PN3.3 (1 of 2) Environmental Impact Assessment - Volume 3

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan Mining Project

SHAAKICHIUWAANAAN

Environmental Impact Assessment



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APPENDIX

4-1 TOS 2 – WET PLANT FLOWSHEET – DMS VS HYBRID

DESIGN
CONSTRUCT
OPERATE

PRIMERO

TOS 2 - WET PLANT FLOWSHEET DMS VS HYBRID

SHAAKICHIUWAANAAN LITHIUM PROJECT FEASIBILITY STUDY

CASN-A 620-GE-MEM-0200

PATRIOT BATTERY METALS INC.

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1 SUMMARY

This memo evaluates the benefits of integrating a flotation circuit alongside the Dense Medium Separation (DMS) process to enhance spodumene recovery. Patriot Battery Metals (PBM) engaged Primero to assess the technical and economic trade-offs of this hybrid approach. Implementing a flotation plant will also necessitate additional permitting requirements and the construction of a water treatment plant to manage process water and ensure environmental compliance.

Key Findings:

- The DMS plant design is based on maximizing recovery using DMS only and includes separate three size class DMS and a recrush DMS.
- Two hybrids plant options are considered,
    - a smaller plant treating the fines fraction that bypasses the DMS, and
    - a larger plant treating the fines and the middlings material from the fine, ultrafines and recrush DMS

HYBRID OPTION COMPARISON	OPTION 1 DMS ONLY	OPTION 2 HYBRID 1	OPTION 3 HYBRID 2
Overall Recovery (%)	66.0	70.7	75.4
Additional concentrate (t)	-	31,074	62,093
CAPEX (additional)	-	$m 139.8	$m 247
OPEX (additional)	-	$m 14.6	$m 22.1
Pre-Tax NPV (8%)	-	-$m 38.0	$m 24.9
Pre-Tax IRR	-	4%	9%
Payback	-	13.3 years	9 years

- The overall recovery is calculated based on testwork values and interpolated for the stream contents, using a conservative value for the DMS base case for comparative and sizing purpose.
- Option 3 recovery aligns with best in class for operational hybrid plants.
- The capital costs include the costs associated with required water treatment.
- Note that the addition of a flotation circuit and a water treatment plant will require additional environmental and operating permits impacting the schedule and complexity to the project development
- The milling duty for the paste plant will be reduced by the inclusion of a flotation circuit, was not factored in the OPEX.
- The analysis indicates marginal increases in recovery for the two options for a relatively large increase in CAPEX.
- The flotation plants are scheduled to be implemented in year 4.
- An additional option (option 4) was considered, which evaluated the impact of a complete scope change for the overall project, reducing the DMS plant complexity and recovery and increasing the flotation plant scope to produce a hybrid plant with the same ultimate recovery. This was a high-level evaluation of the economic impact of lower starting capex and recovery. The lower revenue was not offset by the initial reduced capital cost, and the overall economics is estimated to rank between option 2 and option 3.
- The economics do not favour proceeding with a hybrid plant in any form for the first phase of this project. Largely because of the cost of adding a flotation plant, the current high recovery of the DMS plant and relatively small increase in recovery from a float plant and the delays of being able to implement of float plant.

2 INTRODUCTION

Patriot Battery Metals (PBM) is conducting a Feasibility Study (FS) for a Dense Medium Separation (DMS) lithium concentrator at the Shaakichiuwaanaan property. The study evaluates two identical parallel process trains, each with a capacity of 2.5 million tonnes per annum (Mtpa). To enhance spodumene recovery and improve economic outcomes, PBM has commissioned Primero to assess the impact of integrating a flotation circuit into the existing flowsheet.

This trade-off study specifically examines the benefits of adding flotation to process streams containing lithium bearing materials from the DMS process. These materials include the DMS bypass material (<650 µm) and the middlings streams from the recrush DMS, the fines DMS and the ultrafines DMS. The study looks at two options, one treating only the bypass fraction, and the second treating the bypass and the middlings streams.

The flotation circuit involves grinding the bypass material for option 2 and grinding bypass and middlings fraction for option 3) to a P80 of 200 µm , followed by magnetic separation and spodumene flotation.

The goal is to evaluate the improved recovery from each flotation option, describe the impacts to the project and comment on the economic viability.

The assessment focuses on a single 2.5 Mtpa train and considers a phased implementation strategy , aligning flotation integration with expected production growth three years after the plant's initial operation. Key parameters analysed include capital and operating costs, production impact, and economic feasibility over the life of the mine.

The study does not assess the risk to recovery that variability in feed size distribution poses, where changes in blasting practices, or crushing circuit operation could increase fines generation above those predicted by the testwork. A hybrid plant provides a mitigation against increase fines in the feed material.

3 SCOPE

The scope of the economic assessment includes the following:

- Flotation Testwork
- Design Basis
- Preliminary mechanical equipment list
- Preliminary major equipment sizing
    - Vendor or database quotation for the major equipment
    - Factored Capital Expenditures (CAPEX)
- Operational Expense (OPEX)
- Financial Analysis

This assessment is limited to the addition of the flotation plant and plant support services and does not include local taxes, import duties, or currency fluctuations. The financial analysis relies on current 2025 price projections and cost estimates.

4 FLOTATION TESTWORK

In 2024, four batch flotation tests were conducted on the combined DMS middlings and -0.85 mm fines (DMS bypass) to optimize spodumene recovery. The results of the flotation testwork are displayed in Figure 4-1. The points from right to left represent the rougher and scavenger concentrate, rougher concentrate, 1st cleaner concentrate, 2nd cleaner concentrate, and non-magnetic 2nd cleaner concentrate

Below are key findings from the testwork:

- It was found that over 90% of spodumene grains were liberated at P80 of 214 µm.
- Mica flotation resulted in lithium losses of 14.9% of the lithium distribution, but removal of rougher mica scavenger stage reduced lithium losses to mica concentrate to 10%.
- It is apparent from Figure 4-1 that the best results were from test F2 and forms the basis for process flowsheet design.
    - 2nd Cleaner concentrate Li2O grade was 5.23% with a lithium distribution of 60.9%
    - Final concentrate Li2O grade after magnetic scalping was 5.49% with a lithium distribution of 57.5%

Please refer to the technical report 19005-02 - Final Report that was written by SGS for further detail.

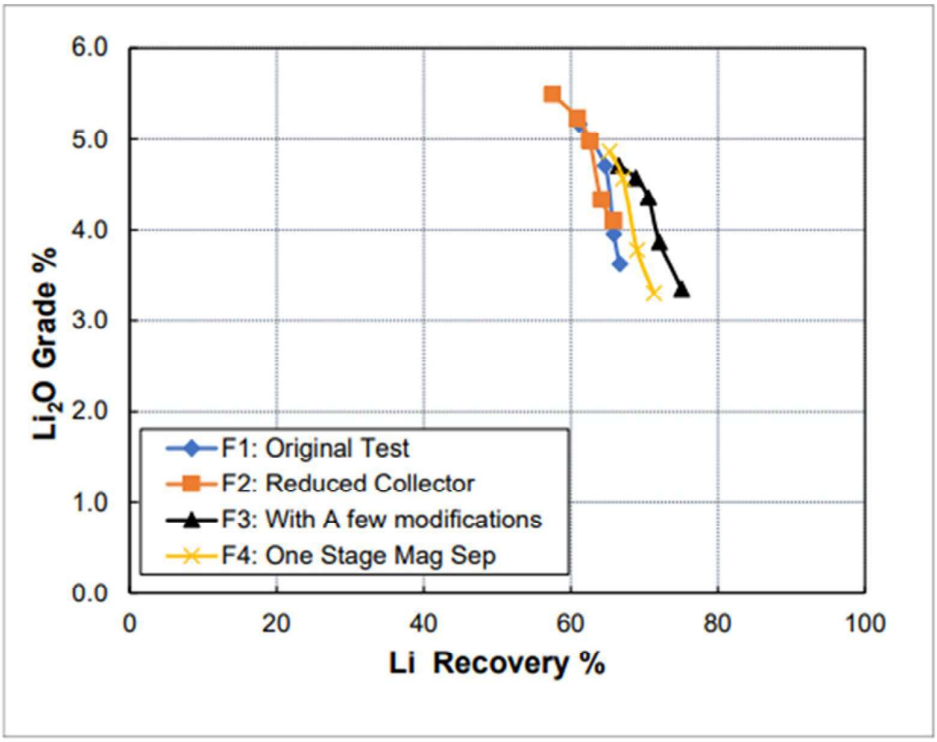


Figure 4-1: 2024 Spodumene Flotation Testwork Results

5 TRADE-OFF OPTIONS

The three options that are available for Shaakichiuwaanaan lithia recovery are:

- Option 1 – DMS Plant (Base option): This option involves using Dense Medium Separation (DMS) to recover spodumene. This is described in detail in the feasibility study.

- Option 2 – Hybrid Plant (DMS and Flotation of DMS bypass): This will focus on discussing the impact of adding flotation plant to treat the DMS bypass only. This option increases lithium recovery by processing DMS bypass material through flotation, potentially improving the project's economic viability but requiring higher capital investment.
- Option 3 – Hybrid Plant (DMS and Flotation DMS bypass with middlings combined): This will focus on discussing the impact of adding flotation plant to treat the DMS bypass along with the middlings from the Recrush DMS, Fine DMS, and Ultrafine DMS modules to improve recovery further. This option aims to further strengthening the economic viability of the flotation plant with additional capital investment compared to Option 2.

5.1 INDUSTRY BENCHMARK

Figure 5-1 displays the correlation between lithium distribution in the DMS bypass and the weight fraction of the DMS bypass relative to DMS plant feed for Shaakichiuwaanaan along with 9 other spodumene projects. Lithium extraction process that was adopted for each project is also illustrated in Figure 5-1 alongside the correlation equation. A generally accepted rule is that as the slope approaches unity, the most effective lithia extraction becomes either flotation only, or hybrid (DMS and flotation). The slope value of 0.6988 for Shaakichiuwaanaan suggests that a DMS-only extraction could be efficient for the deposit . Projects favouring a DMS only flowsheet typically have coarse grained spodumene and well liberated and thus have high recovery using DMS only. However, it's important to note that industry benchmarks are based on historical data and may not fully capture the unique characteristics of the Shaakichiuwaanaan deposit.

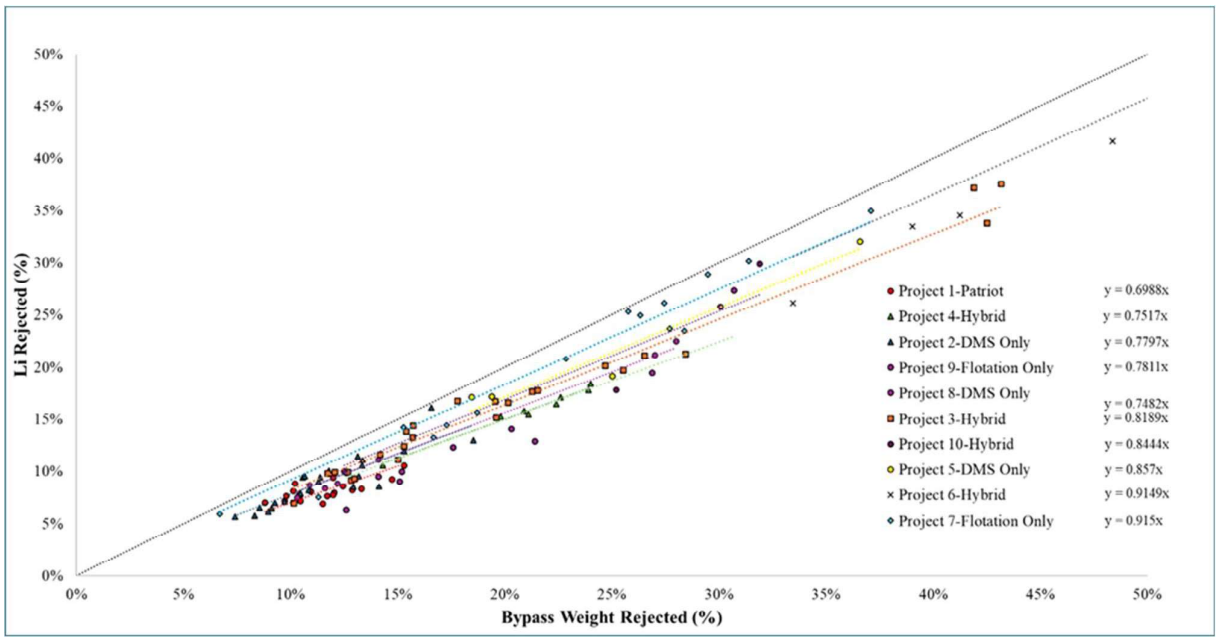


Figure 5-1: Benchmarking of 9 Spodumene Recovery Projects versus Shaakichiuwaanaan

5.2 OPTION 1 – SPODUMENE RECOVERY BY DMS PLANT

This option is currently the focus of the Feasibility study and will not be addressed in this memorandum. The simplified flowsheet of spodumene recovery by DMS only is shown in Figure 5-2 for reference. The proposed flowsheet will have the DMS plant feed split into 3 size fractions: Primary Coarse DMS (-9.5mm, 3.3mm),

Primary Fines DMS (-3.3mm, 1.5mm), and Primary Ultrafine DMS (-1.5mm, 0.65mm). Treating the material into narrow size range will help get the real plant performance close to the DMS pilot plant performance. The Primary DMS will be operated at a conservative cut-point of 2.65 to get rid of most of the gangue material, and the sinks from the Primary DMS will be processed in the Secondary Coarse DMS at a higher cut point of 2.65 to further upgrade the material. The float material from the Secondary Coarse DMS (typically called Middlings) will be recrushed to improve liberation and processed in the Recrush DMS module. The Primary fines and Ultrafine DMS will be processed at one DMS pass with cut point of 2.85. In this flowsheet, the DMS bypass (-0.65mm) is discarded to tailings or sent to the future paste plant along with the DMS float from the four DMS modules discussed above. One potential challenge of this approach is the loss of lithium in the DMS bypass material , which is why the flotation option is being considered.

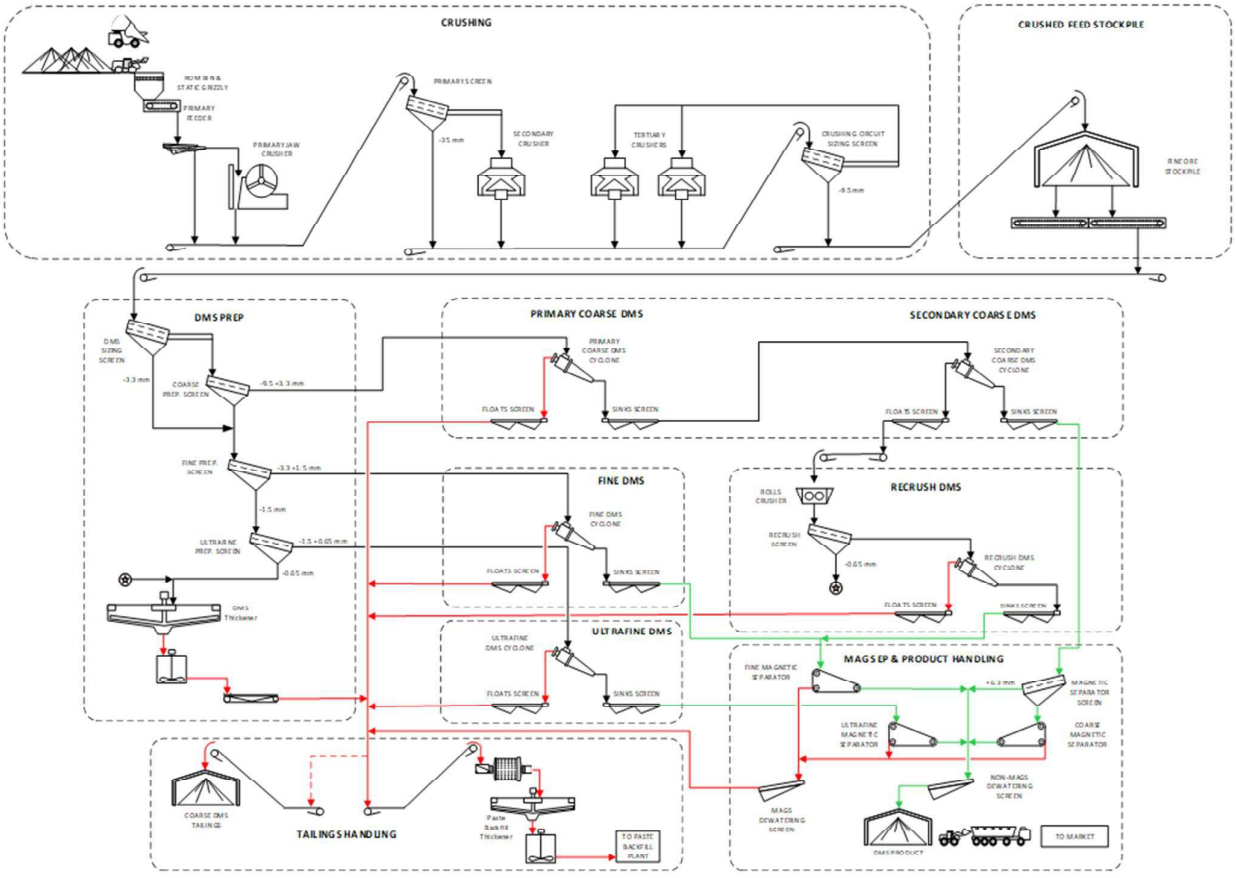


Figure 5-2: Simplified flowsheet of Shaakichiuwaanaan Lithium Project with DMS

5.3 OPTION 2 – HYBRID PLANT TO THE TREAT DMS BYPASS IN FLOTATION

A simplified flowsheet of the processing plant that include spodumene flotation can be seen in Figure 5-3. In this flowsheet, the DMS bypass (-0.65mm) is further beneficiated to improve spodumene liberation ahead of flotation. The objective of integrating flotation into the DMS process is to maximize lithium recovery by processing the DMS bypass material, which contains valuable lithium that would otherwise be lost to tailings. The flotation process involves grinding the DMS bypass material to a P80 of 200 microns, followed by magnetic scalping, desliming and mica flotation to reduce gangue entrainment to spodumene concentrate. Environmental considerations include water usage, reagent management, and water treatment, that are not shown on this flowsheet but are included in the economic analysis of this trade-off.

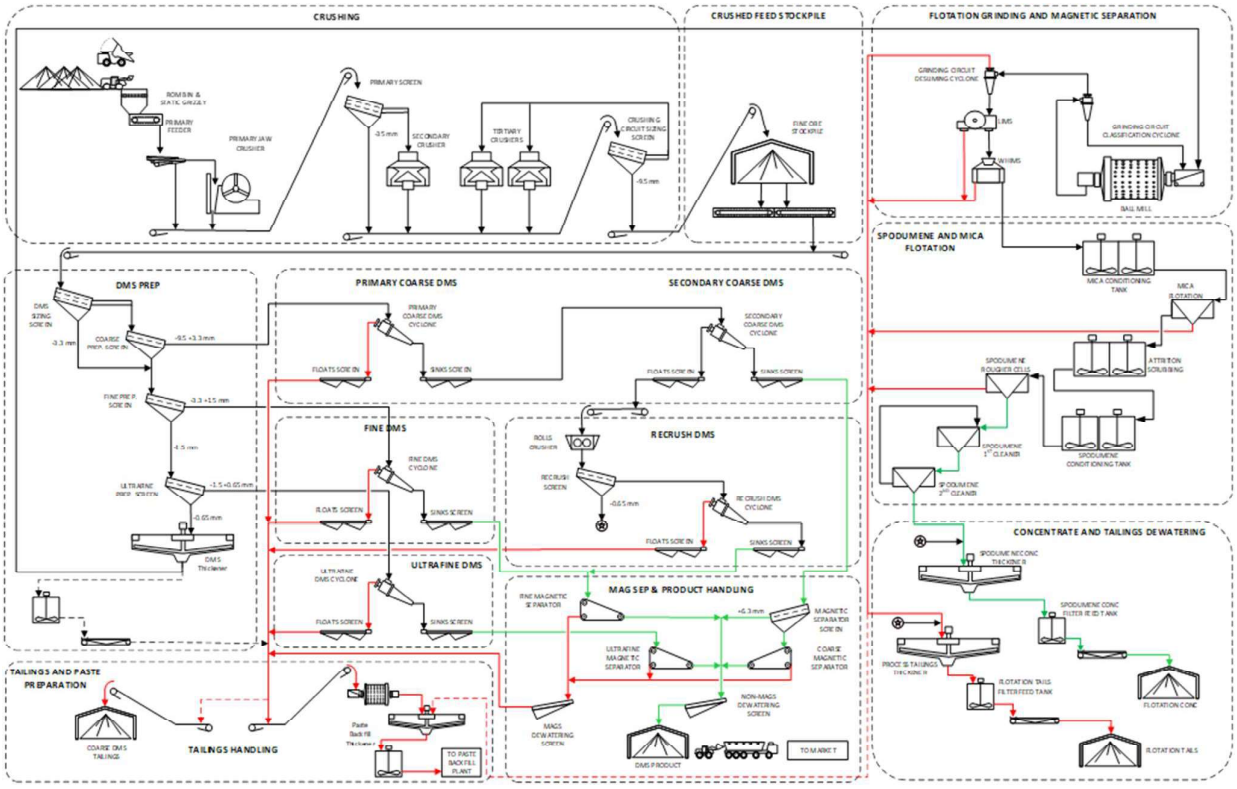


Figure 5-3: Simplified flowsheet of Shaakichiuwaanaan Lithium Project with DMS and Flotation

A high-level block flow diagram of DMS and flotation to treat DMS bypass is displayed in Figure 5-4. Flotation testwork results discussed previously were used to model the proposed flotation plant with application of some factor for scalability and assumption due to the limitation of the flotation testwork.

- A factor of 0.95 was applied to the recovery and grade found in the lab testwork to account for potential scale-up losses and operational inefficiencies in a commercial plant.
- Recovery model for the proposed flowsheet used 33% mass fraction for slimes removal compared to 10% only in the lab bench test. The 33% mass fraction removal was based on sizing distribution of the DMS bypass from the DMS Plant model that had a high fraction of -20 µm.
- Bench test mass pull for mica and magnetic scalping ahead of spodumene flotation was used.
- Magnetic separation of flotation concentrate is not considered in the flowsheet because the flotation feed undergoes magnetic separation and is considered sufficient. Additionally, the mass of flotation concentrates is relatively low (~1/10th of DMS concentrate) and thus has little impact on overall blended concentrate grade. These assumptions and factors introduce some uncertainty into the modelling process, but they are based on industry best practices and available data. A summary of discounting factors applied to bench test results are tabulated in Table 5-1.

Table 5-1: Summary of Discount Factors for Lithia Recovery Model for Option 2

Flotation Circuit Streams	Bench Test	Discount Factor	Flowsheet Model Input
Desliming Mass Pull (%)	11	3	33
Magnetic Concentrate yield (%)	4.1	1	4.1
Mica Concentrate yield (%)	9	1	9
Spodumene Flotation Stage Recovery (%)	88.9	0.95	84.4
Concentrate Grade (%Li2O)	5.2	0.95	4.97

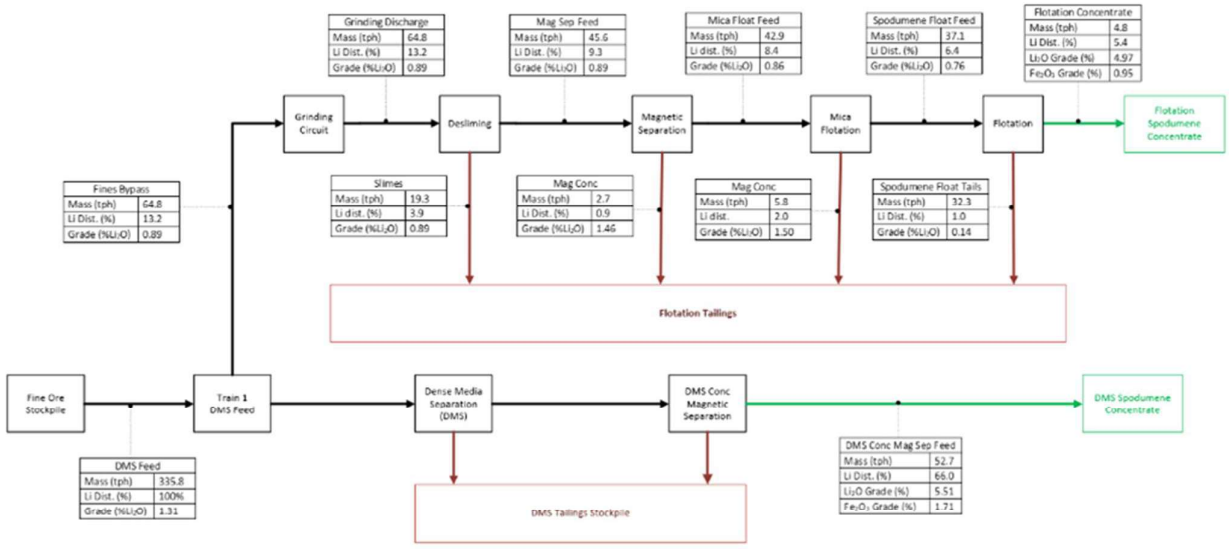


Figure 5-4: Option 2 – High level block flow diagram of DMS and Flotation

Key performance indicators for the hybrid plant flowsheet that treats DMS bypass only in flotation are tabulated in Table 5-2. It can be seen from the table that a DMS only plant is expected to yield 392,266 tons per year of concentrate, at a grade of 5.51 %Li2O and Li recovery of 66%. The addition of spodumene flotation to treat the DMS bypass is expected to yield an additional 31,074 tons per year at 4.97 %Li2O grade for a global annual spodumene concentrate of 423,341 dry tons per year. Overall, Li recovery will be increased by 4.7% by the addition of flotation plant for a global recovery of 70.7%.

Table 5-2: Lithium Deportment and Grade for Option 2

	Concentrate (ton/year)	Li Distribution (%)	Grade (%Li_2O)	Grade (%Fe_2O_3)
DMS	392,266	66.0	5.51	1.71
DMS Bypass Flotation	31,074	4.7	4.97	0.95
Global	423,341	70.7	5.47	1.66

5.4 OPTION 3 – HYBRID PLANT TO TREAT DMS BYPASS AND MIDDLINGS VIA FLOTATION

Option 3 was assessed to further strengthen the economic viability of flotation plant to enhance lithia recovery. This option entails treating the DMS Bypass along with the middlings (DMS floats) from the Fine DMS, Ultrafine

DMS, and Recrush DMS modules. A high-level block flow diagram of DMS and flotation to treat DMS bypass along with the middlings is displayed in Figure 5-5.

A summary of discounting factors applied to bench test results for this option are tabulated in Table 5-3. Unlike Option 2, mass yield for mica concentrate and magnetic concentrate were increased to account for the fact that higher gangue concentrations would impede spodumene flotation and concentrate quality. That is evident by the lower combined grade reporting to the flotation grinding mill. Mass pull of desliming was based on the amount of -20 microns in the overall feed which is expected to be a smaller fraction relative to the mass feed reporting to flotation plant compared to the fraction of slimes in Option 2.

Table 5-3: Lithium Deportment and Grade for Option 3

Flotation Circuit Streams	Bench Test	Discount Factor	Flowsheet Model Input
Desliming Mass Pull (%)	11	1.9	21
Magnetic Concentrate yield (%)	4.1	2.4	10
Mica Concentrate yield (%)	9	1.7	15
Spodumene Flotation Stage Recovery (%)	88.9	0.95	84.4
Concentrate Grade (%Li2O)	5.2	0.95	4.97

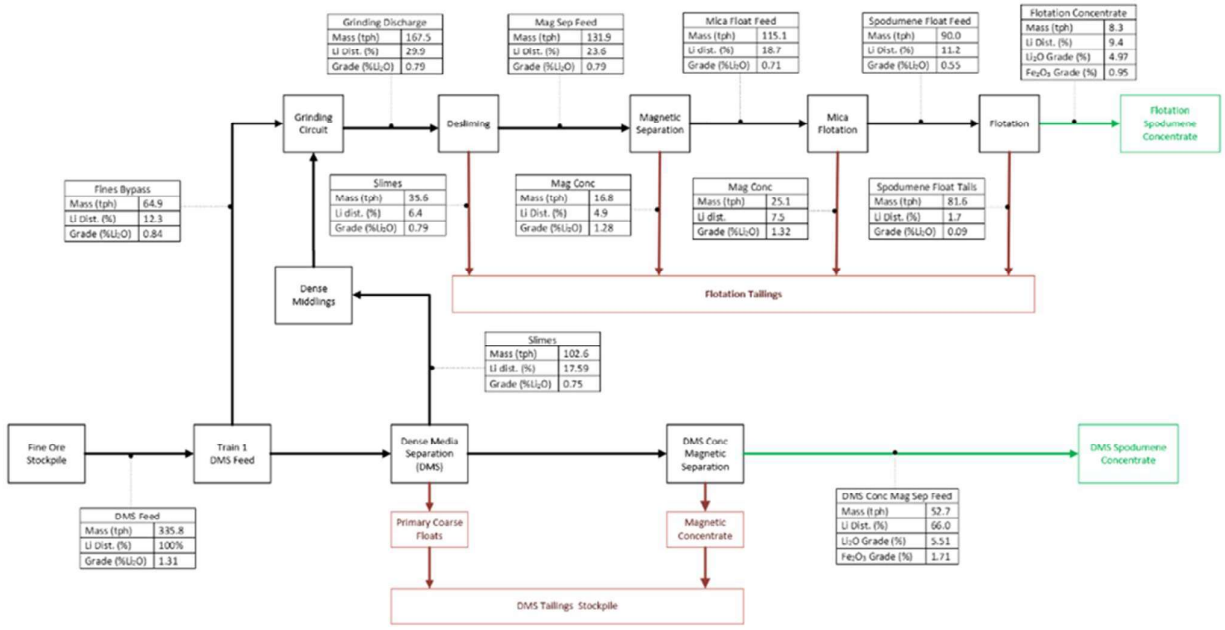


Figure 5-5. Option 3 – High level block flow diagram of DMS and Flotation

Key performance indicators for the hybrid plant flowsheet that treats DMS bypass along with the middlings in flotation are tabulated in Table 5-4. It can be seen from the table that treating DMS bypass and middlings in flotation is expected to yield an additional 62,093 tons per year at 4.97 %Li2O grade for a global annual spodumene concentrate of 454,360 dry tons per year. Overall, Li recovery will be increased by 9.4% with this option for a global recovery of 75.4%.

Table 5-4: Lithium Recovery and Grade for Option 3

	Concentrate (ton/year)	Li Distribution (%)	Grade (%Li_2O)	Grade (%Fe_2O_3)
DMS	392,266	66.0	5.51	1.71
DMS Bypass Flotation	62,093	9.4	4.97	0.95
Global	454,360	75.4	5.43	1.61

6 PROCESS DESIGN

A simplified flowsheet of the proposed flotation circuit with a grinding mill is shown in Figure 6-1.

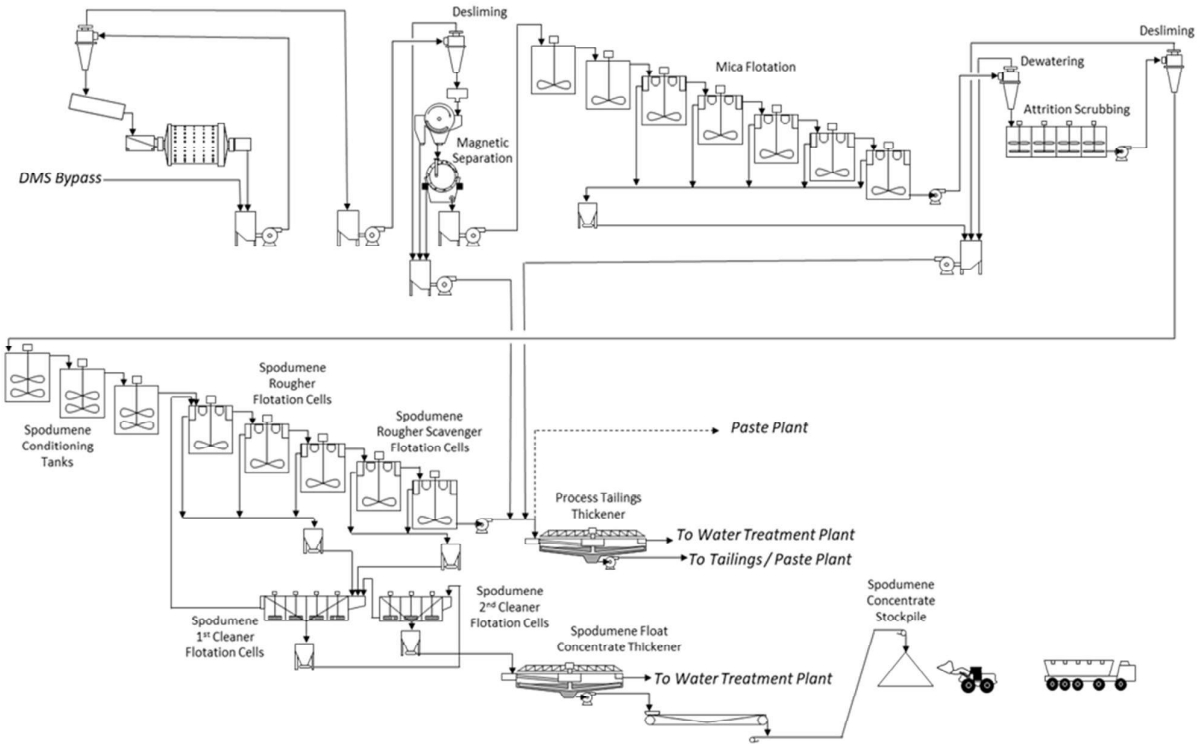


Figure 6-1: Simplified flowsheet of proposed flotation circuit with a grinding mill

6.1 PROCESS DESCRIPTION

The feed source for the flotation circuit can be the DMS bypass (-0.65mm) that account for 19% of total plant feed and 12% of lithium distribution relative to plant feed or the DMS bypass and the middlings from the Recrush DMS, Fines DMS and Ultrafines DMS modules. Material will be milled to achieve a flotation feed P80 of 250 µm where it is expected that nearly 100% of spodumene will be liberated. The first desliming will be used ahead of the Magnetic separators to remove particles finer than 20 µm.

The deslimed material will report to magnetic separation consisting of a one LIMS and one WHIMS to scalp out iron. The mass split will be dictated by the amount Fe2O3 content and mineral deportment to iron which is mostly expected to be waste rock minerals. The non-magnetic material is conditioned at 38% pulp density with the addition of NaOH to raise pH then fed to mica flotation consisting of three rougher cells and 2 scavenger

cells. Mica concentrates report to the process tailings thickener while mica flotation tailings stream from is fed to the dewatering cyclone to thicken the feed to ~50% w/w ahead of attrition scrubbing.

Attrition scrubbing is fed at about 50% w/w with and NaOH is added at this stage to increase pH and activate spodumene. Material discharge from the attrition scrubbing is deslimed and fed to the high-density conditioning tanks where pH is further adjusted using soda ash and spodumene collector is added.

High density conditioning tank discharge reports to the rougher flotation consisting of 3 cells. The tails from the rougher is fed to the rougher scavenger consisting of 2 cells. The combined rougher and rougher scavenger concentrate report to the spodumene cleaner consisting of 3 cells. The tails from the cleaner cells is recycled back to the rougher cells, while the cleaner concentrate is fed to the spodumene recleaner consisting of 3 cells. The tails from the recleaner is recycled back to the cleaner cells, and the recleaner concentrate reports to the concentrate thickener. Spodumene concentrate is dewatered by a vacuum belt filter and conveyed to the flotation concentrate stockpile. Flotation tailings can either be directly pumped to the paste plant or thickened in the process tailings thickener, then pumped to the paste plant, or make use of the existing DMS bypass vacuum belt filter to dry the material for tails stacking.

6.2 PROCESS DESIGN BASIS

Table 6-1 below summarizes the process design basis for the flotation plant from Train 1.

Table 6-1: Process Design Basis for the Flotation Plant

DESCRIPTION	UNITS	NOMINAL	DESIGN
Production Summary			
Throughput	t/h	65	
Average Li2O Feed Grade	% Li2O	0.89	
Concentrate Li2O Grade	% Li2O	5.0	
Global Lithium Recovery	%	41	
Annual Lithium Concentrate at Average Feed Grade	t/a	35,701	
Total Lithium Concentrate over 20 Years	t	714,035	
Process Design Criteria			
Grinding Mill			
Type	-	Ball Mill	
Fresh Feed	t/h	65	80
Feed Size, F100	µm	650	
Feed Size, F80	µm	396	
Product Size, F80	µm	278	
Ball Mill Recirculating Load	%	140	
Ball Mill Discharge Solids	%w/w	70	
Ball mill Cyclone Overflow, P80	µm	200	
Mica Flotation			
Circuit description	-	3 Conditioning Tanks + Rougher & Scavenger + 2 stage Cleaners	
Throughput (dry)	t/h	65	80
Flowrate	m3/h	150	180
Operating pH	-	8.5	
Flotation slurry density	%w/w	35	
Mica Rougher Description	-	Type = 20m^3 Tank cells Quantity = 3 Laboratory flotation time = 2 min Scale-up factor = 2.5 Design retention time = 5 min	

DESCRIPTION	UNITS	NOMINAL	DESIGN
Mica Rougher Scavenger Description	-	Type = 20m^3 Tank cells Quantity = 3 Laboratory flotation time = 1 min Scale-up factor = 2.5 Design retention time = 2.5 min	
Spodumene Flotation			
Circuit description	-	3 Conditioning Tanks + Rougher & Scavenger + 2 stage Cleaners	
Throughput (dry)	t/h	65	80
Flowrate	m^3/h	150	180
Operating pH	-	8.5	
Flotation slurry density	%w/w	35	
Rougher Description	-	Type = 20m^3 Tank cells Quantity = 3 Laboratory flotation time = 1.5 min Scale-up factor = 2.5 Design retention time = 3.8 min	
Rougher Scavenger Description	-	Type = 20m^3 Tank cells Quantity = 2 Laboratory flotation time = 1 min Scale-up factor = 2.5 Design retention time = 2.5 min	
Cleaner Description	-	Type = 12m^3 Tank cells Quantity = 3 Laboratory flotation time = 1.5 min Scale-up factor = 2.5 Design retention time = 3.8 min	
Recleaner Description	-	Type = 12m^3 Tank cells Quantity = 3 Laboratory flotation time = 1 min Scale-up factor = 2.5 Design retention time per cell = 2.5 min	

7 BASIS OF CAPITAL COST ESTIMATES

7.1 GENERAL

The capital cost estimates have a ±35% accuracy and is only for the addition of the flotation plant.

7.1.1 Currency

The base currency of the estimate was in Canadian Dollars (CAD). Where mechanical equipment was quoted in other currencies, they were then converted to CAD using the rates shown in Table 7-1

Table 7-1: Currency Conversion Rates

BASE	CONVERSION
1 CAD	1.00 CAD
1 USD	1.42 CAD
1 CAD	0.9 AUD

BASE CONVERSION

7.1.2 Work Breakdown Structure (WBS)

The WBS used in the estimate was narrowed down to the specific area needed for flotation. Table 7-2 below shows the WBS used for capital expenses.

Table 7-2: Project WBS

WBS	DESCRIPTION
1	Grinding
2	Flotation Feed Magnetic Separation
3	Mica Flotation
4	Attrition Scrubbing
5	Spodumene Flotation
6	Concentrate Dewatering
7	Tailings Dewatering
8	Reagents
9	Plant Services
10	Water Treatment Plant

7.1.3 Inclusions

The capital cost estimate is inclusive of all direct labour, equipment, and materials required to execute the work. All indirect expenses required to support execution are also included.

7.1.4 Exclusions

The following items have been excluded from the cost estimates:

- Local taxes and import duties
- Currency fluctuations
- Financing costs
- Permits
- Escalation
- Force majeure / weather delays / unexpected soil conditions
- Inflation

7.1.5 Factors

The capital cost estimates were prepared based on a combination of vendor quotations, historical data, and engineering judgment. The methodology involved developing a detailed equipment list, obtaining budgetary quotes from vendors for major equipment items, and applying appropriate factors to account for additional discipline costs, freight, installation, indirect costs, and contingencies. Potential risks and uncertainties include fluctuations in commodity prices, and changes in regulatory requirements, which could affect the accuracy of the estimates. Table 7-3 shows a summary of factors applied to the equipment cost to estimate the total capital expenditures. Applied factors totalized to 4.06 (406%) times the equipment supply cost.

Table 7-3: Factors for capital expenditure estimate

CATEGORY	FACTOR	DESCRIPTION
Direct		
Equipment Cost	1.00	
Freight	0.05	Applicable to equipment cost
Installation	0.3	Applicable to equipment cost
Discipline Allowance	2.0	Applicable to installed equipment cost
Indirect		
Indirect	0.15	Applicable to direct cost
Others		
Working Capital, Pre-production Costs	0.30	Applicable to installed equipment cost
Contingency	0.25	Applicable to total cost

7.2 CAPEX SUMMARY FOR OPTION 2

Table 7-4 highlights the total estimated capital expenditures for the addition of a flotation plant to treat the DMS bypass only (Option 2). The total capital cost for this option is $139.80M.

Table 7-4: Summary CAPEX of the Flotation Plant for Option 2

AREA	SUB-AREA	CAPEX (CAD $M)
Direct Costs		
	Grinding	$2.98
	Magnetic Separation	$0.70
	Mica Flotation	$2.46
	Desliming and Attrition Scrubbing	$1.49
	Spodumene Flotation	$1.70
	Concentrate Dewatering	$0.97
	Tailings Dewatering	$0.54
	Reagents	$2.48
	Plant Services	$1.90
	Water Treatment	$6.87
Plant Mechanical Sub-Total		**$22.09**
	Freight	$1.10
	Installation	$6.63
	Subtotal – Plant Mechanical Installed	**$29.82**
	Discipline Allowance	$59.65
Direct Sub-Total		**$89.47**
Indirect Costs		
	Indirect	$13.42
Total Direct and Indirect		**$102.89**
Additional Costs		
	Owners, Preproduction and Working Capital	$8.95
Total Costs without Contingency		**$111.84**
	Contingency	$27.96
Grand Total		**$139.80**

7.3 CAPEX SUMMARY FOR OPTION 3

Table 7-5 highlights the total estimated capital expenditures for the addition of a flotation plant to treat the DMS bypass only and middlings (Option 3). The total capital cost for this option is $247.1M.

Table 7-5: Summary CAPEX of the Flotation Plant for Option 3

AREA	SUB-AREA	CAPEX (CAD $M)
Direct Costs		
	Grinding	$5.27
	Magnetic Separation	$1.23
	Mica Flotation	$4.35
	Desliming and Attrition Scrubbing	$2.63
	Spodumene Flotation	$3.00
	Concentrate Dewatering	$1.71
	Tailings Dewatering	$0.96
	Reagents	$4.39
	Plant Services	$3.36
	Water Treatment	$12.15
Plant Mechanical Sub-Total		**$39.05**
	Freight	$1.95
	Installation	$11.72
	Subtotal – Plant Mechanical Installed	**$52.72**
	Discipline Allowance	$105.44
Direct Sub-Total		**$158.17**
Indirect Costs		
	Indirect	$23.72
Total Direct and Indirect		**$181.89**
Additional Costs		
	Owners, Preproduction and Working Capital	$15.82
Total Costs without Contingency		**$197.71**
	Contingency	$49.43
Grand Total		**$247.14**

8 BASIS OF OPERATING COST ESTIMATES

The operating cost estimate of the flotation plant for Option 2 and Option 3 was prepared and is summarized in Table 8-1 and Table 8-2 respectively. The flotation plant OPEX was estimated on the basis of 482 kt/y of feed that is exclusively DMS bypass for the life of mine. It should be noted that Mobile Equipment, and G&A are not included in this estimate because it is assumed this cost will not increase as a result of adding the flotation plant. Concentrate transportation is included in the economic model. Further detail can be found in the Estimate document CASN-A-620-GE-EST-0200_B. Potential risks and uncertainties include fluctuations in energy prices, changes in reagent costs, and unforeseen maintenance requirements, which could affect the accuracy of the estimates.

Table 8-1: Summary OPEX of the Flotation Plant for Option 2

Cost Center	Estimated Cost		
	CAD $M/year	CAD/t plant feed	CAD/t flotation conc
Plant Labour	$6.380	$2.55	$205.32
Consumables	$1.246	$0.50	$40.09
Reagents	$3.439	$1.38	$110.67
Power	$2.155	$0.86	$69.34
Maintenance	$0.900	$0.36	$28.95
Laboratory	$0.457	$0.18	$14.72
Total Process OPEX	**$14.577**	**$5.83**	**$469.09**

Table 8-2: Summary OPEX of the Flotation Plant for Option 3

Cost Center	Estimated Cost		
	CAD $M/year	CAD/t plant feed	CAD/t flotation conc
Plant Labour	$6.380	$2.55	$102.75
Consumables	$2.608	$1.04	$42.01
Reagents	$8.867	$3.55	$142.80
Power	$2.504	$1.00	$40.33
Maintenance	$1.324	$0.53	$21.32
Laboratory	$0.457	$0.18	$7.37
Total Process OPEX	**$22.140**	**$8.86**	**$356.57**

9 ECONOMIC MODEL AND SENSITIVITY ANALYSIS

It was assumed that the addition of flotation plant can happen after 3 years of DMS only operation. On that basis, the economic analysis was based on 20 years. A detailed before tax financial model and discounted yearly cash flow (DCF) has been developed to complete the economic assessment of the addition of a flotation plant and is based on current 2025 price projections and cost estimates. No provision was made for the effects of inflation, but cost estimates incorporate recent inflationary price increases. The evaluation was carried out using an 8% discount factor over 20 years of mine life with flotation operation.

The pricing information for spodumene concentrate supply are based on a fixed price of $850/t for 5.5% Li2O spodumene concentrate. Concentrate transportation was fixed at $250/t Li2O concentrate and is based on current contractor quotation. It was assumed that spodumene concentrate production will take 3 years to ramp up to nominal production at a rate 5% gradual increase each year starting at 85% nominal production in year 1.

9.1 ECONOMIC MODEL AND SENSITIVITY ANALYIS FOR OPTION 2

Table 9-1 shows the summary of the project economics for option 2 over the life of the mine. The main project economic indicators are presented in

Table 9-2. The economic study shows a before tax profit (EBT) of $73.7M. The net present value of the 20-year based project is -$38.00M at an 8% discount rate and before applicable taxes, inferring that a hybrid plant to treat the DMS bypass only would be a high risk investment. The before-tax internal rate of return (IRR) for the additional flotation stage is only 4% with a long payback time of 13.3 years.

Table 9-1: Project Economics Summary

Flotation Plant Financial Results	Unit of Measure	Value
Pre-Tax NPV @ 8%	CAD $M	-$38.00
Pre-tax IRR	%	4
Pre-Tax Payback Period	years	13.3

Table 9-2: Economic Indicator Summary

Pre-Tax Income Statement	Value (CAD $M)
Gross revenues (Li_2O)	$505.04
Operating Cost Cash Flow	-$291.53
EBITDA	$213.51
Capital Expenditure (pre-production)	-$139.80
Gross profit before tax (EBT)	$73.72

The sensitivity analysis examines the impact of changes in key parameters such as spodumene price, capital costs, and operating costs on the project's economic viability. However, the sensitivity analysis is limited to a certain range of values and does not consider all possible scenarios. Additional scenarios that could be considered include changes in exchange rates, regulatory requirements, and environmental regulations. Sensitivity analysis of the 20-year NPV(8%) and IRR was completed with the following variation:

- Up to 20% increase and decrease in capital expenditure in 5% increment.
- Up to 20% increase and decrease in product price in 5% decrement.

The results for the NPV(8%) and IRR are summarized in Table 9-3 and Table 9-4 respectively.

Table 9-3: Sensitivity Analysis of 20-Year NPV for Option 2

Pre-Tax NPV Sensitivity Analysis		Capital Expenditure (CAD $M)							
		$111.84	$118.83	$125.82	$132.81	$146.79	$153.78	$160.77	$167.76
Spodumene Concentrate Price (USD)	$997	$60.04	$53.05	$46.06	$39.07	$25.09	$18.10	$11.11	$4.12
	$956	$42.53	$35.54	$28.55	$21.56	$7.58	$0.59	-$6.40	-$13.39
	$914	$25.01	$18.02	$11.03	$4.04	-$9.94	-$16.93	-$23.92	-$30.91
	$873	$7.49	$0.50	-$6.49	-$13.48	-$27.46	-$34.45	-$41.44	-$48.43
	$789	-$27.54	-$34.53	-$41.52	-$48.51	-$62.49	-$69.48	-$76.47	-$83.46
	$748	-$45.05	-$52.04	-$59.03	-$66.02	-$80.00	-$86.99	-$93.98	-$100.97
	$706	-$62.57	-$69.56	-$76.55	-$83.54	-$97.52	-$104.51	-$111.50	-$118.49
	$665	-$80.09	-$87.08	-$94.07	-$101.06	-$115.04	-$122.03	-$129.02	-$136.01

Table 9-4: Sensitivity Analysis of 20-Year IRR for Option 2

Pre-Tax IRR Sensitivity Analysis		Capital Expenditure (CAD $M)							
		$111.84	$118.,83	$125.82	$132.81	$146.79	$153.78	$160.77	$167.76
Spodumene Concentrate Price (USD)	$997	14%	13%	12%	12%	10%	9%	9%	8%
	$956	13%	12%	11%	10%	9%	8%	7%	7%
	$914	11%	10%	9%	8%	7%	7%	6%	6%
	$873	9%	8%	7%	7%	5%	5%	4%	4%
	$789	5%	4%	3%	3%	2%	1%	1%	1%
	$748	2%	2%	1%	1%	0%	-1%	-1%	-2%
	$706	-1%	-1%	-2%	-2%	-3%	-3%	-4%	-4%
	$665	-4%	-4%	-5%	-5%	-6%	-6%	-7%	-7%

The results are showing that:

- Both NPV and IRR are highly sensitive to spodumene concentrate selling price.
- Both NPV and IRR are less sensitive to variations in CAPEX.
- The project is still high risk at a 20% increase of spodumene price, an 20% decrease in CAPEX, which is reflected by the 20-year pre-tax IRR of only 14% as shown in Table 9-4.

9.2 ECONOMIC MODEL AND SENSITIVITY ANALYSIS FOR OPTION 3

The summary of the project economics and main economic indicators for Option 3 are displayed in Table 9-5 and Table 9-6 respectively. The economic study shows a before tax profit (EBT) of $319M. The net present value of the 20-year based project is $24.91 M at an 8% discount rate and before applicable taxes. The before-tax internal rate of return (IRR) for additional flotation stage is only 9% with a long payback time of 9 years, inferring that expanding the hybrid plant capacity to treat the DMS bypass and the middlings would still be a very risky investment at the current spodumene concentrate price.

Table 9-5: Project Economics Summary

Flotation Plant Financial Results	Unit of Measure	Value
Pre-Tax NPV @ 8%	CAD $M	$24.91
Pre-tax IRR	%	9
Pre-Tax Payback Period	years	9

Table 9-6: Economic Indicator Summary

Pre-Tax Income Statement	Value (CAD $ M)
Gross revenues (Li_2O)	$1,009.20
Operating Cost Cash Flow	-$442.82
EBITDA	$566.38
Capital Expenditure (pre-production)	-$247.14
Gross profit before tax (EBT)	$319.25

Sensitivity analysis of the 20-year NPV(8%) and IRR was completed for Option 3 with the following variation:

- Up to 20% increase and decrease in capital expenditure in 5% increment.
- Up to 20% increase and decrease in product price in 5% decrement.

The results for the pre-tax NPV(8%) and IRR are summarized in Table 9-7and Table 9-8 respectively.

Table 9-7: Sensitivity Analysis of 20-Year NPV for Option 3

Pre-Tax NPV Sensitivity Analysis		Capital Expenditure (CAD $M)							
		$197.71	$210.06	$222.42	$234.78	$259.49	$271.85	$284.21	$296.56
Spodumene Concentrate Price (USD)	$997	$214.34	$201.98	$189.63	$177.27	$152.56	$140.20	$127.84	$115.49
	$956	$179.34	$166.98	$154.63	$142.27	$117.56	$105.20	$92.84	$80.49
	$914	$144.34	$131.98	$119.63	$107.27	$82.55	$70.20	$57.84	$45.48
	$873	$109.34	$96.98	$84.62	$72.27	$47.55	$35.20	$22.84	$10.48
	$789	$39.34	$26.98	$14.62	$2.26	-$22.45	-$34.81	-$47.16	-$59.52
	$748	$4.33	-$8.02	-$20.38	-$32.74	-$57.45	-$69.81	-$82.16	-$94.52
	$706	-$30.67	-$43.02	-$55.38	-$67.74	-$92.45	-$104.81	-$117.16	-$129.52
	$665	-$65.67	-$78.02	-$90.38	-$102.74	-$127.45	-$139.81	-$152.17	-$164.52

Table 9-8: Sensitivity Analysis of 20-Year IRR for Option 3

Pre-Tax IRR Sensitivity Analysis		Capital Expenditure (CAD $M)							
		$197.71	$210.06	$222.42	$234.78	$259.49	$271.85	$284.21	$296.56
Spodumene Concentrate Price (USD)	$997	20%	19%	18%	17%	15%	14%	13%	13%
	$956	18%	17%	16%	15%	13%	13%	12%	11%
	$914	16%	15%	14%	13%	12%	11%	11%	10%
	$873	14%	13%	13%	12%	10%	10%	9%	8%
	$789	10%	10%	9%	8%	7%	6%	6%	5%
	$748	8%	8%	7%	6%	5%	4%	4%	4%
	$706	6%	5%	5%	4%	3%	2%	2%	2%
	$665	3%	3%	2%	2%	1%	0%	0%	-1%

The results of sensitivity analysis shows that Option 3 would be moderately attractive at a 20% increase of spodumene price, an 20% decrease in CAPEX, which is reflected by the 20-year pre-tax IRR of 20%.

10 ALTERNATIVE HYBRID PLANT CONSIDERATIONS

The review of economics of option 2 and option 3 lead to a question whether a different hybrid plant could improve the economics.

Option 4 is considered where the DMS plant complexity and recovery is reduced, and the flotation plant scope is increased to produce a hybrid plant with the same ultimate recovery. It is recognised that this hypothetical scenario would be a significant disruption to the project.

The basis is that if a flotation plant is being considered, then the DMS plant can be simplified without affecting ultimate overall recovery. Thereby resulting in a reduction of the CAPEX of the DMS plant, an increase in the CAPEX of the flotation plant, but a lower overall cost (compared to option 3)

The same schedule was assumed for the implementation of each of the options, namely installation in year 4 and fully operation in year 5.

The lower revenue due to lower base DMS recovery was not offset by the initial reduced capital cost, and the overall economics is estimated to rank between option 2 and option 3.

11 CONCLUSIONS

The trade-off study for the Shaakichiuwaanaan Lithium Project assessed the technical and economic feasibility of integrating flotation into the existing DMS-based flowsheet. The objective was to assess the impact of a small and large flotation plant to respectively treat the DMS bypass only (Option 2) and the bypass along with middlings from various DMS modules (Option 3).

The study demonstrated that:

- There is a technical rationale for enhancing lithium recovery through the addition of flotation, either as only the bypass fraction or as bypass and middlings.
- Option 2, treating only the DMS bypass in flotation, yields a 4.7% increase in overall lithium recovery and an additional 31,074 t/y of spodumene concentrate. However, it results in a negative pre-tax NPV of -$38.0M and a low IRR of 4%, with a 13.3-year payback, indicating an economically unviable project under current spodumene pricing.
- Option 3, treating both the DMS bypass and middlings in a larger flotation plant compared to Option 2, improves overall lithium recovery by 9.4%, with an additional 62.09 kt/y of spodumene concentrate. It delivers a positive pre-tax NPV of $24.9M and a 9% IRR, the 9-year payback and substantial capital requirement ($247M) suggest high financial risk.
- Considering a change in the DMS flotation split (Option 4), did not improve the economics, but could provide strategic benefits that could be investigated for the second process plant.
- All options would require additional permitting and environmental controls, particularly due to the inclusion of a water treatment plant and increased water usage.
- Sensitivity analyses revealed that project economics are highly dependent on spodumene concentrate pricing, while variations in CAPEX have comparatively less impact. Even under favourable scenarios, the returns remain modest.
- The inclusion of flotation plant is not recommended.

APPENDIX

4-2 NOTE ON THE TRANSPORT OF SPODUMENE CONCENTRATE

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Mémorandum

À : **Pascal Droz**

CC :

De : **Jean-Louis Folio**

Date : **1er avril 2025**

Objet : **NOTE SUR LE TRANSPORT DE CONCENTRÉ DE SPODUMÈNE**

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1. BASE D'ÉVALUATION

1.1. Emplacement

Le site minier est situé sur la route Transtaiga à partir du terminal de transbordement de Matagami :

Route Billy Diamond :	554 km
Route Transtaiga :	290 km
Total :	844 km

1.2. Production

Phase 1 :	400 000 tonnes par année
Phase 2 :	+ 400 000 tonnes par année
Total :	800 000 tonnes par année

Soit 2 191 tonnes par jour.

Ce chiffre a été modifié à 2 300 tonnes par jour.

1.3. Destination / Livraison

Le terminal de transbordement de Matagami se trouve à 844 km du site de chargement de la mine.

2. TRANSPORT PAR CAMION À BENNE

Aux fins de cette étude, nous avons contacté :

- Trois (3) entreprises de transport, sélectionnées en fonction de leur implication dans le transport nordique et de leurs capacités opérationnelles et financières :
  - Transport Fournier.
  - Transport JF Millette.

  - Great River Transport.
- Direction générale MTMD Eeyou Istche Baie James (Chibougamau).
- Société de développement de la Baie James (Matagami).
- Cour de Transbordement de Matagami.

2.1. Capacité routière

Actuellement, considérant une tolérance SDBJ de 10 % sur la route Billy Diamond, le PTC est de 68 750 kg. La charge nette de minerai prévue serait actuellement de 45 à 47 tonnes métriques. Les instructions de Patriot sont de baser les calculs sur une charge de 75 tonnes par camion.

Patriot devra obtenir des permis officiels pour pouvoir transporter une charge de 75 tonnes par voyage.

2.2. Camions et remorques

La confirmation des autorités routières est en attente, mais nous pouvons prévoir d'utiliser :

- Un tracteur à quatre (4) essieux au lieu de trois (3).
- Une remorque bi-train à cinq (5) essieux et à déchargement latéral.

2.3. Rotation / Calendrier / Personnel / Entretien

Tous ces points relèvent de la seule responsabilité des transporteurs, qui doivent faire tout leur possible pour garantir la livraison de 2 300 tonnes par jour à Matagami.

Il est entendu que les installations de la « Cour de Transbordement de Matagami » seront tout à fait adéquates et utilisables.

2.4. Coût du transport du site minier à Bécancour

Destination :	**Bécancour**
Tonnage / Année :	800 000
Tonnage / Jour :	2 300
Tonnage / camion demandé par Patriot :	75

		Entreprises de transport		
		Millette Val d'Or	**Fournier Val d'Or**	**Big River James Bay**
1	**Transport par camion du site minier à Matagami**			
	Coût par tonne métrique :	$95.00	$145.33	$229.33
	Supplément carburant :	À ajouter	À ajouter	À ajouter
2	**Transbordement à Matagami du camion au wagon**			
	Frais de triage :	$2.00	$2.00	$2.00
	Transbordement, y compris le stockage :	$15.00	$15.00	$15.00
3	**Transport ferroviaire de Matagami à Bécancour**			
	Location de wagons / couvercle :	$11.30	$11.30	$11.30
	Transport ferroviaire (699 miles) :	$55.83	$55.83	$55.83
	Total / voyage	$5,192.17	$5,192.17	$5,192.17
	Tonnage / wagon	93	93	93
4	**Total per Ton**	**$179.13**	**$229.46**	**$313.46**

Rail – Matagami Bécancour	
Unité	Gondolas 52’
Tonnes par année	800,000
Capacité portante du Rail (lbs)	286,000
Gondole (67.5K + 1.5 couvercle) (lbs)	69,000
Capacité maximale des wagons (lbs)	217,000
Capacité maximale des wagons (kg)	98,429
Destination	Bécancour
Tonnes par wagon	93
Wagon total par année (50 semaines)	8,602
Nombre de semaines d’opération annuelle	50
Jours d’opération annuel	260
Nombre de jours par cycle	18

Rail – Matagami Bécancour	
Nombre de cycles en 50 semaines	14.44
Nombre de wagon moyen par semaine	172 cars
Quantité de wagons requis	595
Marge d'erreur - réparations	4%
Wagons requis	619.34
Nombre total de wagon requis	620

2.5. Coût du transport du site minier au terminal pétrolier de Québec

Destination :	**Québec POL**
Tonnage / Année :	800 000
Tonnage / Jour :	2 300
Tonnage / camion demandé par Patriot :	75

		Entreprises de transport		
		Millette Val d'Or	**Fournier Val d'Or**	**Big River James Bay**
1	**Transport par camion du site minier à Matagami**			
	Coût par tonne métrique :	$95.00	$145.33	$229.33
	Supplément carburant :	À ajouter	À ajouter	À ajouter
2	**Transbordement à Matagami du camion au wagon**			
	Frais de triage :	$2.00	$2.00	$2.00
	Transbordement, y compris le stockage :	$15.00	$15.00	$15.00
3	**Transport ferroviaire de Matagami à Québec (Beauport)**			
	Location de wagons / couvercle :	$12.56	$12.56	$12.56
	Transport ferroviaire (699 miles) :	$63.82	$63.82	$63.82
	Total / voyage	$5,934.97	$5, 934.97	$5, 934.97
	Tonnage / wagon	93	93	93
4	**Total per Ton**	**$188.38**	**$238.71**	**$322.71**

Rail – Matagami Bécancour	
Unité	Gondolas 52’
Tonnes par année	800,000
Capacité portante du Rail (lbs)	286,000
Gondole (67.5K + 1.5 couvercle) (lbs)	69,000
Capacité maximale des wagons (lbs)	217,000
Capacité maximale des wagons (kg)	98,429
Destination	Québec
Tonnes par wagon	93
Wagon total par année (50 semaines)	8,602
Nombre de semaines d’opération annuelle	50
Jours d’opération annuel	260
Nombre de jours par cycle	20
Nombre de cycles en 50 semaines	13
Nombre de wagon moyen par semaine	172 cars
Quantité de wagons requis	662
Marge d’erreur - réparations	4%
Wagons requis	688.15
Nombre total de wagon requis	689

	Entreprises de transport		
	Millette Val d'Or	**Fournier Val d'Or**	**Big River James Bay**
Opération au port de chargement - Québec (Beauport)			
Coût par tonne du wagon, transfert et entreposage	$15.95	$15.95	$15.95
Récupérer et charger sur le navire	$11.70	$11.70	$11.70
Coût total du site au navire	**$216.03**	**$266.36**	**$350.36**

2.6. Coût du transport du site minier au terminal maritime de Grande Anse (La Baie)

Destination :	**Grande Anse POL**
Tonnage / Année :	800 000
Tonnage / Jour :	2 300
Tonnage / camion demandé par Patriot :	75

		Entreprises de transport		
		Millette Val d'Or	**Fournier Val d'Or**	**Big River James Bay**
1	**Transport par camion du site minier à Matagami**			
	Coût par tonne métrique :	$95.00	$145.33	$229.33
	Supplément carburant :	À ajouter	À ajouter	À ajouter
2	**Transbordement à Matagami du camion au wagon**			
	Frais de triage :	$2.00	$2.00	$2.00
	Transbordement, y compris le stockage :	$15.00	$15.00	$15.00
3	**Transport ferroviaire de Matagami à Grande Anse (La Baie)**			
	Location de wagons / couvercle :	$11.30	$11.30	$11.30
	Transport ferroviaire (699 miles) :	$53.35	$53.35	$53.35
	Total / voyage	$4,961.90	$4,961.90	$4,961.90
	Tonnage / wagon	93	93	93
4	**Total per Ton**	**$176.65**	**$226.98**	**$310.98**

Rail – Matagami Bécancour	
Unité	Gondolas 52’
Tonnes par année	800,000
Capacité portante du Rail (lbs)	286,000
Gondole (67.5K + 1.5 couvercle) (lbs)	69,000
Capacité maximale des wagons (lbs)	217,000
Capacité maximale des wagons (kg)	98,429

Rail – Matagami Bécancour	
Destination	La Baie
Tonnes par wagon	93
Wagon total par année (50 semaines)	8,602
Nombre de semaines d'opération annuelle	50
Jours d'opération annuel	260
Nombre de jours par cycle	18
Nombre de cycles en 50 semaines	14.44
Nombre de wagon moyen par semaine	172 cars
Quantité de wagons requis	596
Marge d'erreur - réparations	4%
Wagons requis	619.44
Nombre total de wagon requis	620

	Entreprises de transport		
	Millette Val d'Or	**Fournier Val d'Or**	**Big River James Bay**
Opération au port de chargement – Grande Anse (Québec)			
Coût par tonne du wagon, transfert et entreposage	$26.00	$26.00	$26.00
Récupérer et charger sur le navire	$5.50	$5.50	$5.50
Coût total du site au navire	**$208.15**	**$258.15**	**$342.8**

Tous ces chiffres peuvent être modifiés en fonction de la durée de détention des wagons à destination.

3. RECOMMENDATION

Compte tenu des coûts présentés précédemment, il est recommandé de retenir l'option de transport routier assurée par **Transport Millette** jusqu'à Matagami, suivie d'un acheminement ferroviaire vers **Grande-Anse**, pour un coût total estimé à **208,15 $ par tonne**.

Au-delà de l'avantage financier, **Transport Millette** se distingue par sa structure organisationnelle robuste et son expertise éprouvée dans ce type d'opérations logistiques. L'entreprise est bien implantée dans la région, dessert déjà plusieurs acteurs industriels et démontre un engagement concret envers l'embauche au sein des communautés des Premières Nations.

Le site de **Grande-Anse** nécessitera l'installation d'un nouvel abri, situé à distance des installations portuaires, afin d'assurer l'entreposage sécuritaire du concentré dès son arrivée par voie ferroviaire. Par ailleurs, un projet de convoyeur reliant l'aire de dépôt au port est actuellement en cours de construction et devrait être achevé d'ici la fin de l'année 2025. La proposition soumise par **QSL** intègre l'ensemble de ces coûts dans le tarif exprimé en dollars par tonne.

Cordialement,

Jean-Louis Folio

Jean-Louis Folio
Directeur Logistique

APPENDIX

4-3 TECHNO-ECONOMIC ANALYSIS OF ELECTRICAL POWER GENERATION AT THE MINING COMPLEX

Ressources PMET

Analyse technicoéconomique

Projet minier Shaakichiuwaanaan, Qc

Rapport technique

Analyse technicoéconomique de la génération d'électricité du complexe minier

N° document BBA / Rév : 7535030-000000-47-ERA-0001 / R00

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Sommaire exécutif

Introduction

Ressources PMET a mandaté BBA afin d'évaluer les options de génération d'électricité pour l'exploitation du projet minier Shaakichiuwaanaan, situé à 53 km du barrage hydroélectrique de La Grande-4 (LG4). La mise en service de la mine est prévue en 2029 pour une durée d'exploitation de 24 années.

Les options à l'étude sont les suivantes :

- 1 – Interconnexion au poste de LG4 d'Hydro-Québec (HQ) comprenant notamment la construction d'une nouvelle ligne aérienne 120 kV de 53 km
- 2A – Centrale autonome au diésel de 65,7 MW
- 2B – Centrale autonome hybride au diésel de 65,7 MW avec un parc éolien de 24,8 MW et un BESS de 12,4 MW/12,4 MWh
- 3A – Centrale autonome au gaz naturel liquéfié (GNL) de 65,6 MW
- 3B – Centrale autonome hybride au gaz naturel liquéfié de 65,6 MW avec un parc éolien de 24,8 MW et un BESS de 12,4 MW/12,4 MWh
- 4A – Centrale autonome au gaz naturel compressé (GNC) de 65,6 MW
- 4B – Centrale autonome hybride au gaz naturel compressé de 65,6 MW avec un parc éolien de 24,8 MW et un BESS de 12,4 MW/12,4 MWh

Ce rapport présente les résultats de l'analyse technicoéconomique préliminaire et des émissions de gaz à effet de serre le tout avec une précision de classe V (-50 %/+100 %).

L'analyse économique de la génération d'électricité a été réalisée basée sur le CAPEX du projet ainsi que des couts d'opération en carburant et en maintenance des différents scénarios. Aucun revenu de la mine n'a été considéré, car dans tous les scénarios, ils seront identiques et dépendront de la production de la mine.

Dans le cas des centrales thermiques, un système de récupération de chaleur peut être utilisé afin de chauffer les bâtiments principaux. Les économies de GNC pour le chauffage épargné par ce système sont donc représentées comme un revenu dans les différents scénarios.

Résultats

Les résultats de l'analyse économique sont présentés dans le tableau suivant :

Tableau 1 : Comparaison économique des différents scénarios

Scénario	CAPEX	VAN	LCOE [$/kWh]	Émissions de GES annuelles [tCO_{2eq}]
1 – Interconnexion avec Hydro-Québec au poste Tilly	195 000 000 $	(398 176 000 $)	0,153	5 500
2A – Centrale autonome au diésel	174 315 000 $	(1 056 763 000 $)	0,405	181 000
2B – Centrale autonome hybride au diésel	269 322 000 $	(904 989 000 $)	0,347	136 000
3A – Centrale autonome au gaz naturel liquéfié	196 683 000 $	(831 936 000 $)	0,319	128 000
3B – Centrale autonome hybride au gaz naturel liquéfié	291 690 000 $	(743 071 000 $)	0,285	95 000
4A – Centrale autonome au gaz naturel compressé	175 575 000 $	(1 057 596 000 $)	0,406	128 000
4B – Centrale autonome hybride au gaz naturel compressé	270 581 000 $	(897 788 000 $)	0,344	95 000

L'évolution de la valeur actuelle nette (VAN) estimée des scénarios durant les années d'opération de la mine est illustrée sur la figure suivante :

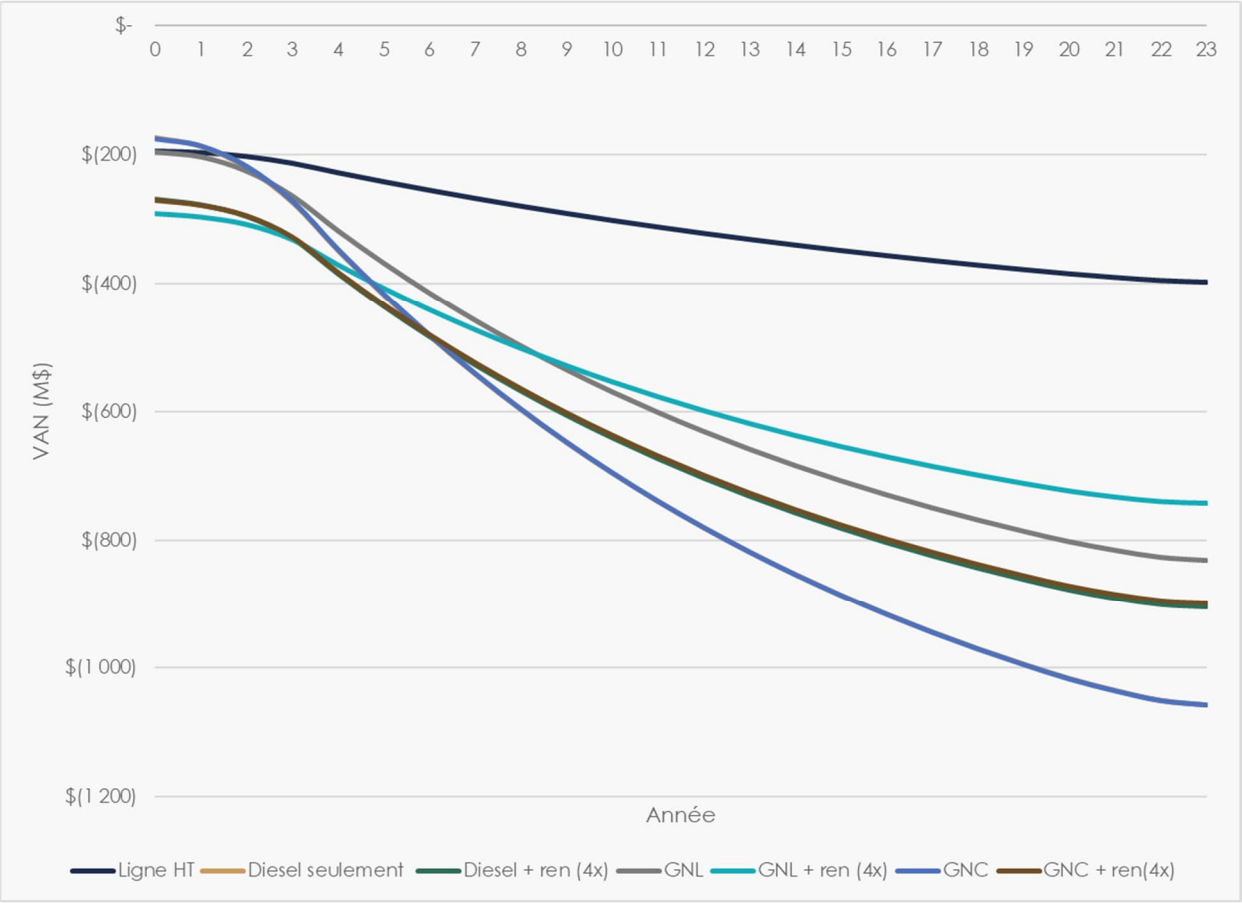


Figure 1 : Évolution de la VAN des différents scénarios

Il est important de noter que ces couts ne devraient être comparés qu'entre eux, car ils sont tous calculés sur la même base. De plus, les couts des droits d'émissions et de carburants utilisés pourraient changer au cours du projet et leurs variations sont difficiles à prévoir. Ils ont donc été considérés comme constants dans cette analyse.

Les hypothèses ou données d'entrées principales utilisées dans cette analyse sont les suivantes :

- Consommation d'énergie électrique annuelle pour les années d'exploitation régulière : 302 658 000 kWh (calculée à partir du profil de charge).
- Une charge moyenne de 34,55 MW est considérée pendant les années d'exploitation normales de la mine.
- Consommation d'énergie thermique annuelle moyenne estimée pour le chauffage des bâtiments principaux : 26 673 000 kWh.

- Durée de vie de la mine : 23 ans.
- Taux d'actualisation (pour calcul de la VAN et du LCOE) : 8 %.
- Taux d'inflation sur le prix du carburant : 0 %.
- Taux d'inflation sur les couts d'exploitation et de maintenance : 2 %.
- Type de contrat considéré : EPC.
- Cout des droits d'émissions fixes sur la durée de vie de la mine : 45 $/tonne CO_{2eq}

Conclusions et recommandations

Afin d'effectuer une comparaison juste des différents scénarios, une matrice de décision a été réalisée selon les aspects critiques liés au projet. Les critères ont été regroupés en quatre sous-sections, soit l'impact économique, l'impact environnemental, l'impact social et l'impact technique. Les résultats par sous-section sont présentés dans le tableau suivant et le détail est disponible à la section 5.

Tableau 2 : Résultats de la matrice décisionnelle

Critère	Poids	1 – Ligne H.T.		2A- Diésel		2B – Diésel + renouvelable		3A – GNL		3B – GNL + renouvelable		4A – GNC		4B – GNC + renouvelable	
		Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage
Impact économique (30%)															
CAPEX	5	2	10	3	15	1	5	2	10	1	5	3	15	1	5
VAN	3	3	9	1	3	1	3	2	6	2	6	1	3	1	3
Prévisibilité	2	3	6	1	2	2	4	1	2	2	4	1	2	2	4
Sous-total	**10**	**25**		**20**		**12**		**18**		**15**		**20**		**12**	
Impact environnemental (30%)															
Émissions GES	4	3	12	1	4	2	8	2	8	2	8	2	8	2	8
Impact sur les plans d'eau et les milieux humides	2	1	2	3	6	2	4	3	6	2	4	3	6	2	4
Impact sur la faune	2	2	4	3	6	1	2	3	6	1	2	3	6	1	2
Empreinte sur le territoire	2	1	2	3	6	2	4	3	6	2	4	2	4	2	4
Sous-total	**10**	**20**		**22**		**18**		**26**		**18**		**24**		**18**	

Critère	Poids	1 – Ligne H.T.		2A- Diésel		2B – Diésel + renouvelable		3A – GNL		3B – GNL + renouvelable		4A – GNC		4B – GNC + renouvelable	
		Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage
Impact social (30%)															
Retombées économiques	3	2	6	1	3	1	3	1	3	1	3	1	3	1	3
Circulation	3	3	9	1	3	2	6	1	3	2	6	1	3	2	6
Autres utilisateurs possibles	2	3	6	1	2	1	2	1	2	1	2	1	2	1	2
Nuisance (bruit, poussière, paysage)	2	3	6	2	4	1	2	2	4	1	2	2	4	1	2
Sous-total	**10**	**27**		**12**		**13**		**12**		**13**		**12**		**13**	
Impact technique (10%)															
Constructibilité et délai	2	3	6	3	6	2	4	3	6	2	4	2	4	2	4
Fiabilité électrique	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Sous-total	**3**	**8**		**8**		**6**		**8**		**5**		**6**		**6**	
Total	**33**	**80**		**62**		**49**		**64**		**51**		**62**		**49**	

Selon les résultats obtenus, le scénario le plus avantageux est celui de l'interconnexion avec le réseau d'Hydro-Québec puisqu'il présente le meilleur pointage pondéré. Il est à noter que si ce scénario est sélectionné, une adhésion au programme de gestion de la demande de puissance (GDP) d'Hydro-Québec est requises et nécessitera une capacité de délestage de la charge d'au moins 30% pour un nombre d'heure déterminé selon le niveau d'engagement choisi. Aucun délestage, ni adhésion au programme de GDP n'a été considéré dans cette étude. Une étude spécifique sera requise pour évaluer les différentes options d'adhésion à la GDP.

Le scénario de production avec une centrale autonome au gaz naturel liquéfié est le deuxième scénario le plus avantageux, principalement en raison de sa VAN inférieure comparativement aux autres scénarios et son impact environnemental. Par ailleurs, en faisant abstraction de son investissement initial, le scénario de la centrale autonome au gaz naturel liquéfié est plus avantageux au niveau des couts d'exploitation.

Aucun scénario intégrant uniquement des énergies renouvelables n'a été considéré dans cette étude. Les énergies renouvelables ne représentent pas une capacité ferme et doivent être jumelées à une source de capacité ferme comme une centrale autonome ou Hydro-Québec pour assurer une opération continue.

Prochaines étapes

La prochaine étape dans le cas de l'interconnexion avec Hydro-Québec serait d'obtenir une entente pour la totalité des MW requis pour la durée du projet afin de démarrer les travaux le plus rapidement possible. Dans ce cas, il serait important d'analyser le niveau d'engagement à utiliser pour le Programme de Gestion de la demande de puissance (GDP) ainsi que les options envisageables pour le délestage.

Pour ce qui est d'une centrale hybride autonome, en cas d'absence d'entente avec Hydro-Québec, la prochaine étape recommandée est de commencer l'ingénierie préliminaire de la centrale aussitôt que possible afin d'aller en appel d'offres rapidement, notamment pour les équipements à longs délais de livraison tels que les génératrices.

Dans l'éventualité où l'intégration des énergies renouvelables serait analysée plus en profondeur dans le but de réduire les couts d'opération (OPEX) et les émissions de GES, tout en augmentant la valeur actuelle nette, les étapes suivantes devront être réalisées :

- Effectuer une évaluation de la ressource éolienne au site en installant une tour de mesurage de vent;
- Effectuer une étude d'intégration d'énergies renouvelables et vérifier les aides financières disponibles.

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1. Introduction

Ressources PMET a mandaté BBA afin d'évaluer les options de génération d'électricité pour l'exploitation du projet minier Shaakichiuwaanaan, situé à 53 km du barrage hydroélectrique de La Grande-4 (LG4). La mise en service de la mine est prévue en 2029 pour une durée d'exploitation de 23 ans.

Les options à l'étude sont une interconnexion au poste de LG4 d'Hydro-Québec (HQ) comprenant notamment la construction d'une nouvelle ligne aérienne 120 kV de 53 km ainsi que la construction et l'opération d'une centrale autonome au diésel, au gaz naturel liquéfié (GNL) ou au gaz naturel comprimé (GNC), avec ou sans intégration d'énergies renouvelables, au site du complexe minier.

L'objectif de ce rapport est de présenter les résultats de l'analyse technicoéconomique et des émissions de gaz à effet de serre liées à chacune des options de génération d'électricité étudiées pour subvenir à la totalité des besoins électriques du complexe minier, le tout à haut niveau avec une précision classe V (-50 %/+100 %). Ces résultats permettent d'évaluer la viabilité et l'impact de chacune de ces options afin de ne cibler que les plus intéressantes dans le cadre de l'analyse de faisabilité du projet minier pour les aspects économique, social, environnemental et technique.

Ce rapport présente donc la méthodologie et les hypothèses utilisées ainsi que la description et l'analyse de tous les scénarios étudiés et les recommandations qui en découlent.

1.1. Abréviation et acronymes

Le tableau ci-dessous énumère certaines abréviations ou acronymes utilisés dans ce document et en donne la définition.

Tableau 3 : Abréviations et acronymes

Abréviation ou acronyme	Définition
BESS	De l'Anglais *Batterie Energy Storage System*, signifiant système de stockage à batterie
EPC	De l'Anglais *Engineering, Procurement and Construction*, signifiant Ingénierie, Approvisionnement et Construction, soit un type de contrat pour un « projet clé en main » à la charge du propriétaire
GES	Gaz à effet de serre

Abréviation ou acronyme	Définition
GNC	Gaz naturel comprimé
GNL	Gaz naturel liquéfié
HQ	Hydro-Québec
LCOE	De l'Anglais *Levelized Cost Of Energy*, signifiant cout actualisé de l'énergie, unité de mesure du cout moyen de l'énergie sur la durée de vie d'installations de production d'énergie électrique
VAN	Valeur actuelle nette

1.2. Étendue de l'étude

À la demande de Ressources PMET, les scénarios conceptuels suivants de génération d'électricité pour le site permanent de Shaakichiuwaanaan ont été analysés par BBA :

Tableau 4 : Description des scénarios

Scénario	Description
1	Raccordement au réseau d'Hydro-Québec (HQ) : interconnexion au réseau de HQ à partir du poste Tilly, près de la centrale hydroélectrique LG-4, par une ligne de 53 km à 120 kV
2A	Centrale autonome au diésel
2B	Centrale autonome hybride au diésel avec intégration d'énergie renouvelable avec un système de stockage à batterie (BESS)
3A	Centrale autonome au GNL
3B	Centrale autonome hybride au GNL avec intégration d'énergie renouvelable avec BESS
4A	Centrale autonome au GNC
4B	Centrale autonome hybride au GNC avec intégration d'énergie renouvelable avec BESS

Le scénario 1, soit celui de la ligne 120 kV, est considéré comme le scénario de base de cette étude, puisqu'il s'agit du scénario prévu au *Preliminary Economic Assessment* (PEA) (rapport BBA nº 7535006-000000-40-ERA-0002-R00). Il est à noter que la ligne haute tension du PEA était cependant à 69 kV, mais une tension de 120 kV est considérée dans la présente étude notamment en raison de l'augmentation de la charge.

2. Hypothèses

Intrants du client

Les intrants utilisés par BBA dans l'exécution de cette analyse consistent en l'information suivante fournie par Ressources PMET :

- Profil de charge électrique : le profil de charge présenté dans la figure suivante prend comme hypothèse une charge constante, tenant compte des facteurs d'utilisation et de disponibilité. Une charge moyenne de 34,55 MW est considérée pour les années d'exploitation régulière, soit de 2032 à 2048. Une augmentation graduelle de la charge entre 2029 et 2032, ainsi qu'une diminution graduelle entre 2048 et 2051 sont considérées;

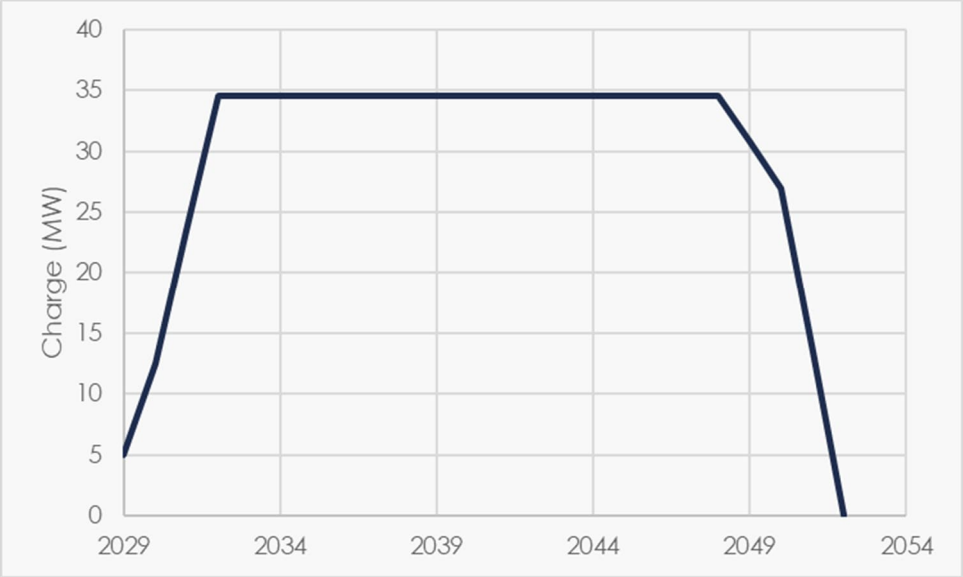


Figure 2 : Profil de charge électrique

- Consommation d'énergie électrique annuelle pour les années d'exploitation régulière : 302 658 000 kWh (calculée à partir du profil de charge);
- Le chauffage des galeries souterraines et des bâtiments principaux (usine de procédé et camp) sera effectué au GNC dans tous les cas de figure. Or, les scénarios de génération d'électricité par une centrale autonome aux hydrocarbures permettent la récupération d'énergie thermique issue de la production d'électricité utilisable pour chauffer les bâtiments principaux, réduisant les besoins en GNC pour cette application. Ainsi, concernant l'énergie thermique, seuls les couts d'OPEX liés au chauffage des bâtiments

principaux (usine de procédé et ses auxiliaires excluant le complexe du camp) seront évalués pour chaque scénario dans cette étude. La consommation de GNC nécessaire pour le chauffage épargné par ce système est donc représentée comme une économie dans les différents scénarios;

- Le CAPEX des installations requises pour le chauffage des bâtiments au gaz naturel comprimé n'est pas inclus dans les calculs, car il ne dépend pas du scénario de génération d'électricité choisi et sera requis dans chacun des cas comme opération principale ou en relève au système de récupération de chaleur;
- La charge thermique à considérer pour cette étude se base sur l'estimation fournie dans le rapport PEA. Le rapport indique que la dépense annuelle de propane pour le chauffage de l'usine de procédé et ses auxiliaires (*crusher*, garage et *paste plant*) sera de 2,5 M$. En considérant le prix du propane présenté au tableau 6, la charge a été estimée comme présentée dans le tableau 5 :

Tableau 5 : Détails des charges thermiques estimées des bâtiments procédés principaux

Secteur	Air chauffé à	Température la plus basse	Puissance
	°C	°C	kW
Usine de procédé	10	-35	**10 900**

- Durée de vie de la mine : 23 ans;
- Taux d'actualisation (pour calcul de la VAN et du LCOE) : 8 %;
- Taux d'inflation sur le prix du carburant : 0 %;
- Taux d'inflation sur les couts d'exploitation et de maintenance : 2 %;
- Type de contrat considéré : EPC;
- Prix des carburants :

Tableau 6 : Prix des carburants

Carburant	Unité	Prix	Hypothèse
Diésel (arctique)	$/L	1,43	Le prix du diésel est un intrant du client pour une unité stationnaire (2025-06)
Gaz naturel comprimé	$/GJ	38,17	Le prix du GNC est une estimation incluant le cout de molécules et le transport au site par Certarus basée sur le Heating fuel source – Trade off Study réalisée par G Mining : CASN-A-MC-TOS-Heating (2025-05)

Carburant	Unité	Prix	Hypothèse
Gaz naturel liquéfié	$/GJ	26,72	Le prix du GNL est une estimation incluant le cout de molécules et le transport au site transport au site basée sur le Heating fuel source – Trade off Study réalisée par G Mining : CASN-A-MC-TOS-Heating (2025-05)
Propane	$/L	0,58	Le prix du propane est une estimation basée sur un projet antérieur de récupération de chaleur (2024-12)

- Spécifications des carburants :

Tableau 7 : Spécifications des carburants

Carburant	Potentiel calorifique inférieur	Efficacité	
Diésel (arctique)	37 300 kJ/L	Génératrice	44 %
		Bouilloire	S. O.
Gaz naturel comprimé	37 890 kJ/m^3	Génératrice	44 %
		Bouilloire	92 %
Gaz naturel liquéfié	37 890 kJ/m^3	Génératrice	44 %
		Bouilloire	S. O.

[1]Condition de référence : 15°C et 101.325 kPa

Émission de GES

Pour les fins de cette étude, il a été déterminé que l'analyse d'impact environnemental de chaque scénario se limiterait à l'estimation des émissions de gaz à effet de serre liées à la production d'énergie électrique et thermique pendant la durée d'opération de la mine.

Pour chacun des scénarios, les émissions de GES ont été déterminées selon le volume de carburant (diésel et gaz naturel) consommé. Le tableau suivant présente les facteurs d'émissions de GES du diésel et du gaz naturel utilisés dans le cadre de cette étude.

Tableau 8 : Facteurs d'émission de GES pour le diésel et le gaz naturel

Carburant	GES	Facteur d'émission[1]	
		Valeur	Unité
Diésel	CO_2	2 681	kg/kL
	CH_4	0,133	kg/kL
	N_2O	0,400	kg/kL

Carburant	GES	Facteur d'émission[1]	
		Valeur	Unité
Gaz naturel	CO_2	1 911	g/m³
	CH_4	0,497	g/m³
	N_2O	0,0497	g/m³

[1] Selon Facteurs d'émission et de conversion, 16 aout 2019, Transition Énergétique Québec

Ensuite, le tableau suivant présente le potentiel de réchauffement global considéré dans cette étude pour chacun de ces GES.

Tableau 9 : Potentiel de réchauffement global des GES

GES	Potentiel de réchauffement global[1]
CO_2	1
CH_4	25
N_2O	298

[1] Note 1 : Selon la Loi sur la tarification de la pollution causée par les gaz à effet de serre (L.C. 2018, c.12, art. 186), annexe

En ce qui concerne le prix au Québec associé à ces émissions de GES, celui-ci est régi par le système de plafonnement et d'échange de droits d'émissions (SPEDE) qui est lié avec celui de la Californie, et est le fruit d'une vente aux enchères régulée par un prix plancher et un prix plafond ayant lieu quatre fois par année. Ainsi, le cout des émissions de GES dépend de la tendance du marché en suivant le principe de l'offre et de la demande. Depuis 2014, la tendance est que le prix de vente est supérieur aux prix minimaux projetés. Les prix minimaux projetés par le Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs sont présentés dans le tableau suivant.

Tableau 10 : Scénario potentiel du prix minimal annuel projeté par le ministère

2025	2026	2027	2028	2029	2030
32,34 $	36,60 $	34,03 $	36,61 $	42,39 $	45,35 $
Note 1 : Référence du SPEDE : Ventes aux enchères					

Pour les fins de cette étude, afin de simplifier les calculs, en raison de la nature spéculative du prix des droits d'émissions au Québec, un cout de 45 $/tonne de $CO_{2\,éq}$ sera utilisé sans inflation. Cela donnera un ordre de grandeur acceptable dans le contexte de cette étude, soit afin de comparer les scénarios de génération d'électricité du site de Shaakichiuwaanaan, mais ce cout

ne devrait pas être utilisé à d'autres fins en raison de son imprévisibilité (dépendance au marché, aux technologies, au changement de gouvernance, etc.). Ce droit d'émission s'applique uniquement si les émissions annuelles totales sont supérieures à 25 000 tonnes de $CO_{2\,éq}$. À partir de ce seuil, l'inscription au SPEDE est obligatoire. Si les émissions sont inférieures à ce seuil, le SPEDE ne sera pas pris en compte dans l'analyse du scénario.

3. Méthodologie

3.1. Interconnexion au réseau d'Hydro-Québec

Le scénario d'interconnexion au réseau d'Hydro-Québec à l'étude est celui impliquant une interconnexion au poste de LG4 d'Hydro-Québec (HQ).

Ce scénario de génération d'électricité implique comme travaux la construction d'un poste supplémentaire 315-120 kV relié au poste Tilly près de LG4 par une nouvelle ligne aérienne 315 kV de 2 km, un poste 120-13,8 kV au complexe minier et d'une ligne aérienne de transmission 120 kV de 53 km entre le poste de la mine et le point d'interconnexion au poste de LG4.

Les caractéristiques de la ligne 120 kV sont présentées dans le tableau suivant :

Tableau 11 : Caractéristiques de la ligne aérienne 120 kV

Caractéristique	Unité	Valeur
Point d'interconnexion		Poste de LG4
Tension	kV	120
Longueur approximative	km	53

3.1.1. Hypothèses financières

Les couts de la ligne électrique sont basés sur trois récents projets de BBA exécutés de 2020 à 2024. L'estimation de couts préliminaire de 2024 est considérée dans le cadre de l'étude.

Tableau 12 : Couts de la ligne électrique

Description	Quantité	Unité	Valeur unitaire k$	Total k$
Ligne 120 kV	53	km	1 500	79 500
Ligne 315 kV	2	km	4 000	8 000
Équipement supplémentaire à Tilly	1	-	12 500	12 500
Sous-station 315-120 kV	1	-	22 900	22 900
Sous-station 120-13,8 kV	1	-	15 000	15 000
			Contingence :	30 %
			Total :	**195 000**

3.1.2. Programme de Gestion de la demande de puissance (GDP)

Le projet de loi 69 du Québec exige maintenant que chaque nouvelle demande de raccordement au réseau d'Hydro-Québec de plus de 5 MW effectue une étude d'efficacité énergétique, démontre des efforts à implanter des mesures d'économies d'énergie, et s'engage à délester un minimum de 30% de sa puissance souscrite dans le cadre du programme GDP Engagement d'Hydro-Québec.

Le programme GDP pour les clients industriels se découpe en deux volets, soit Engagement et Latitude. Le volet Engagement est discuté ici puisqu'il s'agit du volet exigé par Hydro-Québec lors des nouveaux raccordements. Les modalités du programme Engagement ont été proposées dans le document « Condition de tarifs » d'HQ du 1er avril 2025, mais n'ont toujours pas été acceptées par la Régie de l'énergie. Les informations sont donc présentées à titre indicatif seulement.

Pour participer au programme, le client doit s'engager à délester une certaine quantité de puissance, appelée puissance interruptible. Celle-ci est utilisée, à l'aide du profil de consommation, afin d'établir une puissance de base à respecter lors des périodes de délestage. Chaque période de délestage est d'une durée de 4 ou 5 heures consécutives. Les plages horaires ne sont pas fixes, mais le premier événement sera typiquement le matin vers 6h00 et le deuxième événement sera typiquement en fin d'après-midi vers 16h.

Plusieurs choix sont offerts (20 sous-options au total) afin de déterminer le nombre de périodes de délestage annuelles à respecter, ainsi que le crédit applicable, comme suit:

- Heures annuelles : Cinq niveaux d'engagement sont possibles, variant de 20 à 100h annuelles.

- Nombre d'événements par jour : 1 ou 2 événements admissibles par jour. Si 1 événement est sélectionné, le délai minimal entre deux événements est de 12h. Si 2 événements sont sélectionnés, le délai minimal entre deux événements est de 4h.
- Jours de la semaine : Jours de semaine seulement, ou jours de fin de semaine inclus.

Le respect ou non du délestage effectué par le client lors des périodes GDP est confirmé à la fin de chaque mois d'hiver (décembre à mars) en fonction de la puissance maximale (PM) et de la puissance maximale quotidienne moyenne (PMQM) observées lors de chaque mois respectif. Chaque client est donc responsable d'assurer un délestage adéquat à chaque période GDP, sans aide d'HQ

Le crédit se découple en une portion fixe calculée sur la puissance interruptible, en $/kW, ainsi qu'une portion variable calculée sur l'énergie réelle totale délestée lors des périodes GDP, en $/kWh. Le versement des crédits est effectué sur chaque facture des mois d'hiver.

Des pénalités sont également applicables lorsqu'un client ne parvient pas à délester la puissance requise lors des périodes demandées.

Ainsi, le choix du niveau d'engagement doit être fait avec discernement, en s'assurant que les infrastructures et les équipements requis sont adéquats pour garantir l'atteinte du délestage prévu en tout temps. Il serait pertinent d'approfondir cette analyse dans de futures études afin de confirmer l'option la plus avantageuse.

3.2. Centrale autonome

3.2.1. Dimensionnement

Pour les fins de cette étude, BBA a dimensionné la centrale en tenant compte de la charge moyenne, de la charge de pointe et de la plus grosse charge motrice à démarrer, le tout lors de l'année d'opération ayant la charge la plus élevée. De plus, une configuration optimale et standard de ce type d'application de N+2 groupes électrogènes a été considérée, soit un nombre N de groupes en opération pour fournir à la demande, un groupe en réserve pour pallier la perte de l'un de ceux en opération, au besoin, et un groupe en maintenance. Des capacités de génératrices à moyenne vitesse de rotation connues pour ce type d'application ont été utilisées.

Cette configuration de centrale est employée pour chacun des scénarios de génération d'électricité par une centrale autonome considéré dans cette étude (diésel, GNL et GNC, avec

ou sans énergies renouvelables). En effet, les groupes électrogènes sélectionnés peuvent être carburés au diésel, au GNL ou au GNC moyennant quelques modifications à la centrale.

Les caractéristiques électriques de la centrale dimensionnée sont présentées dans le tableau suivant.

Tableau 13 : Caractéristiques des centrales thermiques

Description	Unité	Centrale diésel	Centrale gaz naturel liquide ou compressé
Capacité d'un groupe	kWe	9 388	9 370
Quantité (N+2)		7	7
Capacité installée de la centrale	kWe	65 716	65 590
Capacité en opération (N)	kWe	46 940	46 850

3.2.2. Hypothèses financières

Le CAPEX des génératrices sera similaire entre les différents types présentés dans cette étude. Les estimations ont été réalisées en utilisant des références provenant de projets précédents dans des conditions similaires et en utilisant les normes de l'industrie. L'estimation inclut le bâtiment et sa structure requise pour abriter les génératrices, les salles mécaniques et électriques, les infrastructures de stockage nécessaires et les auxiliaires des génératrices.

En ce qui concerne l'OPEX, celui-ci inclut les couts des carburants selon les prix présentés dans le Tableau 6, les couts d'exploitation et de maintenance des génératrices ainsi que les couts des droits d'émissions de gaz à effet de serre.

3.3. Énergie renouvelable

Dans le cadre de cette étude, les sources d'énergies renouvelables considérées sont le solaire photovoltaïque et l'éolien. Ces deux sources sont matures technologiquement, connues et éprouvées au Québec ainsi que relativement rapides à déployer et à plus faible cout que d'autres sources d'énergie renouvelable telle que l'hydroélectricité. Il est important de noter que l'analyse de ces deux sources d'énergie renouvelable est préliminaire. Une analyse future devra être réalisé pour confirmer le potentiel de ces sources d'énergie. Il existe plusieurs cas d'intégration d'énergie solaire et éolienne sur des sites miniers au Canada et à travers le monde. Le système de stockage d'énergie par batteries (BESS) est aussi discuté dans cette section en complément à l'énergie renouvelable.

Il est important de préciser que l'hydroélectricité n'a pas été prise en compte dans le cadre de cette étude, mais pourrait faire l'objet d'une analyse dans une phase ultérieure.

Les couts CAPEX et OPEX pour les énergies solaire et éolienne ainsi que pour le BESS ont été estimés en utilisant des références provenant de projets précédents dans des conditions similaires à celles de Shaakichiuwaanaan et des standards de l'industrie.

3.3.1. Énergie solaire

3.3.1.1. Solaire – Analyse de ressource

L'analyse de la ressource solaire a été réalisée à l'aide du logiciel spécialisé PVSyst™ selon les meilleures pratiques de l'industrie. Ce logiciel permet de modéliser et de simuler la puissance produite d'un parc solaire pour une année météorologique typique à une granularité d'une heure (8760 heures pour une année) et en tenant compte de pertes typiques. Les données météorologiques typiques basées sur les données historiques à la localisation 53°32'53"N 73°54'48"W ont été utilisées pour réaliser cette étude.

