery gains from processing this fraction would not offset the additional capital and operating costs. As a result, the DMS-only design will remain in place for the full life of the Project.

Regarding tailings management, the design of the paste backfill plant was optimized based on an improved understanding of the required feed material blend. It was determined that the bypass and DMS tailings streams no longer need to be stored in separate stockpiles, as previously recommended. This change has resulted in a more compact and cost-effective tailings handling area, reducing equipment needs, footprint, and overall construction costs.

25.6 Hydrogeology

The review of available data provided information from which to develop the hydrogeological context:

- The study site is surrounded by several unnamed waterbodies and marshy areas.
- The overburden is discontinuous over the study area and is mainly composed of undifferentiated glacial deposits, made up of sand, silt and gravel in variable proportions with boulders and cobbles. Its thickness varies between 0 m and 48 m.
- The bedrock unit generally consists of alternating gray and green amphibolite, metasediments, and pegmatite intrusions. The RQD shows that the rock is generally of good quality with low fracturing. The bedrock outcrops at several locations within the study area.
- The overburden is a hydrostratigraphic unit but is discontinuous and therefore does not constitute a large-scale aquifer. The bedrock constitutes the primary aquifer in the study area. Groundwater levels measured in the bedrock aquifer vary generally between 0.1 m and 3.1 m below ground surface.
- The saturated hydraulic conductivity geometric mean in the overburden is 1.1×10^{-6} m/s and 5.8×10^{-8} m/s in the bedrock. The variation in hydraulic conductivity did not show correlation with RQD, thus suggesting that flow within the bedrock is dominated by discrete features that are not captured by the RQD measurements.
- The groundwater flow follows topography, with groundwater in the overburden unit and shallow bedrock generally flowing from topographic highs, which act as recharge areas, towards the lakes and wetlands. In general, the groundwater flows towards Lake 001 and Lake 027.
- Background groundwater quality was assessed in samples collected during the two (2) sampling campaigns in the monitoring wells located at the site. The results show background concentrations lower than the EDC and RES criteria of the MELCCFP for the majority of the parameters analyzed, except for arsenic, manganese, molybdenum, and antimony exceeding EDC and copper exceeding RES.
- A three-dimensional groundwater flow model was developed to assess the mitigation measures proposed for Stockpile 002 to prevent any significant degradation of the groundwater quality due to any potential seepage, as the material stored in this stockpile was assessed as being a metal leaching risk, particularly arsenic, and potentially acid-generating (Vision Geochemistry, 2025). The mitigation measures evaluated during this study were the installation of a geomembrane below Stockpile 002 during the mine operation, followed by the installation of an impermeable cover following mine closure. The results of the solute transport model suggest that the mitigation system

is effective, with solute concentrations reaching the environmental receptors being below the applicable criteria after 200 years.

25.7 Tailings, Water, Waste Rock and Overburden Management

This section summarizes conceptual design executed in relation to water diversion infrastructure and the management of project waste materials (tailings, waste rock and overburden). The design includes:

- LOM waste storage area lay-outs.
- Related water management infrastructure.
- Water diversion infrastructure.

25.7.1 Mine Waste Management

The anticipated total tailings production during the LOM is evaluated at 71.1 Mt, with approximately 54.0 Mt disposed on surface, and the remaining 17.1 Mt will be used for paste backfill. Surface disposal of tailings will be at Stockpile 002. The tailings will be dewatered at the plant, transported to Stockpile 002, laid down by the hauling equipment, spread with a dozer, and compacted by a vibratory soil compactor. The tailings are assumed to be metal leaching, for the purposes of this study, and therefore the footprint of Stockpile 002 is lined with 1.5 mm double-textured LLDPE geomembrane.

The anticipated total waste rock production during the LOM is evaluated at 155.2 Mt, with 73.1 Mt considered to be non-leaching (Low Risk) waste rock, and 81.0 Mt is PAG waste rock. In addition, a total of 13.7 Mt of overburden and 3.2 Mt of organics will also be produced, 43.5 Mt of the non-leaching waste rock will be stored in Stockpile 001, and the remaining 29.6 Mt will be stored in the open pit. All of the PAG waste rock (81.0 Mt) produced will be stored in Stockpile 002. Two (2) stockpiles are provided for overburden, and these are located to the south of the pit. Two (2) organics piles will be located near Stockpile 001 and used to store organics stripped from the footprints of both Stockpile 001 and 002.

Stockpile 001 has a total footprint area of 80 ha. The stockpile is unlined and has unlined perimeter ditches and ponds for run-off collection. The collected water is returned to the sedimentation ponds at the plant, and once the sediments have settled out, the water is either reused or disposed of to the environment.

The footprint area of Stockpile 002 is approximately 218 ha and is lined. This stockpile is used for both tailings and PAG waste rock. Runoff from the tailings and waste rock will be collected in lined perimeter ditches and discharged to two (2) lined ponds. The collected water will be returned to the plant for treatment and reuse or disposal to the environment.

A layout of all stockpiles is presented in Section 18.

25.7.2 Water Management

As outlined in Section 18, all contact water generated from the mine's infrastructure will be collected through a network of ditches and contact water collection ponds, then pumped to the appropriate treatment systems. Pump stations will be installed at each pond to facilitate water transfer to designated treatment facilities. In total, four (4) ponds will collect runoff from the stockpiles (Table 25.4), and two (2) ponds will manage contact water from the industrial area. Additionally, contact water from dewatering activities (both open pit and underground) will be pumped directly into the treatment facilities.

Two (2) main types of contact water are anticipated in this Project:

- Water with elevated levels of total suspended solids (TSS): Originating from industrial areas, inert waste rock (Stockpile 001), and mine dewatering activities.
- Water with elevated levels of metal leaching and acid-generating materials, primarily arsenic: Generated from potentially acid-generating (PAG) or metal-leaching (ML) waste rock and tailings (Stockpile 002).

Water with elevated levels of TSS will undergo treatment through sedimentation ponds followed by a polishing pond. Water containing metals will be directed to a treatment system, which includes pH adjustment, the use of chelating agents, coagulation, and flocculation. The treated water will then be discharged to the polishing pond for final polishing. Two (2) treatment trains have been designed to reduce the initial CAPEX and to follow the ramp-up of the stockpile footprint and mine development. Space has been left free beside the water treatment system to accommodate another treatment if necessary.

Although lithium has been detected in the runoff water from the tailings section of Stockpile 002, no specific treatment is currently implemented in the water treatment system. This is primarily because lithium is not currently regulated under either Directive 019 or the MDMER. Additionally, optimization of the concentration process is expected to improve lithium recovery, which should, in turn, reduce its concentration in the tailings.

The overall objective is to consolidate all treated water into a single discharge point for the entire site. This discharge will be released into the CE15 stream using a subsurface diffuser, which will promote effective mixing of the effluent with the receiving water body.

Additionally, runoff from the overburden pile will be managed at the source using a paddock system designed to promote infiltration and evaporation, thereby minimizing the volume of water requiring treatment.

Table 25.4: Stockpiles Water Management Infrastructure

Stockpile	Area Managed	Ditch Network (km)	Basins (m^3 +/-)	No. of Basins
Stockpile 001	Low Risk waste rock	4.2	132,000	2
Stockpile 002	Tailings and potentially acid-generating waste rock	7.2	430,000	2
Stockpile 004	Overburden	None. Run-off paddocks utilized	Not required	0
Stockpile 005	Overburden	None. Run-off paddocks utilized	Not required	0

25.7.3 Water Diversion Infrastructure

The open pit will be developed in two (2) phases, the West Pit and East Pit, extending into a portion of Lake 001. To enable this, a lake diversion system is required, consisting of a diversion channel and a series of staged cut-off dams. The proposed diversion channel, approximately 870 m long with a 5 m base and 0.3% gradient, will be constructed in Year 2 and routed through bedrock with 1H:1V side slopes. It will redirect flow from Lake 001 to Lake 05, affecting about 1.9 km^2 or 1.7% of the lake's 106 km^2 watershed. The alignment ensures safe offsets from the pit and nearby lakes (45 m from the pit and 60 m from Lake 002 and Lake 003).

The diversion channel system is designed to maintain normal hydrological function and safely convey the PMF (Probable Maximum Flood) of 94.9 m^3/s with a minimum freeboard of 1.5 m. The channel invert is set at 373 m a.s.l., matching the existing outlet, while the cut-off dam crests are set at 378.4 m a.s.l. The diversion will replace the natural stream flowing from Lake 001 to the CE 15 stream and is engineered to replicate natural hydraulic conditions, thereby minimizing disturbance of fish habitat travelling in the channel. This shall include a series of drops and pools to create variable flow regimes, as well as placement of boulders or rocks to provide shelter during upstream migration. Hydraulic modelling was conducted, and the results confirm that flow velocities and depths within the channel are consistent with those observed in the existing natural stream (north outlet channel, CE 05).

25.7.4 Closure

The mine closure phase aims to ensure a responsible transition toward sustainable post-operation land use, in compliance with Quebec's regulatory requirements and in collaboration with

Indigenous communities. It is based on a progressive and structured planning approach, including the gradual restoration of inactive areas, the dismantling of industrial infrastructure, and the implementation of rehabilitation strategies tailored to each site component.

Industrial facilities will be dismantled following the principles of reduction, reuse, and recycling, while access roads will be either restored or maintained depending on their community use. Non-essential roads will be decommissioned, culverts removed, natural water flows restored, and surfaces recontoured and revegetated with native plant species to prevent erosion.

Waste rock stockpiles will be managed according to their potential for contaminant generation: Stockpile 001 will be restored with a vegetated cover suitable for low-risk waste rock, while Stockpile 002 will be restored using an impermeable clay cover to limit water infiltration and prevent contaminant generation.

The open pit will be naturally flooded to form a stable lake, secured with berms and integrated into the local hydrographic network.

Water management ponds will be backfilled once water quality is confirmed, following sediment characterization and appropriate disposal. Dikes of the stockpile water collection ponds will be breached to prevent water accumulation at the base of the structures.

Finally, overburden and organic soil piles will be used for cover construction and subsequently restored through scarification and revegetation. These measures aim to ensure the environmental stability of the site, promote ecological reintegration of the land, and meet the expectations of stakeholders, particularly local and Indigenous communities.

25.8 Environmental Permitting

To date, none of the inventories carried out have identified any environmental issues posing a risk to the Project. Potential impacts on species at risk confirmed on the territory can be mitigated by specific mitigation measures. Any encroachment of the Project into fish habitat will have to be compensated under applicable regulations. Similarly, any encroachment of the Project into wetlands or the water environment may be subject to a compensation program.

Environmental characterization fieldwork is scheduled for completion in 2025, providing the full set of baseline data necessary to support the environmental and social impact assessments underway at both federal and provincial levels. Concurrently, the stakeholder engagement and consultation process—

encompassing both Indigenous and non-Indigenous communities—will remain ongoing, guided by the core principles of respect, transparency, and collaboration.