Un système de montage fixe est considéré dans cette étude, avec les modules orientés vers le sud pour maximiser la capture d'énergie du lever au coucher du soleil. Un angle d'inclinaison de 35° a été sélectionné pour optimiser la production d'énergie en tenant compte de la latitude du site et en minimisant l'accumulation de neige, tout en restant dans un angle standard de l'industrie.

Cette étude considère l'utilisation d'un système de fondation en béton ne nécessitant pas de profonde excavation.

Des modules photovoltaïques bifaciaux conçus pour capturer l'énergie solaire réfléchie par le sol ont été pris en compte dans les simulations. Étant donné la haute réflectivité de la neige, les panneaux bifaciaux sont particulièrement avantageux dans la région du projet. Les panneaux bifaciaux sont une conception standard de l'industrie. Des onduleurs de chaine seront utilisés en raison de la taille de la centrale, car ils sont généralement plus rentables pour les systèmes photovoltaïques d'une capacité inférieure à 5 MW par rapport aux onduleurs centraux.

Voici les résultats obtenus pour un parc solaire de 1 MW_{cc}, pouvant être considérés comme un bloc modulable au MW_{cc}.

3.3.1.2. Solaire – Résultats

Tableau 14 : Caractéristiques de l'énergie solaire

Description	Unité	Valeur
Paramètres		
Capacité installée	kW_{CC}	1 065
Capacité CA	kW_{CA}	960
Résultats énergétiques		
Production d'énergie nette (année 1)	kWh	1 417 000
Facteur de capacité	%	15
Résultats économiques		
CAPEX (estimation +50 %/-100 %)	k$	2 700
OPEX	k$/an	29
LCOE (23 ans)[1]	$/kWh	0,203

[1] Le LCOE est le cout actualisé de l'énergie représentant le cout moyen par kilowattheure d'électricité produite. Il est obtenu en divisant la somme des couts actualisés totaux (VAN) par la somme de la production d'énergie actualisée.

Le graphique suivant montre la production électrique par mois du parc solaire.

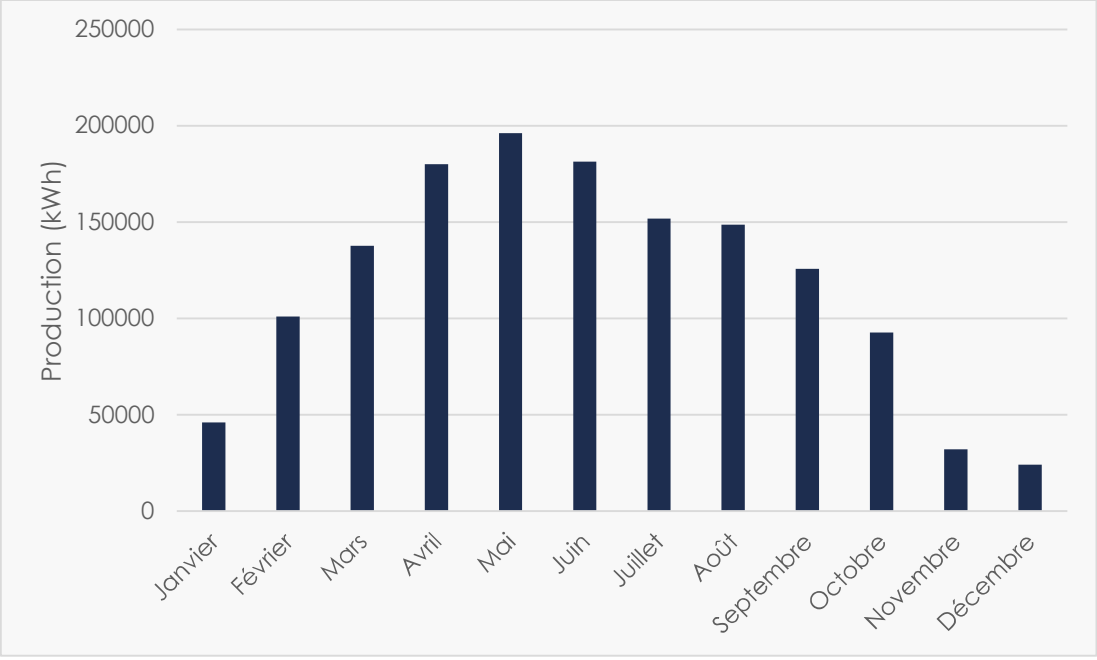


Figure 3 : Production d'énergie du parc solaire de 1 MWcc

Le rapport PVSyst complet est disponible à l'Annexe A du rapport.

3.3.1.3. Solaire – Discussion

L'analyse de la production des panneaux solaires par mois présentée sur la Figure 3 met en évidence une importante variabilité de la production au cours de l'année. En hiver, la production est particulièrement faible en raison du nombre d'heures d'ensoleillement. Cette limitation saisonnière entraine une baisse significative de la production annuelle d'énergie solaire.

De plus, l'espace occupé par un parc solaire d'une capacité de 1 MW est d'environ 4600 m^2. Une grande surface devrait donc être utilisée afin d'y installer un parc solaire d'une capacité suffisante pour décarboner une portion des opérations minières causant une déforestation supplémentaire et complexifiant l'installation de celui-ci.

Au niveau économique, l'utilisation d'énergie solaire présente un LCOE de 0,203 $/kWh. Ce dernier est supérieur à celui de l'énergie éolienne, qui est de 0,140 $/kWh comme présenté au Tableau 15, ce qui rend la technologie moins intéressante.

Comme l'énergie solaire et l'énergie éolienne remplissent la même fonction, soit de réduire la consommation de carburant, et que l'énergie solaire est plus dispendieuse et complexe à installer et moins stable au cours de l'année, elle ne sera pas considérée dans les différents scénarios analysés. Uniquement l'énergie éolienne sera utilisée dans les scénarios intégrant une forme d'énergie renouvelable.

3.3.1.4. Limites de l'étude solaire

Les limites de la présente étude sont les suivantes :

- Aucun emplacement précis de parc solaire n'a été utilisé. Si une étude plus détaillée était réalisée, une zone constructible devrait être sélectionnée, incluant les contraintes environnementales du terrain, afin d'évaluer le potentiel de capacité installée de façon plus précise. Un rapport géotechnique serait également requis pour valider le choix des fondations. L'emplacement final aura un impact sur l'évaluation du potentiel solaire et sur l'estimation des coûts.
- La disponibilité des données météorologiques pour le site est limitée, en particulier pour ce qui est de l'irradiance. Si une étude plus approfondie était menée, il serait recommandé d'acheter des données TMY satellite pour l'évaluation du potentiel solaire.

- Les valeurs du LCOE n'ont pas tenu compte d'un éventuel crédit d'impôt, d'une subvention ou de l'impact de la taxe carbone.

3.3.2. Énergie éolienne

3.3.2.1. Éolien – Analyse de ressource

L'analyse de la ressource éolienne a été réalisée en utilisant les données publiques à l'heure (8760) de ERA5[1] des cinq dernières années et la valeur moyenne de Global Wind Atlas[2]. Les coordonnées utilisées dans le cadre de l'étude sont les suivantes : 53°32'53"N 73°54'48"W. Le terrain identifié est fourni à titre indicatif et correspond à une vitesse de vent moyenne de la zone étudiée.

Plusieurs turbines ont été évaluées pour le projet. La sélection a été basée sur la production annuelle maximale d'une turbine selon leurs courbes de puissance et les données spécifiques du site. La turbine utilisée dans le cadre de cette étude est une turbine de 6,2 MW de classe S, conçue spécifiquement pour les zones de vent faible à moyen. Un facteur de perte de 20 % a été appliqué sur la puissance de sortie de la turbine. Aucune analyse de compatibilité de la turbine au site n'a été réalisée dans le cadre de cette étude.

3.3.2.2. Éolien – Résultats

Les résultats des simulations des analyses de ressource éolienne et économique sont présentés dans le tableau suivant.

Tableau 15 : Caractéristiques de l'énergie éolienne

Description	Unité	Valeur
Paramètres		
Capacité d'une turbine	kW	6,200
Résultats énergétiques		
Vitesse moyenne du vent (à 100 m)	m/s	7,5
Production d'énergie nette (année 1)	kWh	21 800 000
Facteur de capacité net	%	40
Résultats économiques		

[1] ERA5
[2] https://globalwindatlas.info/en/

CAPEX (estimé +50 %/-100 %)	k$	22 000
OPEX	k$/an	115
LCOE (23 ans)[1]	$/kWh	0,140

[1] Le LCOE est le cout actualisé de l'énergie représentant le cout moyen par kilowattheure d'électricité produite. Il est obtenu en divisant la somme des couts actualisés totaux (VAN) par la somme de la production d'énergie actualisée. Dans le cas de l'énergie éolienne, le LCOE est estimé en supposant que 100 % de la production renouvelable est utilisée. Cette estimation ne tient toutefois pas compte des couts liés au système de stockage par batteries (BESS), qui sont nécessaires à l'intégration des énergies renouvelables.

Certains couts CAPEX pourront être réduits lors de l'installation de plusieurs turbines, notamment ceux liés au réseau collecteur et à la location de la grue.

Le graphique suivant représente la production éolienne par mois.

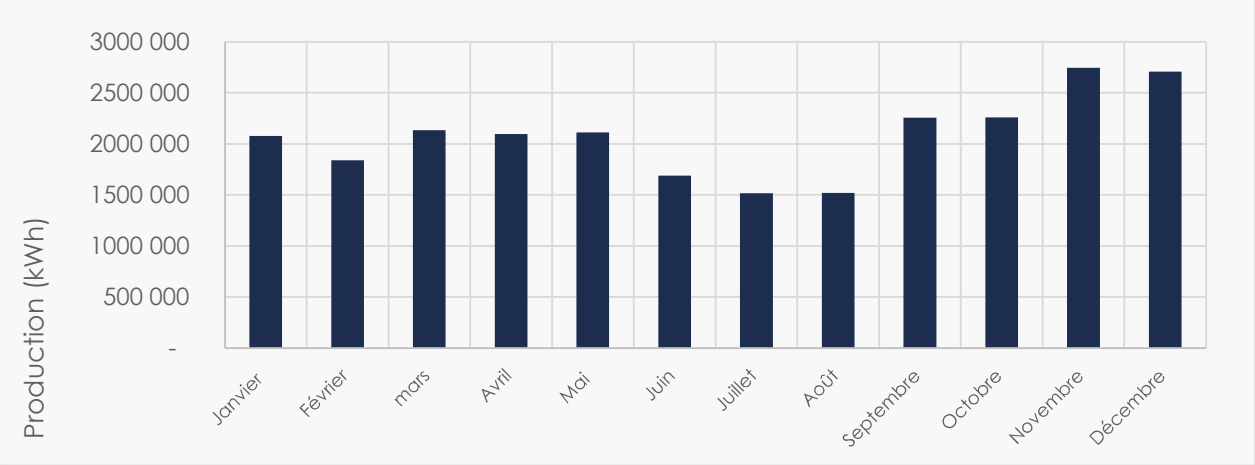


Figure 4 : Production d'énergie éolienne (kWh) par mois

3.3.2.3. Éolien – Discussion

Comme montré sur la Figure 4, la production électrique des éoliennes est relativement constante durant toute l'année et est même à son maximum l'hiver, ce qui est avantageux lorsque comparé à l'énergie solaire qui est très faible en hiver. De plus, l'espace nécessaire pour une éolienne est limité à la base de l'éolienne et aux chemins d'accès dédié aux maintenances ce qui correspond à environ 7000 m^2 pour une éolienne de 6,2 MW. Pour une capacité équivalente, l'empreinte sur le territoire est donc considérablement plus faible que pour l'énergie solaire.

D'un point de vue économique, l'installation d'éoliennes présente un LCOE de 0,140 $/kWh, ce qui est plus bas que celui des centrales thermiques dont le plus faible est de 0,319 $/kWh pour une centrale autonome au gaz naturel liquéfié et que l'énergie solaire dont le LCOE est de

0,203 $/kWh, augmentant ainsi la rentabilité du projet. Pour ces raisons, l'inclusion d'énergie renouvelable sera réalisée au moyen d'éoliennes pour les différents scénarios.

L'intégration d'éoliennes est intéressante, car elle permet de réduire grandement les couts d'exploitation et de maintenance en diminuant l'utilisation des génératrices et donc la consommation de carburant. Cela vient aussi augmenter la prévisibilité des couts d'exploitation comme le prix des carburants est sujet à davantage de fluctuation. Cependant, un investissement initial substantiel est nécessaire afin d'intégrer une quantité d'énergie renouvelable suffisante pour avoir un impact. Sur la durée de vie de la mine, les économies en OPEX permettent de rentabiliser le CAPEX supplémentaire nécessaire. Des subventions et des ententes contractuelles alternatives comme un *Power Purchase Agreement* (PPA) pourraient être intégrées pour réduire le CAPEX, mais elles n'ont pas été considérées dans cette étude.

3.3.2.4. Limites de l'étude

Les limites de cette étude sont les suivantes :

- Aucun emplacement exact pour les parcs éoliens n'a été utilisé. Il serait recommandé, à une étape ultérieure, de sélectionner une zone constructible, car cela aura un impact sur l'évaluation du potentiel éolien et l'estimation des coûts.
- La méthodologie utilisée dans le cadre de cette étude est appropriée à ce stade puisqu'il s'agit d'une analyse préliminaire. Cependant, il sera nécessaire d'obtenir des données de vent plus fiables si une étude plus poussée est envisagée.
- Des facteurs de pertes génériques et typiques ont été appliqués séparément à chaque donnée, ce qui est approprié à ce stade, mais ne donne pas un profil annuel représentatif. Des simulations utilisant un logiciel spécialisé devront être réalisées pour obtenir une estimation plus détaillée des pertes.
- La sélection du modèle d'éolienne a été réalisée à partir des données disponibles, mais aucune étude comparative des technologies afin d'identifier la turbine la plus adaptée au site n'a pas été réalisée, ce qui devrait être fait dans le cadre d'une étude ultérieure.
- Les valeurs du LCOE ne tiennent pas compte des crédits d'impôt potentiels, des subventions pour les projets d'énergies renouvelables ou de l'impact de la taxe carbone.

3.3.3. BESS

Il est recommandé d'avoir un système de stockage d'énergie par batterie (BESS) pour tout cas intégrant l'énergie éolienne à des fins de réserve tournante. Étant donné que la production d'énergie d'une éolienne peut chuter rapidement, un BESS est utilisé pour compenser ces baisses avant que des génératrices additionnelles puissent être démarrées dans le cas d'un réseau

autonome. La dimension du BESS pour la réserve tournante est généralement de 50 % à 100 % de la puissance maximale de l'éolienne avec une durée (capacité de stockage d'énergie) de 1 heure.

Un BESS pourrait aussi être utilisé pour accumuler l'excès d'énergie fournie par les éoliennes si la production est supérieure à la charge du site. Cependant, considérant la charge du site, il faudrait une grande quantité d'éoliennes pour avoir des excès d'énergie à certains moments (production éolienne supérieure à la charge), c'est pourquoi ce scénario n'a pas été analysé dans le cadre de l'étude.

3.3.4. Énergies renouvelables – Discussion

L'intermittence des énergies renouvelables, comme l'éolien ou le solaire, rend difficile l'alimentation continue d'une charge industrielle à partir d'un parc autonome. Ces sources doivent donc être couplées à un système de stockage d'énergie (BESS) ou à une source de production ferme, telle qu'une centrale thermique alimentée. Des périodes prolongées de faible production (vents faibles, faible ensoleillement, etc.) peuvent survenir, durant lesquelles ni le parc renouvelable, ni le système de stockage ne sont en mesure de répondre à la demande. Pour compenser entièrement la production thermique, il serait nécessaire de surdimensionner à la fois le parc renouvelable et le BESS, ce qui entrainerait des couts d'investissement (CAPEX) et d'exploitation (OPEX) très élevés, rendant cette solution peu viable. Afin d'assurer une alimentation continue de la charge, la présence d'une centrale thermique capable de répondre à l'ensemble de la demande, en cas de besoin, demeure essentielle. C'est pourquoi aucune réduction de la capacité de cette centrale n'a été envisagée dans le cadre de l'intégration des énergies renouvelables.

Afin de déterminer le nombre d'éoliennes à intégrer, plusieurs scénarios ont été analysés. Le graphique suivant présente l'impact de l'ajout d'éoliennes sur la VAN du projet selon les différents scénarios.

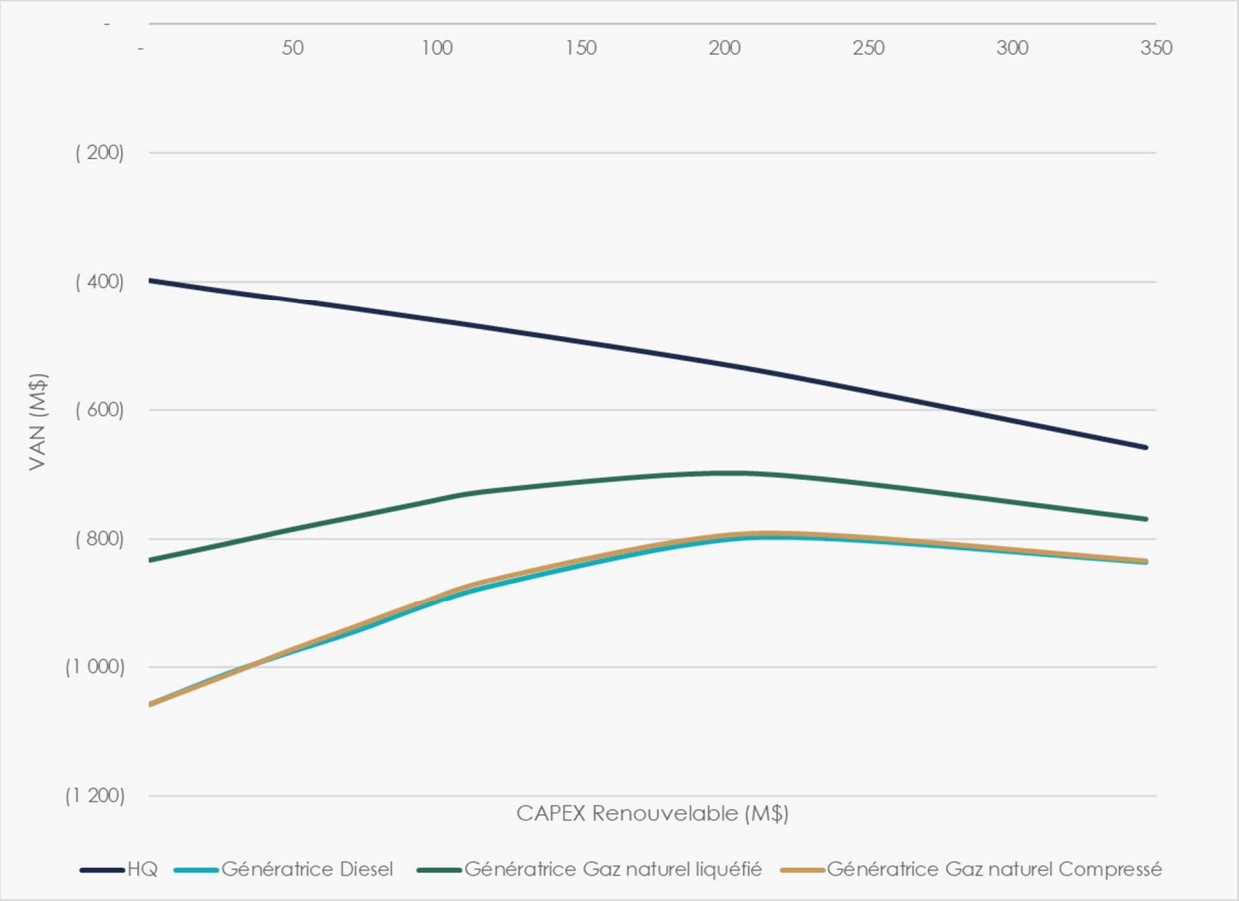


Figure 5 : Évolution de la VAN des différents scénarios selon l'investissement initial en énergie renouvelable

On y remarque que dans le cas d'une interconnexion avec Hydro-Québec, il n'est pas économiquement avantageux d'ajouter des éoliennes et un BESS comme la VAN diminue linéairement avec l'ajout de ceux-ci. Ainsi, aucun scénario combinant l'interconnexion avec HQ et l'énergie éolienne n'est considéré dans cette étude.

Pour les centrales thermiques, on remarque une amélioration de la VAN jusqu'à l'atteinte d'un point d'inflexion à partir duquel l'ajout d'une éolienne n'a plus d'avantage économique. Afin d'optimiser la VAN du projet tout en limitant les investissements CAPEX, l'analyse d'intégration d'énergie renouvelable sera réalisée en considérant l'installation de quatre éoliennes et d'un BESS de 50 % de la capacité des éoliennes dédié à la réserve tournante. Ce scénario permet d'être situé dans la région linéaire de la courbe tout en ayant un investissement initial supplémentaire de moins de 100 M$ pour l'intégration d'énergie renouvelable.

Bref, l'énergie éolienne est la seule énergie renouvelable considérée dans cette étude, mais seulement dans le contexte d'une centrale autonome au diésel ou au gaz naturel et en combinaison avec un BESS pour assurer une réserve « tournante ». Aucune réduction de la capacité de la centrale thermique n'a été envisagée dans le cadre de l'étude.

4. Résultats

4.1. Scénario 1 – Interconnexion avec Hydro-Québec

4.1.1. Description du scénario

Le scénario d'interconnexion au réseau d'Hydro-Québec à l'étude implique un raccordement au poste Tilly situé à proximité de la centrale LG4. Un nouveau poste 315-120 kV relié au poste Tilly par une nouvelle ligne aérienne 315 kV de 2 km serait nécessaire. De plus, une nouvelle ligne 120 kV de 53 km reliés au poste 120-13,8 kV du complexe minier devra aussi être construite.

Pour ce scénario, le chauffage des bâtiments principaux représentant annuellement 26 673 000 kWh_{th} est entièrement au GNC.

Avec cette configuration, voici les indices de performance du scénario 1.

Tableau 16 : Indices de performance de l'interconnexion au réseau d'Hydro-Québec

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (HQ)	kWh_e	302 658 000
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC pour le chauffage des bâtiments	m^3	2 755 000

4.1.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario, les tableaux de la section 2 et les émissions de GES fournies par HQ, les émissions de GES suivantes ont été calculées pour le scénario 1.

Tableau 17 : Émission de GES du scénario 1

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage des bâtiments (GNC)	2 755 000 m^3	63 355 000 m^3	5 300	123 000
Production d'électricité (HQ)	-	-	200	3 800
Total	**2 755 000 m^3**	**63 355 000 m^3**	**5 500**	**127 000**

Comme les émissions annuelles de ce scénario sont inférieures au seuil d'inscription obligatoire au SPEDE de 25 000 tonnes de $CO_{2éq}$, aucun droit d'émission ne devra être payé pour ce scénario.

4.1.3. Estimation des couts

Les couts initiaux (CAPEX) du scénario 1, soit une interconnexion avec le réseau d'Hydro-Québec sont basés sur les projets réalisés par BBA entre 2020 et 2024. Les prix utilisés sont ceux de 2024 et ils sont détaillés dans le tableau suivant :

Tableau 18 : Détail du CAPEX du scénario 1

Description	Quantité	Unité	Valeur unitaire [k$]	Total [k$]
Ligne 120 kV	53	km	1 500	79 500
Ligne 315 kV	2	km	4 000	8 000
Équipement supplémentaire à Tilly	1	-	12 500	12 500
Sous-station 315-120 kV	1	-	22 900	22 900
Sous-station 120-13,8 kV	1	-	15 000	15 000
			Contingence :	30 %
			Total :	**195 000**

Les couts opérationnels (OPEX) du scénario 1 sont basés sur le tarif L d'Hydro-Québec[3]. Ce tarif s'applique pour les clients industriels de grande puissance (plus de 5 000 kW) sur l'énergie totale consommée en kilowattheure (kWh) ainsi que sur la puissance de l'installation en kilowatt (kW). Certains crédits sont disponibles afin de réduire les couts de l'électricité. Les seuls ayant été considérés dans cette étude sont les crédits d'alimentation en moyenne ou haute tension étant donné que le scénario implique l'alimentation par une ligne de 120 kV et le crédit de rajustement pour pertes de transformation. Des économies supplémentaires n'ayant pas été analysées seraient donc possibles sous certaines conditions n'ayant pas été analysées dans cette étude.

Les couts d'exploitation liés au chauffage ne sont pas représentés dans cette section, mais plutôt considérés comme une économie dans le cas des centrales thermiques. Le détail des couts est présenté dans le tableau suivant :

[3] Tarif L d'Hydro-Québec

Tableau 19 : Couts d'opération du scénario 1

OPEX (description)	OPEX à la première année d'opération complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel de l'énergie	13 596 000 $	6 001 000 $
Cout annuel de la puissance	6 325 000 $	2 833 000 $
Total annuel	**19 921 000 $**	**8 834 000 $**
Total sur la durée de vie de la mine (VAN)	**203 176 000 $**	

Une indexation de 2,66 % par année a été appliquée sur les tarifs d'énergie et de puissance, ce qui correspond à la moyenne des augmentations des cinq dernières années.

Le tableau suivant présente le sommaire des couts du scénario 1, soit une interconnexion au réseau d'Hydro-Québec.

Tableau 20 : Sommaire des couts du scénario 1

Description	Unité	Valeur
CAPEX	$	195 000 000
OPEX moyenne actualisée annuelle	$	203 176 000
Total (VAN)	**$**	**(398 176 000)**
LCOE	$/kWh	0,153

4.2. Scénario 2A – Centrale autonome diésel

4.2.1. Description du scénario

Dans ce scénario, la génération d'électricité du complexe minier Shaakichiuwaanaan proviendra en totalité d'une centrale énergétique autonome composée de groupes électrogènes carburant au diésel.

Il est considéré dans cette étude qu'un système permettant de récupérer la chaleur dégagée par la production de l'énergie électrique est implanté afin de chauffer les bâtiments de procédé et auxiliaires. Cela représente un potentiel annuel de récupération de 226 994 000 kWh_{th}, soit environ 75 % de l'énergie électrique produite. Le chauffage des bâtiments est entièrement au GNC, ce qui représente annuellement 26 673 000 kWh_{th}, ce qui serait donc produit en totalité par la récupération de chaleur de la centrale. Toutefois, des opportunités de mitiger davantage les

besoins thermiques au site sont présentes si la ventilation souterraine et les bâtiments administratifs se rajoutent au réseau.

Avec cette configuration, voici les indices de performances clés calculés pour le scénario 2A.

Tableau 21 : Performance de la centrale autonome au diésel pour le scénario 2A

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (diésel)	kWh_e	302 658 000
Consommation de diésel annuelle maximale	L	64 648 000
Énergie thermique maximale produite par récupération de chaleur de la centrale	kWh_{th}	226 994 000
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC sauvée pour chauffage des bâtiments	m^3	2 755 000

4.2.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario et les tableaux de la section 2, les émissions de GES suivantes ont été calculées pour le scénario 2A.

Tableau 22 : Émissions de GES pour le scénario 2A

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage de bâtiment (GNC)[1]	957 000 m^3	980 000 m^3	1 855	1 899
Production d'électricité (diésel)	64 648 000 L	1 316 885 000 L	181 000	3 692 000
Total			**181 000**	**3 694 000**

[1] Le chauffage au GNC est uniquement utilisé pour les premières années lors du démarrage de la mine. Une fois à pleine capacité, les génératrices fournissent l'énergie thermique nécessaire pour le chauffage.

4.2.3. Estimation des couts

Le tableau suivant détaille les couts initiaux (CAPEX) du scénario 2A, soit une centrale autonome au diésel, sans énergies renouvelables.

Tableau 23 : Détail du CAPEX du scénario 2A

Description	Cout
Couts directs des installations (génératrices, contrôle, réseau de récupération de chaleur, bâtiments, fondation, etc.)	95 100 000 $
Cout de stockage de carburant (7 jours d'autonomie)	3 405 000 $
Cout du système de récupération de chaleur (tuyauterie, installation, etc.)	1 104 000 $
Couts indirects (transport, gestion de projet, ingénierie, essais, mise en service, etc.) – 40 %	39 844 000 $
Couts contingence – 25 %	34 863 000 $
Total	**174 315 000 $**

Le tableau suivant détaille les couts d'opération (OPEX) du scénario 2A.

Tableau 24 : Détail des couts d'opération OPEX du scénario 2A

OPEX (description)	OPEX à la première année d'exploitation complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel du diésel	92 446 000 $	34 853 000 $
Cout annuel d'O&M (incl. huile de lubrification)	5 314 000 $	2 284 000 $
Cout annuel des droits d'émission	8 156 000 $	3 075 000 $
Cout épargné de consommation de GNC pour le chauffage	(3 984 000) $	(1 740 000) $
Cout épargné des droits d'émission de consommation de GNC pour le chauffage	(240 000) $	(105 000) $
OPEX total	**101 692 000 $**	**38 367 000 $**
Total sur la durée de vie de la mine (VAN)	**882 448 000 $**	

Le tableau suivant présente le sommaire des couts pour le scénario 2A, soit une centrale autonome au diésel seulement

Tableau 25 : Sommaire des couts du scénario 2A

Description	Unité	Valeur
CAPEX	$	174 315 000
OPEX (diésel, O&M, huile, émissions) sur durée de vie de la mine (VAN)	$	882 448 000
Total (VAN)	**$**	**(1 056 763 000)**
LCOE	$/kWh	0,405

4.3. Scénario 2B – Centrale autonome diésel avec intégration d'énergies renouvelable

4.3.1. Description du scénario

Le scénario 2B comprend la même centrale énergétique au diésel que le scénario 2A avec l'intégration de quatre éoliennes de 6 200 kW chacune pour un total de 24 800 kW afin de réduire la consommation de carburants et d'un BESS de 12 400 kW avec une durée de 1 h pour la réserve tournante.

Comme pour le scénario 2A, un système de récupération de la chaleur dégagée par les groupes électrogènes est utilisé afin de chauffer les bâtiments de procédé et auxiliaires.

Avec cette configuration, voici les indices de performance clés calculés pour le scénario 2B.

Tableau 26 : Indices de performance du scénario 2B

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (diésel)	kWh_e	220 616 000
Consommation de diésel annuelle maximale	L	47 994 000
Production d'énergie électrique annuelle maximale (éolien)	kWh	82 042 000
Pénétration renouvelable totale	%	27,1
Déplacement de carburant (diésel) annuel maximal	L	(16 654 000)
Énergie thermique maximale produite par récupération de chaleur de la centrale	kWh_{th}	165 462 000

Indice de performance	Unité	Valeur
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC épargnée pour chauffage des bâtiments	m^3	2 755 000

4.3.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario et les tableaux de la section 2, les émissions de GES suivantes ont été calculées pour le scénario 2B.

Tableau 27 : Émissions de GES du scénario 2B

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage de bâtiment (GNC)[1]	1 583 000 m^3	3 049 000 m^3	3 000	5 900
Production d'électricité (diésel)	47 994 000 L	955 278 000 L	135 000	2 678 000
Production d'électricité (éolien)	0	0	1 200	25 000
Total			**136 000**	**2 709 000**

[1] Le chauffage au GNC est uniquement utilisé pour les premières et dernières années d'exploitation de la mine. À pleine capacité, les génératrices fournissent l'énergie thermique nécessaire pour le chauffage.

4.3.3. Estimation des couts

Le tableau suivant détaille les couts initiaux (CAPEX) du scénario 2B, soit une centrale autonome au diésel avec intégration d'énergie éolienne.

Tableau 28 : Estimation du CAPEX du scénario 2B

Description	Cout
Centrale autonome au diésel	174 315 000 $
Parc éolien 24,8 MW	78 130 000 $
BESS 12,4 MW/12,4MWh	16 876 000 $
Total	**269 322 000 $**

Le tableau suivant détaille les couts d'opération (OPEX) du scénario 2B.

Tableau 29 : Couts d'opération du scénario 2B

OPEX (description)	OPEX à la première année d'opération complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel du diésel	68 631 000 $	25 194 000 $
Cout annuel d'O&M (incl. huile de lubrification)	4 019 000 $	1 707 000 $
Cout d'entretien annuel des éoliennes	539 000 $	269 000 $
Cout d'entretien annuel du BESS	28 000 $	14 000 $
Cout annuel des droits d'émission	6 106 000 $	2 206 000 $
Cout épargné de consommation de GNC pour le chauffage	(3 984 000) $	(1 654 000) $
Cout épargné des droits d'émission de consommation de GNC pour le chauffage	(240 000) $	(100 000) $
OPEX total	**75 100 000 $**	**27 638 000 $**
Total sur durée de vie de la mine (VAN)	**635 668 000 $**	

Le tableau suivant présente le sommaire des couts d'énergie du scénario 2B, soit une centrale autonome au diésel avec intégration d'énergie éolienne.

Tableau 30 : Sommaire des couts du scénario 2B

Description	Unité	Valeur
CAPEX	$	269 751 000
OPEX (diésel, O&M, huile, émissions) sur durée de vie de la mine (VAN)	$	635 668 000
Total (VAN)	**$**	**(904 989 000)**
LCOE	$/kWh	0,347

4.4. Scénario 3A – Centrale autonome au gaz naturel liquéfié

4.4.1. Description du scénario

Dans ce scénario, la génération d'électricité du complexe minier Shaakichiuwaanaan proviendra en totalité d'une centrale énergétique autonome composée de groupes électrogènes carburant au gaz naturel dont le transport jusqu'au site est effectué sous forme liquide.

Les caractéristiques électriques et la configuration de la centrale sont les mêmes que dans le scénario 2A. Toutefois, des couts initiaux et des couts de consommation de carburant différents sont à prévoir, notamment pour l'installation des équipements requis au site pour l'entreposage et la vaporisation du gaz naturel.

Avec cette configuration, voici les indices de performances clés calculés pour le scénario 3A :

Tableau 31 : Indices de performance du scénario 3A

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (gaz naturel)	kWh_e	302 658 000
Consommation de gaz naturel annuelle maximale	m^3	65 910 000
Énergie thermique maximale produite par récupération de chaleur de la centrale	kWh_{th}	226 994 000
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC sauvée pour chauffage des bâtiments	m^3	2 755 000

4.4.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario et les tableaux de la section 2, les émissions de GES suivantes ont été calculées pour le scénario 2A

Tableau 32 : Émission de GES du scénario 3A

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage de bâtiment (GNC)[1]	957 000 m^3	980 000 m^3	1 900	1 900
Production d'électricité (gaz naturel)	65 910 000 m^3	1 337 494 000 m^3	128 000	2 592 000
Total	**65 910 000m^3**	**1 338 474 000 m^3**	**128 000**	**2 594 000**

[1] Le chauffage au GNC est uniquement utilisé pour les premières années lors du démarrage de la mine. Une fois à pleine capacité, les génératrices fournissent l'énergie thermique nécessaire pour le chauffage des bâtiments principaux.

4.4.3. Estimation des couts

Le tableau suivant détaille les couts initiaux (CAPEX) du scénario 3A, soit une centrale autonome au gaz naturel avec une alimentation en gaz naturel liquéfié, sans énergies renouvelables.

Tableau 33 : Estimation du CAPEX du scénario 3A

Description	Cout
Couts directs des installations (génératrices, contrôle, réseau de récupération de chaleur, bâtiments, fondation, etc.)	95 100 000 $
Cout système de récupération de chaleur (tuyauterie, installation, etc.)	1 104 000 $
Couts indirects (transport, gestion de projet, ingénierie, essais, mise en service, etc.) – 40 %	38 482 000 $
Couts contingence – 25 %	33 671 000 $
Couts des infrastructures de gestion du GNL (dalle de béton, réservoirs, vaporisation, contrôle, tuyauterie, etc.)	28 327 000 $
Total	**196 683 000 $**

Le tableau suivant détaille les couts d'opération (OPEX) du scénario 3A

Tableau 34 : Couts d'opération du scénario 3A

OPEX (description)	OPEX à la première année d'opération complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel du GNL	66 739 000 $	25 051 000 $
Cout annuel d'O&M (incl. huile de lubrification)	5 334 000 $	2 291 000 $
Cout annuel des droits d'émission	5 747 000 $	2 122 000 $
Cout épargné de consommation de GNC pour le chauffage	(3 984 000) $	(1 740 000) $
Cout épargné des droits d'émission de consommation de GNC pour le chauffage	(240 000) $	(105 000) $
OPEX total	**73 596 000 $**	**27 620 000 $**
Total sur la durée de vie de la mine (VAN)	**635 253 000 $**	

Le tableau suivant présente le sommaire des couts de l'énergie pour le scénario 3A, soit une centrale autonome au GNL seulement

Tableau 35 : Sommaire des couts du scénario 3A

Description	Unité	Valeur
CAPEX	$	196 683 000
OPEX (LNG, O&M, huile, émissions) sur durée de vie de la mine (VAN)	$	635 253 000
Total (VAN)	**$**	**(831 936 000)**
LCOE	$/kWh	0,319

4.5. Scénario 3B – Centrale autonome au gaz naturel liquéfié avec intégration d'énergies renouvelable

4.5.1. Description du scénario

Le scénario 3B comprend la même centrale énergétique au gaz naturel liquéfié que le scénario 3A, mais avec l'intégration d'énergie renouvelable afin de réduire la consommation de carburant. L'intégration des énergies renouvelables est faite au moyen de quatre éoliennes de 6 200 kW chacune pour un total de 24 800 kW et d'un BESS de 12 400 kW avec une durée de 1 h.

Comme pour le scénario 3A, un système de récupération de la chaleur dégagée par les groupes électrogènes est utilisé afin de chauffer les bâtiments de procédé et auxiliaires.

Avec cette configuration, voici les indices de performances clés calculés pour le scénario 3B.

Tableau 36 : Indices de performance du scénario 3B

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (gaz naturel liquéfié)	kWh_e	220 616 000
Consommation de gaz naturel annuelle maximale	m^3	48 284 000
Production d'énergie électrique annuelle maximale (éolien)	kWh	82 042 000
Pénétration renouvelable totale	%	27,1
Déplacement de carburant (gaz naturel liquéfié) annuel maximal	m^3	(17 626 000)
Énergie thermique maximale produite par récupération de chaleur de la centrale	kWh_{th}	165 462 000
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC sauvée pour chauffage des bâtiments	m^3	2 755 000

4.5.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario et les tableaux de la section 2, les émissions de GES suivantes ont été calculées pour le scénario 3B.

Tableau 37 : Émissions de GES du scénario 3B

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage de bâtiment (GNC)[1]	1 585 000 m^3	3 054 000 m^3	3 000	5 900
Production d'électricité (gaz naturel liquéfié)	48 284 000 m^3	957 334 000 m^3	94 000	1 855 000
Production d'électricité (éolien)	0 L	0 L	1 200	25 000
Total	**48 284 000 m^3**	**960 388 000 m^3**	**95 000**	**1 886 000**

[1] Le chauffage au GNC est uniquement utilisé pour les premières années lors du démarrage de la mine. Une fois à pleine capacité, les génératrices fournissent l'énergie thermique nécessaire pour le chauffage.

4.5.3. Estimation des couts

Le tableau suivant détaille les couts initiaux (CAPEX) du scénario 3B, soit une centrale autonome au gaz naturel liquéfié avec intégration d'énergie éolienne.

Tableau 38 : Estimation du CAPEX du scénario 3B

Description	Cout
Centrale autonome au gaz naturel liquéfié	196 683 000 $
Parc éolien 24,8 MW	78 130 000 $
BESS 12,4 MW/12,4MWh	16 876 000 $
Total	**291 690 000 $**

Le tableau suivant détaille les couts d'opération (OPEX) du scénario 3B.

Tableau 39 : Couts d'opération du scénario 3B

OPEX (description)	OPEX à la première année d'opération complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel du GNL	48 891 000 $	17 849 000 $
Cout annuel d'O&M (incl. huile de lubrification)	4 027 000 $	1 711 000 $
Cout d'entretien annuel des éoliennes	539 000 $	269 000 $
Cout d'entretien annuel du BESS	28 000 $	14 000 $
Cout annuel des droits d'émission	4 262 000 $	1 535 000 $
Cout sauvé de consommation de GNC pour le chauffage	(3 984 000) $+	(1 654 000) $
Cout sauvé des droits d'émission de consommation de GNC pour le chauffage	(240 000) $	(100 000) $
OPEX total	**53 523 000 $**	**19 625 000 $**
Total sur durée de vie de la mine (VAN)	**451 381 000 $**	

Le tableau suivant présente le sommaire des couts d'énergie du scénario 3B, soit une centrale autonome au gaz naturel liquéfié avec intégration d'énergie éolienne.

Tableau 40 : Sommaire des couts du scénario 3B

Description	Unité	Valeur
CAPEX	$	291 690 000
OPEX (diésel, O&M, huile, émissions) sur durée de vie de la mine (VAN)	$	451 381 000
Total (VAN)	**$**	**(743 071 000)**
LCOE	$/kWh	0,285

4.6. Scénario 4A – Centrale autonome au gaz naturel compressé

4.6.1. Description du scénario

Dans ce scénario, la génération d'électricité du complexe minier Shaakichiuwaanaan proviendra en totalité d'une centrale énergétique autonome composée de groupes électrogènes carburant au gaz naturel dont le transport jusqu'au site est effectué sous forme de gaz naturel compressé.

Les caractéristiques électriques et la configuration de la centrale sont les mêmes que dans le scénario 3A. Toutefois, des couts initiaux et des couts de consommation de carburant différents sont à prévoir comme le gaz naturel arrive au site dans un état différent. Le CAPEX inclut donc le cout des génératrices, du bâtiment de la centrale, des équipements de dépressurisation, de la tuyauterie de raccordement aux remorques et l'espace requis pour l'entreposage des 120 remorques nécessaires afin d'avoir une autonomie de 7 jours de consommation d'énergie.

Tableau 41 : Indices de performance du scénario 4A

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (gaz naturel compressé)	kWh_e	302 658 000
Consommation de gaz naturel annuelle maximale	m^3	65 910 000
Énergie thermique maximale produite par récupération de chaleur de la centrale	kWh_{th}	226 994 000
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC épargnée pour chauffage des bâtiments	m^3	2 755 000

4.6.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario et les tableaux de la section 2, les émissions de GES suivantes ont été calculées pour le scénario 4A.

Tableau 42 : Émissions de GES du scénario 4A

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage de bâtiment (GNC)[1]	957 000 m^3	980 000 m^3	1 900	1900
Production d'électricité (GNC)	65 910 000 m^3	1 337 494 000 m^3	128 000	2 592 000
Total	**65 910 000 m^3**	**1 338 474 000 m^3**	**128 000**	**2 594 000**

[1] Le chauffage au GNC est uniquement utilisé pour les premières années lors du démarrage de la mine. Une fois à pleine capacité, les génératrices fournissent l'énergie thermique nécessaire pour le chauffage.

4.6.3. Estimation des couts

Le tableau suivant détaille les couts initiaux (CAPEX) du scénario 4A, soit une centrale autonome au gaz naturel avec une alimentation en gaz naturel compressé, sans énergies renouvelables.

Tableau 43 : Estimation du CAPEX du scénario 4A

Description	Cout
Couts directs des installations (génératrices, contrôle, réseau de récupération de chaleur, bâtiments, fondation, etc.)	95 100 000 $
Cout d'infrastructure d'entreposage et alimentation de gaz comprimé (tuyauterie, fondation, civil, etc.) – 7 jours d'autonomie	4 000 000 $
Cout système de récupération de chaleur (tuyauterie, installation, etc.)	1 104 000 $
Couts des infrastructures du fournisseur[1]	218 000 $
Couts indirects (transport, gestion de projet, ingénierie, essais, mise en service, etc.) – 40 %	40 082 000 $
Couts contingence – 25 %	35 071 000 $
Total	**175 575 000 $**

[1] Cette estimation est basée sur les couts en CAPEX de Certarus fournis dans le rapport *Heating Energy Source Trade Off Study* réalisé par G-Mining Services (CASN-A-MC-TOS-Heating) pour le complexe minier Shaakichiuwaanaan .

Le cout d'infrastructure supplémentaire réfère principalement pour l'emplacement nécessaire pour le dépôt des remorques de gaz comprimé. Selon le rapport de G-Mining titré Heating Energy Source Trade-Off Study (CASN-A-MC-TOS-Heating), les remorques agiront comme transport principal du carburant sur site et de stockage lors de l'opération des génératrices. Selon les besoins énergétiques au site et en supposant une taille de conteneur de 51 m^3, ceci représente un besoin de 17 remorques par jour. Si nous voulons une autonomie de 7 jours comme décrite dans le scénario 2A, 120 remorques devront être présentes simultanément au site. Le cout d'infrastructure reflète la tuyauterie, la fondation et toutes autres installations nécessaires pour supporter ce besoin.

Il est à noter que les risques opérationnels ne sont pas reflétés dans les couts et devront être considérés durant la sélection d'un scénario.

Le tableau suivant détaille les couts d'exploitation (OPEX) du scénario 4A.

Tableau 44 : Couts opérationnels du scénario 4A

OPEX (description)	OPEX à la première année d'opération complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel du GNC	95 322 000 $	35 780 000 $
Cout annuel d'O&M (incl. huile de lubrification)	5 334 000 $	2 291 000 $
Cout annuel des droits d'émission	5 747 000 $	2 122 000 $
Cout sauvé de consommation de GNC de chauffage	(3 984 000) $	(1 740 000) $
Cout sauvé des droits d'émission de consommation de GNC de chauffage	(240 000) $	(105 000) $
OPEX total	**102 180 000 $**	**38 349 000 $**
Total sur durée de vie de la mine (VAN)	**882 022 000 $**	

Le tableau suivant présente le sommaire des couts de l'énergie pour le scénario 4A, soit une centrale autonome au GNC seulement

Tableau 45 : Sommaire des couts du scénario 4A

Description	Unité	Valeur
CAPEX	$	175 575 000
OPEX (GNC, O&M, huile, émissions) sur durée de vie de la mine (VAN)	$	882 022 000
Total (VAN)	**$**	**(1 057 596 000)**
LCOE	$/kWh	0,406

4.7. Scénario 4B – Centrale autonome au gaz naturel compressé avec intégration d'énergie renouvelable

4.7.1. Description du scénario

Le scénario 4B comprend la même centrale énergétique au gaz naturel compressé que le scénario 4A, mais avec l'intégration d'énergie renouvelable afin de réduire la consommation de carburant. L'intégration des énergies renouvelables est faite au moyen de quatre éoliennes de 6 200 kW chacune pour un total de 24 800 kW et d'un BESS de 12 400 kW avec une durée de 1 h.

Comme pour le scénario 4A, un système de récupération de la chaleur dégagée par les groupes électrogènes est utilisé afin de chauffer les bâtiments de procédé et auxiliaires.

Avec cette configuration, voici les indices de performances clés calculés pour le scénario 4B.

Tableau 46 : Indices de performance du scénario 4B

Indice de performance	Unité	Valeur
Production d'énergie électrique annuelle maximale (gaz naturel compressé)	kWh_e	220 616 000
Consommation de gaz naturel annuelle maximale	m^3	48 284 000
Production d'énergie électrique annuelle maximale (éolien)	kWh	82 042 000
Pénétration renouvelable totale	%	27,1
Déplacement de carburant (GNC) annuel maximal	m^3	(17 626 000)

Indice de performance	Unité	Valeur
Énergie thermique maximale produite par récupération de chaleur de la centrale	kWh_{th}	165 462 000
Énergie thermique maximale nécessaire pour le chauffage des bâtiments	kWh_{th}	26 673 000
Consommation maximale annuelle de GNC sauvée pour chauffage des bâtiments	m^3	2 755 000

4.7.2. Émission de GES

En utilisant la consommation de carburant moyenne calculée pour ce scénario et les tableaux de la section 2, les émissions de GES suivantes ont été calculées pour le scénario 4B.

Tableau 47 : Émissions de GES du scénario 4B

Source des émissions	Consommation de carburant		Émissions de GES [Tonnes $CO_{2.equivalent}$]	
	Maximale annuelle	Total (23 ans)	Maximale annuelle	Total (23 ans)
Chauffage de bâtiment (GNC)[1]	1 585 000 m^3	3 054 000 m^3	3 000	5 900
Production d'électricité (GNC)	48 284 000 m^3	957 334 000 m^3	95 000	1 855 000
Production d'électricité (éolien)	0 L	0 L	1 200	25 000
Total	**48 284 000 m^3**	**960 388 000 m^3**	**95 000**	**1 886 000**

[1] Le chauffage au GNC est uniquement utilisé pour les premières années lors du démarrage de la mine. Une fois à pleine capacité, les génératrices fournissent l'énergie thermique nécessaire pour le chauffage.

4.7.3. Estimation des couts

Le tableau suivant détaille les couts initiaux (CAPEX) du scénario 4B, soit une centrale autonome au gaz naturel compressé avec intégration d'énergie éolienne.

Tableau 48 : Estimation du CAPEX du scénario 4B

Description	Cout
Centrale autonome au gaz naturel compressé	175 575 000 $
Parc éolien 24,8 MW	78 130 000 $
BESS 12,4 MW/12,4MWh	16 876 000 $
Total	**270 581 000 $**

Le tableau suivant détaille les couts d'opération (OPEX) du scénario 4B.

Tableau 49 : Couts opérationnels du scénario 4B

OPEX (description)	OPEX à la première année d'opération complète (2032)	Moyenne annuelle actualisée sur la durée de vie de la mine (23 ans)
Cout annuel du GNC	69 831 000 $	25 494 000 $
Cout annuel d'O&M (incl. huile de lubrification)	4 027 000 $	1 711 000 $
Cout d'entretien annuel des éoliennes	539 000 $	269 000 $
Cout d'entretien annuel du BESS	29 000 $	14 000 $
Cout annuel des droits d'émission	4 262 000 $	1 535 000 $
Cout sauvé de consommation de GNC pour le chauffage	(3 984 000) $	(1 654 000) $
Cout sauvé des droits d'émission de consommation de GNC pour le chauffage	(240 000) $	(100 000) $
OPEX total	**74 463 000 $**	**27 270 000 $**
Total sur durée de vie de la mine (VAN)	**627 208 000 $**	

Le tableau suivant présente le sommaire des couts d'énergie du scénario 3B, soit une centrale autonome au gaz naturel compressé avec intégration d'énergie éolienne.

Tableau 50 : Sommaire des couts du scénario 4B

Description	Unité	Valeur
CAPEX	$	270 581 000
OPEX (GNC, O&M, huile, émissions) sur durée de vie de la mine (VAN)	$	627 208 000
Total (VAN)	**$**	**(897 788 000)**
LCOE	$/kWh	0,344

5. Analyse

Afin d'effectuer une comparaison équitable des différents scénarios étudiés, soit en prenant compte des différents aspects critiques du projet, une matrice décisionnelle a été réalisée. Cette matrice séparée en quatre sous-sections soit, économique, environnement, social et technique, comprend les critères critiques au projet ainsi que leur pondération respective. Une valeur est ensuite attribuée à chaque critère pour chacun des scénarios en fonction des résultats de l'étude de la manière suivante :

1. Faible
2. Moyen
3. Élevé

Pour chaque critère, un pointage est attribué à chaque scénario en multipliant le poids du critère par la valeur associée. Le score final d'un scénario utilisé afin de les comparer entre eux est obtenu en effectuant la somme des pointages de chaque critère. Plus la somme est élevée, le meilleur est le résultat du scénario.

Le tableau ci-dessous présente la description des critères ainsi que leur pondération établie en collaboration avec Ressources PMET.

Tableau 51 : Description et poids des critères de la matrice décisionnelle

Critère	Description	Poids
Impact Économique		**10**
CAPEX	L'investissement initial doit être le plus faible possible	5
VAN	La VAN doit être la plus élevée possible	3
Prévisibilité	Fluctuation possible des couts d'opération en raison du prix des carburants	2
Impact environnemental		**10**
Émission de GES	La quantité d'émissions de GES doit être la plus faible possible	4
Impact sur les plans d'eau et les milieux humides	Perturbation de la qualité de l'eau et destruction ou perturbation de milieux humides pour la construction des installations. Moins les milieux sont affectés, plus le score est élevé	2
Impact sur la faune	Atténuation des effets sur la biodiversité et les habitats naturels.	2
Empreinte sur le territoire	Limitation de la superficie supplémentaire occupée par les installations de production et de transport d'électricité	2
Impact social		**10**
Circulation	Limitation de la circulation de camions nécessaires pour le transport de carburant pour la génération d'électricité et le chauffage	3
Retombée économique pour la région	Création potentielle d'emplois locaux et de contrats pour des entreprises locales	3
Autres utilisateurs possibles	Possibilité d'utiliser les infrastructures électriques pour une autre raison que l'alimentation de la mine	2
Nuisance	Impacts potentiels en termes de bruits, de poussière et d'intégration paysagère	2
Impact technique		**3**
Constructibilité et délais	Complexité de la construction et respect de l'échéancier de la mine	2
Fiabilité électrique	Fréquence des défauts, temps de réponse lors de la perte d'une génératrice, nombre d'étapes avant la production électrique	1
Total		**33**

La matrice décisionnelle obtenue grâce à cette méthode et aux résultats de cette étude est la suivante :

Tableau 52 : Matrice décisionnelle

Critère	Poids	1 – Ligne H.T.		2A – Diésel		2B – Diésel + renouvelable		3A – GNL		3B – GNL + renouvelable		4A – GNC		4B – GNC + renouvelable	
		Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage	Valeur	Pointage
Impact économique															
CAPEX	5	2	10	3	15	1	5	2	10	1	5	3	15	1	5
VAN	3	3	9	1	3	1	3	2	6	2	6	1	3	1	3
Prévisibilité	2	3	6	1	2	2	4	1	2	2	4	1	2	2	4
Sous-total	**10**	**25**		**20**		**12**		**18**		**15**		**20**		**12**	
Impact environnemental															
Émissions GES	4	3	12	1	4	2	8	2	8	2	8	2	8	2	8
Impact sur les plans d'eau et les milieux humides	2	1	2	3	6	2	4	3	6	2	4	3	6	2	4
Impact sur la faune	2	2	4	3	6	1	2	3	6	1	2	3	6	1	2
Empreinte sur le territoire	2	1	2	3	6	2	4	3	6	2	4	2	4	2	4
Sous-total	**10**	**20**		**22**		**18**		**26**		**18**		**24**		**18**	
Impact social															
Retombé économique	3	2	6	1	3	1	3	1	3	1	3	1	3	1	3
Circulation	3	3	9	1	3	2	6	1	3	2	6	1	3	2	6
Autres utilisateurs possibles	2	3	6	1	2	1	2	1	2	1	2	1	2	1	2
Nuisance (bruit, poussière, paysage)	2	3	6	2	4	1	2	2	4	1	2	2	4	1	2
Sous-total	**10**	**27**		**12**		**13**		**12**		**13**		**12**		**13**	
Impact technique															
Constructibi-lité et délai	2	3	6	3	6	2	4	3	6	2	4	2	4	2	4
Fiabilité électrique	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Sous-total	**3**	**8**		**8**		**6**		**8**		**6**		**6**		**6**	
Total	**33**	**80**		**62**		**49**		**64**		**52**		**62**		**49**	

L'explication de chacune des valeurs dans la matrice se trouve à l'Annexe B.

Selon les résultats de la matrice, le scénario 1 (interconnexion avec le réseau d'Hydro-Québec) serait le plus avantageux étant donné qu'il obtient le meilleur pointage pondéré. Il présente les meilleurs résultats économiques et sociaux de tous les scénarios. Cependant, afin que ce scénario soit réalisé, il est nécessaire d'obtenir une entente avec Hydro-Québec pour l'accès au bloc d'énergie nécessaire. Il est à noter que si ce scénario est sélectionné, une adhésion au programme de gestion de la demande de puissance (GDP) d'Hydro-Québec est requises. Advenant le cas où une telle entente ne serait pas possible, une centrale autonome devra être construite.

Le meilleur scénario de centrale autonome serait celui au gaz naturel liquéfié comme son pointage final est le plus élevé. Malgré un investissement initial plus élevé d'environ 22 M$, l'option du gaz naturel liquéfié est beaucoup plus intéressante étant donné qu'en opération complète, les couts du carburant sont inférieurs d'environ 26 M$. Le retour sur l'investissement est donc rapide.

La centrale autonome au diésel n'est pas intéressante, car elle émet davantage de gaz à effet de serre et présente une VAN moins bonne que la centrale au GNL pour un CAPEX inférieur et des impacts sociaux et environnementaux pratiquement identiques.

La centrale autonome de gaz comprimé est le moins favorable des options de centrale autonome en raison des risques d'opération et des enjeux techniques pour le stockage de carburant et l'alimentation des génératrices. Un total de 17 remorques par jour sera nécessaire au site pour l'alimentation des génératrices avec chacune des installations de dépressurisation. De plus, un risque d'augmentation d'OPEX est prévu puisqu'une perte de 5 à 10 % de la quantité dans les remorques est possible en raison de la haute pression requise à l'entrée des génératrices. Finalement, l'empreinte de l'aire d'entreposage est un facteur important à considérer pour l'arrangement du site et les logistiques.

Aucun scénario intégrant uniquement des énergies renouvelables n'a été considéré dans cette étude. Les énergies renouvelables ne représentent pas une capacité ferme et doivent être jumelées à une source de capacité ferme comme une centrale autonome ou Hydro-Québec pour assurer une opération continue.

Selon les résultats de la matrice, les scénarios de centrales autonomes hybrides avec intégration d'énergie renouvelable sont moins intéressants considérant qu'ils obtiennent tous des pointages inférieurs à leur centrale autonome respective. Cela s'explique par leur CAPEX qui est considérablement plus élevé et qui est un facteur important dans la faisabilité du projet.

6. Conclusion et recommandations

Afin d'orienter Ressources PMET par rapport aux différentes méthodes de production d'électricité, BBA a évalué plusieurs options de génération d'électricité pour l'exploitation du projet minier Shaakichiuwaanaan. Les scénarios évalués sont l'interconnexion au poste Tilly d'Hydro-Québec comprenant notamment la construction d'une nouvelle ligne 120 kV de 53 km, une centrale autonome au diésel, une centrale autonome au gaz naturel liquéfié (GNL), et une centrale autonome au gaz naturel compressé (GNC), avec et sans intégration d'énergies renouvelables pour les centrales thermiques.

Ce rapport présente les résultats de l'analyse technicoéconomique et des émissions de gaz à effet de serre liés à chacune des options de production d'électricité étudiées avec un niveau de précision de classe V (-50 %/+100 %).

Le tableau suivant présente l'estimation des couts et des émissions de GES pour tous les scénarios à l'étude.

Tableau 53 : Sommaire des scénarios de génération d'électricité

Scénario	CAPEX	VAN	LCOE [$/kWh]	Émissions de GES annuelles [tCO_{2eq}]
1 – Interconnexion avec Hydro-Québec au poste Tilly	195 000 000 $	(398 176 000 $)	0,153	5 500
2A – Centrale autonome au diésel	174 315 000 $	(1 056 763 000 $)	0,405	181 000
2B – Centrale autonome hybride au diésel	269 322 000 $	(904 989 000 $)	0,347	136 000
3A – Centrale autonome au gaz naturel liquéfié	196 683 000 $	(831 936 000 $)	0,319	128 000
3B – Centrale autonome hybride au gaz naturel liquéfié	291 690 000 $	(743 071 000 $)	0,285	95 000
4A – Centrale autonome au gaz naturel compressé	175 575 000 $	(1 057 596 000 $)	0,406	128 000
4B – Centrale autonome hybride au gaz naturel compressé	270 581 000 $	(897 788 000 $)	0,344	95 000

L'évolution de la valeur actuelle nette (VAN) des scénarios lors des années d'exploitation de la mine est présentée sur la figure suivante :

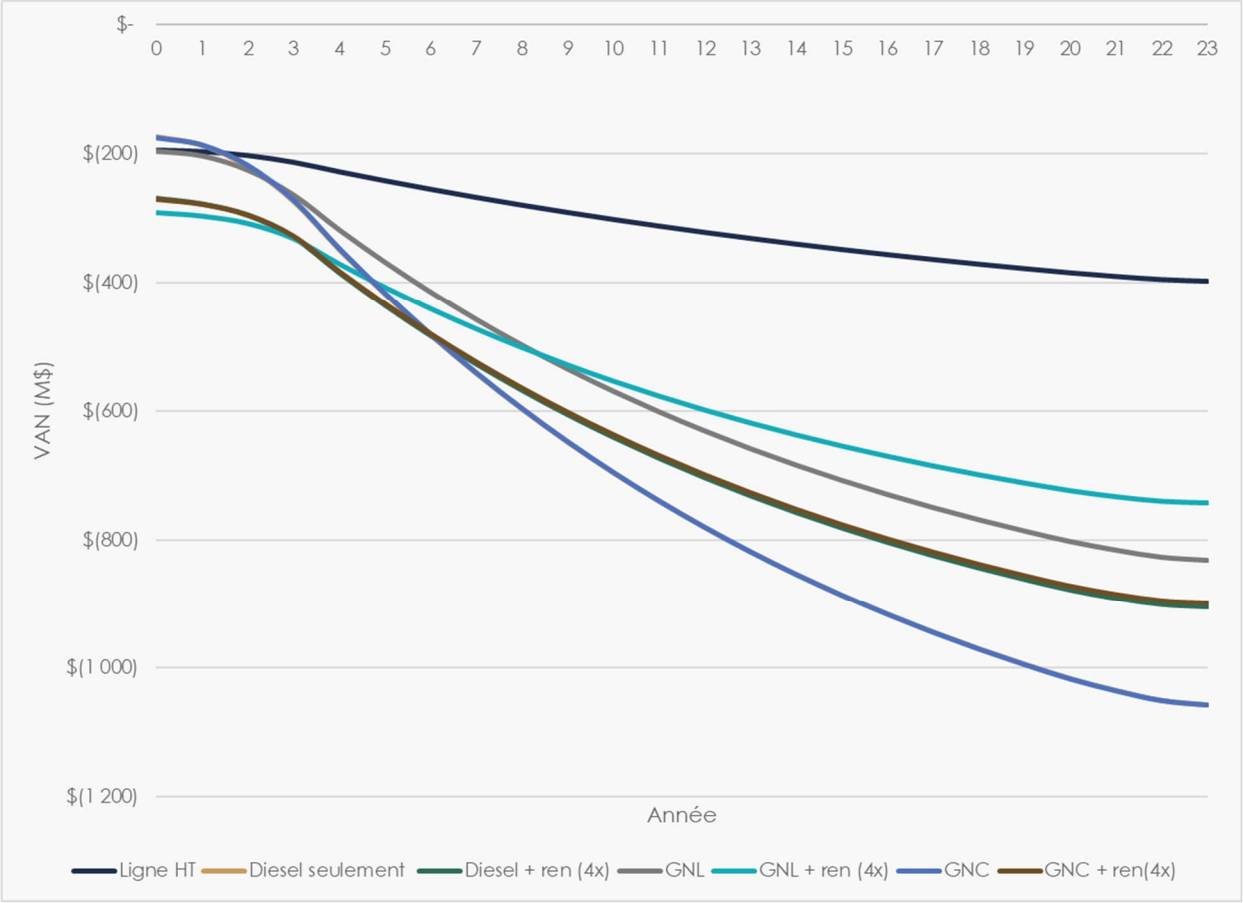


Figure 6 : Évolution de la VAN des différents scénarios

Il est à noter que les couts du chauffage des bâtiments principaux ont été considérés comme une économie dans les scénarios ayant un système de récupération de la chaleur et que cette économie pourrait être plus grande si la ventilation souterraine et les bâtiments administratifs se rajoutent au réseau de récupération de chaleur. De plus, les couts des droits d'émissions de GES sont inclus dans les couts d'exploitation lorsque l'inscription au SPEDE est obligatoire. Ceux-ci ont été considérés comme fixes à un prix conservateur et peuvent changer au courant du projet puisqu'ils dépendent de l'offre et de la demande, ce qui les rend hautement imprévisibles. Pour ces raisons, ces couts ne devraient être comparés qu'entre eux comme ils sont tous calculés sur la même base.

Une matrice décisionnelle a été réalisée afin d'effectuer une comparaison juste des différents scénarios. Les principaux aspects critiques économiques, environnementaux, sociaux et techniques ont été pris en compte dans l'élaboration de cette matrice. Les résultats de l'étude ont permis d'évaluer les scénarios pour chacun des critères et d'obtenir le pointage total suivant dont le plus élevé est le meilleur.

Tableau 54 : Pointage final des scénarios

Scénario	Pointage total
1 – Interconnexion avec Hydro-Québec au poste Tilly	80
2A – Centrale autonome au diésel	62
2B – Centrale autonome hybride au diésel	49
3A – Centrale autonome au GNL	64
3B – Centrale autonome hybride au diésel	52
4A – Centrale autonome au GNC	62
4B – Centrale autonome hybride au GNC	49

Selon ces résultats, le scénario d'interconnexion au réseau d'Hydro-Québec serait le plus avantageux puisque ses couts d'opération sont les plus bas, que son impact social est meilleur et qu'il présente les plus basses émissions de GES de tous les scénarios, le tout pour un CAPEX semblable à celui d'une centrale autonome au GNL qui présente le scénario de génératrice autonome le plus intéressant entre les différentes options.

Advenant l'éventualité où le bloc d'énergie nécessaire ne serait pas alloué par Hydro-Québec, une centrale autonome au gaz naturel liquéfié serait le meilleur choix de production d'électricité. La construction de la centrale autonome au gaz naturel liquéfié présente un CAPEX presque identique à celui de la construction de l'interconnexion avec HQ soit d'environ 22 M$ de plus que la centrale autonome au diésel. Cette option requiert un investissement CAPEX plus important que celui requis pour la centrale diésel. Toutefois, elle présente plusieurs avantages tels que des caractéristiques environnementale et sociale supérieures aux autres options ainsi qu'une réduction des couts en carburant d'environ 26 M$ lors de la pleine opération de la mine. La valeur actuelle nette du projet en est donc grandement améliorée.

7. Prochaines étapes

La prochaine étape dans le cas de l'interconnexion avec Hydro-Québec serait d'obtenir une entente pour la totalité des MW requis pour la durée du projet afin de démarrer les travaux le plus rapidement possible. Dans ce cas, il serait important d'analyser le niveau d'engagement à utiliser pour le Programme de Gestion de la demande de puissance (GDP) ainsi que les options envisageables pour le délestage.

Pour ce qui est d'une centrale hybride autonome, en cas d'absence d'entente avec Hydro-Québec, la prochaine étape recommandée est de commencer l'ingénierie préliminaire de la centrale aussitôt que possible afin d'aller en appel d'offres rapidement, notamment pour les équipements à longs délais de livraison tels que les génératrices.

Dans l'éventualité où l'intégration des énergies renouvelables serait analysée plus en profondeur dans le but de réduire les couts d'opération (OPEX) et les émissions de GES, tout en augmentant la valeur actuelle nette, les étapes suivantes devront être réalisées :

- Effectuer une évaluation de la ressource éolienne au site en installant une tour de mesurage de vent;
- Effectuer une étude d'intégration d'énergies renouvelables et vérifier les aides financières disponibles.

8. Opportunités

Finalement, voici une liste d'opportunités qui n'ont pas été examinées dans le cadre de cette étude, mais qui pourraient s'avérer pertinentes pour de futures analyses :

- **Analyse d'une forte pénétration d'énergie renouvelable :** L'analyse d'une forte pénétration d'énergies renouvelables n'a pas été considérée dans cette étude, car elle était jugée moins avantageuse que le scénario de référence, soit l'interconnexion au réseau d'Hydro-Québec. Toutefois, l'intégration d'une quantité importante d'énergies renouvelables pourrait permettre certaines économies en recourant à une centrale thermique d'appoint équipée de génératrices moins performantes, utilisées uniquement quelques fois par an.
- **Optimisation de la charge :** L'analyse de l'optimisation de la charge n'a pas été considérée dans cette étude qui considère que la charge fournie par Ressources PMET est constante. Cependant, adapter la charge en fonction de la production renouvelable peut réduire le besoin de surdimensionner les équipements lors des journées avec peu de ressource. Cette approche peut aussi inclure des modes d'exploitation distincts, limitant ainsi le surdimensionnement du système de stockage par batteries (BESS).

- **Évaluation du legs des installations énergétiques aux communautés avoisinantes** : L'option de céder les installations énergétiques (plutôt que de les démanteler) à des communautés locales, notamment autochtones, n'a pas été évaluée dans le cadre de cette étude, ni l'option de partenariat avec ces communautés. Une telle approche pourrait contribuer à accroitre les retombées sociales et économiques du projet après la fermeture de la mine, tout en favorisant l'acceptabilité sociale.

Annexe A : Rapport d'analyse PVSyst pour l'intégration d'un parc solaire

PVsyst - Simulation report

Grid-Connected System

Project: 7535030 - Innova

Variant: Innova

Sheds, single array

System power: 1065 kWp

Innova - Canada

Author

Groupe BBA Inc (Canada)

PVsyst V8.0.3
VC9, Simulation date:
25/07/25 12:08
with V8.0.3

Project summary

Geographical Site
Innova
Canada

Situation

Latitude	53.55 °N
Longitude	-73.91 °W
Altitude	387 m
Time zone	UTC-5

Weather data
Innova
Meteonorm 8.2 (2016-2021), Sat=100 % - Synthétique

Monthly albedo values

	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sep.	Oct.	Nov.	Dec.
Albedo	0.50	0.50	0.40	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.27	0.50

System summary

Grid-Connected System

Sheds, single array

Orientation #1
Fixed plane
Tilt/Azimuth 35 / 0 °

Near Shadings
Linear shadings : Fast (table)

User's needs
Unlimited load (grid)

System information

PV Array		Inverters	
Nb. of modules	1704 units	Nb. of units	3 units
Pnom total	1065 kWp	Pnom total	960 kWac
		Pnom ratio	1.109

Results summary

Produced Energy	1416677 kWh/year	Specific production	1330 kWh/kWp/year	Perf. Ratio PR	95.28 %

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General parameters, PV Array Characteristics, System losses 3
Horizon definition 5
Near shading definition - Iso-shadings diagram 6
Main results 7
Loss diagram 8
Predef. graphs 9
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General parameters

Grid-Connected System

Sheds, single array

Orientation #1

Fixed plane

Tilt/Azimuth	35 / 0 °

Sheds configuration

Nb. of sheds	12 units
Single array	

Shading limit angle

Limit profile angle	17.9 °

Sizes

Sheds spacing	6.00 m
Collector width	2.29 m
Average GCR	38.1 %
Top inactive band	0.02 m
Bottom inactive band	0.02 m

Models used

Transposition	Perez
Diffuse	Perez, Meteonorm
Circumsolar	separate

Horizon

Average Height	3.4 °

Near Shadings

Linear shadings : Fast (table)

Bifacial system definition

Orientation #1

Bifacial system

Model	Unlimited Sheds 2D Model

Bifacial model geometry

Sheds spacing	6.00 m
Sheds width	2.31 m
Limit profile angle	17.9 °
GCR	38.5 %
Height above ground	1.50 m
Nb. of sheds	12 units

Bifacial model definitions

Ground albedo	0.30
Bifaciality factor	80 %
Rear shading factor	5.0 %
Rear mismatch loss	10.0 %
Shed transparent fraction	0.0 %

User's needs

Unlimited load (grid)

PV Array Characteristics

PV module

Manufacturer	LONGi Solar
Model	LR7-72HGD 625M
(Custom parameters definition)	
Unit Nom. Power	625 Wp
Number of PV modules	1704 units
Nominal (STC)	1065 kWp
Modules	71 string x 24 In series
At operating cond. (50°C)	
Pmpp	991 kWp
U mpp	979 V
I mpp	1012 A

Inverter

Manufacturer	Sungrow
Model	SG350HX-20A-Preliminary
(Custom parameters definition)	
Unit Nom. Power	320 kWac
Number of inverters	3 units
Total power	960 kWac
Operating voltage	500-1500 V
Max. power (=>30°C)	352 kWac
Pnom ratio (DC:AC)	1.11
Power sharing within this inverter	

Total PV power

Nominal (STC)	1065 kWp
Total	1704 modules
Module area	4603 m²
Cell area	4282 m²

Total inverter power

Total power	960 kWac
Max. power	1056 kWac
Number of inverters	3 units
Pnom ratio	1.11

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Array losses

Array Soiling Losses
Loss Fraction 3.0 %

Thermal Loss factor
Module temperature according to irradiance
Uc (const) 29.0 W/m²K
Uv (wind) 0.0 W/m²K/m/s

DC wiring losses
Global array res. 7.4 mΩ
Loss Fraction 0.7 % at STC

LID - Light Induced Degradation
Loss Fraction 0.6 %

Module Quality Loss
Loss Fraction -0.4 %

Module mismatch losses
Loss Fraction 2.0 % at MPP

Strings Mismatch loss
Loss Fraction 0.1 %

IAM loss factor
Incidence effect (IAM): User defined profile

0°	40°	50°	60°	70°	75°	80°	85°	90°
1.000	1.000	1.000	1.000	0.950	0.890	0.830	0.680	0.000

System losses

Auxiliaries loss
Proportional to Power 4.0 W/kW
0.0 kW from Power thresh.

AC wiring losses

Inv. output line up to MV transfo
Inverter voltage 800 Vac tri
Loss Fraction 1.11 % at STC
Inverter: SG350HX-20A-Preliminary
Wire section (3 Inv.) Copper 3 x 3 x 240 mm²
Average wires length 259 m

MV line up to Injection
MV Voltage 13.8 kV
Average each inverter
Wires Alu 3 x 50 mm²
Length 1000 m
Loss Fraction 0.17 % at STC

AC losses in transformers

MV transfo
Medium voltage 13.8 kV

One transfo parameters
Nominal power at STC 525 kVA
Iron Loss (24/24 Connexion) 0.48 kVA
Iron loss fraction 0.09 % at STC
Copper loss 5.69 kVA
Copper loss fraction 1.08 % at STC
Coils equivalent resistance 3 x 13.23 mΩ

Operating losses at STC (full system)
Nb. identical MV transfos 2
Nominal power at STC 1.05 MVA
Iron loss (24/24 Connexion) 0.96 kVA
Copper loss 11.38 kVA

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Horizon definition

CSV horizon file, lat:-25.075052, lng:-66.809056, exported by solargis.info at 2

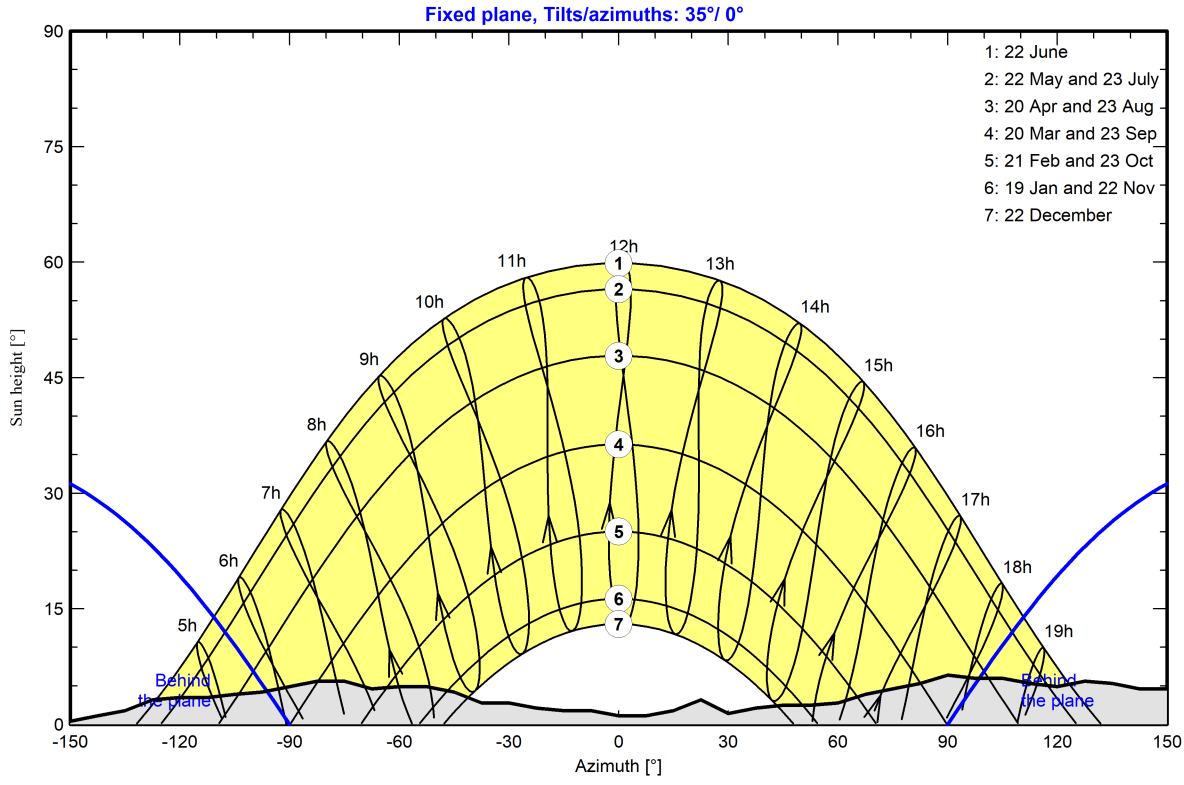
Average Height	3.4 °	Albedo Factor	0.77
Diffuse Factor	0.94	Albedo Fraction	100 %

Horizon profile

Azimuth [°]	-180	-173	-165	-150	-143	-135	-128	-120	-113	-105	-98	-90
Height [°]	1.4	0.7	0.4	0.4	1.1	1.8	3.2	3.5	3.5	3.9	4.2	4.9
Azimuth [°]	-83	-75	-68	-60	-53	-45	-38	-30	-23	-15	-8	0
Height [°]	5.6	5.6	4.6	4.9	4.9	4.2	2.8	2.8	2.1	1.8	1.8	1.1
Azimuth [°]	8	15	23	30	38	45	53	60	68	75	83	90
Height [°]	1.1	1.8	3.2	1.4	2.1	2.5	2.5	2.8	3.9	4.6	5.3	6.4
Azimuth [°]	98	105	113	120	128	135	143	150	158	165	173	
Height [°]	6.0	6.0	5.3	4.9	5.6	5.3	4.6	4.6	4.2	3.2	2.1	

Sun Paths (Height / Azimuth diagram)

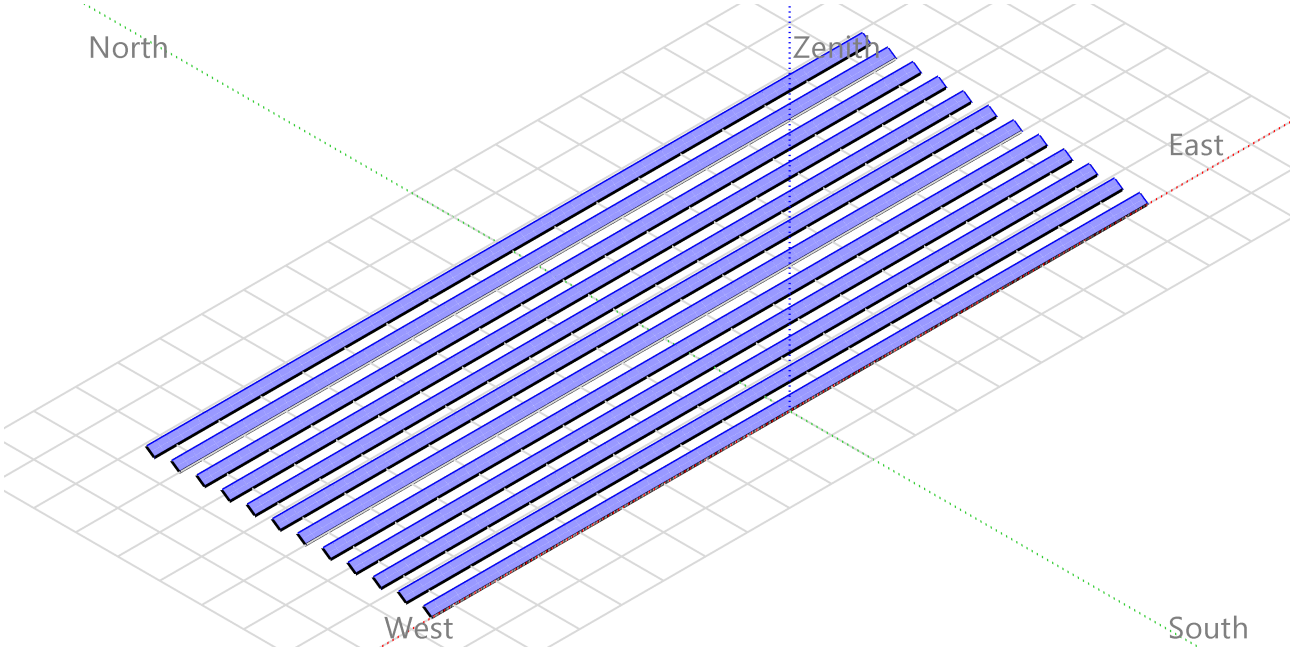
Orientation #1



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Near shadings parameter

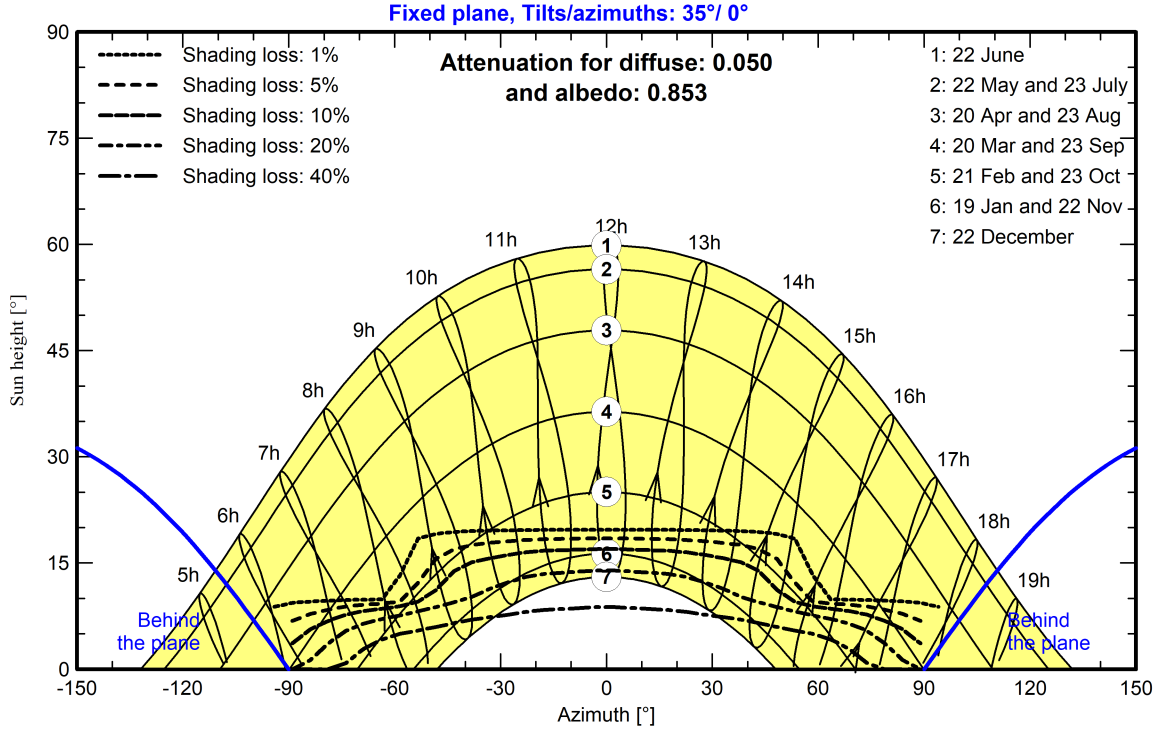
Perspective of the PV-field and surrounding shading scene



Iso-shadings diagram

Orientation #1

Fixed plane, Tilts/azimuths: 35°/ 0°



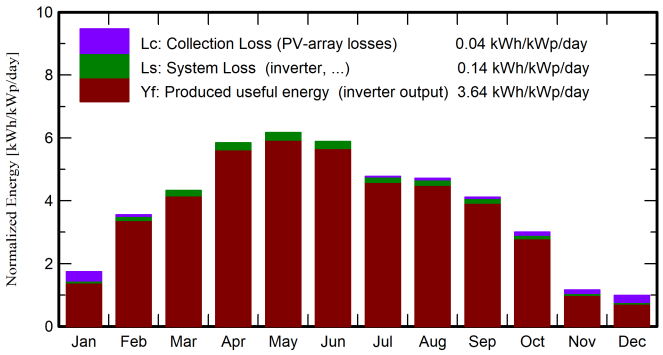
PVsyst V8.0.3
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Main results

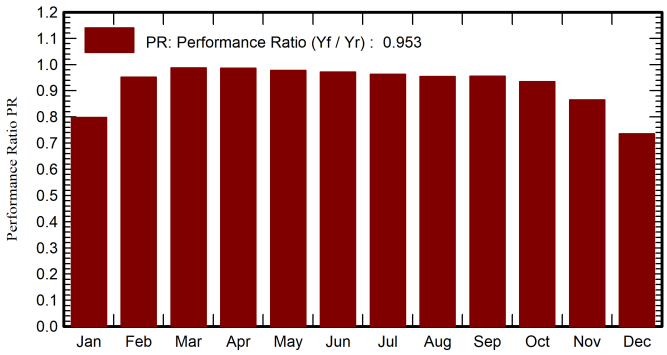
System Production

Produced Energy	1416677 kWh/year	Specific production	1330 kWh/kWp/year
		Perf. Ratio PR	95.28 %

Normalized productions (per installed kWp)



Performance Ratio PR



Balances and main results

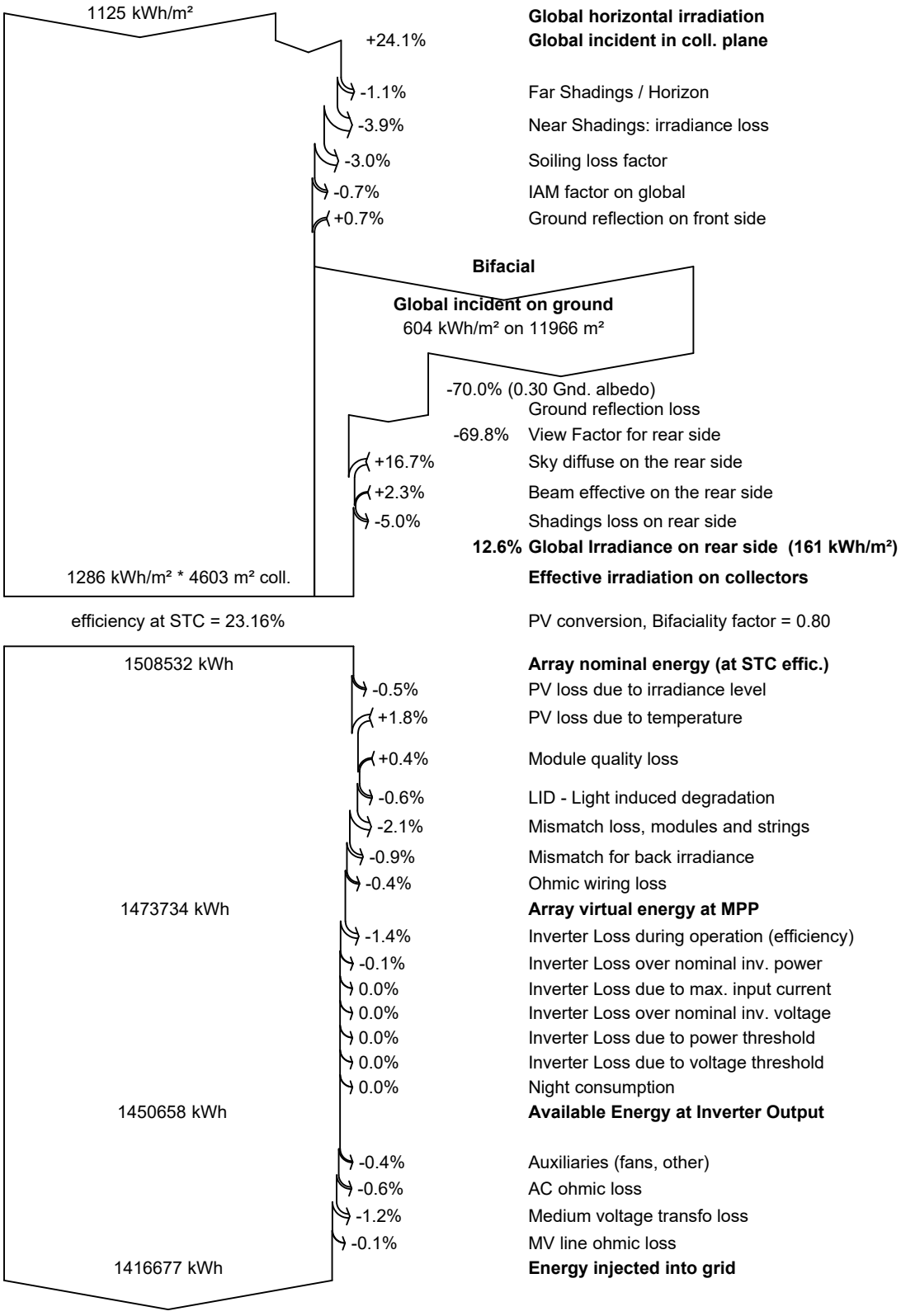
	GlobHor kWh/m²	DiffHor kWh/m²	T_Amb °C	GlobInc kWh/m²	GlobEff kWh/m²	EArray kWh	E_Grid kWh	PR ratio
January	23.7	12.91	-21.65	54.2	41.4	48010	46010	0.797
February	51.6	20.50	-21.52	99.6	89.9	104637	100944	0.951
March	89.3	38.62	-13.66	130.9	121.8	142851	137615	0.987
April	139.6	52.37	-3.93	171.3	161.7	186797	179919	0.986
May	179.6	70.53	5.51	188.5	177.5	203639	196132	0.977
June	175.6	81.10	11.90	175.3	164.5	188175	181290	0.971
July	147.8	88.19	15.36	148.1	138.1	157495	151790	0.963
August	131.7	68.77	14.01	146.2	137.1	154218	148569	0.954
September	94.3	46.60	8.72	123.5	115.6	130505	125751	0.956
October	58.0	27.26	2.86	93.1	86.5	96294	92617	0.934
November	20.1	12.73	-4.97	34.8	29.8	33885	32050	0.865
December	14.0	8.94	-15.62	30.6	22.0	25524	23989	0.735
Year	1125.4	528.50	-1.81	1396.1	1286.1	1472030	1416677	0.953

Legends

GlobHor	Global horizontal irradiation
DiffHor	Horizontal diffuse irradiation
T_Amb	Ambient Temperature
GlobInc	Global incident in coll. plane
GlobEff	Effective Global, corr. for IAM and shadings
EArray	Effective energy at the output of the array
E_Grid	Energy injected into grid
PR	Performance Ratio

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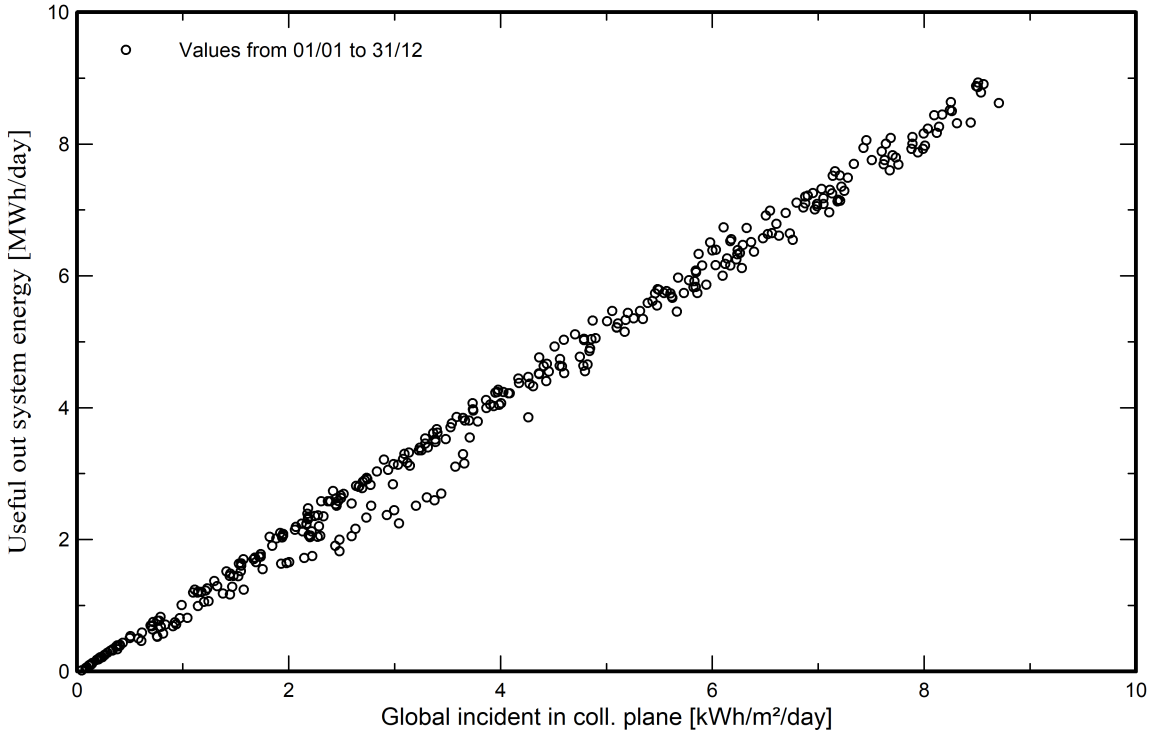
Loss diagram



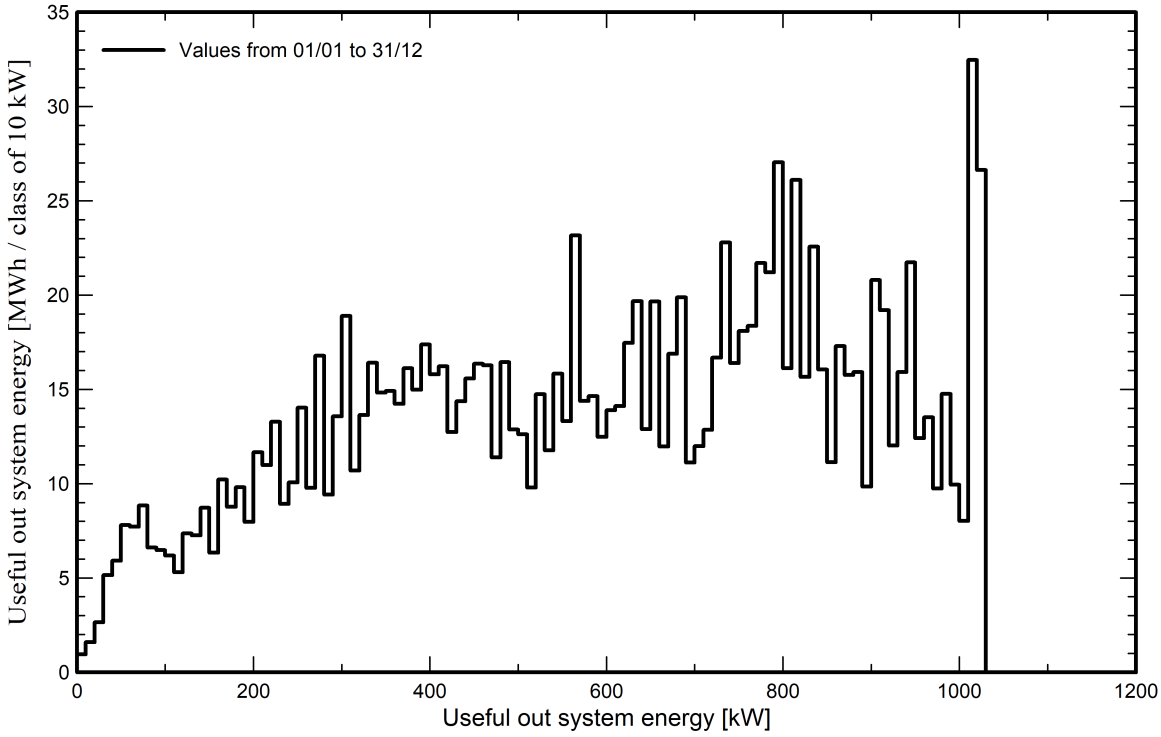
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Predef. graphs

Diagramme d'entrée/sortie journalier

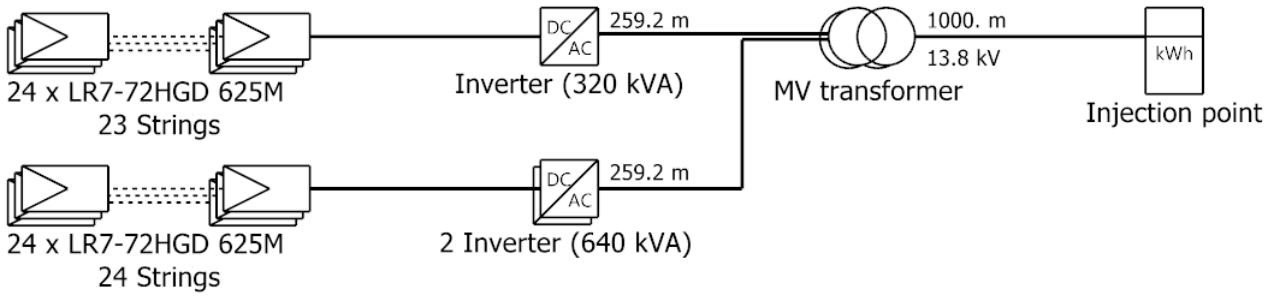


Distribution de la puissance de sortie système



Single-line diagram

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PV module	LR7-72HGD 625M
Inverter	SG350HX-20A-Preliminary
String	24 x LR7-72HGD 625M

	7535030 - Innova	Groupe BBA Inc (Canada)
	VC9 : Innova	25/07/25

Annexe B : Explication de la pondération de la matrice décisionnelle

Critère	Description	Poids	1 – Ligne H.T.	2A- Diesel	2B – Diesel + renouvelable	3A – GNL	3B – GNL + renouvelable	4A – GNC	4B – GNC + renouvelable
			Valeur[2]	Valeur[2]	Valeur[2]	Valeur[2]	Valeur[2]	Valeur[2]	Valeur[2]
Impact Économique									
CAPEX	Limiter l'investissement initial	5	2 - CAPEX de 195 M$	3 - CAPEX de 175M$	1 - CAPEX de 270 M$	2 - CAPEX de 197M$	1 -CAPEX de 292 M$	3 - CAPEX de 176 M$	1 - CAPEX de 271 M$
VAN	Avoir la meilleiure VAN possible basé sur les résultats obtenus	3	3 -VAN de -398 M$)	1 - VAN de - 1 057 M$	1 - VAN de - 905 M$	2 - VAN de -831 M$	2 - VAN de -743 M$	1 - VAN de - 1 058 M$	1 - VAN de -897 M$
Prévisibilité	Fluctuation des prix d'opération	2	3 - Hausse prévisible des côuts d'électricité d'Hydro-Québec	1 - Le coût du diesel pourrait autant monter que descendre lors de la durée de vie de la mine	2 - La production éolienne réduit l'incertitude des OPEX	1 -Le coût du GNL pourrait autant monter que descendre lors de la durée de vie de la mine	2 - La production éolienne réduit l'incertitude des OPEX	1-Le coût du GNC pourrait autant monter que descendre lors de la durée de vie de la mine	2 - La production éolienne réduit l'incertitude des OPEX
Impact environnemental									
Émissions GES	Les émissions de GES doivent être le plus faible possible Basé sur les tonnes de GES émises	4	3 - Émissions annuelles de 5 500 TCO2eq	1 - Émissions annuelles de 181 000 TCO2eq	2 - Émissions annuelles de 136 000 TCO2eq	2 - Émissions annuelles de 128 000 TCO2eq	2- Émissions annuelles de 95 000 TCO2eq	2 - Émissions annuelles de 128 000 TCO2eq	2- Émissions annuelles de 95 000 TCO2eq
Impact sur les plans d'eau et les milieux humide	Destruction des milieux humides pour la construction d'installation.	2	1 - Des milieux humides peuvent être affectés durant la phase de construction	3 - Aucun impact supplémentaire comme elle est installée sur le site minier	2 - Des milieux humides pourraient être affecté lors de la construction des éoliennes et des chemins d'accès, mais leur nombre est limité	3 - Aucun impact comme elle est installée sur le site minier	2 - Des milieux humides pourraient être affecté lors de la construction des éoliennes et des chemins d'accès, mais leur nombre est limité	3 - Aucun impact comme elle est installée sur le site minier	2 - Des milieux humides pourraient être affecté lors de la construction des éoliennes et des chemins d'accès, mais leur nombre est limité
Impact sur la faune		2	2-La faune sera perturbée lors de la construction, mais ensuite l'impact est très faible	3 - Aucun impact supplémentaire comme elle est installée sur le site minier	1- Le bruit des éoliennes et leur rotation peuvent afffecter la faune nottament les chauves-souris et les oiseaux	3 - Aucun impact supplémentaire comme elle est installée sur le site minier	1- Le bruit des éoliennes et leur rotation peuvent afffecter la faune nottament les chauves-souris et les oiseaux	3 - Aucun impact supplémentaire comme elle est installée sur le site minier	1- Le bruit des éoliennes et leur rotation peuvent afffecter la faune nottament les chauves-souris et les oiseaux
Empreinte sur territoire	Production/transport Installation	2	1 - Empreinte pour la servitude d'Hydro-Québec sur la ligne 120 kV de 53 km	3-Situé sur le site minier aucun impact supplémentaire sur le territoire	2- Éoliennes situé à proximité du site	3-Situé sur le site minier aucun impact supplémentaire sur le territoire	2- Éoliennes situé à proximité du site	2-Situé sur le site minier aucun impact supplémentaire sur le territoirem, mais espace important requis sur le site pour l'infrastructure d'entreposage (remorques).	2- Éoliennes situé à proximité du site
Impact social									

Retombé économique	Possibilité d'emplois, entreprise local, etc.	3	2 - La ligne pourrait appartenir à la communauté ou lui être vendue lors de la fermeture de la mine	1 - Aucunes retombées supplémentaires pour la communauté	1 - Aucunes retombées supplémentaires pour la communauté	1 - Aucunes retombées supplémentaires pour la communauté	1 - Aucunes retombées supplémentaires pour la communauté	1 - Aucunes retombées supplémentaires pour la communauté	1 - Aucunes retombées supplémentaires pour la communauté
Circulation	Transport du Diesel/GNL	3	3 - Quelques camions pour le chauffage au GNC	1 - Beaucoup de camions nécessaires pour le diesel	2 - Moins de transport car une partie de l'énergie provient de éoliennes	1 - Beaucoup de camions nécessaires pour le GNL	2 - Moins de transport car une partie de l'énergie provient de éoliennes	1 - Beaucoup de camions nécessaires pour le GNC	2 - Moins de transport car une partie de l'énergie provient de éoliennes
Autres utilisateurs possible	Autre utilisateur des infrastructures électrique	2	3 -D'autres raccordements sur la ligne 120kV seraient possible lors de l'exploitation de la mine	1 -Uniquement pour la mine	1 -Uniquement pour la mine	1 -Uniquement pour la mine	1 -Uniquement pour la mine	1 -Uniquement pour la mine	1 -Uniquement pour la mine
Nuisance (bruit, poussière, paysage)	Plus la valeur est élevée, moins il y a de nuissances	2	3-La ligne sera visible, mais il s'agit d'une structure fréquente dans la région, aucun bruit et aucune poussière	2 - Bruit provenant des génératrices	1-Impact visuel moins habituel pour la région et du bruit provenant des génératrices et des éoliennes	2 - bruit provenant des génératrices	1-Impact visuel moins habituel pour la région et du bruit provenant des génératrices et des éoliennes	2 - Bruit provenant des génératrices	1-Impact visuel moins habituel pour la région et du bruit provenant des génératrices et des éoliennes
Autre									
Constructibilité et délai		2	3 -Construction qui s'étend sur une durée approximative d'un an	3 -Simplicité de la construction, uniquement la centrale et respect de l'échéancier	2 -L'ajout d'éoliennes augmente la complexité de la constructibilité, l'échéancier est toujours respecté	3 -Simplicité de la construction, uniquement la centrale et respect de l'échéancier	2 -L'ajout d'éoliennes augmente la complexité de la constructibilité, l'échéancier est toujours respecté	2 -Simplicité de la construction, uniquement la centrale et respect de l'échéancier. Complexité pour la construction des installations de livraison et de manutention de GNC.	2 -L'ajout d'éoliennes augmente la complexité de la constructibilité, l'échéancier est toujours respecté
Fiabilité électrique		1	2 - Capacité de délestage de la charge de 30% requis (GDP)	2 - Transport et stockage du diesel nécessaire	2 - Transport et stockage du Diesel nécessaire	2 - Transport et stockage du GNLnécessaire	2 - Transport et stockage du GNL nécessaire	2 - Transport et stockage du GNC nécessaire	2 - Transport et stockage du GNC nécessaire
Total		33							

3 Pointage = Valeur x Poids

APPENDIX

4-4 HEATING FUEL SUPPLY – COMPARATIVE ANALYSIS

N° de document : CASN-A-MC-Chauffage	**Type : Étude comparative**	
Département : Mécanique	**Date : 8 mai 2025**	**N° de pages : 10**

SOURCE DE COMBUSTIBLE DE CHAUFFAGE - ÉTUDE COMPARATIVE

Rev	Date	Statut	Modification	Préparé par	Révisé par	Approuvé par
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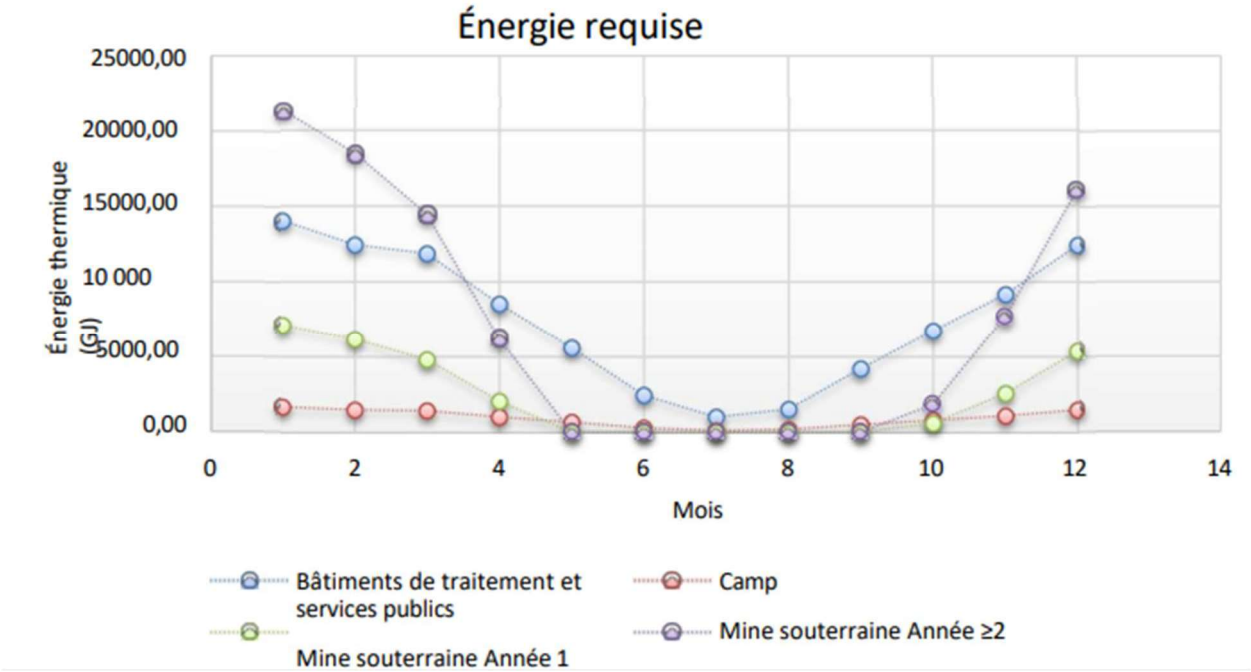
1. DESCRIPTION DES BESOINS

Ce rapport présente les conclusions d'une évaluation technico-économique comparant trois (3) scénarios d'approvisionnement énergétique potentiels (gaz propane liquéfié (GPL), gaz naturel liquéfié (GNL) et gaz naturel comprimé (GNC)) pour fournir de la chaleur au projet d'exploitation minière de lithium Shaakichiuaanaan. Les besoins en chauffage couvrent des infrastructures clés, notamment le campement des travailleurs, les installations de traitement, le bâtiment des équipements auxiliaires et la mine souterraine. L'étude évalue à la fois la faisabilité technique et la viabilité économique de chaque option, dans le but d'identifier la solution la plus appropriée et la plus durable pour soutenir les opérations minières dans la région de la Baie-James au Québec, caractérisée par des conditions hivernales rigoureuses et prolongées.

Les besoins mensuels en énergie thermique (en GJ) pour les bâtiments de traitement, le campement et les deux étapes de l'exploitation minière souterraine sont résumés ci-dessous :

	Énergie requise par bâtiment (GJ)			
Mois	**Bâtiments de traitement et services publics**	**Campement**	**Mine souterraine Année 1**	**Mine souterraine Année ≥ 2**
Janvier	14 008	1 676	7 105	21 317
Février	12 414	1 485	6 151	18 454
Mars	11 832	1 416	4 807	14 422
Avril	8 520	1 019	2 058	6 174
Mai	5 570	666	0	0
Juin	2 435	291	0	0
Juillet	1 006	121	0	0
Août	1 525	183	0	0
Septembre	4 193	502	0	0
Octobre	6 719	804	612	1 835
Novembre	9 142	1 094	2 561	7 684
Décembre	12 392	1 483	5 368	16 105
Total	89 757	10 739	28 662	85 991

Figure 1.1 : Demande en énergie thermique par zone (GJ)



Le graphique inclut le camp, qui présente des besoins en chauffage relativement faibles mais tout au long de l'année, allant d'environ 120 GJ en juillet à près de 1 700 GJ en janvier. Les bâtiments de traitement et les services publics continuent d'afficher des besoins en chauffage importants tout au long de l'année, avec un pic hivernal d'environ 14 000 GJ. La mine souterraine nécessite un chauffage important pendant les mois froids, en particulier d'octobre à avril, sans aucune demande en été. À partir de l'année ≥ 2, les opérations souterraines connaissent une augmentation importante des besoins en chauffage, en raison de la profondeur accrue des opérations et des besoins plus importants en circulation d'air.

2. DESCRIPTION DES OPTIONS

Cette étude examine trois (3) options principales pour l'approvisionnement en énergie thermique du site minier.

2.1 Gaz naturel liquéfié (GNL)

Le GNL est du gaz naturel refroidi à environ -162°C pour le transport et le stockage. Il offre une densité énergétique élevée et est de plus en plus utilisé dans les projets industriels éloignés. Il

nécessite des réservoirs cryogéniques et des systèmes de vaporisation spécialisés, mais présente l'avantage d'avoir une empreinte carbone inférieure à celle du propane.

2.2 Gaz propane liquéfié (GPL)

Le propane est un combustible couramment utilisé pour le chauffage dans les régions éloignées. Il est réputé pour sa fiabilité et sa facilité de stockage dans des réservoirs sous pression. Il est largement disponible au Québec et ne nécessite pas de manipulation cryogénique, ce qui le rend plus simple à installer et à utiliser.

2.3 Gaz naturel comprimé (GNC)

Le gaz naturel comprimé (GNC) est du gaz naturel stocké à haute pression (généralement entre 200 et 250 bars). Contrairement au GNL, il ne nécessite pas de stockage cryogénique, mais il nécessite un équipement de décompression sur place et une infrastructure de sécurité appropriée. Bien que le GNC ait une densité énergétique inférieure à celle du GNL, ce qui entraîne une fréquence de livraison plus élevée, cet aspect logistique est déjà pris en compte dans le prix global par GJ ($/GJ), qui comprend le transport. Ainsi, le GNC offre un processus de manipulation simplifié et des coûts d'exploitation prévisibles.

3. AVANTAGES ET INCONVÉNIENTS DE CHAQUE OPTION

Chaque option énergétique examinée dans cette étude présente des avantages et des limites qui lui sont propres. Le tableau ci-dessous résume les principaux compromis entre le gaz naturel liquéfié (GNL), le propane liquéfié (GPL) et le gaz naturel comprimé (GNC).

Critères	GNL	GPL	GNC
Densité énergétique (GJ/m^3)	Élevé	Moyen	Faible
Complexité du stockage	Élevé (cryogénique)	Faible (pressurisé)	Moyen (PSI élevé)
Fréquence de transport	Faible	Moyen	Élevé
Besoins en CAPEX	Élevé	Moyen	Faible
Émissions (CO_2/GJ)	Faible	Élevé	Bas Faible

Ce tableau permet de visualiser la comparaison des sources d'énergie selon des critères décisionnels importants. Ces compromis qualitatifs seront examinés plus en détail en relation avec les données quantitatives sur les coûts dans les sections suivantes.

4. DESCRIPTION DES INSTALLATIONS ET CARACTÉRISTIQUES DES COMBUSTIBLES

Cette section présente les caractéristiques techniques spécifiques et les implications en matière d'infrastructure de chaque source d'énergie, fournissant ainsi la base pour l'analyse des coûts d'investissement et de la logistique qui suit.

4.1 Gaz naturel liquéfié (GNL)

Le gaz naturel est composé à plus de 97 % de méthane et est liquéfié par refroidissement à -162°C afin de faciliter son transport et son stockage en vrac. Une fois sur site, le GNL doit être dégazé par un processus de vaporisation avant d'être distribué. Il est reconnu pour produire les émissions de CO_2 les plus faibles par unité d'énergie parmi les combustibles fossiles. Son contenu énergétique spécifique est d'environ 53,6 MJ/kg. Le GNL nécessite des réservoirs cryogéniques à proximité de la zone de traitement principale et offre la possibilité d'une vaporisation centralisée et d'un transfert par canalisation vers des zones éloignées telles que l'atelier de réparation des camions. Le coût indiqué est de 14,29 $/GJ, sur la base des prix pratiqués par Énergir. La densité énergétique et les performances en matière d'émissions du GNL en font un candidat sérieux, malgré des besoins en infrastructures plus importants.

4.2 Gaz propane liquéfié (GPL)

Le propane est un gaz hydrocarbure (C_3H_8) qui est liquéfié sous pression modérée pour faciliter son stockage et son transport. Contrairement au GNL, il ne nécessite pas de températures cryogéniques, se liquéfiant à environ -42°C à la pression atmosphérique. Une fois livré sur site, le propane peut être vaporisé à l'aide de vaporisateurs ambiants ou chauffés et distribué par des réseaux de canalisations. Il a une teneur en carbone plus élevée que le méthane, ce qui se traduit par des émissions de CO_2 légèrement plus élevées par unité d'énergie. Son contenu énergétique spécifique est d'environ 49,6 MJ/kg. Le stockage s'effectue généralement dans des réservoirs sous pression placés à proximité de la zone d'utilisation, qui peuvent alimenter directement les équipements de chauffage ou les systèmes de distribution. Le coût indiqué est de 33,77 $/GJ, sur la base des prix pratiqués par Avenir Energy.

4.3 Gaz naturel comprimé (GNC)

Le GNC est acheminé vers des sites éloignés, tels que des mines, à l'aide d'un système de pipeline « virtuel » basé sur des remorques à haute pression, généralement pressurisées à 3 600-4 300 psi (250-300 bars), ce qui élimine le besoin de décompression sur site. Ces remorques servent d'unités de stockage mobiles et sont remplacées selon les besoins.

Une fois sur place, un système dédié de décompression et de régulation de la pression est utilisé pour réduire en toute sécurité la pression du gaz du niveau de la remorque à une pression de distribution adaptée aux systèmes de chauffage des bâtiments, généralement comprise entre 5 et 100 psi. Ce système comprend des échangeurs de chaleur pour éviter le gel, ainsi que des régulateurs de pression, des soupapes de sécurité et des systèmes d'arrêt.

Afin de garantir la sécurité et l'efficacité opérationnelle, un système de surveillance et de contrôle est mis en place. Celui-ci comprend une télémétrie à distance pour la pression de la remorque, la pression de sortie et la surveillance du débit, ce qui permet une logistique proactive d'échange des remorques et un suivi en temps réel de la consommation. Le coût indiqué est de 38,17 $/GJ, sur la base des prix pratiqués par Certarus.

5. CAPEX – DEUX (2) PREMIÈRES ANNÉES

Cette section présente une estimation globale des dépenses d'investissement nécessaires à la mise en œuvre de chacun des trois (3) scénarios énergétiques (GNL, propane liquéfié et GNC) au cours des deux (2) premières années du projet de mine de lithium. Ces estimations comprennent les infrastructures de stockage et les équipements de livraison.

- Gaz naturel liquéfié (GNL) :
  - Réservoir de stockage cryogénique.
  - Système de vaporisation.
  - Réseau de canalisations local.
  - Instrumentation et contrôles.
  - CAPEX estimé : 17 911 572,00 $ selon le devis d'Énergir.
- Gaz propane liquéfié (GPL) :
  - Réservoirs de stockage ASME sous pression.

- Vaporisateurs.
- Réseau de tuyauterie local.
- CAPEX estimé : 4 011 334,03 $ selon le devis d'Avenir Energy.

- Gaz naturel comprimé (GNC) :
  - Mobilisation de l'équipement : aire de déchargement des remorques et skid de décompression.
  - Matériaux nécessaires sur le terrain, tels que la tuyauterie sur site : coût + 15 %.
  - CAPEX estimé : 34 000 $ selon le devis de Certatrus.

6. OPEX – COÛTS D'EXPLOITATION SUR 20 ANS

- Gaz naturel liquéfié (GNL) :
  - L'estimation des coûts d'exploitation pour le scénario GNL est basée sur une consommation annuelle moyenne d'environ 186 000 GJ.
  - Coût des molécules de gaz naturel : 14,29 $/GJ.
  - Coût de livraison : 11 740 $/livraison.
- Gaz propane liquéfié (GPL) :
  - L'estimation des coûts d'exploitation pour le scénario GNC est basée sur une consommation annuelle moyenne d'environ 186 000 GJ.
  - Coût du GPL : 33,77 $/GJ.
  - Coût de livraison : 7 875 $/livraison.
- Gaz naturel comprimé (GNC) :
  - L'estimation des coûts d'exploitation pour le scénario GNC est basée sur une consommation annuelle moyenne d'environ 186 000 GJ. Le prix du scénario GNC est basé sur la structure tarifaire de la plateforme Certarus, y compris les frais de livraison.
  - Coût du GNC : 38,17 $/GJ.
  - Certarus livrerait en moyenne 10 à 15 remorques de GNC par semaine.
  - L'autonomie opérationnelle prévue serait de 4 jours.

7. RÉSUMÉ DES CAPEX ET OPEX

Type de combustible	CAPEX (CAD)	Coût $/GJ	Coût de livraison (CAD/Année)	Opex (CAD/Année)	OPEX sur 20 ans (CAD)	Coût total sur 20 ans (CAPEX+OPEX)
Gaz naturel liquéfié (GNL)	17 911 572,00	14,29	2 312 780,00	4 970 720,00	99 414 400,00	117 325 972,00
Gaz propane liquéfié (GPL)	4 011 334,03	33,77	1 551 375,00	7 832 595,00	156 651 900,00	160 663 234,03
Gaz naturel comprimé (GNC)	34 000,00	38,17	-	7 099 620,00	141 992 400,00	142 026 400,00

8. CONCLUSION

Bien que le gaz naturel liquéfié (GNL) offre un coût total sur 20 ans inférieur, soit 117 325 972,00 $, par rapport au gaz naturel comprimé (GNC), dont le coût s'élève à 142 026 400,00 $, il nécessite un investissement initial très élevé, soit 17 911 572,00 $. En revanche, le GNC a un CAPEX beaucoup plus faible, soit 34 000,00 $, ce qui le rend beaucoup plus accessible pendant les premières phases du projet.

Cela représente une différence de près de 17,9 millions de dollars en coûts initiaux. Ce capital pourrait être réaffecté à d'autres domaines stratégiques du projet ou investi pour générer un rendement plus élevé, en particulier pendant les premières phases, où la flexibilité des flux de trésorerie est cruciale.

Par conséquent, malgré des coûts d'exploitation plus élevés à long terme, nous recommandons d'opter pour le gaz naturel comprimé (GNC) en raison de son coût d'entrée moins élevé et de la possibilité d'optimiser le déploiement du capital pour obtenir de meilleurs rendements initiaux et une plus grande agilité financière.

APPENDIX

4-5 SITE SELECTION ALTERNATIVES FOR STORAGE OF MINE WASTE

Patriot Battery Metals

Shaakichiuwaanaan Project

Eeyou Istchee James Bay Region, QC

Technical Report

Site Selection Alternatives for Storage of Mine Waste

BBA Document No. / Rev.: 7535027-000000-41-ERA-0001-R00

June 20, 2025

FINAL

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APPENDICES

Appendix A: Workshop Presentation

Appendix B: CASN-A-370-GT-DCR-0001-B_Waste and water management Design criteria.pdf (V2)

Appendix C: POETS

1. Project description

The Shaakichiuwaanaan Property (the "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. 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 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.

In September 2024, Patriot Battery Metals ("Patriot") published a NI 43-101 Preliminary Economic Assessment (PEA) for the Shaakichiuwaanaan Project (Patriot, 2024). This resource makes the CV5 spodumene pegmatite one of the ten largest lithium pegmatite deposits in the world, as well as one of the largest tantalum-pegmatite deposits in the world. The Project will be mined using a hybrid open pit and underground mining method.

2. Introduction

It is forecasted that the mining operation will generate a total of 187 Mt of waste rock from the extraction process. Approximately 80 Mt of the waste rock is considered potential acid generating or potentially leaching (high risk) and 107 Mt with low potential (low risk) of waste rock of acid generation. In addition, it is estimated that the process plant will generate 41 Mt of tailings. These waste streams will be managed separately, depending on their characteristics. The quantity expressed here were given by the Client at the moment of the report development. These quantities are all to be placed on surface; the materials used as backfill for underground is out of the scope of this study. Low risk waste rock, also called non-acid generating, or NAG, typically does not produce acid when exposed to air and water. PAG waste rock, on the other hand, tend to react with air and water to produce acid drainage. The volumes of the materials were taken from the document CASN-A-370-GT-DCR-0001-B_Waste and water management Design criteria.pdf (V2).

Table 1: Streams and mitigation methods

Streams	Mitigation Methods	Capacity*	Dry Density	Estimated Volume
		Mt	t/m³	Mm³
Tailings to put on surface	Membrane	41	1.7	24.1
Waste Rock (Low-Risk)	No Membrane	107	2.0	53.5
Waste Rock (Potentially Acid generating)	Membrane	80	2.0	40
Total				**117.6**

Patriot has request BBA's continued participation in developing the site selection for the waste rock and tailings storage facilities of the Shaakichiuwaanaan Project.

This mandate, includes:

- The selection process considering project context and different constraints identified by the Client and BBA.
- A layout for the waste management scenarios summarizing the selected locations for material storage.
- Undertaking a multicriteria selection process resulting in the section of a particular site based on selection criteria and pre-determined scoring. A preliminary version has been developed and used during the site selection workshops. Three (3) site selection workshops were undertaken during the course of this work which have been summarized in the presentation provided in Appendix A.
- A technical report describing the methodology and the results of the site selection.

The associated cost estimates scoped level in nature (±50%) but can be used for decision making purposes.

The site selection process followed directives and best management principles stipulated and recommended by:

- Federal Guidelines for the assessment of alternatives for mine waste disposal;
- Québec Mining Laws and Regulations Applicable to Tailings and Mining Waste;
- The Mining Association of Canada;
- The Global Industry Standard on Tailings Management.

The site selection process prioritized environmental integrity alongside economic and technical factors.

3. Methodology

Project was executed including the Client's sustainability mission and environmental goals as per the following steps:

3.1 Sites and alternatives pre-selection

Validation of proposed locations were analyzed, and a pre-screening process was carried out to select the scenarios to be evaluated. A total of 31 potential sites over a 10 km radius from the mill were identified. Figure 1 illustrates the location of the of the proposed sites forming part of the pre-screening process.

Some details regarding the screening process are as follows:

- Zones of priority were defined for site investigation (May- Sept 2024);
- PMET, WSP and BBA contributed to the pre-selection process;
- Dry stack tailings technology was considered as the basis for disposal of tailings because the envisioned process is coarse Dense Media Separation (DMS);
- Limited to 10 km radius from mill;
- Material waste balances from the PEA study were used as a basis of estimation (not considering backfilling waste into the pit), with the exception of potential metal leaching waste tonnage that is coming from another document;
- Minimum size for waste option (pile) based on DMS tails stream (smallest stream); the minimum volume considered is based on assumptions from the PEA;
- Considered slope angle of 18.4° (3:1 Slope H:V) for design volumes of all options;
- Height of each option considered a minimum 50 m by 50 m area at crest;
- Tailings density of 1.7t/m^3 (compacted in pile);
- Considered minimum offset of 100 m from lakes, rivers and bodies of water (Directive 019 = 60 m offset);
- Claims boundaries not considered as limiting factor for pre-selection process.

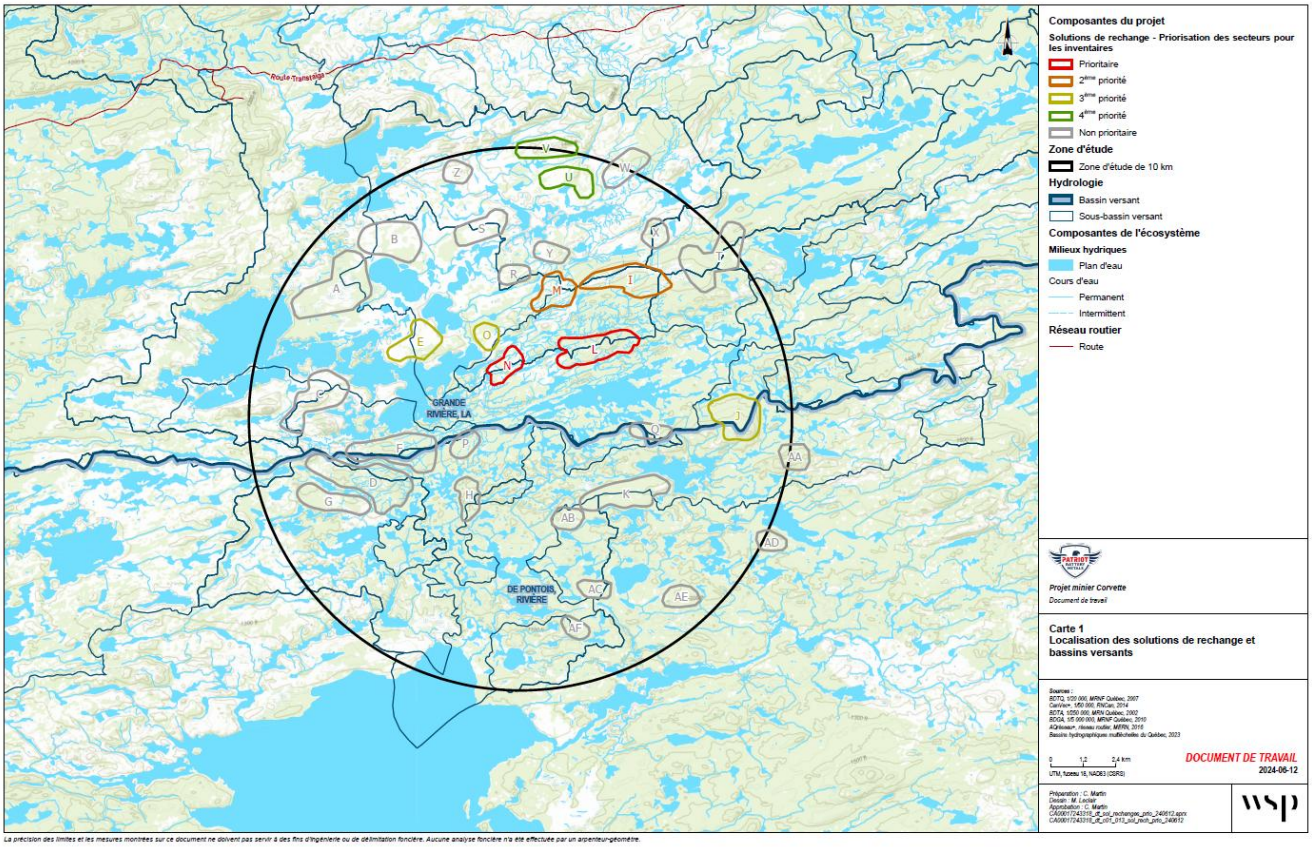


Figure 1: Pre-screening site options

Pre-screening of these potential sites were based on the following criteria:

- Waste deposition in multiple water sheds (basin versant);
  - Limited the waste disposal options to the northern watershed to avoid the impact of multiple watersheds.
- Potential for future mineralization;
  - Options along strike of mineralization zone eliminated (CV 9, CV13…).
- Eliminated options in proximity to known fish habitats, communities and archeological sites based on WSP constraint maps provided March 14th, 2024;
- Options selected in priority order for investigation based on comparison of distance from pit, number of river crossings and presence of favourable topography;
- Four (4) scenarios were identified due to their proximity to the forecasted access road location.

The result of the pre-screening process yielded a total of five (5) credible scenarios which fit the design needs for the project but differed in one or more aspects, as a result of the proposed locations. The retained five (5) scenarios, subsequent to the pre-screening process, are illustrated in Figure 2.

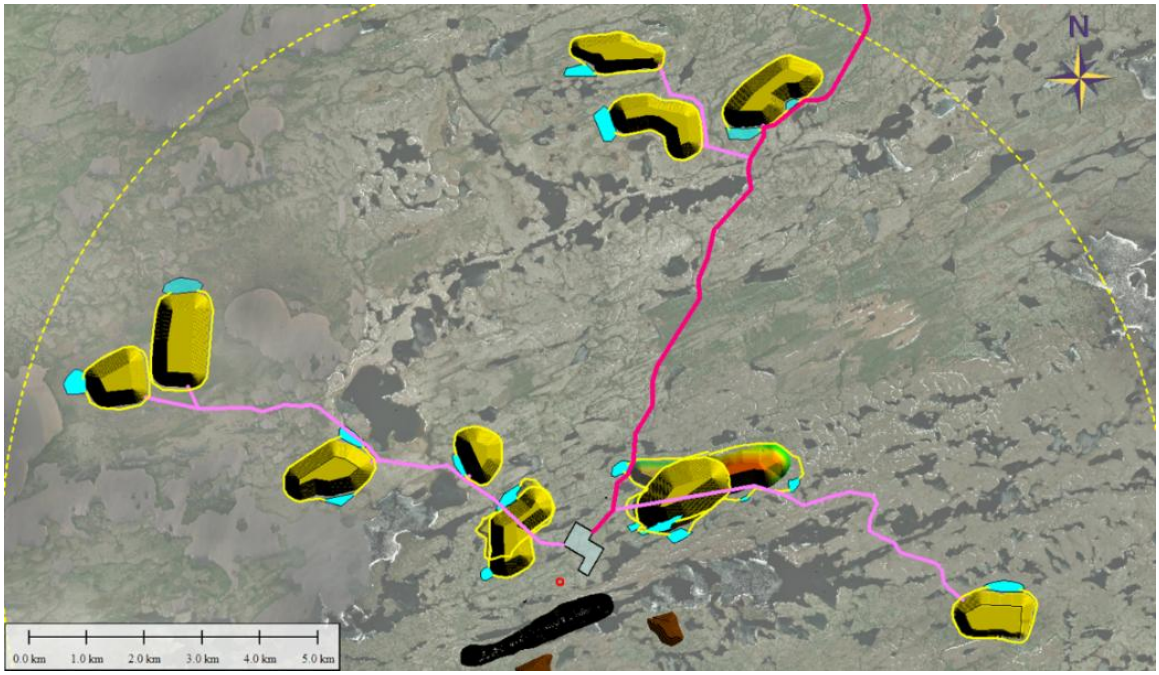


Figure 2: Scenarios retained

3.2 Infrastructure conceptualization

The scenarios retained for analysis were conceptualized in Global Mapper considering the following criteria:

- Waste rock (WR) stockpile sizing, including volumetric compliance;
- Water management consideration for required infrastructure;
- Constraints analysis;
- Layout per each scenario;
- High-level costing for each scenario.

All conceptualized models conform with project requirements and are shown in Figure 3 to Figure 7.

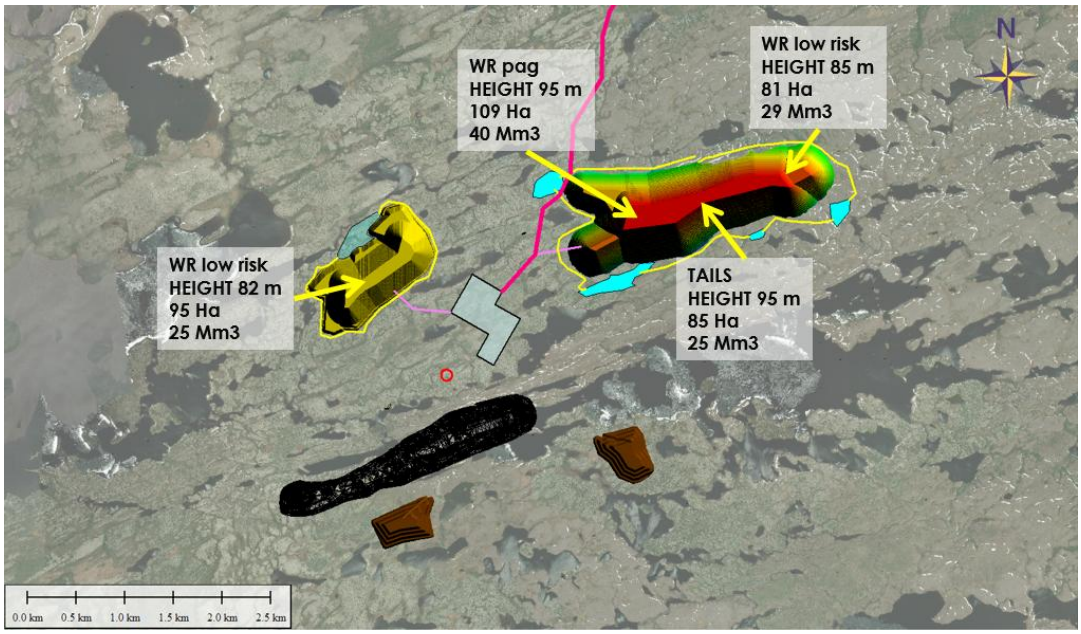


Figure 3: Conceptualized model - Scenario 1

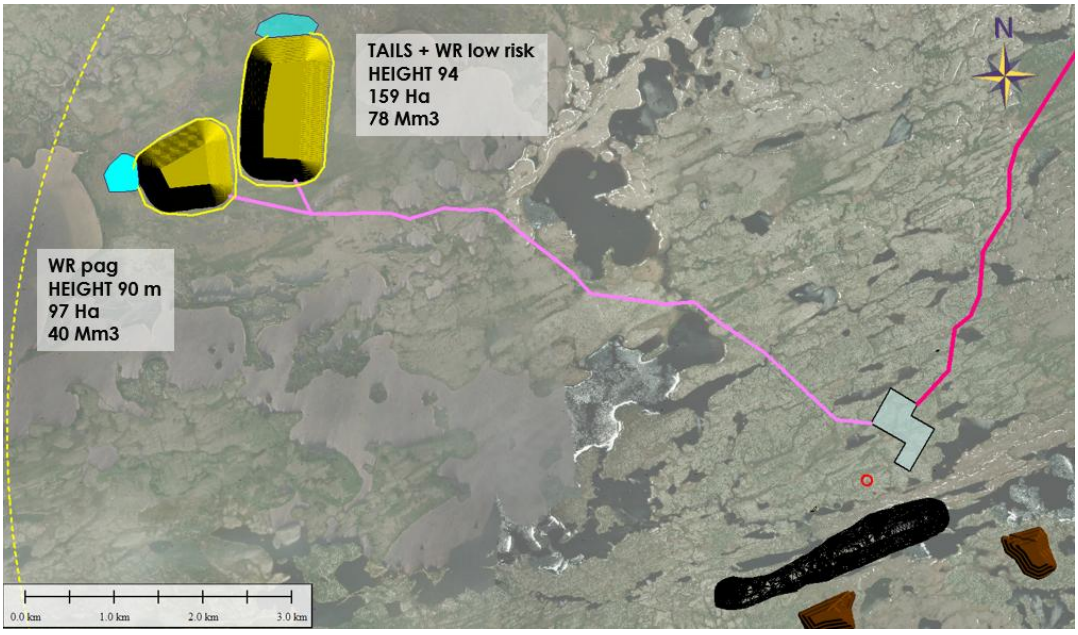


Figure 4: Conceptualized model - Scenario 2

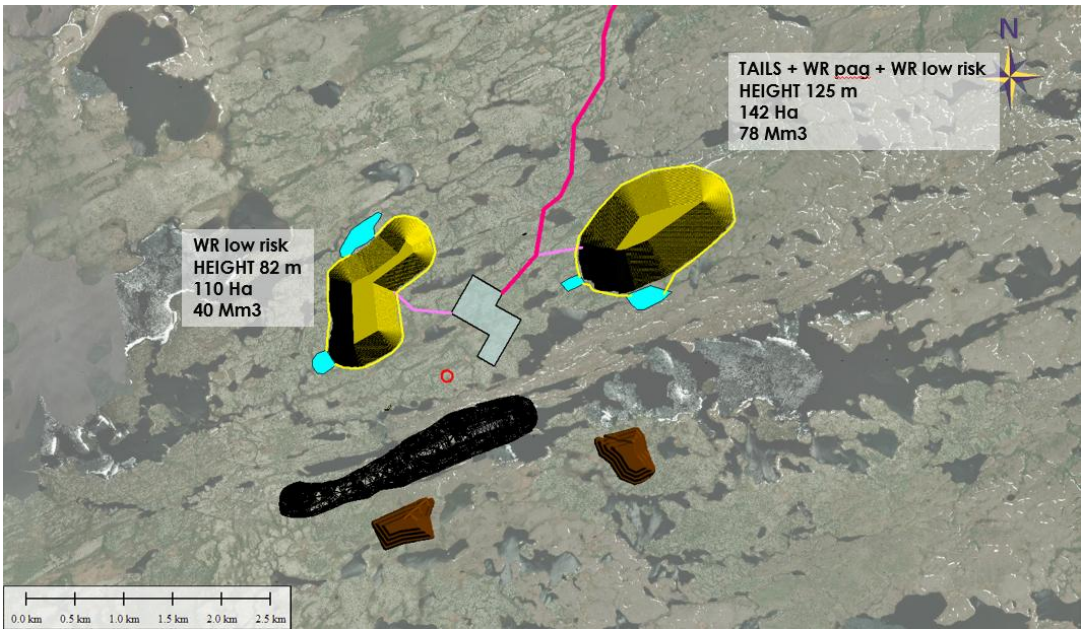


Figure 5: Conceptualized model - Scenario 3

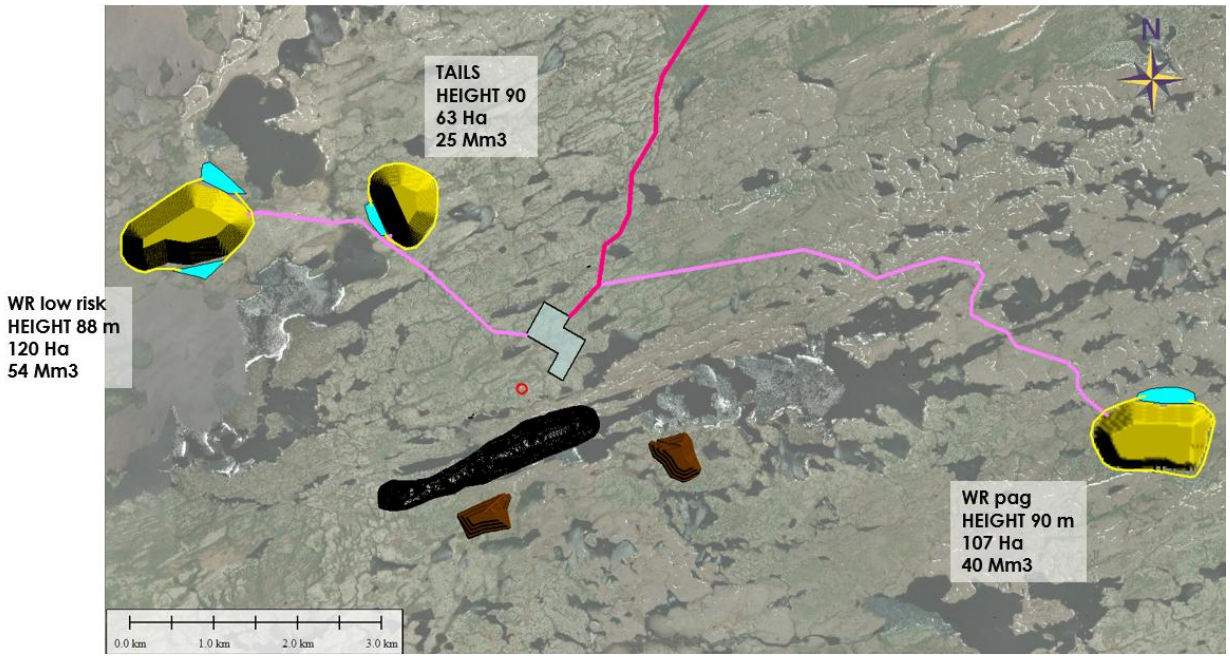


Figure 6: Conceptualized model - Scenario 4

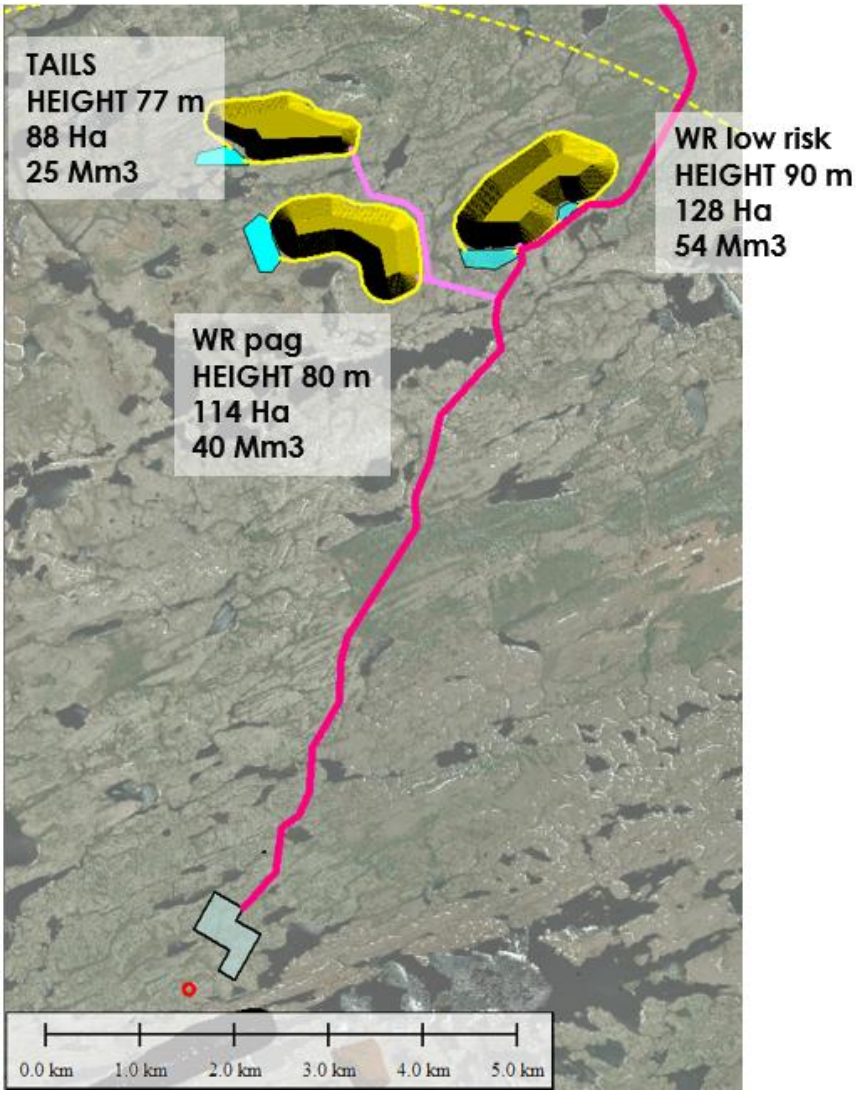


Figure 7: Conceptualized model - Scenario 5

3.3 Input data

Innova provided BBA with project constraints maps. Information consisted of environmental constraints, sensitive and unique ecosystems, surface water bodies, planned mining infrastructure, other infrastructure (roads), property limits, and mineral resources, population distribution (human settlements). Table 2 presents the list of the maps provided by Innova and used in the site selection process.

Table 2: Maps provided by Innova

Map	Plans or document	Notes
Wetlands/ Rivers / Streams	CORVETTE_BIO_carac_milieu_s_011	Has information for all the scenarios but the level of detail is only high for scenarios 1 and 3
	CORVETTE_BIO_hydrog_l_006	Has information for all the scenarios but the level of detail is only high for scenarios 1 and 3
	Fish Habitat_20241023	Only has information on the areas for scenarios 1 and 3
Proximity to sensitive receptors	160846E.pdf	Public information
Moose feeding areas and goose feeding areas	**Confidential**	**Confidential**
Archaeological potential	Aire d'étude et d'inventaire - archaeo	Only has information on the areas for scenarios 1 and 3
Mine description and criteria	CASN-A-370-GT-DCR-0001-B_Waste and water management Design criteria.pdf	

It is to note that some of the maps had a high level of detail in the direct area of influence of the project established in the preliminary phases and mapped during ongoing site investigation performed in 2024. Some of the scenarios analyzed are considered to have a lower level of detail than the areas directly surveyed during the site investigation. This bias in the level of detail between scenarios may impact or influence the final results of site investigation analysis.

The GIS information, the field surveys, stakeholder consultations and the 3D models developed where used to evaluate the sites.

4. Multicriteria decisional matrix set-up

BBA has developed a decision matrix tool that will be used during the risk analysis stage, aimed at facilitating this structured decision. The "POETS" tool offers an easy-to-use interface and offers a complete selection of criteria for an effective evaluation of options.

The POETS follows the philosophy of the federal guidelines for site selection (Guidelines for the assessment of alternatives for mine waste disposal).

In preparation for the workshop, the POETS matrix will be filled in beforehand by BBA and revised jointly with the Client. At this stage, BBA will consolidate the information and the analyses for each option to regroup all the information within POETS. BBA and the Client will then be able to demonstrate the degree of due diligence undertaken of the various concepts and locations.

The decision matrix tool is divided into five (5) sections, each with a drop-down list of selection criteria. The weighting of each component is discussed and decided with the Client, depending on the needs and constraints of the project.

POETS is a decision matrix tool that has enabled BBA to facilitate the Client's decision-making within various high-level studies, PEA, PFS, and FS studies that account for the following sections:

- **P**roject costs
- **O**perations considerations
- **E**nvironmental impacts
- **T**echnical Risks
- **S**ocio-economic impacts

The proposed selection study was conducted at a sufficient level of detail so that the selected scenario can be incorporated into PMET's overall project evaluations.

4.1 Matrix set-up

BBA prepared the structure of the multiple criteria analysis matrix. PMET validated evaluation criteria selection and weights prior to the commencement of the workshops.

The resulting evaluation criteria list and weights are presented in Table 3.

Table 3: Weighting of accounts and sub-accounts

Total Weight %	Weight in total %	Account	Weight in account %	Criteria Description	Individual criteria %	Total Weight %
100%	30%	Project Economics	50%	CAPEX	15.0%	100%
			25%	OPEX	7.5%	
			20%	Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	6.0%	
			5%	Fish habitat compensation	1.5%	
	10%	Operational Consideration	47%	Distance from mill/pit	4.7%	
			32%	Expansion capability	3.2%	
			21%	Operational safety	2.1%	
	25%	Environmental Impacts	30%	STOCKPILE footprint	7.5%	
			20%	CO2 emissions	5.0%	
			30%	Wetlands/ Rivers / Streams	7.5%	
			20%	Fish presence	5.0%	
	10%	Technical Consideration	35%	Water management infrastructure	3.5%	
			20%	water management complexity	2.0%	
			15%	Project schedule & Constructability	1.5%	
			30%	Embankment (earthwork) height	3.0%	
	25%	Social Impacts	25%	Proximity to sensitive receptors (cabins and travel routes)	6.3%	
			25%	moose feeding areas and goose feeding areas	6.3%	
			25%	Archaeological potential	6.3%	
			25%	Fishing areas	6.3%	

In addition to establishing weighting for each of the sub-accounts, and in an effort to ensure that the criteria be applied consistently and without bias, a scorecard applicable to each criterion in the sub-accounts was elaborated and agreed upon with PMET. The scorecard is presented in Figure 8.

TABLE: TSF ACCOUNTS, SUBACCOUNTS AND SCORING CRITERIA

ACCOUNT		SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
PROJECT ECONOMICS	1	CAPEX	cost	Alternative less costly	Increase of cost of 15%	Increase of cost of 30%	Increase of cost of 45%	Increase > 45%
	2	OPEX	cost	Alternative less costly	Increase of cost of 15%	Increase of cost of 30%	Increase of cost of 45%	Increase > 45%
	3	Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	Qualitative	The works can be spread in at least 3 phases with more than 30% of capex beyond year 0	The works can be spread in at least 3 phases with less than 30% of capex beyond year 1	The works can be spread in at least 2 phases with more than 30% of capex beyond year 0	The works can be spread in at least 2 phases with less than 30% of capex beyond year 1	All the works have to be done in one phase in year 0
	4	Fish habitat compensation	Qualitative	No significant risks identified, with all potential costs accounted for or within the applied contingency. Recommended contingency 15%	Limited risks identified which would minimally require an increase of the project estimation contingency. Most costs are well-documented but may need slightly greater contingency. Recommended contingency 25%	Several risks identified which would marginally require an increase of the project estimation contingency. Costs are well-documented but may need greater contingency. Recommended contingency 35%	Several risks identified which would significantly require an increase of the project estimation contingency. Some costs are well-documented but need greater contingency. Recommended contingency 50%	Numerous risks have been identified such that contingencies need to be increased to level beyond the estimation precision. Recommended contingency more than 50%
OPERATIONS	1	Distance from mill/pit	distance in km	0-2 km	2-4 km	4-6 km	6-8 km	> 8 km
	2	Expansion capability	percentage of expansion	40% or more additional volume can be deposited in the site option.	30 - 40% additional volume can be deposited in the site option.	20 - 30% additional volume can be deposited in the site option.	10 - 20% additional volume can be deposited in the site option.	less than 10% additional volume can be deposited in the site option.
	3	Operational safety	distance of roads in km	Minimal Hazard Exposure due to the very short distances (0-2 km) between the sites and the pit or the mill, these distances do not have measurable effect on operator well-being or safety.	Low Hazard Exposure due to the short distances (2-4 km) between the sites and the pit or the mill, these distances have minimal effect on operator fatigue or performance.	Moderate Hazard Exposure due to the intermediate distance (4-6 km) between the sites and the pit or the mill, fatigue and reduced performance are common due to travel distances but manageable with breaks.	High Hazard Exposure due to the long distances (6-8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.	Extreme Hazard Exposure due to the very long distances (>8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.
ENVIRONMENTAL	1	STOCKPILE footprint	Footprint surface	Alternative with less footprint surface	Increase of footprint of 15%	Increase of footprint of 30%	Increase of footprint of 45%	Increase of footprint of 100%
	2	CO2 emissions	Distance of roads in kilometers	The project generates minimal CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (0-1Km)	The project generates below-average CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (1-2Km)	The project generates a moderate amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (2-3Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (3-4Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (> 4Km)
	3	Wetlands/ Rivers / Streams	Surface of wetlands and water streams	less than 10 Ha	10 To 20 Ha	20 to 30 Ha	30 to 40 Ha	More than 40 Ha
	4	Fish presence	Presence of fish	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat
TECHNICAL	1	Water management infrastructure	watershed surface	Alternative with less watershed surface to manage	Increase of watershed of 15%	Increase of watershed of 30%	Increase of watershed of 45%	Increase of watershed of 100% or more
	2	water management complexity	Water surface	Site water management can be done in one water management system and water drainage done by gravity	Site water management can be done in one water management systems requiring on or more sump with ongoing pumping	Site water management can be done in two water management systems and water drainage done by gravity	Site water management can be done in two water management systems requiring on or more sump with ongoing pumping	Site water management would require more than 2 water management system
	3	Project schedule & Constructability	distance, % of wetlands or water streams and area of basins	Site that is closest to the central camp, where the work area impacts the least area of wetlands or water surfaces and has the least surface of basins, and the topography is not hilly. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is within 1km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 20% of the best alternative related to wetlands and basin surface. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is within 3km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 40% of the best alternative related to wetlands and basin surface. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is within 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces within 50% of the best alternative related to wetlands and basin surface. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is more 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces grater 50% of the best alternative related to wetlands and basin surface.
	4	Embankment (earthwork) height	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 degree if height is equal or less than 90m	Increase of height of 20 m Increase the score by 1 degree if height is equal or less than 90m	Increase of height of 30 m Increase the score by 1 degree if height is equal or less than 90m	Increase of height of > 30 m Increase the score by 1 degree if height is equal or less than 90m
SOCIAL	1	Proximity to sensitive receptors (cabins and travel routes)	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km
	2	moose feeding areas and goose feeding areas	distance in km Hunting ground	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km
	3	Archaeological potential	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area
	4	Fishing areas	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

Figure 8: Site evaluation scorecard

4.1.1 Options assessment process

The evaluation was performed in two(2) steps:

- Initially, BBA prepared the matrix and identified the metrics to support scoring and assessment of each sub-account. A draft document was issued to all stakeholders. BBA undertook an initial assessment of all alternatives which served as a basis for discussion in three (3) workshops conducted with stakeholders identified by PMET.
- Final evaluation was performed during workshops with Innova Lithium inc. In these work sessions, participants validate or modify the proposed sub-account scores.

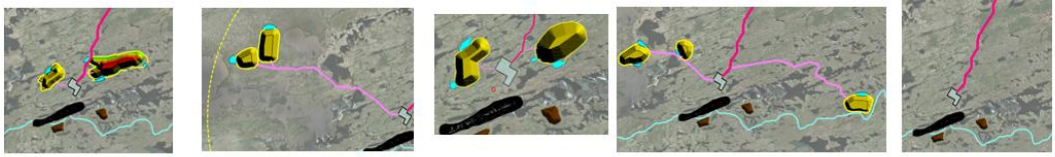
5. Workshop results

5.1 Project economics analysis

Project economics has been evaluated using four (4) sub-accounts, namely:

- **Capital Expenditure (CAPEX):** This sub-account is the economical value to acquire, upgrade or maintain the physical assets in each scenario. To evaluate the CAPEX, the considerations were structure of the foundation, water management infrastructure and roads. The calculation was done using the information provided by PMET, the 3D models developed by BBA and the unit costs from BBA; these unit costs were obtained from similar projects done by BBA with similar characteristics.
- **Operating Expenditure (OPEX)**: This sub-account is the economical factor to maintain operation of the project. To evaluate the OPEX, the consideration was length of road to transport material. The calculation was done using the information provided by PMET, the 3D models developed by BBA and the unit cost from BBA; these unit costs were obtained from similar projects done by BBA with similar characteristics.
- **Phasing possibility**: Cashflow. This sub-account qualifies ability to spread CAPEX over time. This allows to distribute through time, the road construction and the water management construction.
- **Fish habitat compensation:** This sub-account measures the economical value of using fish habitats for the project. This metric is being considered as these costs would need to be expended at the front end of the project.

The scoring for each scenario and criterion is provided in Figure 9 through Figure 12, respectively.



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
CAPEX (MILLON DOLLARS)	$ 184.4	$ 212.9	$ 131.6	$ 188.0	$ 203.6
CAPEX (% OF INCREASE)	40%	62%	0%	43%	55%

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
1	CAPEX	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Very Poor	Very Good	Poor	Very Poor

Figure 9: Site evaluation - CAPEX

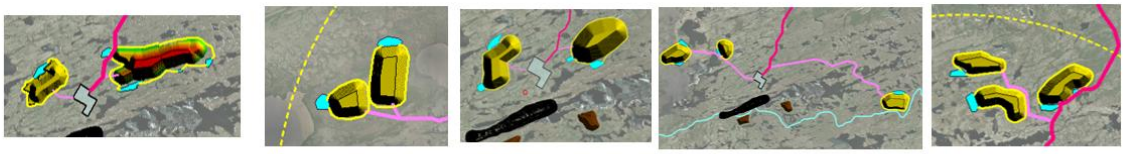


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
OPEX (Km of road to transport material)	1.20	8.54	1.20	12.00	10.90
OPEXO(% Of increase)	0%	612%	0%	900%	808%

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
1	OPEX	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Poor	Very Good	Very Poor	Very Poor

Figure 10: Site evaluation - OPEX

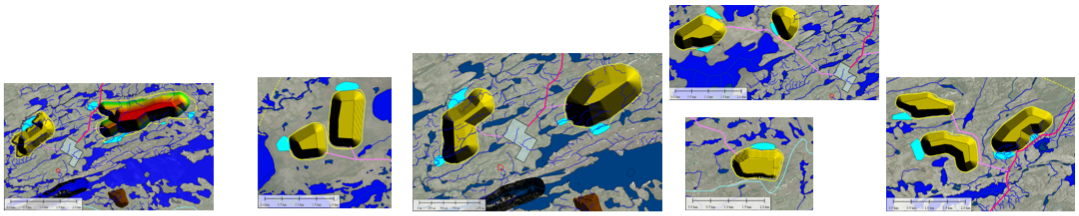


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	3 phases with less than 30% of capex beyond year 1	1 phase	2 phases with less than 30% of capex beyond year 1	2 phases with less than 30% of capex beyond year 1	2 phases with less than 30% of capex beyond year 1

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)**	Qualitative	The works can be spread in at least 3 phases with more than 30% of capex beyond year 0	The works can be spread in at least 3 phases with less than 30% of capex beyond year 1	The works can be spread in at least 2 phases with more than 30% of capex beyond year 0	The works can be spread in at least 2 phases with less than 30% of capex beyond year 1	All the works have to be done in one phase in year 0

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Very Poor	Poor	Poor	Poor

Figure 11: Site evaluation – Project Phasing



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
SURFACE OF WETLANDS (Ha)	45.7	142.1	42.4	9.2	17.6
SURFACE OF WATER STREAMS (Ha)	22.8	0	29.2	0.1	0.7
Total area wetlands and water	68.5	142.1	71.6	9.3	18.3

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	Fish habitat compensation	Surface of wetlands and water streams	less than 10 Ha	10 To 20 Ha	20 to 30 Ha	30 to 40 Ha	More than 40 Ha

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Very Poor	Very Poor	Very good	Good

Figure 12: Site evaluation – Fish habitat compensation

This account represents 30% of the total weight. The result shows that the alternative that scored highest was Scenarios 3, with a score of 84%, followed by Scenario 1, scoring at 62%. The remaining options all scored below 40%.

Whereas Scenarios 1 and 3 rank 1 and 2, respectively, on this criterion, Scenario 3 ranks Very Good for CAPEX, compared to Scenario 1 which ranks Poor.

The total OPEX cost, will be significantly higher for Scenarios 2, 4 and 5 and is reflected in the grading.

Results of the account and sub-account scoring are presented in Table 4.

Table 4: Overall site evaluation – Project economics

Project Economics													
POETS				Rating									
Decision Criteria			Weight	Scenario S1		Scenario S2		Scenario S3		Scenario S4		Scenario S5	
Principal Account	Sub Account												
			100%	Points	Rating	Points	Rating	Points	Rating	Points	Rating	Points	Rating
Project Economics	1	CAPEX	50	Poor	20%	Very Poor	10%	Very Good	50%	Poor	20%	Very Poor	10%
	2	OPEX	25	Very Good	25%	Very Poor	5%	Very Good	25%	Very Poor	5%	Very Poor	5%
	3	Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	20	Good	16%	Very Poor	4%	Poor	8%	Poor	8%	Poor	8%
	4	Fish habitat compensation	5	Very Poor	1%	Very Poor	1%	Very Poor	1%	Very Good	5%	Good	4%
Total			100		62%		20%		84%		38%		27%

5.1.1 Operational considerations

Operational consideration has been evaluated using three (3) sub-accounts, namely:

- **Distance from mill/pit:** This sub-account is an indicator of the distance of the material and its origin.
- **Expansion capability:** This sub-account exposes the potential of each scenario to receive more material if needed.
- **Operational safety**: This sub-account confirms the commitment of PMET with the safety of the operation and the operators, using the factor of time exposure to the hazards during material transportation.

The scoring for each scenario and criterion is provided in Figure 13 through Figure 15, respectively.

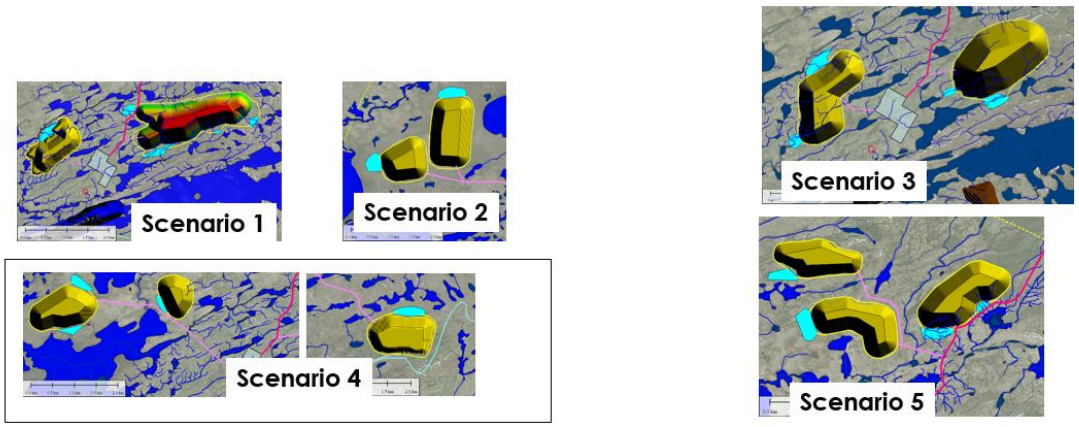


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Distance from mill/pit (the farthest pile) (km)	3.1	8.4	2.5	7.5	9.1

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
1	Distance from mill/pit	distance in km	0-2 km	2-4 km	4-6 km	6-8 km	> 8 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Very Poor	Good	Poor	Very Poor

Figure 13: Site evaluation – Distance from mill



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Expansion capability	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies	10 - 20% going higher and enlarging the footprint and no affecting more water bodies	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
3	Expansion capability	percentage of expansion	40% or more additional volume can be deposited in the site option.	30 - 40% additional volume can be deposited in the site option.	20 - 30% additional volume can be deposited in the site option.	10 - 20% additional volume can be deposited in the site option.	less than 10% additional volume can be deposited in the site option.

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Very Poor	Poor	Very Poor	Very Poor

Figure 14: Site evaluation – Expansion capacity



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Operational safety (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	Operational safety	distance of roads in km	Minimal Hazard Exposure due to the very short distances (0-2 km) between the sites and the pit or the mill, these distances do not have measurable effect on operator well-being or safety.	Low Hazard Exposure due to the short distances (2-4 km) between the sites and the pit or the mill, these distances have minimal effect on operator fatigue or performance.	Moderate Hazard Exposure due to the intermediate distance (4-6 km) between the sites and the pit or the mill, fatigue and reduced performance are common due to travel distances but manageable with breaks.	High Hazard Exposure due to the long distances (6-8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.	Extreme Hazard Exposure due to the very long distances (>8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Poor	Very Good	Very Poor	Very Poor

Figure 15: Site evaluation – Operational safety

This account counts for 10% of the total weight. The result shows that the alternative that scored highest was Scenario 3 with a score 71%, followed by Scenario1, scoring 65%. The remaining alternative scored below 30%.

Scenarios 1 and 3 had similar scores based on distance from the mill, while Scenario 1 scored more poorly on the potential for expandability when compared to Scenario 3. The other options lost most of their grading due to their distance from the mill. Results of the account and sub-account scoring are presented in Table 5.

Table 5: Site evaluation – Operational considerations

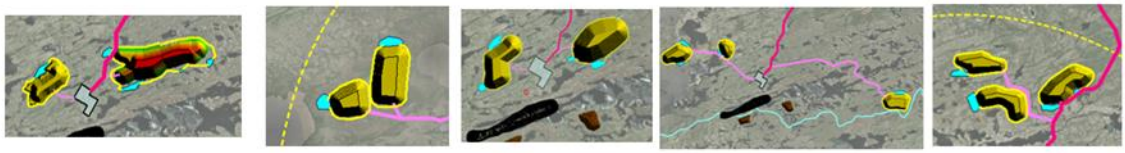
Operational Consideration													
POETS				Rating									
Decision Criteria			Weight	Scenario S1		Scenario S2		Scenario S3		Scenario S4		Scenario S5	
Principal Account	Sub Account												
			100%	Points	Rating	Points	Rating	Points	Rating	Points	Rating	Points	Rating
Operational Consideration	1	Distance from mill/pit	47	Good	38%	Very Poor	9%	Good	38%	Poor	19%	Very Poor	9%
	2	Expansion capability	32	Very Poor	6%	Very Poor	6%	Poor	13%	Very Poor	6%	Very Poor	6%
	3	Operational safety	21	Very Good	21%	Very Poor	4%	Very Good	21%	Very Poor	4%	Very Poor	4%
Total			100		65%		20%		71%		29%		20%

5.1.2 Environmental impacts

Environmental Impacts have been evaluated using four (4) sub-accounts, namely:

- **STOCKPILE footprint**: This sub-account is an indicator of the surface required in each scenario.
- **CO_2 emissions**: This sub-account varies based on the travel distance for each scenario, assuming the mining trucks CO_2 emissions will increase with transport distance.
- **Wetlands/ Rivers / Streams**: This sub-account highlights the importance of the soil classification and the water bodies on the surface of the scenarios.
- **Fish presence**: This sub-account identifies the areas with fish presence and compares it with the footprint of the scenarios. This is a very important sub-account; it was the center of discussions in the workshops and will play a key roll in the conclusions and recommendations in this report.

The scoring for each scenario and criterion is provided in Figure 16 through Figure 19, respectively.

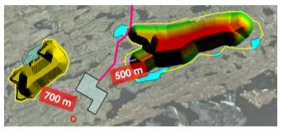


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
STOCKPILE footprint (Ha)	364	256	252	290	330
(% OF INCREASE)	44%	1%	0%	15%	31%

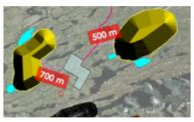
	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**STOCKPILE footprint**	Footprint surface	Smallest footprint surface area	Increase of footprint by 15%	Increase of footprint by 30%	Increase of footprint by 45%	Increase of footprint by 100%

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Very Good	Very Good	Good	Fair

Figure 16: Site evaluation – Stockpile footprint









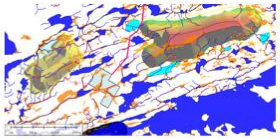


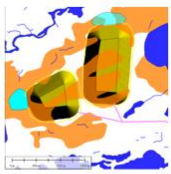
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
CO2 emissions (the longer road) Km	1.2	8.54	1.2	12	10.9

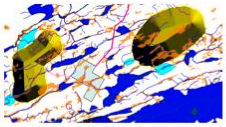
	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
2	**CO2 emissions**	Distance of roads in kilometers	The project generates minimal CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (0-1Km)	The project generates below-average CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (1-2Km)	The project generates a moderate amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (2-3Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (3-4Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (> 4Km)

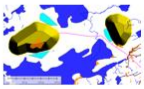
SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Very Poor	Good	Very Poor	Very Poor

Figure 17: Site evaluation – CO_2 emissions

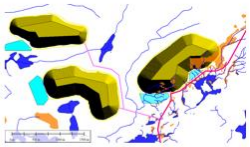










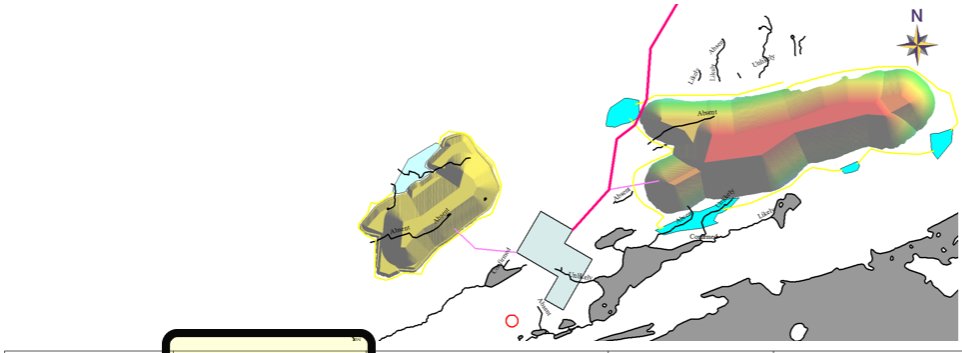


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
TOTAL SURFACE (Ha)	381	268	265	302	344
SURFACE OF WETLANDS	45.7	142.1	42.4	9.2	21.4
SURFACE OF WATER STREAMS	22.8	0	29.2	0.1	0.7
Total area wetlands and water	68.5	142.1	71.6	9.3	22.1

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Wetlands/ Rivers / Streams**	Surface of wetlands and water streams	less than 10 Ha	10 To 20 Ha	20 to 30 Ha	30 to 40 Ha	More than 40 Ha

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Very Poor	Very Poor	Very Good	Fair

Figure 18: Site evaluation – Wetlands



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Fish presence	Confirmed no fish habitat	No information available	Confirmed fish habitat	No information available	No information available

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fish presence**	Presence of fish	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Fair	Very Poor	Fair	Fair

Figure 19: Site evaluation – Fish presence

This account counts for 25% of the total weight. The results show that the alternative that scored highest was Scenario 4 with a rating of 70%, with Scenario 5 coming in second with a rating of 58%. Scenarios 1, 2 and 3 all scored similarly with ratings 54%, 52% and 56%, respectively. These latter scenarios lost most of their scoring due to their perceived impacts on wetlands and potential fish presence when compared to Scenarios 4 and 5.

Results of the account and sub-account scoring are presented in Table 6.

Table 6: Site evaluation – Environmental impacts

Environmental Impacts												
POETS				Rating								
Decision Criteria			Weight	Scenario S1		Scenario S2		Scenario S3		Scenario S4		Scenario S5	
Principal Account		Sub Account	100%	Points	Rating	Points	Rating	Points	Rating	Points	Rating	Points	Rating
Environmental Impacts	1	STOCKPILE footprint	30	Poor	12%	Very Good	30%	Very Good	30%	Good	24%	Fair	18%
	2	CO2 emissions	20	Good	16%	Very Poor	4%	Good	16%	Very Poor	4%	Very Poor	4%
	3	Wetlands/ Rivers / Streams	30	Very Poor	6%	Very Poor	6%	Very Poor	6%	Very Good	30%	Fair	18%
	4	Fish presence	20	Very Good	20%	Fair	12%	Very Poor	4%	Fair	12%	Fair	12%
Total			100		54%		52%		56%		70%		52%

5.1.3 Technical consideration

Technical consideration has been evaluated using four (4) sub-accounts, namely:

- **Water management infrastructure:** This sub-account measures the quantity of infrastructure needed for the water management of the scenarios.
- **Water management complexity:** This sub-account evaluates the number of water systems required for the scenarios.
- **Project schedule & constructability**: This sub-account has three (3) components: the distance to the source of material, the topography factor and the delay risk in the schedule of the project.
- **Embankment (earthwork) height:** This sub-account has two (2) components: the soil stability and the risk of acid generation; the higher the embankment the higher those risks.

The scoring for each scenario and criterion is provided in Figure 20 through Figure 23, respectively.



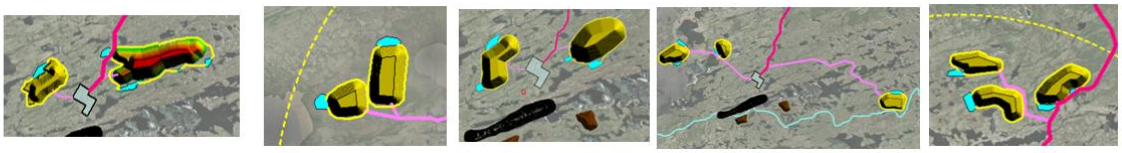
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Water management infrastructure (stock pile+basin(Ha))	381	268	265	302	344
(% OF INCREASE)	44%	1%	0%	14%	30%

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Water management infrastructure**	watershed surface	Alternative with less watershed surface to manage	Increase of watershed of 15%	Increase of watershed of 30%	Increase of watershed of 45%	Increase of watershed of 100% or more

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Very Good	Very Good	Good	Fair

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Figure 20: Site evaluation – Water management infrastructure

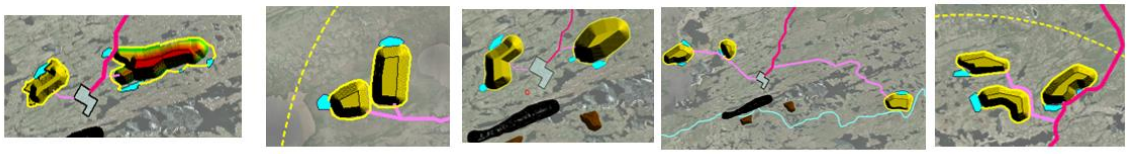


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
water management complexity (number of basins)	5 (by gravity)	2 (by gravity)	4 (by gravity)	4 (by gravity)	4 (by gravity)

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
2	**water management complexity**	Water surface	Site water management can be done in one water management system and water drainage done by gravity	Site water management can be done in one water management systems requiring on or more sump with ongoing pumping	Site water management can be done in two water management systems and water drainage done by gravity	Site water management can be done in two water management systems requiring on or more sump with ongoing pumping	Site water management would require more than 2 water management system

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Fair	Very Poor	Very Poor	Very Poor

Figure 21: Site evaluation – Water management complexity

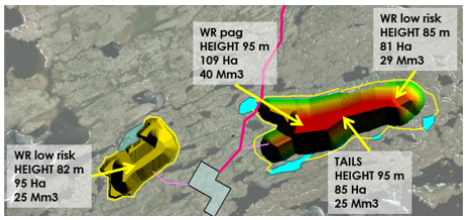


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
DISTANCE TO CENTRAL CAMP (KM)	3.1	8.4	2.5	7.5	9.1
WETLANDS AND WATER SURFACE (HA)	68.5	142.1	71.6	9.2	17.6
BASINS	5	2	4	4	4

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Project schedule & Constructability**	distance, % of wetlands or water streams and area of basins	Site that is closest to the central camp, where the work area impacts the least area of wetlands or water surfaces and has the least surface of basins, and the topography is not hilly. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is within 1km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 20% of the best alternative related to wetlands and basin surface. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is within 3km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 40% of the best alternative related to wetlands and basin surface. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is within 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces within 50% of the best alternative related to wetlands and basin surface. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is more 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces grater 50% of the best alternative related to wetlands and basin surface.

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Poor	Poor	Very Poor	Very Poor

Figure 22: Site evaluation – Schedule and constructability



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Embankment (earthwork) height (m)	TAILS-HEIGHT 95 m WR pag-HEIGHT 95 m WR low risk-HEIGHT 85 m	WR pag-HEIGHT 90 m TAILS + WR low risk-HEIGHT 94	WR (low risk) H=82m TAILS + WR low risk + pag (125m)	WR low risk-HEIGHT 88 m TAILS-HEIGHT 90 WR pag-HEIGHT 90 m	TAILS-HEIGHT 77 WR pag-HEIGHT 80 m WR low risk-HEIGHT 90 m

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Embankment (earthwork) height**	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)	Increase of height of 20 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)	Increase of height of 30 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)	Increase of height of > 30 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Good	Very Poor	Very Good	Very Good

Figure 23: Site evaluation – Embankment height

This account counts for 10% of the total weight. The results show that the alternative that scored highest score was Scenario 2 with a score of 80 %. Scenarios 1 and 4 scored similarly with scores of 63% and 65%, respectively. Scenarios 3 and 5 scored the lowest with 51% and 58%, respectively. Although Scenario 1 scored in the second place grouping it scored favourably with respect to project schedule and constructability compared to all other options.

Results of the account and sub-account scoring are presented in Table 7.

Table 7: Site evaluation – Technical considerations

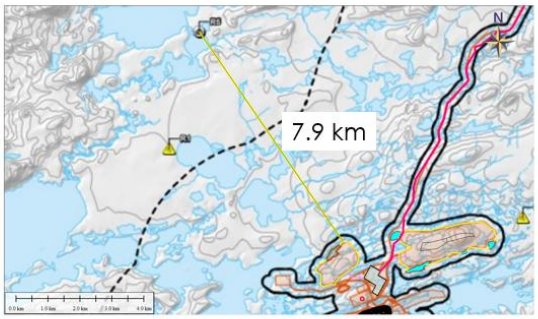
Technical Consideration													
POETS				Rating									
Decision Criteria			Weight	Scenario S1		Scenario S2		Scenario S3		Scenario S4		Scenario S5	
Principal Account		Sub Account											
			100%	Points	Rating	Points	Rating	Points	Rating	Points	Rating	Points	Rating
Technical Consideration	1	Water management infrastructure	35	Poor	14%	Very Good	35%	Very Good	35%	Good	28%	Fair	21%
	2	water management complexity	20	Very Poor	4%	Fair	12%	Very Poor	4%	Very Poor	4%	Very Poor	4%
	3	Project schedule & Constructability	15	Very Good	15%	Very Poor	3%	Poor	6%	Very Poor	3%	Very Poor	3%
	4	Embankment (earthwork) height	30	Very Good	30%	Very Good	30%	Very Poor	6%	Very Good	30%	Very Good	30%
Total			100		63%		80%		51%		65%		58%

5.1.4 Social impacts

Social impacts have been evaluated using four (4) sub-accounts, namely:

- **Proximity to sensitive receptors (cabins and travel routes):** This sub-account measures how far each scenario is from people.
- **Moose feeding areas and goose feeding areas:** This sub-account measures the impact that each scenario could have on hunting.
- **Archaeological potential**: This sub-account evaluates the risk of impacting areas identified in the field with archeological potential.
- **Fishing areas**: This sub-account highlights the importance of the fishing for the communities.

The source of the information was provided by PMET; the scoring for each scenario and criterion is provided in Figure 24 through Figure 27, respectively.



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Proximity to sensitive receptors (distance in km from Cabin or travel route)	7.9	3.3	7.9	5.7	3.7

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Fair	Good	Good	Fair

Figure 24: Site evaluation – Sensitive receptors

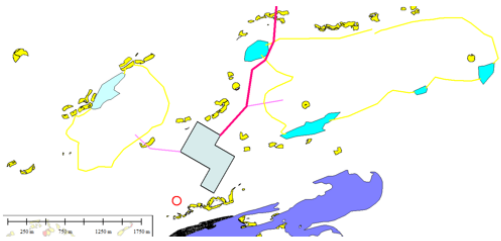


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
moose feeding areas and goose feeding areas (distance in km to goose or moose feeding areas)	2.5	3.4	2.3	0.2	7.9

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
2	**moose feeding areas and goose feeding areas**	distance in km Hunting ground	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Fair	Fair	Fair	Very Poor	Good

Figure 25: Site evaluation – Moose and geese feeding areas

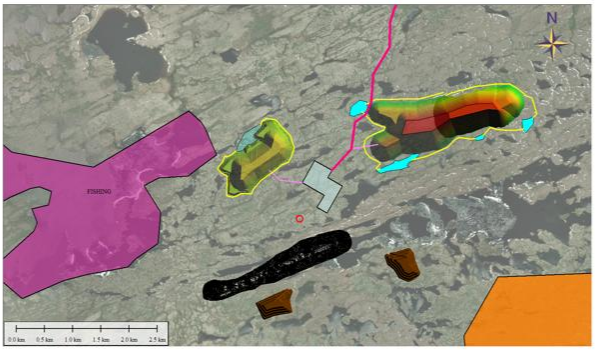


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Archaeological potentia	No high potential areas 2 medium areas inside the sites The closest high is at 0.8 km	No high or medium potential areas in the perimeter The closest high P is at 5.8 km The closest medium P is at 0.2 km	No high potential areas 1 medium area inside the sites The closest high is at 0.47 km	No high or medium potential areas The closest medium is at 0.1 km The closest high is at 1.9 km	No high potential areas 2 medium areas inside the sites The closest high is at 8.0 km

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Good	Good	Very Good	Good

Figure 26: Site evaluation – Archaeology



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
distance to fishing areas (km)	0.15	0.25	0.25	0.12	3.24

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	Fishing areas	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Poor	Poor	Poor	Very Good

Figure 27: Site evaluation – Fishing areas

This account counts for 25% of the total weight. The results show that the alternative that scored the highest was Scenario 5, with a score of 80%. All other scenarios scored in the range of 60%-65%.

Results of the account and sub-account scoring are presented in Table 8.

Table 8: Site evaluation – Social considerations

Social Impacts													
POETS				Rating									
Decision Criteria			Weight	Scenario S1		Scenario S2		Scenario S3		Scenario S4		Scenario S5	
Principal Account		Sub Account	100%	Points	Rating	Points	Rating	Points	Rating	Points	Rating	Points	Rating
Social Impacts	1	Proximity to sensitive receptors (cabins and travel routes)	25	Good	20%	Fair	15%	Good	20%	Good	20%	Fair	15%
	2	moose feeding areas and goose feeding areas	25	Fair	15%	Fair	15%	Fair	15%	Very Poor	5%	Good	20%
	3	Archaeological potential	25	Good	20%	Good	20%	Good	20%	Very Good	25%	Good	20%
	4	Fishing areas	25	Poor	10%	Poor	10%	Poor	10%	Poor	10%	Very Good	25%
Total			100		65%		60%		65%		60%		80%

6. Overall evaluation and sensitivity analysis

The results of POETS matrix assessment are presented in Figure 28. The Overall assessment indicates that Scenario 3 obtained the highest score with an overall rating of 68% and Scenario 1, with a similar rating of 61%. Scenarios 2, 4 and 5 trailed Scenario 3 by 15% to 24%, with scores of 44%, 53% and 49%, respectively. Scenario 1 achieved its overall rating with a more balanced scoring across all accounts while Scenario 3 scored higher in Project Economics and Operational accounts and lost ground on the Environmental and Technical accounts.

Option ID	Option Rating	Overall	P	O	E	T	S
S1	2	61%	62%	65%	54%	63%	65%
S2	5	44%	20%	20%	52%	80%	60%
S3	1	68%	84%	71%	56%	51%	65%
S4	3	53%	38%	29%	70%	65%	60%
S5	4	49%	27%	20%	52%	58%	80%

Figure 28: POETS – Base case – Results site selection

An additional analysis was carried out to evaluate the sensitivity of the account weightings on selection of the preferred alternative. A sensitivity analysis is a quantitative method used to assess the impact of varying input parameters or assumptions on the outcomes of the decision. It helps to understand the degree to which changes in these inputs influence the results and provides insights into the model's robustness or the sensitivity of a decision to different variables.

For this sensitivity analysis, account weightings were changed considering alternatives scenarios as presented in Figure 29 and described as follows:

- **Base Case:** Represents the PMET accounts weighing. This scenario gives half (50%) of total weight to the environment and social impact accounts.
- **Case1:** Represents a situation with social impacts having the same relevance as the cost of the project.
- **Case 2:** Represents a situation with equal weight for all the accounts. All accounts have the same importance.
- **Case 3:** Represents a situation with environmental impacts having the same relevance as the cost of the project.
- **Case 4:** Represents the scenario recommended by Environment Canada where the importance of the environment and social impact is valued more that the other accounts.

POETS - Sensitivity Analysis – Set-Up

Case	Description	Account Weighting				
		P	O	E	T	S
Selected Case	Base Case	30	10	25	10	25
Case 1	Scenario looking to give equal importance to project economics and social	30	10	20	10	30
Case 2	All accounts weighted equally	20	20	20	20	20
Case 3	Scenario looking to give equal importance to project economics and environment	30	10	30	20	10
Case 4	Account weighted with E+S = 66.7% and P/O/T = 11.1% each (33.3%)	11.1	11.1	44.5	11.1	22.2

Figure 29: Weight of accounts per case

In Figure 30, the column "Case" presents the different scenarios evaluated. For each scenario, weights are given to each account (Project economics, Operational, Environment, Technical and Social). It is noted that Scenario 3 achieved the highest score regardless of the sensitivity case being considered, and Scenario 1 ranked second in all cases. Although Scenario 3 achieved the highest score under all sensitivity cases, the difference between Scenarios 3 and 1 ranged from 7% in the Base Case to 3% in Sensitivity Case 2, where all accounts are weighted equally. The difference between these options is also reduced if either Social or Environmental is weighted equally to Project Economics. Lastly, if the account weighting is set to the weighting recommended by Environment Canada the difference between Scenario 1 and Scenario 3 is a mere 2%.

Option ID	Option Rating	Overall	P	O	E	T	S	Case
S1	2	61%	62%	65%	54%	63%	65%	**Selected: Base Case**
S2	4	44%	20%	20%	52%	80%	60%	
S3	1	68%	84%	71%	56%	51%	65%	
S4	3	53%	38%	29%	70%	65%	60%	
S5	4	49%	27%	20%	52%	58%	80%	
S1	2	62%	62%	65%	54%	63%	65%	**Case 1**
S2	4	44%	20%	20%	52%	80%	60%	
S3	1	68%	84%	71%	56%	51%	65%	
S4	3	53%	38%	29%	70%	65%	60%	
S5	4	50%	27%	20%	52%	58%	80%	
S1	2	62%	62%	65%	54%	63%	65%	**Case 2**
S2	4	46%	20%	20%	52%	80%	60%	
S3	1	65%	84%	71%	56%	51%	65%	
S4	3	52%	38%	29%	70%	65%	60%	
S5	4	47%	27%	20%	52%	58%	80%	
S1	2	60%	62%	65%	54%	63%	65%	**Case 3**
S2	4	46%	20%	20%	52%	80%	60%	
S3	1	66%	84%	71%	56%	51%	65%	
S4	3	54%	38%	29%	70%	65%	60%	
S5	5	45%	27%	20%	52%	58%	80%	
S1	2	60%	62%	65%	54%	63%	65%	**Case 4**
S2	4	50%	20%	20%	52%	80%	60%	
S3	1	62%	84%	71%	56%	51%	65%	
S4	3	59%	38%	29%	70%	65%	60%	
S5	4	53%	27%	20%	52%	58%	80%	

Figure 30: POETS – Sensitivity analysis – Results

6.1 Workshop observations

The workshops took place on March 26th, April 10th and April 16th,2025 and were registered in the meeting minutes on presentation number 7535027-00000-41-APP-0003 / R03, included in Appendix A. This work session allows the different interested parties to evaluate and comment on the preliminary multicriteria analysis assessment performed in advance by BBA. The result of the evaluation was discussed, the most relevant observations that were made in the workshop and that impose changes in the sub-account and account assessments are presented as follows:

- The weighting of sub-accounts has changed as a suggested by PMET. This was undertaken in a series of preliminary meetings. It is BBA's opinion that any changes made were consistent and appropriate.
- PMET suggested that an additional criterion be added as a result of previously unavailable geochemical data; more specifically, it has been suggested that due to increased contact time, for piles greater that 90 m in height, the potential for off-site contamination increase. This criterion has been considered in the overall evaluation.
- It was also suggested and deemed appropriate that Closure Costs be replaced with Fish Habitat Compensation cost in the Project Economics account.
- It was also suggested on several occasions to increase the weight (importance) of the criterion of fish presence in the Environmental account, to ensure the commitment of the company to reduce the impacts in the environment; in that regard the last conclusion demonstrated the result of that case.

6.2 Conclusions and recommendations

A rigorous site selection process has been undertaken by PMET based on the available site constraints maps at the time of the analysis. As more detailed information becomes available the analysis can be refined. However, given the results from the current site selection process, it is highly unlikely that any new information will substantively change the conclusion derived from this analysis.

Regardless of the recommendations put forth in this report, the path forward necessarily needs to consider corporate policies and directives as part of a final decision; particularly where environmental stewardship is concerned.

The overall assessment indicates that Scenario 3 obtained the highest score with an overall rating of 68%, and Scenario 1 with a similar rating of 61%. Scenarios 2, 4 and 5 trailed Scenario 3 by 15% to 24%, with scores of 44%, 53% and 49%, respectively. Scenario 1 achieved its overall rating with a more balanced scoring across all accounts while Scenario 3 scored higher in Project Economics and Operational accounts and lost ground on the Environmental and Technical accounts.

An over-arching conclusion that can be drawn from this process is that Scenarios 1 and 3 provided similar scoring while Scenarios 2, 4 and 5 are not competitive alternatives, despite some of these options scoring well on specific criterion (sub-accounts). Should other holistic options need to be considered in the future it is recommended that they be evaluated against Scenarios 1 and 3 alone.

It is noted that Scenario 3 achieved the highest score regardless of the sensitivity case being considered, and Scenario 1 was consistently in second ranking. Although Scenario 3 achieved the highest score under all sensitivity cases, the difference between Scenarios 3 and 1 ranged from 7% in the Base Case to 3% in Sensitivity Case 2, where all accounts are weighted equally. The difference between these options is also reduced if either Social or Environmental is weighted equally to Project Economics. Lastly, if the account weighting is set to the weighting recommended by Environment Canada the difference between Scenario 1 and Scenario 3 is a mere 2%.

The results of the sensitivity analysis indicated that Scenarios 1 and 3 gave essentially the same result considering the nature and precision of the exercise, quality of the constraints data and the level of conceptual design used. It is recommended that Scenarios 1 and 3 be considered in the final decision, as to waste pile location, ensuring alignment with corporate environmental stewardship objectives.

It was also observed that Scenario 3 reliance on economic criteria were paramount in its ability to rank number 1 during this exercise. Although a parametric sensitivity analysis at the sub-account level was not undertaken, it was noted that should a higher weight be given to fish habitats in the environmental account, Scenario 1 would outperform the others in light of PMET's commitment to environmental sustainability; this should be considered in the final decision.

It is suggested to continue with the baseline studies long-term monitoring, community engagement, and ways to integrate sustainable design from the outset, the way it was discussed in the workshops.

If the volume of the materials changes significatively, this may require the need for a new evaluation of the site selection.