25.9 Capital Cost

Capital costs are divided into three (3) categories: initial capital costs, expansion capital and sustaining capital costs during the life of mine. Total cost for the Project is evaluated at $2,914.7M. Note that the amount excludes a pre-production revenue of $101.7M during the Phase 1 construction and the reclamation and closure cost of $248.4M.

The total of the initial cost for the Shaakichiuwaanaan Project is estimated to $1.497.7M, including contingencies and indirect costs and excluding the $101.7M of pre-production revenue. The initial capital cost is made up of Phase 1 costs ($1,349.4M for open pit mine) and 23% of the Phase 2 costs ($148.3M underground mine).

The expansion capital includes the costs related to the construction and development of the underground mine, the camp expansion, the second train of the processing plant and their related indirect costs and contingencies. The expansion capital will occur during the first 2 years of production from June 2030 to March 2032 (Years 1 and 2). All other capital expenses after Year 2 are included in the sustaining capital. The expansion capital cost was estimated to be $480.5M.

Sustaining capital costs include all expenditures necessary to sustain operations throughout the LOM. Sustaining costs start at Year 1 until the end of the mining operations and were estimated to amount to $936.4M over the LOM.

Table 25.5 provides an overview of the capital costs (pre-production, expansion and sustaining) on a cumulative basis for the life of the Project.

Note that Table 25.5 does not consider the amount of $248.4M of the closure cost.

Table 25.5: Project Total Capital Cost Summary

Capital Expenditure	Phase 1 OP Initial Capital Cost ($M)	Phase 2 UG Initial Capital Cost ($M)	Initial Capital Cost ($M)	Phase 2 UG Sustaining Capital Cost ($M)	Total Devel. Capital Cost ($M)	LOM Sustaining Capital Cost ($M)	Total Capital Cost ($M)
100 - Infrastructure	124.9	-	124.9	24.8	149.7	30.8	180.5
200 - Power and Electrical	173.8	-	173.8	46.2	220.0	25.0	245.1
300 - Water Management	128.2	-	128.2	18.7	146.9	100.5	247.4
400 - Surface Operations	18.6	-	18.6	-	18.6	11.9	30.5
500 - Mining	120.0	99.1	219.1	36.4	255.5	550.5	806.0
600 - Process Plant	217.3	20.1	237.4	167.0	404.4	-	404.4
700 - Construction Indirect	262.8	0.1	262.9	123.8	386.7	-	386.7
800 - General Services / Owner's Cost	99.8	4.7	104.5	13.4	117.9	31.6	149.6
900 - Pre-production, Start-up, Comm.	73.3	9.3	82.6	1.5	84.1	186.1	270.2
Total Initial Capital Expenditures Excluding Contingency	**1,218.7**	**133.3**	**1,352.0**	**431.8**	**1,783.8**	**936.4**	**2,720.2**
990 - Contingency	130.7	15.0	145.7	48.7	194.4	-	194.4
Total Initial Capital Expenditures	**1,349.4**	**148.3**	**1,497.7**	**480.5**	**1,978.2**	**936.4**	**2,914.6**
Less: Pre-Prod. Credit Net of TC / RC & Royalties	(101.7)	-	(101.7)	-	101.7	-	(101.7)
Total Initial CAPEX Net of Pre-Production Crédit	**1,247.7**	**148.3**	**1,396.0**	**480.5**	**1,876.5**	**936.4**	**2,813.0**

25.10 Operating Cost

The operating cost estimate is based on a combination of experience, reference projects, firm quotes, and factors as appropriate for a feasibility study. No cost escalation or contingency has been included within the OPEX.

The OPEX in this study was prepared by GMS with inputs from Primero and PMET. The operating costs include the costs to mine and process the mineralized material to produce spodumene concentrate, general and administration expenses (G&A) and transportation. Operation will be implemented by PMET.

The operating cost over the 19-year mine life is estimated to be $6,786M, which represents a unit operating cost of $511.9/t (Cash Operating Cost at Site). The estimated cost for the concentrate transportation from the mine site to Grande-Anse Port is $217.19/t dry. This, however, excludes the costs for the driver's accommodation and transportation to the mine site. Those costs are captured under the G&A costs under the mine operating cost.

25.11 Economic Analysis

The pre-tax base case financial model results in an internal rate of return of 19.9% and an NPV of $2,514M with a discount rate of 8%. The simple pre-tax payback period is 4.9 years. On an after-tax basis, the base case financial model results in an internal rate of return of 18.1% and an NPV of $1,594M with a discount rate of 8%. The simple after-tax payback period is 4.7 years. Table 25.6 shows the financial analysis summary.

Table 25.6: Financial Analysis Summary

Description		CA$ M	US$ M
Pre-Tax	**Discount Rate**		
	0%	8,358	6,237
	5%	3,995	2,981
	8%	**2,514**	**1,876**
	10%	1,805	1,347
	15%	640	478
	Pre-Tax IRR	19.9%	
	Payback Period	4.9 years	
After-Tax	**Discount Rate**		
	0%	5,418	4,043
	5%	2,581	1,926
	8%	**1,594**	**1,190**
	10%	1,115	832
	15%	312	233
	After-Tax IRR	18.1%	
	Payback Period	4.7 years	

The Project is most sensitive to the exchange rate, grade, spodumene concentrate price and process recovery and much less to the operating costs and capital expenditures.

The Shaakichiuwaanaan Project warrants further study to increase the level of engineering to decrease the technical risks.

25.12 Market

Lithium remains indispensable to the global energy transition, driven by robust demand growth from electric vehicles and battery energy storage systems. Despite notable supply expansion in recent years, the industry faces significant hurdles—long project development timelines, permitting, financing, and technical challenges—that are expected to keep the market tight well into the 2030s. A supply-demand gap is projected to emerge after 2030, with deficits increasing through 2040, which is likely to sustain strong long-term pricing for spodumene concentrate, mainly between 2030 and 2035. Rapidly expanding applications such as battery energy storage systems further reinforce lithium's critical market position

beyond transportation. Price forecasts are expected to be in the bracket of US$1,100 to 1,350/t in the long term.

In January 2025, the Company entered into a binding offtake term sheet with Volkswagen's 100%-owned vertically integrated battery manufacturer, PowerCo SE (PowerCo), to supply 100,000 tonnes of spodumene concentrate (SC5.5 target) per year over a 10-year term (PMET, 2024) (PMET, 2025). PMET is working on finding other clients.

25.13 Risks and Opportunities

As with most mining projects, the Shaakichiuwaanaan Project faces risks that could affect its economic viability. Some risks are linked to incomplete information and can be managed through further sampling, testing, engineering, and planning. Table 25.7 lists the key Project-specific risks, their potential impacts, and possible mitigation measures.

The most significant risks include delays in permit approvals, limited labour availability, potential dam failures, and dependence on electricity supply from Hydro-Québec. Many of these can be managed through proper planning, engineering, and proactive engagement with parties.

Other risks are outside the control of the Project, such as fluctuations in commodity prices or exchange rates, and changes in government regulations. These external factors, common to all mining projects, could reduce profitability if they differ from the assumptions used in the economic model.

Conversely, a number of opportunities exist that could improve the Project's economics, schedule, and overall outcomes. These, aside from typical industry-wide factors like commodity prices or exchange rates, are outlined in Table 25.8. Further studies and engineering work are warranted before these opportunities can be reflected in the Project's economics.

Table 25.7: Project Risks

Area	Risk Description and Potential Impact	Mitigation Strategies
Geology	Interpretations of mineralization, grade, geometry and continuity in mineralized zones affecting mine plan.	Continue definition drilling campaigns and perform an underground bulk sample. Improve grade certainty by diamond drilling and validate metallurgical recovery by testing, pilot plant and sampling throughout the mineralized material mined areas (bulk sample).
Project Execution	Failure to obtain all types of permits, licences, and approvals, causing a missing window for deforestation, early works, first dam construction and fish relocation activities. All is required in the first summer.	Maintaining the schedule as the Project progresses towards execution.
General Worker Shortage	Ability to meet construction and production targets due to scarcity of workforce and highly competitive market conditions	Hiring key management team and early next engineering phase.
Project Execution	Fish relocations, dam construction and Lake 001 dewatering delays, impacting critical early steps of open pit pre-production schedule.	Continue evaluation in the next engineering phase and as the Project progresses towards execution.
Geology	The nature, location, and extent of faulting and other geological structural controls could impact the stability of pit walls, crown pillar, underground stopes and underground development.	Progress drilling campaign in the next engineering phase, and as the Project progresses towards execution. Improve geological understanding through the execution of a bulk sample.
Geotechnical and Hydrogeology	A dam breach flooding the mine. More water is coming into the pit than accounted for.	Complete dam breach study. Additional water pumping and treatment capacity.
Mine	Use of paste backfill and operation dilution, rates and target assumptions affecting schedule and deliverables.	Refine evaluation and planning in the next engineering phase, and as the Project progresses towards execution.
Mine	Insufficient sizing of the paste plant buffer silos (tailings and binder).	Refine evaluation and planning in the next engineering phase, and as the Project progresses towards execution.
Mine / Process	Abrasion due to the pegmatite could be higher than anticipated, affecting estimated shutdown periods and ground-engaging tools (GET).	Complete evaluation in the next engineering phase.
Process	Particle size distribution for ROM not meeting DMS plant design criteria, affecting spodumene recovery.	Perform a bulk sample to generate material representative of future mine ROM (via blasting).

Area	Risk Description and Potential Impact	Mitigation Strategies
Waste stockpile	Waste stockpile design and capacity are being affected by fish habitats.	In-pit disposal.
Wastepile	Planning and operational protocols for lining and deposition of the Stockpile 002, affecting mine planning and increasing potential environmental impacts.	Complete operational protocols in the next engineering phase.
Employment	Owner workforce could be hard to complete due to Quebec's current workforce shortage, coupled with unattractive FIFO conditions.	Implementing good public relations and recruitment campaigns, and performing benchmarking in FIFO conditions.
Employment	Operations team salaries use the industry median rate, which might impact the ability to hire skilled workers.	Introduce well-defined contract provisions and consider revising the wage scale to remain competitive.
Employment	Local workforce targets are established at 15% for the construction and 20% during operation.	Initiate local workforce training months ahead in the local communities.
Mine	Blasting near Lake 001 and permanent infrastructures.	Adopt good blasting techniques and QA/QC for both open pit and underground mines.
Mine	Tight process plant head feed grade targeted by blending grade between open pit, underground mine and stockpiles, affecting process plant recovery.	Increase mine planning combined with communication between departments during operation.
Mine	The crown pillar of the underground mine is at 85 m, generating water ingress affecting operation and mine-induced stresses at the end of the mine plan.	Further optimize underground mine plan and increase confidence in crown pillar knowledge (e.g., rock mass and in-situ stress conditions) and operational awareness as the Project progresses towards bulk sample program or execution
Project Execution	Limited timeline to install bridges to cross the discharging river of Lake 001, to access the south portion with Mining and EW equipment, impacting construction and mine schedule, early critical steps.	Continue evaluation and planning in the next engineering phase and as the Project progresses towards execution.
Remote Site	Unexpected road closure of any sort from the south to the site to supply goods and ship concentrate.	Storage at site is designed to sustain sufficient autonomy and storage to pursue operation without impact.
Logistics and Transportation	LG4 airport usage is not currently allowed.	Continue discussion with HQ and AHJ to confirm usage or confirm airport strategy.
Long Lead Items	The actual lead time for the main transformers of the required size is over 24 months of delivery.	For the construction schedule, establish a good long lead-time item list to avoid construction / ramp-up delays.