Appendix A: Workshop Presentation

Shaakichiuwaanaan Project

POETS

WORKSHOP 1 – 2025-03-26
WORKSHOP 2 - 2025-04-10
WORKSHOP 3 – 2025-04-16

Site selection alternatives for storage of mine waste

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Patriot Battery Metals

Site selection alternatives for mine waste rock

James Bay , Qc

Shaakichiuwaanaan (Corvette) Project

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BBA Document No. / Rev. 7535027-00000-41-APP-0003 / R03
May 01, 2025

For Comment
Not to be used for construction

REVISION HISTORY

Revision	Document Status – Revision Description	Date
RAA	Preliminary	2025-03-13
R00	Draft - For Comment	2025-03-27
R01	Second presentation	2025-04-10
R02	Final Draft – Workshop 3	2025-04-23
R03	Final Draft – Updated the archaeology	2025-05-01

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Reference List

Subaccount or subject	Plans or document	notes
Wetlands/ Rivers / Streams	CORVETTE_BIO_carac_milieu_s_011	Has information for all the scenarios but the level of detail only is high for scenarios 1 and 3
	CORVETTE_BIO_hydrog_l_006	Has information for all the scenarios but the level of detail only is high for scenarios 1 and 3
	Fish Habitat_20241023	Only has information on the areas for scenarios 1 and 3
Proximity to sensitive receptors	160846E.pdf	Public information
moose feeding areas and goose feeding areas	**confidential**	**confidential**
Archaeological potential	Aire d'étude et d'inventaire - archaeo	Only has information on the areas for scenarios 1 and 3
Mine description and criteria	CASN-A-370-GT-DCR-0001-B_Waste and water management Design criteria.pdf	

Mine description

5.2.4 Projected Mine Life	23 years
5.2.6 Type of Mine	Open pit and underground
5.2.10 Mine Waste Production	Waste Rock: 186.9 Mt Overburden: 16 Mt Tailings: 73.5 Mt Backfill: TBD
5.2.11 Backfill	32.5 Mt of tailings will be used for underground backfill
	80 Mt of waste rock (Acid Generating)
5.3.4 Precipitation	Average annual precipitation is 772 mm which includes 518 mm of rainfall and 254 cm of snowfall
5.3.5 Evapotranspiration	Average annual lake evaporation is 421 mm and maximum evapotranspiration under average conditions is 390 mm
5.4.4 Filtered tailings moisture content	12-14% (M_{water}/M_{total})
5.4.10 Tailings Deposition	Filtered tailings are transported with mine trucks from the concentrator to the final disposal area or the paste backfill plant with a front-end loader.
5.5.2.6 Average in-situ dry density	Waste rock: 2 t/m^3 Tailings: 1.7 t/m^3
Overall slope: 3 (Horizontal): 1 (Vertical) for all types (i.e. tailings, and Waste rock)	
5.5.2.3 Base Lining (Only in Acid generating rock and tailings portion)	Single liner: 1.5 mm HDPE Liner Cushion Layer: 1m Overdrainage System: 1m

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DESIGN CRITERIA

M2 : HYBRID UG/OP MINING METHOD

Material		Capacity*	Dry Density	Estimated Volume
		Mt	t/m^3	Mm^3
Tailings to put on surface	*MEMBRANE*	*41*	*1.7*	**24.1**
Waste Rock (Low-Risk)	***NO** MEMBRANE*	*107*	*2.0*	**53.5**
Waste Rock (Potentially Acid generating)	*MEMBRANE*	*80*	*2.0*	**40**
TOTAL				***117.6***

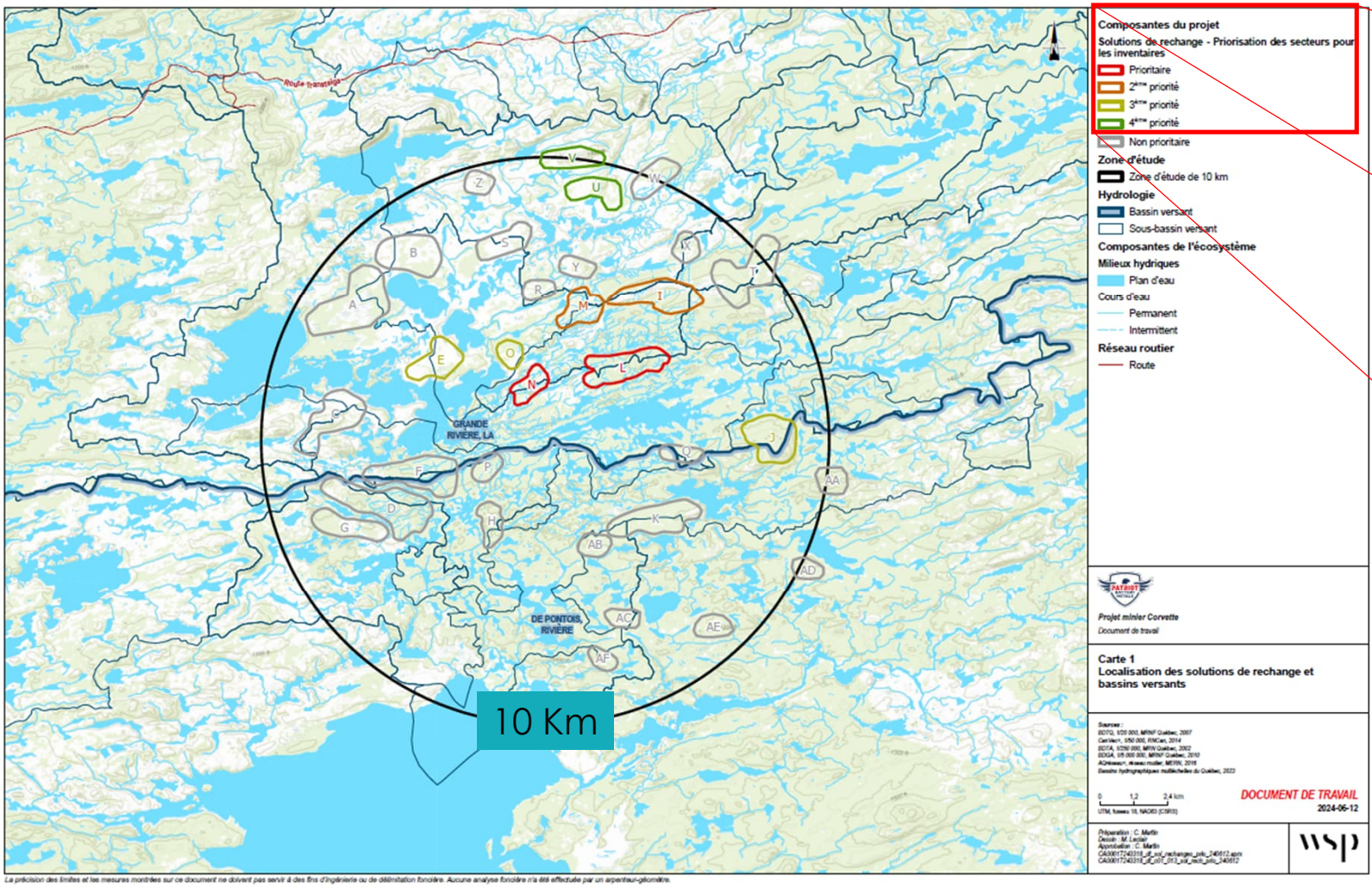
- *Estimated tonnages based on dry tailing technology*

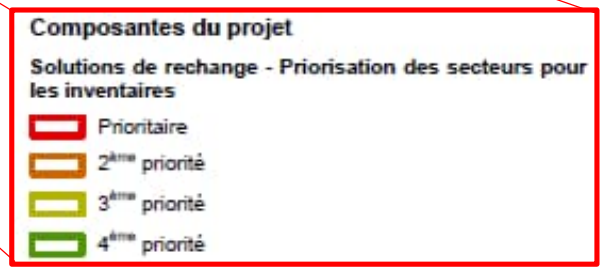
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PRE-SCREENING

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PRE-SCREENING





31 potential sites evaluated

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PRE-SCREENING PROCESS

Took place Feb-Apr 2024 to identify the potential waste disposal options for the Shaakichiuwaanaan Lithium project:

- Zones of priority were defined for site investigation (May- Sept 2024)
- PMET, WSP and BBA contributed to the pre-selection process
- Dry stack tailings technology was considered as the basis for disposal of tailings
- Limited to 10km radius from mill
- Material waste balances form PEA study used a s a basis of estimation (not considering backfilling waste into the pit)
- Minimum size for waste option(pile) based on DMS tails stream (smallest stream)
- Considered slope angle of 18.4° (3:1 Slope H:V) for design volumes of all options
- Height of each option considered a minimum 50m by 50m area at crest
- Tailings density of 1.7t/m3 (compacted in pile)
- Considered minimum offset of 100m from known lakes, rivers (Directive 019 requires a minimum 60m offset)
- Claims boundaries not considered as limiting factor for pre-selection process

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PRIORITY SITES SELECTED FOR SITE INVESTIGATION

Screening of options based following considerations:

- Waste deposition in multiple water sheds (basin versant)
  - Limited waste disposal options to northern water shed to eliminate risk of potential contamination of multiple watersheds
- Potential for future mineralization
  - Options along strike of indicated mineralization zone eliminated (CV 9, CV13…)
- Environment and social impacts
  - Eliminated options in proximity to known fish habitats, communities and archeological sites based on WSP constraint maps provided March 14th 2024
- Options selected in priority order for investigation based on comparison of distance from pit, number of river crossings and presents of favourable topography
- Priority 4 option retained due to proximity forecasted site access road

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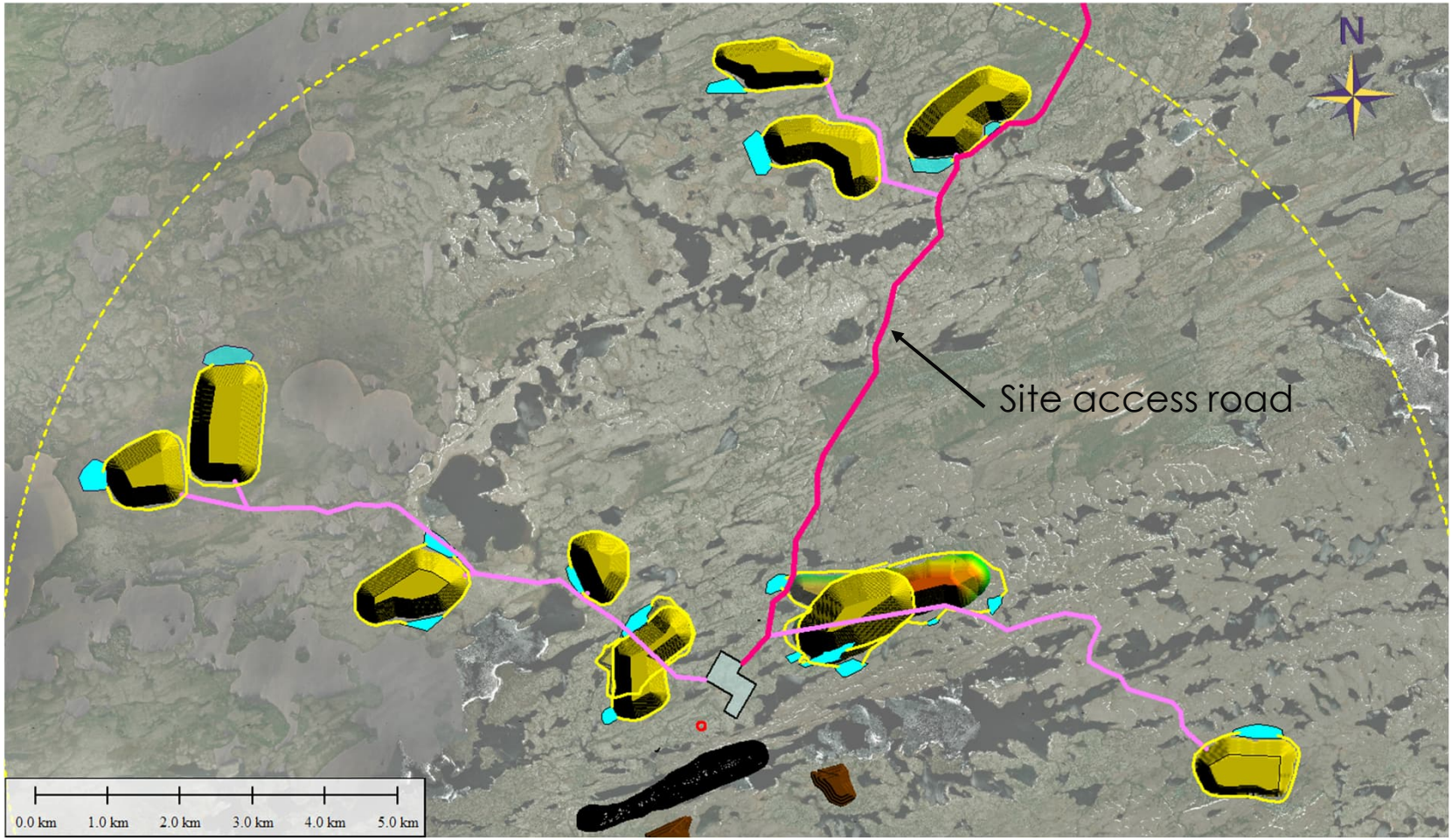
Project Scenarios – Scope definition

Cases	Location options	Hybrid Solution
S1	Base Case	2024 PEA Sites L,N
S2	Avoid fish habitat and wetlands	Sites A
S3	Possibility to reduce project cost related to stockpiles	Extended Sites L and N
S4	Priority 3 (selected by the client)	Sites **E,O,J**
S5	Priority 4 (selected by the client)	Sites **U,V,**W

E,O,J as defined by priority 3 (see slide 7)
U,V as defined by priority 4 (see slide 7)

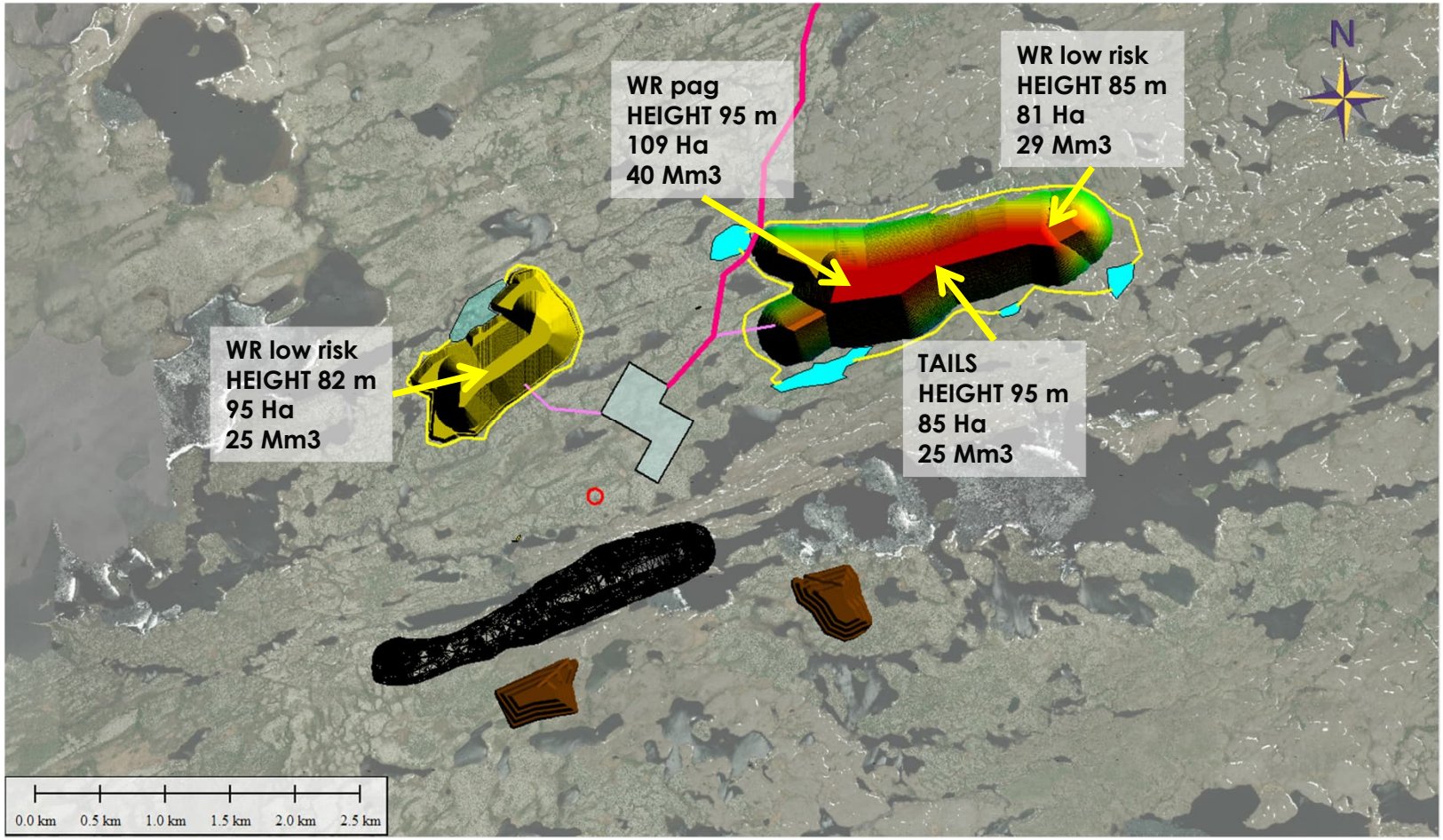
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Sites chosen



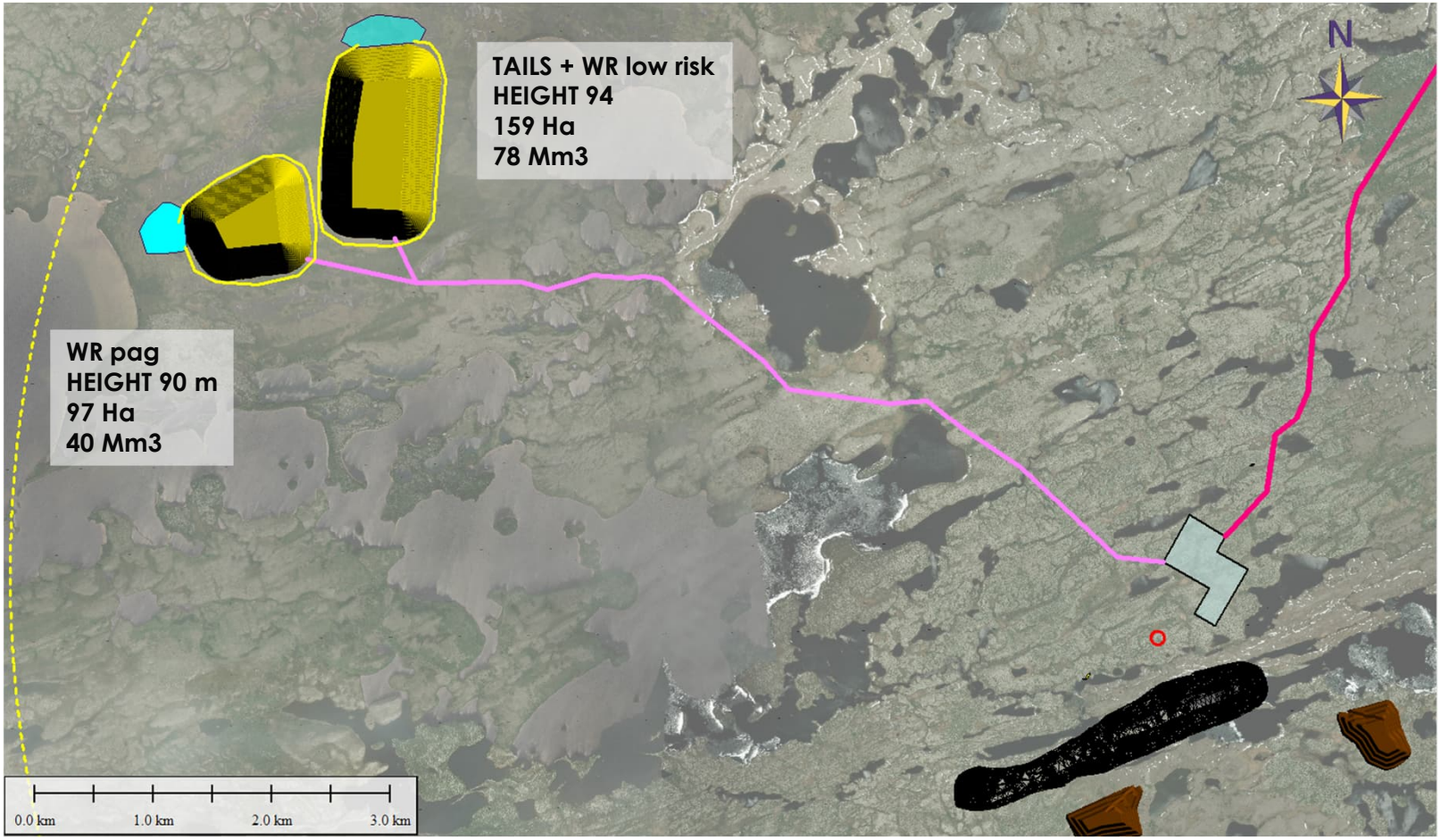
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SCENARIO 1 (BASE CASE - SITES N,L)



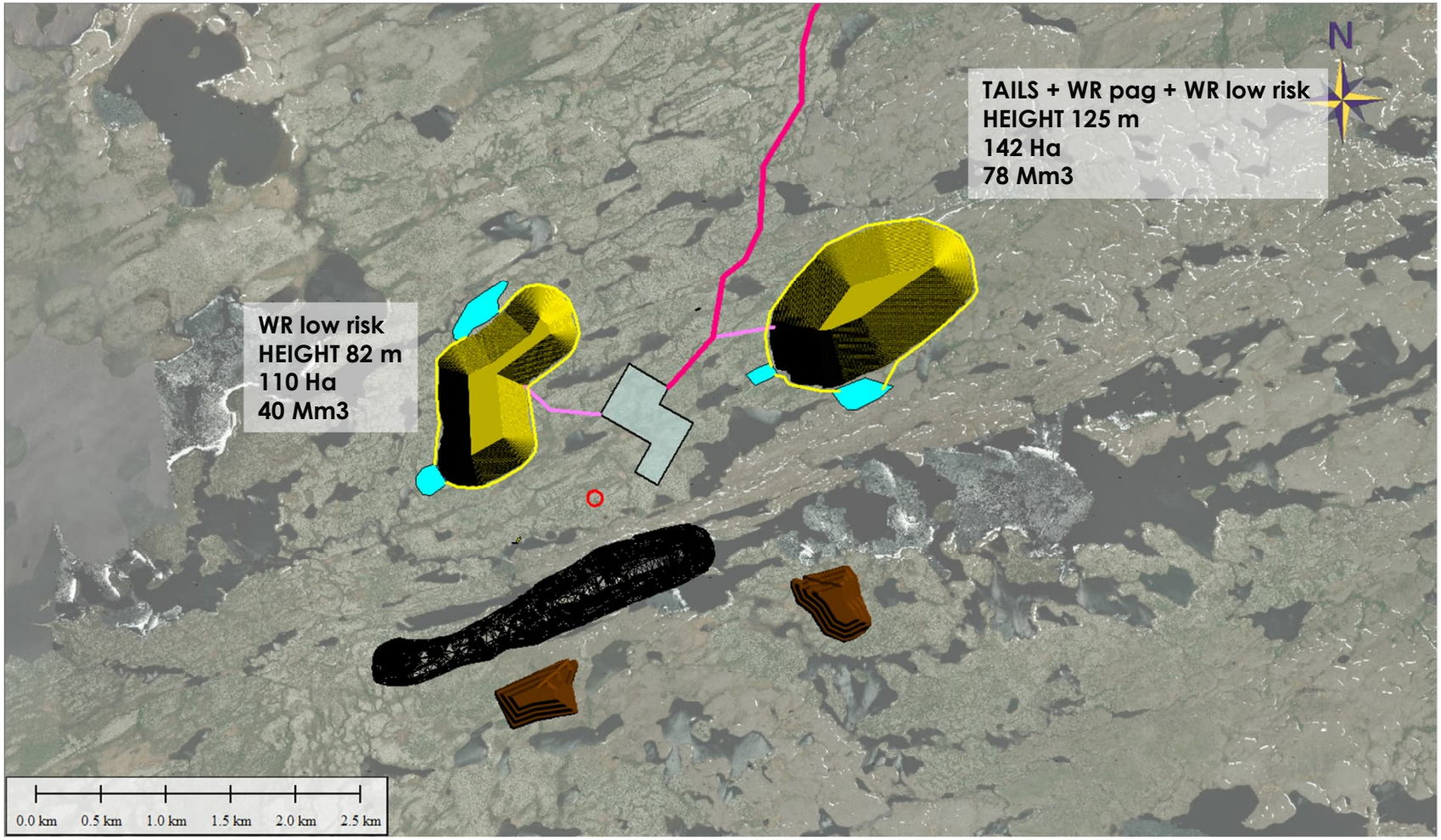
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SCENARIO 2 (AVOID FISH HABITATS AND WETLANDS – SITE A)



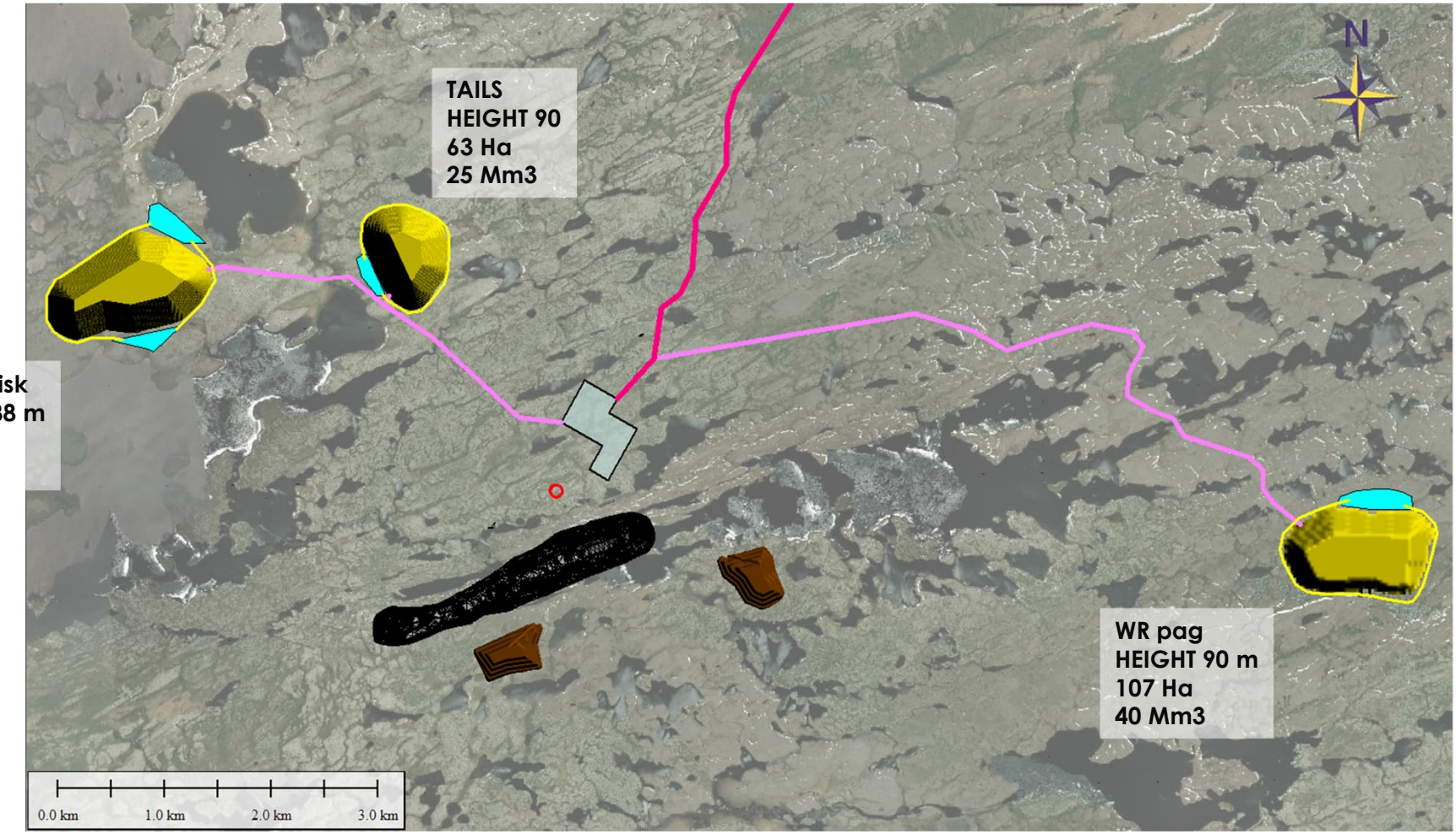
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SCENARIO 3 (REDUCE PROJECT COST RELATED TO STOCKPILES - SITES N,L EXTENDED)



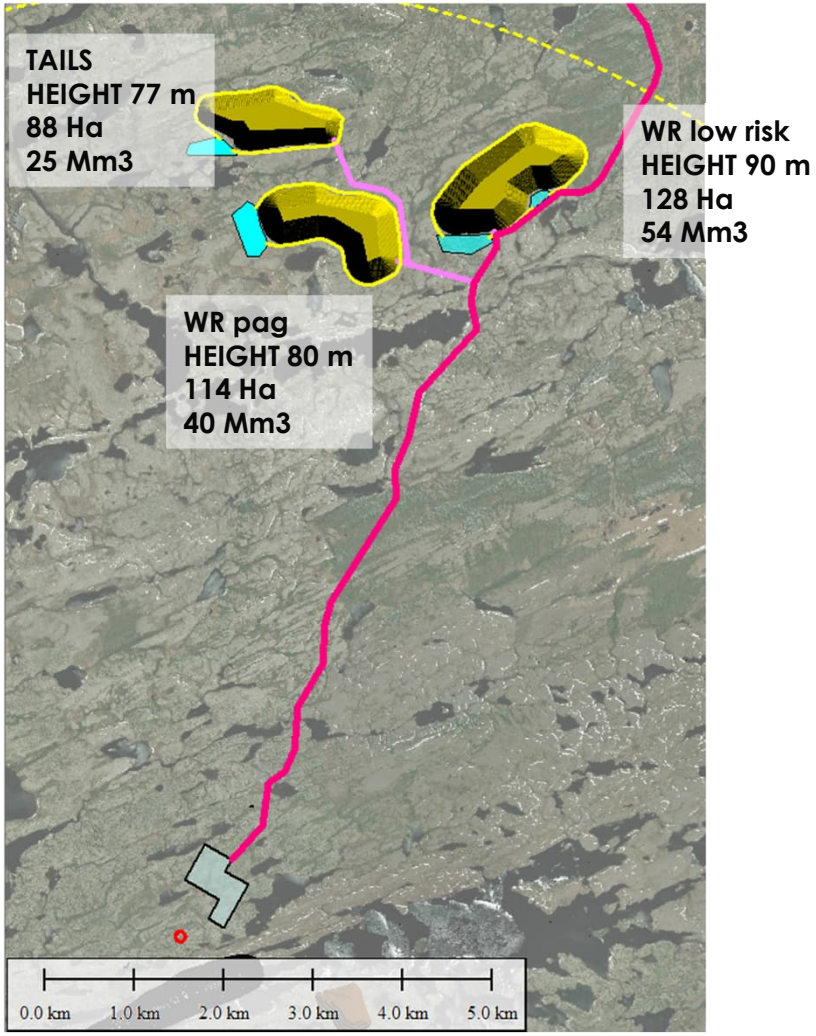
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SCENARIO 4 (PRIORITY 3 – SITES E,O,J)



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SCENARIO 5 (PRIORITY 4 – SITES U,V,W)



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SELECTION CRITERIA

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POETS CRITERIA

Total Weight %		Weight in total %	Account	Weight in account %	Criteria Description	Individual criteria %		Total Weight %
100%	→	30%	Project Economics	50%	CAPEX	15.0%	→	100%
				25%	OPEX	7.5%	→	
				20%	Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	6.0%	→	
				5%	Fish habitat compensation	1.5%	→	
	→	10%	Operational Consideration	47%	Distance from mill/pit	4.7%	→	
				32%	Expansion capability	3.2%	→	
				21%	Operational safety	2.1%	→	
	→	25%	Environmental Impacts	30%	STOCKPILE footprint	7.5%	→	
				20%	CO2 emissions	5.0%	→	
				30%	Wetlands/ Rivers / Streams	7.5%	→	
				20%	Fish presence	5.0%	→	
	→	10%	Technical Consideration	35%	Water management infrastructure	3.5%	→	
				20%	water management complexity	2.0%	→	
				15%	Project schedule & Constructability	1.5%	→	
				30%	Embankment (earthwork) height	3.0%	→	
	→	25%	Social Impacts	25%	Proximity to sensitive receptors (cabins and travel routes)	6.3%	→	
				25%	moose feeding areas and goose feeding areas	6.3%	→	
				25%	Archaeological potential	6.3%	→	
				25%	Fishing areas	6.3%	→	

Score card

TABLE: TSF ACCOUNTS, SUBACCOUNTS AND SCORING CRITERIA

ACCOUNT		SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
PROJECT ECONOMICS	1	**CAPEX**	cost	Alternative less costly	Increase of cost of 15%	Increase of cost of 30%	Increase of cost of 45%	Increase > 45%
	2	**OPEX**	cost	Alternative less costly	Increase of cost of 15%	Increase of cost of 30%	Increase of cost of 45%	Increase > 45%
	3	**Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)**	Qualitative	The works can be spread in at least 3 phases with more than 30% of capex beyond year 0	The works can be spread in at least 3 phases with less than 30% of capex beyond year 1	The works can be spread in at least 2 phases with more than 30% of capex beyond year 0	The works can be spread in at least 2 phases with less than 30% of capex beyond year 1	All the works have to be done in one phase in year 0
	4	**Fish habitat compensation**	Qualitative	No significant risks identified, with all potential costs accounted for or within the applied contingency. Recommended contingency 15%	Limited risks identified which would minimally require an increase of the project estimation contingency. Most costs are well-documented but may need slightly greater contingency. Recommended contingency 25%	Several risks identified which would marginally require an increase of the project estimation contingency. Costs are well-documented but may need greater contingency. Recommended contingency 35%	Several risks identified which would significantly require an increase of the project estimation contingency. Some costs are well-documented but need greater contingency. Recommended contingency 50%	Numerous risks have been identified such that contingencies need to be increased to level beyond the estimation precision. Recommended contingency more than 50%
OPERATIONS	1	**Distance from mill/pit**	distance in km	0-2 km	2-4 km	4-6 km	6-8 km	> 8 km
	2	**Expansion capability**	percentage of expansion	40% or more additional volume can be deposited in the site option.	30 - 40% additional volume can be deposited in the site option.	20 - 30% additional volume can be deposited in the site option.	10 - 20% additional volume can be deposited in the site option.	less than 10% additional volume can be deposited in the site option.
	3	**Operational safety**	distance of roads in km	Minimal Hazard Exposure due to the very short distances (0-2 km) between the sites and the pit or the mill, these distances do not have measurable effect on operator well-being or safety.	Low Hazard Exposure due to the short distances (2-4 km) between the sites and the pit or the mill, these distances have minimal effect on operator fatigue or performance.	Moderate Hazard Exposure due to the intermediate distance (4-6 km) between the sites and the pit or the mill, fatigue and reduced performance are common due to travel distances but manageable with breaks.	High Hazard Exposure due to the long distances (6-8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.	Extreme Hazard Exposure due to the very long distances (>8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.
ENVIRONMENTAL	1	**STOCKPILE footprint**	Footprint surface	Alternative with less footprint surface	Increase of footprint of 15%	Increase of footprint of 30%	Increase of footprint of 45%	Increase of footprint of 100%
	2	**CO2 emissions**	Distance of roads in kilometers	The project generates minimal CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (0-1Km)	The project generates below-average CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (1-2Km)	The project generates a moderate amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (2-3Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (3-4Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (> 4Km)
	3	**Wetlands/ Rivers / Streams**	Surface of wetlands and water streams	less than 10 Ha	10 To 20 Ha	20 to 30 Ha	30 to 40 Ha	More than 40 Ha
	4	**Fish presence**	Presence of fish	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat
TECHNICAL	1	**Water management infrastructure**	watershed surface	Alternative with less watershed surface to manage	Increase of watershed of 15%	Increase of watershed of 30%	Increase of watershed of 45%	Increase of watershed of 100% or more
	2	**water management complexity**	Water surface	Site water management can be done in one water management system and water drainage done by gravity	Site water management can be done in one water management systems requiring on or more sump with ongoing pumping	Site water management can be done in two water management systems and water drainage done by gravity	Site water management can be done in two water management systems requiring on or more sump with ongoing pumping	Site water management would require more than 2 water management system
	3	**Project schedule & Constructability**	distance, % of wetlands or water streams and area of basins	Site that is closest to the central camp, where the work area impacts the least area of wetlands or water surfaces and has the least surface of basins, and the topography is not hilly. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is within 1km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 20% of the best alternative related to wetlands and basin surface. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is within 3km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 40% of the best alternative related to wetlands and basin surface. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is within 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces within 50% of the best alternative related to wetlands and basin surface. Reduce the scoring by one category if the option creates a permitting delay risk	Site that is more 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces grater 50% of the best alternative related to wetlands and basin surface.
	4	**Embankment (earthwork) height**	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 degree if height is equal or less than 90m	Increase of height of 20 m Increase the score by 1 degree if height is equal or less than 90m	Increase of height of 30 m Increase the score by 1 degree if height is equal or less than 90m	Increase of height of > 30 m Increase the score by 1 degree if height is equal or less than 90m
SOCIAL	1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km
	2	**moose feeding areas and goose feeding areas**	distance in km Hunting ground	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km
	3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area
	4	**Fishing areas**	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

Project Economics (30%)

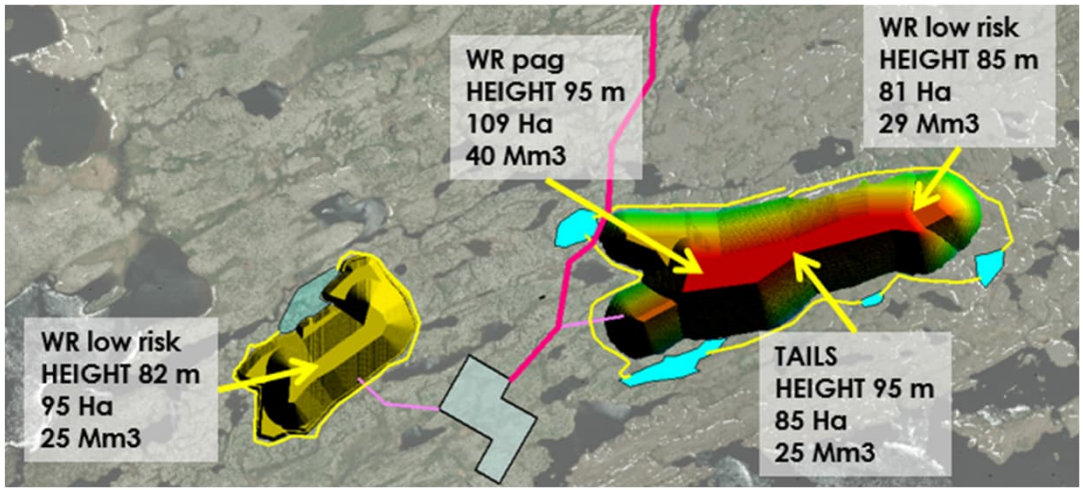
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CAPEX (scenario 1)

			QUANTITY	UNIT PRICE	COST
MEMBRANE	TAILS 24.1Mm3	SURFACE OF TAILINGS PILE (Ha)	98	$ 505,000	$ 49,490,000
		LENGTH OF DITCHES (m)		$ 1,685	$ -
		SURFACE OF BASIN (Ha)		$ 2,060,000	$ -
		VOLUME OF THE BASIN (m^3)	105,900	$ -	$ -
		ROADS (m)	0	$ 818	$ -
		BRIDGES FOR ROADS (unit)	0	$ 2,000,000	$ -
NO MEMBRANE	WR (LOW RISK) 25Mm3	SURFACE OF WASTE ROCK PILE (LOW RISK) (Ha)	95	$ 15,000	$ 1,425,000
		LENGTH OF DITCHES (m)	3700	$ 1,215	$ 4,495,500
		SURFACE OF BASIN (Ha)	8.3	$ 2,060,000	$ 17,098,000
		VOLUME OF THE BASIN (m^3)	111,600	$ -	$ -
		ROADS (m)	700	$ 818	$ 572,833
		BRIDGES FOR ROADS (unit)	1	$ 2,000,000	$ 2,000,000
NO MEMBRANE	WR (LOW RISK) 29Mm3	SURFACE OF WASTE ROCK PILE (LOW RISK) (Ha)	81	$ 15,000	$ 1,215,000
		LENGTH OF DITCHES (m)	3,100	$ 1,215	$ 3,766,500
		SURFACE OF BASIN (Ha)	5	$ 2,060,000	$ 10,094,000
		VOLUME OF THE BASIN (m^3)	92,800	$ -	$ -
		ROADS (m)	0	$ 818	$ -
		BRIDGES FOR ROADS (unit)	0	$ 2,000,000	$ -
MEMBRANE	WR (PAG) 40Mm3	SURFACE OF WASTE ROCK PILE (PAG) (Ha)	109	$ 505,000	$ 55,045,000
		LENGTH OF DITCHES (m)	4200	$ 1,685	$ 7,074,900
		SURFACE OF BASIN (Ha)	15.4	$ 2,060,000	$ 31,724,000
		VOLUME OF THE BASIN (m^3)	134,400	$ -	$ -
		ROADS (m)	500	$ 818	$ 409,167
		BRIDGES FOR ROADS (unit)	0	$ 2,000,000	$ -
					$ 184,409,900

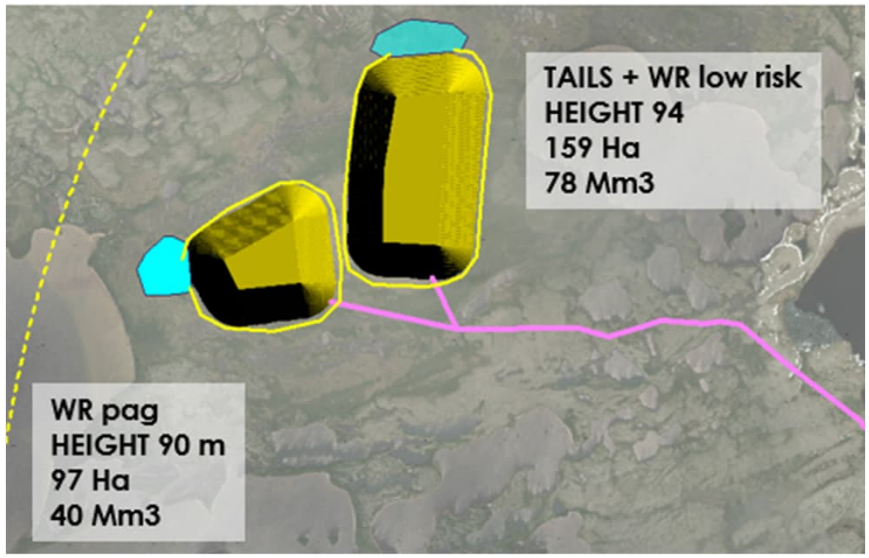
13 Ha are added here to separate the Tails from the WR low risk

Ditches and basin was considered in the WR pag pile, not here



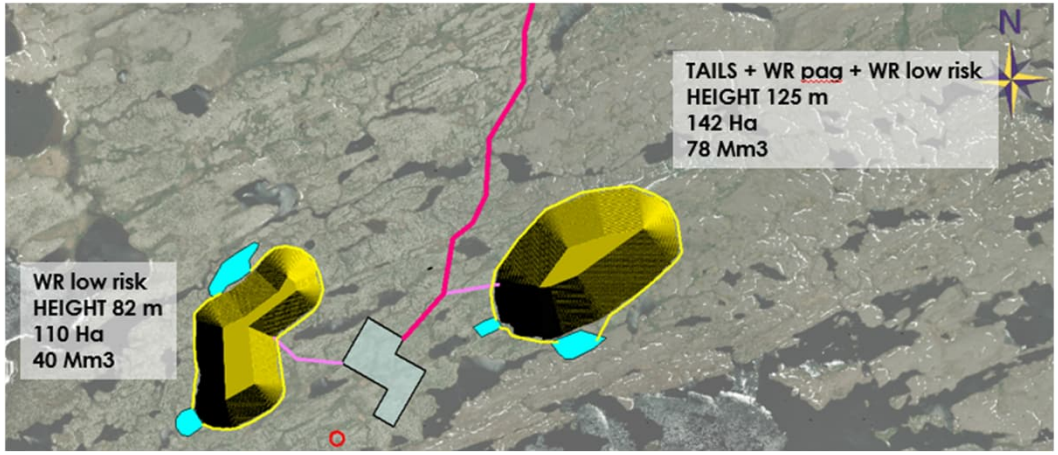
CAPEX (scenario 2)

			QUANTITY	UNIT PRICE	COST
MEMBRANE	TAILS + WR (LOW RISK) 77.6Mm3	SURFACE OF TAILINGS PILE + WASTE ROCK PILE (LOW RISK) (Ha)	158.5	$ 505,000	$ 80,042,500
		LENGTH OF DITCHES (m)	4400	$ 1,685	$ 7,411,800
		SURFACE OF BASIN (Ha)	14.5	$ 2,060,000	$ 29,870,000
		VOLUME OF THE BASIN (m^3)	186,900	$ -	$ -
		ROADS (m)	7,600	$ 818	$ 6,219,333
		BRIDGES FOR ROADS (unit)	3	$ 2,000,000	$ 6,000,000
NO MEMBRANE		SURFACE OF WASTE ROCK PILE (LOW RISK) (Ha)		$ 15,000	$ -
		LENGTH OF DITCHES (m)		$ 1,215	$ -
		SURFACE OF BASIN (Ha)		$ 2,060,000	$ -
		VOLUME OF THE BASIN (m^3)		$ -	$ -
		ROADS (m)		$ 818	$ -
		BRIDGES FOR ROADS (unit)		$ 2,000,000	$ -
MEMBRANE	WR (PAG) 40Mm3	SURFACE OF WASTE ROCK PILE (PAG) (Ha)	97.1	$ 505,000	$ 49,035,500
		LENGTH OF DITCHES (m)	3400	$ 1,685	$ 5,727,300
		SURFACE OF BASIN (Ha)	13.5	$ 2,060,000	$ 27,810,000
		VOLUME OF THE BASIN (m^3)	119,500	$ -	$ -
		ROADS (m)	940	$ 818	$ 769,233
		BRIDGES FOR ROADS (unit)	0	$ 2,000,000	$ -
					$ 212,885,667



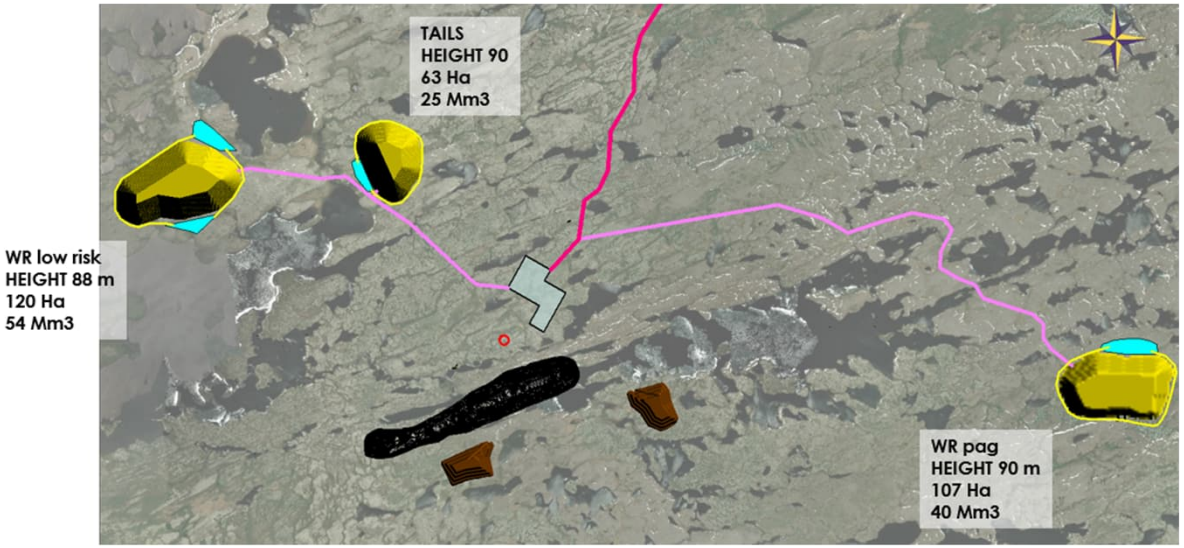
CAPEX (scenario 3)

			QUANTITY	UNIT PRICE	COST
MEMBRANE	TAILS + WR (LOW RISK) + (low risk) 77.6Mm3	SURFACE OF TAILINGS PILE + WASTE ROCK PILE (LOW RISK) (Ha)	142	$ 505,000	$ 71,710,000
		LENGTH OF DITCHES (m)	4200	$ 1,685	$ 7,074,900
		SURFACE OF BASIN (Ha)	9	$ 2,060,000	$ 18,540,000
		VOLUME OF THE BASIN (m^3)	163,100	$ -	$ -
		ROADS (m)	700	$ 818	$ 572,833
		BRIDGES FOR ROADS (unit)	1	$ 2,000,000	$ 2,000,000
NO MEMBRANE	WR (low risk) 40Mm3	SURFACE OF WASTE ROCK PILE (LOW RISK) (Ha)	110	$ 15,000	$ 1,650,000
		LENGTH OF DITCHES (m)	3900	$ 1,215	$ 4,738,500
		SURFACE OF BASIN (Ha)	12.1	$ 2,060,000	$ 24,926,000
		VOLUME OF THE BASIN (m^3)	131,900	$ -	$ -
		ROADS (m)	500	$ 818	$ 409,167
		BRIDGES FOR ROADS (unit)	0	$ 2,000,000	$ -
MEMBRANE				$ 505,000	$ -
				$ 1,685	$ -
				$ 2,060,000	$ -
				$ -	$ -
				$ 818	$ -
				$ 2,000,000	$ -
					$ 131,621,400



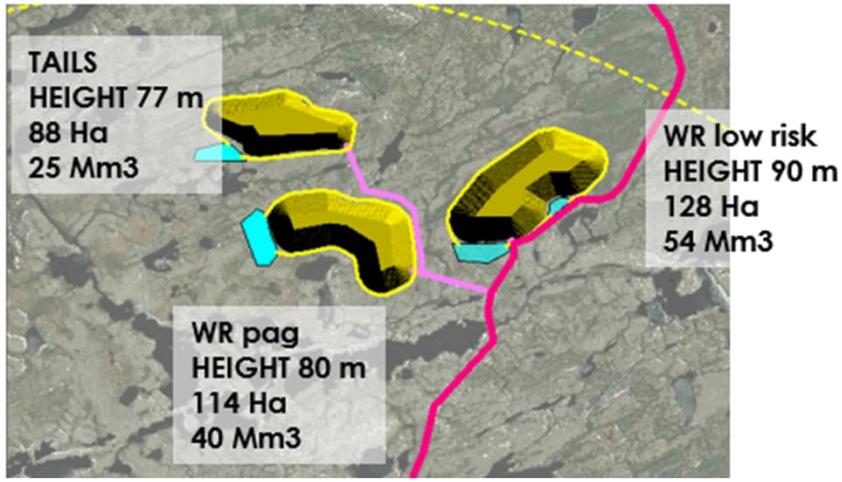
CAPEX (scenario 4)

			QUANTITY	UNIT PRICE	COST
MEMBRANE	TAILS 24.1Mm3	SURFACE OF TAILINGS PILE (Ha)	63.3	$ 505,000	$ 31,966,500
		LENGTH OF DITCHES (m)	2560	$ 1,685	$ 4,312,320
		SURFACE OF BASIN (Ha)	4.7	$ 2,060,000	$ 9,682,000
		VOLUME OF THE BASIN (m^3)	73,500	$ -	$ -
		ROADS (m)	2,200	$ 818	$ 1,800,333
		BRIDGES FOR ROADS (unit)	2	$ 2,000,000	$ 4,000,000
NO MEMBRANE	WR (LOW RISK) 53.5Mm3	SURFACE OF WASTE ROCK PILE (LOW RISK) (Ha)	119.6	$ 15,000	$ 1,794,000
		LENGTH OF DITCHES (m)	3300	$ 1,215	$ 4,009,500
		SURFACE OF BASIN (Ha)	15.6	$ 2,060,000	$ 32,136,000
		VOLUME OF THE BASIN (m^3)	146,100	$ -	$ -
		ROADS (m)	1,800	$ 818	$ 1,473,000
		BRIDGES FOR ROADS (unit)	1	$ 2,000,000	$ 2,000,000
MEMBRANE	WR (PAG) 40Mm3	SURFACE OF WASTE ROCK PILE (PAG) (Ha)	106.8	$ 505,000	$ 53,934,000
		LENGTH OF DITCHES (m)	3500	$ 1,685	$ 5,895,750
		SURFACE OF BASIN (Ha)	10.9	$ 2,060,000	$ 22,454,000
		VOLUME OF THE BASIN (m^3)	127,200	$ -	$ -
		ROADS (m)	8,000	$ 818	$ 6,546,667
		BRIDGES FOR ROADS (unit)	3	$ 2,000,000	$ 6,000,000
					$ 188,004,070

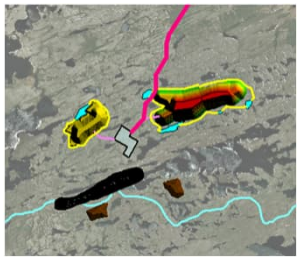


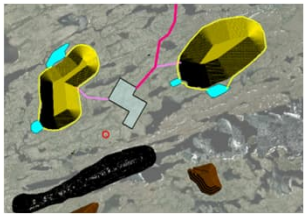
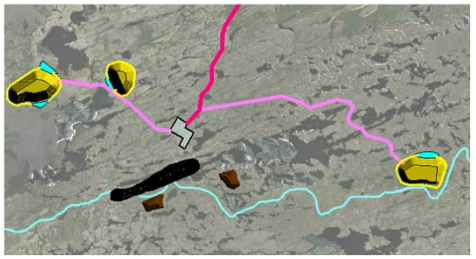
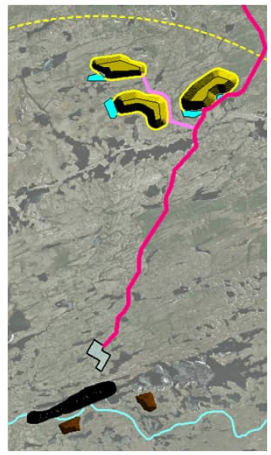
CAPEX (scenario 5)

			QUANTITY	UNIT PRICE	COST
MEMBRANE	TAILS 24.1Mm3	SURFACE OF TAILINGS PILE (Ha)	88.2	$ 505,000	$ 44,541,000
		LENGTH OF DITCHES (m)	3800	$ 1,685	$ 6,401,100
		SURFACE OF BASIN (Ha)	8.8	$2,060,000	$ 18,128,000
		VOLUME OF THE BASIN (m^3)	104,800	$ -	$ -
		ROADS (m)	1,800	$ 818	$ 1,473,000
		BRIDGES FOR ROADS (unit)	1	$2,000,000	$ 2,000,000
NO MEMBRANE	WR (LOW RISK) 53.5Mm3	SURFACE OF WASTE ROCK PILE (LOW RISK) (Ha)	128.3	$ 15,000	$ 1,924,500
		LENGTH OF DITCHES (m)	3840	$ 1,215	$ 4,665,600
		SURFACE OF BASIN (Ha)	13.2	$2,060,000	$ 27,192,000
		VOLUME OF THE BASIN (m^3)	152,900	$ -	$ -
		ROADS (m)	8,000	$ -	$ -
		BRIDGES FOR ROADS (unit)	6	$ -	$ -
MEMBRANE	WR (PAG) 40Mm3	SURFACE OF WASTE ROCK PILE (PAG) (Ha)	113.5	$ 505,000	$ 57,317,500
		LENGTH OF DITCHES (m)	4250	$ 1,685	$ 7,159,125
		SURFACE OF BASIN (Ha)	14.5	$2,060,000	$ 29,870,000
		VOLUME OF THE BASIN (m^3)	138,300	$ -	$ -
		ROADS (m)	1,100	$ 818	$ 900,167
		BRIDGES FOR ROADS (unit)	1	$2,000,000	$ 2,000,000
					$ 203,571,992



CAPEX (15% weight)



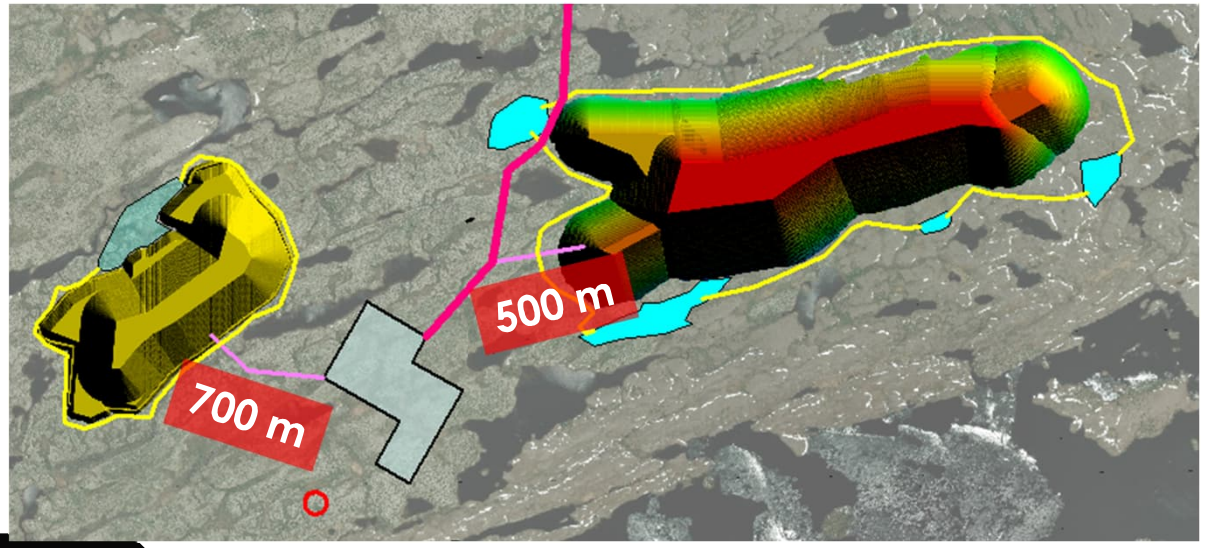




	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
CAPEX (MILLON DOLLARS)	**$ 184.4**	**$ 212.9**	**$ 131.6**	**$ 188.0**	**$ 203.6**
CAPEX (% OF INCREASE)	40%	62%	0%	43%	55%

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
1	CAPEX	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Very Poor	Very Good	Poor	Very Poor

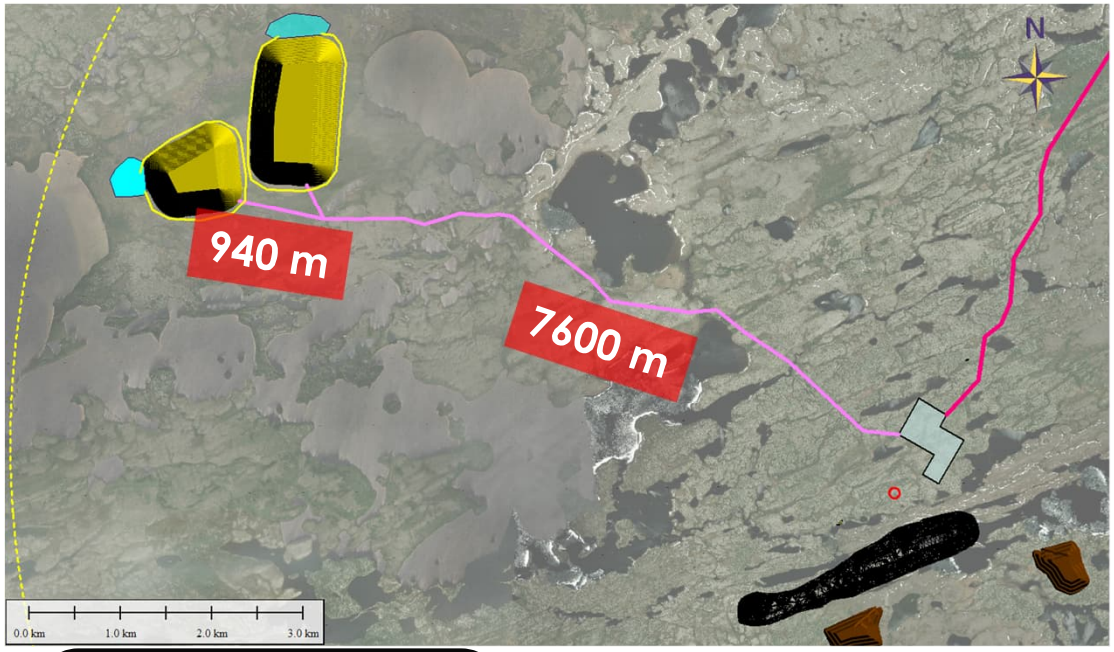
OPEX (7.5% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
OPEX (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**
OPEXO(% Of increase	0%	612%	0%	900%	808%

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**OPEX**	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

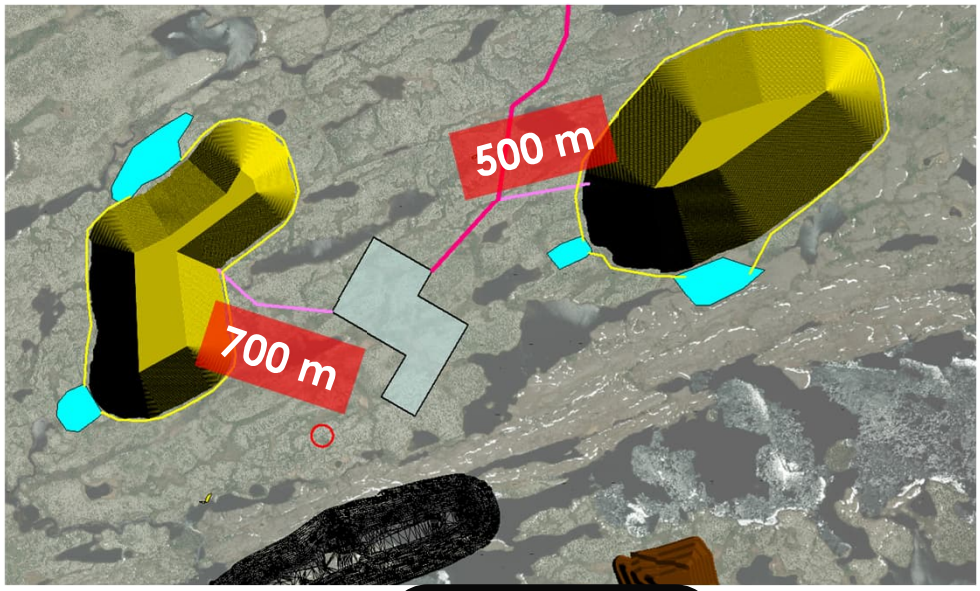
OPEX
(7.5% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
OPEX (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**
OPEXO(% Of increase)	0%	612%	0%	900%	808%

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**OPEX**	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

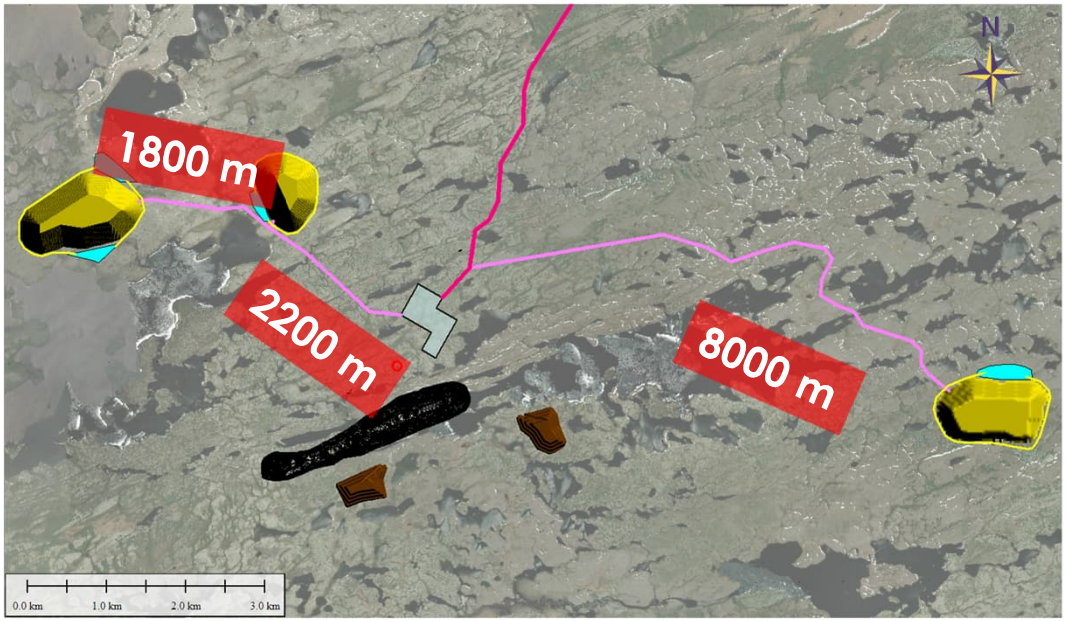
OPEX
(7.5% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
OPEX (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**
OPEXO(% Of increase)	0%	612%	0%	900%	808%

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**OPEX**	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

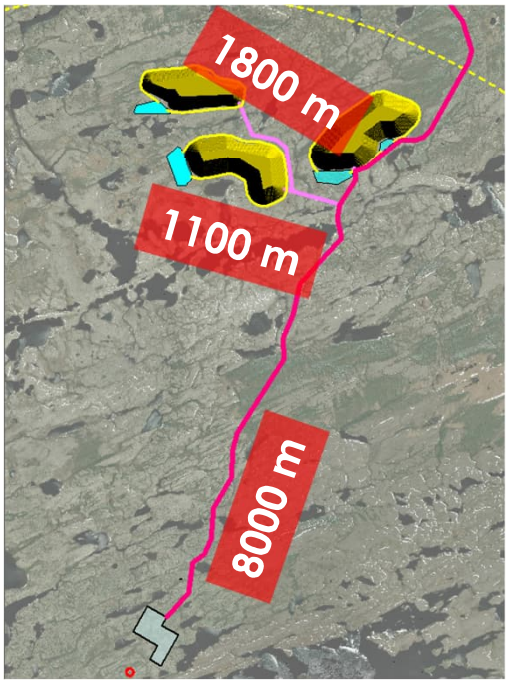
OPEX
(7.5% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
OPEX (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**
OPEXO(% Of increase)	0%	612%	0%	900%	808%

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**OPEX**	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

OPEX
(7.5% weight)

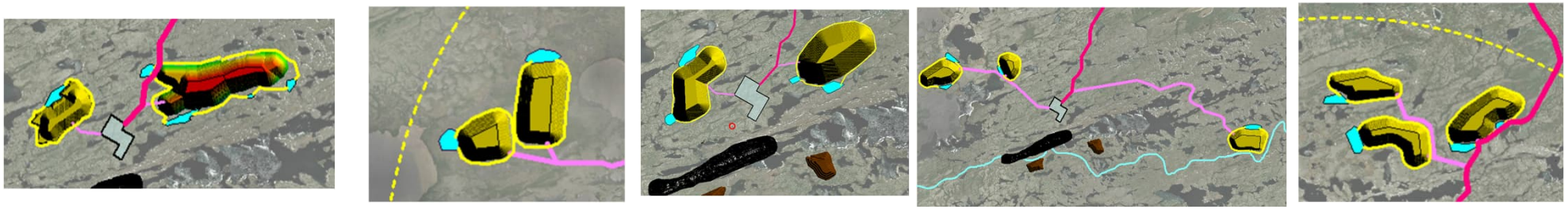


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
OPEX (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**
OPEXO(% Of increase)	0%	612%	0%	900%	808%

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
1	**OPEX**	cost	least costly alternative	Increase of cost 15%	Increase of cost 30%	Increase of cost 45%	Increase > 45%

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Poor	Very Good	Very Poor	Very Poor

Phasing possibility - Cashflow (ROADS, DITCHES, BASINS) (6% weight)

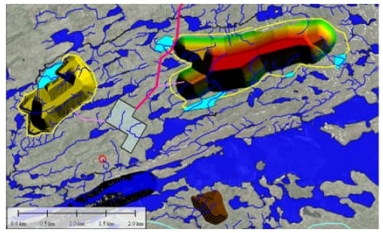
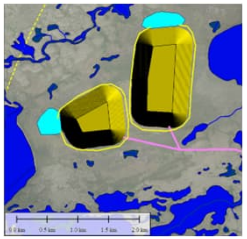
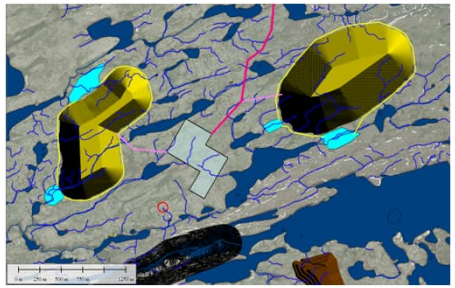
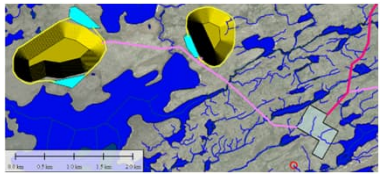
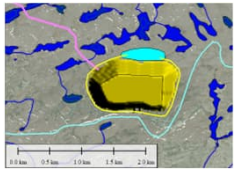
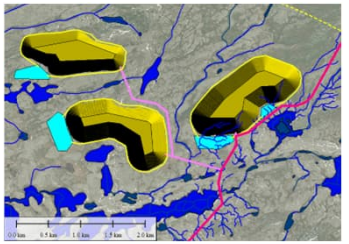


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	3 phases with less than 30% of capex beyond year 1	1 phase	2 phases with less than 30% of capex beyond year 1	2 phases with less than 30% of capex beyond year 1	2 phases with less than 30% of capex beyond year 1

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)**	Qualitative	The works can be spread in at least 3 phases with more than 30% of capex beyond year 0	The works can be spread in at least 3 phases with less than 30% of capex beyond year 1	The works can be spread in at least 2 phases with more than 30% of capex beyond year 0	The works can be spread in at least 2 phases with less than 30% of capex beyond year 1	All the works have to be done in one phase in year 0

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Very Poor	Poor	Poor	Poor

Fish habitat compensation (1.5% weight)

	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
SURFACE OF WETLANDS (Ha)	45.7	142.1	42.4	9.2	17.6
SURFACE OF WATER STREAMS (Ha)	22.8	0	29.2	0.1	0.7
Total area wetlands and water	68.5	142.1	71.6	9.3	18.3

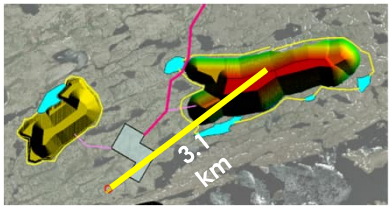
	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	Fish habitat compensation	Surface of wetlands and water streams	less than 10 Ha	10 To 20 Ha	20 to 30 Ha	30 to 40 Ha	More than 40 Ha

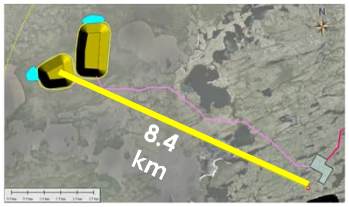
SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Very Poor	Very Poor	Very good	Good

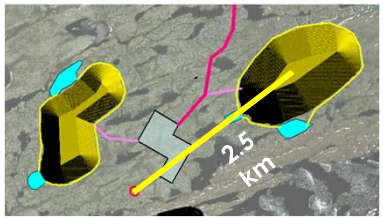
Operational consideration (10% weight)

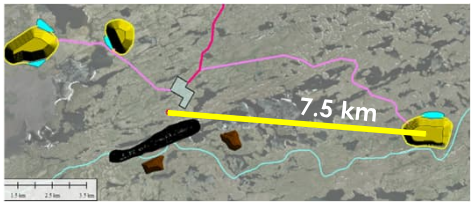
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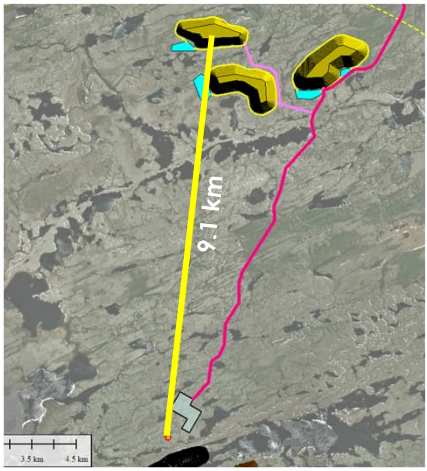
Distance from mill/pit (4.7% weight)









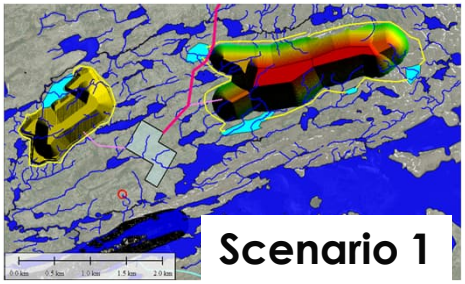


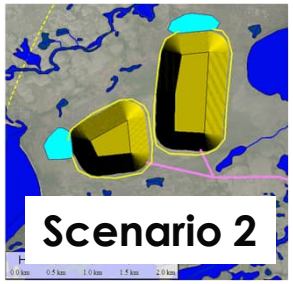
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Distance from mill/pit (the farthest pile) (km)	3.1	8.4	2.5	7.5	9.1

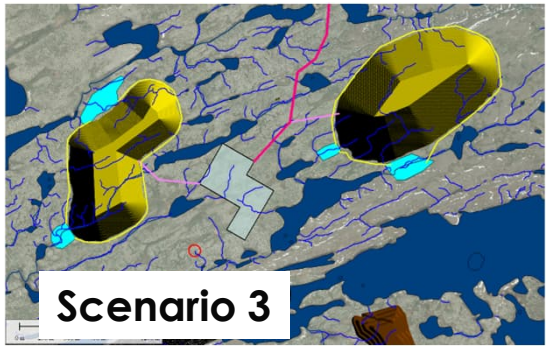
SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
1	Distance from mill/pit	distance in km	0-2 km	2-4 km	4-6 km	6-8 km	> 8 km

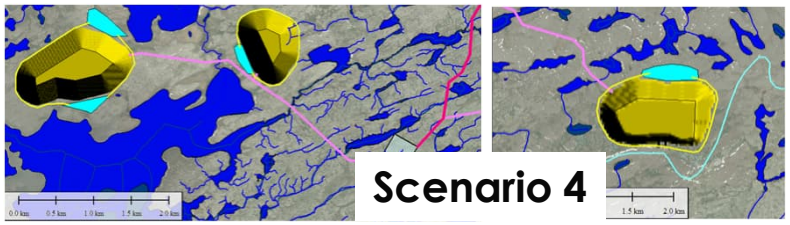
SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Very Poor	Good	Poor	Very Poor

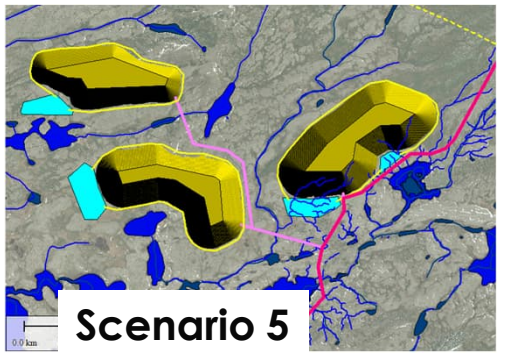
Expansion capability (3.2% weight)









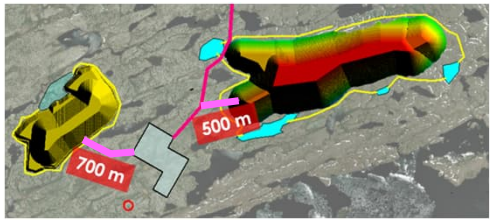


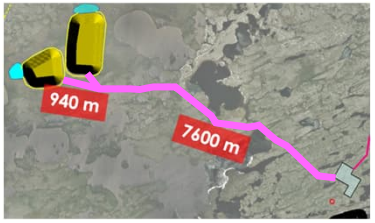
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Expansion capability	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies	10 - 20% going higher and enlarging the footprint and no affecting more water bodies	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies	It is possible to gain maximum 10% of capacity by going higher and avoiding to affect more water bodies

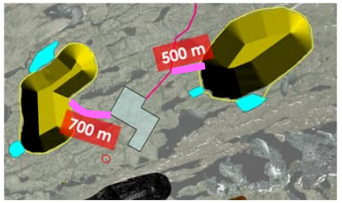
SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
3	Expansion capability	percentage of expansion	40% or more additional volume can be deposited in the site option.	30 - 40% additional volume can be deposited in the site option.	20 - 30% additional volume can be deposited in the site option.	10 - 20% additional volume can be deposited in the site option.	less than 10% additional volume can be deposited in the site option.

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Very Poor	Poor	Very Poor	Very Poor

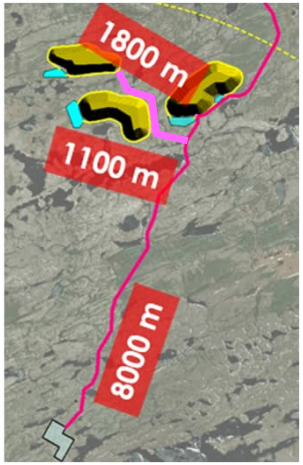
Operational safety (2.1% weight)











	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Operational safety (Km of road to transport material)	**1.20**	**8.54**	**1.20**	**12.00**	**10.90**

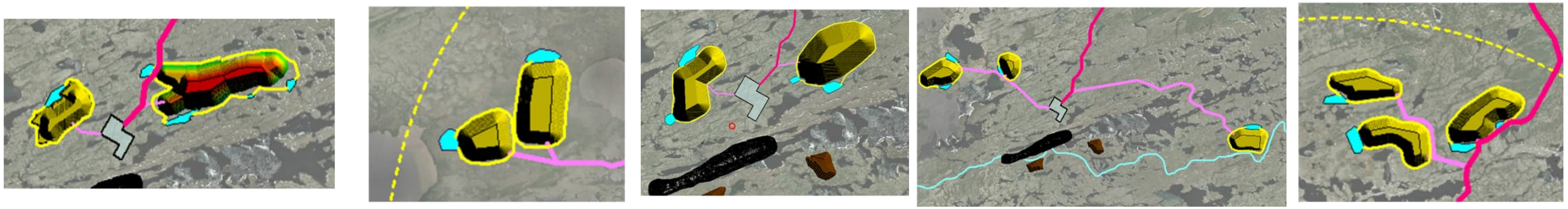
SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Operational safety**	distance of roads in km	Minimal Hazard Exposure due to the very short distances (0-2 km) between the sites and the pit or the mill, these distances do not have measurable effect on operator well-being or safety.	Low Hazard Exposure due to the short distances (2-4 km) between the sites and the pit or the mill, these distances have minimal effect on operator fatigue or performance.	Moderate Hazard Exposure due to the intermediate distance (4-6 km) between the sites and the pit or the mill, fatigue and reduced performance are common due to travel distances but manageable with breaks.	High Hazard Exposure due to the long distances (6-8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.	Extreme Hazard Exposure due to the very long distances (>8km) between the sites and the pit or the mill, these long travel distances significantly increase operator fatigue, leading to safety concerns.

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Poor	Very Good	Very Poor	Very Poor

Environmental impacts (25% weight)

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STOCKPILE footprint (7.5% weight)

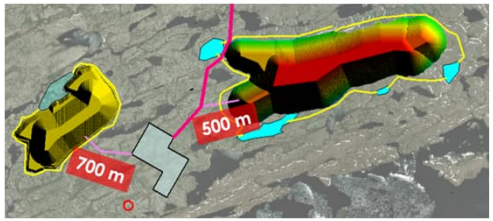


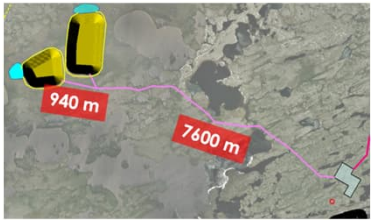
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
STOCKPILE footprint (Ha)	364	256	252	290	330
(% OF INCREASE)	44%	1%	0%	15%	31%

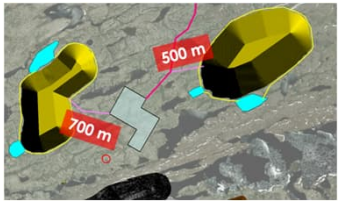
	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**STOCKPILE footprint**	Footprint surface	Smallest footprint surface area	Increase of footprint by 15%	Increase of footprint by 30%	Increase of footprint by 45%	Increase of footprint by 100%

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Very Good	Very Good	Good	Fair

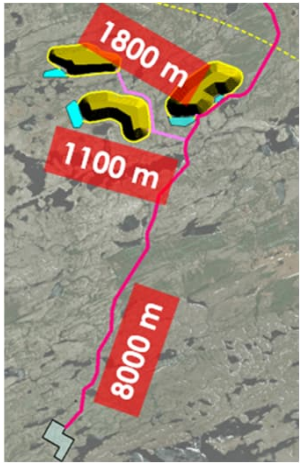
CO_2 emissions (5.0% weight)









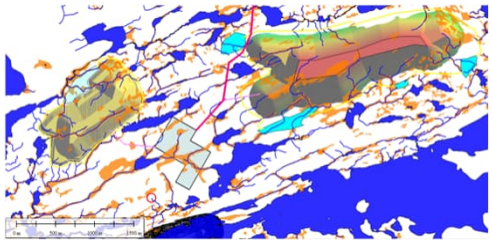
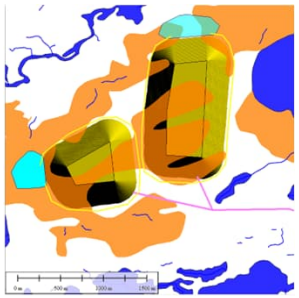
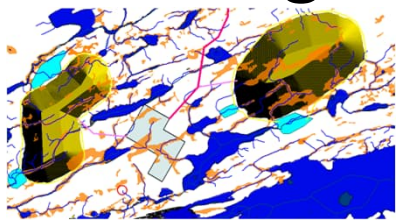
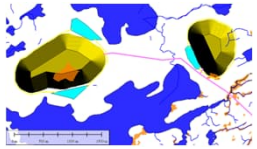
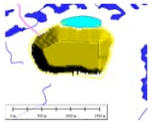
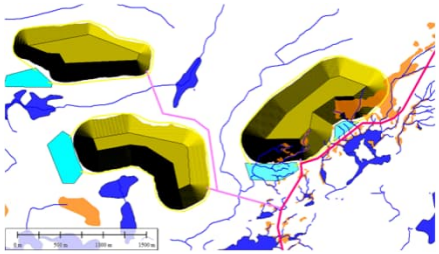


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
CO2 emissions (the longer road) Km	1.2	8.54	1.2	12	10.9

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
2	**CO2 emissions**	Distance of roads in kilometers	The project generates minimal CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (0-1Km)	The project generates below-average CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (1-2Km)	The project generates a moderate amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (2-3Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (3-4Km)	The project generates a large amount of CO_2 emissions, which are based in the haulage distance between the pit and the point of deposition (> 4Km)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Very Poor	Good	Very Poor	Very Poor

Wetlands/ Rivers / Streams (7.5% weight)

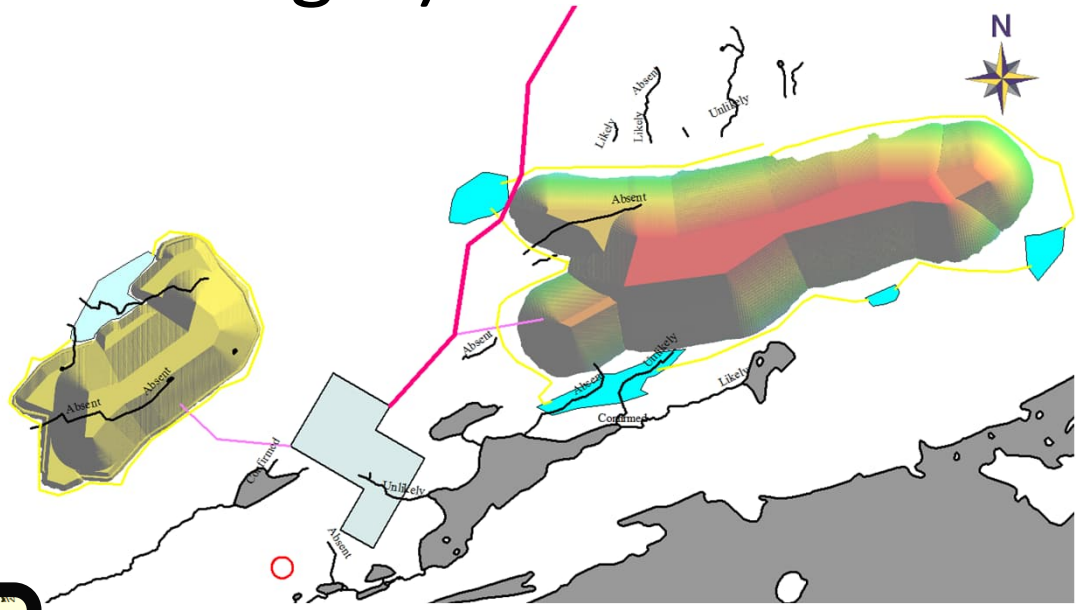







	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
TOTAL SURFACE (Ha)	381	268	265	302	344
SURFACE OF WETLANDS	45.7	142.1	42.4	9.2	21.4
SURFACE OF WATER STREAMS	22.8	0	29.2	0.1	0.7
Total area wetlands and water	68.5	142.1	71.6	9.3	22.1

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Wetlands/ Rivers / Streams**	Surface of wetlands and water streams	less than 10 Ha	10 To 20 Ha	20 to 30 Ha	30 to 40 Ha	More than 40 Ha

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Very Poor	Very Poor	Very Good	Fair

Fish presence (5.0% weight)

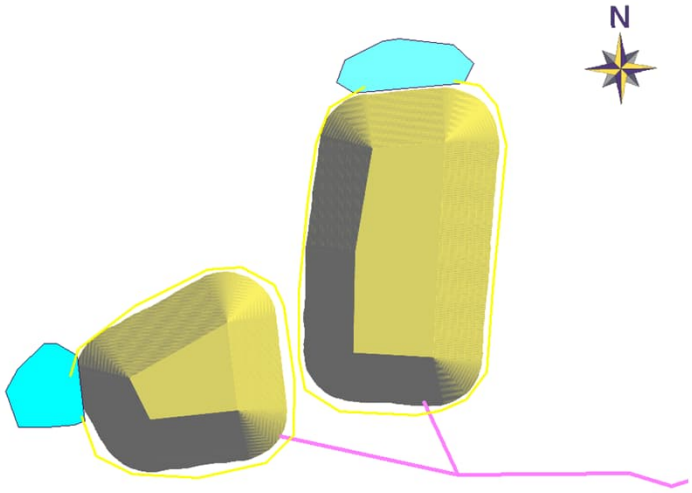


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Fish presence	Confirmed no fish habitat	No information available	Confirmed fish habitat	No information available	No information available

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fish presence**	Presence of fish	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Fair	Very Poor	Fair	Fair

Fish presence (5.0% weight)

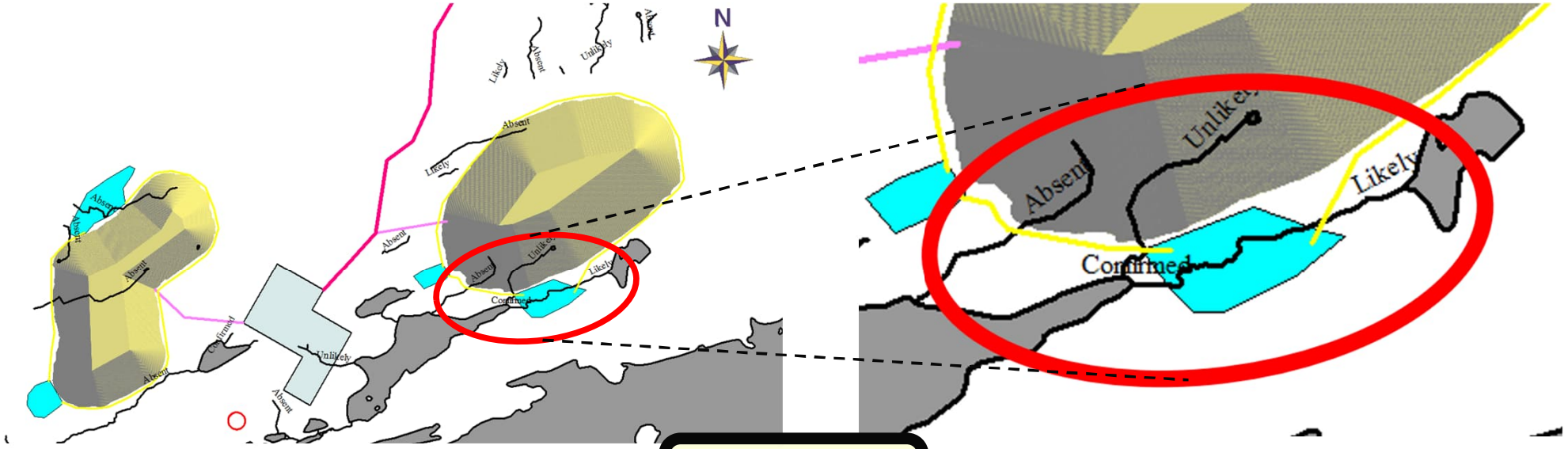


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Fish presence	Confirmed no fish habitat	No information available	Confirmed fish habitat	No information available	No information available

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fish presence**	Presence and distance in km	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	**Fair**	**Very Poor**	**Fair**	**Fair**

Fish presence (5.0% weight)

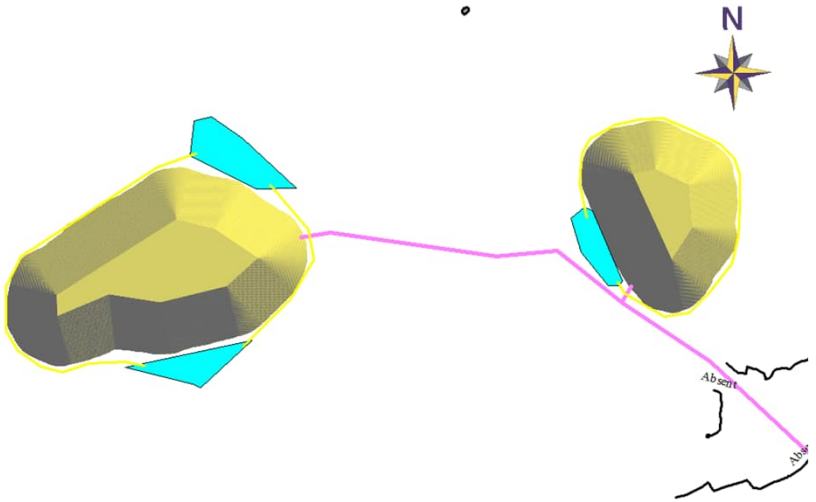


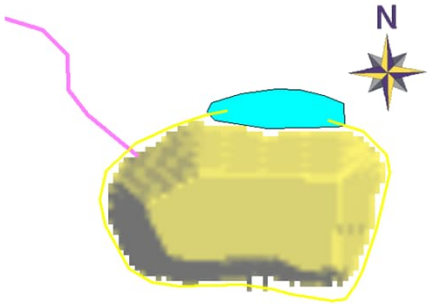
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Fish presence	Confirmed no fish habitat	No information available	Confirmed fish habitat	No information available	No information available

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fish presence**	Presence and distance in km	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Fair	Very Poor	Fair	Fair

Fish presence (5.0% weight)



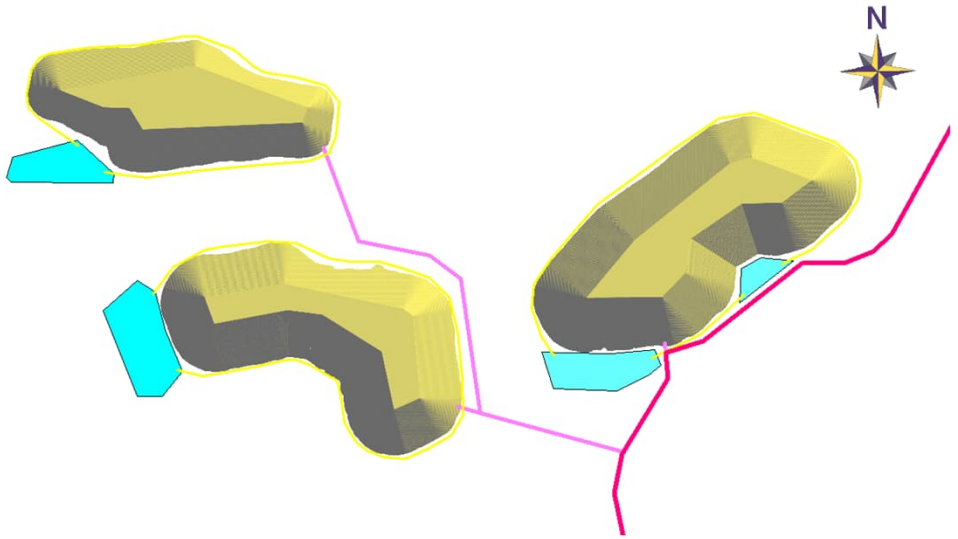


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Fish presence	Confirmed no fish habitat	No information available	Confirmed fish habitat	No information available	No information available

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fish presence**	Presence and distance in km	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	**Fair**	**Very Poor**	**Fair**	**Fair**

Fish presence (5.0% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Fish presence	Confirmed no fish habitat	No information available	Confirmed fish habitat	No information available	No information available

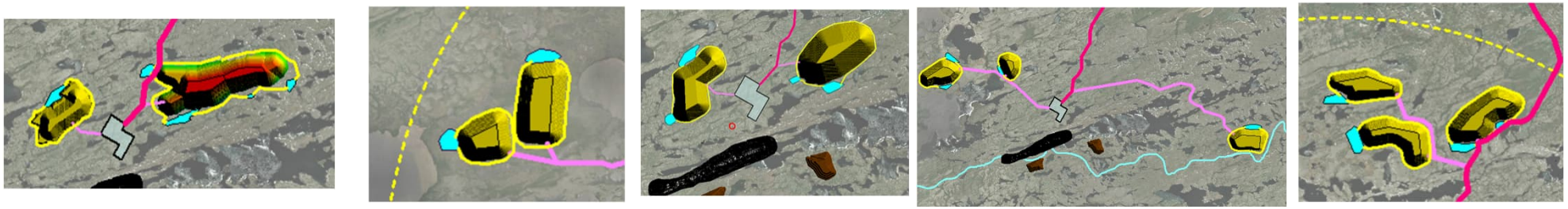
SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fish presence**	Presence and distance in km	Confirmed no fish habitat	likely no fish habitat	No information available	Likely fish habitat	Confirmed fish habitat

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	**Fair**	**Very Poor**	**Fair**	**Fair**

Technical consideration (10% weight)

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Water management infrastructure) (3.5% weight)

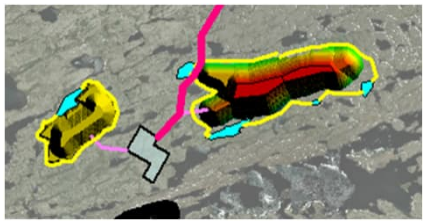
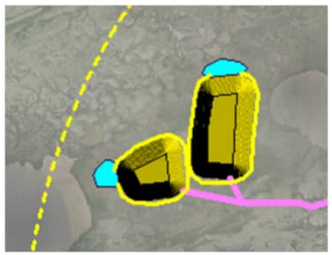
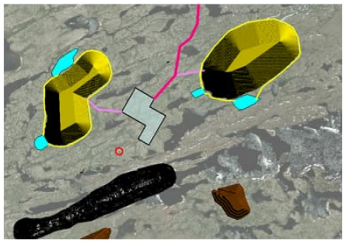
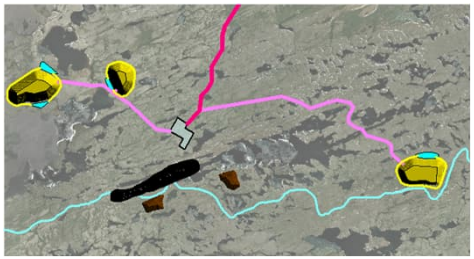
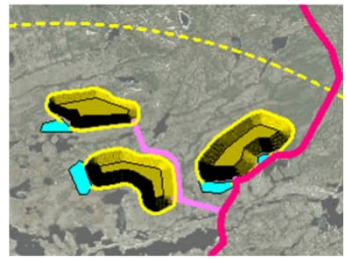


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Water management infrastructure (stock pile+basin(Ha))	381	268	265	302	344
(% OF INCREASE)	44%	1%	0%	14%	30%

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Water management infrastructure**	watershed surface	Alternative with the least watershed surface area to manage	Increase of watershed by 15%	Increase of watershed by 30%	Increase of watershed by 45%	Increase of watershed by 100% or more

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Very Good	Very Good	Good	Fair

Water management complexity (2.0% weight)

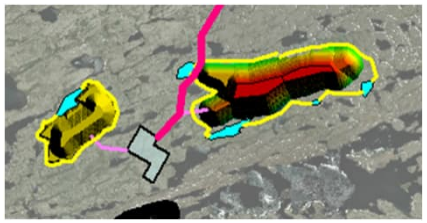
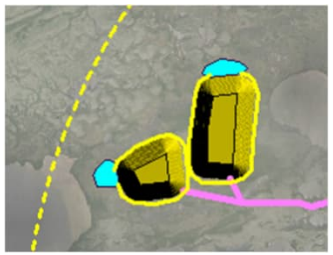
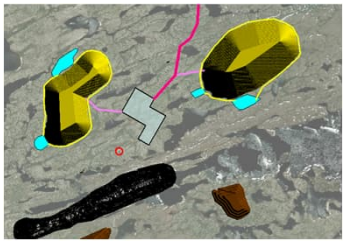
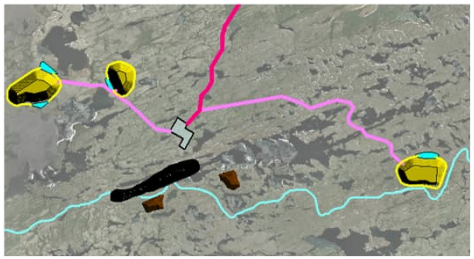
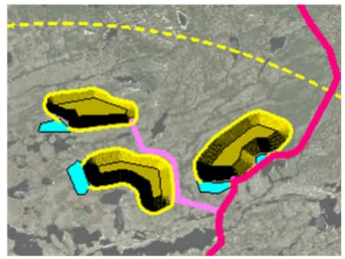






	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
water management complexity (number of basins)	5 (by gravity)	2 (by gravity)	4 (by gravity)	4 (by gravity)	4 (by gravity)

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
2	**water management complexity**	Water surface	Site water management can be done in one water management system and water drainage done by gravity	Site water management can be done in one water management systems requiring on or more sump with ongoing pumping	Site water management can be done in two water management systems and water drainage done by gravity	Site water management can be done in two water management systems requiring on or more sump with ongoing pumping	Site water management would require more than 2 water management system

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Poor	Fair	Very Poor	Very Poor	Very Poor

Project schedule & Constructability (1.5% weight)

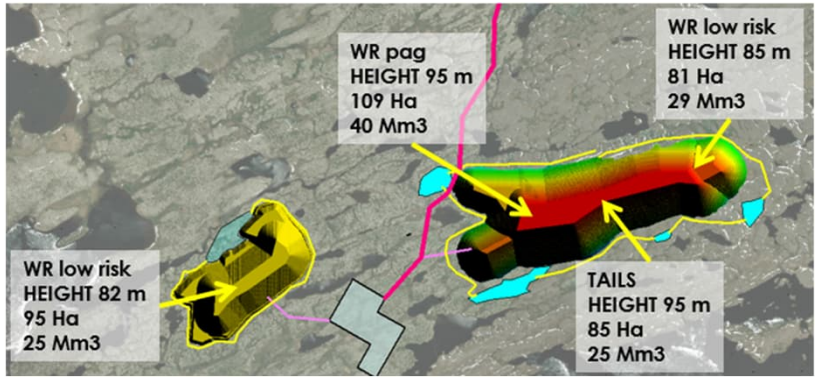






	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
DISTANCE TO CENTRAL CAMP (KM)	3.1	8.4	2.5	7.5	9.1
WETLANDS AND WATER SURFACE (HA)	68.5	142.1	71.6	9.2	17.6
BASINS	5	2	4	4	4

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Project schedule & Constructability**	distance, % of wetlands or water streams and area of basins	Site that is closest to the central camp, where the work area impacts the least area of wetlands or water surfaces and has the least surface of basins, and the topography is not hilly. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is within 1km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 20% of the best alternative related to wetlands and basin surface. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is within 3km to the closest alternative, where the work area impacts the least area of wetlands or water surfaces or within 40% of the best alternative related to wetlands and basin surface. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is within 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces within 50% of the best alternative related to wetlands and basin surface. Reduce the scoring by 2 category if the option creates a permitting delay risk	Site that is more 6km to the closest alternative, where the work area impacts the area of wetlands or water surfaces grater 50% of the best alternative related to wetlands and basin surface.