Area	Risk Description and Potential Impact	Mitigation Strategies
Electrical Power Requirements	The Hydro-Québec (HQ) network requires improvements to deliver the required power to the Project. Delays to that critical item will impact Project sequence and economics if not closely tracked.	Confirm with HQ in the early stage the power requirement for the Project and pursue the power connection demand.
CAPEX	No redundancy for the main site transformer in the first two (2) years of Project execution.	Build a strong inspection plan with improved warranty clauses for the transformer supplier.
Processing	Not hitting the iron content specification in concentrate because of higher than planned dilution.	Modification of the flow sheet with the addition of ore sorting and/or magnetic separation. Further metallurgical testing with additional material (bulk sample).
Concentrate Transportation	The Project assumed a 75-t load capacity for road haulage of concentrate from site to Matagami is above the current approved limits.	Progress permitting approval process with local Authorities to use 75 t or use smaller trucks with higher operational cost.
Weather Condition	Winter cold weather is affecting various stages of material handling due to freezing (storage areas, pads, trucks or trains).	Heat pads and storage bins where required. Trucks and train are to be provided with adequate systems or features.
Financial	Commodity volatility. Foreign exchange. Use of the CMT-ITC.	MOU, LOIs or other midstream and downstream product purchase agreements. Engage qualified tax specialists and align estimates with government guidance to ensure compliance and consistency in determining eligible capital expenditures.
Closure	Clay cover quantity to cover the waste rock Stockpile 002.	Perform additional investigation and borrow pits identification to supply the required volume.

Table 25.8: Project Opportunities

Area	Opportunities	Strategies
Geology	Conversion of remaining Inferred Mineral Resources at CV5 to Indicated Resources and then to Probable Reserves, extending the LOM.	Continued infill drilling of the current Mineral Resource at CV5 and CV13.
Geology	Conversion of Exploration Target material to Mineral Resources, extending the LOM.	Continued exploration drilling at the Property, including the other known LCT pegmatite clusters.

Area	Opportunities	Strategies
Financial	Improve Initial CAPEX by using various financing instruments at FID.	Explore equipment leasing, consignment options, and other such financial instruments to reduce funding requirements at FID.
Closure	Clay cover quantity reduction to cover the waste rock Stockpile 002.	Optimize the geochemistry modelling of piles to possibly reduce clay volume.
Mine	Further refine mine development strategy to benefit from the very high-grade Nova Zone earlier in the LOM.	Explore strategies to accelerate development towards the Nova Zone.
Mine	Optimize pastefill binder content to reduce operating costs of the UG mine.	Reevaluate the binder recipe during operation with in-situ data to better inform pastefill strength design requirements.
Geology / Process	Tantalum as a co-product of lithium production.	Progress geological understanding, metallurgical testwork, technical feasibility and supporting economic studies to include tantalum in Project valuation.
Geology / Process	Caesium as a co-product to lithium production.	Progress geological understanding, metallurgical testwork, technical feasibility and supporting economic studies to include caesium in Project valuation.
Geology / Process	Gallium as a co-product to lithium production.	Progress geological understanding, metallurgical testwork, technical feasibility and supporting economic studies to include gallium in Project valuation.
Geology / Process	Rubidium as a co-product to lithium production.	Progress geological understanding, metallurgical testwork, technical feasibility and supporting economic studies to include rubidium in Project valuation.
Process	Improve spodumene recovery by adding a flotation circuit.	Investigate incremental recovery gains from flotation to process flowsheet and evaluate economic potential.
Process	Improve spodumene recovery by pre-concentration of marginal ore (mineralized waste) with the use of ore sorting.	Investigate incremental material gains from ore sorting on OP and UG marginal material and evaluate the economic potential.
Power	120 kV transmission power line construction requires a high capital cost up front.	Developing a partnership with experienced local communities for its construction and maintenance could improve.
Mine	Optimize water management infrastructure design and operation, as well as the closure plan concept, by improving segregation of materials in operations.	Improve geochemical understanding of the various rock units and explore opportunities for optimized segregation during operations. Maximize potential for in-pit disposal.

Area	Opportunities	Strategies
Transportation	Evaluate a new deep-sea port in James Bay to reduce shipping distance.	Engage with other stakeholders, most notably the Cree and the Government of Quebec.
Transportation	Completion of the proposed Route 167 extension north of Renard to the LG4 dam complex.	Engage with other stakeholders, most notably the Cree and the Government of Quebec.
Transportation	Completion of the proposed rail line running parallel to the James Bay Road.	Engage with other stakeholders, most notably the Cree and the Government of Quebec.
Mine / Process	Improve Project economics by using automation.	Investigate the potential introduction of a Remote Operations Centre to maximize the benefits of automation and reduce the requirement for a FIFO workforce.
Spodumene Treatment	Optimize Project logistics by pursuing the development of an electrified process for conversion at site of spodumene concentrate into a value-added lithium industrial salt with a higher ratio of lithium contributing to mass, reducing the requirement for trucking.	Explore partnerships to support the development of the process and technology (Refiners, OEMs, Provincial and Federal Agencies, Hydro Quebec and others) and progress the Project towards feasibility.

25.14 Conclusions

G Mining Services, BBA, Primero, AtkinsRéalis, Vision, WSP, Alius Mine Consulting, Mailloux Hydrogéologie, CGM Expert, and Vision Geochemistry were engaged by PMET Resources to complete a Feasibility Study on the Shaakichiuwaanaan Project to evaluate its economic viability.

This Technical Report summarizes the key results, covering Mineral Reserves, mine design, metallurgy, processing, infrastructure, environmental management, capital and operating costs, and overall economic analysis for a lithium-only mining operation at the CV5 Pegmatite.

The study was prepared by qualified independent consultants using recognized engineering practices. The conclusion is that the Project contains sufficient detail to support a positive economic outlook. Section 26 provides recommendations to mitigate risks and take advantage of potential opportunities identified during the Study.

In summary, the Qualified Persons recommend that the Company advance to the next stage of development, including basic engineering and procurement of long-lead items as recommended in Section 26, to achieve commercial production by June 2030.

26. RECOMMENDATIONS

Based on the results of the technical and financial analyses of this lithium-only Feasibility Study (FS) on the CV5 Pegmatite, which demonstrates positive Project economics, GMS recommends proceeding with the basic engineering for the Project and initiating limited notice to proceed (LNTP) with suppliers for critical items and equipment of long delivery during 2027 prior to the full construction release once Project financing is finalized and the main permits are issued.

Table 26.1 summarizes the proposed budget of $83.1M to advance the Project, as budgeted in the FS from Jan 2027 to Dec 2027, and to progress into the basic engineering stage and perform the recommendations discussed in this section to achieve commercial production by the end of May 2030.

Table 26.1: Cost Estimate Associated with Recommendations

Description	Amount $M
Down payment for camp, water management and other infrastructure.	18.6
LNTP for long-lead items Progress on power connection to HQ, for transmission line and main transformer procurement	33.6
Down payment on equipment and contractor	3.8
Progress basic engineering and initiate detailed engineering	16.4
Owners Cost	2.6
Contingency 11%	8.1
Total Project Costs for 2027	**83.1**
Geology and Exploration[1]	15.0
Geochemistry Testing[1]	0.5
Metallurgical Testwork[1]	0.3
Exploration Camp Improvement[1]	2.0
Total Anticipated Costs	**100.9**

**Note:*
(1) This cost is not included in the Project Feasibility Study Capital Estimate.

26.1 Geology / Exploration

Although the Property remains strongly prospective for copper-gold-silver at the Maven Trend, and gold at the Golden Trend, the results of the 2021 through 2024 drilling have firmly focused exploration on lithium pegmatite at the CV Trend, and the prospectivity of the Property to host additional sizable occurrences.

Significant and continued lithium pegmatite exploration, including a combination of surface work and drilling, is warranted and recommended.

Recommended activities would include support for advancing the CV5 Pegmatite through detailed engineering, as well as exploration focused on other LCT pegmatites at the Property. The activities would be completed as a single phase and specifically include:

- Additional diamond drilling at the CV5 Pegmatite targeting early mine-life and areas of relatively higher geological complexity to support ongoing modelling efforts.
- Geotechnical, geomechanical, and hydrogeological drilling at CV13 to support future economic studies.
- Additional step-out and delineation drilling.
- Drill testing of the other LCT pegmatite clusters at the Property and corridors between.
- Continued surface mapping, prospecting and channel sampling over mineralized LCT pegmatite outcrops, and other unexplored areas of the Property.

The estimated budget is $15M and is summarized in Table 26.2. It includes all supporting costs to operate the program.

Table 26.2: Phase 1 Estimated Budget

Task	Estimated Cost ($)
Drill Exploration in Support of Detailed Engineering (CV5)	5,000,000
Geotechnical, Geomechanical, & Hydrogeological Drilling at CV13	2,000,000
Drill Exploration (CV5, CV13, CV9)	6,000,000
Drill Exploration (other LCT Pegmatites)	1,500,000
Surface Exploration, Geophysics	500,000
Total	**15,000,000**

26.2 Mining

The recommendations for the open-pit and underground mining aspects of the Project are described in the following sections.

26.2.1 Open Pit

26.2.1.1 Blasting Mitigation Near Lake 001 and Permanent Infrastructure

A variety of major infrastructures (dams, lakes, administration buildings, process plant, roads, stockpiles, etc.) are located around the pit, making controlled blasting techniques mandatory. A review of blasting techniques and mitigation methods for both the open-pit and underground mines is recommended to ensure a safe operation and to mitigate impacts on the nearby environment.

26.2.1.2 Other Nearby Mineralized Deposits

The feasibility study only considered CV5 in the mining sequence and the process plant head grade. As resources evolve, the integration of other mineralized deposits in the near vicinity of CV5 should also be considered in the next engineering phases.

26.2.1.3 Include Other Elements in the Blending Schedule

The feasibility study only considered lithium in the mining sequence and the process plant head grade. Other identified relevant elements, such as caesium, tantalum, and gallium should also be considered in the process plan feed grade to optimize potential recovery.

26.2.1.4 Early Work Mass Balance

The mine plan identified a potential additional need for overburden and waste material during the early phases of construction works. Based on the most recent data available, assumptions were used on the removal rate and to define whether the composition of the overlaying till is suitable for construction. More geotechnical drilling is recommended to better understand the composition and disposition of the glacial till across the property and provide better estimates of quantity.