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Poor	Poor	Very Poor	Very Poor

Embankment (earthwork) height (3.0% weight)

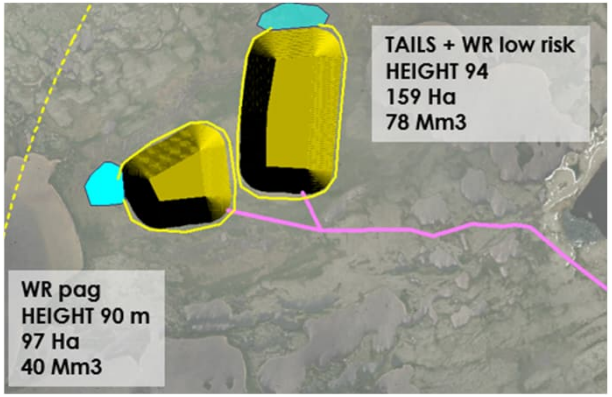


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Embankment (earthwork) height (m	**TAILS-HEIGHT 95 m WR pag-HEIGHT 95 m WR low risk-HEIGHT 85 m**	**WR pag-HEIGHT 90 m TAILS + WR low risk-HEIGHT 94**	**WR (low risk) H=82m TAILS + WR low risk + pag (125m)**	**WR low risk-HEIGHT 88 m TAILS-HEIGHT 90 WR pag-HEIGHT 90 m**	**TAILS-HEIGHT 77 WR pag-HEIGHT 80 m WR low risk-HEIGHT 90 m**

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Embankment (earthwork) height**	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)	Increase of height of 20 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)	Increase of height of 30 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)	Increase of height of > 30 m Increase the score by 1 category if height is equal or less than 90m (WR-low risk)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Good	Very Poor	Very Good	Very Good

Embankment (earthwork) height (3.0% weight)

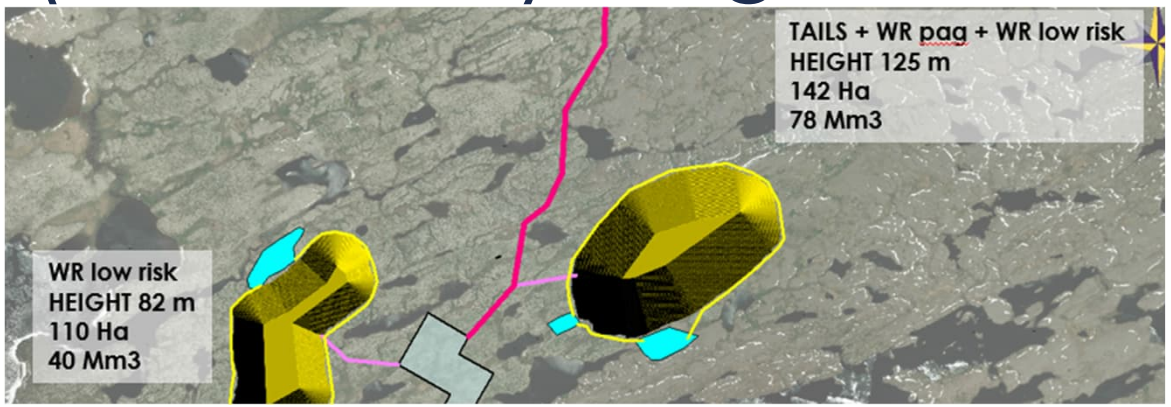


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Embankment (earthwork) height (m)	**TAILS-HEIGHT 95 m WR pag-HEIGHT 95 m WR low risk-HEIGHT 85 m**	**WR pag-HEIGHT 90 m TAILS + WR low risk-HEIGHT 94**	**WR (low risk) H=82m TAILS + WR low risk + pag (125m)**	**WR low risk-HEIGHT 88 m TAILS-HEIGHT 90 WR pag-HEIGHT 90 m**	**TAILS-HEIGHT 77 WR pag-HEIGHT 80 m WR low risk-HEIGHT 90 m**

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Embankment (earthwork) height**	Height	Alternative with the least height	Increase of height of 10 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 20 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of > 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Good	Very Poor	Very Good	Very Good

Embankment (earthwork) height (3.0% weight)

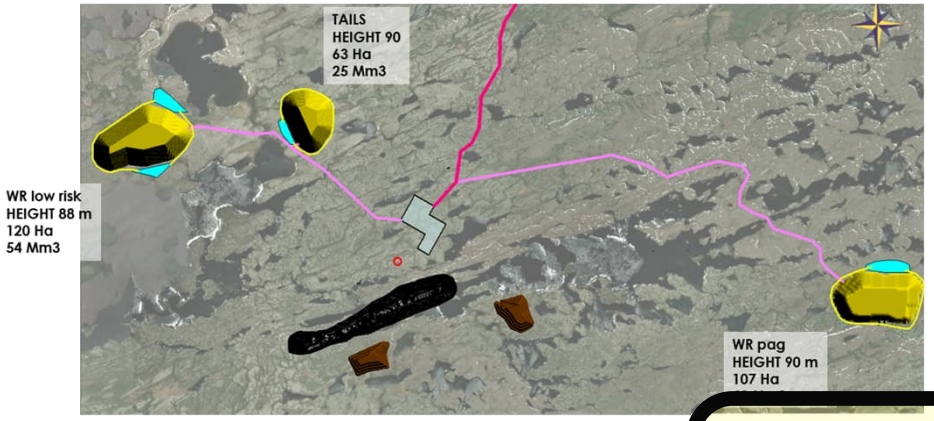


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Embankment (earthwork) height (m)	**TAILS-HEIGHT 95 m WR pag-HEIGHT 95 m WR low risk-HEIGHT 85 m**	**WR pag-HEIGHT 90 m TAILS + WR low risk-HEIGHT 94**	**WR (low risk) H=82m TAILS + WR low risk + pag (125m)**	**WR low risk-HEIGHT 88 m TAILS-HEIGHT 90 WR pag-HEIGHT 90 m**	**TAILS-HEIGHT 77 WR pag-HEIGHT 80 m WR low risk-HEIGHT 90 m**

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Embankment (earthwork) height**	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 20 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of > 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Good	Very Poor	Very Good	Very Good

Embankment (earthwork) height (3.0% weight)

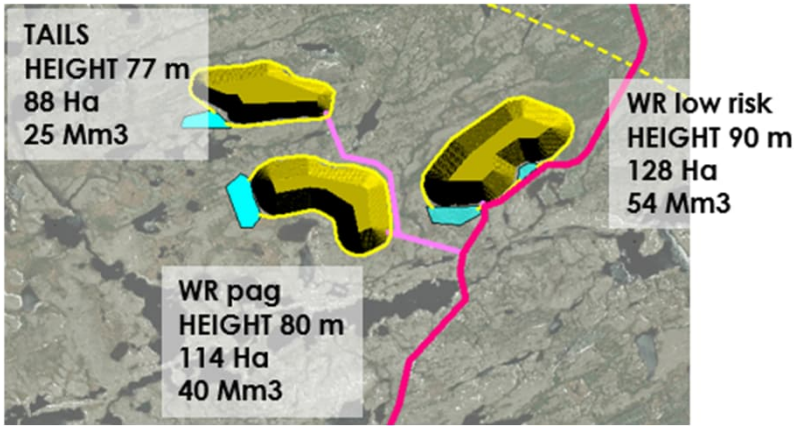


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Embankment (earthwork) height (m)	**TAILS-HEIGHT 95 m WR pag-HEIGHT 95 m WR low risk-HEIGHT 85 m**	**WR pag-HEIGHT 90 m TAILS + WR low risk-HEIGHT 94**	**WR pag-HEIGHT 82 m TAILS + WR low risk HEIGHT 125 m**	**WR low risk-HEIGHT 88 m TAILS-HEIGHT 90 WR pag-HEIGHT 90 m**	**TAILS-HEIGHT 77 WR pag-HEIGHT 80 m WR low risk-HEIGHT 90 m**

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Embankment (earthwork) height**	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 20 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of > 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Good	Very Poor	Very Good	Very Good

Embankment (earthwork) height (3.0% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Embankment (earthwork) height (m)	**TAILS-HEIGHT 95 m WR pag-HEIGHT 95 m WR low risk-HEIGHT 85 m**	**WR pag-HEIGHT 90 m TAILS + WR low risk-HEIGHT 94**	**WR pag-HEIGHT 82 m TAILS + WR low risk HEIGHT 125 m**	**WR low risk-HEIGHT 88 m TAILS-HEIGHT 90 WR pag-HEIGHT 90 m**	**TAILS-HEIGHT 77 WR pag-HEIGHT 80 m WR low risk-HEIGHT 90 m**

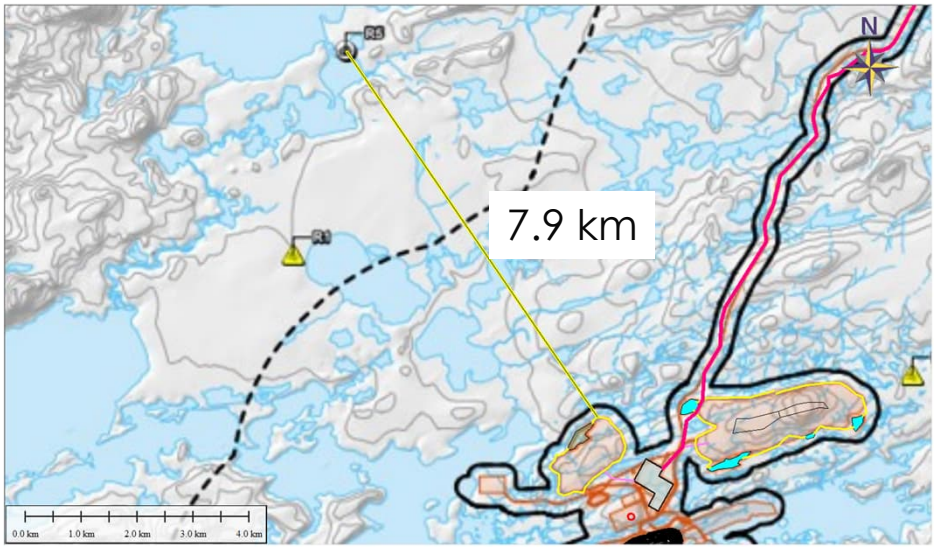
	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Embankment (earthwork) height**	Height	Alternative with the less height	Increase of height of 10 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 20 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)	Increase of height of > 30 m Increase the score by 1 category if height is equal or less than 90m (WR-pag)

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Very Good	Very Good	Very Poor	Very Good	Very Good

Social impacts(25% weight)

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Proximity to sensitive receptors (cabins and travel routes) (6.3% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Proximity to sensitive receptors (distance in km from Cabin or trave route)	7.9	3.3	7.9	5.7	3.7

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Fair	Good	Good	Fair

Proximity to sensitive receptors (cabins and travel routes) (6.3% weight)

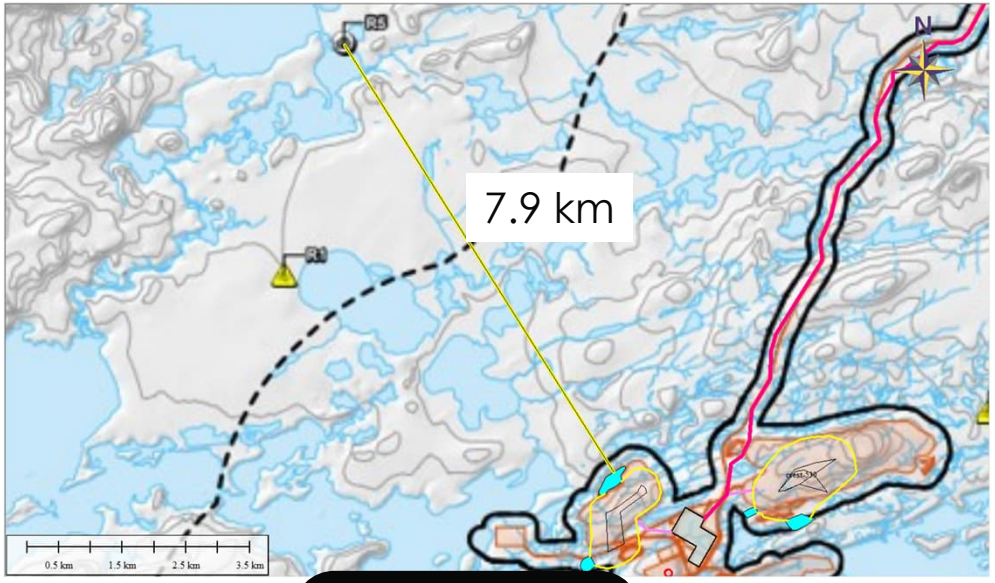


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Proximity to sensitive receptors (distance in km from Cabin or travel route)	7.9	3.3	7.9	5.7	3.7

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Fair	Good	Good	Fair

Proximity to sensitive receptors (cabins and travel routes) (6.3% weight)

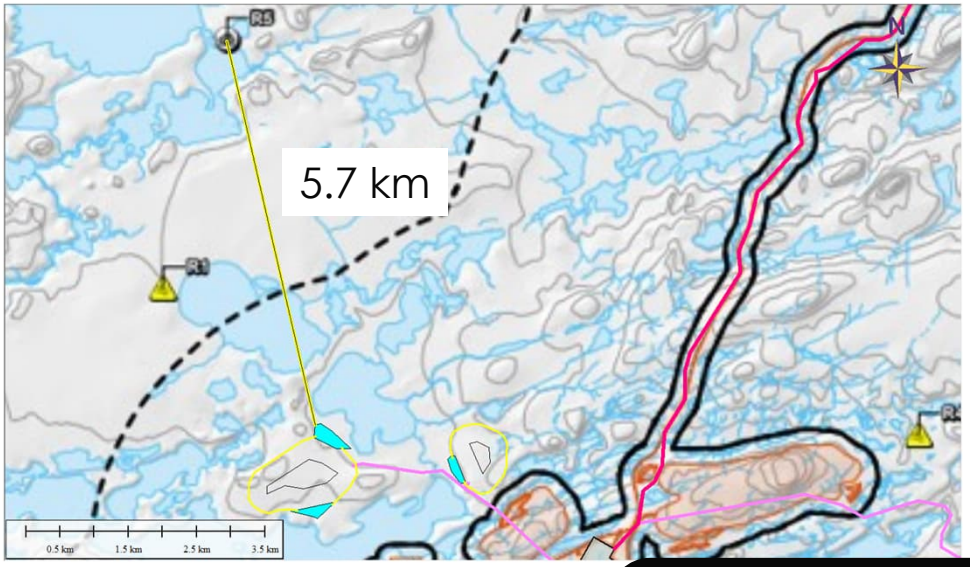


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Proximity to sensitive receptors (distance in km from Cabin or travel route)	7.9	3.3	7.9	5.7	3.7

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Fair	Good	Good	Fair

Proximity to sensitive receptors (cabins and travel routes) (6.3% weight)

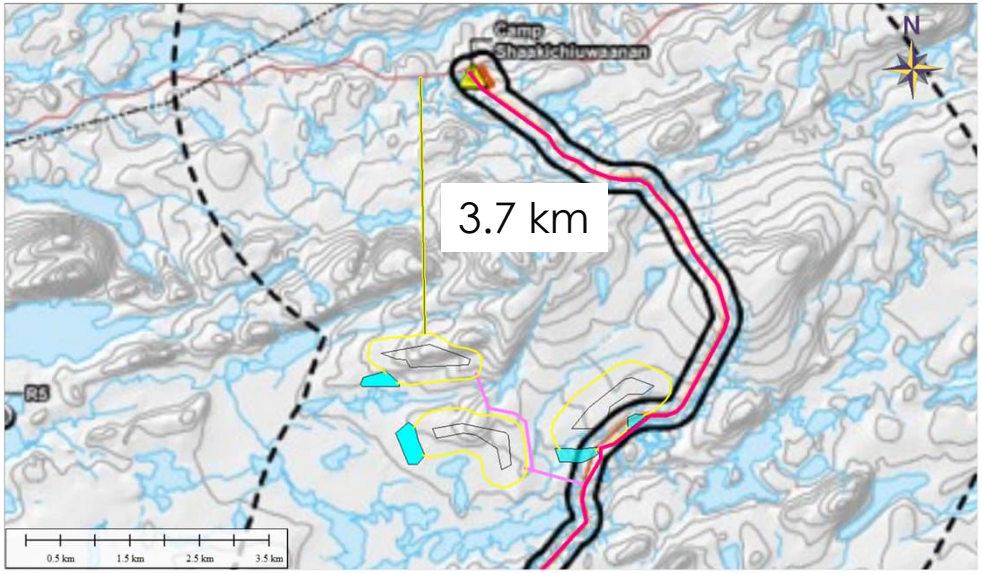


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Proximity to sensitive receptors (distance in km from Cabin or travel route)	7.9	3.3	7.9	5.7	3.7

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Fair	Good	Good	Fair

Proximity to sensitive receptors (cabins and travel routes) (6.3% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Proximity to sensitive receptors (distance in km from Cabin or travel route)	7.9	3.3	7.9	5.7	3.7

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
1	**Proximity to sensitive receptors (cabins and travel routes)**	distance in km from Cabin or travel route	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Fair	Good	Good	Fair

Moose feeding areas and goose feeding areas(6.3% weight)

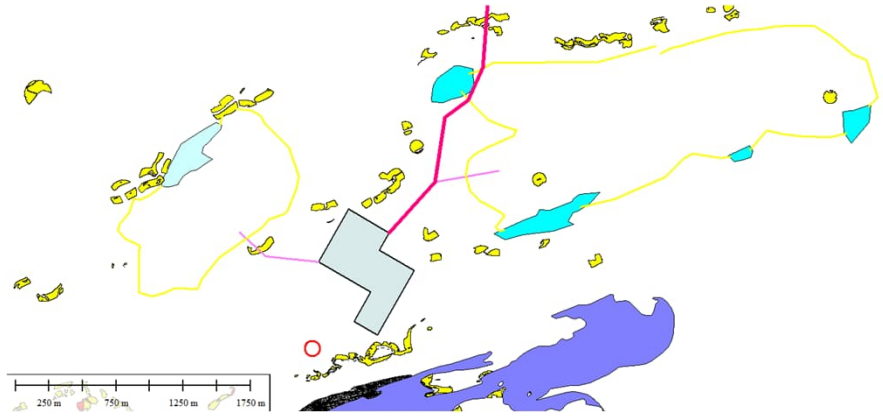


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
moose feeding areas and goose feeding areas (distance in km to goose or moose feeding areas)	2.5	3.4	2.3	0.2	7.9

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
2	**moose feeding areas and goose feeding areas**	distance in km Hunting ground	more than 10 km	5 - 10 km	2- 5 km	1 - 2 km	less than 1 km

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Fair	Fair	Fair	Very Poor	Good

Archaeological potential (6.3% weight)

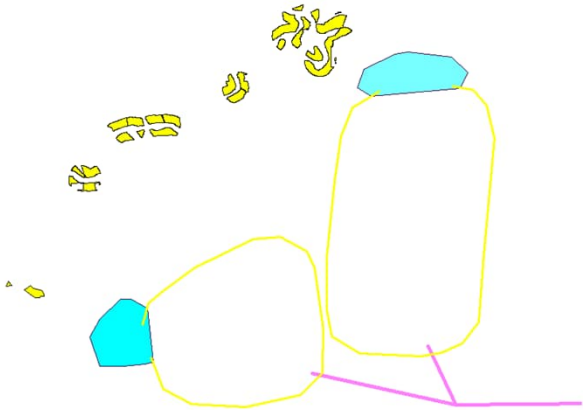


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Archaeological potentia	No high potential areas 2 medium areas inside the sites The closest high is at 0.8 km	No high or medium potential areas in the perimeter The closest high P is at 5.8 km The closest medium P is at 0.2 km	No high potential areas 1 medium area inside the sites The closest high is at 0.47 km	No high or medium potential areas The closest medium is at 0.1 km The closest high is at 1.9 km	No high potential areas 2 medium areas inside the sites The closest high is at 8.0 km

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Good	Good	Very Good	Good

Archaeological potential (6.3% weight)

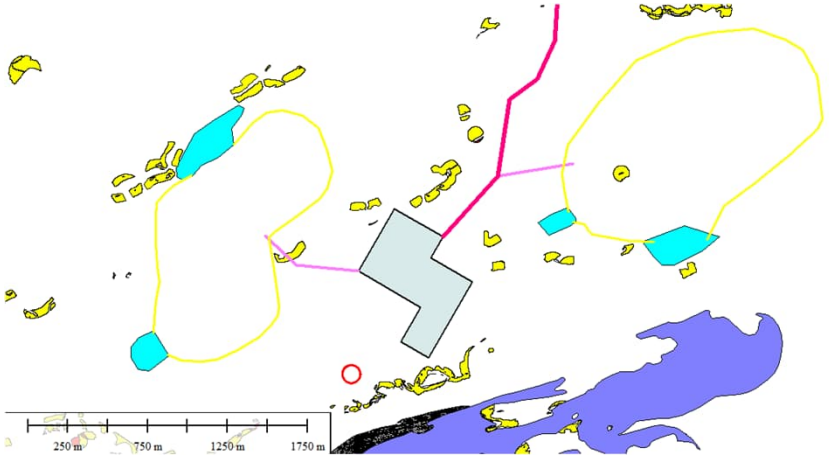


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Archaeological potential	No high potential areas 2 medium areas inside the sites The closest high is at 0.8 km	No high or medium potential areas in the perimeter The closest high P is at 5.8 km The closest medium P is at 0.2 km	o high potential areas medium area inside the sites e closest high is at 0.47 km	No high or medium potential areas The closest medium is at 0.1 km The closest high is at 1.9 km	No high potential areas 2 medium areas inside the sites The closest high is at 8.0 km

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Good	Good	Very Good	Good

Archaeological potential (6.3% weight)

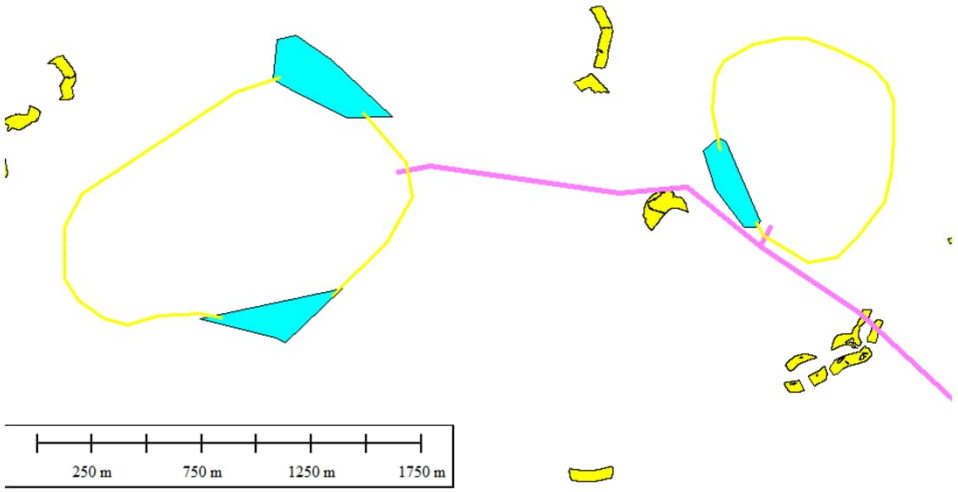


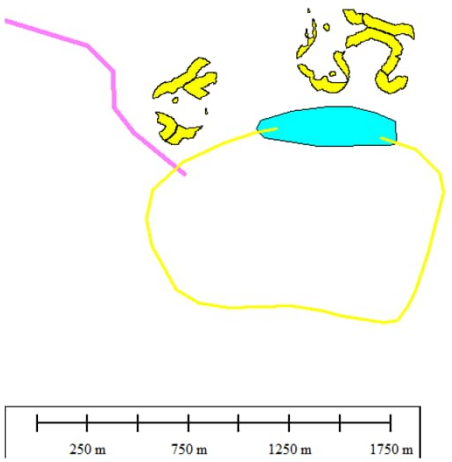
	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Archaeological potential	No high potential areas 2 medium areas inside the sites The closest high is at 0.8 km	No high or medium potential area in the perimeter The closest high P is at 5.8 km The closest medium P is at 0.2 km	No high potential areas 1 medium area inside the sites The closest high is at 0.47 km	o high or medium potential areas e closest medium is at 0.1 km e closest high is at 1.9 km	No high potential areas 2 medium areas inside the sites The closest high is at 8.0 km

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Good	Good	Very Good	Good

Archaeological potential (6.3% weight)



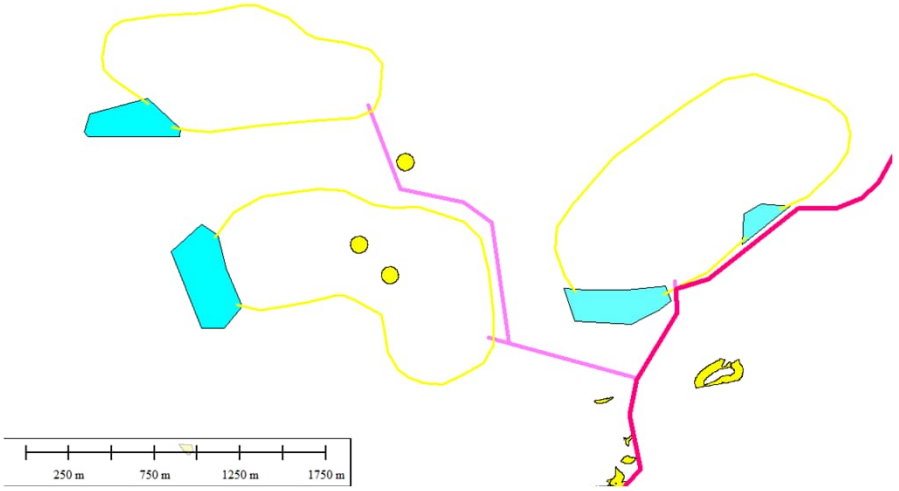


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Archaeological potential	No high potential areas 2 medium areas inside the sites The closest high is at 0.8 km	No high or medium potential areas in the perimeter The closest high P is at 5.8 km The closest medium P is at 0.2 km	No high potential areas 1 medium area inside the sites The closest high is at 0.47 km	No high or medium potential areas The closest medium is at 0.1 km The closest high is at 1.9 km	No high potential areas 2 medium areas inside the sites The closest high is at 8.0 km

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Good	Good	Very Good	Good

Archaeological potential (6.3% weight)

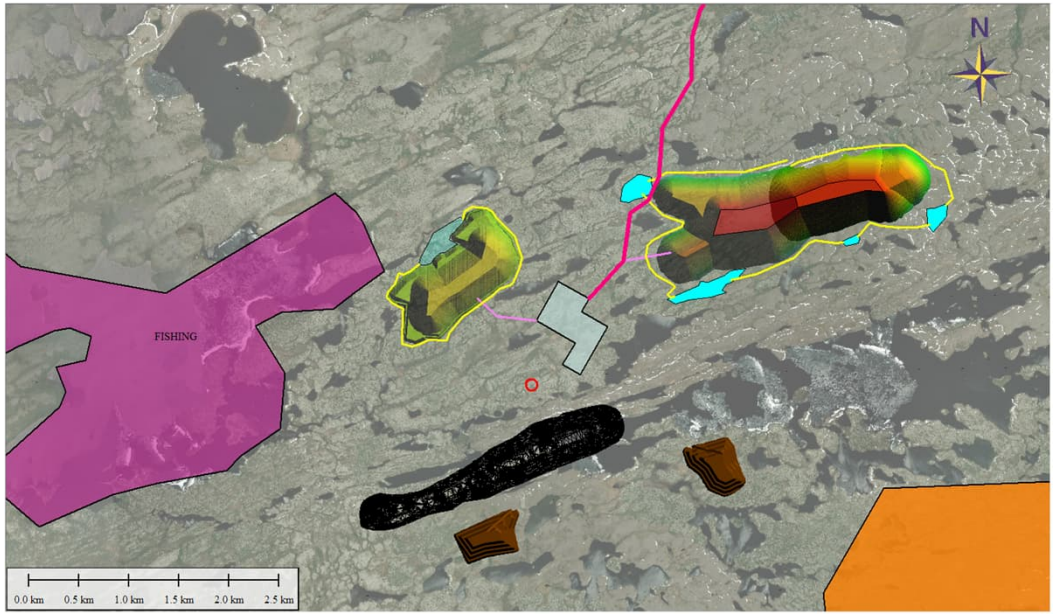


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
Archaeological potential	No high potential areas 2 medium areas inside the sites The closest high is at 0.8 km	No high or medium potential areas in the perimeter The closest high P is at 5.8 km The closest medium P is at 0.2 km	No high potential areas 1 medium area inside the sites The closest high is at 0.47 km	No high or medium potential areas The closest medium is at 0.1 km The closest high is at 1.9 km	No high potential areas 2 medium areas inside the sites The closest high is at 8.0 km

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
3	**Archaeological potential**	distance in m	No high or medium potential areas identified within 500m	No high potential areas identified within 500m	No high potential areas identified within 250 m	High potential area identified within less than 50m of waste rock placement	High potential area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Good	Good	Good	Very Good	Good

Fishing areas (6.3% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
distance to fishing areas (km)	0.15	0.25	0.25	0.12	3.24

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fishing areas**	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Poor	Poor	Poor	Very Good

Fishing areas (6.3% weight)

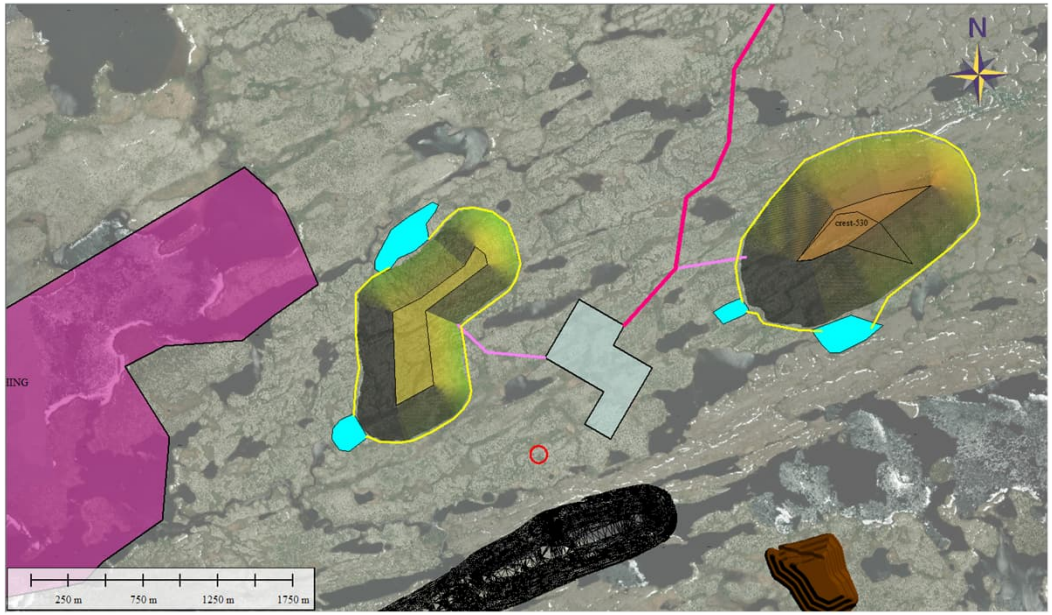


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
distance to fishing areas (km)	0.15	0.25	0.25	0.12	3.24

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fishing areas**	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Poor	Poor	Poor	Very Good

Fishing areas (6.3% weight)

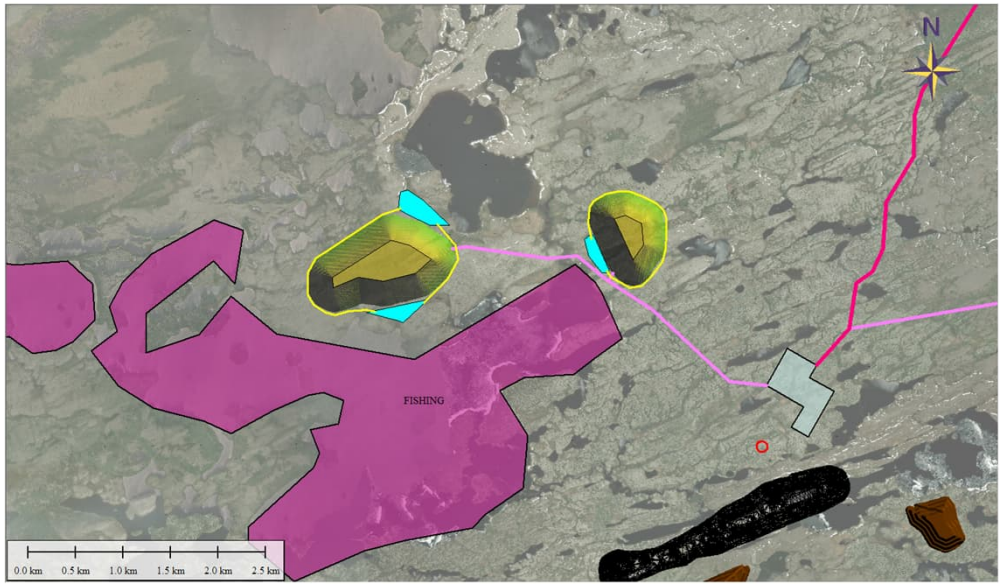


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
distance to fishing areas (km)	0.15	0.25	0.25	0.12	3.24

SUB-ACCOUNT		Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fishing areas**	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Poor	Poor	Poor	Very Good

Fishing areas (6.3% weight)

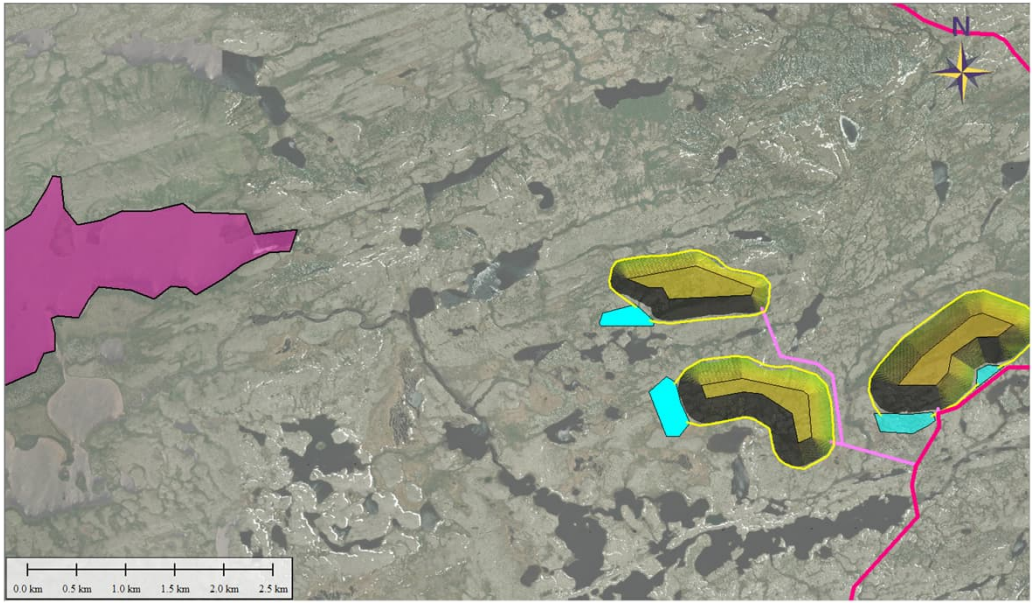


	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
distance to fishing areas (km)	0.15	0.25	0.25	0.12	3.24

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fishing areas**	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Poor	Poor	Poor	Very Good

Fishing areas (6.3% weight)



	SCENARIO 1	SCENARIO 2	SCENARIO 3	SCENARIO 4	SCENARIO 5
distance to fishing areas (km)	0.15	0.25	0.25	0.12	3.24

	SUB-ACCOUNT	Unit	Very Good	Good	Fair	Poor	Very Poor
4	**Fishing areas**	distance in m	None identified within subwatershed	Fishing area identified within 1000m of waste rock placement	Fishing area identified within 500m of waste rock placement	Fishing area identified within less than 500m of waste rock placement	Fishing area identified within the proposed waste rock area

SCORE	Option S1	Option S2	Option S3	Option S4	Option S5
	Poor	Poor	Poor	Poor	Very Good

POETS - BASE CASE RATING

Option ID	Option Rating	Overall	P	O	E	T	S
S1	2	61%	62%	65%	54%	63%	65%
S2	5	44%	20%	20%	52%	80%	60%
S3	1	68%	84%	71%	56%	51%	65%
S4	3	53%	38%	29%	70%	65%	60%
S5	4	49%	27%	20%	52%	58%	80%

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POETS - Sensitivity Analysis – Set-Up

Case	Description	Account Weighting				
		P	O	E	T	S
Selected Case	Base Case	30	10	25	10	25
Case 1	Scenario looking to give equal importance to project economics and social	30	10	20	10	30
Case 2	All accounts weighted equally	20	20	20	20	20
Case 3	Scenario looking to give equal importance to project economics and environment	30	10	30	20	10
Case 4	Account weighted with E+S = 66.7% and P/O/T = 11.1% each (33.3%)	11.1	11.1	44.5	11.1	22.2

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POETS - RESULT OF SESITIVITY ANALYSIS

Option ID	Option Rating	Overall	P	O	E	T	S	Description / Comment
S1	2	61%	62%	65%	54%	63%	65%	Selected: Base Case
S2	4	44%	20%	20%	52%	80%	60%	
S3	1	68%	84%	71%	56%	51%	65%	
S4	3	53%	38%	29%	70%	65%	60%	
S5	4	49%	27%	20%	52%	58%	80%	
S1	2	62%	62%	65%	54%	63%	65%	Case 1
S2	4	44%	20%	20%	52%	80%	60%	
S3	1	68%	84%	71%	56%	51%	65%	
S4	3	53%	38%	29%	70%	65%	60%	
S5	4	50%	27%	20%	52%	58%	80%	
S1	2	62%	62%	65%	54%	63%	65%	Case 2
S2	4	46%	20%	20%	52%	80%	60%	
S3	1	65%	84%	71%	56%	51%	65%	
S4	3	52%	38%	29%	70%	65%	60%	
S5	4	47%	27%	20%	52%	58%	80%	
S1	2	60%	62%	65%	54%	63%	65%	Case 3
S2	4	46%	20%	20%	52%	80%	60%	
S3	1	66%	84%	71%	56%	51%	65%	
S4	3	54%	38%	29%	70%	65%	60%	
S5	5	45%	27%	20%	52%	58%	80%	
S1	2	60%	62%	65%	54%	63%	65%	Case 4
S2	4	50%	20%	20%	52%	80%	60%	
S3	1	62%	84%	71%	56%	51%	65%	
S4	3	59%	38%	29%	70%	65%	60%	
S5	4	53%	27%	20%	52%	58%	80%	

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Appendix B: CASN-A-370-GT-DCR-0001-B_Waste and water management Design criteria.pdf (V2)

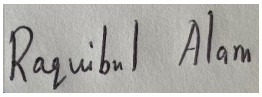
AtkinsRéalis	TECHNICAL NOTE Shaakichiuwaanaan Project – Waste and Water management		
	Document No.: CASN-A-370-GT-DCR-0001 *AtkinsRéalis: 702381-0000-4GEA-0001_PB*	**Rev.** B	**Date:** January 29, 2025

Title of document: **DESIGN CRITERIA – WASTE and WATER MANAGEMENT**

Client: **PATRIOT BATTERY METALS INC.**

Project: **SHAAKICHIUWAANAAN PROJECT – WASTE AND WATER MANAGEMENT**

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(MM-NT-00ER_01 ANG)

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List of Revisions

#	Prep.	Rev.	App.	Date	Pages	Remarks
B	RA	PA	BM	2025-01-29	All	Issue for client's comments

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Appendix A – Site Location and Layout

1. Introduction

AtkinsRéalis has been retained by Innova, a wholly-owned Québec subsidiary of Patriot Battery Metals Inc., for a feasibility study for waste and water management for the Shaakichiuwaanaan Mining Project. The site is in the Eeyou Istchee James Bay region of Québec, Canada, with coordinates of 53°32'00" N, 73°55'00" W. It is approximately 220 km east of Radisson and about 240 km north-northeast of Nemaska. The Property comprises 463 mineral claims, covering roughly 23,710 hectares across two main claim groups. The largest of these groups extends nearly continuously for about 51 km in an east-west direction.

The project will extract spodumene from a pegmatite deposit using a hybrid method that includes open-pit and underground mining. In the initial phase (2029-2034), the estimated annual production rate will be 400 ktpa at a grade of 5.5% lithium oxide (Li2O) concentrate. This will increase to 800 ktpa in the expansion phase (2031-2051). The operating life of the project is currently estimated to be 23 years.

The project is expected to generate approximately 186.9 million tonnes (Mt) of waste rock, 16 Mt of overburden, and 73.5 Mt of tailings. Two types of waste rock will be produced: metal-leaching or potentially acid generating waste rock (estimated at 80 Mt) and non-leaching "Low Risk" waste rock (106.9 Mt). Stockpile 001, located north of the open pit, will be designated for a portion of the "Low Risk" waste rock, about 60 Mt. Stockpile 002, situated northeast of the pit, will receive the remaining "Low Risk" waste rock (46.9 Mt), all of the potentially metal-leaching or acid generating waste rock (80 Mt), and a portion of the filtered tailings (41 Mt). The remaining tailings, estimated as 32.5 Mt, will be disposed as paste backfill once the underground mine is operational. Overburden will be managed in Stockpile 004 and Stockpile 005, located south of the pit.

The development of the open pit is planned in two phases. Phase 1 will be from 2029 to 2034, while Phase 2 is scheduled for 2034 to 2045. Underground mining operations will run from 2031 to 2051. Before the commencement of Phase 1, a cutoff dyke (Stage 1) will be constructed in Lake 001 together with a diversion ditch. This will allow for the pit development within the western portion of Lake 001 and continued discharge from Lake 001 to Lake 007, around the open pit. A second cutoff dyke and ditch, located to the east will be required for the Phase 2 pit development. The dyke and diversion ditch will maintain the water level of Lake 001 at the current elevation of approximately 373 meters above sea level (masl) and redirect water around the northern perimeter of the pit. The general site layout is provided in **Appendix A**.

2. Scope

This document forms the design criteria and design basis for tailings management, the stockpiles, the cut-off dyke, and the diversion ditch, geotechnical aspects, water management and hydrogeology (in the stockpiles area) for the Shaakichiuwaanaan Mining Project.

3. Units

The following units are used (the study is carried out using metric units):

- Distance – meter (m)
- Elevation – metres above mean sea level (masl)
- Volume – cubic meter (m^3)
- Pressure – kilo pascal (kPa)
- Unit Weight – kilo pascal (kPa)
- Temperature – Celsius (^oC)
- Time – seconds (sec), years (yr)

4. Site Geology

The Shaakichiuwaanaan Property, located in the Archean Superior Province of the Canadian Shield, lies in the La Grande sub-province, near the Opinaca sub-province, and is renowned for its robust exploration potential for commodities, including base metals, precious metals, and lithium (PEA, 2024). The La Grande sub-province, a geological marvel, showcases a volcano-plutonic structure aligned with the WemindjiCaniapiscau corridor. It comprises two primary domains, the Eastmain River Belt and the La Grande River Belt, each with its unique geological features (PEA,2024).

The Property is situated within the La Grande River Belt, characterized by volcanic and sedimentary rocks, including two ancient volcanic sequences and interstratified metasediments (PEA, 2024). The La Grande sub-province is bordered to the north by the plutonic Bienville sub-province and to the south by the metasedimentary and plutonic Opinaca sub-province (Lucas & St-Onge, 1998; Houle, 2004; Percival, 2007). The La Grande and Opinaca sub-provinces contain many of Québec's known spodumene pegmatite occurrences (PEA, 2024).

Metamorphism in the region varies from greenschist facies at the center to amphibolite facies at the edges (Card, 1986; Houle, 2004). Structural trends shift from east-west in the southwest to northeast-southwest in the north, primarily formed between 2700 and 2680 Ma (Percival et al., 2012). The area is not just about potential; it is about significant discoveries that have already been made. It contains significant nickel-copper deposits associated with komatiitic flows, ultramafic intrusions, and notable spodumene pegmatite occurrences (PEA, 2024).

5. Design Criteria and Design Basis

The design criteria and basis summary for the stockpile and the dams are provided in **Table 5-1**.

Table 5-1: Design Criteria and Basis Summary

Item	Design Criteria and Basis	Reference
5.1 Applicable Legislation, Regulations and Best Practice Guidelines		
5.1.1 Provincial Standards	Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP) ○ The Provincial Directive 019 on the Mining Industry Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP) ○ Dam Safety Regulation ○ Dam Safety Act Ministère des Ressources Naturelles et des Forêts (MRNF) ○ Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec -	MDDEP, 2012 s-3.1.01, r. 1 - Dam Safety Regulation S-3.1.01 - Dam Safety Act MRNF, 2024

Item	Design Criteria and Basis	Reference
5.1.2 Canadian Dam Association Guidelines	Canadian Dam Association (CDA), Dam Safety Guidelines	CDA, 2013
	CDA Technical Bulletins	
	○ Inundation, Consequences, and Classification for Dam Safety	CDA, 2007a
	○ Surveillance of Dam Facilities	CDA, 2007b
	○ Flow Control Equipment for Dam Safety	CDA, 2007c
	○ Dam Safety Analysis and Assessment	CDA, 2007d
	○ Hydrotechnical Considerations for Dam Safety	CDA, 2007e
	○ Seismic Hazard Considerations for Dam Safety	CDA, 2007f
	○ Geotechnical Considerations for Dam Safety	CDA, 2007g
	○ Dam Safety Reviews	CDA, 2016
	○ Application of Dam Safety Guidelines to Mining Dams	CDA, 2019
	○ Tailings Dam Breach Analysis	CDA, 2021
	○ Environmental Consequence Classification	CDA, 2023
	CDA Guidelines	
	○ Public Safety Around Dams	CDA, 2011
5.1.3 Others	Global Industry Standard on Tailings Management	GISTM, 2020
	Federal Metal and Diamond Mines Effluent Regulation	MDMER, 2024
	A Guide to the Management of Tailings Facilities	MAC, 2021a
	USBR Design of Small Dams	USBR, 1987
	USBR Design Standards No. 13: Embankment Dams	USBR, 2021
	USBR Design Standards No. 14: Appurtenant Structures for Dams (Spillways and Outlet Works) Design Standard	USBR, 2022
	USBR Design of Stilling Basins and Energy Dissipators	USBR, 1984
	FEMA Technical Manual: Overtopping Protection for Dams	FEMA, 2014
	USACE Hydraulic Design of Flood Control Channels (EM 1110-2-1601)	USACE, 1994
	ICOLD Bulletin 157 Small Dams, Design, Surveillance and Rehabilitation	ICOLD 2016 Hawley and Cunning, 2017.

Item	Design Criteria and Basis	Reference
	Guidelines for Mine Waste Dump and Stockpile Design.	

5.2 Mine Description

5.2.1 Name of Mine	Shaakichiuwaanaan	
5.2.2 Facility location	Category III lands of the Eeyou Istchee Cree Territory, James Bay, Quebec.	See Appendix A
5.2.3 Reserves	Total Ore: 87.2 Mt Average LiO_2% grade:1.26 Tonnes of concentrate @5.5% LiO2: 13.7 Mt	Project Charter (GMining)
5.2.4 Projected Mine Life	23 years	Project Charter (GMining)
5.2.5 Mineralogy of the Main Ore Body	Spodumene ($LiAlSi_2O_6$)	PEA (2024)
5.2.6 Type of Mine	Open pit and underground	PEA (2024)
5.2.7 Process and Reagents	Ferrosilicon 270 (DMS) Magnafloc 10 (Thickening and dewatering)	PEA (2024)
5.2.8 Treatment Processes	Fresh ferrosilicon (“FeSi”) will be added to a mixing hopper, which will then be fed continuously via screw feeders to each DMS’s respective medium header box. Flocculant mixing will be completed in a designated area within the plant with a dedicated sump pump that recycles spillage to the thickener feed tank.	PEA (2024)
5.2.9 Processing Capacity	Stage 1: 2.5 Mtpa DMS Plant Stage 2: 5.0 Mtpa DMS Plant dual train	Project Charter (GMining)
5.2.10 Mine Waste Production	Waste Rock: 186.9 Mt Overburden: 16 Mt Tailings: 73.5 Mt Backfill: TBD	Project Charter (GMining)
5.2.11 Backfill	32.5 Mt of tailings will be used for underground backfill	Assumed by AtkinsRealis

Item	Design Criteria and Basis	Reference
5.2.12 Coordinate Reference System	UTM-NAD83-ZONE18	BBA, 2024
5.2.13 Topographical Surveys	LiDAR and orthophoto survey in August 2022. A LiDAR and orthophoto survey, and heliborne magnetic and radiometric survey have also been completed in 2024 over the JBN-57 claim block.	NI 43-101 Technical Report, PEA

5.3 Climate Conditions and Meteorology

5.3.1 Climate Classification	Sub-arctic climate, Dfc in terms of the Köppen climate classification	
5.3.2 Weather Station(s)	Environment and Climate Change Canada (ECCC) Stations near site are: ○ La Grande IV station (7093GJ5), La Grande IV A station (7093GJ3) ○ La Grande 4 A station (7093714) ○ Nitchequon station (7095480)	AtkinsRéalis, 2025
5.3.3 Temperature	Average annual temperature is -3.1 degrees Celsius	AtkinsRéalis, 2025
5.3.4 Precipitation	Average annual precipitation is 772 mm which includes 518 mm of rainfall and 254 cm of snowfall	AtkinsRéalis, 2025
5.3.5 Evapotranspiration	Average annual lake evaporation is 421 mm and maximum evapotranspiration under average conditions is 390 mm	AtkinsRéalis, 2025
5.3.6 Wind	The predominant wind direction is from the southwest (see Figure 6-1 for wind roses)	Weather Spark, 2020
5.3.7 Climate Change Effect	Climate change increase factor of 18% for extreme precipitation Climate change increase factor of 5% for extreme snow cover Climate change increase factor of 8% for annual and monthly precipitation	AtkinsRéalis, 2025

5.4 Tailings Management

Item	Design Criteria and Basis	Reference
Description	**Tailings**	
5.4.1 Tailings Production	73.5 Mt	Project Charter (GMining)
5.4.2 Particle Size Distribution	TBD	TBD
5.4.3 Specific Gravity	~2.75	Primero, e-mail correspondence 5 December 2024
5.4.4 Filtered tailings moisture content	12-14% (M_{water}/M_{total})	Primero e-mail correspondence 5 December 2024 Note that the optimum moisture content must be determined
5.4.5 Plasticity	Non-plastic	Assumed by AtkinsRealis
5.4.6 Insitu dry density (compacted)	1.70 t/m³	Selected by AtkinsRealis, based on an estimated maximum dry density of 1.8 t/m^3
5.4.7 Waste Profile Definition	TBD for tailings	TBD
5.4.8 Consolidation Properties	No data	N/A
5.4.9 Hydraulic Conductivity and Anisotropy of Fresh and Compacted Tailings	TBD	TBD
5.4.10 Tailings Deposition	Filtered tailings are transported with mine trucks from the concentrator to the final disposal area or the paste backfill plant with a front-end loader.	PEA (2024)

5.5 Stockpiles

5.5.1 Stockpile 001		
5.5.1.1 Mine Waste	60 Mt (Low-Risk waste rock)	Designed by AtkinsRealis

Item	Design Criteria and Basis	Reference
5.5.1.2 Hazard Classification	Moderate hazard (WHC III)	Preliminary, based on Hawley and Cunning, 2017
5.5.1.3 Base Lining	Not required	Selected by AtkinsRealis, based on geochemical characterization of waste rock
5.5.1.4 Stockpile Geometry	Bench Height: 5 to 10m Intermediate bench width: TBD Overall slope: 3 (Horizontal): 1 (Vertical)	GMining (meeting discussions 14 January 2025) -Selected by AtkinsRealis
5.5.1.5 Maximum Height	90 m (measured from the max elevation to the downstream toe)	Designed by AtkinsRealis
5.5.1.6 Average in-situ dry density	Waste rock: 2 t/m^3	Selected by AtkinsRealis
5.5.1.7 Water Management	Runoff collected by perimeter ditches at the toe of the stockpile and directed to the stockpile pond	Selected by AtkinsRealis
5.5.1.8 Minimum offset from toe of stockpile to an existing water body	60 m	MDDEP, 2012
5.5.2 Stockpile 002		
5.5.2.1 Mine Waste	80 Mt of waste rock (Acid Generating) 47 Mt of waste rock (Low Risk) 42.5 Mt of tailings	AtkinsRealis, the 80 Mt total of potentially leaching or acid-generating waste rock is based on discussions with Vision and Patriot Battery Metals on 7 January 2025
5.5.2.2 Hazard Classification	Waste Rock - Moderate hazard (WHC III)- Tailings stack - TBD-	Preliminary, based on Hawley and Cunning, 2017
5.5.2.3 Base Lining (Only in Acid generating rock and tailings portion)	Single liner: 1.5 mm HDPE Liner Cushion Layer: 1m Overdrainage System: 1m	PEA (2024) PEA (2024)
5.5.2.4 Stockpile Geometry	Bench Height: 5 to 10 m Bench width: TBD	GMining (meeting discussions 14 January 2025)

Item	Design Criteria and Basis	Reference
	Overall slope: 3 (Horizontal): 1 (Vertical) for all types (i.e. tailings, and Waste rock)	Selected by AtkinsRealis
5.5.2.5 Maximum Height	125 m (measured from the max elevation to the downstream toe)	Designed by AtkinsRealis
5.5.2.6 Average in-situ dry density	Waste rock: 2 t/m^3 Tailings: 1.7 t/m^3	Selected by AtkinsRealis Selected by AtkinsRealis
5.5.2.7 Water Management	Runoff and seepage are directed to ditches at the toe of stockpile and directed to stockpile ponds	Selected by AtkinsRealis
5.5.2.8 Minimum offset from the toe of stockpile to an existing water body	60 m	MDDEP, 2012
5.5.3 Stockpile 004 and 005		
5.5.3.1 Mine Waste	Stockpile 004: 8 Mt of overburden Stockpile 005: 8 Mt of overburden	PEA (2024)
5.5.3.2 Hazard Classification	Low Hazard (WHC II)	Preliminary, based on Hawley and Cunning, 2017
5.5.3.3 Base Lining	None	
5.5.3.4 Stockpile Geometry	Bench Height: TBD Intermediate bench width: TBD Overall slope: 3 (Horizontal): 1 (Vertical)	TBD
5.5.3.5 Maximum Height	60 m (measured from the max elevation to the downstream toe)	Designed by AtkinsRealis
5.5.3.6 Average in-situ dry density	Overburden: 2 t/m^3	Selected by AtkinsRealis
5.5.3.7 Water Management	Runoff directed to ditches at the toe of the stockpile and then to open pit or store and seep	Selected by AtkinsRealis
5.6 Water Management for Stockpiles		
5.6.1 Perimeter drainage system		
5.6.1.1 Design Storm	1 in 100 yr 24-hr summer rainfall event	MDDEP, 2012

Item	Design Criteria and Basis	Reference
5.6.1.2 Design Flow Calculation Methodology	Rational Method	Selected by AtkinsRéalis
5.6.1.3 Runoff Coefficient	Considering high moisture conditions	Selected by AtkinsRéalis
5.6.1.4 Ditch Dimensions	Trapezoidal Side slope: 2 H: 1 V or flatter	Selected by AtkinsRéalis
5.6.1.5 Ditch Lining	Stockpile 001: Unlined Stockpile 002: 1.5 mm LLDPE geomembrane for run-off collection from leaching/PAG rockfill and tailings, unlined for run-off from Low Risk waste rock	based on preliminary results of the geochemical characterization by Vision Geochemistry as provided at meeting on Jan 21st, 2025.
5.6.1.6 Erosion protection	Stockpile 001: Rip-rap and non-woven needle punched geotextile Stockpile 002: 1.5 mm LLDPE geomembrane plus rip-rap and non-woven needle punched geotextile on the base of the ditch. Erosion protection as per Stockpile 001 ditches for run-off collection from Low Risk waste rock	Selected by AtkinsRéalis
5.6.1.7 Minimum Freeboard	0.8 m	USACE, 1994
5.6.2 Collection Ponds		
5.6.2.1 Dam Dimensions	Crest Width: 10 m Dam Slopes: 2.5 H: 1 V; protected by riprap	Selected by AtkinsRéalis
5.6.2.3 Seepage Control	1.5 mm LLDPE geomembrane on the upstream slope	Selected by AtkinsRéalis
5.6.2.4 Basin Lining	Stockpile 001 Pond: Unlined Stockpile 002 Ponds: 1.5 mm LLDPE geomembrane Stockpile 002 Pond for Low Risk Waste Rock: Unlined	based on preliminary results of the geochemical characterization by Vision Geochemistry as provided at meeting on Jan 21st, 2025.
5.6.2.5 Dam Hazard Classification	Pond 1 (Stockpile 001): TBD Pond 2 (Stockpile 002): TBD Pond 3 (Stockpile 002): TBD Pond 4 (Stockpile 002): TBD	Classification to be completed in accordance with CDA, 2013 and MELCCFP

5.6.2.6 Return Period for Design Seismic Event	1 in 2,475-year event (minimum)	MDDEP, 2012. To be reviewed following CDA (2013) hazard classification
5.6.2.7 Inflow Design Flood (IDF)	Probable Maximum Flood (PMF) based on Probable Maximum Precipitation (PMP). The more critical one of the spring or summer PMF	MDDEP, 2012
5.6.2.8 Environmental Design Flood (EDF)	Stockpile 001 Pond: • 1 in 25-year snowmelt in 13 days followed by a 100-year 24-hr rainfall event Stockpile 002 Ponds 2, 3 and 4: • Unlined section: 1 in 25-year snowmelt in 13 days followed by a 100-year 24-hr rainfall event • Lined section: 1 in 100-year snowmelt in 13 days followed by a 1, 000-year or 2,000-year 24-hr rainfall event (pending information from Vision Geochmistry)	MDDEP, 2012
5.6.2.9 Freeboard	Minimum Freeboard for EDF: • For Stockpile 001 ponds and the ponds containing runoff from unlined section of the stockpile 002: 1.0 m • For the pond(s) containing runoff from the lined portion section of the stockpile 002: 1.5 m. Minimum Freeboard for IDF: The greater of 0.9 m or the sum of the setup and runup generated by wind.	MDDEP, 2012 USBR, 2021
5.6.2.10 Spillway Geometry	TBD	TBD

5.7 Lake 001 Cut-off Dykes

5.7.1 Dyke Dimensions	Crest Width Upstream berm: TBD Crest Width Downstream berm: TBD Minimum Dam crest width: 10 m Berm Slopes (upstream and downstream): TBD	TBD
5.7.2 Dyke Height	TBD	
5.7.2 Dyke Hazard Classification	Extreme	Preliminary, selected by AtkinsRealis, based on CDA, 2013
5.7.3 Dam Class	Class B	Quebec Dam Safety Act, chapter S-3.1.01, r. 1
5.7.4 Return Period for Design Seismic Event	1/10,000 or Maximum Credible Earthquake event for “Extreme” Dam Classification	CDA, 2013 and GISTM, 2020
5.7.5 Inflow Design Flood (IDF)	Probable Maximum Flood (PMF) based on Probable Maximum Precipitation (PMP). The more critical one of the spring or summer PMF	CDA, 2013
5.7.6 Minimum Freeboard	Minimum Freeboard for IDF: The greater of 0.9 m or the sum of the setup and runup generated by wind.	CDA, 2013
5.7.7 Erosion Protection	Riprap, D_{50} = TBD, Thickness: $2xD_{50}$	Selected by AtkinsRealis
5.7.8 Seepage management	Sheet piles	Selected by AtkinsRealis

5.8 Diversion Channel for Lake 001

5.8.1 Average Longitudinal Gradient	Minimum longitudinal gradient: TBD	TBD
5.8.2 Design Flow	Probable Maximum Flood (PMF) based on Probable Maximum Precipitation (PMP) The more critical one of the spring or summer PMF	Selected by AtkinsRéalis

5.8.3 Channel Geometry	Trapezoidal	Selected by AtkinsRéalis
5.8.4 Dimensions	Side slope: 2 H: 1 V, steeper in blasted rock sections	Selected by AtkinsRéalis
5.8.5 Erosion Protection	RipRap lining for ditch in overburden	Selected by AtkinsRealis
5.8.6 Minimum Freeboard	0.8 m	USACE, 1994

5.9 Seismic Design Criteria

5.9.1 Site Class	Class D	NBCC, 2020
5.9.2 Peak Ground Acceleration (PGA)	0.0642g for 1 in 2,475-year event (HPC: High) 0.1g for 1 in a 10,000-year event (HPC: Extreme)	NBCC, 2020 Extrapolated by AtkinsRealis from NBCC, 2020. A site-specific hazard assessment is required for more accuracy

5.10 Geotechnical Design Basis

5.10.1 Seismic Liquefaction		
5.10.1.1 Susceptibility	Saturated and near-saturated (S_r > 80%) sand-like soils/tailings.	Selected by AtkinsRéalis
5.10.1.2 CRR for Sand-like material	Based on (S) SPT method: Boulanger and Idriss (2014) simplified method	Boulanger and Idriss, 2014
5.10.1.3 Fines Content Correction	Applied only to foundation soil, not to tailings	James, 2009
5.10.1.4 Factor of Safety against Liquefaction of Tailings and Sand-Like Soils (FOS_{Liq} = CRR/CSR)	FOS_{liq} < 1.0: Liquefiable FOS_{liq} between 1.0 to 1.3: Potentially liquefiable FOS_{liq} > 1.3: Not liquefiable	Selected by AtkinsRéalis
5.10.2 Cyclic softening		
5.10.2.1 Susceptibility	Saturated and near-saturated (S_r > 80%) clay-like soils meet the following plasticity index (PI) limits:	Idriss and Boulanger, 2008

	• Clays: PI < 7 • Silts: PI < 12	Bray and Sancio, 2006
5.10.2.2 CRR for Clay-like material	Based on the simplified method: Idriss and Boulanger (2008) simplified method.	Idriss and Boulanger, 2008
5.10.2.3 Factor of Safety against Cyclic Softening of Clay-Like Soils (FOS_{cs} = CRR/CSR)	FOS_{cs} < 1.0: Susceptible to cyclic softening	Selected by AtkinsRéalis
5.11 Geotechnical Parameters	To be determined after analysis of BH information and laboratory data	TBD
5.12 Loading and Seepage Conditions		
5.12.1 Seepage Model Limits / Boundary Conditions (Lake 001 Cut-off Dam)	Upstream Boundary Condition: Lake 007 EL 373m Downstream Boundary Condition: towards Open Pit	Selected by AtkinsRéalis
5.12.2 Seepage Model Material Parameters	To be determined after analyzing laboratory and field investigation data	TBD

5.13 Geotechnical Design Criteria

5.13.1 Limit Equilibrium Analysis – Minimum Factors of Safety (Lake 001 Cut-off Dyke)

MDDEP, 2012

Zone	Loading Conditions	Minimum Factor of Safety
Upstream and downstream	Slope stability at the end of each construction phase (short-term)	1.3 to 1.5
Upstream and downstream	Slope stability in the presence of the project flood (short-term)	1.3
Upstream and downstream	Slope stability under stationary conditions (long-term)	1.5
Upstream	Quick drain	1.3
Upstream and downstream	Pseudo-static or dynamic analysis for seismic loading, with filled basin and reduced properties if liquefaction possible	1.1
	(post-seismic value)	(1.3)
Central	Horizontal slippage of the dike under static loading	1.5
Central	Horizontal slippage of dike under liquefaction pressures	1.3
Central	Bearing capacity of soil under fill	1.5

5.13.2 Limit Equilibrium Analysis – Minimum Factors of Safety (Stockpiles)

Loading Conditions	Minimum Factor of Safety
Dump/Soil Surface (short-term)	1.0
Dump/Soil Surface (long-term)	1.2
Overall global stability short-term (static), post-construction	1.3 to 1.5
Overall global stability long-term (static)	1.5
Pseudo-static or dynamic analysis for seismic loading	1.1 to 1.3

BCMWRPRC, 1991a
Hawley and Cunning, 2017

5.13.3 Piezometric Conditions	• Geotechnical analysis for the current condition is based on readings from the existing piezometers/BHs water level • Geotechnical analysis for ultimate/closure condition is based on seepage analysis results.	Selected by AtkinsRealis

5.14 Closure

5.14.1 Objectives	• Eliminate unacceptable health risks and ensure the safety of persons. • 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 follow-up. • Return the site to a visually acceptable condition. • To remediate the infrastructure site (excluding tailings and tailings accumulation areas) in a state compatible with future use	Based on MRNF, 2024

5.14.2 Minimum safety factor targeted for stability analyses under closure conditions

Loading Condition	Minimum Factors of Safety			Minimum Factors of Safety for this Project (Closure)
	MDDEP (2012)	CDA (2019)	MRNF(2024)	
Static – End of construction	1.3 to 1.5	<1.3	1.3 to 1.5	1,5
Static – Long term (with NPS)	1.5	1.5	1.5	1.5
Static – Design Flood	1.3	-	1.3 to 1.5	1.3
Pseudo-static	1.1	1.0	1.1	1.1
Post-seismic	1.3	1.2	1.3	1.3
Horizontal Sliding (static) (1)	1.5	-	1.5	1.5
Horizontal Sliding (post-liquéfaction) (1)	1.3	-	1.3	1.3
Note : This condition will be included in the stability analysis (composite sliding surfaces)				

MDDEP, 2012
CDA, 2019
MRNF, 2024

5.13.3 Earthquake Loading	The return interval for the earthquake loading for closure design will be either based on the hazard consequence classification as per CDA (2013) for the active and passive closure phases or as prescribed by MDDEP (2012) which is 1 in 2,475 years for acid-generating residues (the greater loading shall apply).	CDA, 2013

5.13.4 Hydrotechnical Closure Criteria	MDDEP (2012) recommends a design flood with a minimum return period of 100 years, for drainage systems with structures without water accumulation, on and around mine sites. Therefore, the following design criteria were adopted in the present study: • The ditches or culverts design flood is generated by a summer-fall rainfall with a 100-yr return period. This return period should also be used in the Maintenance Program for the site. • The transformation of rainfall into runoff is based on a conservative runoff coefficient of 1.0. (recommended by MTQ methodology)	MDDEP, 2012
5.13.4 Vegetative Cover	There are generally no formal criteria for the evaluation of the vegetation management. Vegetation success on the different infrastructures and cover systems contribute to their integrity as it protects the top layers against erosion and contribute to the integration of the site to its surroundings. However, the roots should not penetrate so deeply as to affect the cover system, such as a geomembrane liner. Also, vegetation growth should not lead to blocking the ditches of the site.	
5.13.5 Cover System	TBD – Cover system will be selected based on the waste profile and can include granular and geosynthetic constituents	TBD
Notes:		

6. Supplemental Information

Table 6-1: Minimum Required Freeboard (USBR, 1987)

Fetch Length [km]	Minimum Freeboard[m]
<1.6	0.9
1.6	1.2
4.0	1.5
8.0	1.8
16.1	2.1

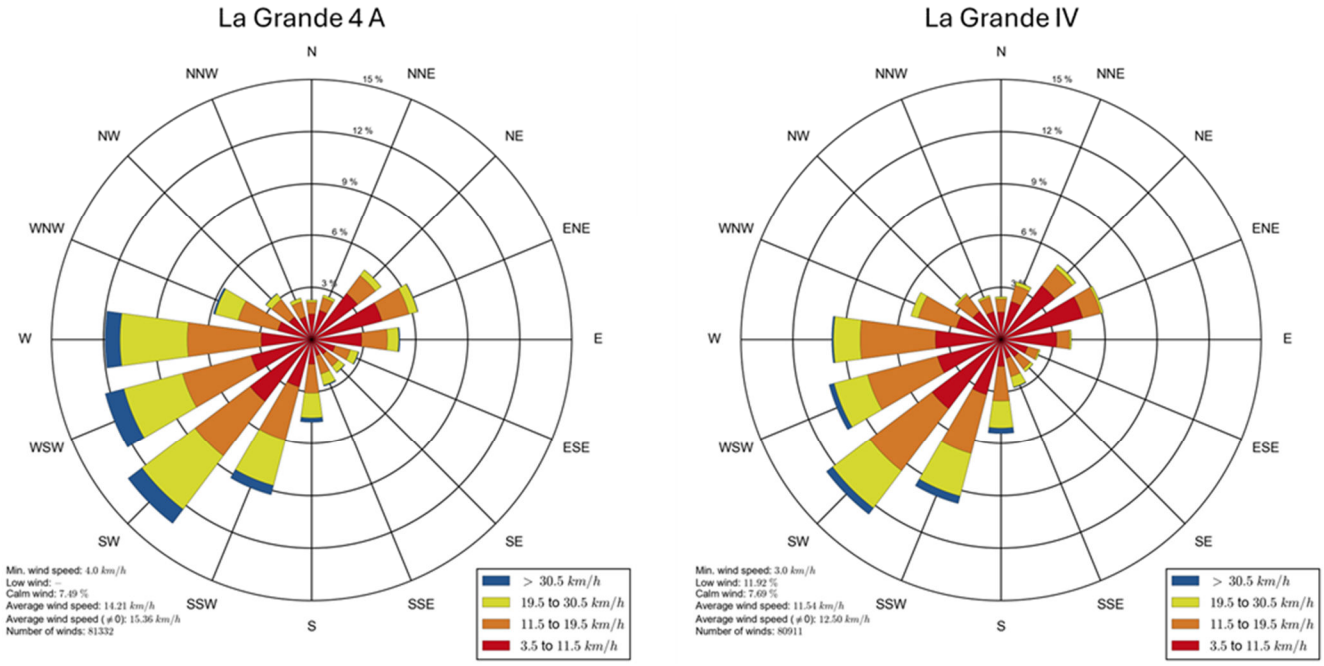


Figure 6-1 Wind Rose plots for La Grande 4 A and La Grande IV stations (established based on wind data from 2014 to 2024) (Provided by Client)

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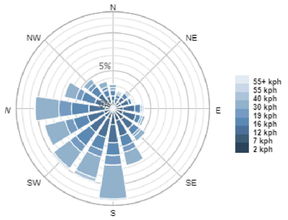
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Appendices

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(MM-NT-00ER_01 ANG)

Appendix A. Site Location and Layout

PAPER SIZE FORMAT : A1



ANNUAL WIND ROSE



569 000 E
570 000 E
571 000 E
572 000 E
573 000 E
574 000 E
566 000 E
567 000 E
568 000 E
5 934 000 N
5 933 000 N
5 932 000 N
5 931 000 N
5 930 000 N

EXISTING ACCES ROAD
STOCKPILE 002
DMS TAILINGS
DMS MIDDLINGS
PAG WASTE ROCK
MINE WASTE
ROCK DISPOSAL
GATE
CAMP
CONSTRUCTION PAD
BATCH CONCRETE PLANT
FIRST NATION
CULTURAL CENTER
EMERGENCY
HALL
PROCESS PLANT
EL. SUBSTATION
U/G MINE EROOM / COMPRESSED AIR
U/G VENTILATION INFRASTRUCTURES
PASTE PLANT
FUEL STATION
ADMIN / DRY
TRUCKSHOP
WAREHOUSE
U/G MINE LAYDOWN 20 000m2
WTP ACCESS ROAD & PIPING TRAIL
HG / LG MIXING STOCK PILE
U/G WASTE TRANSFER PAD
AGGREGATE PLANT
U/G MINE PORTAL / BOXCUT
WATER TREATMENT PLANT
MINE RUN OFF
SEDIMENTATION POND
ROM
PAD
UNDERGROUND MINE
(DESIGN TO BE COMPLETED)
BLASTING RADIUS 500m
BLASTING RADIUS 300m
UNDERGROUND RAMP
DIVERSION DITCH PHASE 1 & 2
PHASE 1 DAM
PHASE 2 OPEN PIT
PHASE 2 DAM
EMULSION PLANT
EXPLOSIVE MAG
PHASE 1 OPEN PIT
STOCKPILE 005
OVERBURDEN
STOCKPILE 004
OVERBURDEN

NOT FOR CONSTRUCTION
NE PAS UTILISER POUR
CONSTRUCTION

REV	DESCRIPTION	PAR BY	ING ENG	DATE
B1	REVISION GÉNÉRAL GENERAL REVISION	Y.B.	P.D.	25-01-22
B	ÉMIS POUR REVUE DE PROJET ISSUED FOR PROJECT REVIEW	Y.B.	P.D.	24-12-17
A	REVUE INTERNE INTERNAL REVIEW	Y.B.	P.D.	24-12-12

PATRIOT BATTERY METALS

Mining Services

CONCEPTEUR/DESIGN: P. DROZ	24-09-18
DESSINATEUR/DRAWN: Y. BERGER	24-08-01
VÉRIFIÉ PAR/CHECKED: P. DROZ	24-09-18
INGÉNIEUR/ENGINEER: P. DROZ	24-09-18
ÉCHELLE/SCALE: 1 : 10 000	DATE

PROJET/PROJECT:
SHAAKICHIUWAANAAN

PHASE:
ÉTUDE DE FAISABILITÉ
FEASIBILITY STUDY

TITRE:
PLAN DU SITE
ARRANGEMENT GÉNÉRAL
PLAN PRINCIPAL
VUE DE PLAN

TITLE
SITE LAYOUT
GENERAL ARRANGEMENT
MAIN PLAN
PLAN VIEW

DESSIN NUM./DRAWING NO.: CASN – A

001 – GE – GA – 0002 – B1
AREA DISC. TYPE SEQ. REV.

COORDINATE SYSTEM: NAD83(CSRS) / UTM zone 18N

SAVED ON: 25-01-22 2:05 PM BY Yannick Berger
PATH: Z:\ACTIVE_PROJECTS\PATRIOT BATTERY METAL\SHAAKICHIUWAANAAN (CASN-A) Feasibility Study\03-Drawing\001 General\GE General\CASN-A-001-GE-GA-0002.dwg

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ATKINSRÉALIS - Sensitive

Appendix C: POETS

Project No: 7535-0027
Client: Patriot Battery Metals
Project Name: Shaakichiuwaanaan

Prepared by: BBA
Approved by: BBA
Date: 2025-05-28

POETS List- Account & Sub Account

Selection # from 1 to 10 of the sub account only are required

Project Economics

#	Description	Notes
	Project Costs NPV - CAPEX & OPEX	
4	Fish habitat compensation	
	Sustaining Capital Cost: structures, equipment	
1	CAPEX	
	Environmental Compensation Cost	
	Post-Closure & Maintenance cost	
	Project costs estimation risk	
	Construction Time & Complexity (risk)	
	Private land acquisition (utilities)	
3	Phasing possibility - Cashflow (ROADS, DITCHES, BASINS)	
	Profitability / Taxes	
	Economic Risk	
2	OPEX	

Operational Consideration

#	Description	Notes
1	Distance from mill/pit	
	Distance from portal	
	TSF Elevation	
	TSF Capacity	
	Ease of access	
	Disposal option possible	
	Operational flexibility -middlings location	
	Allows for integrated site water management	
2	Expansion capability	
	Operational difficulties	
	Ease of construction and schedule	
	Closure	
	Impact of deposition schedule	
	Ore sterilisation (proven reserves area)	
3	Operational safety	
	Start-up complexity and process implementation / known technology	
	System simplicity	
	Reliability, availability & redundancy	
	Operability / flexibility	
	Impact of weather on operation	

Technical Consideration

#	Description	Notes
	Integration with topographic features - Pipelines	
	Presence of active geological faults	
	Technological risk	
	Feasibility of upstream raises	
2	water management complexity	
4	Embankment (earthwork) height	
	Availability of construction and reclamation materials	
	TSF Technical risks	
	Geotechnical stability (side slope, foundations)	
	Geotechnical stability (foundation)	
1	Water management infrastructure	
	Integration with topographic features - TL	
	Permeability of foundation	
	Proximity to rail station / Finished product network	
	Road geometry: alignment, slopes, cross section	
	Pad elevation and risk of inundation	
	Service network: streams, rivers, creeks crossing	
	Pad surface, earthworks	
	Proximity to existing public services, project connectivity	
	Electrical network: length, footprint	
	Gas network: length, footprint	
	Water supply and discharge networks / Electrical network: length, footprint	
3	Project schedule & Constructability	
	Project schedule - Equipment delivery	
	Project schedule - Construction	
	Constructability	
	Non-shutdown Works	
	Construction dependance with operation	
	Integrability of fine iron recovery plant	
	Critical testwork, investigations and scalability available	
	TSF dam requirement	
	TSF dam requirement – Height and Risk	
	Dam geotechnical stability related to foundation	
	Presence of active geological faults or other geological features	

Environmental Impacts

#	Description	Notes
	Exposure to prevailing winds	
1	STOCKPILE footprint	
	Groundwater, downstream water wells users	
	Surface/ground water aquifer communication	
	Flood plains	
	Wetlands	
	Downstream groundwater wells	
	Acoustics, noise and vibrations	
	Surface water quality / vulnerability	
	Streams and rivers impacted (hydrography)	
	Endangered species territories	
	Wildlife habitat safety and quality	
	Fishery and aquatic habitat safety and quality	
	Presence of sensitive or unique ecosystems	
	Hydrology and watershed management	
	Terrestrial flora and fauna	
	Surface water vulnerability	
	Air quality / Dust generation	
2	CO2 emissions	
	Acid rock drainage and metal leaching	
	Avian fauna	
	Amphibians and reptiles	
4	Fish presence	
	Permitting challenges	
	Environmental risks	
	Dam failure risk	
	Climate	
	Terrain & Soils	
	Vegetation	
	Wildfire	
	Sustainability, Durability & Greenhouse gases	
	Process water recovery	
	Ground water	
3	Wetlands/ Rivers / Streams	
	Surface water	
	Terrestrial habitat	
	Biodiversity	
	brown field vs green field	

Social Impacts

#	Description	Notes
	Soil, land use and land capability	
	Economic land-use (soil use, land capability)	
	Roads or other public facilities	
	Proximity and size of human habitation	
	Archeological and burial sites	
2	moose feeding areas and goose feeding areas	
1	Proximity to sensitive receptors (cabins and travel routes)	
	Impact on navigable waters	
	Economic benefit and employment	
	Worker / Community health & safety	
3	Archaeological potential	
	Site zonage	
	Income (direct/indirect)	
	Taxes (direct/indirect)	
	Regional Government Development	
	Government Expenditures	
	Labour Market Analysis	
	Population	
	Housing	
	Transportation & Traffic	
	Community Services	
	Land Tenure	
4	Fishing areas	
	Outdoor Recreation & Tourism	
	contaminated water spill and consequences to local population	
	Surface exploitation rights / permitting	
	Employment	
	archeology & tourism impact	
	Nuisances	
	Health and safety	
	Socioeconomic risk	
	Landscape / scenery	
	Archaeological potential	



APPENDIX

4-6 DIVERSION CHANNEL LAKE 01 TO LAKE 05 – DESIGN OPTIONS SUMMARY

AtkinsRéalis	TECHNICAL NOTE Shaakichiuwaanaan Project – Feasibility Study		
	Document Number: CASN-A-329-HY-MEM-0001 *AtkinsRéalis: 702381-5000-4HER-0002 PC*	**Rev.** C	**Date:** February 20, 2026

Document title: **DIVERSION CHANNEL LAKES 001 TO 005/ SUMMARY OF DESIGN OPTIONS**

Client: **PATRIOT BATTERY METALS INC.**

Project **SHAAKICHIUWAANAAN PROJECT – WATER AND RESIDUE MANAGEMENT**

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Senior Mine Tailings Engineer
#OIQ: 6076892

Approved by: Bruno Morrissette, P.Eng.
Project Manager
#OIQ: 125157

(F-40ER-0012-MMG-F)

Notice to the reader

This report is a translation of the English version of the report bearing the same number. In the event of any discrepancy between the two reports, the English version shall prevail.

This report was prepared and the work described herein was performed by AtkinsRéalis Canada Inc. (AtkinsRéalis) for the exclusive use of Patriot Battery Metals Inc., which participated in the development of the scope of work and understands its limitations. The methodology, results, conclusions, and recommendations contained in this report are based solely on the scope of work and are subject to the time and budget considerations described in the proposal and/or contract under which this report was published. Any use, reliance, or decision made by a third party based on this report is the sole responsibility of that third party. AtkinsRéalis disclaims any liability for damages that may be suffered or incurred by any third party as a result of the use, reliance, or any decision made based on this report.

The findings, conclusions, and recommendations contained in this report (i) have been prepared in a manner consistent with the level of skill normally exercised by professionals currently practicing in similar circumstances in the field, and (ii) reflect AtkinsRéalis' best judgment based on the information available at the time of preparation of this report. No other warranty, express or implied, is given with respect to the professional services provided to Patriot Battery Metals Inc. or the results, conclusions, and recommendations contained in this report. The results and conclusions contained in this report are valid only as of the date of the report and may be based, in part, on information provided by third parties. If certain information proves to be inaccurate, if new information is discovered, or if project parameters change, modifications to this report may be necessary.

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List of revisions

#	Prep.	Rev.	App.	Date	Pages	Comments
C	AM	PA	BM	2026-02-19	All	Issued for feasibility study

Patriot Battery Metals Inc.
Shaakichiuwaanaan Feasibility Study

Diversion Channel Lake 001 to 005
Design Options Summary

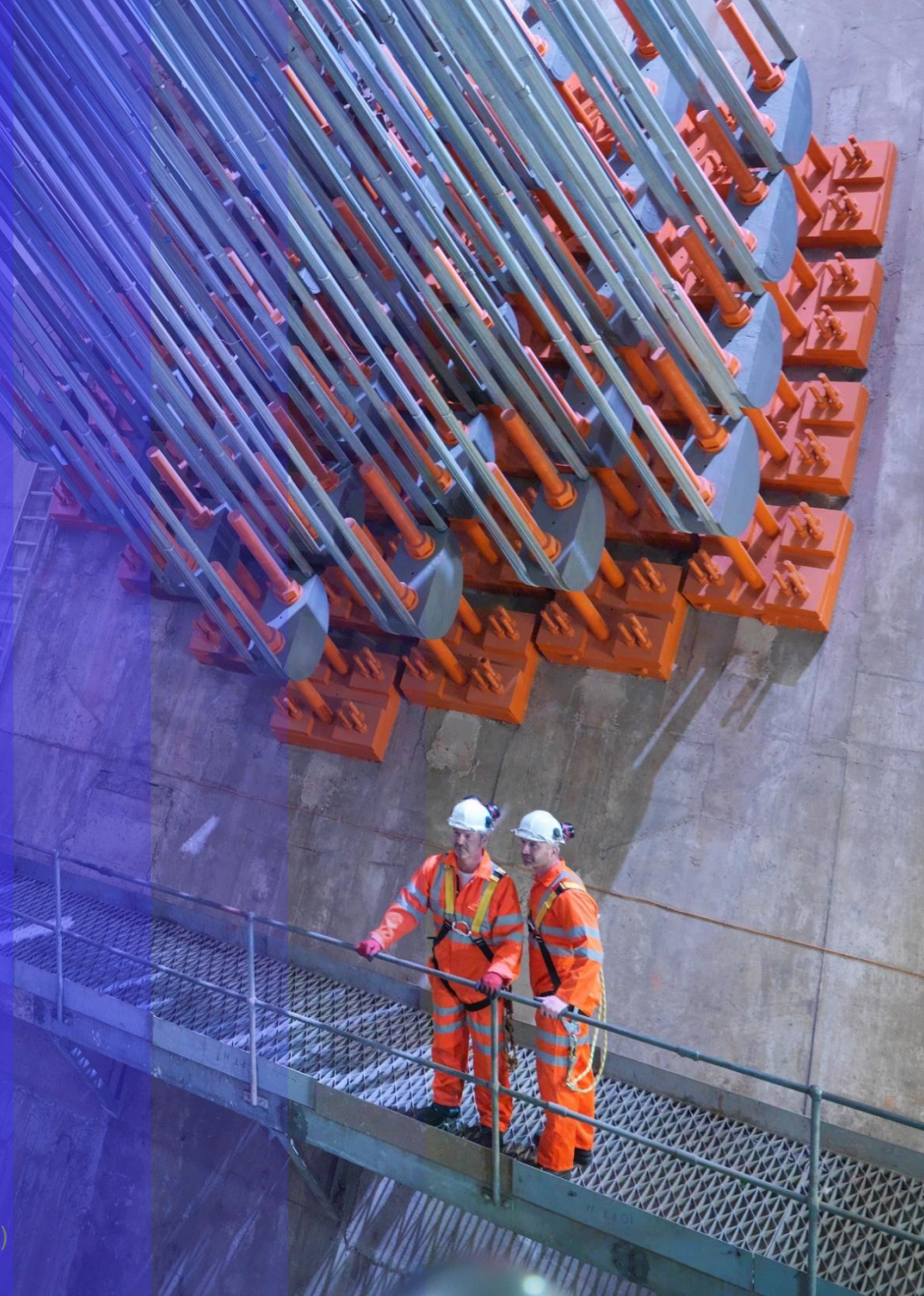
Prepared by: Abdolvahid Mohammadian, Ph.D., PEng
Holman Tellez, M.Sc., PEng.

Reviewed by: Philip Addis, PEng.

Document no.: CASN-A-329-HY-MEM-0001 rev.C
AtkinsRéalis Document no.: 702381-5000-4HER-0002 PC

Oct 01st, 2025

AtkinsRéalis



AtkinsRéalis - Sensitive / Sensible (FR)

Background

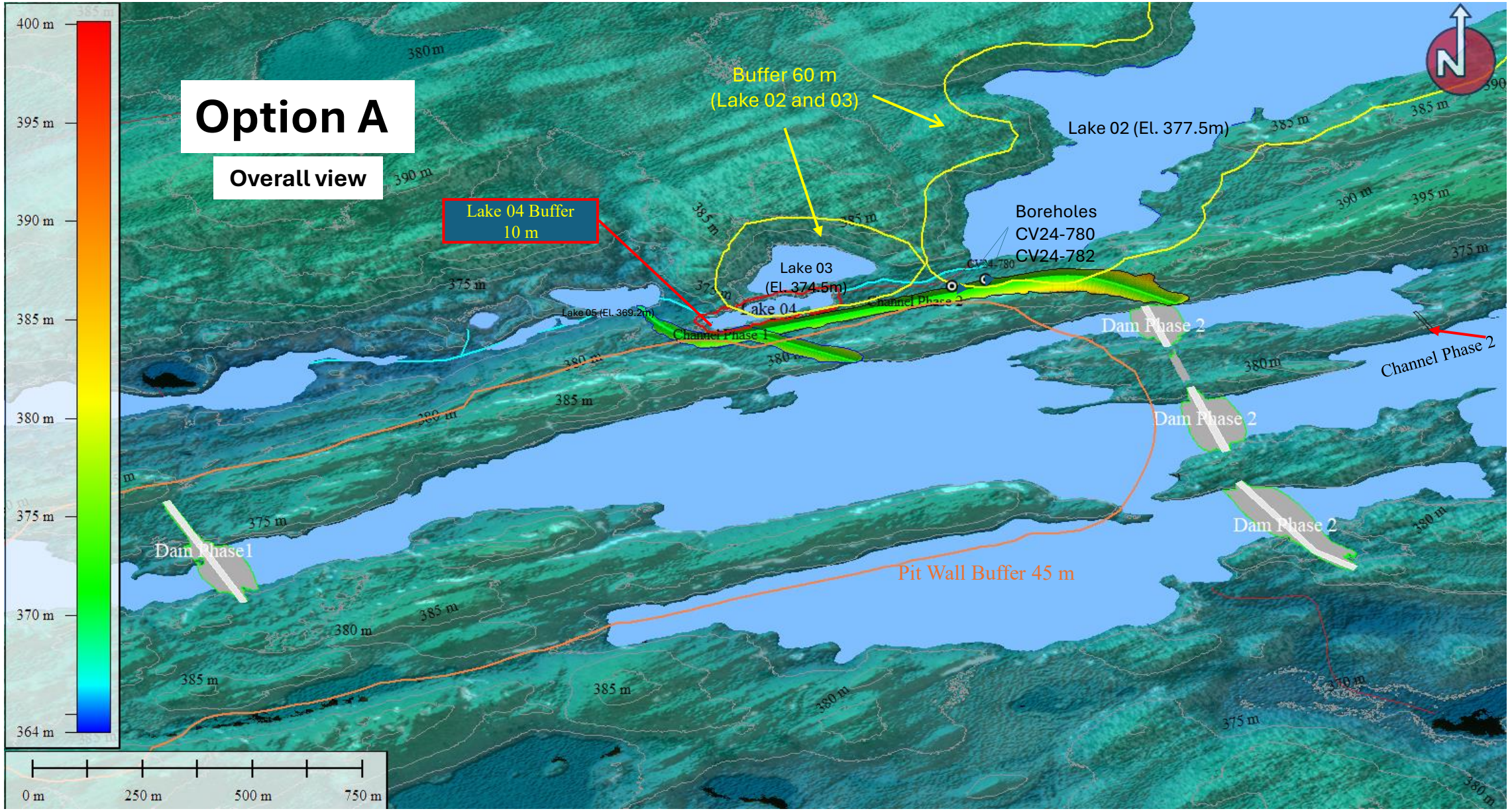
- Preferred design alternatives for FS diversion channel(s) evaluated
- Project Charter's *phased* approach for dams and diversion channel utilized in the evaluation (continuous attempts to reduce fish habitat impacts are within design work)
- Phase 1 dam and Phase 1 diversion channel first built; and then Phase 2 larger dam & diversion channel built several years later
- Consideration in evaluation is to include fish habitat/passage impacts, offset required from Open Pit for water infiltration (pillar), offset from existing lakes, construction & seasonal difficulty influences

Summary of Scenarios

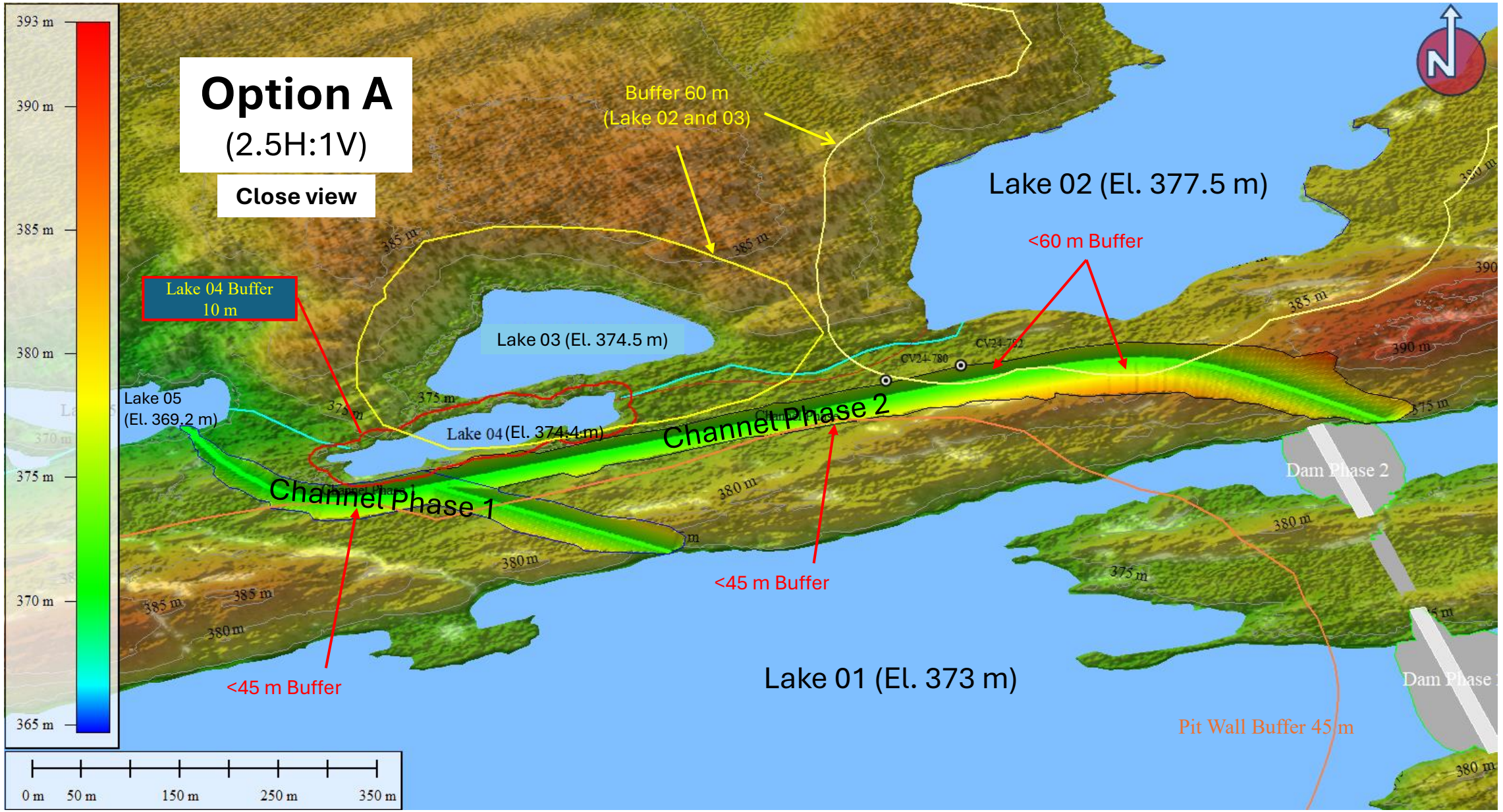
Diversion Channel Option Description	Channel Length (m)	Side Slope (Width/Height Ratio)
A - First Design Concept	793	2.5H/1.0V
B - Shortened: Phase 2 Channel (Through Lake 04)	738	2.5H/1.0V
C - Shortened Further: (Phase 1 Channel Also Through Lake 04)	726	2.5H/1V & 1.5H/1.0V
D - Respect Open-Pit /Lake Offset Distances & Reduce Fish Habitat/Passage Impact	870	1.0H/1.0V

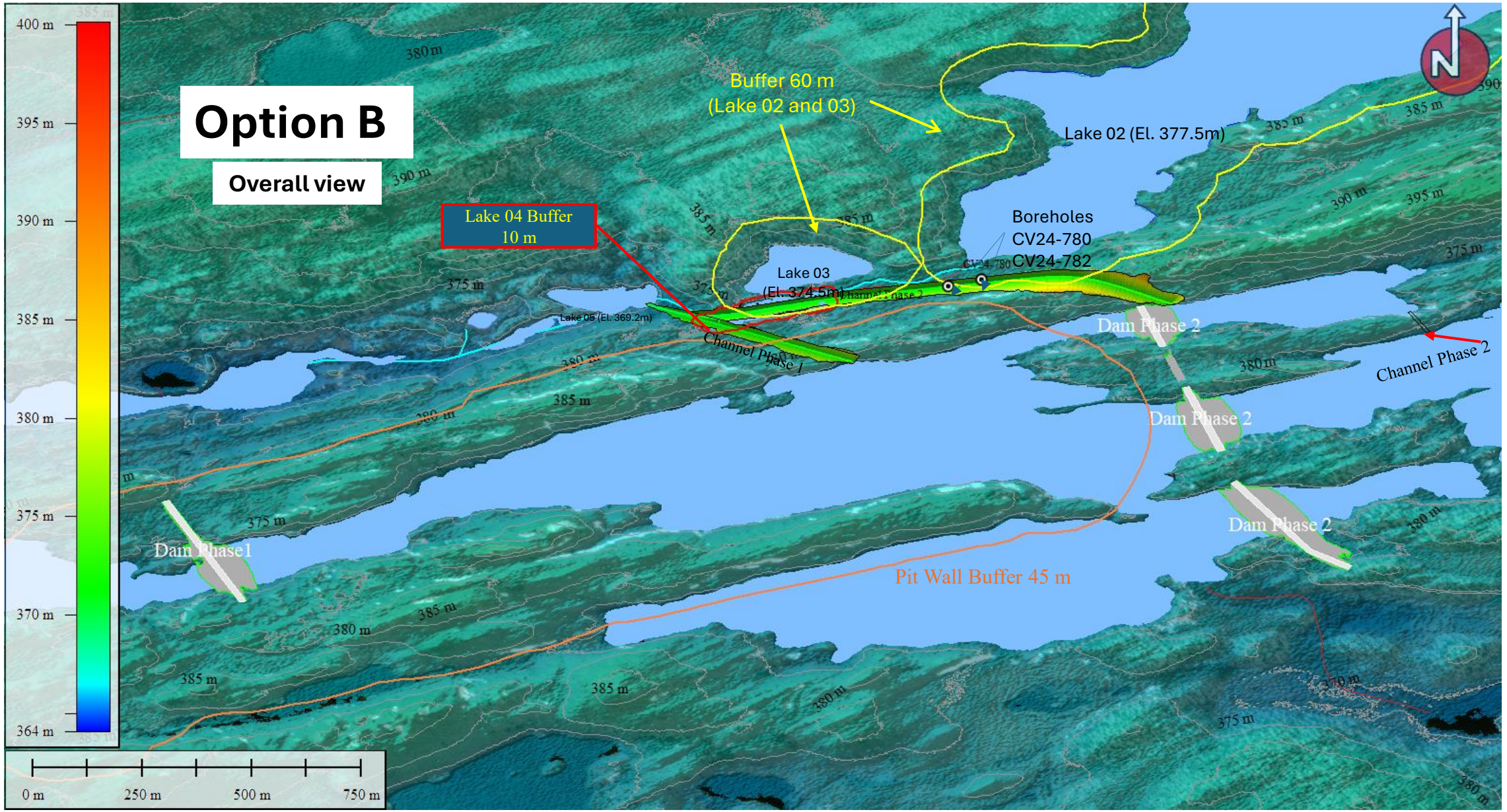
Additional Considerations

- Options A, B and C were initially considered; however, it was concluded afterwards that there would only be a single stage diversion (Option D – Phase 2 only).
- Additional hydraulic considerations were necessary for fish passage through the channel, such as creating pools and drops to replicate current outlet variations.
- Option D (selected), a single-phase channel aligned through the offset boundaries of the pit and the lakes.

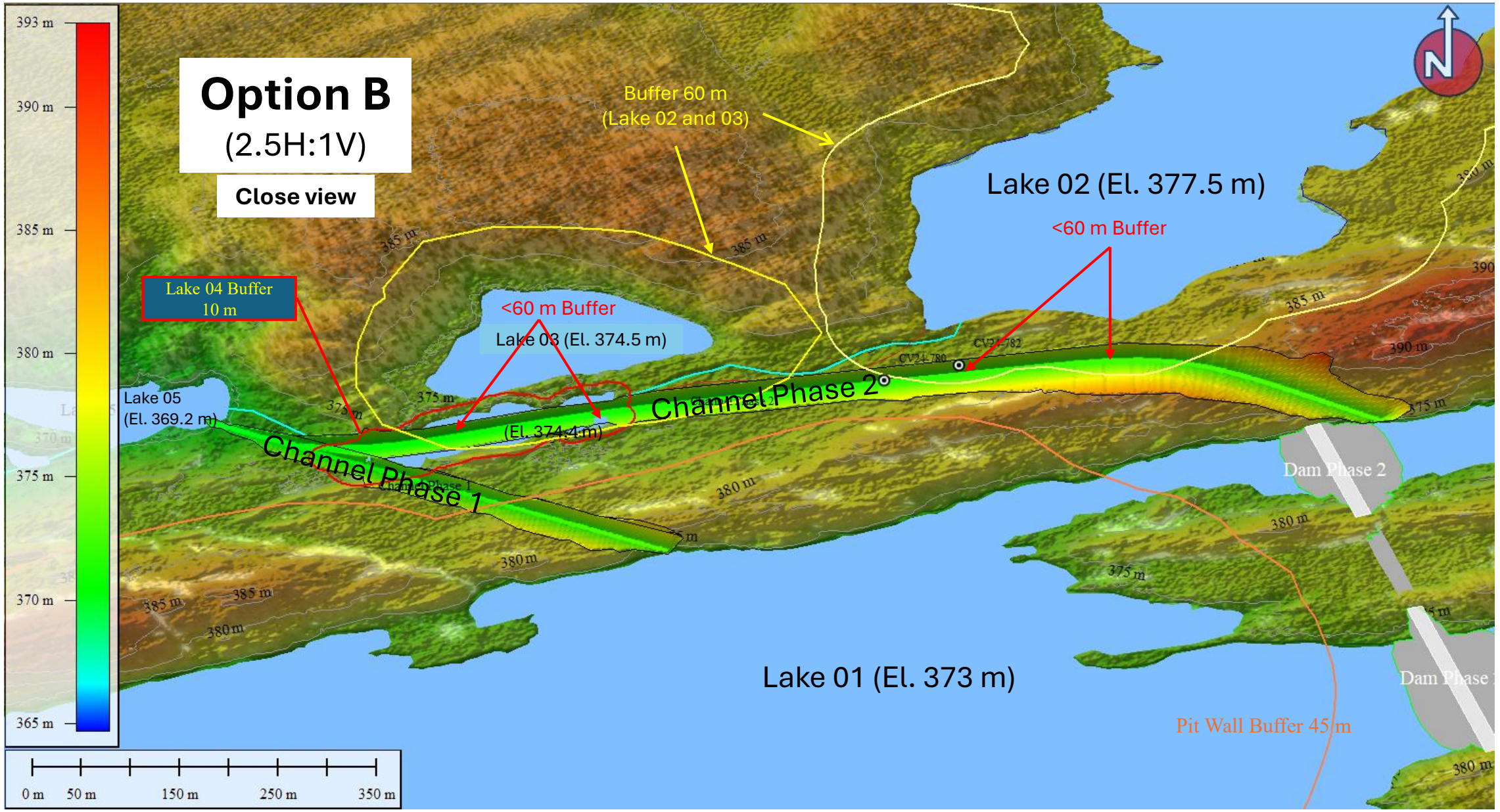


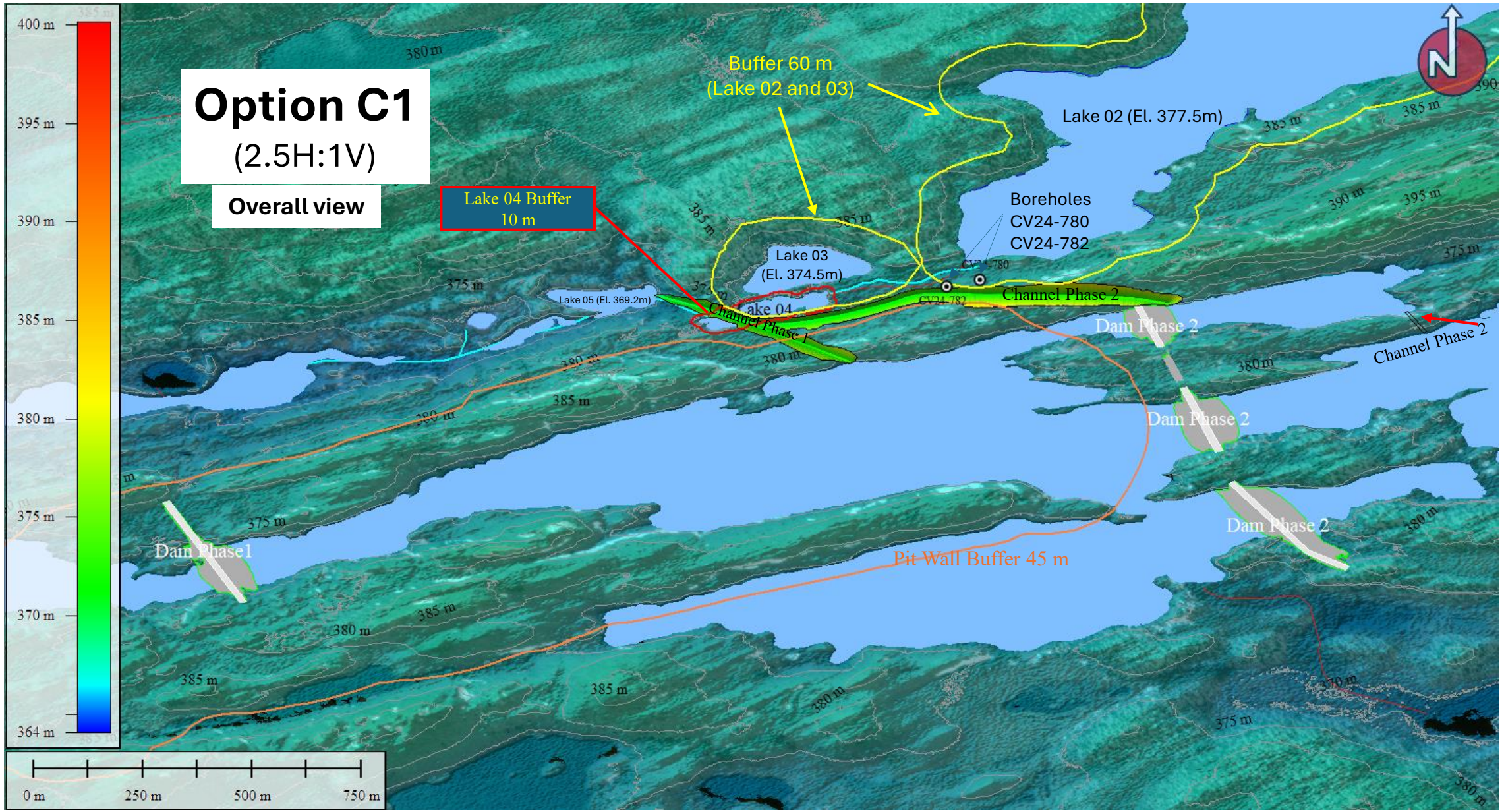
Water Level in the lakes (based on raster): Lake 02 = 377.5 m ; Lake 03 = 374.5 m ; Lake 04 = 374.4 m ; Lake 05 = 369.2 m


Option A
(2.5H:1V)
Close view
Buffer 60 m
(Lake 02 and 03)
Lake 02 (El. 377.5 m)
<60 m Buffer
Lake 04 Buffer
10 m
Lake 03 (El. 374.5 m)
Lake 05
(El. 369.2 m)
Lake 04 (El. 374.4 m)
Channel Phase 2
Channel Phase 1
<45 m Buffer
<45 m Buffer
Dam Phase 2
Lake 01 (El. 373 m)
Pit Wall Buffer 45 m
393 m
390 m
385 m
380 m
375 m
370 m
365 m
0 m
50 m
150 m
250 m
350 m
N



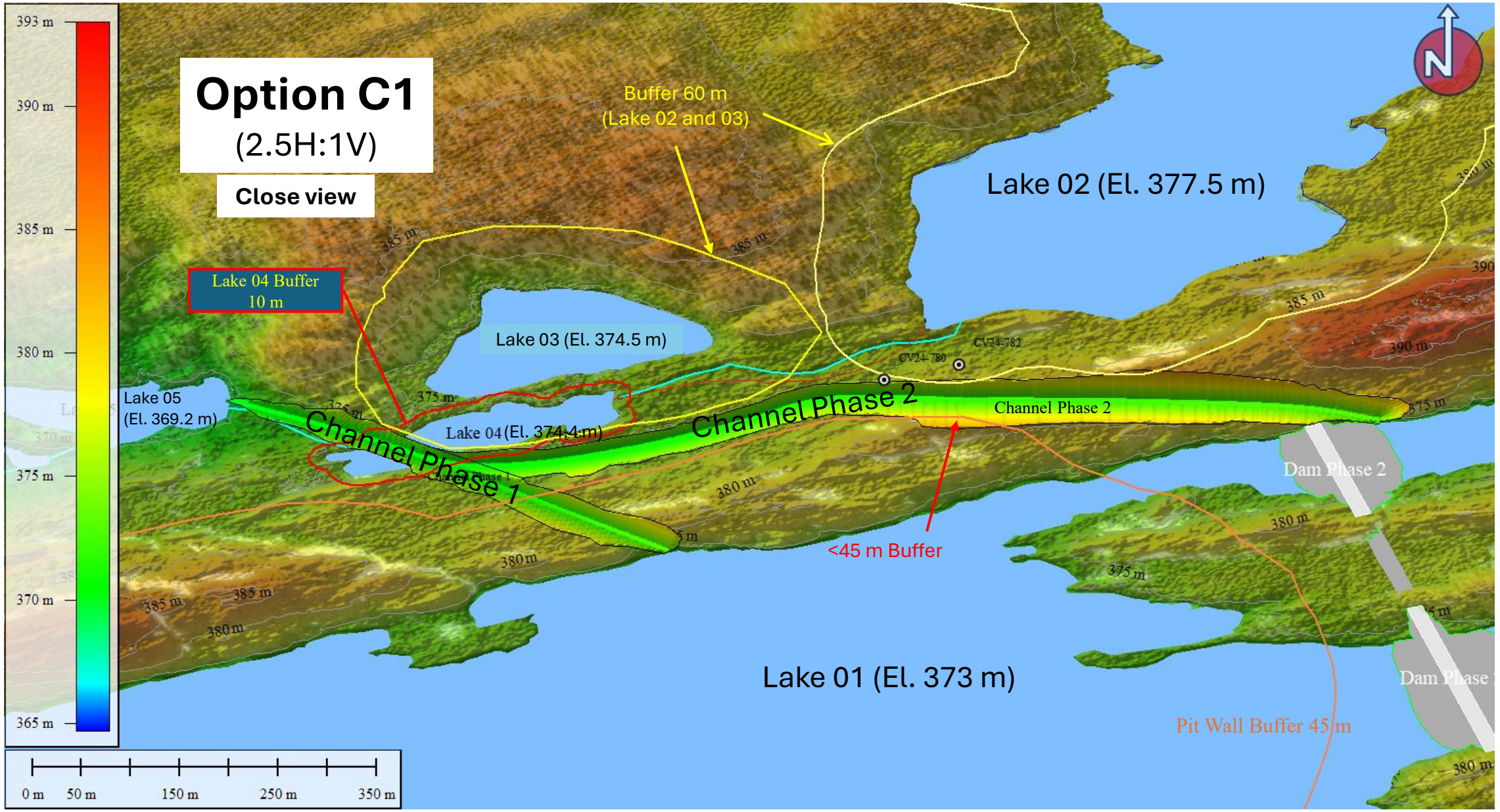
Water Level in the lakes (based on raster): Lake 02 = 377.5 m ; Lake 03 = 374.5 m ; Lake 04 = 374.4 m ; Lake 05 = 369.2 m

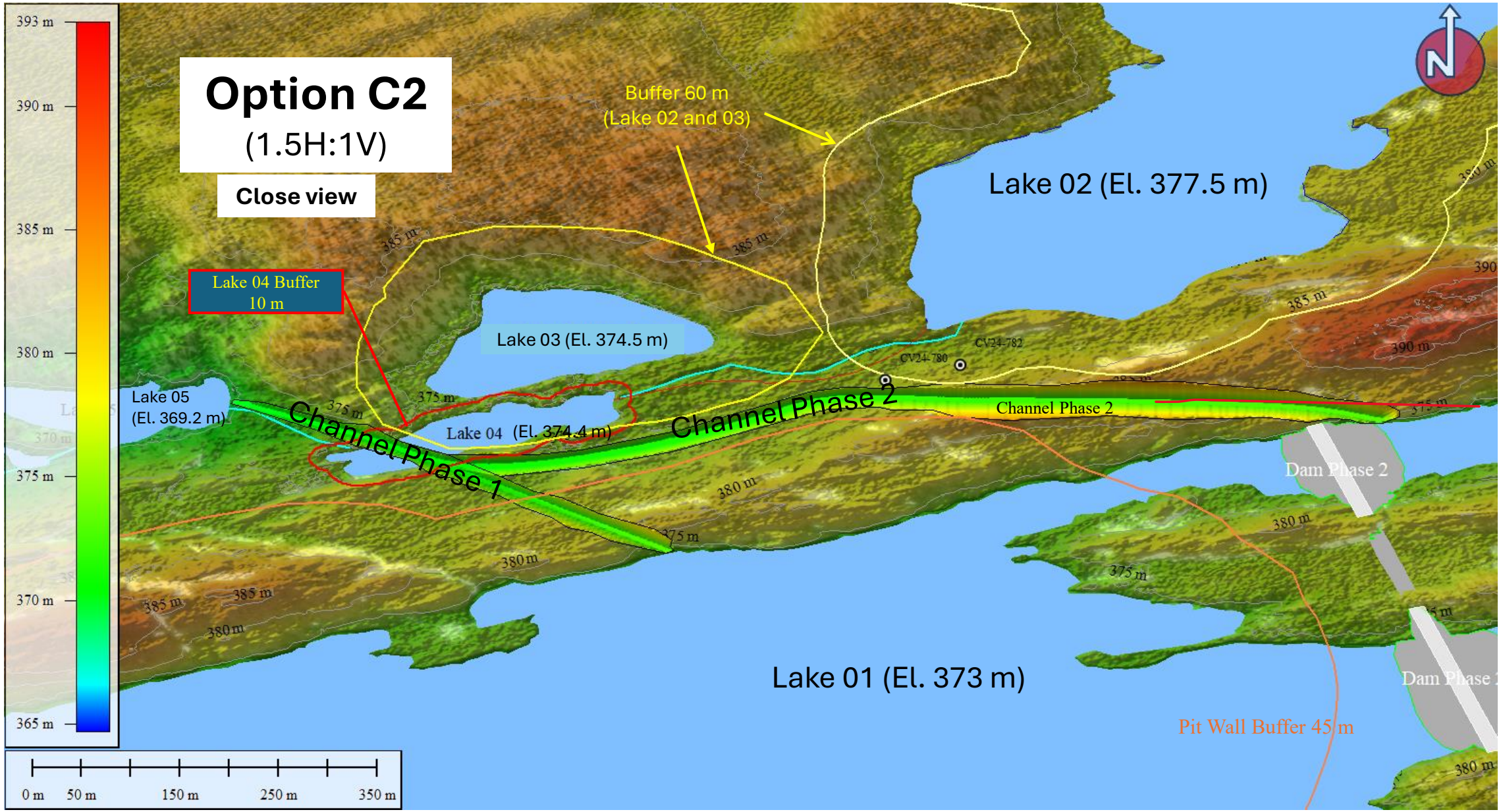

Option B
(2.5H:1V)
Close view
Buffer 60 m
(Lake 02 and 03)
Lake 02 (El. 377.5 m)
<60 m Buffer
Lake 04 Buffer
10 m
<60 m Buffer
Lake 03 (El. 374.5 m)
Lake 05
(El. 369.2 m)
Channel Phase 2
(El. 374.4 m)
Channel Phase 1
Dam Phase 2
Lake 01 (El. 373 m)
Pit Wall Buffer 45 m
393 m
390 m
385 m
380 m
375 m
370 m
365 m
0 m
50 m
150 m
250 m
350 m
N



Water Level in the lakes (based on raster): Lake 02 = 377.5 m ; Lake 03 = 374.5 m ; Lake 04 = 374.4 m ; Lake 05 = 369.2 m

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Option C1
(2.5H:1V)
Close view
Buffer 60 m
(Lake 02 and 03)
Lake 02 (El. 377.5 m)
Lake 04 Buffer
10 m
Lake 03 (El. 374.5 m)
Lake 05
(El. 369.2 m)
Lake 04 (El. 374.4 m)
Channel Phase 1
Channel Phase 2
Channel Phase 2
CV24-782
CV24-780
Dam Phase 2
<45 m Buffer
Lake 01 (El. 373 m)
Pit Wall Buffer 45 m
Dam Phase 1
393 m
390 m
385 m
380 m
375 m
370 m
365 m
0 m
50 m
150 m
250 m
350 m
N


Option C2
(1.5H:1V)
Close view
Buffer 60 m
(Lake 02 and 03)
Lake 02 (El. 377.5 m)
Lake 04 Buffer
10 m
Lake 03 (El. 374.5 m)
Lake 05
(El. 369.2 m)
Lake 04 (El. 374.4 m)
Channel Phase 1
Channel Phase 2
Channel Phase 2
Dam Phase 2
Lake 01 (El. 373 m)
Pit Wall Buffer 45 m
CV24-782
CV24-780
393 m
390 m
385 m
380 m
375 m
370 m
365 m
0 m
50 m
150 m
250 m
350 m
N

Summary of Results: Options A, B and C

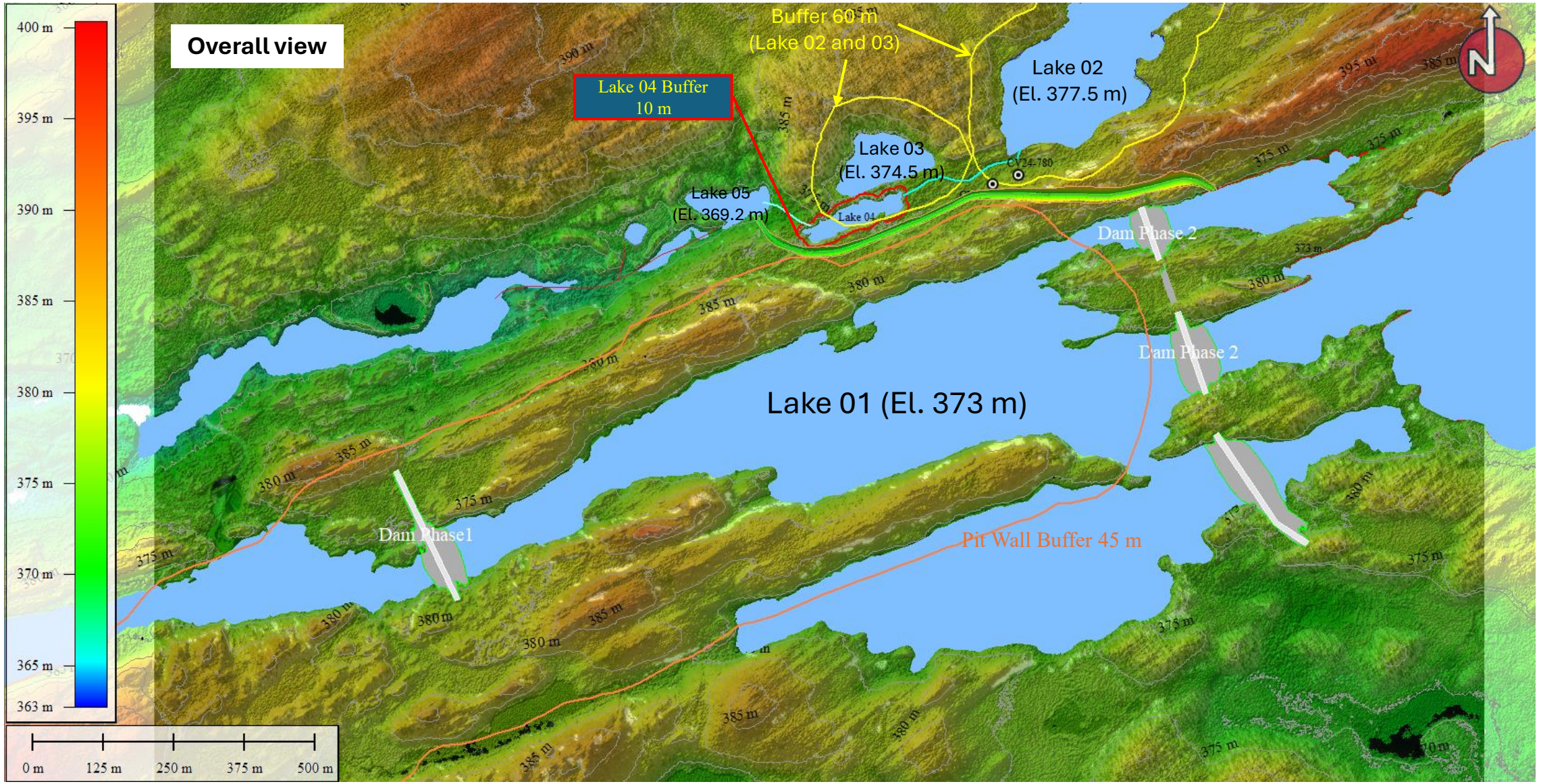
Option	Length	Side Slope	Offset Intrusion respected? (Y/N)				Volume (1000m^3)		
			Pit Wall (45 m)	Lake 02 (60 m)	Lake 03 (60 m)	Lake 04 (10 m)	Phase 1	Phase 2	Total
A	793 m	2.5H:1V	N	N	Y	N (≈5 m)	31	84	115
B	738 m	2.5H:1V	Y	N	N	N	20	85	105
C1	726 m	2.5H:1V	Y	Y	Y	N	20	93	113
C2		1.5H:1V	Y	Y	Y	N	15	66	81

It should be noted that after further discussions with the team, it was decided that these options are no longer feasible, and option D was selected as the preferred option. Option D is described in the next slides.