Additional potential quarries along the road connecting the mine site to the Trans-Taiga Road have been identified to generate the required additional material. At this stage, no geochemical characterization of these potential sites is available to assess quality or whether the material excavated from these sites is potentially acid-generating rock (PAG) or non-potentially acid-generating rock (non-PAG). It is recommended to perform these tests in the next design phase to ensure that suitable material is readily available in the early work stage.

26.2.2 Underground Infrastructure

26.2.2.1 Bulk Sample Program

Performing a bulk sample in an underground mine provides a degree of confidence in the geological aspects. A bulk sample, typically composed of several thousand tonnes, can capture a representative portion of the mineralization, thereby providing valuable information on grade, metallurgical recovery and underground conditions. Such detailed information acquired from in-situ observation, qualification and quantification of various factors is expected to validate a number of key project design assumptions, such as (but not limited to):

- Grade spatial distribution: used to validate the geological interpretation of the mineralised zone, with emphasis on grade distribution and contacts. Especially important for the high grade (and high margin) Nova Zone.
- Rock mass properties: used to inform on the in-situ stress conditions, stope stability and dilution performance, crown pillar stability, excavation sizing, ground support considerations. Optimization of ROM Pad Rehandling and Direct Back-tip Dumping by UG Trucks.

The primary crusher grizzlies are not sized to allow direct dumping from underground trucks. This assumption generated additional rehandling costs by open-pit equipment. This assumption is suggested to be reviewed in the next phase of engineering to ensure that the most viable and economical way to manage ore on surface is identified.

26.2.2.2 Battery Electric Vehicles (BEVs) and Optimization of the Contract Development Phase

It is planned to use a mining contractor for the portal ground support and the pre-production development period, which is estimated to be two (2) years. BEV technologies are evolving rapidly, and as the Project moves towards operation, it is suggested to review all BEV selection and the required operational needs to implement such equipment technologies and reevaluate if using a mining contractor is still the best approach.

26.2.2.3 Mine Schedule and Design Optimization

The underground mine schedule has been optimized based on the known properties of the ore body. As more drilling is performed, the life of mine plan (LOM) and the thickness of the crown pillar should be reviewed accordingly to ensure the best economics. Specific engineering studies regarding, among others, geotechnical (soil), rock mass strength, structural domaining and hydrogeology will be needed to fully fulfill

the “Regulation respecting occupational health and safety in mines Article 77” as the Project progresses towards execution. Furthermore, operational awareness mechanisms such as live feed monitoring and a trigger action response plan (TARP) are additional means by which the company can further ensure safe and sustainable operation, particularly when operations take place within 100 metres of the water body.

26.2.2.4 Mineralized Areas Under the Open Pit

Mineralized areas under the open pit have been identified and evaluated as non-economic. As more drilling is performed and the economic context evolves, it is suggested to review the potential of adding these lenses to the LOM.

26.2.2.5 Portal and Ventilation Raise Location

Portals and ventilation raise locations are based on preliminary infrastructure layout but should be reviewed in detail with a site visit and field investigation, mainly to optimize the portal’s location and adjust it for operational efficiency. Ventilation raises location should be optimized to avoid the lake and ensure there are no water connections on surface.

26.2.2.6 Mine of the Future

The mine of the future is one that leverages technology to improve the transversal effectiveness of the collaboration of the different actors of the process. These measures can enhance the overall equipment effectiveness (OEE) of the value chain and increase productivity, safety and sustainability. Therefore, it is recommended to study the impact of these technologies on operations and planning.

Here are some examples of relevant use cases that leverage technology and their impact on processes in the mining industry:

- Geolocation, tracking and fencing: Mining companies can use GPS, RFID, and other wireless technologies to track the location and status of people, equipment, and materials in real time.
- Augmented reality: Mining companies can use augmented reality (AR) to overlay digital information onto the physical environment, such as instructions, diagrams, or data.
- Predictive asset maintenance: Mining companies can use sensors, data analytics, and artificial intelligence (AI) to monitor the condition and performance of equipment and machinery.
- AI and continuous systems planning using data centralization: Mining companies can use AI to optimize their systems planning by integrating and analyzing data from various sources, such as sensors, cameras, drones, geolocation, and tracking devices.

- Cybersecurity and security operation center (SOC) are important aspects of the mining industry, as they can help to protect the critical infrastructure and data from cyber threats and incidents.

26.3 Processing

26.3.1 Metallurgical Testing

As the Project advances, further test work will be required to develop a DMS-only flowsheet. These tests are:

- Comminution test work for crushing (e.g., Bond Crushing Work Index “CWi”). This work serves to confirm the crusher sizing and provides an indication of the size distribution feeding the plant.
- Additional magnetic separation test work, particularly in the coarse concentrates, following the results from CRIMMs. Further larger-scale testwork is recommended to evaluate the unit's ability to operate at elevated throughputs and quantify the potential performance impacts of overloading the unit. Even if performance declines slightly under higher loads, this may still be a preferable option compared to bypassing the coarse fraction entirely, particularly since the coarse material performed well when included in testing.
- Metallurgical testwork on CV13 Pegmatite material is highly recommended for spodumene and tantalite testwork, with a focus on the Rigel and Vega caesium zones for caesium and rubidium testwork.
- Due to the width and orientation of the CV5 Pegmatite lenses, the expected dilution of the plant feed is expected to be relatively low. However, there may be opportunities to maximize the extraction of spodumene concentrate from the deposit if parts of the deposit with higher dilution are directed to an ore sorting processing solution. Ore sorting test work is planned for the next phase of test work.

26.3.2 Processing Plant

The following opportunities exist for the processing plant. Further study of the following items is recommended in the next stage of the study:

- The iron specification of <2.0% Fe_2O_3 based on the typical value as of the time of the study. Investigating the future implications of a +/-0.1% Fe_2O_3 on the sales price of the concentrate will determine the feasibility of certain project options. For example, depending on the sensitivity of iron contamination on spodumene, the sales price will determine whether the Project should install ore sorting or additional magnetic separation units.

- Further optimization of the concentrate magnetic separation circuit is recommended. Additional testwork should be undertaken to evaluate alternative technologies and operating conditions. In particular, further testing by the Changsha Mining and Metallurgy Research Institute (CRIMMS) is warranted to assess the unit's performance at elevated throughputs. Demonstrating the unit's ability to operate under overloaded conditions could enable simplification of the circuit by allowing all concentrate to be treated without bypassing separation on the coarser material, potentially improving both efficiency and product consistency.
- With the current understanding of the geological body (i.e., its substantial width and its vertical orientation), an integrated ore sorting plant within the crushing circuit is not required. However, there is an opportunity to target blocks that contact the host rock to be directed to a distinct storage pile. Spodumene can be recovered from this pile via a modular ore sorting plant. This would allow for more recovery of spodumene from the deposit without feeding high amounts of external dilution (the main source of final iron contamination) to the plant.
- To increase revenue generation, there is potential to incorporate by-product recovery circuits into the overall processing strategy. Additional testwork and trade-off studies are recommended to evaluate the feasibility of producing tantalum, caesium, and rubidium products. In particular, caesium recovery could be integrated with the modular ore sorting plant concept. While the ore sorter is primarily being considered for intermittent use in processing high-dilution material, it could also be repurposed, during low dilution periods, for producing a saleable caesium product from a separate deposit. This dual application would enhance the plant's flexibility and strengthen the justification for investment in the ore sorting unit.
- The concentrator design at Project start-up is recommended to be a DMS-only plant (as currently described in this study). The start-up of DMS only plants for both feed tonnages and recovery has consistently been shown to be very quick. Installing a processing plant for the lithium contained in the bypass fraction (i.e., the -0.65 mm fraction) as a phase 2 later in the Project's life cycle is an opportunity for the Project. Test work will ultimately dictate the nature of this processing plant.
- Currently, the tailings (i.e., the DMS bypass fraction and the DMS tailings) are being stored in separate piles. This is to offer maximum flexibility to the generation of a correct mixture of material for the paste backfill plant. As the correct recipe of paste backfill feed material is ascertained, there will be an opportunity to optimize the tailings handling area.

26.4 Infrastructure

26.4.1 Main Site Layout – General

The feasibility optimized the locations of the main infrastructure, such as stockpiles, plant and camp area, considering natural topography, safety and water body limitations. During the next steps, the infrastructure locations, including buildings and waste stockpiles, need to be confirmed by condemnation and geotechnical drilling, as well as various additional studies and field surveys.

26.4.2 Hydrogeology

Hydrogeology recommendations relate to groundwater contamination prevention and monitoring.

- Effective quality control / assurance should be implemented during geomembrane installation to minimize the number of defects and therefore decrease the percolation rate through the geomembrane at the Stockpile 002 facility and reduce potential for groundwater contamination.
- Implementation of a groundwater monitoring program to monitor the groundwater quality surrounding Stockpile 002 with periodic groundwater monitoring and an appropriate trigger level set to meet groundwater quality protection objectives.
- The projections obtained from the hydrogeological modelling studies must be compared with the groundwater monitoring results collected in the field. The hydrogeological modelling study should be updated and recalibrated as needed based on field data to refine the predictions.

Hydrogeology recommendations relate to mine dewatering.

- Conduct packer testing along the planned ramp particularly along shear zones.
- Map the faults at surface and conduct packer testing on those faults to validate the model assumptions.
- Continue long-term groundwater level monitoring.

26.4.3 Water Management

Site-Wide Water Balance and Quality:

- Water deviation infrastructures were designed to transfer Lake 001’s outflows to Lake 05, located in the northern area of the open pit. A flood analysis has been conducted using a 1D hydrodynamic simulation (HEC-RAS) to evaluate the likelihood of flooding along and down gradient of the diversion

channel. Hydraulic simulations revealed that the diversion channel effectively contains the flood flow events, including the Probable Maximum Flood. Across all modelled flood scenarios, inundation is primarily confined to vegetated areas, the river's main channel, adjacent floodplains, and downstream lakes connected to the diversion channel. However, in the existing natural channel, localized erosion is evident in specific areas such as rapids, drops, and channel constrictions. Based on the erosion study findings, zones with high erosion potential should be identified and prioritized for targeted protection measures.

- An enhanced flood assessment using 2D modelling is recommended in future analyses to refine the results, and Seasonal water level fluctuations in Lake 001 should be monitored and used to confirm or update the proposed elevation for the diversion dam crest.
- The diversion channel design supports fish passage by simulating pools and drops using a 1D HEC-RAS model, as a high-level approach. The purpose is to maintain ecological connectivity, support aquatic habitat function, and reduce potential impacts on fish migration. For future studies, a detailed analysis is recommended, incorporating the full configuration of drop and pool structures to assess flow conditions throughout the channel and ensure long-term fish passage viability.
- A dynamic site-wide water balance model was developed using GoldSim™, simulating daily flows under various climate scenarios. It is recommended to refine the model using updated climate data and field measurements, and to conduct probabilistic simulations to assess uncertainty and support operational decision-making.
- Cut-off dams were designed to isolate Lake 001 from the open pit and safely retain water under PMF conditions, with crest elevations sized to maintain a minimum freeboard of 1.5 m. For the next phase, a dam breach analysis and detailed flood risk assessment should be conducted to evaluate downstream impacts and confirm emergency response protocols. Additionally, a wind setup analysis is recommended to validate the adequacy of the freeboard under extreme wind conditions, particularly given the exposed geometry and fetch length of the lake.
- Contact water ponds around Stockpiles 01 and 02 were designed with embankments to manage runoff. For the next phase, a dam breach analysis is recommended to evaluate potential downstream impacts in the event of failure, particularly for ponds associated with run-off from PAG materials.
- Emergency spillways were designed for all ponds to safely convey the PMF. For the next phase, a full hydraulic design of the spillways should be completed, including the layout and geometry of the spillway chutes, dissipation basin, erosion protection, and confirmation of spillway location relative to pond embankments and downstream receptors.
- Catchment paddocks were designed around overburden and organic piles using low-permeability dikes to temporarily store runoff and promote sediment settlement, sized for a 1 in 10-year rainfall

event. For the next phase, it is recommended to confirm the paddock sizing and performance through hydrological modelling and field validation, and to assess long-term erosion control and maintenance requirements.

26.4.4 Water Treatment

The water treatment system has been designed based on the most recent data from the water balance and geochemical models. It is recommended that the design basis be updated in the next phase of the Project to reflect the latest geochemical findings.

The system has been developed to meet the criteria of Directive 019 and the MDMER regulations. The Project should receive the environmental discharge objectives from the MDDEFP, which will include the receiving environment's capacity to accept the effluent. Modifications to the treatment system may be required once these objectives are received.

Certain parameters, such as lithium, are not addressed by the current water treatment design. Continuous monitoring of both the influent and effluent is required, and the treatment system should be updated as needed based on the monitoring results.

Space has been reserved adjacent to the water treatment infrastructure to accommodate a potential future treatment unit. This was done to ensure compliance with regulatory requirements for TSS or ammonium, if additional treatment becomes necessary.

A single discharge point is currently planned for this Project, located in the CE15 stream. A more detailed investigation should be conducted to confirm the feasibility of this design and to validate the stream's capacity to receive the effluent year-round.

26.4.4.1 Potable Water

The potable production system has been designed based on the projected population of workers in the permanent camp. If the capacity needs to be adjusted, the design of the treatment system must be updated accordingly.

The groundwater exploration program did not yield sufficient water to meet the Project's potable water requirements. As a result, the freshwater intake for potable water production will be in Lake 308, north of the infrastructure, to avoid any contamination from runoff originating from stockpiles near the freshwater source.

The water from Lake 308 has not been fully characterized in accordance with the MDDEFP guidelines. Nonetheless, the treatment proposed in this project has been designed to ensure the production of potable water, even in the event of microbiological contamination. To validate the system and ensure regulatory compliance, it remains essential to sample the freshwater from Lake 308 following the MDDEFP protocols.

The availability of water in Lake 308 has been assessed at a high level to confirm the general availability of the resource. However, a detailed hydrological study is required to ensure the long-term availability of freshwater for potable water production, including under drought conditions

26.4.4.2 Sewage System

The sewage treatment system has been designed based on the projected population of workers in the permanent camp. If the capacity needs to be adjusted, the design of the treatment system must be updated accordingly.

26.4.5 Geochemical Testing and Modelling

Recommended and continued geochemical testing to improve the understanding of geochemical risk associated with mine waste materials includes the following:

- Field barrel testing (FBT) is scheduled to commence in Fall 2025 on-site to assess the geochemical behaviour of waste rock material at an intermediate field scale, with the intent of bridging the gap between laboratory-based kinetic tests and full-scale field conditions. The FBTs include five (5) tests in total – four (4) FBTs, each containing representative materials from each of the four (4) main lithology types (amphibolite, barren pegmatite, metasediments, and ultramafic material), and one (1) composite FBT has been prepared to represent Stockpile 001 which is made-up of the appropriate lithological proportions and the segregation criteria (sulfur and arsenic cut-off values) to classify the stockpile is Non-PAG / Non-ML.
- Initiation of a dry and submerged column test is scheduled to commence in Fall 2025 to serve as a confirmatory test simulating Stockpile 001 waste rock material. The column tests are made up of the appropriate lithological proportions and contain the segregation criteria (sulfur and arsenic cut-off values) to classify the stockpile as Non-PAG / Non-ML. The dry column will represent Stockpile 001 on surface, and the submerged (water-saturated) column will represent backfilling of the open pit with Stockpile 001 waste rock.
- An additional column test containing PAG waste rock material from Stockpile 002 is recommended for testing under water-saturated conditions to explore the potential storage of PAG waste rock via backfilling of the open pit.

- Geochemical modelling is recommended to develop source terms for water-saturated column leach tests that represent potential storage of Stockpile 001, and blended Stockpile 001 / Stockpile 002 PAG, by backfilling the open-pit. Geochemical modelling may also include potential in-pit water treatment. This modelling effort will support the evaluation of risks associated with underwater storage of PAG waste rock, help estimate the timescales over which such risks may manifest, and inform the design of appropriate mitigation measures.
- Due to the hybrid mine design with underground workings, backfill test samples from a 180-day cure period are expected to initiate in Fall 2025 for static and kinetic testing to investigate the metal leaching risks for long-term underground storage.

26.4.6 Mine Closure

As outlined in Section 20.8 of Chapter 20, a 1.5 m thick clay cover will be implemented for the restoration of Stockpile 002. This impermeable layer is designed to limit water infiltration and oxygen diffusion to the underlying materials (PAG and CND waste rock) to avoid the generation of contaminants. The recommendations for the feasibility of the cover system are:

- Potential Borrow pit Sources:
  - Investigations should be conducted to identify suitable clay materials from potential borrow pits for use in the construction of the cover system.
- Required Geotechnical Properties:
  - The technical feasibility of the clay cover depends on specific properties of the material. According to Benson et al. (1994) and Marcoen et al. (2000), as cited in Bussière and Guittonny (2020), the recommended geotechnical criteria for clay used as a hydraulic barrier include:
    - Hydraulic conductivity ≤ 10^{-9} m/s.
    - Fine particle content > 30%.
    - Clay fraction (< 2 µm) > 15%.
    - Plasticity index (PI) > 7%, but < 20% if freeze–thaw and wet–dry cycles are expected.
    - Liquid limit > 20%.
    - Gravel content < 50%.

- Numerical Validation:
    - It is strongly recommended to perform flow modeling using SEEP/W software to validate the hydraulic performance of the clay cover and integrated in the modal the projected climate change effect.
- Optimization of Waste Rock Management:
    - A review of the waste rock management should be done to evaluate the possibility of additional segregation to reduce the quantity of waste rock which requires an impermeable cover system (i.e. Reducing Stockpile 002).
- Laboratory and Field Testing:
    - Laboratory and field tests should be considered early in the mine's development to optimize and evaluate the performance of the proposed cover systems.

26.4.7 Geotechnical

Additional geotechnical investigations are required to support the next phase of work. The foundations for the stockpiles and site infrastructure must be further characterized. This work includes defining the depth to bedrock within the proposed diversion channel for Lake 001, and the collection ditches and ponds around Stockpile 001 and Stockpile 002. During the detailed design phase, additional geotechnical investigations along the critical sectional line used in the stability analysis will be necessary. This will help determine if there are any thick, weak layers beneath the stockpile footprints that could be at risk of liquefaction.

There is currently no soil profile or geotechnical information available at the location of the proposed Water Treatment Plant. For future design phases, it is recommended to conduct geotechnical boreholes within the footprint of the Water Treatment Plant.

The geotechnical investigations must include the proposed locations of the West Pit and East Pit Dams within Lake 001 to determine the thickness of the lake sediments.

Potential borrow pits must be identified for granular materials, and these should be sampled and tested.

A site-specific seismic hazard assessment should be completed to determine the appropriate loading for the seismic stability analyses.

26.4.8 Main Site Access Road

Identify non-PAG and non-metal leaching waste rock to be used for road improvement.

26.4.9 Power

Continue the negotiation with Hydro-Québec to secure the 42.5 MW peak power block and confirm the Tilly Hydro-Québec 315 kV modification required for Project connection. The permitting and authorizations to build the 120 kV powerline are critical in the timeline and develop partnership with local communities for its construction and maintenance.

26.4.10 Concentrate Transportation

26.4.10.1 Road

Implement a comprehensive bulk material testing program to characterize the concentrate's flowability and freezing behavior under operational conditions. The results will be used to optimize the selection and design of the bulk material handling systems and equipment.

Obtain the required annual special transportation permit from the Quebec Ministry of Transportation (MTQ) and the James Bay Development Corporation (SDBJ), authorizing the operation of 75-ton haul trucks for concentrate transport from the mine site to Matagami.

26.4.10.2 Matagami Concentrate Transloading Site

Conclude an agreement with the owner of the land for the Matagami Transshipment Centre site.

26.4.10.3 Concentrate Transportation Optimization

In addition to the established strategy for transporting spodumene concentrate, there are substantial opportunities to enhance logistics and decrease transport costs for the Project. A key initiative involves capitalizing on infrastructure advancements facilitated by the La Grande Alliance, a collaboration between the Cree Nation and the Government of Québec. Specifically:

- Stakeholder Engagement: During the next phases of the Project, it is recommended to collaborate with stakeholders from La Grande Alliance, encompassing First Nations and government entities. Goals include:
    - Synchronizing the Project timeline with the planned expansion of Highway 167, as detailed in the Alliance's Feasibility Study.
    - Evaluating the potential cost savings of transporting spodumene concentrate from the Shaakichiuwaanaan Project site to Chibougamau's CN rail infrastructure, for onward transport to the Port of Grande-Anse, QC.

26.4.10.4 Decarbonation

Usage of fossil fuel remains important for the OP mine fleet and building, and underground heating. Identification of efficient systems or alternative methods of heating, such as heat recovery, electrification, heat pumps and/or bi-energy and others can be investigated. Such a trade-off study could benefit from government financial incentives offering up to 75% coverage of the cost of the study.

26.5 Environmental Fieldwork and Studies

Continued environmental work is recommended to support the Project as it advances through economic and development studies.

Project development should continue to avoid the deposition of mine waste in fish habitat to avoid a listing on Schedule 2 of the Metal and Diamond Mining Effluent Regulations (SOR/2002-222).

Project optimization should aim to minimize encroachment on fish habitat, wetlands and hydrous environments (watercourses, lakes, shorelines, floodplains) to lower the need for habitat compensation and associated costs.

26.6 Detail Engineering

It is expected that the Detail Engineering by the integrator and several specialized firms for specific facilities, such as the paste plant, waste and water management and processing plant, will start prior to the full funding of the Project. The expected cost for the early engineering programs is estimated at $16.4M during 2027.