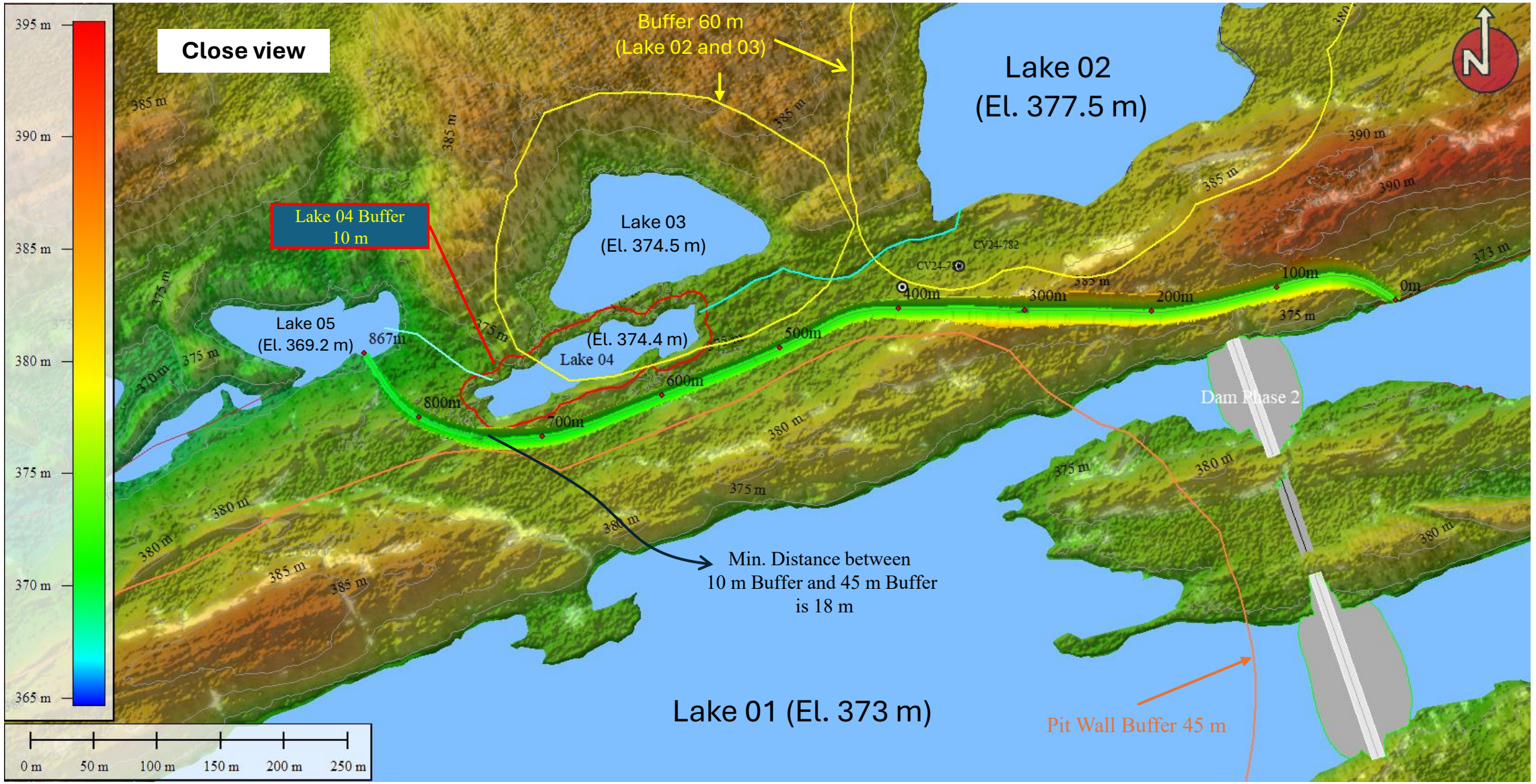
Option D:

- Only a single-stage diversion channel.
- Suggest a 5 m wide and 1H:1V side slope to allow passing the PMF with adequate freeboard (0.6 m). 1H:1V side slopes were selected due to construction constraints due to blasting limitations.
- Model validation for fish hydraulic conditions, including 3 drops of 0.2 m and channel bed slope of 0.3%.
- AtkinsRéalis also suggests adding pools of 0.4 m depth in the channel bed to allow flow velocities close to zero. A single micromodel is included to show the hydraulic conditions within the pools.

Option D: Single-phased option

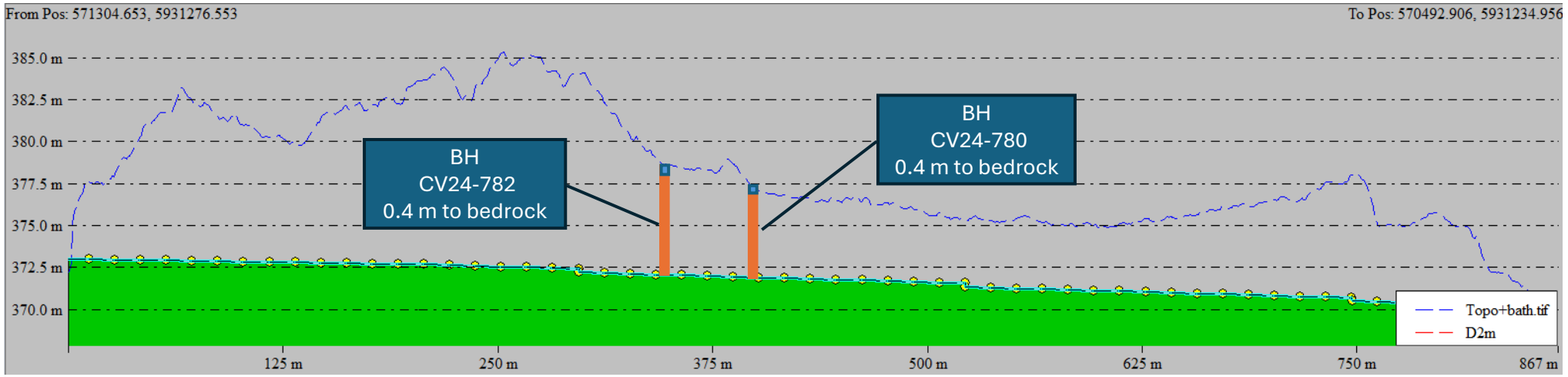


Option D: Single-phased option



Option D

Channel Profile – From Lake 01 to Lake 05



Length ≅ 870 m

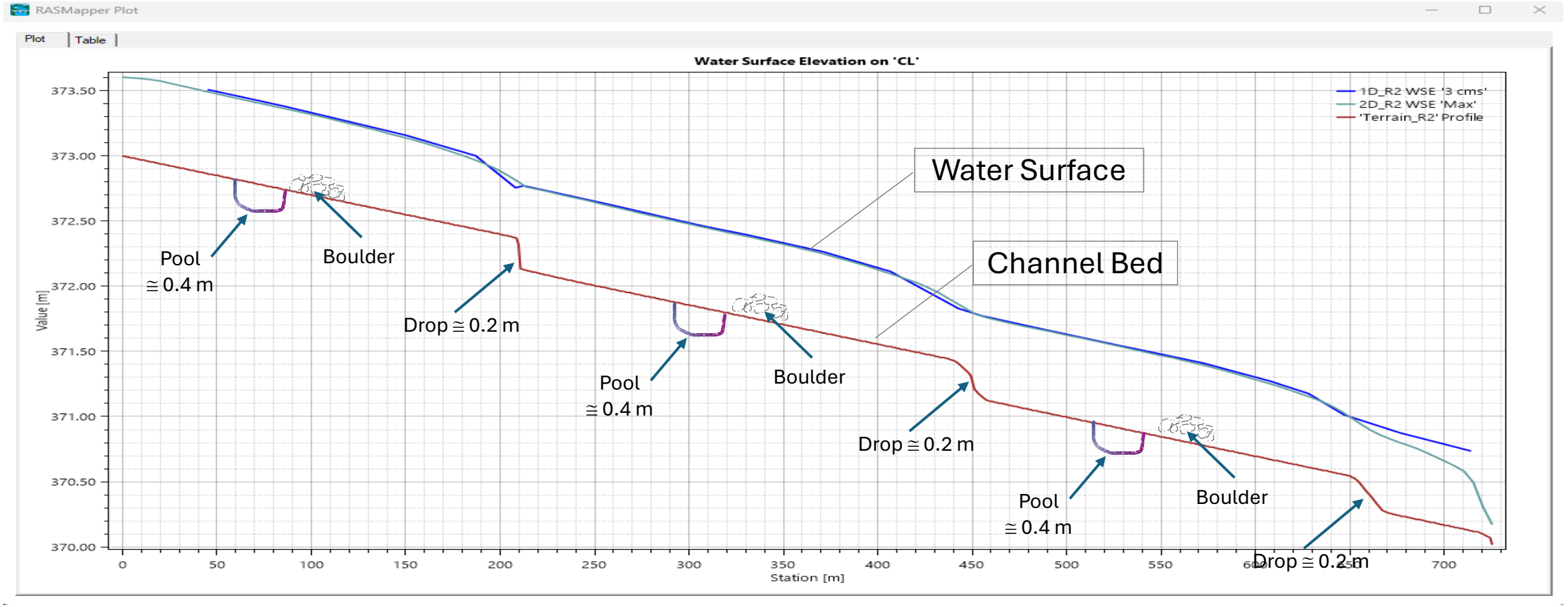
	Sandy Gravel		Bed Rock

Summary of Results – Option D

Option	Length	Side Slope	Offset Intrusion respected? (Y/N)				Volume (1000m^3)		
			Pit Wall (45 m)	Lake 02 (60 m)	Lake 03 (60 m)	Lake 04 (10 m)	Phase 1	Phase 2	Total
D	870 m	1H:1V	Y	Y	Y	Y	One phase only		67

- Option D (enhanced detail for estimations)
  - Typical Section Size: 5 m wide and side slopes 1H:1V (steep slopes due to blasting work).
  - Total Excavation Volume: ~67k m^3 (61k m^3 corresponds to rock and 6k m^3 to overburden).
  - With the side slopes of 1H:1V the results indicate no intrusion into the offset boundaries of the pit and lakes.

Water Surface Elevation (flow = 3m^3/s) – Diversion Channel



AtkinsRéalis has proposed the installation of three or four drops and pools along the channel bed to create varying hydraulic conditions that replicate those at the natural outlet of Lake 01. These hydraulic conditions will facilitate rapid flows and water depths, both upstream and downstream of the drops. Additionally, it is recommended to place boulders between sections of the channel where high velocities are present to provide shelter for fish during their migration from downstream to upstream. The figure above shows a schematic representation of the proposed fishway channel. Slides 19 and 20 explain this in further detail. Furthermore, it is expected that at the location of the pools, flow velocity will be close to zero (especially at the locations near the bed). This aspect is investigated in Slide 21. Therefore, the required conditions for the fish habitat/passage will be satisfied.

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Flow Value Estimations – Diversion Channel

Table 1. Flow and substrate characteristic of CE1 (Current natural outlet of Lake 01)

Segment	Main flow Type	Width (m)		Current velocity (m/s)	Mean Depth (m)	Dominant Substrate
		Wetted	Littoral boundary			
SH1	Lentic channel	15.0	17.0	0.03	0.80	Boulder, coarse boulder, cobble
SH2	Runs	7.0	9.2	0.50	0.50	Boulder, coarse boulder, cobble
SH3	Pool	60.0	65.0	0.05	0.40	Cobble. Boulder, coarse boulder
SH4	Runs	7.0	9.0	0.60	0.60	Boulder, coarse boulder, cobble
SH5	Cascade	5.0	7.0	1.00	0.30	Boulder, coarse boulder

Note: It is unknown the total flow passing through the natural channel at the time of this survey; however, based on the report pictures, it was assumed average conditions (i.e., between July and October)

Table 1 from PMET Report – Technical Note Diversion Channel and Fish Habitat Requirements, Feb 11, 2025.

Based on the Monthly Ave Flows provided by the Client on Jan 28, 2025

- Monthly Min (January) = 0.6 m^3/s
- Monthly Ave = 2.1 m^3/s (Adopted to validate the results)
- Monthly Max (May) = 6.9 m^3/s

Hydraulic modelling of scenario D is conducted using these flows, and results are presented in Slides 20 and 21. The results of flow velocities and depths were found to be consistent with the table above. For instance:

- For the mean monthly flow (2.1 m^3/s), the measured velocity and depth at the runs are 0.6 m/s and 0.6 m (Table 1- above), which is in agreement with the simulated value at the runs (Slide 20 – Section 1), which are 0.6 m/s and 0.6 m (Slide 20; Table 2).
- Similarly, for the mean monthly flow, at the cascade/drop, the measured depth and velocity are 0.3 m and 1.0 m/s (Table 1 – above), and the corresponding simulation results at the cascade/drop (Slide 20 – Section 2) are 0.4 m and 0.9 m/s (Slide 20; Table 2). The difference between the measured and simulated values is insignificant, and the results seem adequate.
- Therefore, it is expected that the proposed diversion channel can replicate the hydraulic conditions of the current natural stream (CE1).

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Hydraulic Model Results

Table 1

Q = 0.6 m³/s (Min monthly flow)	WSE [m]	Depth [m]	Velocity [m/s]
Upstream of drop (run) (Section 1)	373.2	0.3	0.4
At drop/cascade (Section 2)	371.7	0.2	0.8

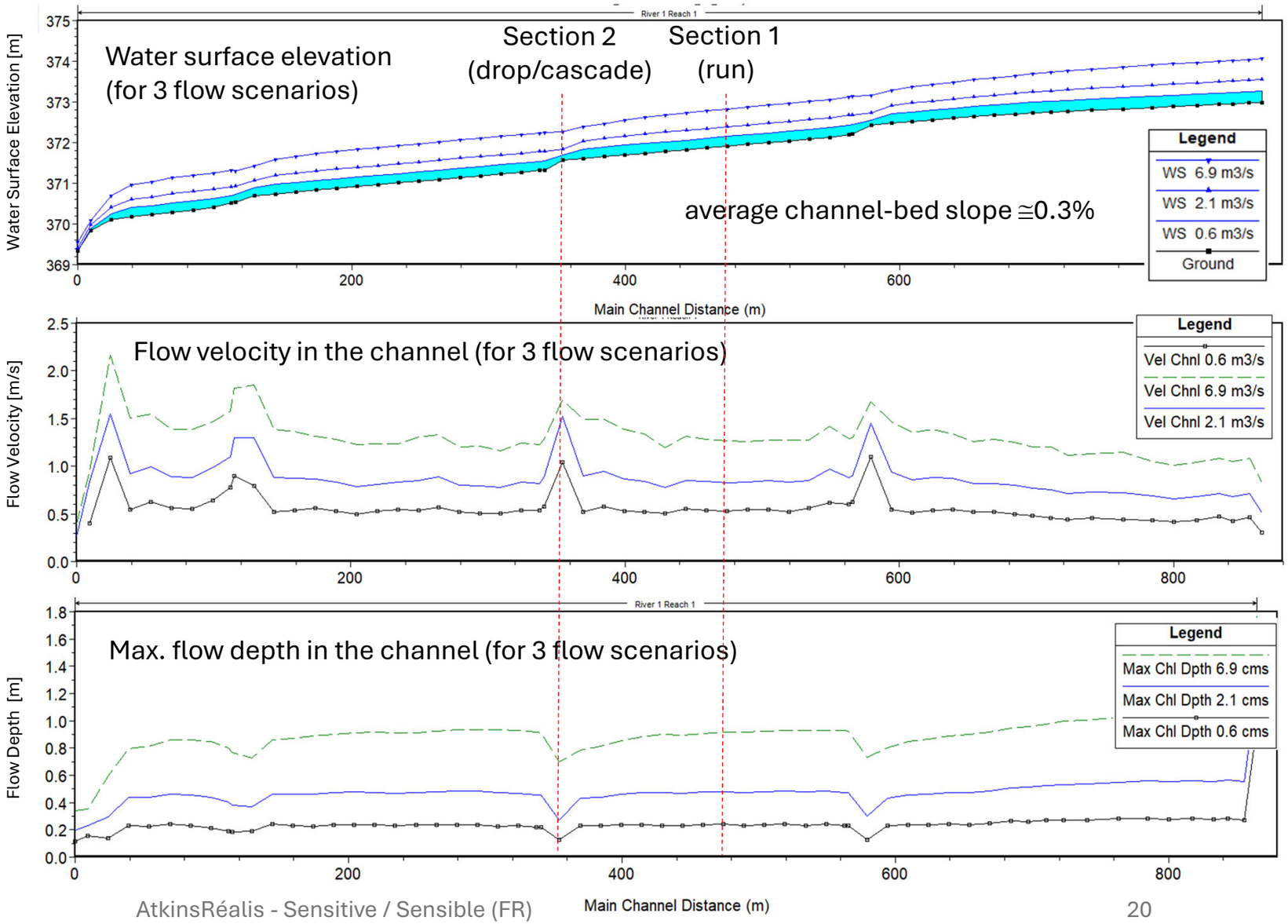
Table 2

Q = 2.1 m³/s (Mean monthly flow	WSE [m]	Depth [m]	Velocity [m/s]
Upstream of drop (run) (Section 1)	372.5	0.6	0.6
At drop/cascade (Section 2)	372.0	0.4	0.9

Table 3

Q = 6.9 m³/s (Max monthly flow)	WSE [m]	Depth [m]	Velocity [m/s]
Upstream of drop (run) (Section 1)	373.1	1.2	0.9
At drop/cascade (Section 2)	372.6	1.0	1.0

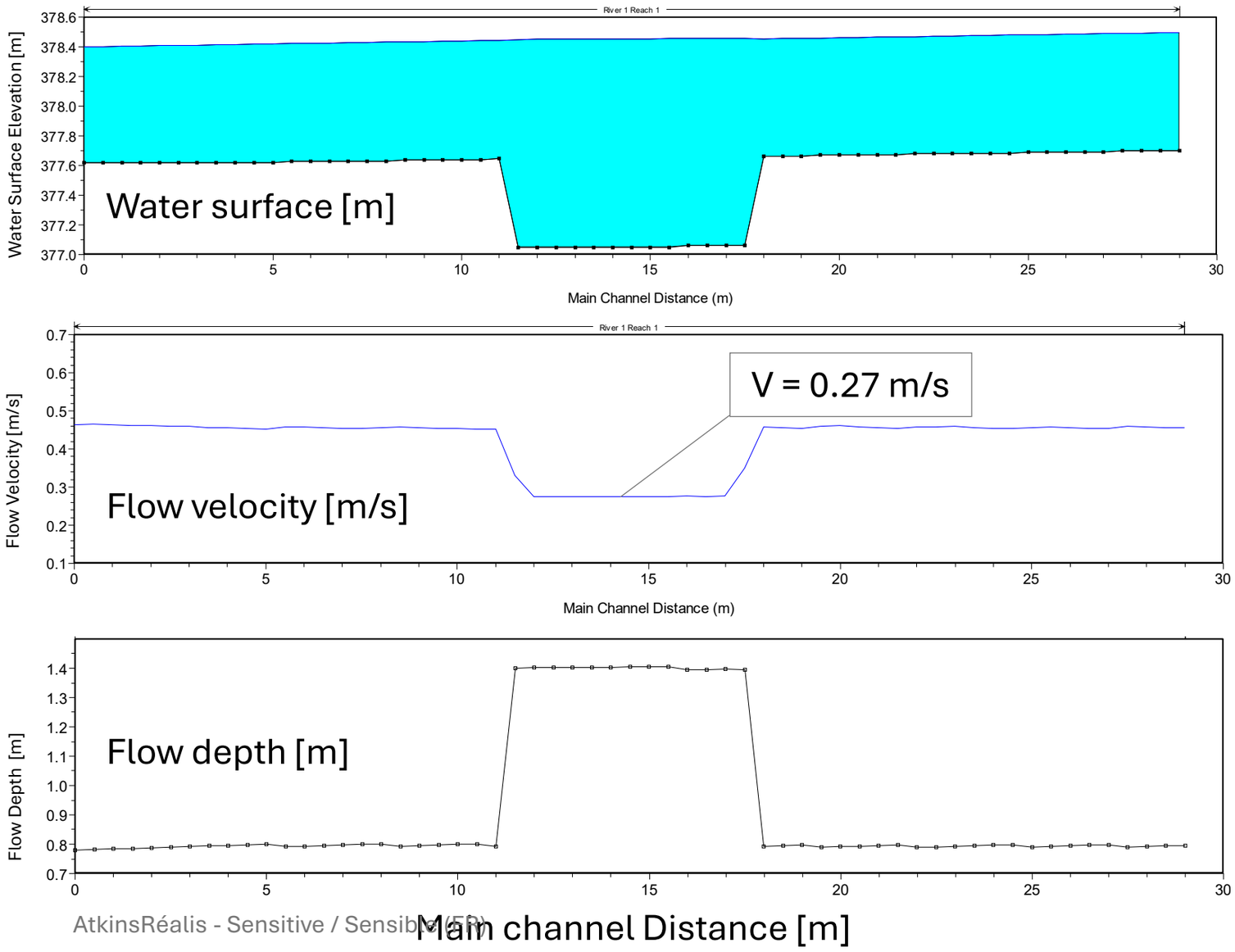
HEC-RAS results: water surface (top), flow velocity (middle), and flow depth (down) along the channel (Option D with 3 drops) : 3 flow scenarios



Simulation results of pools:

- To investigate the impact of a pool on flow conditions, a HEC-RAS micromodel was developed for a typical pool in the channel. The total length modelled is 30 meters, with one pool simulated.
- The results of the model, including water surface elevation, flow velocity, and flow depth, along the channel are presented in these figures.
- It is observed that flow velocities are significantly reduced at the location of the pool. Since this is a 1D depth-averaged model, the velocity magnitude represents the average value across the depth. Therefore, it is expected that velocities near the bottom of the channel are low and close to zero.

Profiles of water surface, flow velocity, and flow depth for "Mean flow" scenario (i.e., Q= 2.1 m^3/s)



Discussion

Option	Cons	Pros
A	• Intrusion to buffer boundary 60 m of Lake 02 • Intrusion to buffer boundary 45 m of Pit • Only 5 m gap between the right embankment to the water body of Lake 04	Not selected by client/Options below address concerns
B	• Intrusion to buffer boundary 60 m of Lake 02 • Intrusion to buffer boundary 60 m of Lake 03 • Impact Lake 04 and possible Lake 02's natural channel	Not selected by client/Options below address concerns
C1	• Minor Intrusion to buffer boundary 60 m of Lake 02 • Minor Intrusion to buffer boundary 45 m of Pit • Impact Lake 04 (west section of the lake). Lake will discharge into ditch. • Difficult to construct in Lake 04 footprint.	• Shorter ditch • Good hydraulic capacity • Side slopes of 2.5:1 are considered suitable for the construction of conventional foundation soil
C2	• Impact Lake 04 (west section of the lake). Lake 04 will discharge into ditch • Side slopes of 1.5:1 are considered too steep for conventional foundation soils (Armoring of side slopes will be required). • Less hydraulic capacity • Difficult to construct in Lake 04 footprint.	• Shorter ditch • Less excavation
D	• ≅ 5 m between the top channel boundary (both sides ≅ 10m) and to edge of the buffer of Lake 04 and the Pit. The total distance between the pit and lake buffers is 18 m. • No phased ditch is considered. The ditch needs to be constructed at the beginning of the Lake 01 dewatering.	• Only one Phase is considered (Phase 2) • No Intrusions to the buffer boundary of 45 m of the pit. • No Intrusion to buffer boundary **10 m** of Lake 04. • Adequate hydraulic capacity. • Existing waterways between Lake 04 and Lake 05 are undisturbed • Minimum excavation volume among all scenarios

Recommendation

- Option D is recommended *(with1H:1V side slope throughout the channel):*
  - Reason for choosing Option D:
    - This scenario maintains the 45m Open Pit wall pillar on the entire length of the channel, where feasible
    - Less excavation volume (due to narrow side slope) – also easier to implement (rock blasting)
    - Allows for the least disturbance of existing Lakes and streams to the North
    - Easier construction on land vs. in marshland
    - Best respects the need for the Open Pit pillar and also the Lake offset requirements
    - With drop-pool scenarios associated with boulders, the required considerations for fish migration toward upstream are well satisfied.

Costs

A full cost evaluation was not completed on the Options nor factored into the decision-making.

- It was determined that the cost differences between the options would not significantly justify impacting the surrounding watershed.
- Secondly, any reduction in costs due to shorter channels (less rock & soil excavation) would be offset by significantly increased construction costs associated with working in marshland.

Questions?

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APPENDIX

4-7 120 KV TRANSMISSION LINE STUDY

Innova Lithium Inc.

Shaakichiuwaanaan project

James Bay Region, Québec

Technical Report

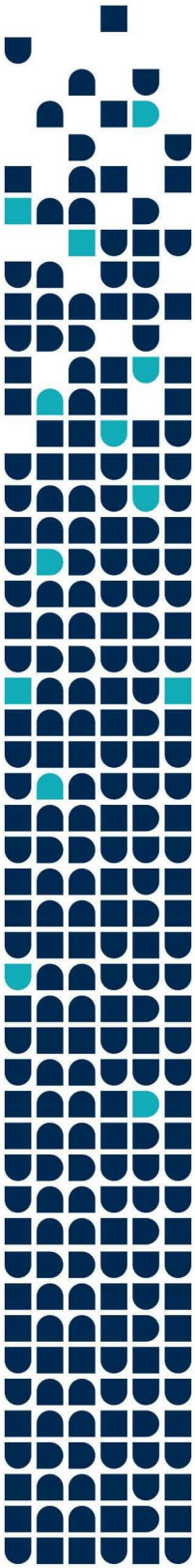
120 kV Transmission Line Study

BBA Document No.-Rev.: 7535031-000000-4L-ERA-0001-RAB

September 5, 2025

FINAL-STUDY

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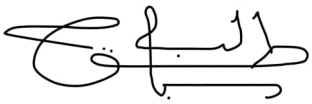




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Florence Nolette



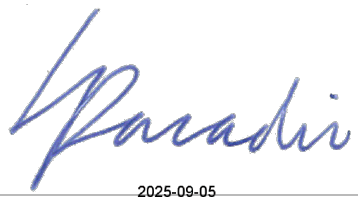
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REVISION HISTORY

Revision	Document Status – Revision Description	Date
RAB	Final-Study	2025-09-05

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Executive Summary

This document is a prefeasibility report for a projected 120 kV transmission line (TL) supplying 50 MW to exploit an open pit lithium mine, located approximately 50 km south-west from Hydro-Québec (HQ) Tilly substation in James Bay, for the development of its Shaakichiuwaanaan project. The project was originally planned at 69 kV but was increased to 120 kV due to additional power demand.

Preliminary design criteria, along with structure, foundation and main component selections were performed. The context allows for H-frame type wooden structures and direct-buried foundations. The average height of structure is 70 ft (61 ft above ground), with spans in the range of 175 m between structures, on a 38 m wide cleared right of way.

Two (2) 120 kV preliminary line routing options have been studied considering the available screening features/constraints:

- Option 1: Mainly along the Trans-Taiga Road as much as possible to minimize the construction of access roads.
- Option 2: More direct line between projected 315-120 kV substation and projected 120-13.8 kV mine substation.

Based on available information at this stage, option 1 seems to be more advantageous. Though it presents fewer constraints, it still passes relatively close to locations that were identified as sensitive sites. While it is important to keep both options at this point to provide flexibility for future steps, this study considers option 1 as the selected line routing.

A preliminary structure spotting and line routing adjustment were performed for line option 1, following the integration of lidar data as well as surficial deposits mapping. It consists of a total of 318 wood structures distributed on a 54.8 km long line routing. It should be noted that adjustments will be needed in the following phases (for example due to wetlands delineation, social impacts, land usage, coordination with HQ, additional engineering precision).

A material take-off was prepared, based on assumptions described in this report.

A preliminary schedule was also prepared, and the next key steps were identified. It demonstrates that the line could be commissioned in mid 2029, providing that all the steps highlighted in the schedule are performed within the appropriate timeframe.

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APPENDICES

Appendix A: Typical ROW Section for Tangent H-Frame Structures
Appendix B: Typical ROW Section for Dead-End / Angle H-Frame Structures
Appendix C: Typical Foundation Drawings
Appendix D: Preliminary Schedule

1. Introduction

Innova Lithium Inc. aims to exploit an open pit lithium mine located in the Eeyou Istchee James Bay region in northern Québec at the coordinates 53.52°N, 73.94°W. Based on the feasibility study, the maximum load at the mine is expected to be 43.1 MW. To provide a safety margin, this study considers a 50 MW load.

The project site is located approximately 50 km south-west from Hydro-Québec (HQ) Tilly substation in James Bay. At this location, there are two access points to the HQ network:

- at 25 kV from HQ distribution network near La Grande complex airport;
- at 315 kV at HQ Tilly substation, near La Grande-4 (LG-4) generating station.

The existing 25 kV power supply at the airport cannot meet the mining complex's projected demand for 50 MW. A 25 kV line does not have the capacity to transmit that much power over the required distance. Building a 315 kV line would be too costly and time-consuming for this relatively small load. An intermediate tension level is therefore required. Due to the increased power demand versus the initial PEA evaluation, a 120 kV transmission line will be required, instead of the originally planned 69 kV line.

2. Codes, standards and regulations

The works to be accomplished shall be capable of meeting all requirements specified and shall conform to the latest revisions of applicable codes, standards, and regulations including, but not limited to the following:

Table 1: Codes, standards, and regulations

Document Code	Document title
CSA (Canadian Standards Association)	
CAN/CSA – C22.3 No. 1	Overhead Systems
CAN/CSA – C22.3 No. 60826	Design Criteria of Overhead transmission Lines
CAN/CSA C411.1	AC Suspension Insulators
CAN/CSA -C61089	Round Wire Concentric lay Overhead Electrical Stranded Conductors
CAN/CSA C83	Communication and Power Connectors for use in Overhead Line Conductors
CSA-C57	Electric Power Connectors for use in Overhead Line Conductors
IEC (International Electrotechnical Commission)	
IEC 60071	Insulation coordination part 2: Application guide
IEC / CIGRE	Technical Brochure 273

2.1. Reference documents

The information included in this report must be used in conjunction with these documents:

Table 2: Reference documents

Document	Description
7535016-000000-30-CPP-0001	Project Execution plan for 25 kV Powerline Camp connection
7535031-000000-4L-EES-0001	Technical Specification -120 kV Transmission Line – LiDAR Survey
7535031-000000-4L-EMO-0001	LiDAR / Survey zone (KMZ file)
7535031-000000-4L-EMO-0002	Preliminary Routing Options (KMZ file)
7535031-000000-4L-EMO-0004	Preliminary line routing and structure spotting (KMZ file)
CASN-A-001-GE-GA-0002_RevB4(ID 12397)	Site Layout - General Arrangement - Main plan – Plan View
HQ TET-EMP-N-GEN0001	Normes calculs d'emprises (Hydro-Québec)
HQ TET-LIA-N-GEN0100	Critères de conception des lignes aériennes (Hydro-Québec)
Corvette_hauteur_250424_Optimized	Elevation and terrestrial habitat types

3. Interconnection scenario

In order to feed the mining operations, a 315-120 kV substation will be built near the 735-315 kV Tilly substation at the LG-4 site. A short 315 kV line will power the substation where the tension will be converted at 120 kV. The 120 kV transmission line will then reach the mine site, where a 120-13.8 kV substation will be built, with a capacity of 50 MW and above, for the permanent needs of the Shaakichiuwaanaan mining complex. The new 315 kV line should be less than 2 km long to avoid triggering an EA study and causing additional delays on the project schedule.

4. 120 kV transmission line (TL)

4.1. Preliminary Design criteria

Based on the applicable standards, BBA determined preliminary design criteria for the transmission line and structure designs, as presented in the following tables.

4.1.1. Climatic conditions

BBA has conducted a preliminary review of the available climatic data for the region to establish initial environmental conditions.

Based on the assessment, the area is not subject to harsh climatic conditions, and the occurrence of rime ice is considered highly unlikely. Rime ice typically forms when supercooled water droplets freeze upon contact with a solid surface, often in high-altitude or severe weather environments. However, such conditions are not expected along the proposed routing. As a result, rime ice will not be included in the structural load calculations due to its low probability of occurrence.

A detailed climatic study may still be conducted during the detailed engineering phase to confirm these assumptions and ensure the design remains robust under all expected environmental conditions.

The line will be designed for loads with a return period of 50 years as per IEC 60826 Standard. The reliability corresponds to level 1 as per CSA C22.3 No. 60826 Table 1. This level is selected as the expected design life of the mine is below 50 years.

The preliminary climatic conditions considered for this study are the following:

Table 3: Climatic conditions

Condition	Value
Ice (50 years)	
Maximum ice thickness (including adjustment factor)	30 mm (HQ TET-LIA-N-GEN0100)
Wind (50 years)	
Maximum reference wind speed	90 km/h
Temperature	
Mean annual	-3 °C (Environment Canada web site)
Maximum recorded	33 °C (Environment Canada web site)
Minimum recorded	-51 °C (Environment Canada web site)
Average of the coldest month	-23 °C (Environment Canada web site)
Maximum operating temperature for the conductor	100 °C
Elevation	
Above the mean sea level	From 260 m to 440 m
Terrain	
Category	B (CSA-C22.3 No. 60826)
Mean annual snow depth	1.0 m (Table D.1 CSA 22.3 No. 1-20)

4.1.2. Ground clearances

The following clearance requirements from CSA 22.3 No. 1 apply between the conductor at its lowest position and the closest point on the ground.

The clearance will be calculated with the conductor at the following conditions:

- Maximum conductor temperature (100 °C)
- Maximum ice thickness on conductors (30 mm)

Design and construction tolerance of 1.2 m are typically added to vertical minimum code clearances to account for approximations in design, and inaccuracies in survey and installation.

Table 4: Round Clearance

Description	Minimum Code Clearance (m)	Snow depth (m)	Tolerance (m)	Minimum Ground Clearance (m)
Vertical Clearances				
Over roads, or alongside land (within the maximum horizontal swing plus travelled by road vehicles (including roadways, streets, lanes, alleys, driveways and entrances); H=4.15 m	5.5	0	1.2	6.7
Over walkways or land normally accessible only to pedestrians alone, snowmobiles, and personal use ATV; H = 2.4 m	4.0	1.0	1.2	6.2

4.1.3. Insulation coordination

A medium pollution level was considered to account for the proximity to road infrastructure as well as the mining complex.

Table 5: Insulator properties

Description	Suspension string
CSA Type	CSA CS-5 Ball and Socket type
Materiel Type	Toughened Glass
Pollution level (mm/kV)	20 (Medium)
Number of units per string	1 x 9
Mechanical failing load (kN)	120
Creepage distance/insulator (mm)	320
Diameter per units (mm)	255
Section length (mm)	1 314
Effective leakage distance (mm)	2 880

4.1.4. Conductor selection

With the expected capacity of 50 MW, the use of a 636 MCM ACSR Rook conductor is considered at this stage. The optimal line conductor size is based on the generally accepted current density criteria of 1A/mm^2 to operate within the economic range of the line. The following table and figures show some of the conductor's properties as well as expected electrical performance to supply the expected loads assuming a 53 km long TL.

Table 6: Expected Electrical Performance

Description	Single Conductor 1 x 636 kcmil ACSR ROOK
Build	24 x A1 Aluminum strands / 7 x S1A Steel strands
Diameter	24.8 mm
Rated strength	101 kN
Rated Ampacity (@75C)	709A
Ampacity (@50MW)	264 A
Supply voltage	120 kV
Length of the line	53 km
Load power factor	0.95
Phase disposition	Horizontal
Load active power	50 MVA
Surge Impedance Loading (SIL)	38.6 MW
Line maximum power (@75C)	116 MW
Line active power losses	1.12 MW

Description	Single Conductor 1 x 636 kcmil ACSR ROOK
Annual energy losses for 8000 hours	8 960 MWh
Annual energy losses equivalent cost ($0.06/kWh, 0.8 annual loss factor)	0.43 M$
Useful life loss evaluation (10%, 24 years)	3.8 M$
Voltage drop (@50MW)	4.91 %

4.1.5. Structure type selection

The transmission line will be composed of H-Frame wooden structures. These consist of two wood poles with approximately 4.4 metres horizontal separation. It supports a crossarm near the top of the poles which holds the three phases. These structures are expected to perform well in the context of this project and to be an economical solution. Average span lengths of 175 m and average pole height of 70 ft (61 ft above ground) are estimated.

Dead-end or angle structures will be composed of 3 guyed wood poles and will have a bigger footprint at ground level.

4.1.6. Foundations

It is expected that most of the foundations will consist of the direct embedment of the poles. Typically, poles will be buried at 10% of their length plus 2 ft, which is standard practice.

Special foundations such as piles or corrugated steel pipes might be needed for wetland applications or soils with very low bearing capacity.

Anchors consisting of steel plates or anchor rods will be used for the guying of structures respectively in overburden and rock.

4.1.7. Right-of-way

The right-of-way (ROW) limits will be confirmed during the design phase to respect the local constraints and the tree heights, to obtain the targeted reliability of supply.

Based on the report "Elevation and terrestrial habitat types," the preliminary ROW width for typical alignment structures is established at 38 m, typically 19 m on each side of the centreline. An average mature tree height of 12 m was considered, assuming that all danger trees outside the ROW limits will be removed during tree clearing work.

Preliminary layout of an alignment and dead-end/angle structure, including the right-of-way (ROW) configuration, is shown in appendix A and B. Drawings of typical foundations in overburden soil and rock are shown in Appendix C.

4.1.8. TL main characteristics

Table 7: TL main characteristics

Characteristics	Description
Conductor Type	Single conductor 636 ACSR Rook 24/7 (3 phases)
Phase disposition	Horizontal
Phase transposition	None expected, to be confirmed
Structure type	Wood H-frame(alignment) 3 guyed poles (dead-end/angle structures)
Foundation type	Direct embedment (in overburden or rock)
Grounding	Perimetric grounding and continuous counterpoise between structures
Average poles height	70 ft (±21.3 m)
Overhead shield wires	1 optical ground wire (fibre optic) + 1 steel cable shield wire
Average Span (m)	175 m
ROW (m)	38 m most of the line Larger wood clearing boxes around dead-end structures

4.2. High-level corridor screening

BBA reviewed publicly available information to understand the land use and environmental context for the corridors. A study area of ±1 200 km² wide was created over which screening feature data were collected (Table 3 below).

Table 8: Screening features

Feature	Data source
Lakes, rivers, streams	GRHQ (Géobase du réseau hydrographique du Québec)
Wetlands	MELCCFP (Ministère de l'Environnement, de la Lutte contre les Changements Climatiques de la Faune et des Parcs
Sensitive sites	Innova Lithium inc.
Species with special status	CDPNQ (Centre de Données sur le Patrimoine Naturel du Québec)
Airports	MTMD (Ministère des Transports et de la Mobilité Durable)
Roads, forest, contour lines, sensitive soil deposits, slopes	Natural Resources and Forest (Québec)
Existing transmission lines	Open Infrastructure Map

Land use, environmental features and minimum setbacks were considered to determine the preliminary TL routings and corridors.

Table 9: Land and environmental features and their minimum setback

Feature	Minimum Setback (m)	Comment
735 kV Transmission line (existing ROW)	1,000	Where practicable
La Grande-4 Airport	600	Preliminary value, to be validated
Cabin, cottage, sensitive sites	TBD	From feature edges to TL centreline
Road	20	From road edge to TL centreline
Watercourse, waterbody	20	From feature edge to TL structures, to be validated
Wetlands	10	From feature edge to TL structures, where practicable

4.3. Potential line routings

The area considered for the preliminary TL routing is bordered to the west of the study zone by two existing 735 kV TL, the LG-4 airport and a hydroplane base. Two (2) preliminary line routing options have been identified to start at the projected 315-120 kV substation (located 1.5 km from the 735-315 kV Tilly substation) and head south between the existing 315 kV lines and the LG-4 Dam. They then cross the road, the 315 kV lines and turn off to the south-west towards their respective routes, as described below.

4.3.1. Option 1 – Mainly along the existing road

The first TL routing option (52.9 km long) minimizes access road construction by running along the Trans-Taiga Road before branching off towards the mine. This option consists of three distinct sections:

- **Section 1** (near LG-4 Dam). After crossing the Trans-Taiga Road and the 315 kV line, the route heads into a natural area of approximately 18 km long, through trees, hills and wetlands where the construction of access roads will be required. Otherwise, there would be a considerable line length increase in addition to two 735 kV line crossings. It then rejoins the Trans-Taiga Road about 2 km south of the LG-4 airport.
- The **second section** runs along the existing Trans-Taiga Road for about 17 km before heading to the mine (about 2.3 km before the camp location).
- The **third section** crosses and follows the existing mine access roads. It finally joins the projected 120-13.8 kV mine substation.

Main challenges, constraints and risks:

- **Construction of access roads:** The first section requires the construction of access roads, which involves logistical and environmental considerations.
- **Proximity to airport infrastructure:** TL routing is located at ±1.2 km from LG-4 Airport and 300 m from hydroplane base. Minimum setbacks should be validated at a later stage.
- **Wetland, forest, hills, unfavourable soils:** The routing passes through wetlands, forests, hills and unfavourable soil deposits that may require adaptation of the routing and/or require specialized foundation designs.
- **Proximity to sensitive sites:** The TL routing passes at a minimum of 300 m to locations identified as sensitive sites.

4.3.2. Option 2 – More direct line

The second TL routing option follows a more direct path to the mine substation for most of its total length of 50.5 km. However, this routing option is farther from the Trans-Taiga Road and would therefore necessitate more access roads.

This option features three different sections:

- **Section 1:** With an initial TL routing very similar to the one for option 1, the routing for option 2 deviates from it approximately 11.5 km after the road crossing, then joins it again approximately 13.5 km further on, close to the Trans-Taiga Road.
- **Section 2:** After approximately 2.5 km along the road and TL routing of option 1, option 2 branches off towards the mine for about 12 km to rejoin the option 1 TL routing.
- **Section 3:** For the last 7 km, the TL routing runs alongside option 1 near the winter road to finally connect to the projected 120-13.8 kV mine substation.

Main challenges, constraints and risks:

- **Construction of a new access road:** The second option requires construction of more access roads (sections 1 and 2), which may involve logistical and environmental challenges.
- **Wetland, forest, hills, unfavourable soils:** With a longer portion of routing away from the Trans-Taiga Road, it may pass through increased length of wetlands, forests, hills and unfavourable soil deposits that may require adaptation of the routing and/or require specialized foundation designs.
- **Proximity to sensitive sites:** The TL routing passes at a minimum of 200 m to locations identified as sensitive sites.

The preliminary route options are shown on Figure 1 and are available with the provided komuz file (ref. doc. No. 7535031-000000-4L-EMO-0002).

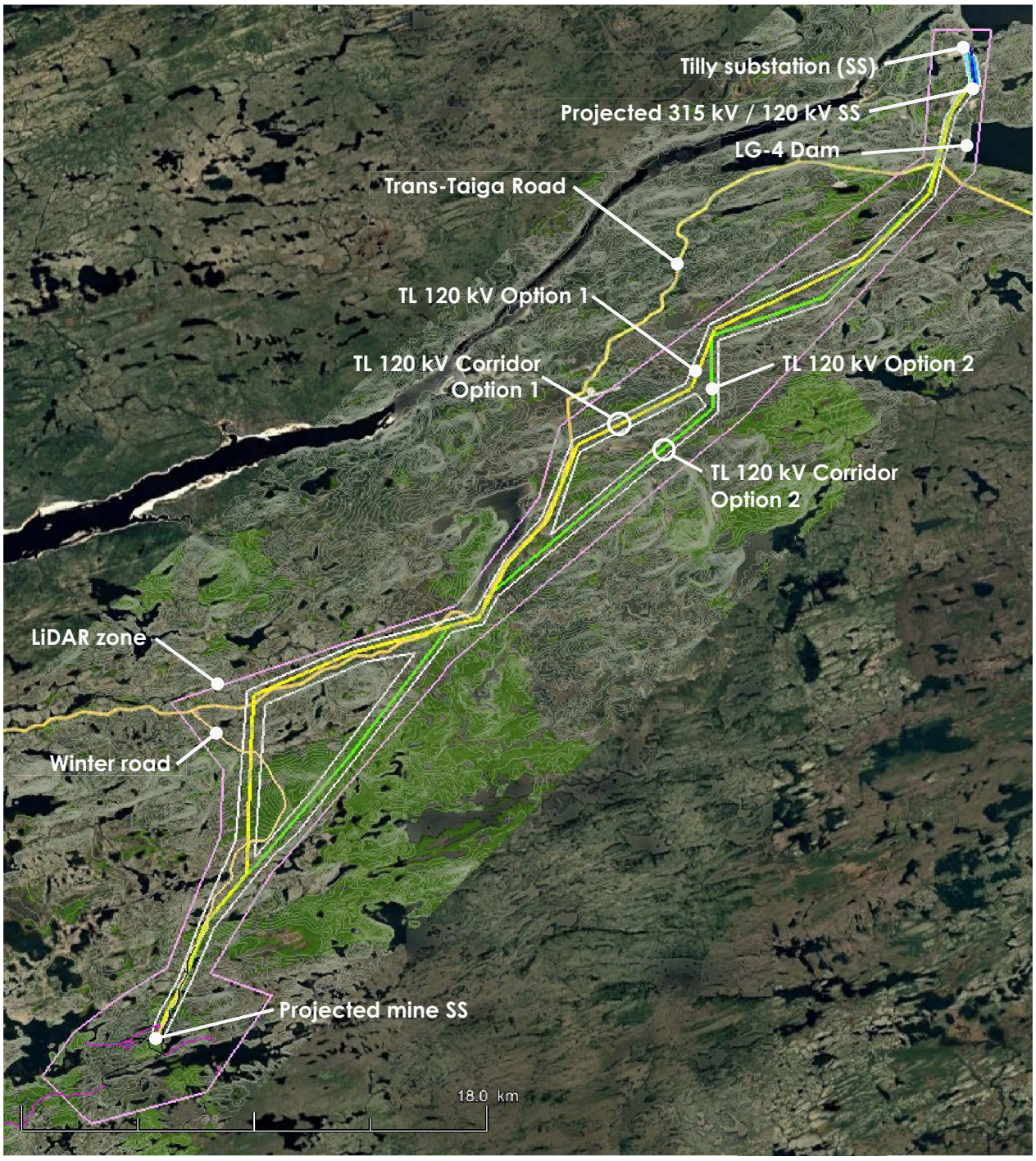


Figure 1: Preliminary route options for projected 120 kV TL

4.3.3. Comparative analysis

The table below shows a comparative analysis of both line routing options.

Table 10: Comparative analysis

Description	Option 1	Option 2	Comment
Length	52.9 km	50.5 km	
Access roads	Less access road construction	More access road construction	Lengths TBD in future phase
Bodies of water	Both options comparable, no major crossings		
Wetlands	Both options comparable		More wetlands area expected post field surveys
Unfavourable soils	32 % of line length	29 % of line length	To be refined with geomorphology study
Slopes	Both options comparable		To re-evaluate with Lidar Data
Forests	Shorter length through old-growth forest (8.9 km)	Longer length through old-growth forest (11.6 km)	
Sensitive sites	Less proximity to sensitive sites	More proximity to sensitive sites	Discussions with communities needed

While the total line length for both options is similar, there are considerable challenges associated with option 2 whereas option 1 seems more favourable. While it is important to keep both options at this point to provide flexibility for future steps, the following sections of the report consider option 1 as the selected line routing for study purposes.

4.4. Preliminary routing and structure repartition

Following, the desktop analysis of line routing options, a lidar survey was performed (see 7535031-000000-4L-EES-0001 and 7535031-000000-4L-EMO-0001 for technical specification and lidar survey area).

Using the lidar study and other available information, a specialized firm underwent a geomorphology study of the preliminary routing corridor. The aim of this study is to identify surficial deposits, which allows determining favourable soils for structure repartition, in consideration with the foundation installation.

Following the integration of lidar data and the information on surficial deposits on a 3D model (using PLS-CADD software), BBA revised the line routing (option 1) and proceeded to the preliminary spotting of structures along the line route. The revised line routing, along with structure locations and numbers, are available with the provided .kmz file (ref. doc. No. 7535031-000000-4L-EMO-0004). It consists of a total of 318 wood structures distributed on a 54.8 km long line routing.

5. Material take-offs

A preliminary material take-off is presented for option 1 routing. Quantities and type of structures are based on the preliminary routing and repartition performed by BBA. Foundation type (overburden vs. rock) was assessed using surficial deposits mapping provided in the geomorphological study report. It should be noted that access road locations and quantities are yet to be assessed and might have a significant impact on the overall estimated cost of the TL.

Table 11: Preliminary MTO

Characteristics	Description
Tree clearing area (ha)	212
Access road construction (km)	TBD
Wood poles	▪ 60' class 1 x 9 ▪ 60' class 2 x 12 ▪ 65' class 1 x 41 ▪ 65' class 2 x 74 ▪ 70' class 1 x 99 ▪ 70' class 2 x 308 ▪ 75' class 1 x 81 ▪ 80' class 1 x 38 ▪ 85' class H1 x 2
Pole foundations (2 poles - Alignment)	▪ 251 x in overburden (with 1200 mm diameter steel bearing plate) ▪ 39 x in rock
Pole foundations (3 poles – Dead-end /Angle)	▪ 27 x in overburden (with 1200 mm diameter steel bearing plate) ▪ 1 x in rock
Anchors	▪ 12 x rock anchor (expanding rock anchor) ▪ 313 x overburden anchor (900 po^2 steel plate)
Structures	▪ 290 x alignment ▪ 28 x dead-end / angle
636 ACSR Rook 24/7 Conductor (km)	173 (5 % buffer)
Shield wire (km)	57 (5 % buffer)
Optical ground wire (OPGW) (km)	57 (5 % buffer)
Insulator CSA CS-5 Ball and Socket type	10 602 units
Grounding (Copper Clad #4 AWG)	Perimetric grounding and continuous counterpoise between structures
OPGW Splice Boxes	15 units

6. Next steps

A preliminary schedule was prepared and is available in appendix D. It demonstrates that the line could be commissioned in mid 2029, providing that all the steps highlighted in the schedule are performed within the appropriate timeframe.

Here are some of the key activities that should be undertaken in a timely manner to ensure the success of the project:

- **Project schedule update.** Coordinate with all involved stakeholders to update the overall project schedule, including permitting, procurement and construction for the power line and substations, as well as HQ interconnection infrastructure.
- **First nation communities.** Initiate or continue communications with first nation communities as early as possible to explore potential business partnerships that may be beneficial and of interest to the project. This process could lead to multiple iterations to obtain a line routing that is suitable for all stakeholders.
- **Wetland desktop study.** A delineation of all wetlands in the area of interest should be performed based on Lidar survey data as well as aerial photography. This will serve as an input in line routing and structure spotting adjustments pre-permitting.
- **Access road and borrow pits study.** The study will allow identifying potential borrow pits to be used during construction and potential access roads. In addition, a higher degree of precision will be put in the interpretation of surficial deposits at every structure location. Another result of this study is a geotechnical field investigation plan. This step is crucial for construction planning and cost assessment.
- **Forestry.** Prepare a wood clearing specification for the permit application. Even though there is no commercial wood at this latitude, *Regulation respecting the sustainable development of forests in the domain of the State* still applies since it is situated below the northern limit of the forest tundra.
- **Field surveys.** Additional field surveys will most likely be needed during the summer of 2026 to prepare permitting applications. Features such as wetlands limits, high water line for waterbodies, protected species will be catalogued.
- **Permitting.** Make a thorough assessment of permits required and prepare applications. For information, the Project Execution Plan prepared by BBA (ref. document 7535016-000000-30-CPP-000) provides useful information on the regulatory framework. It will, however, need to be revised for a voltage level of 120 kV instead of 25 kV.
- **Land usage and leasing.** Based on available information, the area of interest for the TL is on public land pertaining to the domain of the State. However, coordination will be needed with Hydro-Québec due to proximity to their property such as a 315 kV TL, the Trans-Taiga road, and LG4 complex. A land surveyor should be hired post-permitting preparing the necessary documentation for the TL's ROW to be officially recognized. Surveys on lands in the domain of

the State must be preauthorized by the *Bureau de l'arpenteur général du Québec (BAGQ)* and a request will need to be filed for that matter.

- **Preliminary engineering.** Engineering of the line and substations will need to progress to a higher level of precision. Critical steps include the following:
  - Reassess actual and future power demand to confirm the main design parameters of the TL.
  - Prepare required technical documentation for permitting (crossing or proximity to HQ's Trans-Taiga Road, airport proximity with NAV Canada, line crossings with HQ, water bodies crossing with NAV Canada/Transport Canada, tree clearing limits).
  - Structure repartition and adjustments for permitting.
  - Structure and foundations preliminary engineering.
  - Soil resistivity measurements along the line and at future substations locations to determine grounding strategy.
  - Substations preliminary engineering.
  - Define telecommunications requirements for the project as well as possible interconnection points.
- **Continue discussions with MEIE (*Ministère de l'Économie, de l'Innovation et de l'Énergie*).** Continue to follow up on the connection request with the MEIE to ensure that the file progresses and obtains authorization.
- **Continue discussions with Hydro-Québec.** After obtaining authorization from the MEIE, proceed with the connection process with Hydro-Québec. Confirm power requirements and interconnection scenario, as well as expected costs and contributions from Hydro-Québec.
- **Monitor Critical Minerals Infrastructure Fund.** Monitor new applications window, as the previous one closed on June 27, 2025. Substantial funding could be obtained through this program.

7. Conclusion

Preliminary design criteria, along with structure, foundation and main component selections were performed. The context allows for H-frame wooden structure configurations and direct-buried foundations. Additional technical constraints may be identified during the engineering process. If this is the case, special designs will be executed in the detailed engineering phase.

Two-line routing options were successfully identified and analyzed. Based on available information at this stage, option 1 seems to be more advantageous. Though it presents fewer constraints, it still passes relatively close to locations that were identified as sensitive sites. Both options should still be retained at this stage to provide for more flexibility in future steps.

A preliminary structure spotting and line routing adjustment were performed for line option 1, following the integration of lidar data as well as surficial deposits mapping. It should be noted that it is most likely that adjustments will be needed in the following phases (for example due to wetlands delineation, social impacts, land usage, coordination with HQ, additional engineering precision).

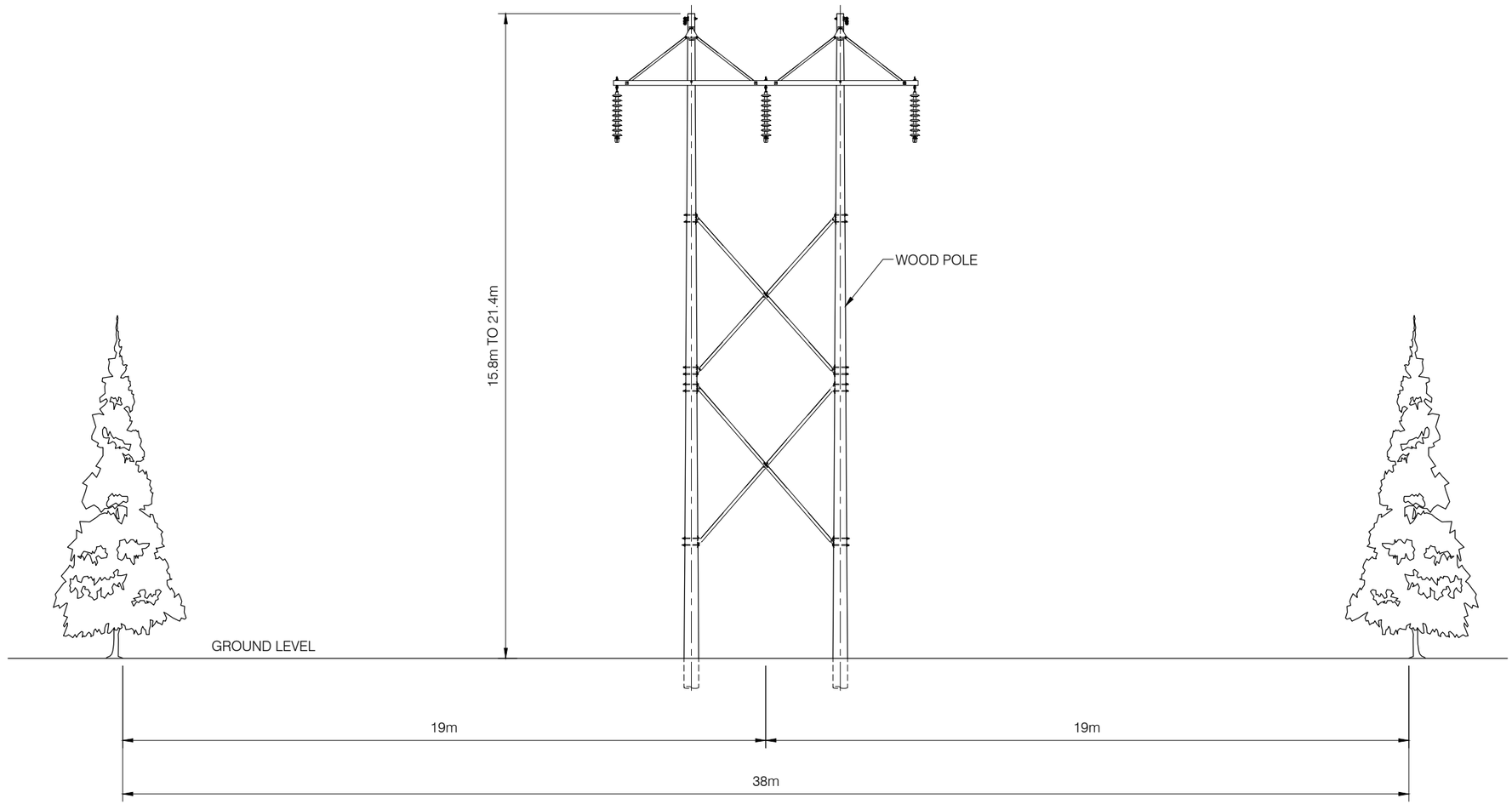
A high-level MTO was prepared, based on assumptions described in this report.

A preliminary schedule was also prepared, and the next key steps were identified.

Appendix A: Typical ROW Section for Tangent H-Frame Structures

NOTES:

DIMENSIONS ARE IN METERS (m) UNLESS OTHERWISE INDICATED.



RIGHT-OF-WAY

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION

REV	DESCRIPTION	PREPARED BY	CHECKED BY	DATE
.	-	:	:	-

REVISIONS

CLIENT: PATRIOT BATTERY METALS

DESIGNED BY: M. LÉGER, Eng. OIQ No. 122782	DRAFTED BY: V. MORAIS
PREPARED BY: M. LÉGER, Eng. OIQ No. 122782	CHECKED BY:
SCALE: N.T.S.	DATE: 2025-06-06

SEAL:

PROJECT: SHAAKICHIUWAANAAN

TITLE: RIGHT-OF-WAY ALIGNEMENT STRUCTURE

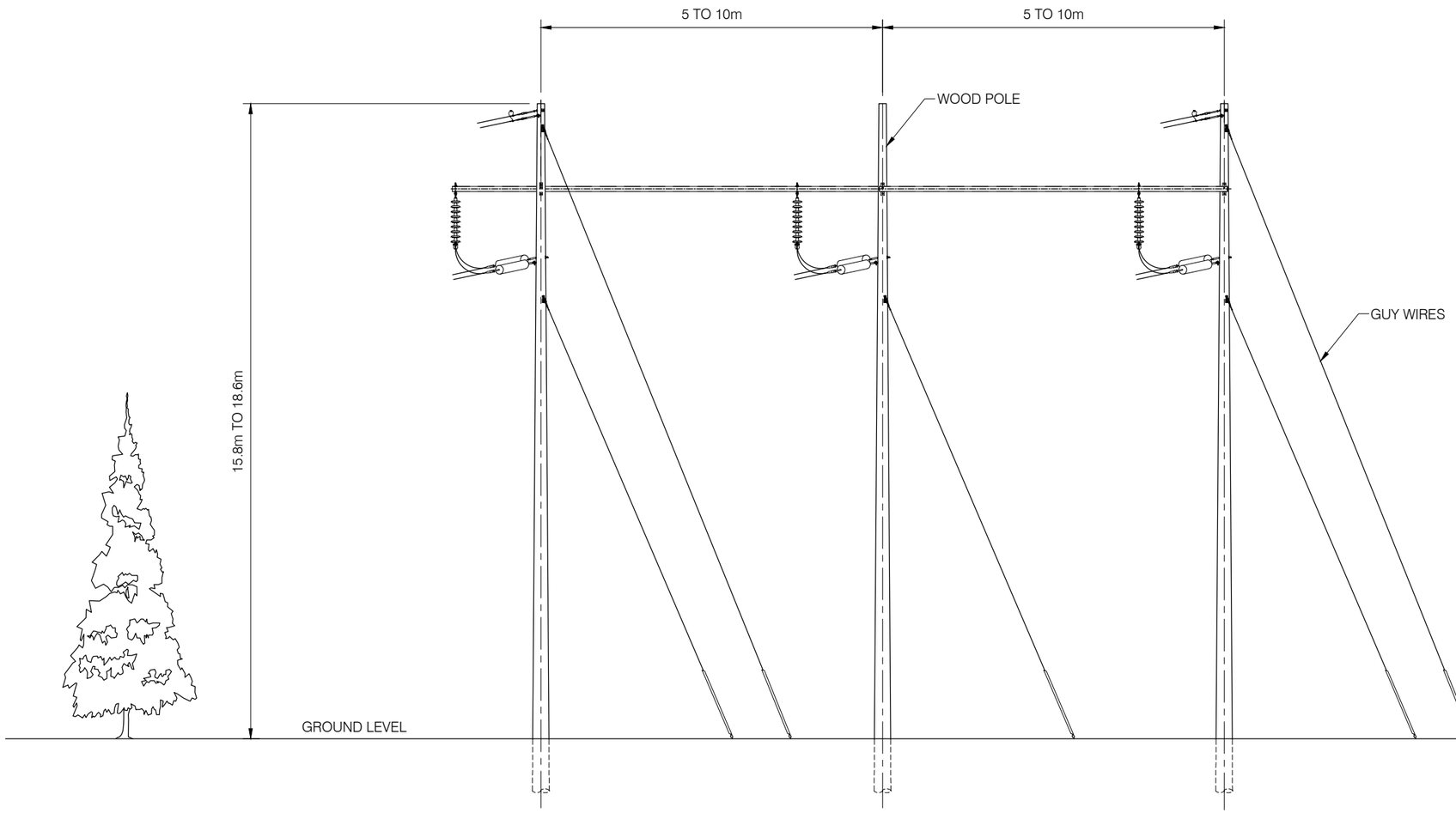
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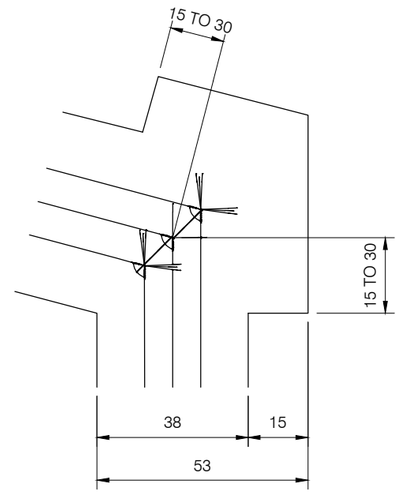
Appendix B: Typical ROW Section for Dead-End / Angle H-Frame Structures

NOTES:

DIMENSIONS ARE IN METERS (m) UNLESS OTHERWISE INDICATED.



RIGHT-OF-WAY



PLAN VIEW

TREE CLEARING BOX - DEAD-END STRUCTURES

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION

REV	DESCRIPTION	PREPARED BY	CHECKED BY	DATE
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REVISIONS

CLIENT: PATRIOT BATTERY METALS

DESIGNED BY: M. LÉGER, Eng. OIQ No. 122782	DRAFTED BY: V. MORAIS
PREPARED BY: M. LÉGER, Eng. OIQ No. 122782	CHECKED BY:
SCALE: N.T.S.	DATE: 2025-06-06

SEAL:

PROJECT: SHAAKICHIUWAANAAN

TITLE: RIGHT-OF-WAY
DEADEND 0° TO 90° STRUCTURE

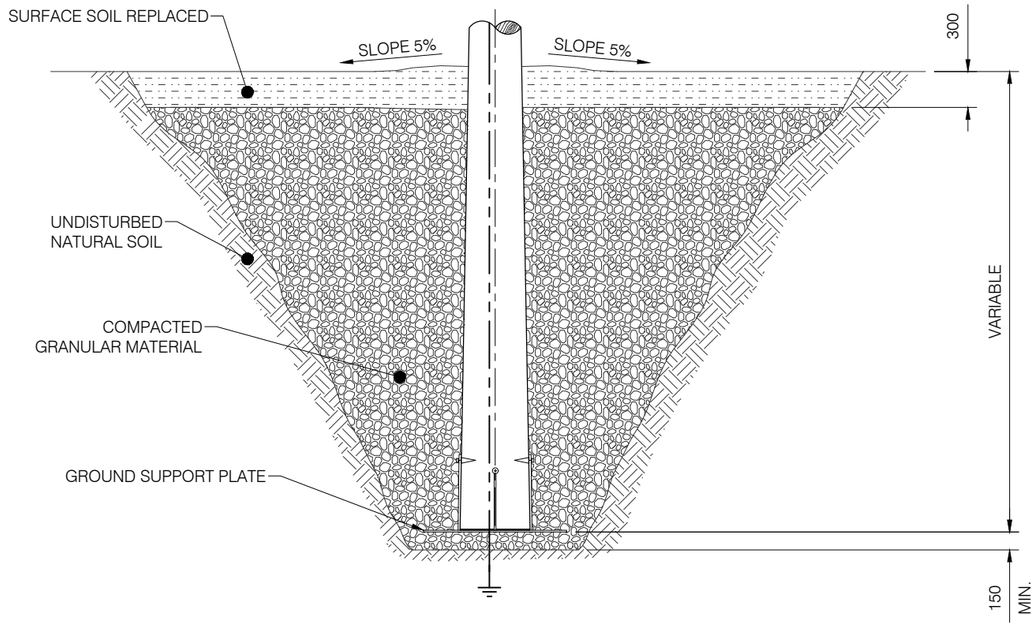
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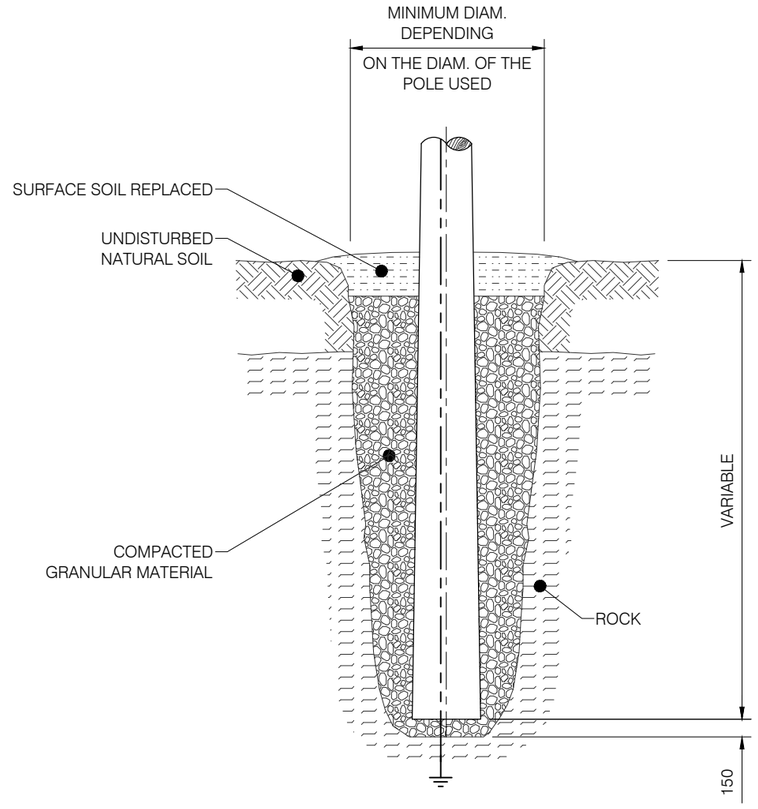
Appendix C: Typical Foundation Drawings

NOTES:

DIMENSIONS ARE IN MILLIMETERS (mm)
UNLESS OTHERWISE INDICATED.



TYPICAL FOUNDATION
FOR OVERBURDEN SOILS



TYPICAL FOUNDATION
FOR ROCK

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION

REV	DESCRIPTION	PREPARED BY	CHECKED BY	DATE
.	-	:	:	.

REVISIONS

CLIENT: PATRIOT BATTERY METALS

DESIGNED BY: M. LÉGER, Eng. OIQ No. 122782	DRAFTED BY: V. MORAIS
PREPARED BY: M. LÉGER, Eng. OIQ No. 122782	CHECKED BY:
SCALE: 1:50	DATE: 2025-06-06

SEAL:

PROJECT: SHAAKICHIUWAANAAN

TITLE: TYPICAL FOUNDATIONS

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Appendix D: Preliminary Schedule

Shaakichiuwaanaan project
120 kV Transmission Line Study

Preliminary Schedule

ID	Phase	Activity	Conditions \ Timing	Month N°																																								
				-	-	-	nov-26	déc-26	janv-27	févr-27	mars-27	avr-27	mai-27	juin-27	juil-27	août-27	sept-27	oct-27	nov-27	déc-27	janv-28	févr-28	mars-28	avr-28	mai-28	juin-28	juil-28	août-28	sept-28	oct-28	nov-28	déc-28	janv-29	févr-29	mars-29	avr-29	mai-29						
				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34						
01	Hydro-Québec	Energy block & Hydro-Québec interconnection	Continue discussions with MEIE, HQ planification study and "Avant-projet." Should be completed before start of procurement phase.																																								
02	Study	Access roads study	Could start immediatly.																																								
03	Prelim. eng.	Final adjustments on routing and repartition for permitting	Could start immediatly and may not be completed before getting environmental study results.																																								
04	Env. and permitting	Environmental and social impact assessment process (for overall mining project)																																									
05		Submit permit application for tree clearing	Submission is conditional to environmental and social evaluation of mine project approval (ministerial decree). Strategic decision to take : deposit permitting for the line interconnection only (not the entire mine.)																																								
06		Lease of Public Land Act Respecting the Lands in the Domain of the State – Sec. 47	For the 315/120 kV Substation																																								
07		Preparation of documents for the Ministerial authorization submission																																									
08		Ministerial authorization submission and approval (wetland, line, substation, environmental authorization)	Submission is conditional to environmental and social evaluation of mine project approval (ministerial decree). Strategic decision to take : deposit permitting for the line interconnection only (not the entire mine.)																																								
09	Prelim. eng.	Preliminary engineering (GAs, structures, foundations, etc.)																																								
10	Detail eng.	Engineering documents For Tender																																								
11		Engineering documents For Construction																																								
12	Construction support	Support from Office																																								
14		Site visits and inspections																																								
15	Procurement	Procurement - Substation equipment (transformers, breakers, etc.)								C.f.T		C.A																											E.o.D.				
16		Procurement - Wood poles												C.f.T		C.A					E.o.D.																					
17		Procurement - Hardware												C.f.T		C.A							E.o.D.																			
18		Procurement - Conductor														C.f.T		C.A						E.o.D.																		
19		Procurement - OPGW and hardware											C.f.T		C.A										E.o.D.																	
20		Procurement - Insulators															C.f.T		C.A																						
21		Procurement - Steel braces and crossarms													C.f.T		C.A							E.o.D.																	
22		Procurement - Wood clearing									C.f.T		C.A																													
23		Procurement - Construction of OHL													C.f.T			C.A																							
24	Env. permitting	Authorization for construction	Deadline for obtaining the authorization																																								
25		Wood clearing and access road construction	Conflict with ministerial authorization									Clearing restricted												Clearing restricted																	
26		Mob./Demob.																																								

Shaakichiuwaanaan project
120 kV Transmission Line Study

Preliminary Schedule

ID	Phase	Activity	Conditions \ Timing	Month N°																																						
				-	-	-	nov-26	déc-26	janv-27	févr-27	mars-27	avr-27	mai-27	juin-27	juil-27	août-27	sept-27	oct-27	nov-27	déc-27	janv-28	févr-28	mars-28	avr-28	mai-28	juin-28	juil-28	août-28	sept-28	oct-28	nov-28	déc-28	janv-29	févr-29	mars-29	avr-29	mai-29					
				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34					
27	Construction	Pole setting and anchors																																								
28		Framing																																							
29		Cable installation																																							
30		Site rehabilitation																																							
31		Substations construction																																							
32		Commissioning																																							

	Engineering/Study
	Environment/Permitting
	Procurement
	Construction
C.f.T	Call for Tender
C.A.	Contract Award
E.o.D.	End of Delivery

APPENDIX

5-1 GEOMECHANICAL DESIGN STUDY OF THE CV5 PEGMATITE



TECHNICAL REPORT

Feasibility-Level Geomechanical Design Study of the CV5 Pegmatite

Shaakichiuwaanaan Project
Québec, Canada

Submitted to	John Drapack P.Eng, PMP Director Studies PMET Resources
Date	February 2 2026
Reference	SHA-001-R-20260202-03

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Feasibility-Level Geomechanical Design Study of the CV5 Pegmatite

Shaakichiuwaanaan Project
Québec, Canada

Client: PMET Resources

February 2, 2025

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Changelog

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Signatures

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Executive Summary

Introduction

PMET Resources is advancing the development of the Shaakichiuwaanaan CV5 Pegmatite Project, a spodumene-bearing lithium deposit located within the Shaakichiuwaanaan Property in the James Bay region of Québec, Canada. The CV5 spodumene pegmatite lies approximately 220 km east of Radisson and 240 km north-northeast of Nemaska. The Shaakichiuwaanaan CV5 open pit (OP) design extends southwest–northeast over 2.8 km in length and up to 480 m in width, with depths up to 200 m, partially under Lake 001. The underground mine (UG) follows the strike of the 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 and extends to approximately 490 m below surface.

Alius Mine Consulting was retained by G Mining Services in September 2024 to provide geomechanical support for the Feasibility Study of this project mine design. This work was intended to build upon the recommendations provided in the Preliminary Economic Assessment of the Project (BBA, 2024e), which relied on the work initially provided by BBA (BBA, 2023a; BBA, 2024d).

The present study encompassed a series of complementary geomechanical investigations, including core logging and laboratory testing based on BBA (2024a), BBA (2024b) and Alius Mine Consulting (2025). The objective of these investigations was to generate a comprehensive geomechanical dataset to support the geomechanical characterization and the development of FS-level design guidelines for both the future open pit and underground mine included in the mine plan.

Mine Plan

The Shaakichiuwaanaan project comprises both an open pit and an underground mine, with a combined production rate of approximately 5.1 Mtpa (14,000 tpd). The open pit will be mined using conventional methods, while the underground operation will employ mechanized long-hole open stoping. Most stopes (90-95%) will follow a transverse orientation in the wider portions of the orebody, with longitudinal stopes (5-10%) applied in narrower sections.

Underground access will be provided via a surface portal and ramp system. Ore will be extracted from a series of sublevels at a targeted spacing of 30 m, with multiple crosscuts developed to access the orebody.

Because underground mining will progress concurrently with open pit operations, interaction between the two is expected and has been assessed in this study.

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Site Characterization

All available information was reviewed and consolidated to update the geotechnical and geomechanical characterization of the soils and rock masses associated with the Project. Key conclusions are summarized below (detailed discussion is provided in Sections 4 and 5):

01. **Soils**
    - → Soils in the vicinity of the open pit have not been thoroughly investigated.
    - → Residual soils are predominantly coarse-grained, with localized thin layers of silt, sand, and gravel.
    - → Given the granular and loose nature of the material, shear strength was assessed using Standard Penetration Tests (SPT) only; no laboratory strength testing was performed.
02. **Rock Units / Lithologies**
    - → Four (4) principal lithologies were the focus of the rock mass characterization: amphibolite (AMP), metasediments (META), undifferentiated ultramafic (UMU), and pegmatite (PEG).
    - → The open pit walls will predominantly expose amphibolite and metasediments, with localized exposures of ultramafic rock and pegmatite.
    - → Underground mine excavations, particularly stope surfaces, are expected to expose pegmatite, with AMP, META, and UMU forming the hanging wall and footwall.
03. **Geomechanical Domains**
    - → Structural data and discontinuity characterization were used to define representative geomechanical domains for the open pit. Sixteen (16) geomechanical domains were identified (Section 6).
    - → For the underground mine, four (4) geomechanical domains were defined, each corresponding to the major lithological units. Additional spatial subdivision was not required due to relatively uniform rock mass conditions.
04. **Discontinuity Orientations**
    - → Discontinuities – including joints, foliation, faults, shears, and contacts – were logged for spacing, roughness, alteration, and infill. Direct shear testing of 21 discontinuities provided peak and residual shear strength parameters.
    - → High-resolution structural data from sixty-five (65) televiewer surveys were incorporated (forty-four for the open pit assessments, twenty-six for the underground assessments).
    - → Spatial variability in joint orientations was evaluated by subdividing the open pit into eight (8) structural zones; metasediments were further subdivided spatially relative to pegmatite intrusion.
    - → Joint sets were interpreted using stereonets for each geomechanical domain.
    - → Foliation within AMP, META, and UMU is one of the dominant structural features, dipping moderately to steeply toward the south–southeast.

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05. Rock Mass Quality

- Rock mass quality was evaluated using fourteen (14) geomechanical drillholes. RMR_{76}, Q', Q, and GSI classifications were applied using first and second quartiles to bracket variability.
- Across all four major lithologies, rock mass conditions range from FAIR to GOOD in both the open pit and underground areas.
- The 315 drillholes logged by Dahrouge Geological exhibit Q' distributions that are consistent with those obtained from the geomechanical logging completed by Alius.

06. Large-Scale Structural Model

- The CV5 Project area has experienced multiple tectonic events; the regional Guyer Group fabric generally dips moderately to steeply south, whereas the CV5 pegmatite is non-concordant and dips steeply north. No significant pegmatite deformation has been observed (refer to Section 2.2.3).
- Six subvertical faults intersect the open pit, with FLT_1025 (NW–SE trending, central pit) showing the most significant displacement.
- Fault characterization is limited by the predominant drillhole orientation.
- Large-scale structural modelling was completed only for the open pit; no model is yet available for the underground mine.

Design Guidelines

Design guidelines to support the updated Feasibility Study was provided in a number of key areas. A summary is provided below.

01. Open Pit – Design Guidelines

Bench-scale and global-scale analyses were performed for ten design sectors using stereographic interpretation, kinematic analysis, bench-scale probabilistic modelling, and 3D numerical modelling. Key findings include:

- Bench face angles range from 40 to 75°, depending on lithology and structural setting.
- Double benching (20 m bench height) is feasible across most sectors due to good rock mass conditions.
- Bench widths ranging from 8.5 to 12 m are required to retain, in most cases, more than 70% of potential spills.
- Resulting inter-ramp angles vary between 32 and 55°, depending on the geomechanical domain and wall location.
- Geotechnical berms (16 m wide) are recommended for walls exceeding 160 m in height, starting 80 m below surface and ending 80 m above pit bottom.
- 3D modelling results confirm factors of safety greater than 1.5 for all pit walls in fresh rock. Localized shallow damage near sharp wall transitions and areas where UMU is exposed should be monitored during operations.
- Inter-ramp angles of 36° are recommended in soils.

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02. Underground – Stope Design and Dilution Estimates

Stope stability was assessed using the Potvin stability chart and ELOS (Equivalent Linear Overbreak/Slough). Key results:

- A uniform stope design is recommended:
  - Transverse stopes 17 mL × 25 mW × 30–36 mH
  - Longitudinal stopes 17 mL × 7 mW × 30–36 mH
- Stope backs should be cablebolted to limit instability and dilution.
- Planed dilution (ELOS) for transverse and longitudinal stope surfaces is:
  - Hanging wall 0.75 m
  - Footwall 1.25 m / 0.50 m
  - Back None (cablebolted)
  - Endwalls None (ore)
  - Backfill 0.50 m

03. Underground – Mine Sequence

- Approximately 90-95% of stopes are transverse, following a primary-secondary sequence; 5-10% are mined using a longitudinal retreat sequence.
- The full-sequence 3D numerical modelling (years 2031–2049) indicates that stress redistribution results primarily in localized yielding near mined stopes, with no large-scale instabilities expected under the planned sequence.
- Continued refinement of the sequence is recommended as the design progresses, particularly in upper mining areas where crown pillar interactions occur.

04. Underground – Backfill

A combination of empirical methods and numerical modelling was used to define cemented paste backfill (CPB) strength:

- CPB strengths between 210–540 kPa are typically required, depending on stope type and sequencing.
- A higher factor of safety (FoS = 2.0) is recommended at the start of mine life, reducing to 1.5 over time as operational experience is gained.
- Underhand stoping requires conservative backfill strength at this stage (1.2–1.6 MPa), and should be minimized, when possible.
- Stopes located within 100 m of exposed open pit walls require a strength of 2.0 MPa (Section 8.10).

05. Underground – Crown Pillar

A surface crown pillar overlies underground workings, beneath Lake 001. A pillar height of 85 m was evaluated using the Scaled Span empirical method and 3D numerical modelling.

- Empirical results estimate an average probability of failure of approximately 1.5%, within the range for Class F (long-term) pillars.
- Numerical modelling indicates the development of minor plastic strain late in the mining sequence (years 2047–2049), suggesting a need for sequencing refinement.

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→ Monitoring systems and operational controls (instrumentation and TARPs) will be required.

Important Regulatory Note: Although the empirical and numerical evaluations indicate that an 85 m pillar can meet stability requirements with appropriate monitoring and sequencing controls, the mine plan does not currently comply with Article 77 of Québec's Regulation Respecting Occupational Health and Safety in Mines (RSSTM). This regulation specifies that when a crown pillar is less than 100 m thick and located beneath a body of water, specific engineering studies must be completed – studies that have not yet been undertaken at this stage of the Project.

Accordingly, the 85 m value should be regarded as a preliminary assessment for mine-planning flexibility, rather than a regulatory-compliant design. Additional studies will be required in the next project phase to establish a fully compliant crown pillar design.

06. Underground – Ground Support Design

Ground support recommendations were developed using empirical design charts, engineering judgment, and in-house analytical tools.

→ Typical lateral development (<7.5 m span) requires 2.4 m rebar (20 mm) on a 1.2 × 1.2 m pattern, with 1.5 m split-sets in the secondary rows, supplemented by 6-gauge, galvanized welded-wire mesh (4 inch aperture).

→ Typical intersections require 6 m cablebolts on a 2 × 2 m pattern.

→ Stope backs and brows require cablebolting (6 m, 2 × 2 m pattern).

→ Portal and major infrastructure support design will need to be completed during detailed engineering.

07. Underground – Portal (Box Cut)

The portal is located where overburden thickness is approximately 5 m, with limited local geomechanical data.

→ Overburden slopes of 2H:1V are recommended.

→ Portal slope angles are controlled by the major joint set (JS1), requiring a 50°dip on the north wall; the south wall may remain steeper (80°).

→ A 10 meter thick crown pillar above the portal was assessed using Scaled Span, showing acceptable PoF (2.4%) with support.

08. Underground – Infrastructures

The stability of underground infrastructures—including haulage drifts, ore passes, draw-points, and access ramps—was evaluated using a 3D *FLAC3D* numerical model incorporating the open pit and underground stopes:

→ Yielding and deformation are limited to areas immediately surrounding stopes, draw-points, and certain development headings.

→ No large-scale deformation zones or global instabilities were identified.

→ Infrastructure stability should be revisited as additional geotechnical and sequencing data become available during detailed engineering.

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09. Open Pit–Underground Interaction

2D limit equilibrium analyses (Slide2) were used to evaluate the stability of pit walls where underground stopes daylight, followed by a validation through 3D numerical modelling:

- → Exposed backfill on the pit wall requires a minimum strength of 2.0 MPa.
- → A strengthened paste zone extending 100 m behind the pit wall is required to achieve a global FoS > 1.5.
- → This corresponds to reinforcing a width of approximately six stopes from the pit–UG interface.

Design Guideline Limitations

Based on the completed analyses and the current level of available information, the following limitations should be recognized when applying the design guidelines.

01. Litho-Structural Model

- → Lithological contacts were modelled based on current interpretations and drilling density; revisions should be expected as mining advances.
- → The location and continuity of major structures (e.g., FLT_1525, diabase dyke) remain uncertain in areas with sparse drilling, particularly near the portal box cut.
- → The structural model covers only the open pit area; structural conditions in the underground mine have not been modelled.
- → The UMU unit is comparatively under-characterized, reducing confidence in design parameters where this lithology is exposed.
- → Foliation orientation varies along the pit and may differ locally from the assumed averaged sets. Although this was accounted for by defining multiple geomechanical domains, real variability may be greater and should be confirmed as exposures develop.

02. Geotechnical Characterization

- → No laboratory testing of soils was performed; strength envelopes for overburden rely solely on SPT results and empirical correlations.
- → Limited drilling exists around the open pit, especially near the portal and pit boundaries, increasing uncertainty in soil thickness and properties.

03. Geomechanical Characterization

- → Joint persistence values are assumed to be 15 m until validated by field measurements (e.g., pit wall mapping, LiDAR).
- → Strength parameters for foliation planes remain uncertain.
- → Joint sets observed on the south wall are considered less restrictive with respect to the design criteria than those on the north wall, due to their wider spacing.
- → Excavations were assumed dry or with minor inflow; groundwater pressures were not explicitly incorporated in classification or modelling.

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04. Open Pit Design

- → Rock mass disturbance factors (blast damage, stress relief) were not incorporated into the stability envelopes.
- → Achieving the recommended bench widths requires controlled drilling and blasting; excessive backbreak may reduce catch bench capacity.
- → Large-scale structures were not included in the numerical models; potential instabilities associated with these features are therefore not captured.
- → Hydrogeological conditions were assumed to have limited impact based on current data; pore pressures were not included in the numerical analyses.
- → Guidelines apply only to the geometries and lithologies evaluated; deviations may require reassessment.

05. Underground Mine Design

- → Large-scale structures were not included in the continuum numerical models, limiting assessment of structurally controlled failure mechanisms.
- → In-situ stress assumptions introduce uncertainty; variations in the major principal stress orientation or magnitude may affect stope stability and crown pillar performance.
- → Hydrogeological effects were not explicitly modelled; water pressures may affect excavation stability.
- → Seismic hazard was not assessed due to the relatively shallow mine depth, but localized stress concentrations may still develop near stopes or in pillars.
- → The portal area lacks geotechnical drilling and requires additional investigation before final design and execution.
- → Stability of large underground infrastructure was not evaluated.
- → The current 85 m crown pillar thickness does not meet the regulatory requirements of Québec's RSSTM Article 77 and must be re-evaluated through the required engineering studies before mining (or development) beneath the water body can commence.

06. Operational Variability

- → Slope performance is highly dependent on drilling accuracy, blasting quality, and scaling practices. Good practices were assumed throughout the assessments.
- → Excessive crest loss or overbreak may reduce effective catch bench widths below design assumptions.
- → Operational deviations may reduce stability and require design modifications.

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Recommendations for Future Work

Based on the analyses completed in this study, the following work is recommended to advance the Project toward detailed engineering and execution.

01. **Litho-Structural Model**
    - → Improve the definition of lithological contacts, particularly those with unfavourable orientations.
    - → Acquire additional structural data (surface mapping, oriented core) in key pit wall sectors.
    - → Update the structural model to better define large-scale structures, especially in areas with limited drill coverage (e.g., underground mine footprint).
    - → Conduct a focused site characterization program for the portal area. This investigation should confirm overburden depth near the box cut and along the ramp axis, as well as the soil and rock slopes to be used in the design.

02. **Geotechnical Characterization**
    - → Develop a site-specific geotechnical model for overburden in the portal and pit-periphery areas.
    - → Plan a geotechnical investigation campaign to increase data density and improve the understanding of soil stratigraphy around the open pit.
    - → Perform laboratory testing on representative soils to establish appropriate shear strength envelopes.

03. **Geomechanical Characterization**
    - → Prioritize data collection for the UMU unit, which currently has the greatest uncertainty.
    - → Expand laboratory testwork (intact strength, shear strength) for UMU, pegmatite, altered zones and foliation planes.

04. **Open Pit Design**
    - → Update the 3D numerical models as litho-structural interpretations are refined.
    - → Reassess structural persistence assumptions (currently 15 m) once exposed pit walls allow direct measurement (e.g., LiDAR scanning).
    - → For Sector F-2, evaluate the observed faults and open joints against the updated structural model to confirm their potential impact on design parameters.
    - → As mining exposes south walls, further assess the actual impact of the north-dipping joint sets on bench-scale wall integrity, and review the assumption that these joints do not significantly influence the expected IRA.

05. **Underground Mine Design**
    - → Update the 3D numerical models as new litho-structural information becomes available.
    - → Incorporate major structures into the underground stability assessments once their geometries are confirmed.

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- → The current 85 m crown pillar thickness does **not** comply with Québec RSSTM Article 77 for mining beneath a water body. Additional engineering studies (hydrogeology, soil mechanics, geomechanics) are legally required before approval. These studies must be submitted to the CNESST before any excavation under the influence of the water body.
- → For the portal design, drill at least three (3) additional drillholes (approx. 90 m total) to confirm the rock mass conditions at the portal location.
- → Consider performing in-situ stress measurements (e.g., Doorstopper testing) early in development to validate the far-field stress tensor.