27.REFERENCES

AADNC. (2025). *First Nations Profiles: Cree Nation of Mistissini (Band No. 75).* Government of Canada.

AARQ. (2023). *Société d'histoire naturelle de la vallée du Saint-Laurent / Écomuseum.* Atlas des amphibiens et des reptiles du Québec. Retrieved from https://www.atlasamphibiensreptiles.qc.ca/

Air Tunilik. (2023, October 18). *Qu'est-ce que les vacances du Goose Break au Canada (2023).* . Retrieved from https://airtunilik.com/quest-ce-que-les-vacances-du-goose-break-au-canada/

Archéoconsultant Inc. (2024). *Étude de potentiel archéologique – Métaux de Batterie Patriot Inc. (Innova Lithium Inc.), Projet minier Shaakichiwaanaan – Corvette, Eeyou Istchee Baie-James, Nord-du-Québec, Québec. Préliminaire.* Saint-François-de-la-Rivière-du-Sud, Québec, Canada: Archéoconsultant Inc.

Archéoconsultant Inc. (2025). *Étude de potentiel archéologique – Métaux de Batteries Patriot Inc. (Innova Lithium Inc.), Projet minier Shaakichiwaanaan – (anciennement Corvette), Eeyou Istchee Baie-James, Nord-du-Québec, Québec. Préliminaire.* Saint-François-de-la-Rivière-du-Sud, Québec, Canada: Archéoconsultant Inc.

Archer, P., & Oswald, R. (2008b - GM63695). *Rapport géologique et recommandations, Travaux de Terrain 2006, Projet FCI, Québec, Félicie - Corvette Ouest - Island Lake.* Mines Virginia Inc.

AtkinsRéalis. (2025). *Geotechnical Laboratory Analysis of Tailings.* AtkinsRéalis.

Azimut Exploration. (2024 - GM 73760). *2023 prospecting campaign, JBN-57 Property.*

Bambic, P. (1997). *Corvet Quest Property Work Summary.* Minex Consulting Enr. Report, for Virginia Gold Mines Inc.

Bandyayera, D., Burniaux, P., & Chapon, B. (2013). Géologie de la région du lac Magin (33G09), de la colline Captel (33G15) et du lac Fontay (33G16), Baie-James, Québec. *Ministère des Ressources naturelles, Québec, RP 2013-01*, 32.

Bandyayera, D., Burniaux, P., & Morfin, S. (2011). Géologie de la région du lac Brune (33G07) et de la Baie Gavaudan (33G10). *Ministère des Ressources naturelles et de la Faune, Québec, RP 2011-01*, 25.

BBA. (2024a). *Geotechnical Campaign for Phase 1 Diversion Dams prepared for Innova Lithium.* Shaakichiuwaanaan Project: BBA.

BBA. (2024a). *Geotechnical Slope Design Basis prepared for Innova Lithium.*

BBA. (2024b). *Geotechnical Campaign (Phase 2) prepared for Innova Lithium. Factual Report of 2024 Geotechnical (Soil) Investigation.* Shaakichiuwaanaan Project: BBA.

BBA. (2024d). *PEA Geotechnical Slope Design Basis prepared for Innova Lithium.*

BBA. (2025). *Hydrogeological Characterisation – CV5 mining site, Shaakichiuwaanaan Lithium Property. Technical report submitted on July 4 2025. Document no.: 535019-000000-41-ERA-0002-R00.*

BBA. (2025). *Hydrogeological characterisation – Water supply well, Shaakichiuwaanaan Lithium Property. Technical report submitted on July 9, 2025. Document no.: 7535019-000000-41-ERA-0003-R00.*

BBA. (2025). *Packer Testing Factual Report – Shaakichiuwaanaan Project. Technical report submitted on Date of submission. Document no.: 7535019-000000-41-ERA-0001-R00.*

BBA. (2025a). *Packer test factual report.* BBA.

BBA/Vision Geochemistry. (2024a). *Analysis and Interpretation of the CV5 Static and Kinetic Geochemistry Program.* Prepared by BBA Inc. and Vision Geochemistry Ltd.

BBA/Vision Geochemistry. (2024b). *Geochemical Modelling of Arsenic Leaching Risk for CV5 Waste Rock Stockpiles.* Prepared by BBA Inc. and Vision Geochemistry Ltd.

BCMWRPRC. (1991b). *Mine Rock and Overburden Piles 2* (Vol. May 1991). (O. a.–I. Guidelines, Ed.) Klohn Leonoff Ltd.

Belem, T., Gélinas, L.-P., & Oke, J. (2022). *A new generalized solution for the required strength of the three types of cemented mine backfill.* Houston, TX, USA: American Rock Mechanics Association.

Benchmark Minerals. (2025). *Lithium Forecast Report, Q2 2025.*

Bird, D. (2023). *Lithium - Deficits Still on the Horizon But the Pace of New Supply is Picking Up, Lithium Commodity Market Report.* RFC Ambrian.

Bouazza, Z., Mahdi, A., & Fournier, A. (2024). *Technical Report for Patriot Battery Metals. Water Management Plan and Water Diversion. BBA Doc. No. 7535006-014000-41-ERA-0002-R00, dated September 11, 2024.*

Bradley, D., McCauley, A., & Stillings, L. (2017). *Mineral-deposit model for lithium-cesium-tantalum pegmatites.* US Geological Survey Scientific Investigations Report 2010-5070-O.

Card, K. (1986). Geology and Tectonics of the Archean Superior Province, Canadian Shield. In *Workshop on Early Crustal Genesis: The World's Oldest Rocks* (pp. 27-29). LPI Technical Report 86-04.

Carter, T. G. (2014). An update on the scaled span concept for dimensioning surface crown pillars in hard rock mines. In H. Petros (Ed.), *Proceedings of the 1st International Symposium on Surface Crown Pillars* (pp. 1-20). Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

CDA. (2013). *CDA Dam Safety Guidelines.* Ottawa, Canada: Canadian Dam Association.

CDA. (2019). *Technical Bulletin: Application of Dam Safety Guidelines to Mining Dams* . Canadian Dam Association.

CDPNQ. (2023, November 1). *Centre de données sur le patrimoine naturel du Québec*. Retrieved from Data on Species at risk. Interactive map.: https://services-mddelcc.maps.arcgis.com

Cerny, P., & Ercit, T. (2005). The Classification of Pegmatites revisited. *Canadian Mineralogist*, 2005-2026.

Champion Electric Metals. (2024a, July 16). *Champion Electric Confirms Spodumene-Bearing Pegmatite Dikes in Multiple Trenches at Quebec Lithium Project, James Bay Territory*. Retrieved from News: https://www.champem.com/news/champion-electric-confirms-spodumene-bearing-pegmatite-dikes-in-multiple-trenches-at-quebec-lithium-project-james-bay-territory

Champion Electric Metals Inc. (2024b, September 10). *Champion Electric Cuts Significant Widths of Lithium Pegmatite in Trenching at Quebec Lithium Project, James Bay Territory 10.5 metres at 0.96% Li2O and 8.5 metres at 1.17% Li2O*. Retrieved from News & Media / News: https://www.champem.com/news/champion-electric-cuts-significant-widths-of-lithium-pegmatite-in-trenching-at-quebec-lithium-project-james-bay-territory-10-5-metres-at-0-96-li2o-and-8-5-metres-at-1-17-li2o

Chisasibi Cree Nation. (2024).

Choinière, J., & Beaumier, M. (1997). *Bruits de fond géochimiques pour différents environnements géologiques au Québec.* Ministère des Ressources naturelles, Service des minéraux industriels et de l'assistance à l'exploration.

CNG. (2024). *Grand Council of the Crees (Eeyou Istchee) / Cree Nation Government.* Retrieved from https://www.cngov.ca/community-culture/communities/

CNG. (2025). *Wildlife Management and Conservation*. Retrieved from https://cngov.ca/en/environment/wildlife-management-and-conservation/

COMEV. (2025). *Committee on the Assessment of Environmental and Social Impact – Evaluating Committee (COMEV).* Gouvernement du Québec.

COMEX. (2022). *Demande de renseignements supplémentaires - Projet de mine de lithium Baie James par Galaxy Lithium (Canada) Inc. Dossier 3214-14-055.* COMEX, Québec.

Commission de toponymie Québec. (2016).

COSEWIC. (2023). *COSEWIC Annual Report 2023-2024.* Committee on the Status of Endangered Wildlife in Canada. Retrieved from https://www.canada.ca/en/environment-climate-change/services/species-risk-public-registry/cosewic-annual-reports/2023-2024.html

Cree Geoportal. (2023). *CTA Trapline Maps. Retrieved from: https://www.creegeoportal.ca/. Consulted on October 18, 2023.*

Cree Nation of Chisasibi. (2023). *Aéroport de Chisasibi – YKU. https://chisasibi.ca/departments-services/airport/ Consulted October 18, 2023.*

Cree Trappers Association. (2022). *Annual Report 2021-2022. 85 p.*

Cree Trappers' Association. (2021). *Ishtchiikun. https://creetrappers.ca/about/ishthchiikun. Consulted on October 18, 2023.*

Cree-Québec Forestry Board. (2018). *Lands, MUS and traplines. Retrieved from: http://www.ccqfcqfb.ca/fr/le-regime-forestier-adapte/le-territoire-dapplication/sites-et-territoires-dinteret-pour-les-cris/. Consulted on October 18, 2023.*

CRNTBJ. (2010). *Commission sur les ressources naturelles et le territoire de la Baie-James. Portrait faunique de la Baie-James, C09-07.*

D'Amours, I. (2011 - DP 2011-08). *Synthèse des levés magnétiques de la Baie-James.* Gouvernement du Québec.

de Chavigny, P. (1999 - GM 56161). *Reconnaissance Geologique, Permis JVT/Boreale (PEM 1308-1309-1310-1294), SNRC 33G, Baie James, Quebec, Canada.* Mines D'OR Virginia Inc/Exploration Boreale Inc.

Demers, M., & Blanchet, C. (2001). *Propriete Lac Erlandson-Ta Reconnaissance Geologique Aout 2001.* Mines d'Or Virginia.

EcoTec Consultants. (2024). *Retombées économiques de l'industrie minière au Québec en 2022.* Association minière du Québec & EcoTec Consultants. Retrieved from https://www.amq-inc.com/wp-content/uploads/2025/10/amq_rapport_retombeeseconomiques_2024_fr-compresse.pdf

Ekstrom, R. (1960 - GM10515). *Report of Exploration in the Corvette Lake - La Grande River Area, New Quebec, June-October, 1959.* Tyrone Mines Ltd (GM10515).

Environment Canada. (2023). *Récupéré sur Cadre écologique du Canada. Écorégions du Canada. Collines La Grande*. Retrieved from http://www.ecozones.ca/francais/region/72.html#:~:text=Cette%20%C3%A9cor%C3%A9gion%20s%27%C3%A9tend%20sur,%27environ%20%2D4%20%C2%B0C

EPC Chisasibi, 2017, cited in WSP, 2024. (n.d.).

Gaia Metals Corp. (2020, April 16). *Gaia Metals Corp. Identifies New Targets and Unlocks Potential at Golden Gap, Corvette-FCI Property, James Bay Region, Quebec*. Retrieved from Baystreet.ca: https://www.baystreet.ca/viewarticle.aspx?id=585238&msclkid=4b2b1ee4ae1b11eca84362ff6bf8e17f

GISTM. (2020). *GLOBAL INDUSTRY STANDARD ON TAILINGS MANAGEMENT.* International Council on Mining and Metals (ICMM), United Nations Environment Programme (UNEP) and Principles for Responsible Investment (PRI).

GNCC. (2023).

Goldak Airborne Surveys. (2009 - DP 2009-01). *Levé spectrométrique et magnétique aéroporté sur le territoire de la Baie-James – blocs NW and SW du secteur LG-4.* Gouvernement du Québec.

Goutier, J., & Dion, C. (2004). *Géologie et minéralisation de la Sous-province de La Grande, Baie-James, in Québec Exploration 2004.* Ministère des Ressources naturelles, de la Faune et des Parcs, Québec, DV 2004-06.

Goutier, J., Dion, C., Quellet, M.-C., David, D., David, J., & Parent, M. (2002). Géologie de la région du lac Guyer (33G/05, 33G/06 et 33G/11). . *Ministère des Ressources naturelles, Québec, RG 2001-15*, 53.

Goutier, J., Gigeon, J., Burniaux, P., Dion, C., Takam, F. T., Chartier-Montreuil, W., & Bandyayera, D. (2021 - RP 2020-01). *Géologue de la région du lac de la Corvette, Eeyou Istchee Baie-James.* Gouvernment du Québec.

Gouvernement du Québec - Ministère des Affaires municipales et de l'Habitation. (2025). *Carte administrative régionale – Région 10.* Retrieved from https://cdn-contenu.quebec.ca/cdn-contenu/adm/min/affaires-municipales/publications/cartes/region/10.pdf

Gouvernement du Québec. (2023a, November 1). *Gouvernment du Québec*. Retrieved from Zone de végétation et domaines bioclimatiques du Québec: https://mffp.gouv.qc.ca/documents/forets/FE_zones_vegetation_bioclimatiques_MRNF.pdf

Gouvernement du Québec. (2023d, November 1). Retrieved from Espèces floristiques menacées ou vulnérables. Outil Potentiel. Base de données MIcrosoft Access.: https://www.environnement.gouv.qc.ca/biodiversite/especes-designees-susceptibles/

Gouvernement du Québec. (2023c, November 1). *Gouvernement du Québec*. Retrieved from Liste des espèces désignées menacées ou vulnérables au Québec (LEMVQ).: https://www.quebec.ca/agriculture-environnement-et-ressources-naturelles/faune/gestion-faunehabitats-fauniques/especes-fauniques-menacees-vulnerables/liste

Government of Canada. (2022, 04 12). *La Grande Rivière Airport, Quebec*. Retrieved from https://weather.gc.ca/:

https://weather.gc.ca/past_conditions/index_e.html?station=ygl&msclkid=111e1565ba8e11ec8720853f5e5ba60f

Government of Québec. (2022, April 6). *Société du Plan Nord*. Retrieved from https://www.quebec.ca/gouvernement/ministeres-et-organismes/societe-plan-nord

Grand Québec. (2025). *Radisson.* Retrieved from https://grandquebec.com/nord-du-quebec/radisson/

GREIBJ. (2020). Retrieved from Gouvernement régional d'Eeyou Istchee Baie-James: https://greibj-eijbrg.ca

GREIBJ. (2025a). *Gouvernement régional d'Eeyou Istchee Baie-James*. Retrieved from https://greibj-eijbrg.ca/accueil

Hawley, M., & Cunning, J. (2017). *Guidelines for Mine Waste Dump and Stockpile Design.* CSIRO Publishing.

Henning, J., Gonzalaz, C., & Hamediazad, F. (2024). *Geotechnical Investigations prepared for Innova Lithium. Factual Report of 2024 Geotechnical Diamond Drill Investigation, Corvette CV5 Site. BBA Doc. No. 7535017-000000-4M-ERA-0001-001, July 22, 2024, 67 pages with appendices.*

Hopky, D. W. (1998). *Guidelines for the Use of Explosives in or Near Canadian Fisheries Waters.* Canadian Technical Report of Fisheries and Aquatic Sciences 2107: iv + 34 p.

Houle, P. (2004). James Bay Region Central Superior Province (Opatica, Opinaca, Nemiscau, and La Grande Subprovinces). In *Report on Mineral Exploration Activities in Quebec 2004* (pp. 9-16). MERN.

Hydro-Québec. (2022, 03 24). *Hydro-Québec Production*. Retrieved from Hydro-Québec: https://www.hydroquebec.com/generation/generating-stations.html?msclkid=355698ccabbf11ecb0bd2805430e7368

Institut de la Statistique du Québec. (2024b).

Investissement Québec. (2023, August 16). *Why Quebec?, Power Generating Capacity*. Retrieved from https://www.investquebec.com/international/en/why-quebec/hydroelectricity.html

Johnson, M. (1996 - GM56869). *Summary report, Sakami Project, Lac Guyer and Lac de la Corvette areas, Baie-James region.* Phelps Dodge Corporation of Canada.

Knox, A. (2022). *NI 43-101 Technical Report on the Corvette Property, Quebec, Canada.* Patriot Battery Metals Inc.

Lecavalier, C. (2017). *Des centaines de camps abandonnés dans le Nord.* Le Journal de Montréal. Retrieved from https://www.journaldemontreal.com/2017/03/05/des-centaines-de-camps-abandonnes-dans-le-nord

LGA. (2022a, 03 24). Retrieved from La Grande Alliance: https://www.lagrandealliance.quebec/en/

LGA. (2022b). *Infrastructure Program Feasibility Studies, Interim Report.* Cree Development Corporation.

LGA. (2023a, October). Retrieved from Études de faisabilité du Programme d'Infrastructures. Octobre 2023.: https://www.lagrandealliance.quebec/wp-content/uploads/2023/10/LGA-Executive-Summary-2023-FINAL-FR_compressed.pdf

LGA. (2023b). *La Grande Alliance. Études de faisabilité du programme d'infrastructures. Executive summary. 38 p.*

Li, L., & Aubertin, M. (2012). A modified solution to assess the required strength of exposed backfill in mine stopes. *Canadian Geotechnical Journal, 49*(8), 994–1002. doi:10.1139/t2012-061

Li, T., Ducruc, J.-P., Côté, M.-J., Bellavance, D., & Poisson, F. (2019). *Les provinces naturelles : première fenêtre sur l'écologie du Québec.* Québec: Ministère de l'Environnement et de la Lutte contre les changements climatiques, Direction de la connaissance écologique.

Localité de Radisson. (2024). Retrieved from L'histoire. Localité de Radisson.: https://www.localiteradisson.com/fr/localite-de-radisson/l-histoire/

Lucas, S., & St-Onge, M. (1998). Geology of the Precambiran Superior and Grenville Provinces and Precambrian Fossils in North AMerica. In *Geology of Canada, No. 7, Volume C-1* (Vols. C-1, p. 394). Geological Survey of Canada.

Makivvik. (2025). Retrieved from https://www.makivvik.ca/

Matagami. (2024). *Ville de Matagami - Portrait économique.*

Matagami. (2025). *Ville de Matagami - Portrait économique.*

McCracken, T. and Cunningham, R. (2023). *NI 43-101 Technical Report for Patriot Battery Metals. Mineral Resource Estimate for the CV5 Pegmatite, Corvette Property. September 8, 2023. 284 pages.*

McCracken, T., & Cunningham, R. (2023). *NI 43-101 Technical Report for Patriot Battery Metals. Mineral Resource Estimate for the CV5 Pegmatite, Corvette Property.*

McCracken, T., & Cunningham, R. (2023). *NI 43-101 Technical Report for Patriot Battery Metals. Mineral Resource Estimate for the CV5 Pegmatite, Corvette Property. September 8, 2023. 284 pages.*

McCracken, T., & Cunningham, R. (2025). *NI 43-101 Technical Report, Mineral Resource Estimate for the Shaakichiuwaanaan Project, James Bay Region, Quebec, Canada.* Patriot Battery Metals Inc.

McCracken, T., Latulippe, H., Ghouralal, S., Piciacchia, L., Cunningham, R., & and Fortin, N. (2024). *NI 43 101 Preliminary Economic Assessment for the Shaakichiuwaanaan Project, James Bay Region, Québec, Canada. Prepared for Patriot Battery Metals Inc. September 12, 2024 (effective date of August 21, 2024).* Patriot Battery Metals Inc.

MDDEP. (2012). *Directive 019 on the mining industry.* Gouvernement du Québec. Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP).

MDDEP. (2025). *Directive 019 on the mining industry.*

MELCCFP. (2023). *Récupéré sur Aires protégées au Québec. Les provinces naturelles. Niveau I du cadre écologique de référence du Québec*. Retrieved from Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs: https://www.mamh.gouv.qc.ca/fileadmin/publications/organisation_municipale/cartotheque/Region_10.pdf

MELCCFP. (2025). *Directive 019 sur l'industrie minière.* Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs. Gouvernement du Québec.

MELCCFP. (2025). *Quebec Air Quality Standards and Criteria [NCQQA v9].*

MELCCFP, 2023; previously MELCC, 2020. (n.d.).

MERN. (2019). *Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec. 51 p. and appendices.* Ministère de l'Énergie et des Ressources Naturelles.

MERN. (2022). *Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec.*

Mitchell, R. J., Olsen, R. S., & Smith, J. D. (1982). Model studies on cemented tailings used in mine backfill. *Canadian Geotechnical Journal, 19*, 14-28. doi:10.1139/t82-002

MRNF. (2023). *Ministère des Ressources naturelles et des Fôrets.* Québec: GESTIM.

MRNF. (2023). *Personal communication, in WSP*.

MRNF. (2023, August 14). *Québec Ministère des Ressources naturelles et des Forêts*. Retrieved from GESTIM des titres miniers: https://gestim.mines.gouv.qc.ca

MRNF. (2024). *Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec.* Ministère des Ressources naturelles et des Forêts. Gouvernement du Québec.

MRNF. (2024). *Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec* . Québec: Ministère des Ressources naturelles et des Forêts.

NBCC. (2020). *National Building Code of Canada Seismic Hazard Tool.* Government of Canada. Retrieved from https://earthquakescanada.nrcan.gc.ca/hazard-alea/interpolat/index-en.php

NIIGAAN. (2022). *Technical Note. Environmental, social, and hydrological surveys.*

Niigaan. (2025, February 25). *Aquatic Environment Baseline Conditions for the Shaakichiwaanaan Lithium Project – Surface Water and Sediment Quality.* (P. B. Inc., Ed.)