06. Operational Monitoring and Controls

- → Develop a site-specific Ground Control Management Plan (GCMP) that formalizes geotechnical monitoring requirements, data review frequency, and decision-making protocols.
- → Develop Trigger Action Response Plans (TARPs) for both open pit and underground operations, defining action levels and required responses for changes in displacement, pore pressure, or observed ground conditions.
- → Update the GCMP and TARPs according to operational observations and monitoring results.
- → Implement real-time monitoring systems (prisms, radar, LiDAR) in sectors influenced by major structures or sensitive pit geometries.
- → Implement a hydrogeological monitoring program to track groundwater levels and pore pressures in both the pit and underground mines:
  - Install vibrating-wire piezometers (VWP) in key locations, including pit walls, the crown pillar, and areas where groundwater inflow could influence stability.
  - Establish automated data logging and alarm thresholds to support early detection of abnormal pore-pressure trends.

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1 Introduction

1.1 Project Description and Stage

In 2024, Alius Mine Consulting ("**Alius**") was mandated by G Mining Services ("**GMS**") to complete the feasibility-level geomechanical study ("**FS**") of the Shaakichiuwaanaan CV5 Project ("**Shaakichiuwaanaan**" or "**Project**") for PMET Resources ("**PMET**"), located in the James Bay region of Québec, Canada. The objective of the mandate is to develop geomechanical recommendations to support the NI 43-101 feasibility study prepared by GMS. As part of this work, Alius completed complementary geomechanical core logging, laboratory testing, and the development of design guideline recommendations for both the open pit and underground mine.

The CV5 spodumene pegmatite is part of the Shaakichiuwaanaan Property, which lies approximately 220 km east of Radisson and 240 km north-northeast of Nemaska, in the province of Québec, as shown in Figure 1.

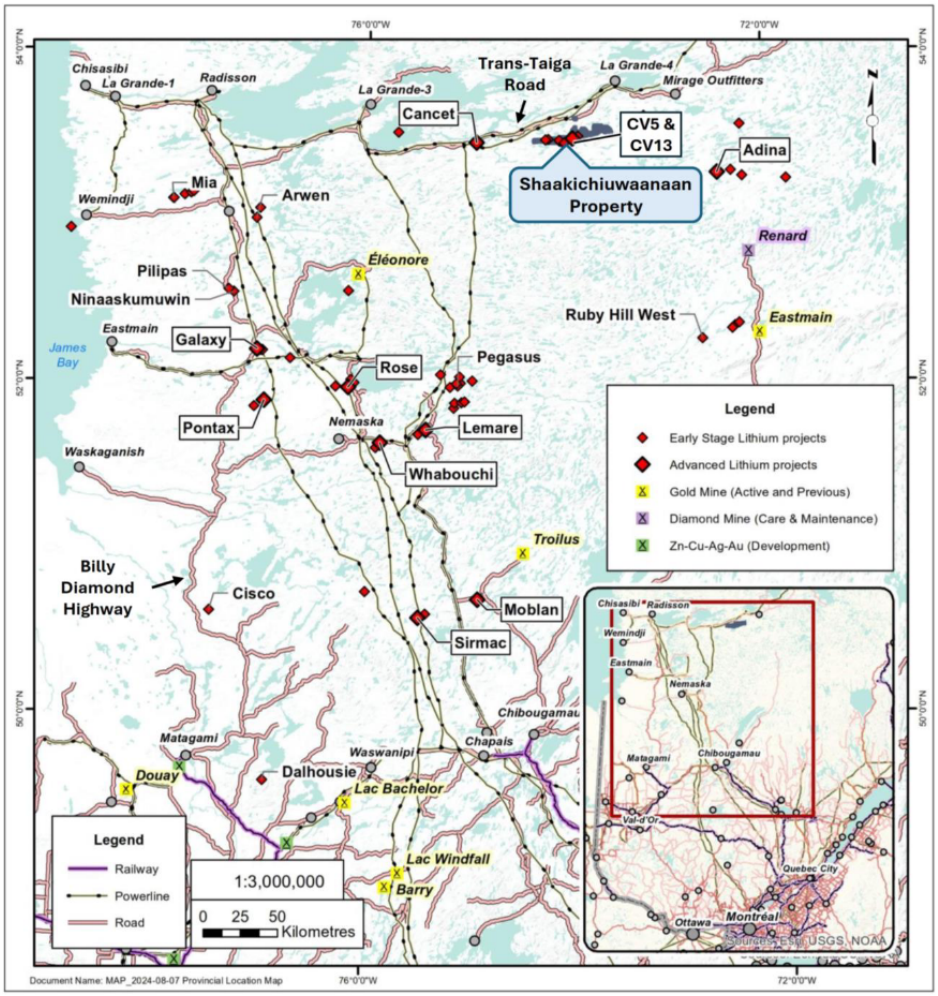


Figure 1. General location of Shaakichiuwaanaan property (BBA, 2024e).

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According to the project development framework presented by Read and Stacey (2009), the present project stage and its general objectives are defined as follows:

> ***Stage 3 – Feasibility Design***
>
> *The objective of this stage is to establish a clear picture of the anticipated costs of mine development and operation. At the completion of the study, alternative interpretations may be possible, but in the view of a 'competent person', these would be unlikely to affect the potential economic viability of the project. To achieve this level of accuracy, overall slope designs in the order of ±5° are necessary.*

The main objectives of this technical report are therefore to provide design criteria for the proposed open pit and underground mines, to identify and assess key geomechanical risks and opportunities that influence the business case and to recommend additional work to further increase confidence in the design.

1.2 Scope of Work

The scope of work includes:

- Planning and executing a complementary geomechanical drilling campaign, conducting laboratory rock testing, and confirming that the available data is sufficient to provide FS-Level recommendations.
- Updating the geotechnical and geomechanical domain definitions.
- Providing FS-Level soil and rock engineering slope design guidelines for the open pit (OP) mine based on kinematic, limit equilibrium analyses, and numerical modelling. Recommendations should cover bench, inter-ramp and overall slope configurations.
- Providing FS-Level rock engineering design guidelines for the underground (UG) mine based on empirical methods and numerical modelling, covering:
  - Stope design and dilution
  - Backfill requirements
  - Mining sequence
  - Ground support
  - Pillars design
  - Main portal
  - Underground infrastructures
  - Interaction with the open pit

Design recommendations for both the open pit and underground mines are presented in this report. Details of the complementary geomechanical investigation are provided in a separate factual report (Alius Mine Consulting, 2025).

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1.3 Available Information

The study relies on the following sources of information:

01. **Geological Model**
    - → Lithological units include amphibolite (composed of Amphibolite 01 to 03), the metasediments group, the ultramafic undifferentiated and pegmatite (composed of the solids named Pegmatite, Pegmatite CV and Pegmatite Feldspar). Smaller diabase and iron formation units (Marbot and garnet + Fe-ox) are also present.
    - → Latest lithological model received on January 29, 2025.
    - → CV5 structural model (i.e., faults) received on January 27, 2025 (modelled only in the open pit area).
    - → RQD wireframes received on December 23, 2024.
02. **Geotechnical and Geomechanical Investigations and Characterization**
    - → Geotechnical:
        - Phase 2 investigation campaign (BBA, 2024c)
    - → Geomechanical:
        - PEA-level geomechanical characterization (BBA, 2023a).
        - Additional geomechanical investigation campaigns (BBA, 2024a; BBA, 2024b).
03. **Open Pit Design**
    - → PEA pit design (Figure 3a) for design guideline assessment.
    - → Final FS pit design (Figure 3b) received March 17, 2025 , used for numerical modelling.
04. **Underground Mine Design**
    - → PEA underground mine design (Figure 4a) used for design guideline assessment.
    - → Intermediate stope design (Figure 4b) based on the July 2024 block model (inferred and indicated resources), excluding stopes south of the main zone.
    - → Final FS underground design (Figure 5) received March 5, 2025, used for numerical modelling.

Unless stated otherwise, the design guidelines presented in this report are based on PEA or intermediate designs, while numerical modelling incorporates the final FS designs that include updated slope and stope recommendations.

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2 General Context and Site Conditions

This section provides a site description and a description of the geological conditions.

2.1 Site Description

The Project site is located in a sub-arctic climate. Climate data from La Grande Rivière Airport (LG2) near Radisson, Quebec, about 220 km west of the site, indicate annual temperatures ranging from -27 to 20°C, with occasional extremes of -35 and 26°C. Rainfall peaks in late August and early September with an average monthly maximum of approximately 90 mm, while snow cover typically persists from mid-October to late May (BBA, 2024e).

The terrain in the area of Shaakichiuwaanaan consists of gently undulating, forested hills interspersed with streams and wetlands, with elevations ranging from approximately 260 to 350 m, characteristic of the James Bay Region. The future Project area is partially located over a lake and is surrounded by numerous waterbodies, as shown in Figure 2.

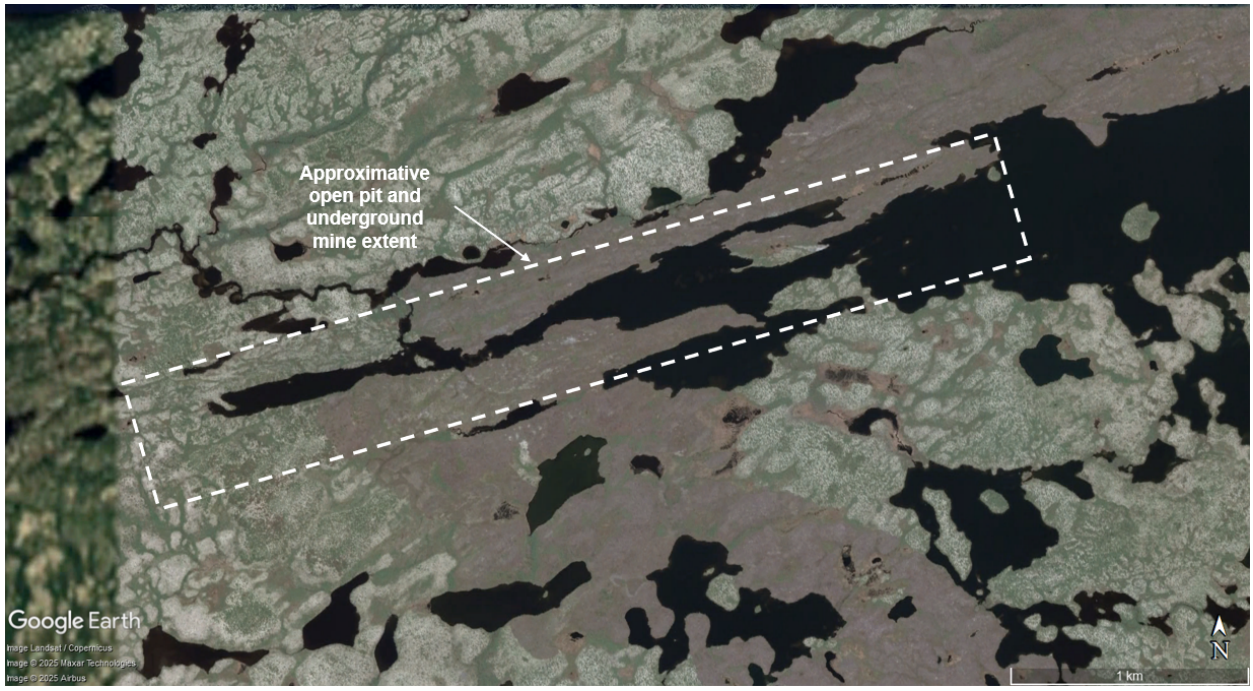


Figure 2. Approximate extent of the open pit and underground mine with the surrounding lakes.

The Project deposit consists of a large main pegmatite dyke (CV5), accompanied by several smaller dykes striking approximately east–west. Planned mining operations will incorporate both conventional open pit mining and mechanized long-hole open stoping for the underground 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.

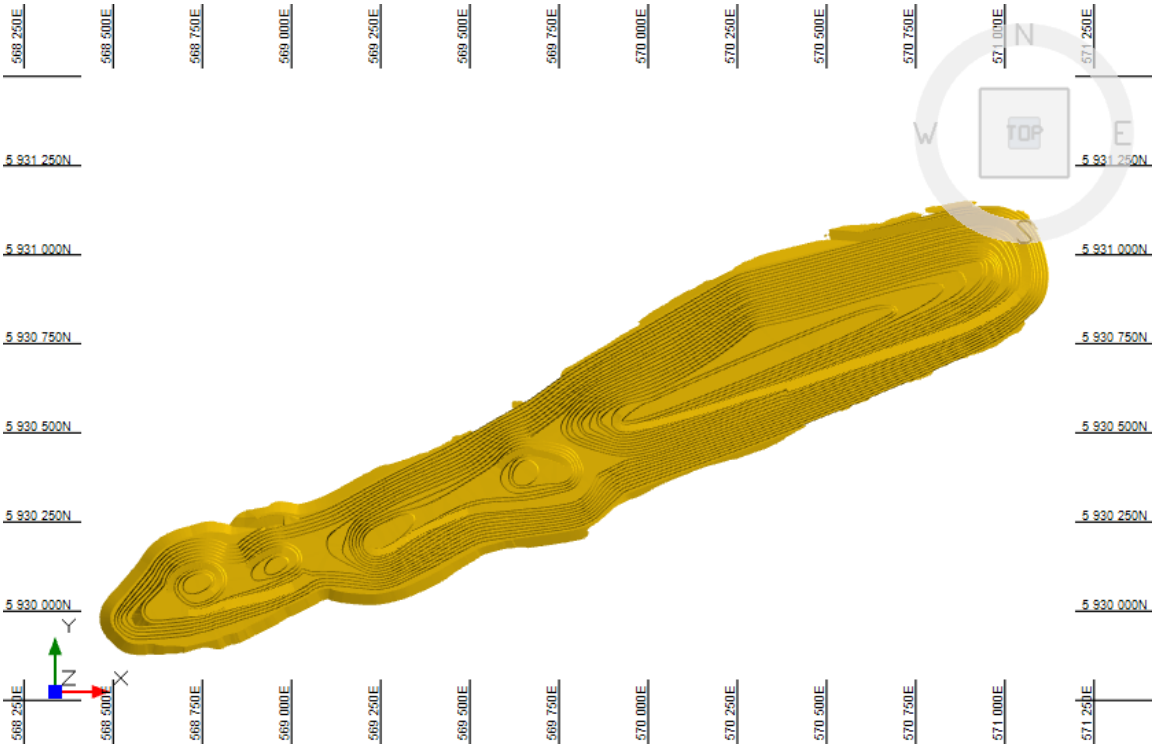
According to the PEA design shown in Figure 3a, the open pit orientation is southwest–northeast

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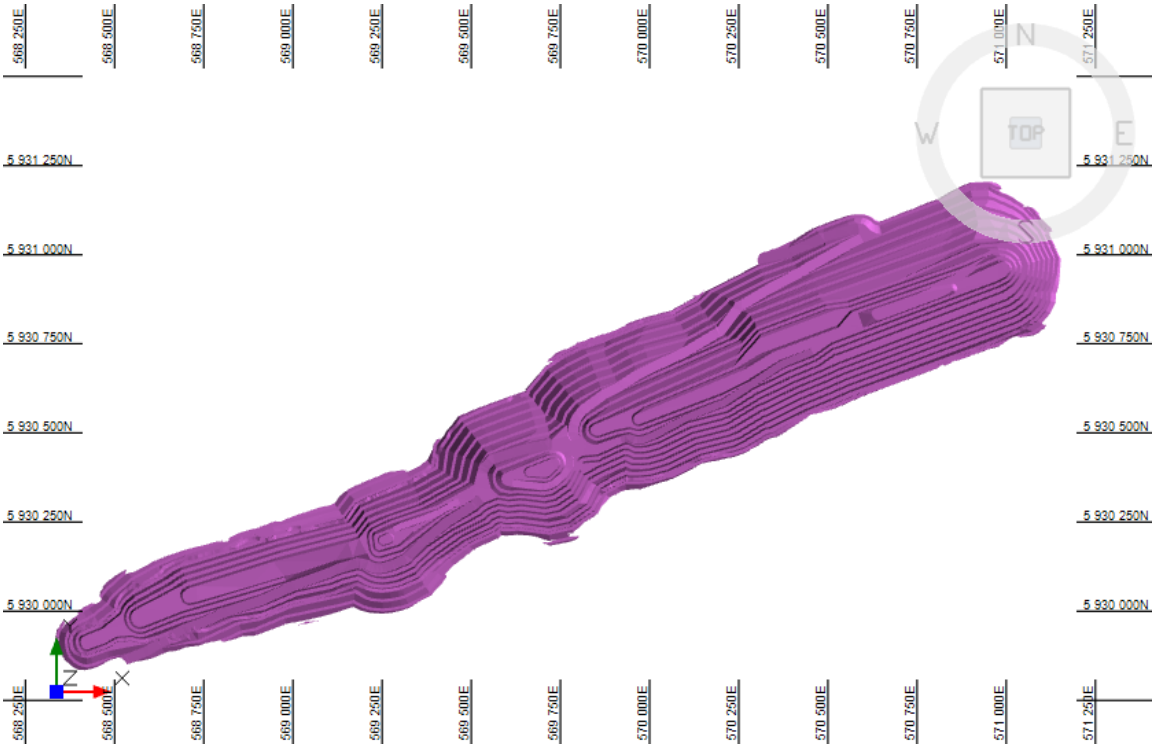
and measures approximately 2.8 km in length and up to 480 m in width. The southwest zone of the pit is about 70–150 m deep, while the northeast zone is wider and deeper, reaching a depth of approximately 200 m. As previously noted, a significant portion of the orebody lies beneath Lake 001 and will therefore require dewatering to enable open pit mining. Given that surface water generally flows westward, water diversion and management will be important aspects of the Project.

According to the PEA underground design shown in Figure 4a, the underground mine is located at the northeast end of the pit and extends to a depth of approximately 490 m below surface.

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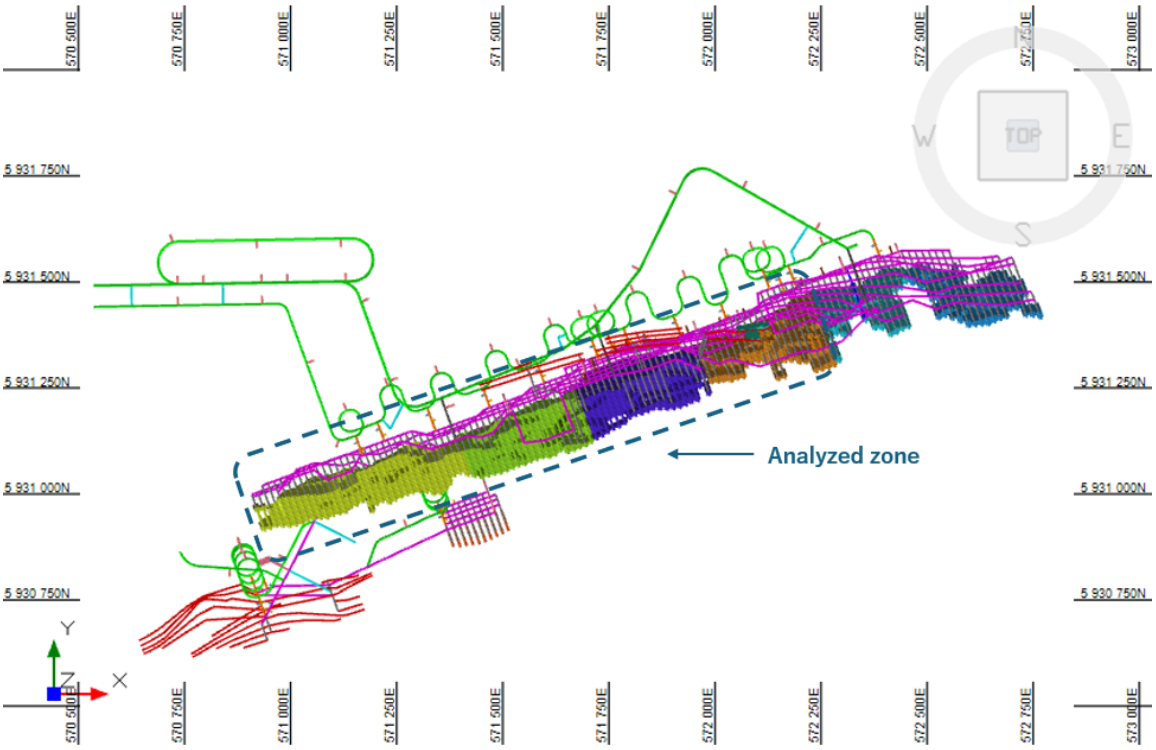
(a) PEA design.



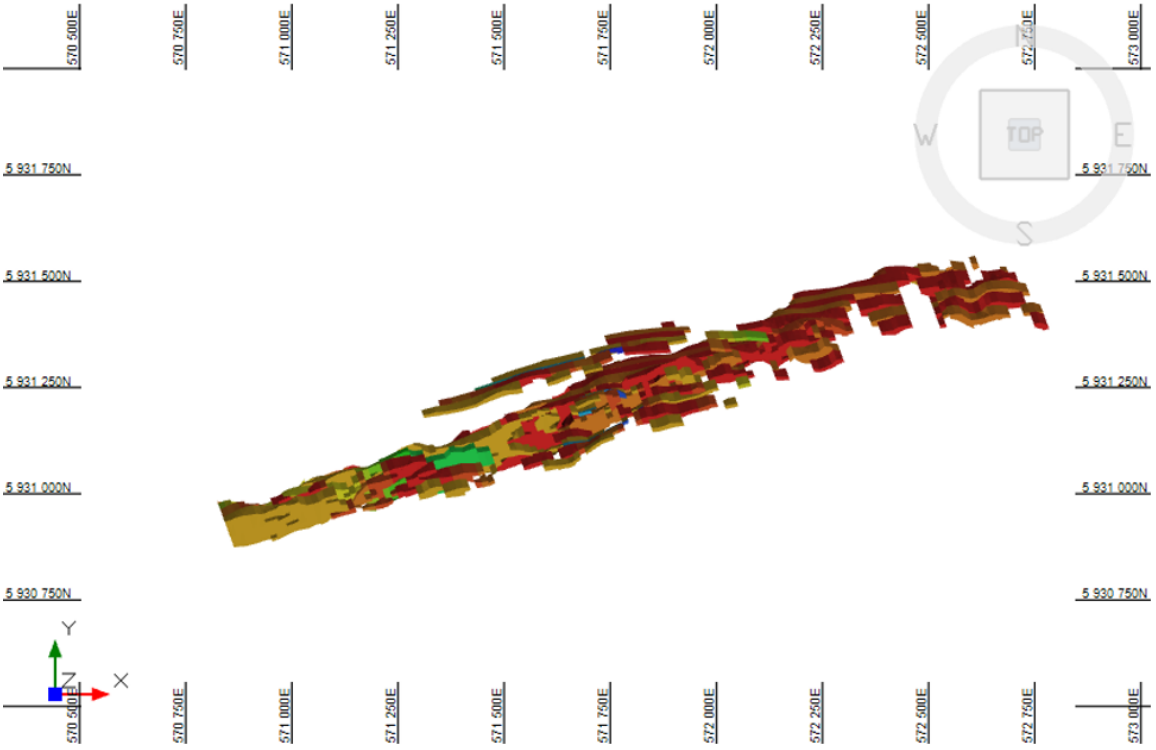
(b) FS design.

Figure 3. Open pit designs as provided by GMS – Plan views.

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(a) PEA design.



(b) Intermediary design (based on July 2024 block model).

Figure 4. Underground mine designs as provided by GMS – Plan views.

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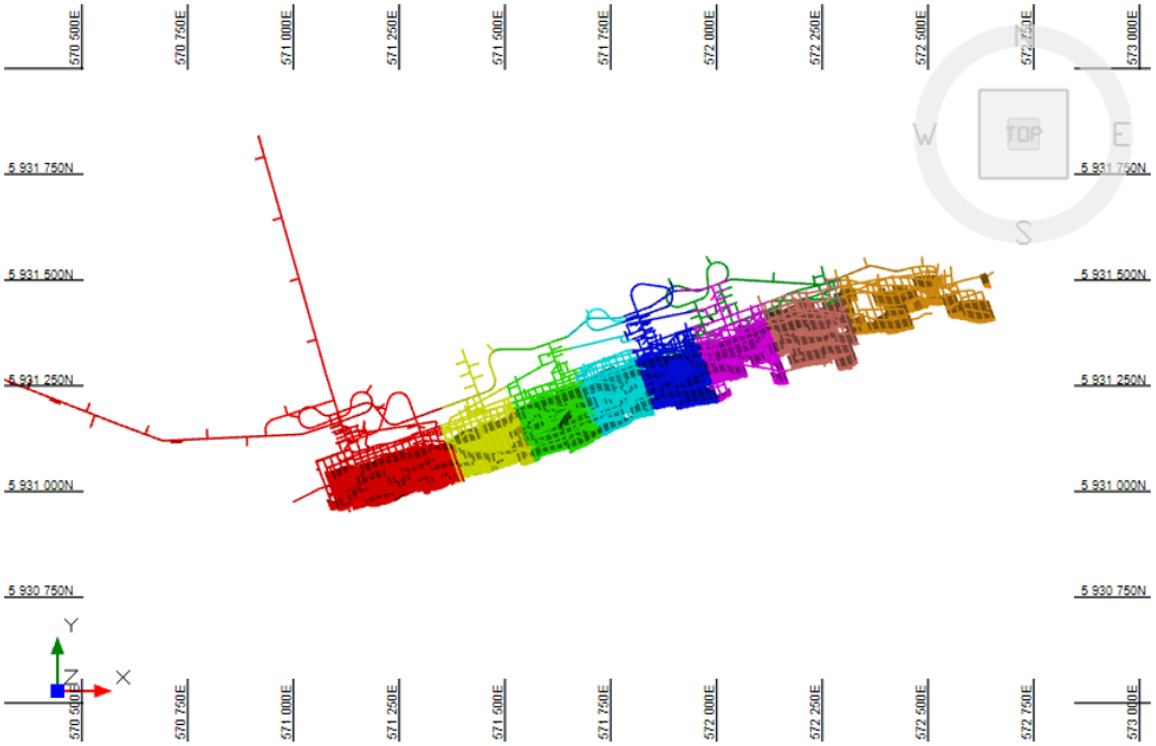


Figure 5. Underground mine designs as provided by GMS – FS design.

2.2 Geological Conditions

2.2.1 Regional Geology

Regionally, the CV5 Pegmatite orebody is located within the La Grande subprovince of the Archean Superior Province in the Canadian Shield, as shown in Figure 6 (BBA, 2024e). The Project area lies within the La Grande River Greenstone Belt, a volcano-sedimentary assemblage dominated by basaltic to felsic volcanic rocks interlayered with metasedimentary units that have been metamorphosed to amphibolite facies.

The La Grande subprovince trends parallel to the Wemindji–Caniapiscau structural corridor and is bounded to the south by the Opinaca subprovince. The transition zone between La Grande and Opinaca hosts several spodumene-bearing pegmatites, including the CV5 Pegmatite.

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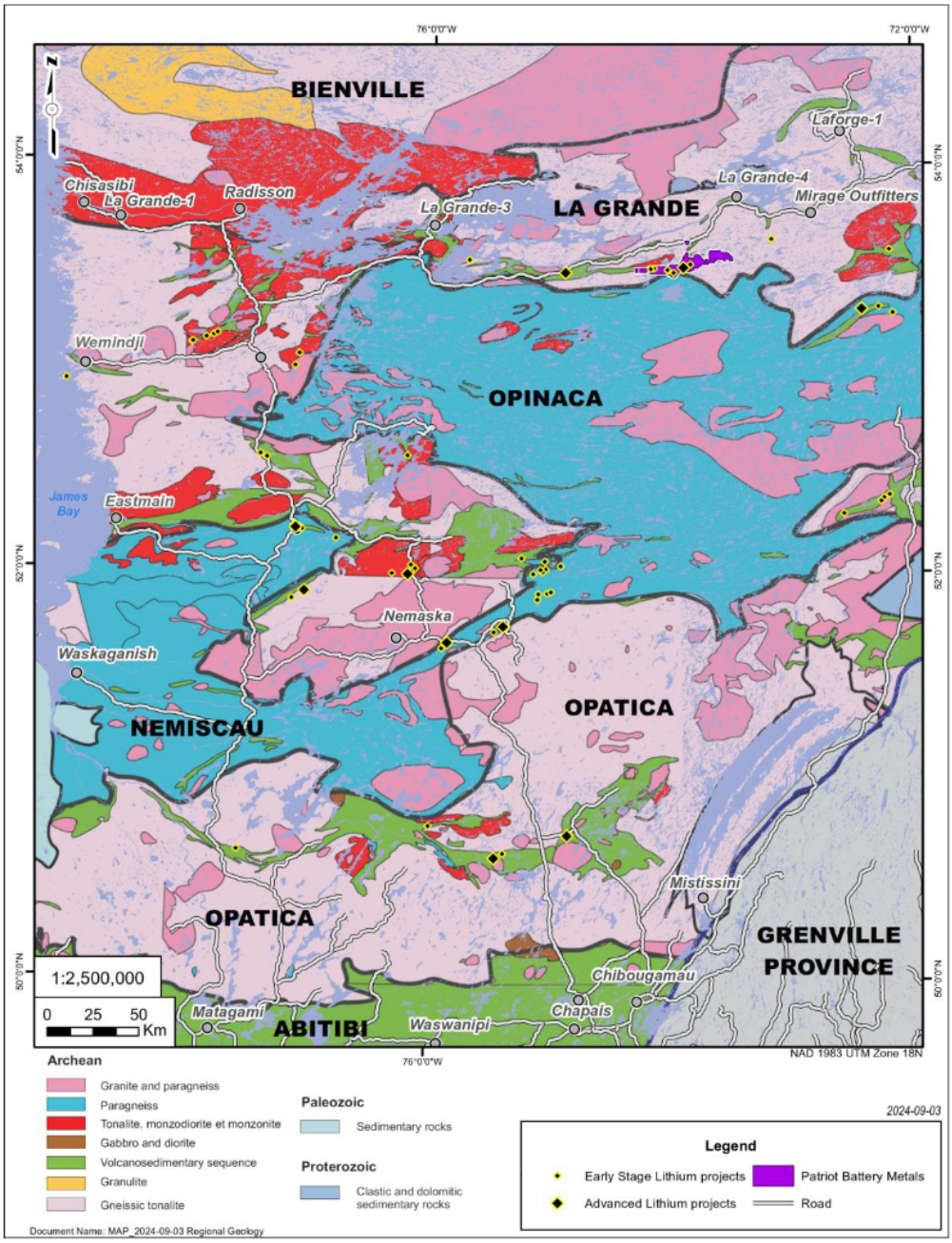


Figure 6. Regional geology surrounding the Shaakichiuwaanaan Project (BBA, 2024e).

2.2.2 Local Geology

Locally, the CV5 Pegmatite orebody is hosted within the Guyer Group of the Lac Guyer Greenstone Belt, part of the broader La Grande River Greenstone Belt (BBA, 2024e). The Guyer Group consists of a volcano-sedimentary sequence dominated by amphibolite, metasedimentary rocks, and intermediate to mafic volcanic rocks, along with minor ultramafic units such as peridotite, pyroxenite, and komatiite, and felsic volcanic tuffs. These units generally trend east–west

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and dip moderately to steeply to the south. The Guyer Group is bounded to the north by the Magin Formation and to the south by tonalite, granodiorite, and diorite intrusions, as well as the metasedimentary rocks of the Marbot Group. Proterozoic gabbroic dykes of the Lac Spirt and Senneterre swarms locally intrude these units.

The CV5 Pegmatite lies within the CV Lithium Pegmatite Trend, an approximately 1-km-wide and more than 25-km-long east–west corridor hosting lithium-bearing Li-Cs-Ta (LCT) pegmatites. The CV5 dyke has an approximate strike length of 4.6 km and is primarily hosted within amphibolite and metasedimentary rocks. At surface, the pegmatite appears either as high-relief “whale back” outcrops or as low-relief flat exposures.

Based on the January 2025 lithological model, the codes of the main lithological units are presented in Table 1, as well as the lithology-coding scheme used throughout this report. Note that when the expression "ultramafic" is used in this report, it refers specifically to the ultramafic undifferentiated group.

Table 1. Codes for main lithologies.

Code	Description
AMP	Amphibolite
DIA	Diabase
IRONF	Iron formation
META	Metasediments
OVB	Overburden
PEG	The pegmatite and the spodumene pegmatite are modelled separately but are grouped in this report.
UMU	Undifferentiated ultramafic ("ultramafic")

As illustrated in Figure 7, the ultimate pit walls primarily expose amphibolite and metasediments. Ultramafic rocks are exposed locally, while pegmatite occurrences are expected near the pit bottom and in isolated sections of the walls. Underground stopes are mainly within the pegmatite, but various lithologies may be present within the immediate hanging wall and footwall. The geomechanical characterization, therefore, focuses on the four principal units: AMP, META, UMU, and PEG.

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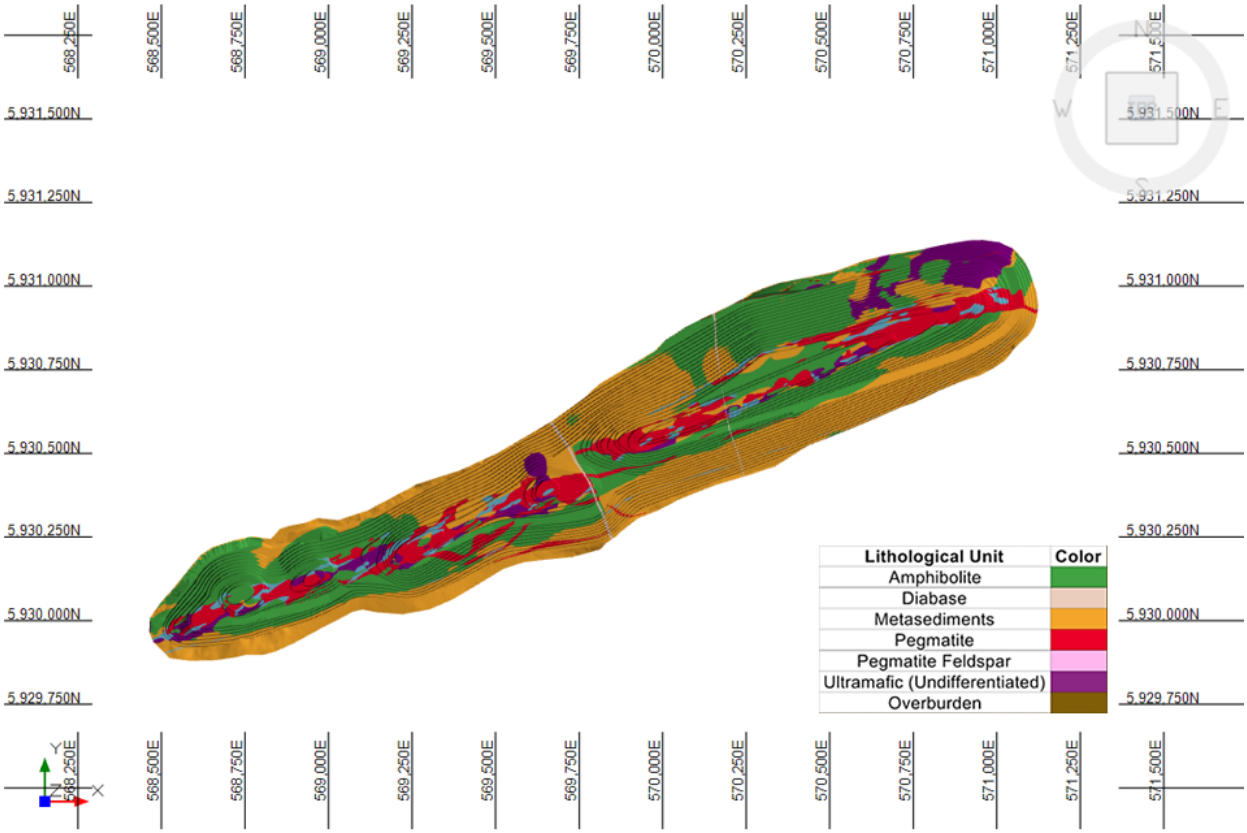


Figure 7. Ultimate pit shell coloured according to lithologies exposed in walls.

2.2.3 Structural Geology

The Shaakichiuwaanaan Project area has undergone multiple tectonic events that produced major east–west–trending shear zones, including the Pontois-Sud Fault to the north and the Nochet Shear Zone to the south. Within this structural framework, the apparent fabric of the metasediments (Guyer Group) generally dip moderately to steeply south. The CV5 pegmatite is non-concordant to the regional foliation and dips steeply to the north. No significant deformation of the pegmatite has been identified to date, aside from minor folding observed elsewhere on the property (BBA, 2024e).

In the geological model provided by GMS, six interpreted sub-vertical faults intersect the open pit, as shown in Figure 8. Among these, fault FLT_1025 – a northwest–southeast-trending structure passing through the central part of the pit – shows measurable displacement within the modelled lithologies and is the only fault exhibiting a significant interpreted offset.

At this stage, it is difficult to accurately define the position and extent of several structures, as many of their traces are subparallel to the orientation of the geological drillholes. This geometry reduces the likelihood that faults intersect the core, potentially leading to underrepresentation or uncertainty in fault characterization. Faults oriented approximately perpendicular to the pit walls should therefore be interpreted with caution and closely monitored during pit development. This

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limitation does not apply to faults oriented perpendicular to the drillhole traces.

Currently, structural modelling covers only the open pit area; no structures have yet been modelled for the underground mine. Developing a more detailed structural model will be important to ensure that no major fault zones adversely affect open pit or underground operations.

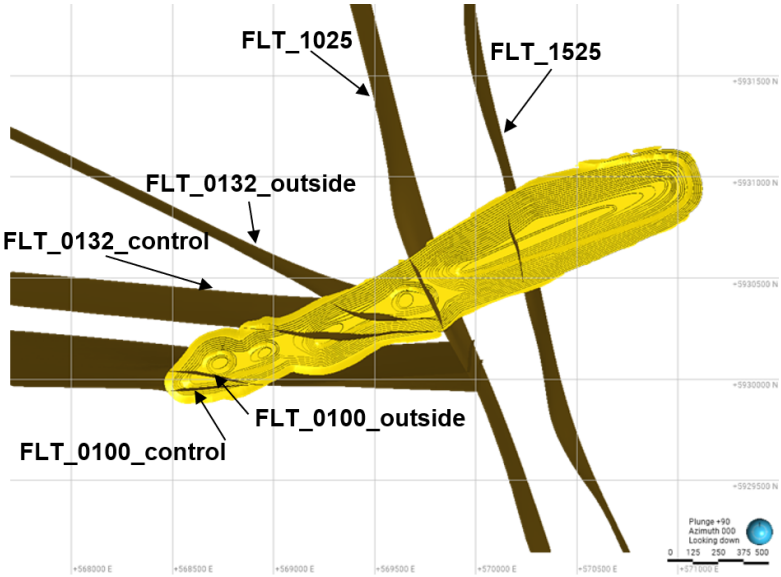


Figure 8. Location of the six sub-vertical faults intersecting the open pit.

2.3 Hydrogeology

The hydrogeology baseline for the Project is established in BBA (2023b). This study focuses on the footprint of the planned open pit mine at CV5, which at the time (2023), corresponded to a pit encompassing both the current feasibility study open pit and underground mine designs.

The report covers the hydraulic conductivity, groundwater-level, and groundwater-quality for three main hydrostratigraphic units: overburden, lake sediments, and bedrock. The characteristics of each unit presented by BBA (2023b) are the following:

- Overburden: Thin overburden deposits form the vadose zone. Given the nature of the materials encountered (largely surface glacial deposits/unconsolidated till composed of boulders and cobbles in a matrix of sand, silt, and gravel), the loose surface deposits are assumed to have high permeability.
- Lake sediments: This unit was not characterized during the mentioned study.
- Bedrock: Bedrock constitutes the main aquifer at the site, characterized as an unconfined aquifer within fractured rock. The rock mass generally exhibits low-intensity fracturing, with an average hydraulic conductivity of 3.2×10^{-7} m/s, indicative of medium permeability. Groundwater in the northern and southern parts of the (2023) planned pit generally flows toward the lake located at the center of the pit, whereas in the western portion of the site, groundwater flows northwest toward the lake outlet.

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3 Investigation Campaigns

To support the feasibility study, the available investigations are grouped into two main categories: geotechnical investigations for soil characterization and geomechanical investigations for rock mass characterization. Both investigation types are summarized in the following subsections.

3.1 Geotechnical Programs

The geotechnical investigation program was prepared by BBA, and soil sample laboratory testing was undertaken by Englobe in 2024.

3.1.1 Fieldwork

A geotechnical field campaign was conducted during the summer and fall of 2024 to support soil characterization and hydrogeological investigations (BBA, 2024c). The program, initiated and logged by BBA, involved the description and classification of sixty-seven (67) drillholes. Among these, eleven (11) drillholes located within 250 m of the PEA open pit footprint were identified as representative of the regional overburden conditions and were considered as inputs for the process to determine the geotechnical design guidelines. The locations of the selected drillholes are shown in Figure 9, and their details are provided in Table 2.

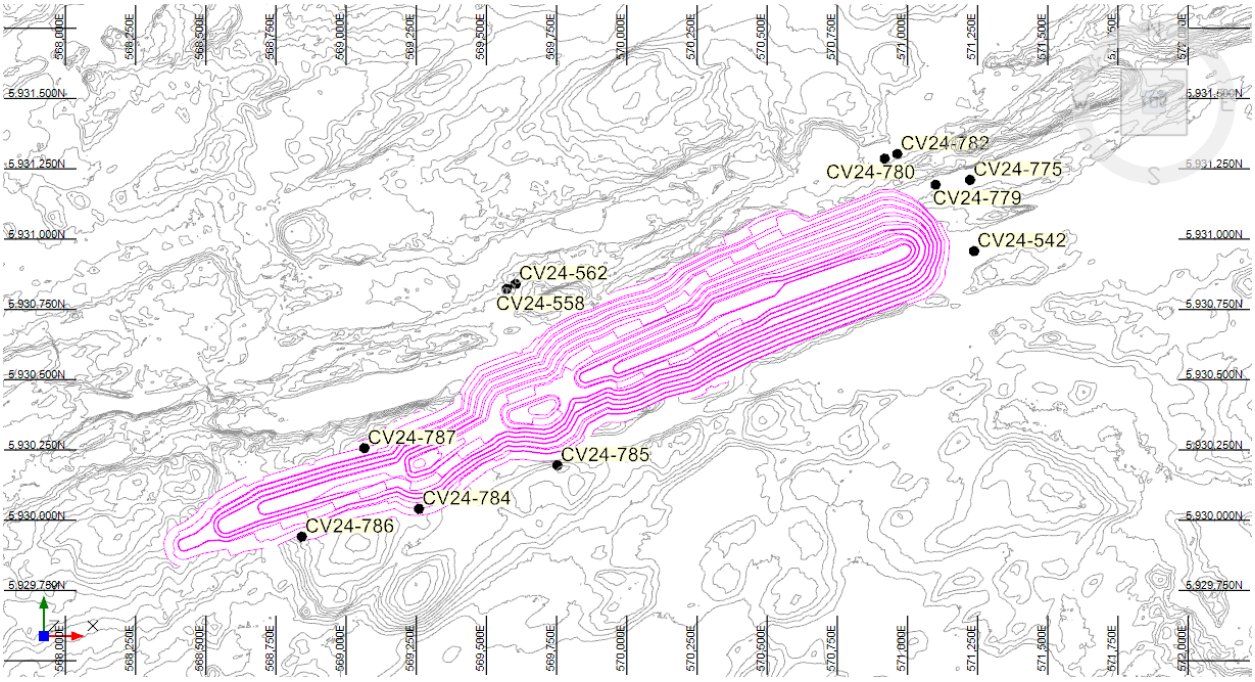


Figure 9. Plan view of the FS open pit and soil geotechnical drillholes.

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Table 2. Geotechnical drillhole details.

DH ID	Collar Coordinates			Core	Azimuth	Plunge	Length
(CV24–)	**East** (m)	**North** (m)	**Elev.** (m)	**Size**	(°)	(°)	(m)
542	571 235.6	5 930 959.1	94.86	NQ	0	-90	11.3
558	569 570.6	5 930 824.8	94.86	HQ	0	-90	11.1
562	569 602.1	5 930 842.2	112.10	HQ	0	-90	11.0
775	571 221.3	5 931 212.4	102.07	NTW	0	-90	11.0
779	571 098.5	5 931 194.9	81.42	NTW	0	-90	13.0
780	570 917.1	5 931 288.1	140.26	NTW	0	-90	9.1
782	570 962.2	5 931 304.3	132.56	HQ	0	-90	10.2
784	569 258.0	5 930 042.0	75.38	HQ	0	-90	32.0
785	569 751.0	5 930 197.0	79.69	NQ	0	-90	3.8
786	568 840.8	5 929 943.0	115.20	NQ	0	-90	25.4
787	569 063.2	5 930 258.0	106.99	NQ	0	-90	18.5

Codes **DH ID** Identification of drillhole.
Notes Coordinates in NAD83 / UTM zone 18N.
HQ diameter: 63.5mm; NQ diameter: 47.6mm; NTW diameter: 64.0mm.

The field program consisted of conducting Standard Penetration Tests (SPT) along each drillhole until bedrock was encountered, as well as visually classifying the materials recovered from the split-spoon samplers. Soil and rock samples were also collected during the campaign for subsequent laboratory testing.

3.1.2 Laboratory Testwork

Laboratory testing of soil samples from the selected drillholes was limited to grain-size distribution, sedimentation, and water content analyses. These tests were primarily performed to validate the material classification.

3.2 Geomechanical Programs

In 2024, several logging campaigns were conducted to acquire data to support the development of a geomechanical model suitable for both pit slope and underground stability analyses. One program, planned and logged by BBA, was carried out early during the feasibility study phase (BBA, 2024a; BBA, 2024b). The most recent geomechanical investigation program was designed and executed by Alius, with field logging performed by Alius personnel (Alius Mine Consulting, 2025). The objective of the Alius complementary investigation program was to address data gaps identified in the previous rock mass characterization.

Laboratory testing of samples collected during the Alius geomechanical logging campaign was completed at the Alius Rock Mechanics Laboratory in Québec City (Québec), except for direct shear tests, which were performed by WSP. Samples collected by BBA during their campaign were tested at the Queen's University Rock Mechanics Laboratory in Kingston (Ontario), except for direct shear

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tests, which were also performed by WSP.

A historical database of drillholes drilled between 2021 and 2024 is also available. The geology team of PMET and Dahrouge Geological Consulting Ltd. ("**Dahrouge**") logged this core and collected high-level geomechanical data. This database comprises a geomechanical quick-log: RQD, estimated joint set number, joint roughness and alteration, and field intact rock strength, as determined by geological hammer testing. Alius did not implement QA/QC procedures to thoroughly review this historical database. Nevertheless, these data were used for comparison purposes with the results of the 2024 geomechanical investigations.

3.2.1 Fieldwork

Since the Shaakichiuwaanaan Project encompasses both open-pit and underground mining operations, geomechanical drilling targeted specific areas accordingly, based on the PEA pit shell and PEA stope extents at the time. The geomechanical field investigations were carried out in three successive phases:

01. **Phase 1**
    - → By BBA in April 2024
    - → Six (6) drillholes were logged: CV24-447, CV24-456, CV24-459, CV24-469, CV24-479 and CV24-496
    - → Details in BBA (2024a)
02. **Phase 2**
    - → By BBA in September 2024
    - → Four (4) drillholes were logged: CV24-663, CV24-679, CV24-684 and CV24-689
    - → Details in BBA (2024b)
03. **Phase 3**
    - → By Alius in October 2024
    - → Four (4) drillholes were logged: CV24-769, CV24-777, CV24-781, and CV24-783
    - → Details in Alius Mine Consulting (2025)

Field data collection consisted primarily of geomechanical core logging, rock core sampling, intact rock strength estimation using a geological hammer and on-site point load testing (PLT), and the characterization of discontinuities intersecting the NQ and NQ3 core recovered from the fourteen (14) non-oriented drillholes. The identification of natural versus mechanical joints was carried out directly at the drill rig by the Dahrouge geological team, using split-tube core handling to ensure optimal differentiation. During geomechanical logging, this identification ensured that data were collected exclusively from natural joints. Faults and natural rubble or broken core were not marked; therefore, their inclusion in the geomechanical log is based on the loggers' interpretation.

Details of these drillholes are provided in Table 3. Note that the logged intervals do not always correspond to the full drilled length, as logging focused on the most relevant core sections and was not consistently performed over the entire hole.

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During the core logging campaigns, the core was not oriented; therefore, no manual joint orientation measurements were taken. Instead, structural information was obtained through Televiewer surveys carried out by DGI Geoscience Inc. A total of sixty-five (65) Televiewer (TV) surveys – both optical (OTV) and acoustic (ATV) – were completed, primarily in geological drillholes. Additional details are provided in Section 5.1.2. Figure 10 presents the locations of the drillholes and associated downhole Televiewer surveys relative to the FS open-pit and underground mine designs.

Table 3. Geomechanical drillhole details.

DDH ID	**Collar Coordinates**			**Core**	**Azimuth**	**Plunge**	**Length**
(CV24–)	**E** (m)	**N** (m)	**Z** (m)	**Size**	(°)	(°)	(m)
447	571 152.3	5 931 101.1	375.1	NQ	130	-55	308.0
456	570 174.5	5 930 836.0	378.3	NQ	200	-55	456.9
459	571 508.9	5 930 921.8	374.6	NQ	296	-60	314.1
469	571 572.0	5 930 953.4	373.2	NQ	40	-60	407.0
479	570 355.0	5 930 476.9	379.2	NQ	16	-55	467.1
496	571 529.1	5 931 440.2	390.7	NQ	113	-55	509.0
663	570 784.8	5 930 905.2	371.8	NQ	160	-60	215.0
679	570 814.8	5 931 002.4	372.0	NQ	340	-60	124.9
684	570 932.4	5 930 996.7	371.9	NQ	27	-60	161.0
689	569 705.4	5 930 476.0	380.2	NQ	11	-51	167.0
769	571 579.6	5 931 234.4	374.9	NQ3	170	-68	374.0
777	568 849.4	5 930 131.6	395.1	NQ3	340	-75	101.1
781	569 283.5	5 930 125.5	388.4	NQ3	330	-84	200.1
783	571 927.4	5 931 447.3	377.6	NQ3	145	-68	416.0

Codes **DDH ID** Identification of diamond drillhole.
Notes Coordinates in NAD83 / UTM zone 18N.
NQ Diameter: 47.6mm; NQ3 diameter: 45.0mm.

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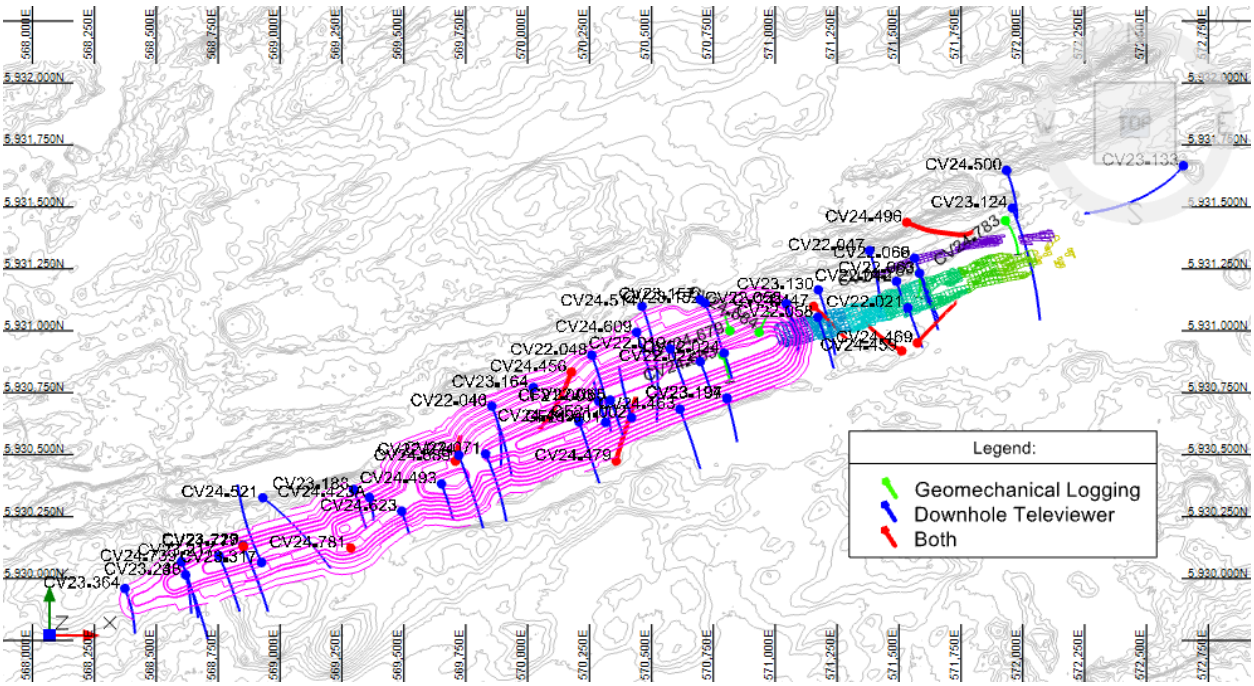


Figure 10. Geomechanical drillholes and televiewer surveys location on the FS open pit and underground mine design.

The geomechanical core logging includes the collection of the following data:

- Interval logging (position, lithology, total core recovery (TCR), rock quality designation (RQD), weathering and alteration, feature count, joint set number and field intact rock strength)
- Discontinuity logging (depth, type, aperture, alteration, roughness and condition)
- Punctual data (point load tests)

During the 2024 geomechanical campaigns, the total logged core length is approximately 3 575 m. Table 4 presents the logged length per hole, along with the corresponding factual reports covering each hole.

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Table 4. Geomechanical logged lengths per hole.

DDH ID (CV24–)	Logged Intervals			Source
	Depth From (m)	**Depth To** (m)	**Length** (m)	
447	11	308	297	BBA (2024a)
456	6	457	451	BBA (2024a)
459	5	314	309	BBA (2024a)
469	8	407	399	BBA (2024a)
479	5	467	462	BBA (2024a)
496	8	509	501	BBA (2024a)
663	8	215	207	BBA (2024b)
679	8	125	117	BBA (2024b)
684	8	161	153	BBA (2024b)
689	14	167	153	BBA (2024b)
769	92	318	226	Alius Mine Consulting (2025)
777	25	101	76	Alius Mine Consulting (2025)
781	22	167	145	Alius Mine Consulting (2025)
783	257	413	156	Alius Mine Consulting (2025)

Codes **DDH ID** Identification of diamond drillhole.

The four main lithologies logged are amphibolite, pegmatite, metasediments, and ultramafic, which are also the four main lithologies exposed on the pit and stope walls. Table 5 and Figure 11 present the logged length per lithology type. Figure 12 summarizes the lithologies intersected in each logged hole.

Table 5. Logged lengths per lithology type.

Lithology	Logged Core	
	Total Length (m)	**Relative Frequency** (%)
Metasediments	1 555.5	43.5
Amphibolite	1 319.4	36.9
Pegmatite	456.2	12.8
Ultramafic	169.2	4.7
Iron formation	63.3	1.8
Diabase	9.1	0.3
Overburden	3.0	0.1
Total	**3 575.7**	**100**

Notes Discrepancies in calculations might occur due to round-off errors.

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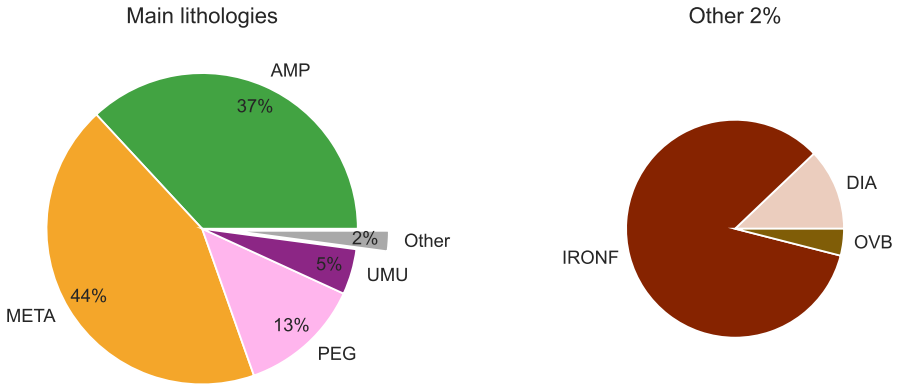


Figure 11. Logged lithology distribution.

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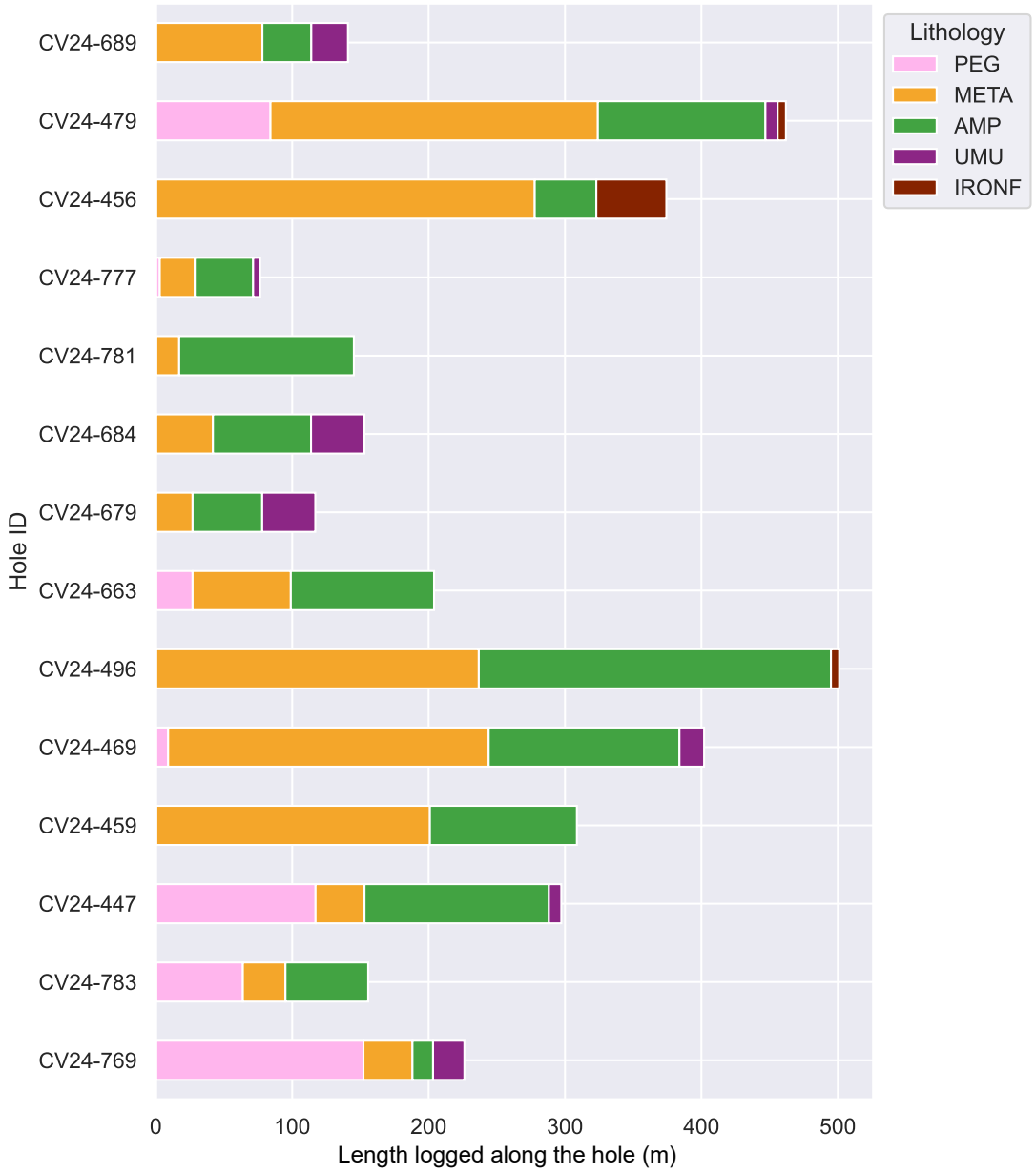


Figure 12. Lithology logged length per hole.

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3.2.2 Laboratory Testwork

Representative samples of each lithology were collected from the geomechanical drillholes by BBA and Alius personnel during core logging for laboratory testing on intact rock and open discontinuities. The program included axial and diametral point-load testing (PLT), Brazilian tensile strength (BTS) tests, uniaxial compressive strength (UCS) tests, triaxial compressive strength (TCS) tests on rock samples and direct shear tests (DST) on open discontinuities.

The factual reports of BBA (2024a), BBA (2024b), and Alius Mine Consulting (2025) present the complete laboratory results. The total number of tests available, by type and lithology, for all failure types, is presented in Table 6.

Table 6. Summary of laboratory test frequencies on rock samples by lithology.

Lithology	BTS	UCS	TCS	PLT		DST
				Axial	Diametral	
Amphibolite	44	17	23	2	5	12
Metasediments	36	23	27	3	7	4
Pegmatite	20	14	15	10	6	0
Ultramafic	26	17	16	0	0	6

Codes **BTS** Brazilian tensile strength, **UCS** Uniaxial compressive strength, **TCS** Triaxial compressive strength, **PLT** Point-Load test, **DST** Direct shear test.
Notes Total number of PLT's does not include the tests performed directly on site.
The DST were performed with 3 or 4 stages each.
The total number of tests is based on valid test results only.

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4 Geotechnical Characterization

This section presents the physical and mechanical properties of the overburden materials located in the vicinity of the mine.

4.1 Stratigraphic Units

The overburden materials present on site were classified into distinct stratigraphic units based on their composition and characteristics. The identified stratigraphic units are listed below:

- Organic soil
- Silt and sand
- Sand and gravel
- Till

The bedrock is located beneath the till layer. Drilling was extended by an average of four (4) metres into the rock to confirm that the underlying material was indeed bedrock rather than a large boulder. The stratigraphic units were primarily differentiated based on grain size. Stratigraphic columns were established for all geotechnical drillholes considered in the study, and the depth and position of each unit are presented in Table 7. This table also includes the distance of each drillhole from the slope and its position relative to the water bodies.

Table 7. Location information and top depth of each stratigraphic unit in geotechnical drillholes.

DH ID (CV24–)	Distance From Slope (m)	Drilled on Water	Stratigraphic Top Depth (m)					
			Water	Organic Soil	Silt and Sand	Sand and Gravel	Till	End of Overburden
542	120	Yes	0	–	5.0	–	–	5.7
558	250	No	–	0	–	0.1	–	2.6
562	250	No	–	0	–	0.4	–	1.3
775	100	Yes	0	–	4.0	4.8	5.8	7.5
779	100	Yes	0	–	4.0	–	5.1	9.1
780	150	No	–	0	–	0.1	–	0.4
782	170	No	–	0	–	0.1	–	0.4
784	0	No	–	0	0.1	1.8	12.2	27.4
785	0	No	–	0	0.1	–	–	0.3
786	0	No	–	0	0.1	–	3.3	20.6
787	30	No	–	0	0.2	–	1.4	14.0

Codes **DH ID** Drillhole identification.

Based on the top depth of each stratigraphic unit, the corresponding thicknesses were calculated and are presented in Table 8.

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Table 8. Thickness of each stratigraphic unit in geotechnical drillholes.

DH ID (CV24–)	Thickness of Stratigraphy (m)			
	Organic Soil	Silt and Sand	Sand and Gravel	Till
542	–	0.8	–	–
558	0.1	–	2.5	–
562	0.4	–	1.0	–
775	–	0.9	1.0	1.7
779	–	1.1	–	4.0
780	0.1	–	0.3	–
782	0.1	–	0.3	–
784	0.1	1.8	10.4	15.3
785	0.1	0.2	–	–
786	0.1	3.2	–	17.3
787	0.2	1.2	0	12.6

Codes **DH ID** Drillhole identification.

The proportions of each stratigraphic unit in the overburden columns are generally similar, and the average values provide a reasonable representation of the site conditions. The average thickness of each stratigraphic unit is presented in Table 9.

Table 9. Average thickness of each stratigraphic unit in geotechnical drillholes.

Stratigraphic Unit	Average Thickness (m)	Relative Presence (%)
Organic soil	0.1	2
Silt & sand	0.8	12
Sand & gravel	1.4	20
Till	4.6	66

Grain size, a key criterion for distinguishing stratigraphic units, also influences the strength characteristics of the materials in the context of slope stability. The following section describes the approach used to determine the strength parameters for each stratigraphic unit.

4.2 In-Situ Testing

The only in situ testing conducted in the geotechnical drillholes consisted of Standard Penetration Tests (SPT), which provided N-values along the drillhole profiles. These N-values were subsequently used to estimate the strength parameters of the various stratigraphic units. Table **??** summarizes the number of SPT tests performed in each drillhole and the corresponding average N-values.

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Table 10. Number of SPT performed in each stratigraphic unit per geotechnical drillholes and average N-values.

DH ID (CV24–)	Standard Penetration Resistance (N-Value)							
	Organic Soil		Silt & Sand		Sand & Gravel		Till	
	N	Avg.	N	Avg.	N	Avg.	N	Avg.
542	–	–	1	10	–	–	–	–
558	–	–	–	–	4	51	–	–
562	1	33	–	–	1	61	–	–
775	–	–	1	6	2	7	1	60
779	–	–	2	2	–	–	1	64
780	–	–	–	–	1	71	–	–
782	–	–	–	–	1	51	–	–
784	–	–	3	11	10	37	1	59
785	–	–	1	54	–	–	–	–
786	–	–	3	49	–	–	–	–
787	–	–	2	25	–	–	–	–
Total	**1**	**33**	**13**	**23**	**19**	**41**	**3**	**61**

Codes DH ID Identification of drillhole, N number of SPTs, Avg. average N-value.

To carry out the stability analyses, strength parameters are required for each stratigraphic unit. As no laboratory shear strength testing was performed to directly establish the strength envelopes of the stratigraphic materials, the envelopes were derived from correlations between the average N-values obtained from the Standard Penetration Tests and reference values from the literature (Terzaghi et al., 1996). The correlation tables used for this purpose are summarized in Table 11.

Table 11. Correlation between friction angle and N-value (adapted from Terzaghi et al., 1996 and Day, 2005).

N-Value	Associated Friction Angle (°)		
	Sand	Silt	Sand and Gravel
4	28	–	–
10	30	28	36
30	36	30	40
50	41	34	48

Note that the field N-values were used directly, without correction for hammer efficiency or overburden pressure. As a result, the derived friction angles should be considered approximate and suitable for preliminary design levels.

The strength parameters used to model soil slopes are presented in Table 12.

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Table 12. Strength parameters of stratigraphic units.

Stratigraphic Unit	Cohesion (kPa)	Friction Angle (°)
Silt & sand	0	28
Sand & gravel	0	40
Till	0	44

4.3 Laboratory Testing

In the area surrounding the open pit, residual soils are predominantly coarse-grained, with localized thin layers of silt, sand and gravel. Considering the granular and loose nature of the material on site, soil shear strength was assessed using SPT tests only; no laboratory strength testing was performed. Instead, laboratory testing of soil samples was conducted to validate the material characterization. The soil samples were collected from split spoons and sent to Englobe for laboratory testing. A total of 31 tests were performed on 11 relevant drillholes to determine particle-size distribution and water content. As the purpose of those tests was to validate the material type, and was part of the process to provide good stratigraphic log, the results are not presented in this report.

4.4 Geotechnical Design Parameters

The geotechnical parameters used in the stability analyses are summarized in Table 13.

Table 13. Summary of soil properties used for stability analyses.

Stratigraphic Unit	Density [1] (kN/m^3)	Cohesion (kPa)	Friction Angle (°)
Silt & sand	20.5	0	28
Sand & gravel	21.1	0	40
Till	22.8	0	44

Notes [1] The material densities were evaluated based on literature (Terzaghi et al., 1996).

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5 Geomechanical Characterization

This section outlines the physical and mechanical properties of the rock mass present at the Shaakichiuwaanaan Project, as determined from the investigation campaigns conducted by BBA (2024a), BBA (2024b), and Alius Mine Consulting (2025).

Geomechanical characterization was carried out for the four main lithologies, namely amphibolite, pegmatite, metasediments, and ultramafic rock. These units are composed of sub-lithologies; for example, there are several types of pegmatite, including spodumene, feldspar, and tourmaline. It was therefore assumed that all sub-lithologies included in these units have similar geomechanical properties.

5.1 Rock Mass Characterization

A rock mass is a natural geological material composed of two primary components: the intact rock matrix and its discontinuities. The following sections present the intact rock properties, the rock mass classification, and the orientation and characteristics of discontinuities. Rock mass strength estimates and final design parameters are then outlined. The resulting geomechanical domains are provided in Section 6.

5.1.1 Intact Rock Properties

Intact rock properties are generally assessed using a combination of field observations and laboratory testing. The laboratory-derived parameters apply to both the open-pit and underground mine designs. The key intact rock parameters include:

- Density
- Uniaxial compressive strength
- Triaxial compressive strength components: σ_{ci} and m_i
- Tensile strength
- Elastic properties: Young's modulus and Poisson's ratio

Density

Density (ρ) was measured in the laboratory as part of rock testing preparation. The number of samples tested and average densities are presented in Table 14, as well as the corresponding unit weights.

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Table 14. Summary of average density and unit weight per lithology.

Lithology	N	Density (kg/m^3)	Unit Weight (kN/m^3)
Amphibolite	42	2 984	29.3
Metasediments	27	2 876	28.2
Pegmatite	39	2 636	25.8
Ultramafic	21	2 875	28.2

Codes **N** number of samples.

Field Intact Rock Strength

Intact rock strength is generally quantified using the uniaxial compressive strength (UCS) parameter. UCS may be obtained from field tests or through laboratory testing (ASTM, 2014). The most common field methods include the field intact rock strength (FIRS) hammer test, as described by ISRM (1978), and point-load testing (PLT) on core samples (ASTM, 2016). Whenever possible, PLT results are preferred, as they provide a more consistent and reliable assessment of intact rock strength.

The point-load tests were conducted in accordance with ASTM D5731: *Standard Test Method for Determination of the Point Load Strength Index of Rock and Application to Rock Strength Classifications* (ASTM, 2016). The tests were performed on each logging interval directly on site. Because the point-load strength index (I_s) depends on the equivalent core diameter, a size correction was applied to normalize it to a reference diameter of 50 mm, yielding the standard value Is_{50}. The size-corrected point-load strength index is calculated as follows:

$$Is_{50} = \frac{P}{De^2} \times \left(\frac{De}{50}\right)^{0.45} \qquad \textit{(eq. 1)}$$

Where Is_{50} = point-load strength index corrected to 50 mm (MPa)
P = failure load (N)
De = equivalent core diameter (mm)

The uniaxial compressive strength can be estimated from Is_{50} using a conversion factor k:

$$UCS = k \times Is_{50} \qquad \textit{(eq. 2)}$$

Using laboratory UCS results together with both laboratory and on-site PLTs, conversion factors were derived for each lithological unit and are presented in Table 15. Although typical conversion factors range from 20 to 25, greater variability may occur in anisotropic rocks (ISRM, 2007), where values may fall within the 15–50 range.

The conversion factor obtained for the ultramafic undifferentiated unit is lower than commonly expected. This is attributed to the scattered nature of the data and the absence of a clear trend,

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as reflected by the relatively low coefficient of determination (R^2 = 0.40). Additional details on the derivation of conversion factors are provided in Appendix A.

Table 15. PLT to UCS conversion factors.

Lithology	Conversion Factor	R^2
Amphibolite	20.90	0.80
Metasediments	17.44	0.71
Pegmatite	15.89	0.84
Ultramafic	8.04	0.40

Codes R^2 R-squared statistical measure.

Applying these conversion factors to the measured I_{s50} values, a total of 205 diametral PLT tests were converted to UCS estimates. This dataset forms the basis for the lithology-based statistical analysis of UCS. Results for the ultramafic unit were calculated but not used in design due to insufficient data. Table 16 summarizes the first (Q1), second (Q2, median), and third (Q3) quartiles of the UCS values derived from the PLT data.

The high coefficients of variation reflect significant variability in intact rock strength. Based on the median values (Q2), and in accordance with the ISRM (1978) rock strength classification, the amphibolite and metasediments are classified as *very strong* rocks, the pegmatite as a *strong* rock, and the ultramafic as a *medium–strong* rock.

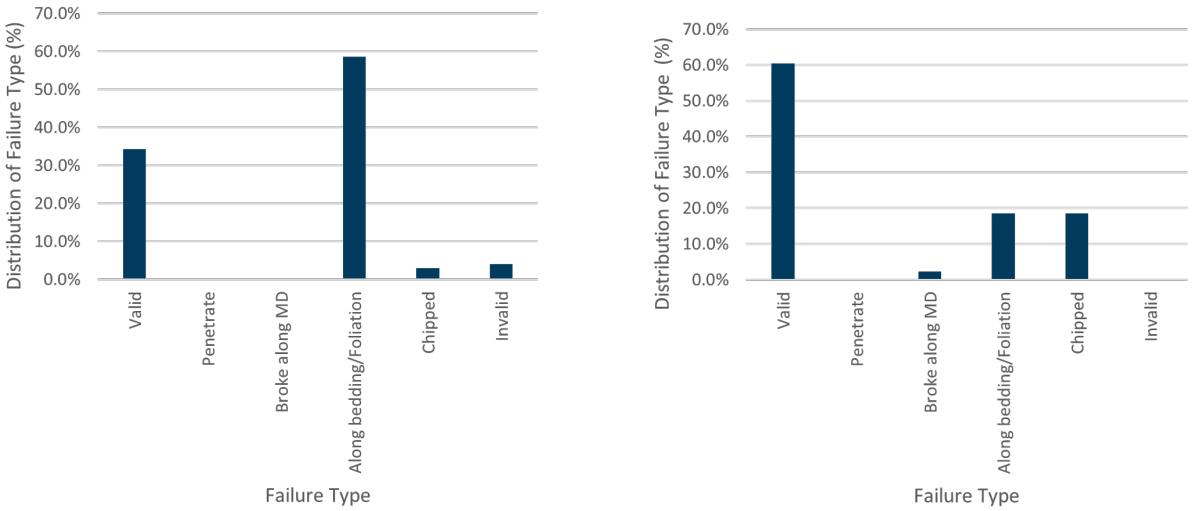
Table 16. Uniaxial compressive strength from on-site point load testing.

Lithology	UCS from PLT (MPa)					
	N	Q1	Q2	Q3	Avg.	COV (%)
Amphibolite	92	107	152	190	154	42
Metasediments	35	76	133	193	134	53
Pegmatite	70	62	73	90	79	46
Ultramafic	8	14	30	41	28	60

Codes **N** number of tests, **Q1** First quartile, **Q2** Median, **Q3** Third quartile, **Avg.** Average, **COV** coefficient of variation.
Notes **Low** 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.
UCS values are obtained with the conversion factors presented in Table 15.

When examining failure types from the PLT program, a significant proportion occurred along foliation planes. Figures 13a and 13b illustrate the distribution of failure types for the amphibolite and metasediments.

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(a) Amphibolite. *(b) Metasediments.*

Figure 13. Distribution of point-load test failure types.

Uniaxial Compressive Strength

To define representative uniaxial compressive strength (UCS) values for each lithological unit, all available laboratory test results were compiled (Alius Mine Consulting, 2025; BBA, 2024a; BBA, 2024b). A total of seventy-one (71) valid UCS results were retained for analysis. All UCS values were normalized to a reference specimen diameter of 50 mm following the size-correction approach of Hoek and Brown (1980).

In accordance with the interpretation framework proposed by Bewick et al. (2015), the UCS laboratory results were categorized according to failure mode (e.g., splitting, shear, failure along a discrete feature, or specimen-scale matrix failure). These were subsequently grouped into three failure-type classes: intact, combined, and discrete failure. Only UCS values associated with intact and combined failure types were retained for geomechanical characterization. A summary of all tests is presented in Table 17, while the filtered data set (intact + combined failures) is summarized in Table 18.

Table 17. Summary of uniaxial compressive strengths – All failure types.

Lithology	N	Laboratory UCS (MPa)			COV [1]
		Min.	Max.	Avg.	(%)
Amphibolite	17	16	277	171	38
Metasediments	23	58	251	129	38
Pegmatite	14	20	105	70	35
Ultramafic	17	11	202	66	80

Codes **N** number of valid tests, **Min** minimum (MPa), **Max** maximum (MPa), **Avg** average (MPa), **COV** coefficient of variation.
Notes [1] COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.

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Table 18. Summary of uniaxial compressive strengths – Intact and combined failure types.

Lithology	$N_i + N_c = N$	Laboratory UCS (MPa)			COV [1]
		Min.	Max.	Avg.	(%)
Amphibolite	10 + 5 = 15	107	277	188	24
Metasediments	7 + 5 = 12	58	251	142	42
Pegmatite	6 + 3 = 9	51	105	80	25
Ultramafic	8 + 3 = 11	30	202	88	62

Codes **N** number of valid tests, N_i number of tests with intact failure type, N_c number of tests with combined failure type, **Min** minimum (MPa), **Max** maximum (MPa), **Avg** average (MPa), **COV** coefficient of variation.
Notes **[1]** COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.

Table 19 presents the first and second quartiles for laboratory UCS values derived from tests exhibiting intact or combined failure types.

Table 19. Uniaxial compressive strength distribution from laboratory testing – Intact and combined failure types.

Lithology	Laboratory UCS (MPa)	
	Q1	Q2
Amphibolite	167	186
Metasediments	101	130
Pegmatite	62	86
Ultramafic	48	55

Codes **Q1** First quartile. **Q2** Median.

Comparison of the laboratory UCS results with the field-scale intact strength estimates derived from point-load tests (Section 5.1.1) indicates generally good agreement across the amphibolite, metasediment, and pegmatite units. Slight discrepancies in the ultramafic unit, where PLT-derived strengths tend to be lower.

Triaxial Compressive Strength

In triaxial compressive strength (TCS) tests, specimens are loaded axially to failure under constant confining pressure. From multiple TCS tests, a curve-fitting procedure can be used to determine the unconfined compressive strength (σ_{ci}) and m_i, two intact rock material constants required to assess the intact rock strength as well as the rock mass strength (more details in Section 5.1.4). As suggested by Hoek and Brown (2018), laboratory tests were conducted using the single-stage method rather than the multi-stage method. The tests were conducted across a range of confining pressures (σ_3), yielding a well-distributed dataset, representative of the expected stress levels.

TCS tests were completed on the four main lithological units. Table 20 summarizes the valid results from BBA (2024a) and BBA (2024b), and Alius Mine Consulting (2025). The results presented for

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the amphibolite, metasediments, and pegmatite reflect only tests exhibiting intact-type failure. In contrast, the results for the ultramafic unit include all observed failure modes (intact, combined, and discrete). These results form the basis for developing the failure envelopes.

Table 20. Summary of triaxial compressive strength tests.

Lithology	Confinement, σ_3 (MPa)	Peak Axial Stress, σ_1 (MPa)		
		N	Avg.	COV [1] (%)
Amphibolite	1.0	2	265.6	30
Amphibolite	1.5	1	170.3	–
Amphibolite	2.0	1	270.6	–
Amphibolite	3.0	2	194.6	21
Amphibolite	5.0	4	237.3	34
Amphibolite	15.0	2	392.7	2
Amphibolite	25.0	2	408.1	18
Metasediments	1.3	1	126.3	–
Metasediments	1.5	1	189.8	–
Metasediments	5.0	1	234.2	–
Metasediments	15.0	3	250.1	36
Pegmatite	1.3	1	63.6	–
Pegmatite	1.5	2	125.8	52
Pegmatite	1.7	1	132.8	–
Pegmatite	5.0	3	254.5	5
Pegmatite	25.0	3	300.3	43
Ultramafic	1.0	2	84.2	87
Ultramafic	1.2	2	23.9	21
Ultramafic	1.3	2	39.8	31
Ultramafic	5.0	3	119.8	13
Ultramafic	15.0	2	129.5	21
Ultramafic	25.0	3	164.9	5

Codes **N** number of valid tests, **Avg** average (MPa), **COV** coefficient of variation.
Notes [1] COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.
Amphibolite, Metasediments and Pegmatite: Intact failure type. Ultramafic: All failure types.

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Tensile Strength

Indirect tensile strength can be measured in laboratory using the Brazilian Tensile Strength (BTS) test. To estimate the direct tensile strength (DTS) from BTS tests, a correction factor must be applied, as BTS results tend to overestimate true tensile capacity. Perras and Diederichs (2014) suggests DTS/BTS correction ratios of 0.86, 0.82 and 0.70 for metamorphic, igneous and sedimentary rocks, respectively. Although the corrected DTS values may be used when developing failure envelopes, Hoek and Brown (2018) notes that applying a tension cutoff is generally more appropriate than directly incorporating tensile strength data into the Hoek-Brown envelope-fitting process.

As part of the BBA (2024a), BBA (2024b), and Alius Mine Consulting (2025) campaigns, a total of 126 BTS tests were performed (Table 21). Only a limited number of tests yielded valid results, as many samples failed diagonally across the loading points or along foliation planes. Sixteen (16) valid results were obtained and are summarized in Table 22. The corresponding DTS values, derived using the correction factors above, are presented at the end of this section, in Table 24.

Table 21. Summary of Brazilian tensile strength – All failure types.

Lithology	N	Laboratory BTS (MPa)			COV [1]
		Min.	Max.	Avg.	(%)
Amphibolite	44	4	25	10	48
Metasediments	36	2	14	10	22
Pegmatite	20	3	10	7	25
Ultramafic	26	3	14	8	40

Codes **N** number of valid tests, **Min** minimum (MPa), **Max** maximum (MPa), **Avg** average (MPa), **COV** coefficient of variation.
Notes [1] COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.

Table 22. Summary of Brazilian tensile strength – Intact failure types.

Lithology	N	Laboratory BTS (MPa)			COV [1]
		Min.	Max.	Avg.	(%)
Amphibolite	6	8	16	11	26
Metasediments	6	8	13	10	18
Pegmatite	4	6	10	8	23

Codes **N** number of valid tests, **Min** minimum (MPa), **Max** maximum (MPa), **Avg** average (MPa), **COV** coefficient of variation.
Notes [1] COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.

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Elastic Properties

Elastic properties, namely Young's modulus (E_i) and Poisson ratio (ν), are typically obtained by measuring axial and lateral strains during uniaxial compression tests.

For the Shaakichiuwaanaan project, the intact Young's modulus was estimated based on modulus ratios (MR) and the unconfined compressive strengths (σ_{ci}). The modulus ratios were estimated from the literature, based on guidelines (Hoek, 2006), and σ_{ci} was obtained from failure envelopes (presented in the upcoming section) using average UCS values and individual TCS test results, as presented in previous sections. In summary, the intact Young's modulus is calculated using:

$$E_i = MR \times \sigma_{ci} \qquad \textit{(eq. 3)}$$

Where E_i = intact Young's modulus
MR = modulus ratio
σ_{ci} = unconfined compressive strength

Finally, the resulting intact Young's moduli were compared to literature values (Read and Stacey, 2009), as shown in Table 23.

Poisson's ratio for lithological units interpreted using the IMASS constitutive model was assigned as a function of GSI (see equation eq. 4). For other materials – such as overburden and host rock – Poisson's ratio was assumed to be 0.20, based on Read and Stacey (2009), and project experience.

$$v = 0.32 - 0.0015 \times GSI \qquad \textit{(eq. 4)}$$

Where GSI = Geological Strength Index rock mass classification (section 5.1.2)

Table 23. Summary of elastic properties.

Lithology	Modulus Ratio (–)		Intact Young's Modulus (GPa)		Similar Rock Type Found in Literature
	Literature [1]	Retained	Obtained [2]	Literature [3]	
Amphibolite	400–500	450	89.2	30-90	Amphibolites
Metasediments	200–1 100	250	38.3	N/A	Metasandstone/ Schistes
Pegmatite	300–550	425	51.2	35-75	Granite
Ultramafic	250–300	275	15.9	N/A	Igneous/Peridotite

Codes N/A no data available.
Notes [1] From Hoek (2006).
[2] Calculated from eq. 3.
[3] From Read and Stacey (2009).

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Failure Envelopes

The intact Hoek–Brown failure envelopes developed for this study are presented in Figure 14. The resulting Hoek–Brown parameters – uniaxial compressive strength (σ_{ci}) and the material constant m_i – are summarized in Table 24.

The intact Hoek-Brown failure envelopes were developed for each relevant lithology of the Project, using valid triaxial compressive strength test data, including intact failure type for the amphibolite, metasediments, and pegmatite; and all failure types for the ultramafic unit. Through comparison with combination of different failure types and with values from literature for the resulting σ_{ci} and the m_i, the above-mentioned failure types were used to establish the most representative failure envelopes. Although the inclusion of discrete failures is not a conventional approach, doing so for the ultramafic unit resulted in a more realistic failure envelope, consistent with the expected mechanical behaviour of this lithology. The failure envelope was elaborated using average uniaxial compressive strength values for both intact and combined failure types for each lithology.

Failure envelopes were developed for each relevant lithology of the Shaakichiuwaanaan project using valid triaxial compressive strength test data. Only intact-type failures were used for amphibolite, metasediments, and pegmatite. For the ultramafic unit, all failure types (intact, combined, and discrete) were included due to the limited number of tests with intact and combined failure types.

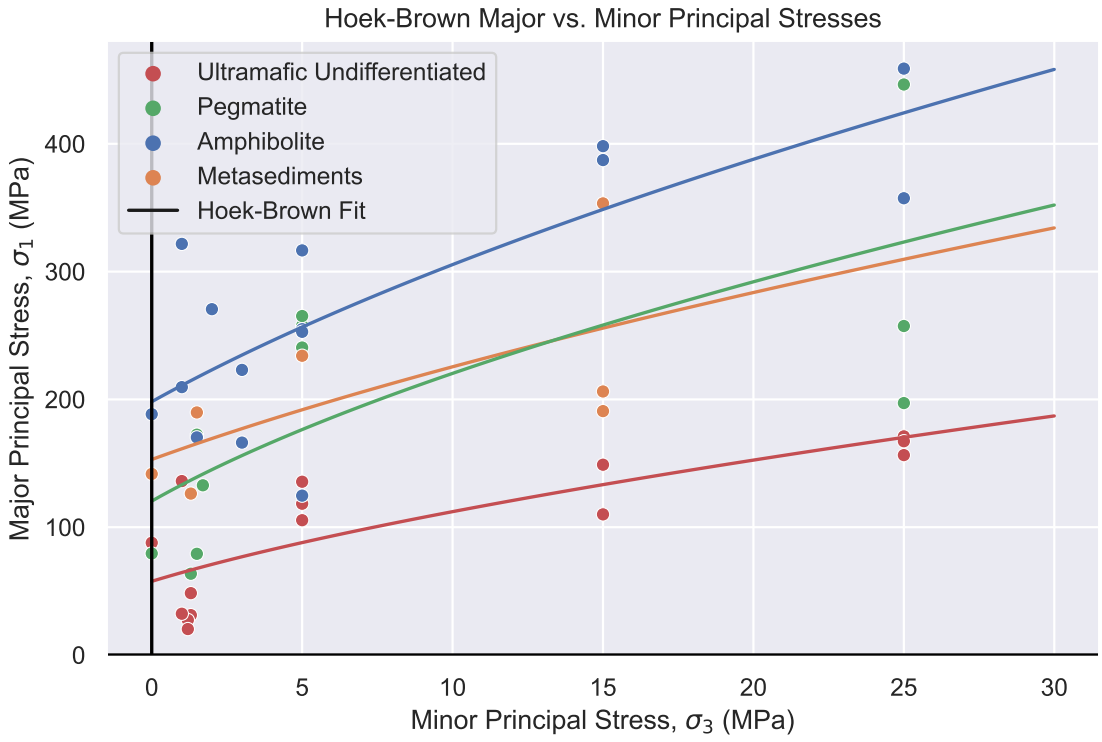


Figure 14. Intact rock Hoek-Brown failure envelopes.

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Summary of Intact Rock Properties

A summary of the intact rock properties of the main lithologies is presented in Table 24.

Table 24. Summary of intact rock strength properties of main lithologies.

Lithology	Density (kg/m^3)	Compressive Strength (MPa)	Tensile Strength (MPa)	Young Modulus (GPa)	σ_{ci} (MPa)	Constant m_i (–)
Amphibolite	2 984	188	9.5	89.2	198	24.2
Metasediments	2 636	142	8.6	38.3	153	15.1
Pegmatite	2 876	80	6.6	51.2	120	24.7
Ultramafic	2 875	88	–	15.9	57.7	12.3

Codes σ_{ci} intact rock unconfined compressive strength, m_i Hoek-Brown intact rock constant.
Notes Compressive strength corresponds to the average uniaxial compressive strength from intact or combined failure types only. Tensile strength corresponds to the corrected average Brazilian tensile strength with suggested ratios of Perras and Diederichs (2014).

5.1.2 Rock Mass Classification

The quality of rock masses can be assessed from core logging of geomechanical drillholes or from mapping, using one or more classification methods. For the Shaakichiuwaanaan project, the quality of the rock masses was assessed from core logging of the fourteen (14) geomechanical drillholes presented in Section 3.2. Given that this Project includes both open pit and underground mines with an extent of 4.2 km, separate rock mass classifications were developed for each. Although the two mining areas are largely spatially distinct, local interaction occurs near the northeastern end of the open pit where underground workings approach the pit footprint (see Section 8.10). The drillholes used to establish the rock mass classifications are listed in Table 25.

Table 25. Summary of drillholes used in the open pit and underground mines rock mass quality assessment.

Mine	Drillholes Considered
Open pit	**(CV24-)** 456, 479, 663, 679, 684, 689, 777, 781
Underground	**(CV24-)** 447, 459, 469, 496, 663, 679, 684, 769, 783

Notes Some drillholes are considered in both the open pit and underground mine analyses due to their location.

The rock mass quality was evaluated using three classification systems:

- Rock Mass Rating, or RMR_{76} (Bieniawski, 1976)
- Q' and Q indices (Barton et al., 1974)
- Geological Strength Index, or GSI (Hoek, Carter, and Diederichs, 2013)

Details of RMR_{76} (Bieniawski, 1976) and the Norwegian Geotechnical Institute (NGI) Q' and Q systems (Barton et al., 1974) are presented below, starting with the equations used to compute

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each classification.

$$RMR_{76} = A + B + C + D + E \quad \textit{(eq. 5)}$$

Where A = rating for intact rock strength
B = rating for drill core rock quality designation (RQD)
C = rating for spacing of discontinuities
D = rating for condition of discontinuities
E = rating for groundwater

$$Q = Q' \times \frac{J_w}{SRF} = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \quad \textit{(eq. 6)}$$

Where RQD = rock quality designation
J_n = joint set number
J_r = joint roughness number
J_a = joint alteration number
J_w = joint water reduction factor
SRF = stress reduction factor

For simplification, a J_w/SRF ratio of 1/1 was used for both open pit and underground classifications.

In this study, the GSI was evaluated using the joint condition rating and RQD, using the following equations:

$$GSI = 1.5 \times JCond_{89} + \frac{RQD}{2} \quad \textit{(eq. 7)}$$

Where $JCond_{89}$ = joint condition rating as defined by Bieniawski (1989)
RQD = rock quality designation as defined by Deere et al. (1967); the RQD value shall not exceed 80

$$JCond_{89} = 1.3 \times JCond_{76} \quad \textit{(eq. 8)}$$

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Where $JCond_{76}$ = joint condition rating as defined by Bieniawski (1976)
and $JCond_{76}$ = 25, Very rough surf.; not continuous; no separation; hard joint wall
= 20, Slightly rough surfaces; separation < 1 mm; hard joint wall
= 12, Slightly rough surfaces; separation < 1 mm; soft joint wall
= 06, Slickensided surfaces or gouges < 5 mm thick
or
Open joints 1-5 mm; continuous joints
= 00, Soft gouge > 5 mm thick
or
Open joints > 5 mm; continuous joints

Finally, RMR_{76}, GSI, Q, and Q' values were weighted by run length to reflect variation in logged interval lengths. To account for inherent rock mass variability, classification inputs are presented using lower and upper bounds (first and second quartiles). When sufficient data were available, the 25th and 50th percentiles of the cumulative distribution functions were used to support the underground mine design recommendations.

Open Pit

Tables 26, 27 and 28 summarize the rock mass classifications for the four main lithologies expected to be exposed in the final open pit walls.

Table 26. Summary of RQD – Open pit.

Lithology	RQD (%)				
	Q1	Q2	Q3	Avg.	COV [1] (%)
Amphibolite	69.0	87.5	96.4	79.4	28
Metasediments	56.9	77.3	89.7	68.8	41
Pegmatite	85.7	89.0	93.8	87.5	11
Ultramafic	62.5	74.6	84.7	72.1	26

Codes **RQD** rock quality designation, **Q1** first quartile, **Q2** median, **Q3** third quartile, **Avg.** average, **COV** coefficient of variation.
Notes [1] COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. **Moderate** 10% < COV < 30%. **High** COV > 30%.

Table 27. Rock mass classifications – Open pit.

Lithology	RMR_{76}		GSI		Q		Q'	
	Q1	Q2	Q1	Q2	Q1	Q2	Q1	Q2
Amphibolite	60	64	64	71	8	12	8	12
Metasediments	50	59	50	64	4	12	4	12
Pegmatite	56	64	64	64	11	15	11	15
Ultramafic	53	60	59	64	6	9	6	9

Codes **Q1** First quartile, **Q2** Second quartile.

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Table 28. Rock mass classification ratings – Open pit.

Lithology	RMR_{76} Rating		Q Rating	
	Q1	Q2	Q1	Q2
Amphibolite	Fair	Good	Fair	Good
Metasediments	Fair	Fair	Fair	Good
Pegmatite	Fair	Good	Good	Good
Ultramafic	Fair	Fair	Fair	Fair

Codes Q1 First quartile, Q2 Second quartile.

Underground Mine

During rock mass characterization of the underground mine area, trends in Q' indices were examined along the planned stopes, and no clear trend was established for lower values. The rock mass classification presented below is therefore deemed applicable across the entire underground area and and forms the basis for defining the geomechanical domains associated with the lithological units. Tables 29, 30, and 31 summarize the rock mass classifications for the four main lithologies in the underground mine area.

Table 29. Summary of RQD – Underground mine.

Lithology	RQD (%)				
	Q1	Q2	Q3	Avg.	COV [1] (%)
Amphibolite	78.2	88.8	95.6	83.8	20
Metasediments	71.3	84.5	93.7	80.1	22
Pegmatite	54.5	89.3	98.7	75.0	40
Ultramafic	59.9	78.8	91.1	76.1	24

Codes RQD rock quality designation, Q1 First quartile, Q2 Median, Q3 Third quartile, Avg. Average, COV coefficient of variation.
Notes [1] COV = σ/μ = standard deviation (MPa) / average (MPa).
Low 0% < COV < 10%. Moderate 10% < COV < 30%. High COV > 30%.

Table 30. Rock mass classifications – Underground mine.

Lithology	RMR_{76}		GSI		Q		Q'	
	Q1	Q2	Q1	Q2	Q1	Q2	Q1	Q2
Amphibolite	61	68	64	64	11	18	11	18
Metasediments	56	59	62	64	10	16	10	16
Pegmatite	46	69	44	78	7	18	7	18
Ultramafic	52	60	61	64	5	11	5	11

Codes Q1 first quartile, Q2 second quartile.

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Table 31. Rock mass classification ratings – Underground mine.

Lithology	RMR_{76} Rating		Q Rating	
	Q1	Q2	Q1	Q2
Amphibolite	Good	Good	Good	Good
Metasediments	Fair	Fair	Fair	Good
Pegmatite	Fair	Good	Fair	Good
Ultramafic	Fair	Fair	Fair	Good

Codes Q1 first quartile, Q2 second quartile.

Although the rock mass quality classifications were derived from the geomechanical logging of BBA (2024a), BBA (2024b) and Alius Mine Consulting (2025) geomechanical logging, they were compared against the quick-log performed by Dahrouge on 315 drillholes from 2022-2024 covering the underground area. The comparison showed similar Q' values, supporting the adopted classification. Figure 15 shows shows the distribution of Q' values for the drillholes evaluated by Dahrouge, regardless of the lithology. The majority of the Q' values range from 4 to 40.

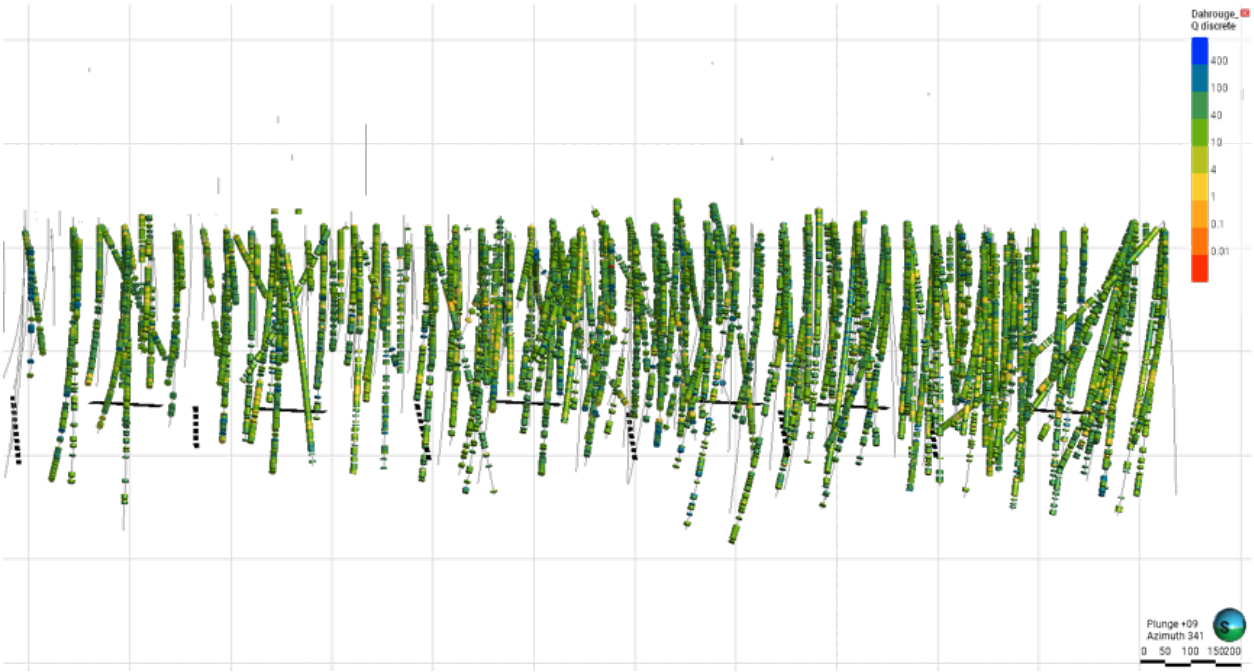


Figure 15. Evaluated Q' distribution from Dahrouge logged holes.

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RQD Model

RQD wireframes were provided by Dahrouge to represent zones of lower rock quality, defined as areas with RQD values below 40%. The model was built using the entire geological database (approximately 228 000 m of logged core). Several limitations were noted:

- Some low-RQD intervals observed in drillholes do not correspond to any modelled low-RQD lenses.
- Certain modelled lenses are not directly supported by low RQD measurements in adjacent drillholes, potentially overstating their lateral continuity.
- Most low-RQD lenses are oriented east–west, whereas the dominant fault trends strike northwest–southeast.

Despite these limitations, the model was used to examine the extent and continuity of low-RQD zones across both the open pit and underground mine areas. It is noted that the majority of the geological database displays high RQD values (Q1 = 97%; Q2 = 99%). The model is expected to be refined further during the detailed engineering phase.

For the open pit, when compared with the PEA pit design shown in Figure 16, some low RQD zones intersect the pit wall, though their extent is generally limited. The exception is a large lens extending from the top to the bottom of the pit along the north wall, which approximately corresponds to the location of fault plane FLT_1025. A review of the drillhole logs in the vicinity of this lens indicates that very few drillholes are drilled behind the pit wall. Indeed, only one hole behind the pit wall display a small interval with RQD values below 40%. Nevertheless, due to its spatial association with the fault, this feature should be taken into account during mining operations to adapt working practices accordingly.

For the open pit, comparison with the PEA pit design (Figure 16) shows that some low-RQD zones intersect the pit walls, but these occurrences are generally limited in size. The main exception is a larger lens extending from the top to the bottom of the north wall, spatially associated with fault plane FLT_1025.

A review of the drillhole logs in the vicinity of this lens indicates that very few holes are drilled behind the pit wall. Indeed, only one drillhole behind the pit wall displays a small interval with RQD values below 40%. Nonetheless, due to its spatial association with the fault, this feature should be taken into account during mining operations so that mining practices can be adjusted as needed.