Niigaan. (2025). *Avifaune et herpétofaune – Étude de référence – Projet minier Shaakichiuwaanaan (Corvette). Rapport du projet 151022002.*

Niigaan. (2025). *État de référence de l'environnement aquatique – Caractérisation aquatique – Campagne terrain 2022-2024. Rapport du projet 22-0095.*

OSECC. (2012, 2022, 2023). Office de la sécurité économique des chasseurs cris (OSECC). *Rapport annuel*.

Patriot. (2024a, August 5). *Patriot Battery Metals Inc. Significant Mineral Resource Upgrade at Shaakichiuwaanaan Lithium Project to Underpin Impending PEA*. Retrieved from Patriot Battery Metals / News: https://patriotbatterymetals.com/significant-mineral-resource-upgrade-at-shaakichiuwaanaan-lithium-project-to-underpin-impending-pea/

Patriot. (2025a, July 20). *World's Largest Pollucite-Hosted Caesium Pegmatite Mineral Resource Defined at Shaakichiuwaanaan*. Retrieved from Patriot Battery Metals / News: https://patriotbatterymetals.com/worlds-largest-pollucite-hosted-caesium-pegmatite-mineral-resource-defined-at-shaakichiuwaanaan/

Percival, J. (2007). Geology and Metallogeny of the Superior Province, Canada. In W. Goodfellow, *Mineral Deposits of Canada. A synthesis of major deposit types, district metallogeny, the evolution of geological provinces and exploration methods* (pp. 903-928). Special Publication 5.

Percival, J., Skulski, T., Sanborn-Barrie, M., Stott, G. L., Corkery, M., & Boily, M. (2012). Chapter 6 - Geology and Tectonic Evolution of the Superior Province, Canada. In *Tectonic Styles in Canada: The Lithoprobe Perspective* (pp. 312-378). Geological Association of Canada, Special Paper 49.

PMET. (2023a, July 30). *Patriot Battery Metals Inc. Announces the Largest Lithium Pegmatite Resource in the Americas at CV5, Corvette Property, Quebec, Canada*. Retrieved from News: https://patriotbatterymetals.com/patriot-announces-the-largest-lithium-pegmatite-resource-in-the-americas-at-cv5-corvette-property-quebec-canada/

PMET. (2023b, October 18). *Patriot Battery Metals Inc. Discovers New High-Grade Zone at CV13 Spodumene Pegmatite, Corvette Property, Quebec, Canada*. Retrieved from News: https://patriotbatterymetals.com/patriot-discovers-new-high-grade-zone-at-the-cv13-spodumene-pegmatite-corvette-property-quebec-canada/

PMET Resources. (2024b, August 5). *Patriot Battery Metals Inc. Exploration Target for the Shaakichiuwaanaan Lithium Project Outlines District Scale Opportunity, Quebec, Canada*. Retrieved from News: https://patriotbatterymetals.com/exploration-target-for-the-shaakichiuwaanaan-lithium-project-outlines-district-scale-opportunity-quebec-canada/

PMET Resources. (2025b, May 12). *Significant Mineral Resource Upgrade at Shaakichiuwaanaan Lithium Project to Underpin Impending Feasibility Study*. Retrieved from Patriot Battery Metals / News: https://patriotbatterymetals.com/significant-mineral-resource-upgrade-at-shaakichiuwaanaan-lithium-project-to-underpin-impending-feasibility-study-representing-an-increase-of-30-and-306-in-indicated-resources-at-cv5-and-cv13-resp/

Pride, C. (1978 - GM34044). *Lake Sediment Geochemistry, 1974.* MRN.

Primero. (2024). *Corvette PEA – Process Design Criteria (45705-DCR-PR-001-REVE). February 2024.*

Québec, Gouvernement du. (2022b). *Maps of fur-bearing animal management units ("UGAF" or unités de gestion des animaux à fourrure) numbers 91 and 94. Retrieved from: https://www.quebec.ca/en/tourism-and-recreation/sporting-and-outdoor-activities/trapping/maps-ugaf. Consulted on October 18.*

Québec, Gouvernement du. (2022c). *Hunting zone maps. Retrieved from: https://www.quebec.ca/en/tourism-and-recreation/sporting-and-outdoor-activities/sport-hunting/hunting-zone-maps. Consulted on October 18, 2023.*

Quellette, J.-F., & Vachon, D. (2014 - GM68359). *Technical Report and Recommendations, 2013 Exploration Program, FCI Project, Quebec.* Komet Resources Inc, Virginia Mines Inc.

RCAANC. (2025). *Profil de la Nation crie de Chisasibi.* Gouvernement du Canada. Relations Couronne-Autochtones et Affaires du Nord Canada (RCAANC).

Romain, D., & Larivière, J.-F. (2021 - GM 72626). *Report on the 2021 Exploration Campaigns on the Mythril Regional Project: Prospecting.* Midland Exploration, MRN.

Roscoe, S., & Donaldson, J. (1988). Uraniferous pyrite quartz pebble conglomerate and layered ultramafic intrusion in a sequence of quartzite, carbonate, iron formation and basalt of probable Archean age at Lac Sakami, Quebec. In *Recent Research, Part C, Paper 88-1C* (pp. 117-121). Geological Survey of Canada.

Roy, I., & Archer, P. (2010 - GM65536). *Technical Report and Recommendations, Fall 2009 Exploration Program, FCI Property, Quebec.* Virginia Mines Inc., Odyssey Resources Ltd.

Schmidt, P. T., Samson, I. M., William-Jones, A. E., & Smith, D. L. (2017). HFSE evolution of the Eldor Carbonatite. *SGA 2017 confernce abstract*.

Séguin, A., Gagnon, E., Lepage, C., & Thomassin, Y. (2022). *Initial Environmental and Social Scoping Report - Preliminary Economic Assessment – Socio-environmental Requirements.* James-Bay Region, Northern Quebec: BBA.

SGS. (2022, 04 01). *Mining Geochemistry*. Retrieved from SGS Canada: https://www.sgs.ca/en/mining/analytical-services/geochemistry

SGS. (2023a). *The Mineralogical Characterization of Twenty-Two Pegmatite Samples from the Corvette-FCI Property.* Project 19005-01. James-Bay, Québec: SGS Canada Inc.

SGS. (2023b). *An Investigation into Scoping Level Study Metallurgical Testwork on Samples from the Corvette-FCI Property*. Project 19005-02. James-Bay, Québec: SGS Canada Inc.

SGS. (2023c). *An Investigation into Variability HLS Testwork on Samples from the Corvette Property.* Project 19005-04. James-Bay, Québec: SGS Canada Inc.

SGS. (2023d). *An Investigation into the HLS Testwork on Five Variability Samples from the Corvette-FCI Property.* Project 19005-06. James Bay, Québec: SGS Canada Inc.

SGS. (2025). *DMS concentrate hydrometallurgical program for the Zeppelin master drill core composite (CV5 Pegmatite): Flowsheet development for lithium hydroxide conversion.* SGS Canada Inc.

Smith, D. L. (2018 - GM70744). *Simplified Exploration Work Report - Corvette Property, GM70744.* 92 Resources Corp.

Smith, D. L. (2019 - GM71513). *Simplified exploration work report, FCI West Property.* O3 Mining Inc., MERN.

Smith, D. L. (2019). *2018 Exploration of the Corvette Property, Quebec.* 92 Resources Corp., MERN.

Smith, D. L. (2020 - GM71564). *2019 Exploration of the FCI property, Quebec.* Gaia Metals Corp., MERN.

Smith, D. L. (2021 - GM72176). *2019 Exploration of the Corvette Property, Quebec.* Gaia Metals Corp, MERN.

Smith, D. L., Delporte, R., & Mickelsen, P. (2025 - GM74460). *The 2023 Exploration of the Shaakichiuwaanaan Property, Quebec.* Patriot Battery Metals Inc.

Smith, D. L., Mickelson, P., & Blu, F. (2023 - GM73402). *2021 Exploration of the Corvette Property.* Patriot Battery Metals Inc., MRNF (pending).

Smith, D. L., Schmidt, P., Mickelsen, P., & Ullrich, A. (2024 - GM73931). *2022 Exploration of the Corvette Property.* Patriot Battery Metals Inc.

Société de développement de la Baie-James. (2009). *Retrieved from: https://www.sdbj.gouv.qc.ca/fr/aeroport/a-propos/. Consulted on October 18, 2023.*

Statistics Canada. (2022). Retrieved from Indigenous Population Profile, 2021 Census of Population.: https://www12.statcan.gc.ca/census-recensement/2021/dp-pd/ipp-ppa/index-eng.cfm

Statistics Canada. (2023a). **Cree Nation of Chisasibi [Region of First Nation or Indian band or Tribal Council area], Québec (table).* Ottawa: Indigenous Population Profile, 2021 Census of Population.

St-Seymour, K., & Francis, D. (1988). Magmatic interaction between mantle and crust during the evolution of the Archean Lac Guyer greenstone belt. *Canadian Journal of Earth Sciences, 25*, 691-700.

Tourisme Eeyou Istchee Baie-James. (2016). *Maps and itineraries. Retrieved from: https://www.escapelikeneverbefore.com/en/discover-the-region/maps-and-routes/. Consulted on October 18, 2023.*

VEI-WSP. (2024). Technical Note 3 – Land Use (Final Version). *Cree Development Corporation*.

Vision Geochemistry. (2025). *Geochemical Characterization of Mine Waste Materials & Modelling of Waste Rock Stockpiles for the Shaakichiuwaanaan Project (CV5 Pegmatite): Feasibility Study Update.* Vision Geochemistry.

Weather Spark. (2020, November 26). Retrieved from Weather Spark: https://weatherspark.com/y/147066/Average-Weather-at-La-Grande-Rivi%C3%A8re-Airport-Canada-Year-Round

WSP. (2025). *Projet minier Shaakichiuwaanaan. Rapport sectoriel – Ambiance sonore et vibrations. Eeyou Istchee Baie-James. Rapport produit pour Métaux de batteries Patriot inc. Référence WSP : CA0001724.3318. 27 pages et annexes.* .

WSP. (2025). *Projet minier Shaakichiuwaanaan. Rapport sectoriel – Chiroptères. Eeyou Istchee Baie-James. Rapport produit pour Métaux de Batteries Patriot Inc. Référence WSP : CA0001724.3318_006_R_RevA.*

WSP Canada Inc. (2025). *Métaux de Batteries Patriot Inc., Projet minier Shaakichiwaanaan – Rapport sectoriel: Hydrologie. Version préliminaire, confidentiel. Référence WSP: CA0001724.3318.* Montréal, Québec, Canada: WSP Canada Inc.

WSP Canada Inc. (2025). *Métaux de Batteries Patriot Inc., Projet minier Shaakichiwaanaan – Rapport sectoriel: Végétation et milieux humides. Réf. WSP: CA0001724-3318_005_R_REVA_SC_VEGMHH.* Sherbrooke, Québec, Canada.