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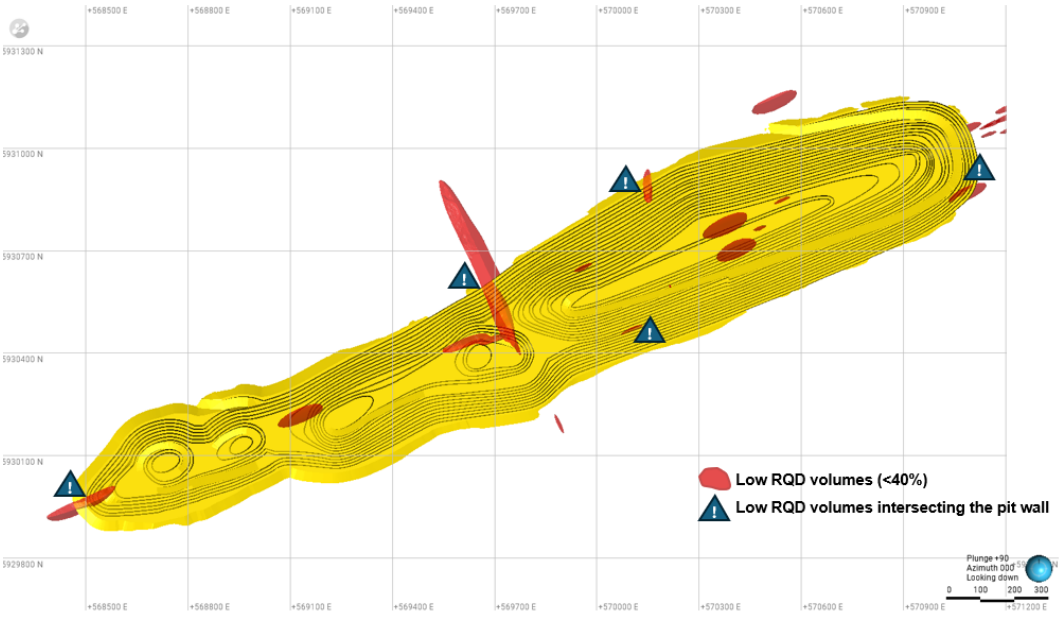


Figure 16. Modelled low RQD zones (<40%) in relation to the PEA open pit design.

In the underground mine area (Figure 17), several discontinuous low-RQD lenses are present, which is consistent with the scattered nature of low-RQD measurements in the drillhole data. No coherent structural or spatial trend can be established based on the available information.

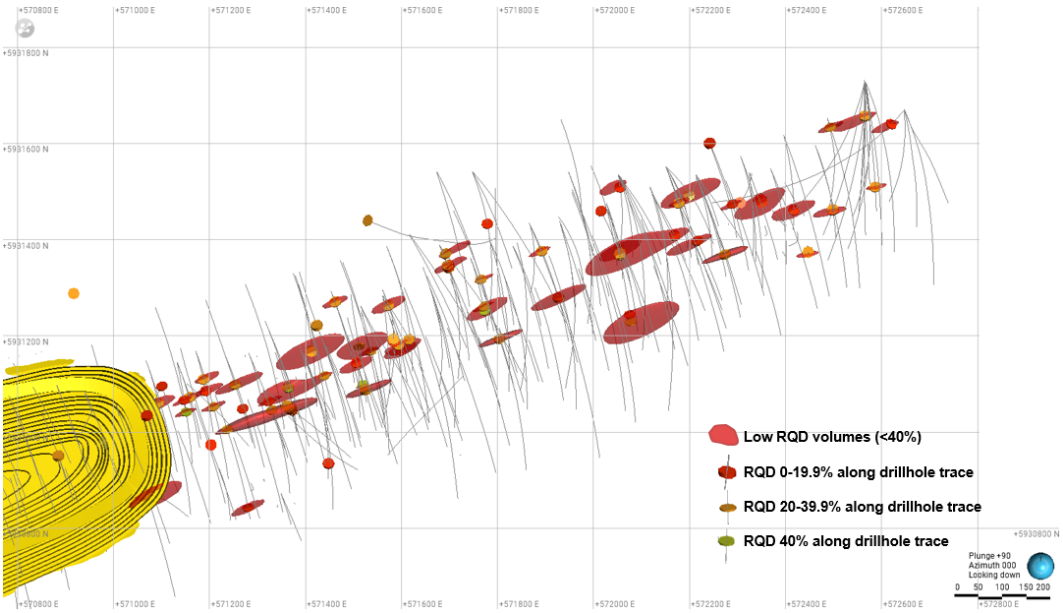


Figure 17. Modelled low RQD zones (<40%) within the underground mine area.

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5.1.3 Rock Mass Structures

Rock mass structures can be divided into two main categories: major structures – such as faults, folds, and metamorphic layering – and minor structures, including micro-bedding, small-scale folding, minor faults, joints, schistosity, and cleavage (Read and Stacey, 2009). Whereas major structures are discussed in Section 2.2.3, this section focuses on the discontinuities present within the rock mass, describing their properties, orientations, and general characteristics.

Shear Strength Properties

Shear strength of discontinuities is generally defined in terms of peak friction angle and cohesion, following the Mohr-Coulomb failure criterion (eq. 10). These parameters are determined through laboratory direct shear (DS) testing, in which a normal load is applied (perpendicular) to a discontinuity between two rock blocks while the shear stress required to induce relative displacement is continuously recorded.

$$\tau_{dp} = \sigma_N \cdot \tan \phi_{dp} + c_{dp} \qquad \textit{(eq. 9)}$$

Where τ_{dp} = discontinuity peak shear strength (MPa)
c_{dp} = discontinuity peak cohesion (MPa)
φ_{dp} = discontinuity peak friction angle (°)

In the residual condition – after peak strength has been exceeded and significant displacement has occurred along the discontinuity – the shear strength is given by:

$$\tau_{dr} = \sigma_N \cdot \tan \phi_{dr} + c_{dr} \qquad \textit{(eq. 10)}$$

Where τ_{dr} = discontinuity residual shear strength (MPa)
c_{dr} = discontinuity residual cohesion (MPa)
φ_{dr} = discontinuity residual friction angle (°)

For the Shaakichiuwaanaan project, twenty-one (21) discontinuities were tested in the amphibolite, metasediments, and ultramafic units (Alius Mine Consulting, 2025; BBA, 2024a; BBA, 2024b). No samples were collected from the pegmatite due to its relatively limited exposure on the final pit wall. Figure 18 presents the direct shear test results together with the interpreted peak and residual friction angles. The regression analyses shown in the figure were constrained to a y-intercept of 0 (i.e., zero cohesion), consistent with bench-scale conditions where blast-induced damage, fractured surface conditions, and low confining stresses are expected (Read and Stacey, 2009).

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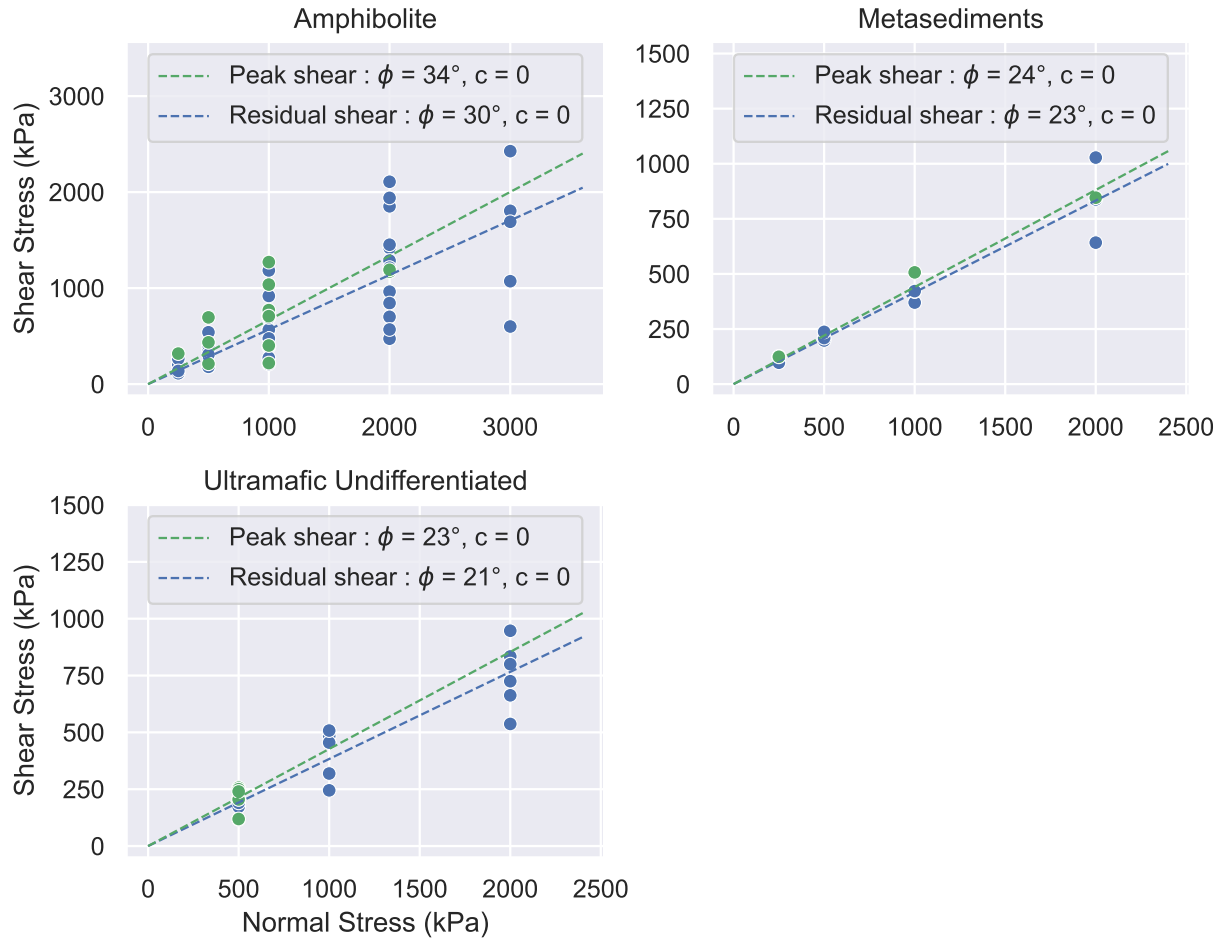


Figure 18. Direct shear test results for peak (green) and residual (blue) failures.

Table 32 summarizes the retained friction angles, defined as the average of the peak and residual values. Where multiple lithologies are interlayered along an exposed pit wall and cannot be reasonably distinguished in the analysis, a retained friction angle of 30° is applied. This assumption is particularly relevant for the eastern and western end walls of the pit.

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Table 32. Summary of shear strength properties.

Lithology	Cohesion (kPa)	Peak Friction Angle (°)	Residual Friction Angle (°)	Retained Friction Angle (°)
Amphibolite	0	34	30	32
Metasediments	0	24	23	23.5
Pegmatite	0	N/A	N/A	32 [1]
Ultramafic	0	23	21	22.5

Codes N/A not applicable.
Notes [1] From conservative assumption based on Read and Stacey (2009): $\phi_{\text{Coarse-grained igneous rocks}}$ = 31-48°.

Orientation and Description

Televiewer surveys – optical televiewer (OTV) and acoustic televiewer (ATV) – are widely used to acquire high-resolution structural data directly from drillholes, including fracture orientations, spacing, and aperture characteristics. These data are used to define the geomechanical domains.

For the Shaakichiuwaanaan project, sixty-five (65) televiewer surveys were completed across both the open pit and underground mine areas. Table 33 summarizes the surveys considered in the structural analyses.

In the open pit area, forty-four (44) surveys from drillholes located within the pit footprint or within 100 m of the pit limit were included. In addition, three surveys from drillholes CV22-042, CV22-066, and CV22-069 were incorporated specifically to characterize the ultramafic undifferentiated unit. Although these drillholes are situated approximately 400–500 m east of the pit boundary, they were included due to the limited structural information available for this lithology.

For the underground mine area, twenty-six (26) televiewer surveys were available and included in the analysis.

Table 33. Summary of televiewer surveys used for structural analysis of the open pit and underground mine sectors.

Mine Sector	Drillholes Considered
Open pit	**(CF21-)** 001, 002, 003 **(CV22-)** 019, 022, 024, 040, 043, 048, 052, 055, 058, 065, 071, 074 **(CV23-)** 130, 152, 157, 164, 188, 194, 197, 217, 219, 223, 236, 243, 317, 364 **(CV24-)** 423A, 447, 456, 463, 479, 489, 493, 514, 521, 609, 623, 689, 739, 777, 781
Underground	**(CV22-)** 019, 021, 022, 024, 042, 047, 052, 055, 058, 063, 066, 069 **(CV23-)** 124, 130, 133, 152, 157, 194, 197 **(CV24-)** 447, 459, 463, 469, 496, 500, 514

Notes Some drillholes are considered in both the open pit and underground mine analyses due to their location.

Irrelevant features, such as veins, water levels, and casing bottoms, were excluded from the database because they do not contribute to the structural interpretation. The analysis focused on the following discontinuity types: open joints, closed joints, foliation, broken zones, faults, shears,

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and contacts.

Table 34 presents the number of entries associated with each discontinuity type, while Table 35 summarizes the total number of entries for the four main lithologies. Both tables compile data from the sixty-five (65) televiewer surveys.

Table 34. Summary of structural data available per discontinuity type.

Discontinuity Type	Number of Entries
Closed joint	6 725
Open joint	1 906
Foliation	1 535
Fault	436
Contact	417

Note From the 65 televiewer surveys.

Table 35. Summary of structural data available for the main lithology type.

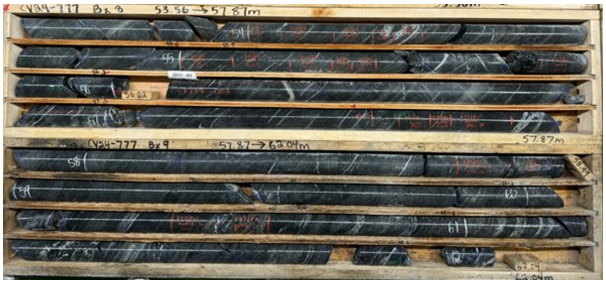
Lithology	Number of Entries
Amphibolite	9 785
Metasediments	2 969
Pegmatite	1 853
Ultramafic	856

Note From the 65 televiewer surveys.

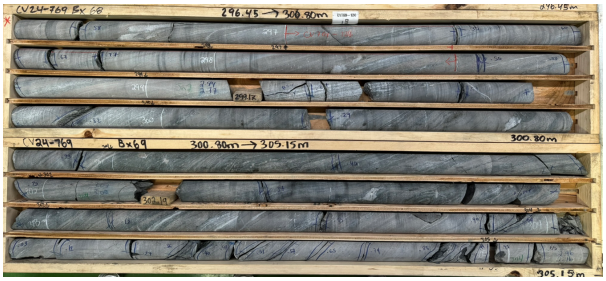
Foliation is a structural fabric that introduces anisotropy in the rock mass and can significantly affect the stability of mine excavations. As shown in Table 34, foliation was identified during the televiewer surveys; however, the actual occurrence is likely higher due to limitations inherent to the picking process during televiewer data interpretation. This primarily affects the estimation of average joint spacing for joint sets composed of foliation planes. In practice, when foliation is present, the true joint spacing is typically smaller than indicated, resulting in less favourable conditions for stability analysis.

At the Shaakichiuwaanaan project, foliation was identified across all lithological units, but is most prominent in the amphibolite, metasedimentary rocks, and the ultramafic undifferentiated unit. Figures 19a to 19c show representative core photographs where foliation is clearly visible.

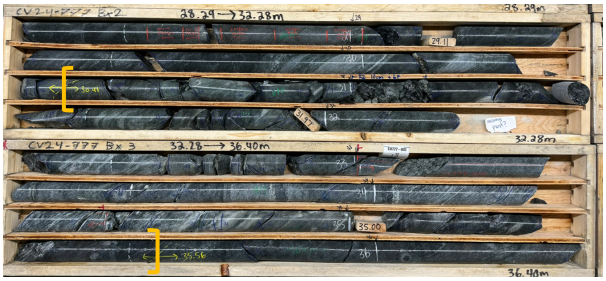
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(a) Amphibolite.



(b) Metasediments.



(c) Ultramafic undifferentiated.

Figure 19. Foliation visible on the core picture of different rock types.

The description of discontinuities, including surface shape, roughness, alteration, infill thickness, and the number of joint sets, was recorded during logging. Figure 20 presents the distributions of the joint roughness (J_r), the joint alteration (J_a) and the joint set rating (J_n) by lithology, compiled from the fourteen (14) geomechanical drillholes.

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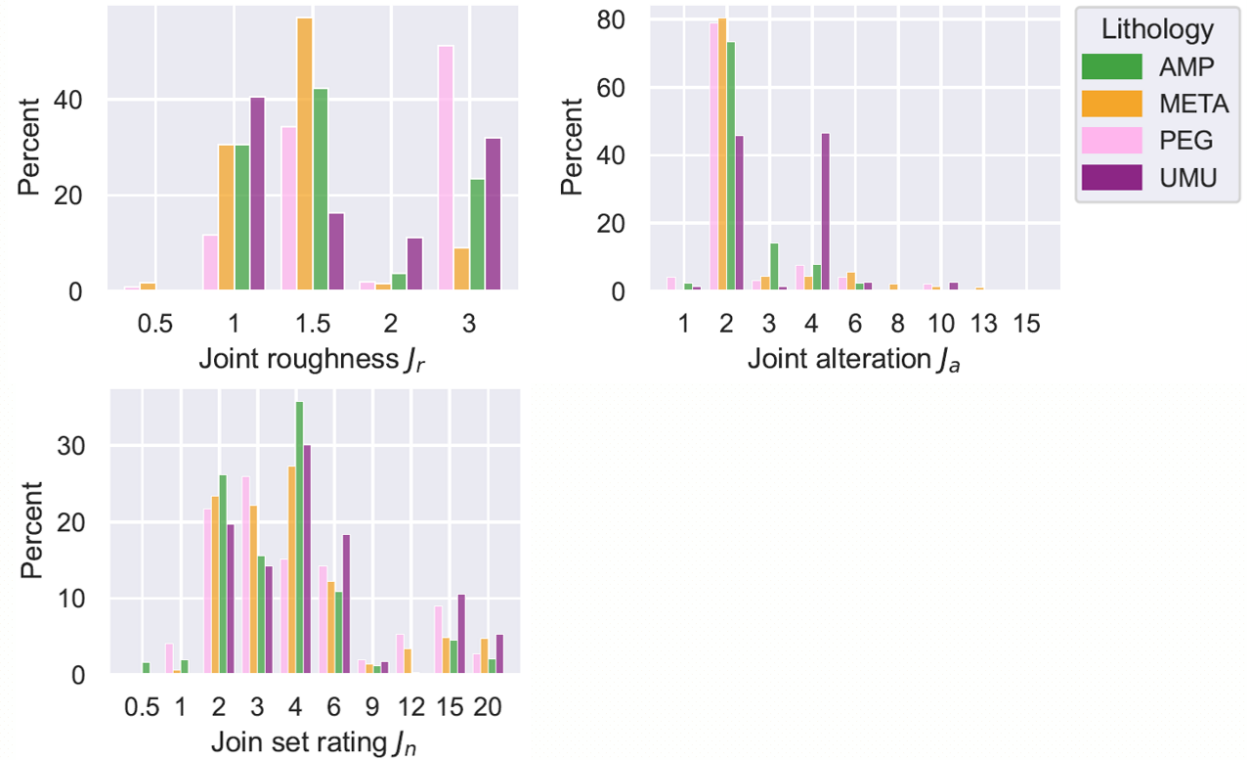


Figure 20. Characterization of discontinuities by lithology.

Open Pit

The interpretation of surveyed discontinuities is a critical component of the structural analyses required to support pit design. This section presents the structural characterization of the rock that will be exposed in the open pit walls and forms the basis for defining the geomechanical domains described in Section 6.

The Shaakichiuwaanaan open pit extends approximately 2.8 km in length. Given this scale, a detailed assessment of the spatial variability in joint set orientations (dip and dip direction) was required. To facilitate this analysis, the pit was subdivided into eight (8) zones, as shown in Figure 21. For each zone, stereographic projections of discontinuity poles were generated for the four main lithological units. The results indicate that joint-set orientations vary spatially across the pit, particularly from west to east.

The pegmatite unit exhibits a steep dip that effectively divides the host rock into a hanging wall to the northwest (north pit wall) and a footwall to the southeast (south pit wall). To evaluate potential differences in joint set orientations across this intrusive body, drillholes were classified based on whether they intersected the host rock north or south of the pegmatite. This approach allowed comparison of discontinuity orientations within the same lithology on both sides of the intrusion and supported subdividing the metasedimentary unit into northern and southern domains.

Appendix C presents the stereonets generated for each zone and lithological unit.

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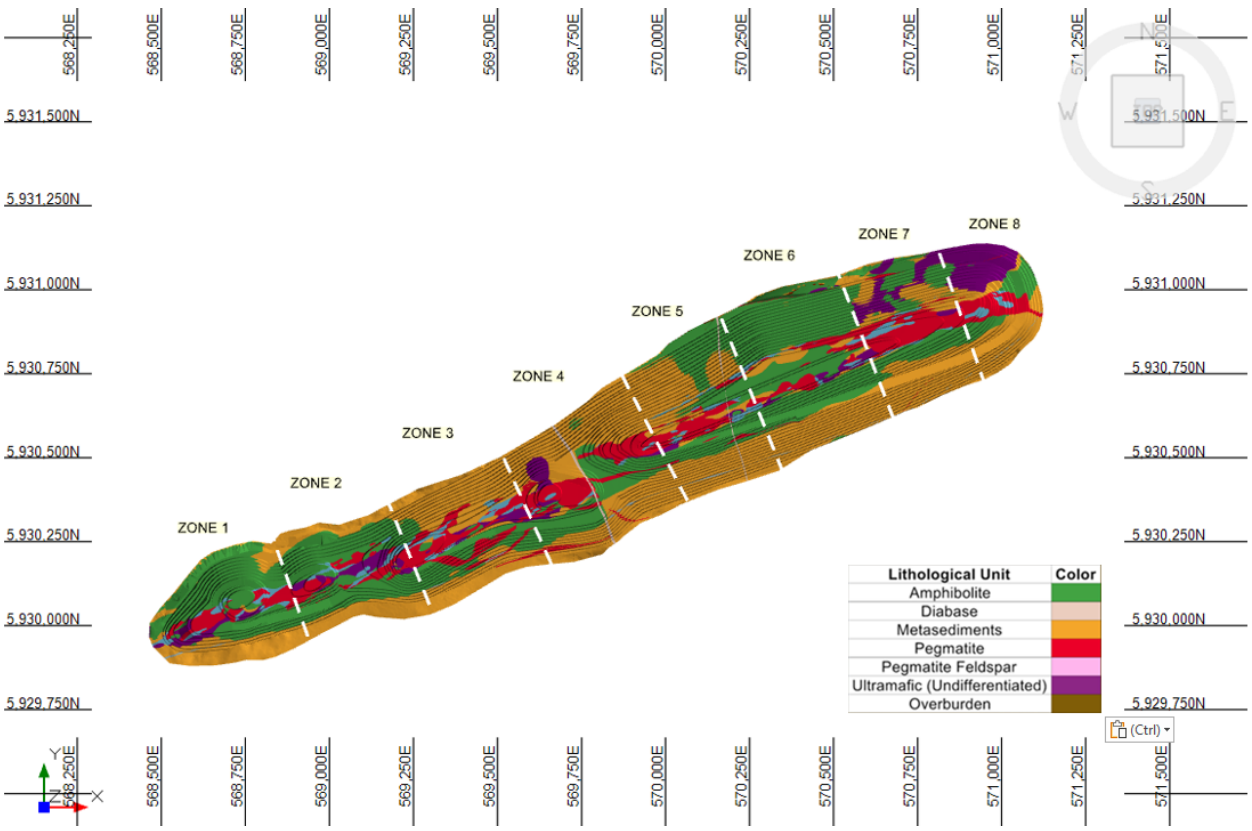


Figure 21. Open pit zones used for geomechanical domain definition.

This detailed spatial analysis of joint set orientations provided the basis for establishing geomechanical domains and for determining which televiewer surveys should be combined to develop representative structural datasets for subsequent kinematic analyses. For reference, the defined geomechanical domains and associated televiewer surveys are listed in Table 40 of Section 6. To spatially locate the geomechanical domains, the open pit was divided into three (3) general areas, as shown in Figure 22. The following nomenclature is adopted:

$$X_1X_2 - Lithology$$

Where X_1: general area (W=West, C=Central, E=East).
X_2: open pit wall (N=North, S=South, E=East, W=West, All= both south and north walls).
Lithology: corresponds to the lithology code; if more than one lithology is concerned, combination of first code letters is used, APU (**A**MP, **P**EG, **U**MU) and AMPU (**A**MP, **M**ETA, **P**EG, **U**MU).

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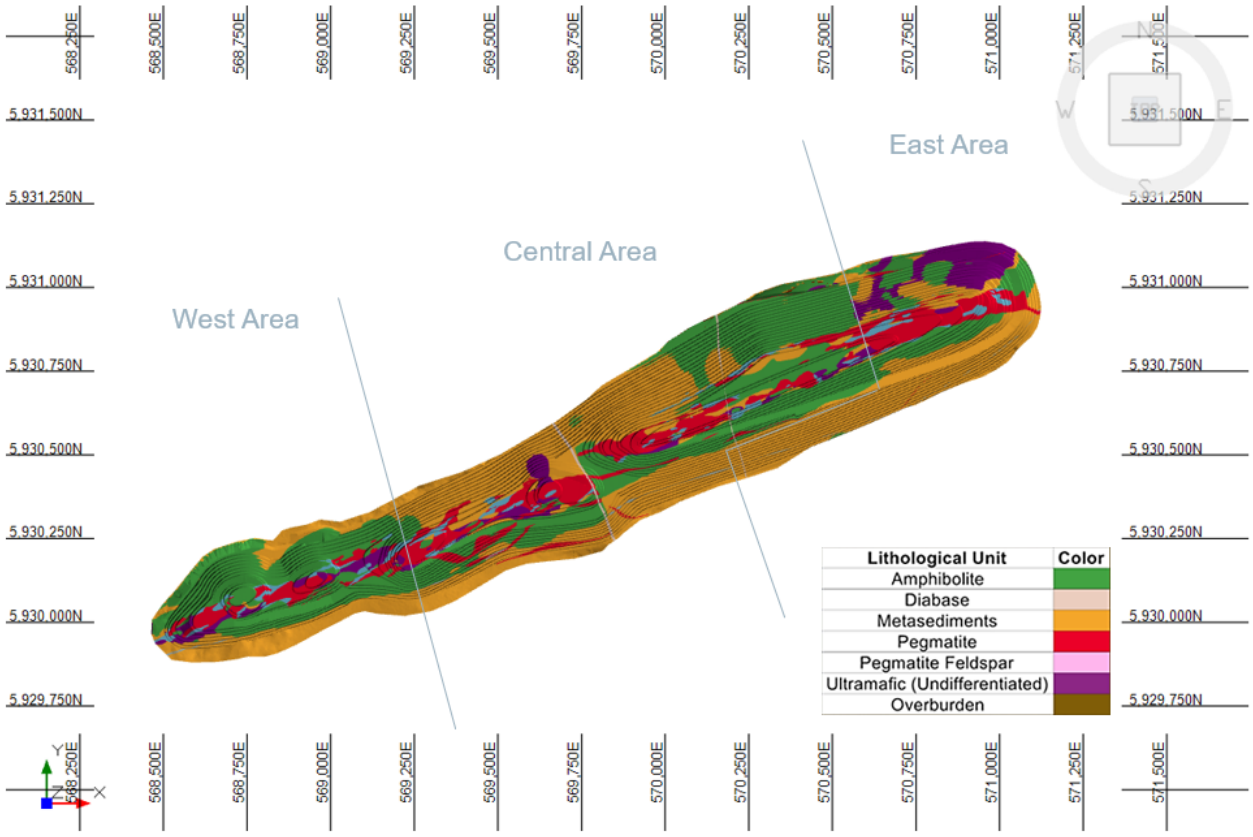


Figure 22. Open pit general area division.

The subdivision into eight (8) zones, together with the separation of the metasedimentary units into northern and southern sectors relative to the pegmatite intrusion, highlighted several areas with limited structural data. In several cases, these gaps occur because the televiewer-surveyed drillholes do not intersect the lithological units that will be exposed in the final pit wall. Consequently, for some geomechanical domains, data from adjacent zones were used to supplement the available information.

- → WS-META — No data available for Zones 1 and 2 META (South); data from Zones 1 and 2 META (North) were used instead.
- → CN-UMU & EN-UMU — Data for the UMU are limited; information from adjacent zones was used where available.

Table 36 presents a summary of the interpreted joint sets within the geomechanical domains. Along with the average pole dip and dip direction of the joint sets, this table displays the mean joint spacing and the number of poles for each defined set. These values, combined with other factors, such as comparisons with the previous geomechanical characterization (PEA), were used to classify the joint sets as major or minor. During the analysis, greater emphasis was placed on the major sets due to their significant influence on bench design, while the minor sets were considered to have limited potential impact. Note that JS1 sets typically include foliation planes; therefore, the

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true average joint spacing for these sets is likely smaller than the value shown in this table.

Table 36 summarizes the interpreted joint sets within the geomechanical domains. The table includes the mean dip and dip direction for each joint set, mean joint spacing, and the number of poles used to define each set. These parameters, along with comparisons with previous geomechanical characterizations (PEA), were used to classify joint sets as major or minor. For the analyses, greater emphasis was placed on the major sets, while minor sets were assumed to have a limited impact. Note that JS1 typically includes foliation planes; therefore, the true average spacing of this set is likely smaller than the values shown.

Appendix B presents the stereographic projections (lower hemisphere) of the discontinuities and the interpreted joint sets for each of the geomechanical domains within the open pit.

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Table 36. Interpreted joint sets per geomechanical domain – Open pit.

GMD (Fresh Rock)	Joint Set	Dip (°)	Dip Direction (°)	Mean Joint Spacing (m)	Number of Poles
WAll-AMP	JS1	71	157	2.30	241
	JS2	21	347	4.86	145
WW-APU	JS1	72	157	1.31	400
	JS2	17	002	3.40	284
WS-META	JS1	48	156	1.47	50
	JS2	31	002	3.90	83
	JS3	78	163	1.21	43
CN-META	JS1	49	164	1.03	194
CN-AMP	JS1	60	150	1.59	308
	JS2	22	332	3.03	112
	js3	29	159	2.27	89
CN-UMU	JS1	51	160	0.53	33
	js2	58	212	2.31	5
CS-META	JS1	55	152	0.71	125
	JS2	21	356	0.98	110
CS-AMP	JS1	53	157	1.50	604
	JS2	22	337	5.49	292
	JS3	77	334	3.59	227
EN-UMU	JS1a	29	162	1.24	21
	JS1b	58	152	0.42	39
EAll-AMP	JS1	47	158	1.31	350
	JS2	23	357	3.74	261
EN-META	JS1	51	163	2.37	36
	JS2	51	346	3.19	35
	js3	58	233	2.75	8
EE-AMPU	JS1	46	160	1.09	299
	JS2	22	350	3.87	243
ES-META	JS1	55	151	2.30	140
	JS2	30	343	5.76	72
WCE-PEG	JS1	77	156	3.47	92
	JS2	08	013	2.00	427
	JS3	77	340	3.26	137

Codes GMD geomechanical domain. Refer to Table 40 for GMD presentation.
Notes Capitalized JS: major joint set. Lowercase js: minor joint set.

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Underground Mine

The main lithology considered for the underground analysis is the pegmatite; however, amphibolite, metasediments, and ultramafic rocks can also be exposed in the hanging wall and footwall. To identify the relevant joint sets, only the discontinuities surveyed in the twenty-six (26) drillholes that are located in the underground mine area were considered. Table 37 presents the interpreted joint sets defined for the underground analysis. Again, note that JS1 sets are usually the ones that include the foliation planes. Consequently, the average joint spacing for these sets is expected to be smaller than the value presented in this table.

Appendix B also shows the stereographic projections (lower hemisphere) of the discontinuities and the interpreted joint sets for each of the geomechanical domains in the area of the underground mine, prepared with *Dips* (Rocscience, 2024a).

Table 37. Interpreted joint sets per geomechanical domain – Underground mine.

GMD (Fresh Rock)	Joint Set	Dip (°)	Dip Direction (°)	Mean Joint Spacing (m)	Number of Poles
Metasediments	JS1	48	153	2.56	422
	JS2	26	343	4.66	145
Amphibolite	JS1	47	153	2.44	885
	JS2	25	344	7.97	568
	js3	64	340	10.73	310
Ultramafic	JS1	56	148	2.35	39
	JS2	29	153	3.73	63
Pegmatite	JS1	02	346	2.41	258
	JS2	87	341	4.78	62
	js3	89	042	6.80	14

Codes GMD Geomechanical Domain (fresh rock). Refer to Table 41 for GMD definitions.
Notes Capitalized JS: major joint set. Lowercase js: minor joint set.

5.1.4 Rock Mass Properties

Rock mass properties include the Young's modulus, Poisson's ratio and strength properties. The Young's modulus of the rock mass can be estimated using the following equation (Hoek and Diederichs, 2006):

$$E_{RM} = E_i \times \left[0.02 + \frac{1 - \frac{D}{2}}{1 + \exp\left(\frac{60 + 15D - GSI}{11} \right)} \right] \qquad \textit{(eq. 11)}$$

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Where E_{RM} = Young's modulus of rock mass (GPa)
E_i = Young's modulus of intact rock (GPa)
D = disturbance factor (due to stress redistribution and blasting damage)
Where D = 0.5, Controlled presplit or smooth wall blasting
= 0.7, Mechanical excavation effects of stress redistribution damage
= 1.0, Production blasting
* a transitional D relationship can be utilized (Rose et al., 2018)

The rock mass and intact rock Poisson's ratio are assumed to be equal. The resulting elastic rock mass properties are presented in Table 38.

The rock mass strengths are represented by either the Mohr-Coulomb or the Hoek-Brown criteria. The former criterion can be represented as follows (Coulomb, 1776):

$$\tau = \sigma_N \cdot \tan\ \phi + c \qquad \textit{(eq. 12)}$$

Where τ= shear strength (MPa)
σ_N = normal stress (MPa)
φ= friction angle (°)
c = cohesion (MPa)

In terms of the major and minor principal stresses (i.e., σ_1 and σ_3), the Mohr-Coulomb criterion is given by:

$$\sigma_1 = k \cdot \sigma_3 + \sigma_{cm} = \left(\frac{1+\sin\phi}{1-\sin\phi}\right) \cdot \sigma_3 + \left(\frac{2\ c\ \cos\phi}{1-\sin\phi}\right) \qquad \textit{(eq. 13)}$$

Where σ_1 = major principal effective stress at failure (MPa)
σ_3 = minor principal effective stress at failure (MPa)
σ_{cm} = rock mass compressive strength (MPa)

The Hoek-Brown criterion takes the following form (Hoek and Brown, 2018):

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b\frac{\sigma_3}{\sigma_{ci}} + s\right)^a \qquad \textit{(eq. 14)}$$

Where σ_{ci} = unconfined compressive strength of intact rock (MPa)
m_b, s, a = rock mass material constants

With

$$m_b = m_i\ \exp\left[\frac{GSI-100}{28-14D}\right] \qquad \textit{(eq. 15)}$$

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$$s = \exp\left[\frac{GSI - 100}{9 - 3D}\right] \quad \textit{(eq. 16)}$$

$$a = \frac{1}{2} + \frac{1}{6}\left[\exp\left(\frac{-GSI}{15}\right) - \exp\left(\frac{-20}{3}\right)\right] \quad \textit{(eq. 17)}$$

Where m_i = intact rock material constant.

The rock mass compressive strength (σ_{RM}) and the rock mass tensile strength derived from the Hoek-Brown criterion can be evaluated using the following equations:

$$\sigma_{RM} = \sigma_{ci} \cdot s^a \quad \textit{(eq. 18)}$$

$$\sigma_t = -\frac{s \cdot \sigma_{ci}}{m_b} \quad \textit{(eq. 19)}$$

The rock mass failure envelopes were derived using disturbance factors of 0.0 (undisturbed) and 0.7, combined with the GSI values obtained for the underground mine rock mass quality assessment. The resulting rock mass envelopes are presented in Figure 23.

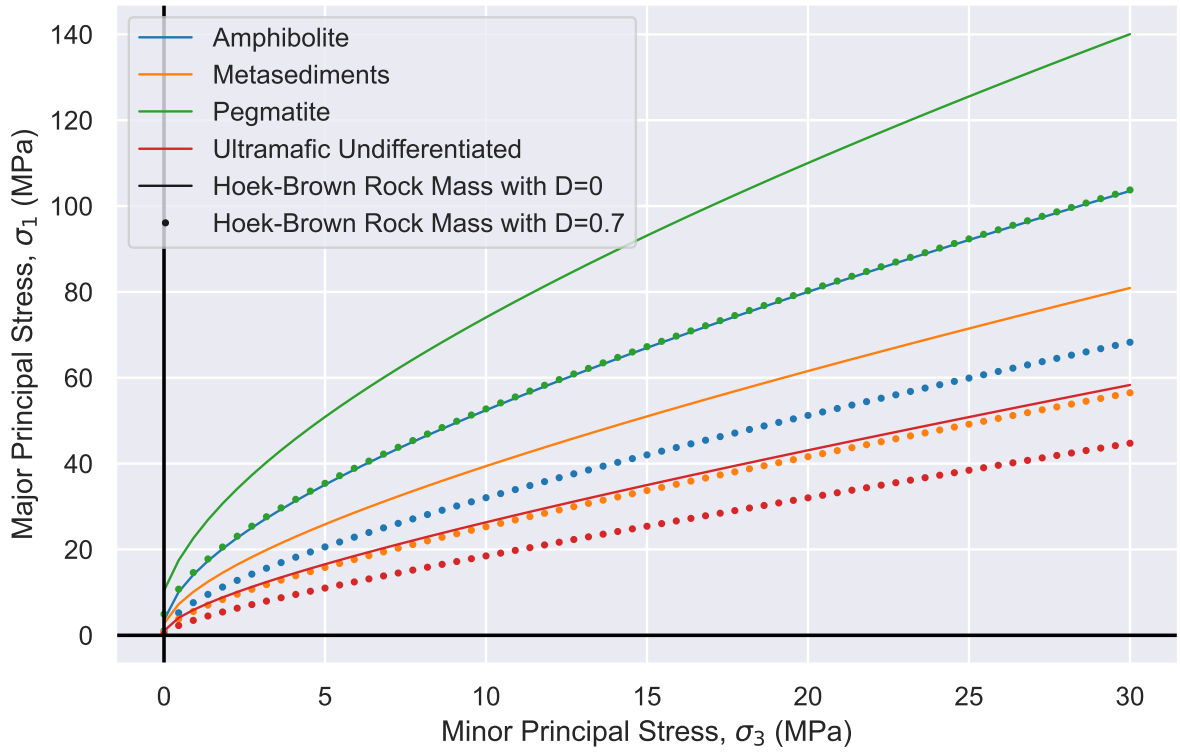


Figure 23. Rock mass failure envelopes.

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Summary of Rock Mass Properties

The resulting rock mass properties are presented in Table 38.

Table 38. Summary of rock mass strength properties of main lithologies.

Lithology	Disturbance Factor (–)	s (–)	a (–)	Tensile Strength (MPa)	σ_{RM} (MPa)	Constant m_b (–)	E_{RM} (GPa)
Amphibolite	0.0	0.0183	0.502	-0.54	26.59	6.69	54.4
	0.7	0.0054	0.502	-0.32	14.43	3.35	22.4
Metasediments	0.0	0.0183	0.502	-0.67	20.53	4.17	23.3
	0.7	0.0054	0.502	-0.40	11.12	2.09	9.6
Pegmatite	0.0	0.0868	0.501	-0.93	35.41	11.26	43.9
	0.7	0.0412	0.501	-0.67	24.40	7.37	23.1
Ultramafic	0.0	0.0129	0.502	-0.31	7.74	3.40	9.7
	0.7	0.0054	0.502	-0.18	4.20	1.70	4.0

Codes **s, a** Hoek-Brown rock mass material constant, σ_{RM} rock mass compressive strength, m_b Hoek-Brown rock mass constant, E_{RM} Young's modulus of the rock mass.
Notes Rock mass tensile strength obtained from eq. 19.

5.2 In Situ Stress Conditions

As no in situ stress measurements were conducted at Shaakichiuwaanaan, stress states have been estimated based on two (2) nearby sites: Éléonore mine (Guido, 2019) and Hydro-Quebec's LG2 Project (Heidbach et al., 2025). The major principal stress (S1) 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 (S2) is assumed to be horizontal and perpendicular to the orebody, with a magnitude of 0.7 times the vertical stress component and a locked-in value of 7 MPa. The minor stress (S3), defined as the vertical stress (σ_V), is calculated from the material weight. Table 39 summarizes the in situ stress tensor considered for the Project. Figure 24 presents the retained k-ratio (i.e., σ_H/σ_v) against nearby measurements and literature.

Note that the principal stress being expected to be parallel to the orebody is favourable from a stope stability perspective. When the maximum principal stress aligns with the orebody, the potential for stress-induced block detachment or kinematic failure along joints is reduced, and the rock mass is generally better supported by the in situ stress field. This orientation, therefore, contributes to enhanced overall stability.

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Table 39. In situ stress state assumptions.

Stress Component	Principal Stress Component	Magnitude Gradient (MPa)	Orientation	
			Trend (°)	**Plunge** (°)
Horizontal max (σ_H)	Major (S1)	0.040 × Z + 10	070	00
Horizontal min (σ_h)	Intermediate (S2)	0.019 × Z + 7	160	00
Vertical (σ_v)	Minor (S3)	0.027 × Z	–	90

Codes Z depth below surface, in meters.

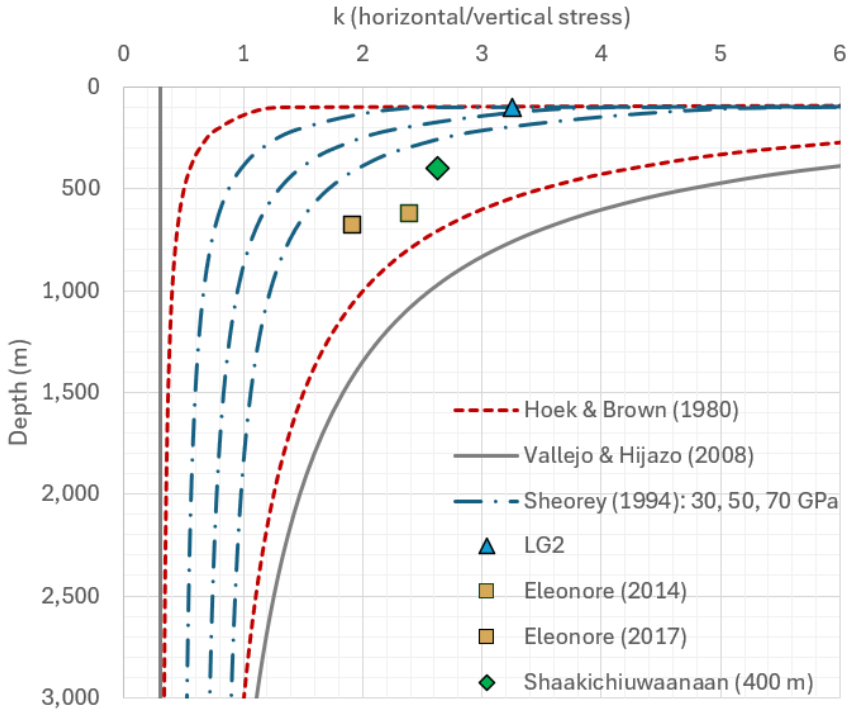


Figure 24. Comparison of the resulting k-ratio (σ_H/σ_v) at a depth of 400 meters against two nearby sites and literature.

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6 Geotechnical and Geomechanical Domains

Geotechnical and geomechanical domains refer to the division of the geotechnical/geomechanical model into several zones within the proposed limits of open pit and underground mines, which are also within the main weathering domains (i.e., soil and fresh rock). Geotechnical domains refer to materials that exhibit soil-like behaviour and share similar geological, structural, and material characteristics. Geomechanical domains refer to rock materials categorized based on one or several of the following: rock mass classifications, structure orientations (interpreted from stereographic projections), laboratory-derived strength envelopes, and/or expected rock mass behaviour. This section summarizes the geotechnical and geomechanical domains for open pit and underground mines.

Geotechnical and geomechanical domains represent subdivisions of the geotechnical/geomechanical model within the limits of the proposed open pit and underground mines, corresponding to the main weathering units (i.e., soil and fresh rock). *Geotechnical domains* refer to materials that exhibit soil-like behaviour and share similar geological, structural, and other characteristics. *Geomechanical domains* refer to rock materials expected to behave similarly based on rock mass classifications, structural orientations (interpreted from stereographic projections), and laboratory-derived strength envelopes. This section summarizes the geotechnical and geomechanical domains defined for the open pit and underground mines.

6.1 Open Pit

Within the pit footprint, only one geotechnical domain was identified, as the overburden materials across the site share similar characteristics. In contrast, several geomechanical domains were defined, each assigned to more than one lithological unit. These distinctions reflect spatial variations in geomechanical properties across the site, particularly related to structural orientations. Details of the open pit geotechnical and geomechanical domains are presented in Table 40.

The geomechanical domains were defined through a detailed analysis of spatial variations in discontinuity orientations across the open pit. The methodology involved subdividing the 2.8 km pit into eight zones and generating stereographic projections for each major lithological unit within each zone. Because the pegmatite intrusion separates the host rock into distinct north and south sectors, drillholes were classified accordingly, allowing comparison of structural orientations within the same lithology across the intrusion. This approach helped identify structural differences, which guided the subdivision into geomechanical domains, and highlighted areas with limited data coverage. Further details are presented in Section 5.1.3.

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Table 40. Open pit geotechnical and geomechanical domains.

Domain (GTD/GMD)	Weathering Domain	Lithology Code	Considered Drillholes
GTD-ALL	Overburden	N/A	**(CV24-)** 542, 558, 562, 775, 779, 780, 782, 784, 785, 786, 787
WAll-AMP	Fresh rock	AMP	**(CV23-)** 217, 219, 223, 236, 243, 317, 364 **(CV24-)** 521, 739, 777
WW-APU	Fresh rock	AMP, PEG, UMU	**(CV23-)** 217, 219, 223, 236, 243, 364 **(CV24-)** 739, 777
WS-META	Fresh rock	META	**(CV23-)** 217, 219, 223, 236, 243, 317, 364 **(CV24-)** 521, 739, 777
CN-AMP	Fresh rock	AMP	**(CF21-)** 001, 002, 003 **(CV22-)** 048, 065 **(CV23-)** 164 **(CV24-)** 456, 463, 479, 489, 514, 609
CN-META	Fresh rock	META	**(CV22-)** 040, 043 **(CV23-)** 164, 188 **(CV24-)** 423A, 456, 689
CS-AMP	Fresh rock	AMP	**(CF21-)** 001, 002, 003 **(CV22-)** 040, 043, 048, 065, 071, 074 **(CV23-)** 164, 188 **(CV24-)** 423A, 456, 463, 479, 489, 493, 514, 609, 623, 689, 781
CS-META	Fresh rock	META	**(CV22-)** 071, 074 **(CV24-)** 489, 493, 623, 781
CN-UMU	Fresh rock	UMU	**(CV22-)** 040, 043, 071, 074 **(CV23-)** 188 **(CV24-)** 423A, 493, 623, 689, 781
EAll-AMP	Fresh rock	AMP	**(CV22-)** 019, 022, 024, 055, 058 **(CV23-)** 130, 152, 157, 194, 197 **(CV24-)** 447
EN-UMU	Fresh rock	UMU	**(CF21-)** 001, 002, 003 **(CV22-)** 019, 022, 024, 055, 058, 048, 065, 066, 069, 042 **(CV23-)** 130, 152, 157, 194, 197 **(CV24-)** 447, 463, 479, 514, 609
EN-META	Fresh rock	META	**(CV22-)** 048, 052, 055 **(CV23-)** 152, 157 **(CV24-)** 514, 609
ES-META	Fresh rock	META	**(CF21-)** 002 **(CV22-)** 022, 024, 058, 065 **(CV23-)** 194, 197 **(CV24-)** 447, 463, 479
EE-AMPU	Fresh rock	AMP, PEG, META, UMU	**(CV22-)** 024, 052, 055, 058 **(CV23-)** 130, 152 **(CV24-)** 447
WCE-PEG	Fresh rock	PEG	**(CF21-)** 001, 002, 003 **(CV22-)** 022, 024, 040, 043, 048, 052, 055, 058, 065, 071, 074 **(CV23-)** 130, 188, 194, 197, 217, 219, 223, 236, 243, 317, 364 **(CV24-)** 423A, 447, 456, 463, 479, 489, 493, 521, 609, 739

Codes N/A Not applicable; GTD Geotechnical Domain; GMD Geomechanical Domain.

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6.2 Underground Mine

As described in Section 5.1.2, the geomechanical domains for the underground mine are based solely on rock mass classification and are therefore directly aligned with the lithological units, without further spatial subdivision. Details of the underground mine geomechanical domains are provided in Table 41.

Table 41. Underground mine geomechanical domains.

Domain (GMD)	Lithology Code	Weathering Domain	Considered Drillholes
UG-AMP	AMP	Fresh rock	**(CV24-)** 447, 459, 469, 496, 663, 679, 684, 769, 783
UG-META	META		
UG-PEG	PEG		
UG-UMU	UMU		

Code GMD Geomechanical Domain.

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7 Open Pit Design Guidelines

The development of slope design guidelines for open pits aims to achieve the steepest feasible slope angles to maximize resource recovery and reduce waste handling, while managing safety risks to personnel and equipment. These guidelines are established using a risk-based approach to derive slope recommendations that align with an acceptable level of risk. The open pit stability analyses incorporate this framework by applying design targets for factors of safety (FoS), spill containment requirements (catch bench sizing), and probabilities of failure (PoF), recognizing that zero risk is neither realistic nor desirable. In practice, open pit slopes are often considered overly conservative if no instabilities occur during operations. Consequently, a certain level of instability is expected and should be managed appropriately throughout the pit's development, operation, and closure phases.

7.1 Terminology and Definitions

The open pit wall terminology used in the upcoming sections is detailed in Figure 25.

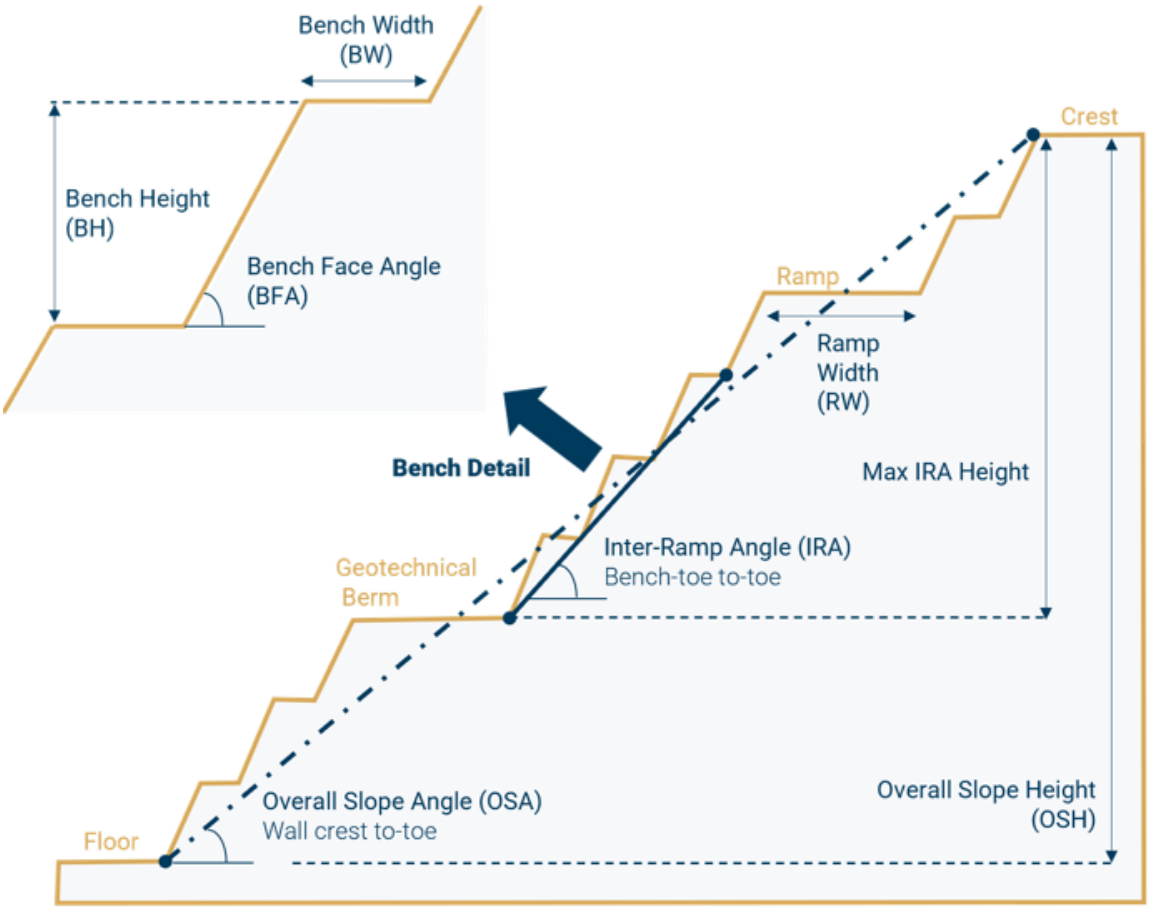


Figure 25. Pit wall terminology used in the report (modified from Read and Stacey, 2009).

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In addition to the terminology described above, the proposed design approach relies on several key concepts and probability definitions:

- **Blocks** – Used as a generic term for either planar or wedge failures.
- **Crest Loss** (backbreak) – Represents the portion of the catch bench width removed during blasting or scaling operations. This lost width is deducted from the designed bench width to determine the effective catch bench width available to retain rockfall debris.
- **Probability of Occurrence** (PoO) – Defined as the probability of encountering kinematically valid blocks. The criteria for kinematic validity depend on the failure mechanism. For planar failures, a block is considered valid if the strike of the discontinuity falls within a lateral tolerance of 20° from the slope orientation (Goodman, 1989), and if the dip of the plane is shallower than the bench face angle. For wedge failures, a block is valid if the line of intersection formed by two discontinuities daylights toward the interior of the pit and if its plunge is shallower than the bench face angle. The approach does not assume an initial bench face angle, allowing a full range of geometries to be kinematically tested.

$$PoO = \frac{N_v}{N_t} = \frac{Number\ of\ valid\ blocks}{Total\ number\ of\ blocks} \qquad \textit{(eq. 20)}$$

- **Probability of Sliding** (PoS) – Defined as the probability of a kinematically valid block to fall, which corresponds to a factor of safety below one (1). The factor of safety of each valid block is calculated using a limit-equilibrium method.

$$PoS = \frac{N_f}{N_v} = \frac{Number\ of\ sliding\ blocks\ (FoS < 1)}{Number\ of\ valid\ blocks} \qquad \textit{(eq. 21)}$$

- **Probability of Failure** (PoF) – Defined as the product of the probability of occurrence and the probability of sliding, representing the probability of generating unstable blocks.

$$PoF = PoO \times PoS = \frac{N_f}{N_t} = \frac{Number\ of\ sliding\ blocks\ (FoS < 1)}{Total\ number\ of\ blocks} \qquad \textit{(eq. 22)}$$

- **Spill Contained** – Refers to the probability that failed blocks (i.e., falling blocks) are completely retained on the effective catch bench.

$$Spill\ Contained = \frac{Number\ of\ contained\ blocks\ on\ catch\ bench}{Total\ number\ of\ failed\ blocks} \qquad \textit{(eq. 23)}$$

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→ **Spill Length for a Planar Failure** (S_p) – The distance travelled downslope by failed material following a planar sliding event.

$$S_p = H \times \left(\frac{\sin(90° - \phi)}{\tan(\phi)} + \cos(90° - \phi) \right) \quad \textit{(eq. 24)}$$

Where H = maximum length perpendicular to hypotenuse of the failed material (m)
ϕ = angle of repose of failed material (°)

$$H = \sqrt{\frac{2 \times A_S}{\frac{1}{\tan(\phi)} + \frac{1}{\tan(\theta - \phi)}}} \quad \textit{(eq. 25)}$$

Where A_S = area of the failing plane section (m^2)
θ = bench face angle (°)

→ **Spill Length for a Wedge Failure** is defined as the distance reached by the broken material if the block fails and is calculated as the largest of the two potential spill lengths, S_{w1} and S_{w2}, as described in Gibson et al. (2007):

$$S_{w_1} = \sqrt[3]{\frac{6KV}{\pi} \times \left(\frac{\tan\theta - \tan\phi}{\tan\phi \times \tan\theta} \right)} \quad \textit{(eq. 26)}$$

$$S_{w_2} = \sqrt{\frac{6KV}{L} \times \left(\frac{\tan\theta - \tan\phi}{\tan\phi \times \tan\theta} \right)} \quad \textit{(eq. 27)}$$

Where S_{w_1} = spill length corresponding to a conical spill (m)
S_{w_2} = spill length corresponding to a pyramidal spill (m)
K = swell factor
V = volume of block (m^3)
L = length of wedge (m)
θ = bench face angle (°)

→ **Risk** is defined as the probability that a failed block will travel beyond the effective catch bench width. It represents the combined likelihood of a block both failing and not being retained on the catch bench. It is calculated as:

$$Risk\ (\%) = \frac{PoF\ (\%) \times [100 - Spill\ Contained\ (\%)]}{100} \quad \textit{(eq. 28)}$$

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7.2 Design Acceptance Criteria

The design criteria used for the stability analyses are presented below.

Overburden

A minimum factor of safety (FoS) of 1.3 was targeted for the static analyses (1.1 for dynamic analyses), as the soil slopes pose a limited risk to the overall pit stability due to their relatively low height and the possibility of implementing corrective measures after excavation, if required. In addition, since the lower and thickest stratigraphic unit is till, it is assumed that the groundwater level will gradually decline as mining progresses, remaining within the base of the till layer.

Fresh Rock

The recommendations for the open pit slope angles are developed in accordance with the design acceptance criteria proposed by Read and Stacey (2009). Table 42 presents the static and pseudo-static (dynamic) factor of safety (FoS) and the probability of failure (PoF) criteria for various scales (i.e., bench, inter-ramp and overall).

Table 42. Design acceptance criteria (reproduced from Read and Stacey, 2009).

Slope Scale	Consequences of Failure [2]	Acceptance Criteria [1]		
		Minimum Static FoS	Minimum Dynamic FoS	Maximum PoF P[FoS<1]
Bench	Low-High	1.1	N/A	25–50%
Inter-ramp (IRA)	Low	1.15–1.2	1.0	25%
	Medium	1.2	1.0	20%
	High	1.2.–1.3	1.1	10%
Overall (OSA)	Low	1.2–1.3	1.0	15–20%
	Medium	1.3	1.05	5–10%
	High	1.3-1.5	1.1	<5%

Codes N/A not applicable, FoS factor of safety, PoF probability of failure.
Notes [1] Needs to meet all acceptance criteria.
[2] Acceptable risk can be defined using a semi-qualitative risk matrix: consequence vs. likelihood.

Regarding the catch bench containment, Read and Stacey (2009) state that: "In the case of retention capacity, it is neither practicable nor economical to design the benches to contain the spillage from every potential sliding plane or wedge failure in the benches above. Instead, the limit is set to the failed volume that can reasonably be contained on the bench. Usually, this is 70–85% of the potential failed volumes.".

When designing benches, the bench width (BW) is the principal parameter used to ensure adequate catchment and containment of potential rockfall. Bench width (BW) is generally assessed using the empirical equation proposed by Ryan and Pryor (2000):

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$$BW = 0.2 \times BH + 4.5 \quad \textit{(eq. 29)}$$

Where BH = bench height (m)

A minimum safety factor of 1.5 is targeted for both the IRA and the OSA. Compliance with this requirement is verified through modelling that incorporates bench parameters and ramp design for the IRA, as well as geotechnical berms for the OSA.

Through a risk-based approach, presented in section 7.3, bench design parameters aim to meet several design criteria:

- As indicated in Table 42, the target probability of failure (PoF) for planar or wedge failures in bench design is limited to a maximum of 25 to 50%.
- At least 70% of spills resulting from failed blocks must be fully contained on the catch benches.
- The minimum recommended bench width is 8.5 m, based on eq. 29, assuming a bench height of 20 m.
- The maximum bench face angle, as defined in accordance with PMET, is set to 75°.
- The maximum acceptable risk – which governs the spill containment requirements in eq. 28 – is defined in accordance with PMET and set to 5%.

For the Project, the adopted bench design criteria are summarized in Table 43.

Table 43. Summary of adopted bench design criteria and assumptions.

Design Criteria	Criteria
Bench face angle range	40–80°
Maximum predefined BFA	75° for the main pit walls
Maximum PoF	Target of 30% [1]
Backbreak assumption	1.5 m [2]
Bench height	20 m (double bench configuration)
Minimum bench width	8.5 m
Minimum spill containment	70% of spills to be completely retained
Maximum acceptable risk	5% [2]

Notes [1] Can be higher depending on other factors: risk evaluated, unit exposure on wall, spill width evaluation.
[2] For the design criteria standardization step, a backbreak of 1 m was assumed acceptable on the south walls.
[3] Can be higher depending on other factors: unit exposure on wall, involved joint sets' characteristics.

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7.3 Methodology

For the purposes of this Project, different design approaches have been applied depending on the material type and the scale of analysis. The materials are divided into two primary domains: soils (overburden) and fresh rock. Accordingly, the open pit slope stability has been assessed using the following methodology:

01. **Soils – Overburden Slopes**
    → Limit equilibrium analysis (LEA)
02. **Rock – Bench Scale**
    → Kinematic analyses
    → Limit equilibrium analysis (LEA)
03. **Rock – Inter-Ramp Scale & Overall Scale**
    → 3D numerical modelling

Details of the analyses methodology are presented in the following sections.

Overburden Slopes

Slope design criteria for the overburden were established through 2D limit equilibrium analyses using *Slide2* (Rocscience, 2025b). This software enables the evaluation of slope stability under a range of geological, groundwater, and loading conditions by incorporating the geometry, stratigraphy, and shear strength parameters of each material unit. External forces such as pore pressures, seismic loading, and reinforcement elements can also be included.

The analyses were performed using the Spencer and GLE/Morgenstern-Price methods, both of which account for interslice forces. These methods provide a realistic representation of complex geometries and variable material conditions, enabling a reliable estimate of the factor of safety.

Because the stratigraphic layers were not sufficiently defined to support realistic cross-sections around the pit area, stability analyses were completed using theoretical slope profiles. A slope height of 20 m was adopted, representing the typical maximum height expected near the mine. However, the specific height has limited influence on the results, as cohesion was not included in the material properties.

Bench Scale

Understanding and identifying the potential failure modes at the bench scale is a key component of an appropriate design. The three (3) basic mechanisms of structurally controlled failures in rock slopes are planar failures, wedge failures and toppling failures, as described below:

01. **Planar Failure** is kinematically possible when a discontinuity plane is less inclined than the slope face (daylights) and at an angle steeper than the friction angle. Planar failures will generally only develop to a significant extent if the strike of the joint is within ±20° from the strike of the rock slope.

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02. **Wedge Failure** is kinematically possible when the plunge of the intersection of two planes (sliding vector) is less inclined than the slope face (daylights) and at an angle greater than the combined friction angle, which is determined from the characteristics of each plane that forms the wedge. Wedge failures will develop to a significant extent if the azimuth of the line of intersection is within ±45º from the dip direction of the slope face.

03. **Toppling Failure** may develop when a rock mass contains multiple, parallel, steeply dipping continuous geological structures, such as continuous joints/foliation planes, that strike nearly parallel to the strike of the rock slope face. Toppling failure will generally only develop when the strike of the structures is within ±20º from the azimuth of the slope face. Kinematically, the potential for toppling failure is determined by the spacing (separation), inclination, and continuity of the toppling blocks, as well as the slope angle. Wide spacing or discontinuous structures will mitigate the potential for toppling.

A clear understanding of the geomechanical context is required before performing bench-scale stability assessments. For the Shaakichiuwaanaan project, geomechanical characterization (Section 5) identified the geomechanical domains and major discontinuity sets through stereographic projection analysis. Expected lithologies along the pit walls were then reviewed to subdivide the pit into sectors based on both geomechanical domains and slope orientations.

Following this sectorization, bench-scale stability analyses were completed using the risk-based approach described below and illustrated in Figure 26.

01. **Assessment of PoO, PoS and PoF**
    - → For a defined range of bench face angles, the probabilities of planar and wedge occurrence, sliding and failure (eq. 20, eq. 21 and eq. 22) are calculated using limit equilibrium analyses. Planar failures are evaluated using an Alius in-house *Python* code, while wedge failures are evaluated using *SWedge* (Rocscience, 2024b). Post-processing of all results is also completed using an in-house *Python* script.
    - → For planar sliding assessments, all surveyed discontinuities are initially considered, followed by a targeted analysis that includes only discontinuities belonging to major sets.
    - → For wedge analyses, when a sector contains more than 500 discontinuities, random sampling is used to reduce the dataset to 500 entries for computational efficiency. The sampled dataset is validated to ensure major joint sets remain adequately represented.

02. **Evaluation of Potential Spill Length**
    - → At each BFA, spill lengths are computed for both planar and wedge failures (eq. 24, eq. 26, and eq. 27). Only wedges exceeding 2 m^3 are considered at this stage, as smaller blocks are assumed to be manageable operationally.

03. **Identification of the Limiting Design Factor**
    - → The limiting factor corresponds to one of the following:
        - Planar or wedge failure probabilities exceed an acceptance criterion.
        - Spill width requirements exceed the minimum allowable bench width.
        - The predefined maximum bench face angle is reached.

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04. **Selection of Bench Face Angle**
    - → The BFA associated with the higher IRA that respects all predefined criteria is retained as the "Selected BFA".
05. **Bench Width Requirement**
    - → The evaluation of spill containment requirement is obtained using the risk equation (eq. 28), based on the maximum acceptable risk and the PoF associated with the Selected BFA.
    - → The required bench width is the greater of the minimum catch bench width (i.e., 8.5 m), and the spill containment requirement plus a 1.5 m allowance for crest loss (backbreak).
06. **Evaluation of the Inter-Ramp Angle**
    - → The resulting IRA can be calculated using the parameters retained above.

A distinct workflow is executed for each pit sector. Within this framework, the bench face angle is the principal variable being optimized. After optimization, the proposed design guidelines are standardized within a design sector or, when appropriate, with an adjacent design sector.

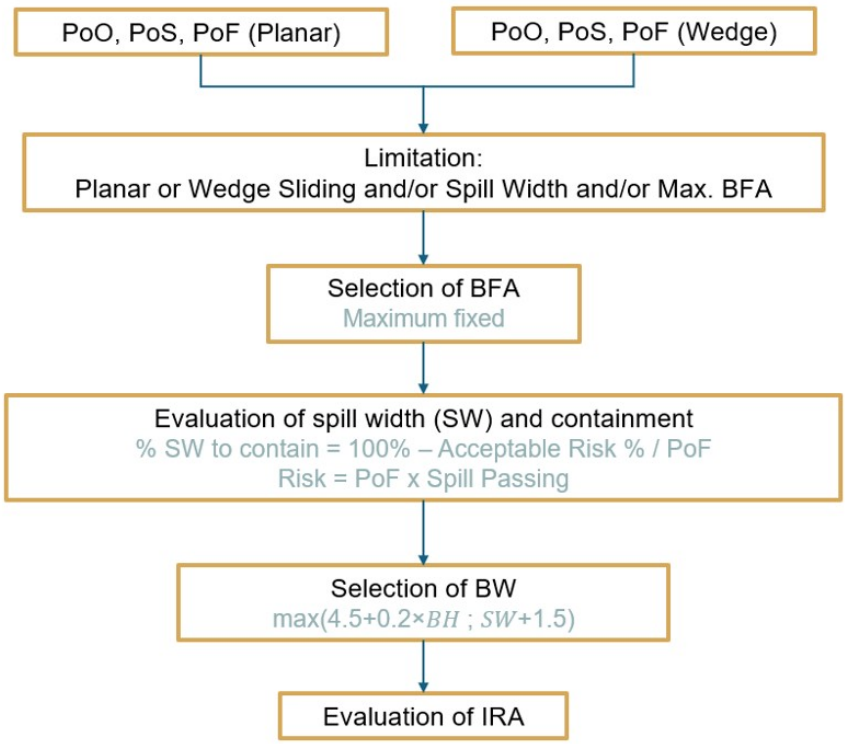


Figure 26. Bench scale design optimization flow chart.

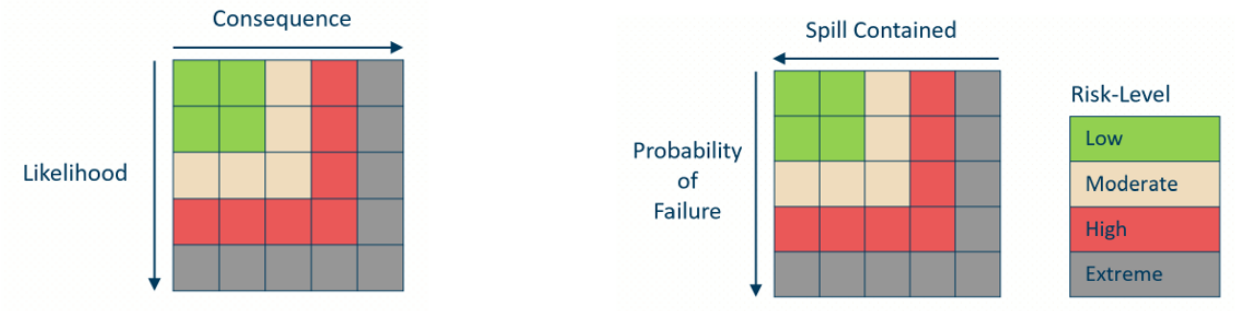
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In summary, the geomechanical risk-based approach evaluates risk as a function of two key components:

01. **Probability of failure (PoF), representing the likelihood of a block failing,**
02. **Containment capacity of the catch bench, representing the consequence of failure.**

This relationship is typically illustrated using a risk matrix, as shown in Figure 27. Likelihood is derived from the PoF associated with a given wall orientation and its structural data. The consequence is assessed by comparing the estimated failure volume with the effective width of the catch bench. When failure volumes exceed the bench's containment capacity, the consequence level increases, thereby increasing operational hazard.

The matrix allows for both quantitative and qualitative risk classification. Quantitatively, risk can be computed as the product of PoF and containment effectiveness (Spill Contained). Risk can also be presented using qualitative terms (e.g., low, moderate, high, extreme), both typically represented using a colour-coded scheme. The risk tolerance thresholds can be modified to reflect the acceptable level of operational risk.



(a) Conceptual matrix. *(b) Qualitative risk matrix.*

Figure 27. Typical risk matrices.

Other relevant assumptions and/or limitations include:

- Depending on the continuity of structural features, kinematic failures may affect slopes at multiple scales, from bench to overall. Persistence is therefore an important consideration when evaluating failure potential. Until mining exposes the structures and their actual persistence can be measured, a default persistence of 15 m is assumed for design purposes.
- The primary failure mechanisms considered are planar and wedge sliding. Although toppling is acknowledged as a potential mechanism, it is treated as an operational hazard rather than a design-limiting mode for bench geometry.

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Inter-Ramp & Overall Scale

Given the good rock mass conditions, design limitations are expected to be structurally controlled. After applying the risk-based framework and defining the bench design parameters, no further optimization of the inter-ramp or overall slopes was undertaken, as steeper configurations are not feasible with the fixed bench height and required catch bench widths. Nonetheless, verification of pit stability remains essential. Inter-ramp and overall slope stability were evaluated using 3D numerical modelling to confirm that the proposed design meets the required factors of safety. Compared with traditional 2D methods, the 3D analysis provides a more realistic representation of the pit geometry and the influence of the spatial distribution of geological units. It also enables assessment of potential failure mechanisms associated with structural features – such as contacts or weaker units – that may extend across multiple pit sectors. The modelling results were used to confirm global pit stability under static conditions, provided in Section 7.6.

7.4 Overburden Stability Assessment

7.4.1 Design Methods

Overburden slope stability was evaluated using numerical modelling. An initial model was developed by combining all stratigraphic units according to their average proportions observed on site. Additional models were prepared for each stratigraphic unit individually, as well as for the lake-bottom sediments alone, to account for scenarios in which a single unit dominates the slope profile. The slope geometry was then iteratively adjusted until the target factor of safety (FoS) was achieved.

7.4.2 Results

Figure 28 presents the representative section and the limit equilibrium analysis results used to support the slope design optimization. An inter-ramp angle (IRA) of 36° meets the target factor of safety of 1.3.

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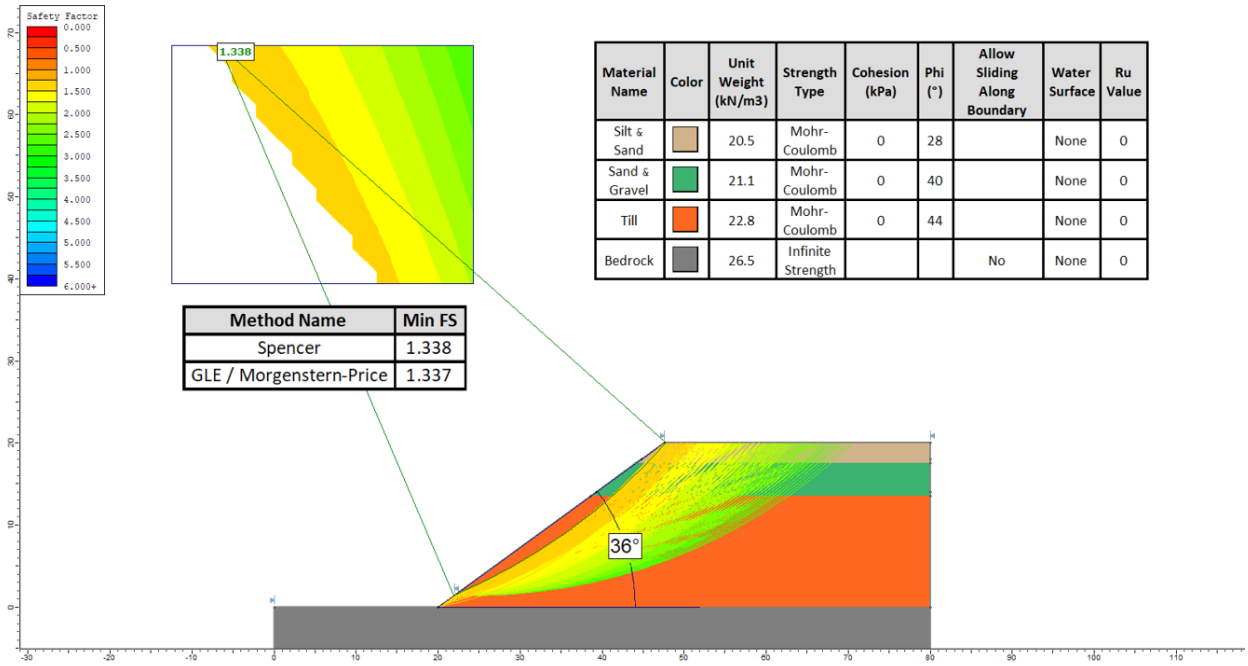


Figure 28. Stability analyses result for a typical overburden slope.

The recommended overburden slope design is presented in Table 44. These guidelines ensure that a FoS of 1.3 is reached under static conditions and 1.1 under dynamic conditions. In addition, it is recommended to establish and maintain a catch bench at the interface between the overburden and the bedrock. The bench width should be 5 m when the overburden thickness is less than 5 m, and 10 m when the overburden thickness exceeds 5 m. This feature will facilitate the installation of a collector berm and help prevent overburden exposure at the pit crest if the bedrock surface differs from the anticipated geometry.

Table 44. Design guidelines for overburden.

Slope Composition	**BFA** (°)	**Max BH** (m)	**BW** (m)	**IRA** (°)
Typical Section	36	25	0	36
Silt & sand only	24	25	0	24
Sand & gravel only	34	25	0	34
Till only	38	25	0	38
Lake bottom sediments	38	25	0	38

Codes **BFA** bench face angle, **Max BH** maximum bench height, **BW** bench width, **IRA** inter-ramp angle.
Notes For bench heights over 25 m, include a 5 m width catch bench at mid-slope.

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7.5 Bench Stability Assessment

7.5.1 Design Methods

For each sector presented in the following section, the potential for both planar and wedge failures was evaluated and used as a basis for the bench design optimization.

For planar failures, a distinction is made between major, minor, and random joint sets. The analysis is therefore carried out in two steps: first, using all available poles; and second, focusing specifically on the poles belonging to major sets that present failure potential within the considered sector. In both cases, only poles satisfying the kinematic criteria are included. Ultimately, the design recommendations are derived from the analysis restricted to the major sets.

For wedge failures, the number of discontinuity combinations assessed in each geomechanical domain is summarized in Table 45.

Table 45. Wedge failure analysis – Joints combinations.

GMD	Lithology Unit Code	Number of Discontinuities	Number of Discontinuities After Sampling	Number of Combinations
WAll-AMP	AMP	708	500	124 750
WW-APU	AMP, PEG, UMU	1 265	500	124 750
CN-AMP	AMP	972	500	124 750
CN-META	META	423	423	89 253
CS-AMP	AMP	2 221	500	124 750
CS-META	META	324	324	52 326
EAll-AMP	AMP	1 106	500	124 750
EN-UMU	UMU	191	191	18 145
ES-META	META	325	325	52 650
EE-AMPU	AMP, PEG, UMU, META	1 046	500	124 750

Codes GMD Geomechanical Domain (fresh rock).
Notes Note that the wedge analysis was not performed for GMD WS-META, CN-UMU, EN-META and WCE-PEG. Refer to Table 40 for GMD presentation.

7.5.2 Open Pit Sectorization

The open pit sectorization is based on the pit slope orientation and the exposed geomechanical domains. The PEA 3D design and the lithological model provided by GMS on January 29, 2025, were used to define the design sectors presented in this section. The pit sectors are illustrated in Figure 29 and summarized in Table 46. A total of fifteen (15) sectors were evaluated during the bench-scale analyses; however, where appropriate, these sectors were subsequently grouped to provide simplified design guidelines for the pit walls.

An updated version of the open pit design guidelines reflecting the latest FS pit design available in March 2025 is provided in Appendix D. This version, which also forms the baseline case for the numerical modelling, is further discussed in Section 7.6.2.

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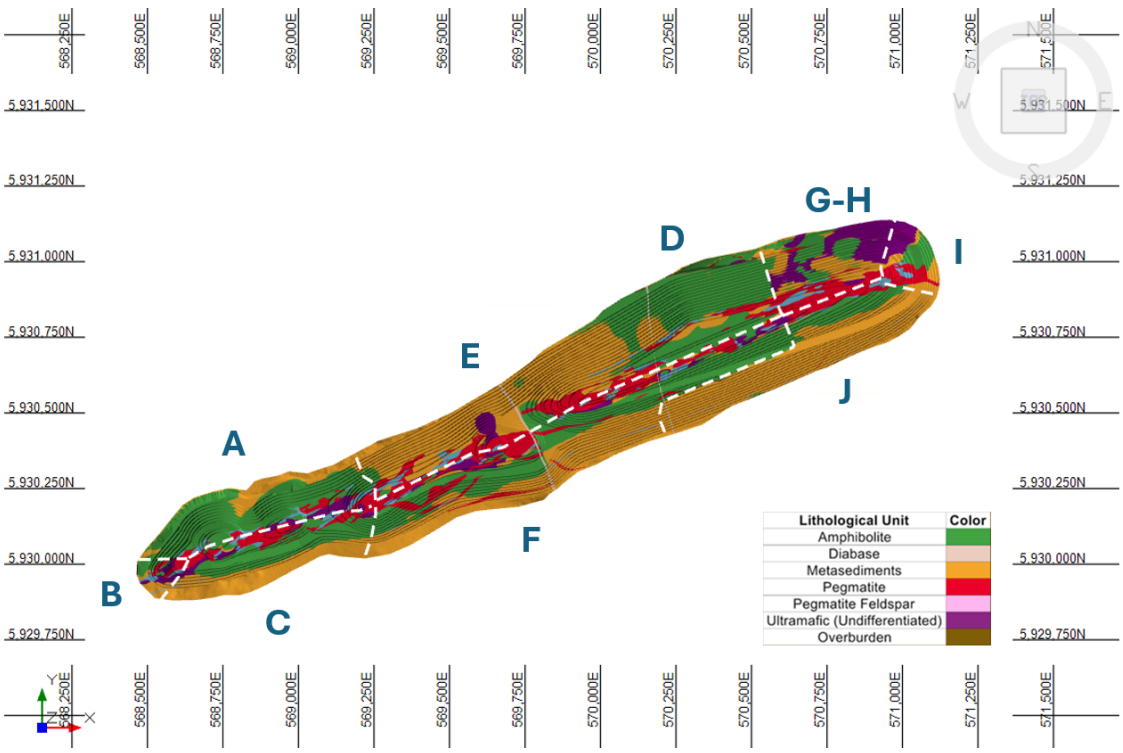


Figure 29. Open pit design sectors.

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Table 46. Pit design sector characteristics.

Sector	Lithology	Location	Main Dip Direction (°)	Dip Direction Range (°)	Corresponding GMD
A	AMP	North wall (West)	160	130–200	WAll-AMP
B	AMP, PEG, UMU	West wall (West)	060	005–130	WW-APU
C-1	AMP	South wall (West)	340	340–005	WAll-AMP
C-2	META	South wall (West)	340	340–005	WS-META
D	AMP	North wall (Central)	160	150–165	CN-AMP
E-1	META	North wall (Central)	155	140–155	CN-META
E-2	UMU	North wall (Central)	155	140–155	CN-UMU
F-1	AMP	South wall (Central)	335	330–350	CS-AMP
F-2	META	South wall (Central)	335	330–350	CS-META
G-1	AMP	North wall (East)	165	165–180	EAll-AMP
G-2	META	North wall (East)	165	165–180	EN-META
H	UMU	North wall (East)	165	165–180	EN-UMU
I	AMP, PEG, UMU, META	East wall (East)	250	180–310	EE-AMPU
J-1	AMP	South wall (East)	335	310–335	EAll-AMP
J-2	META	South wall (East)	335	310–335	ES-META

Codes GMD Geomechanical Domain (fresh rock).
Notes Main Dip Direction: Dip direction used in the kinematic analysis.
Refer to Table 40 for GMD presentation.

As shown in Figure 29, pegmatite exposure along the final pit walls is relatively limited. It is primarily present on the two end walls (sectors B and I), although it is interlayered with other lithological units. Pegmatite also occurs locally in the central wall area, where it appears as a thin intrusive band over only a few benches. Due to the impracticality of applying different guidelines to such small-scale exposures, pegmatite was not used as a basis for defining sectors. Nevertheless, the stability of benches within the WCE-PEG geomechanical domain was evaluated using practical experience, and guidance for localized pegmatite exposures is provided should operational implementation be

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considered feasible.

Figure 29 also shows that certain geomechanical domains – such as UMU in sector E, META in sector D, and UMU in sector H – occur only locally along pit walls otherwise dominated by other domains. These minor occurrences were also included in the stability assessment.

Finally, faults and the diabase dyke were not included in the sectorization process.

7.5.3 Results

For each open pit design sector, the workflow illustrated in Figure 26 was applied to determine BFA results ranging from 40 to 80°, in 2° increments. When the recommended BFA fell between the evaluated angles, interpolation was performed. Detailed results for all sectors are provided in Appendix E.

Overall, design constraints are primarily driven by the potential for planar failures – particularly along the north wall, where foliation is expected to control the bench face angles. Minor wedge failures may occur throughout the pit, but they do not limit the design in any sector.

An example of the stability analysis process is presented in Figure 30, which illustrates the results for Sector A (AMP) in the west area of the north wall. The upper plot shows the PoO, PoS, and PoF; the middle plot shows the spill width of the failed material; and the lower plot presents the resulting inter-ramp angle. Together, these plots highlight the trade-off between inter-ramp angle (IRA) and bench stability as a function of the bench face angle (BFA). While a steeper BFA may increase the IRA and improve project economics, it also increases both the probability of failure and the likelihood that failed material will exceed the bench containment width. In such cases, the bench width must be increased to ensure containment. Table 47 presents the retained design parameters for this sector, along with all associated inputs and results.

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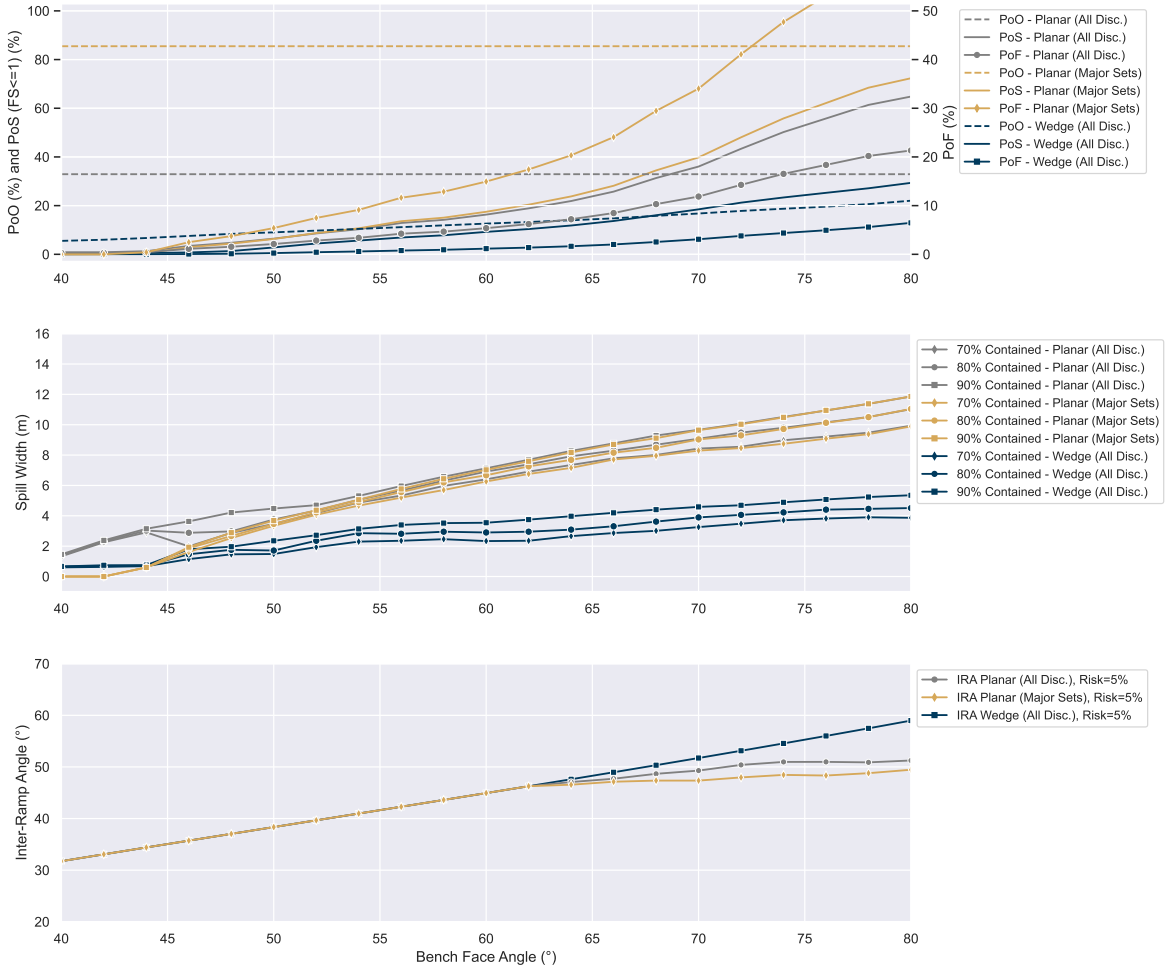


Figure 30. Bench scale stability analyses results - Sector A AMP.

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Table 47. Bench design optimization parameters – Sector A (AMP)

Parameter	Sub-Parameter	Value
Probabilities [1]		
Planar Failures [2]	PoO	85.5%
	PoS	31.3%
	PoF	29.5%
Wedge Failures	PoO	15.8%
	PoS	16.0%
	PoF	2.5%
Design Constraints		
Limitation		Planar sliding & Minimum bench width
Design Outputs		
Bench Face Angle (BFA)		68°
Spill Contained (% Contained, Width) [3]		85%, 9.0 m
Bench Width (BW)		10.5 m *from max(8.5 ; 9.0 + 1.5)*
Inter-Ramp Angle (IRA)		47°

Notes [1] Probabilities for selected BFA.
[2] Probabilities considering the major joint set assessment.
[3] Based on the limiting spill length at the selected BFA, either from planar or wedge failure.

In **Sector A**, within the amphibolite:

- One major discontinuity set (JS1) poses potential issues from a planar-sliding perspective. JS1 consists primarily of foliation, open joints, and closed joints, with an average spacing of 2.3 m.
- Wedge sliding is not identified as a limiting mechanism for this sector and lithology. The selected BFA is therefore governed by the planar PoF, and the catch bench width is adjusted to retain 85% of spill events.
- Under these conditions, the resulting risk (eq. 28) is 4.4%.

Within the pegmatite geomechanical domain (WCE-PEG):

- A bench face angle of 75° is considered acceptable over a limited extent, based on stereonet projection review.
- Three joint sets were interpreted within this geomechanical domain.
  - Joint sets JS1 and JS3 may represent the same structural set with variable dip directions around 90°. However, for conservatism, they were treated as two distinct sets with an average dip of 77°.
  - JS2 has an average dip of 8°. Based on the interpreted joint set orientations, no kinematic structural limitations are anticipated for the proposed bench design.

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- Nevertheless, the pegmatite should be closely monitored during operations, as its coarse-grained texture and pronounced mineral cleavage provide preferential planes of weakness that can contribute to localized breakage when exposed on the pit wall.

For the ultramafic undifferentiated unit:

→ Kinematic analysis indicates significant design limitations due to unfavourable joint orientations.

→ In addition, this lithology exhibits lower rock mass quality than the other units; therefore, ultramafic exposures on the pit walls should be minimized wherever practical.

The following content summarizes the bench-scale design guidelines and the associated risks:

01. **Recommendations on Bench Width**

→ For a double benching configuration, with total bench heights of 20 m, recommended bench widths vary from 8.5 m to 12 m, depending on the pit sector.

→ The proposed bench widths account for a non-usable width of 1.5 m, which includes crest losses and hard-toes. For the standardized design criteria, a backbreak of 1 m was considered acceptable on the south walls. Achieving the recommended widths will rely on the consistent application of best practices in drilling and blasting. Backbeak should be monitored closely, and design adjustments evaluated based on operational performance.

02. **Recommendations on Bench Face Angle**

→ Based on available structural data, recommended bench face angles vary from 40° to 75° depending on the pit sector.

Figure 31 presents the risk representation for the pit design sectors, based on the relationship between the probability of failure derived from the stability analyses and the percentage of spills contained, which reflects the efficiency of the catch bench in retaining material in the event of failure. Each sector is shown as a data point indicating its PoF and spill containment for both the optimized and standardized design. To identify the sectors on the chart, the pit sector ID (A-H) is used, with the associated geomechanical domain indicated. If the geomechanical domain exists for both the north and south walls, the position is specified on the graph.

If standardized design guidelines differ from the optimized guidelines, an arrow linking the two points is shown on the chart. The risk curves displayed are derived as a function of PoF and spill containment (eq. 28) and represent reference thresholds corresponding to different risk levels: **very low** (2.5%), **low** (5%), **moderate** (10%), and **high** (15%). Risk is visually assessed by comparing each sector's position relative to these coloured curves.

Therefore, the position of each point on the chart reflects the relative risk of each design sector, with higher PoF values and lower spill containment indicating greater overall risk. Based on the plotted data, most design sectors lie within the very low to low risk range, demonstrating that the proposed design guidelines generally achieve acceptable risk levels. A few points fall outside the light-green envelope, indicating that they exceed the acceptable design criteria related to PoF or

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spill containment, as defined in the referenced guidelines and in Table 43. One point also plots within the "high" risk zone, above the "moderate" risk curve. However, when considering operational constraints and the applied risk-management approach, the design guidelines associated with these sectors are still deemed acceptable, despite their location outside the reference thresholds:

Accordingly, the position of each point on the chart reflects the relative risk of each design sector, with higher PoF values and lower spill containment indicating greater overall risk. Based on the plotted data, most design sectors fall within the very low to low risk range, highlighting that the proposed design guidelines generally achieve acceptable risk levels. A few points lie outside the light-green envelope, indicating that they exceed one or more of the acceptable design criteria related to PoF or spill containment, as defined in the guidelines and in Table 43. One point also lies within the "high" risk category, above the moderate risk curve. However, given operational constraints and the overall risk-management approach, the associated design guidelines are still considered acceptable, despite falling outside the reference thresholds:

- **C-1** and **F-1** – Corresponding to amphibolite exposures on the south walls in the west and central areas. Their spill containment reduced after the standardization process; however, it is deemed reasonable due to the low planar and wedge PoF associated with these sectors. The resulting evaluated risks are 3.3% and 5.8%, respectively.
- **C-2** – This sector corresponds to metasediment exposed on the south wall of the western area. It exhibits an high PoF, and the standardization process results in 70% spill containment. This combination yields a high risk rating (10.8%). Nonetheless, the limited exposure along the final pit wall reduces its overall significance from a design-risk perspective.
- **E-1** and **J-2** – These sectors correspond to metasediment exposures on the north wall of the central area and the south wall of the eastern area. Both sectors have PoF values exceeding 50%. However, the higher PoF is mitigated by applying spill containment targets of 95% and 87.5%, respectively, which keep the resulting evaluated risks within the moderate to very low ranges.

Design Sector E-2 (CN-UMU) is not included in this chart because it is recommended to remove the ultramafic unit locally exposed along the walls of Sector E.

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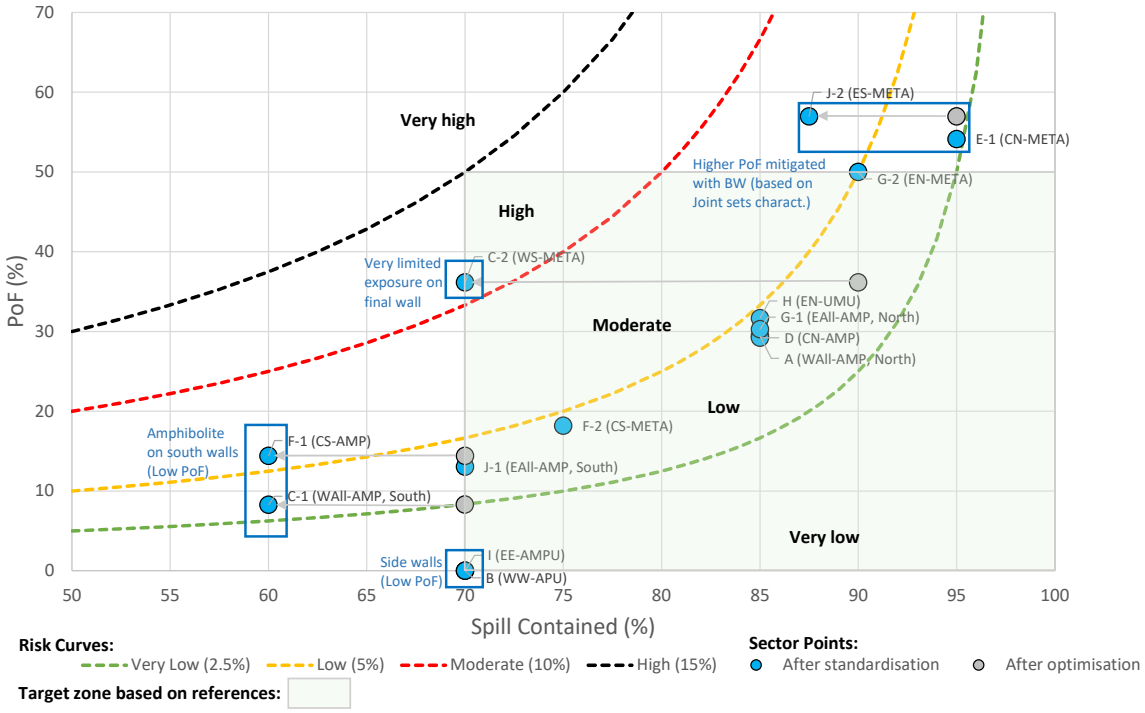


Figure 31. Risk representation for the design sectors compared to design criteria.

Double Benching

Double benching, resulting in a total bench height of 20 m, is considered feasible given the generally good rock quality and relatively low fracture frequency. Double benching is not appropriate in the following situations (Ryan and Pryor, 2000):

- The geological structures vary widely in dip. In this case, it is often difficult to get final dig faces to match from one working level to another, creating offsets between working levels, local overhangs, and other safety hazards.
- The rock mass is so fractured that it can be freely dug without the aid of blasting.
- There are many long joints or faults that daylight in the slope, which increase the potential for large failure geometries to develop when mining the lower bench level. Such failure geometries can be readily accommodated in operation when the slope height in failure is 15 m, as the equipment can readily scale the full bench height. However, when daylighted structures cause a slope failure of up to 30 m in height, the entire face cannot be adequately scaled, posing a safety hazard. For a given sliding plane, the failure volume actually increases geometrically rather than proportionally. The failure volume for a double bench can be three to four times that of a single bench for the same structures.

At this stage of the Project, none of the above conditions are considered likely. This conclusion is supported by the results of the numerical modelling at the inter-ramp and overall scales, presented in the following sections.

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7.6 Inter-Ramp and Overall Slope Stability

Slope stability at both the inter-ramp and overall scales was assessed using a numerical model, as described in the following sections.

7.6.1 Design Methods

The 3D numerical modelling conducted in this Project to assess the inter-ramp and overall slope stability was completed using *FLAC3D* (Itasca Consulting Group, 2025a), a modelling software designed to simulate the response of rock mass to stress variations induced by excavation, making it particularly suitable for mining applications. The IMASS (Itasca Model for Advanced Strain Softening) constitutive model (Itasca Consulting Group, 2025b) was used to characterize the strength behaviour of the rock masses surrounding the pit area. This model accounts for the progression of damage, from intact rock to disintegrated material, taking into consideration the effects of dilatancy and bulking during plastic deformation. The progressive weakening of properties in two stages distinguishes plastic damage from subsequent fracturing, allowing for a realistic representation of the rock mass post-peak behaviour. Overburden was modelled using an elastic model.

7.6.2 Numerical Model

A 3D model was built, and extents along the E-N-Z axes are approximately 4.9 $\times$ 3.0 $\times$ 0.6 km. This model incorporates the lithological model and the planned open pit. Additionally, host rock material is added as a filler around the model to ensure that boundary conditions are far away and do not affect the results.

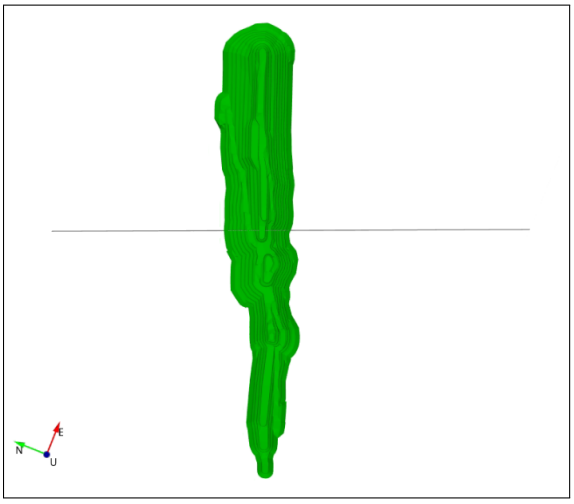
Geometry

The modelled open pit geometry is based on the March 2025 pit design. Geological solids derived from the lithological model were used to assign appropriate material properties to each zone within the numerical model. Faults identified on site were not incorporated in the current model due to their predominant northwest strike, which is approximately perpendicular to the pit slopes. Moreover, structural characterization of these faults is still in progress. It is recommended that the final interpretation of the fault network be integrated into a subsequent 3D model to assess and validate its potential impact on overall slope stability.

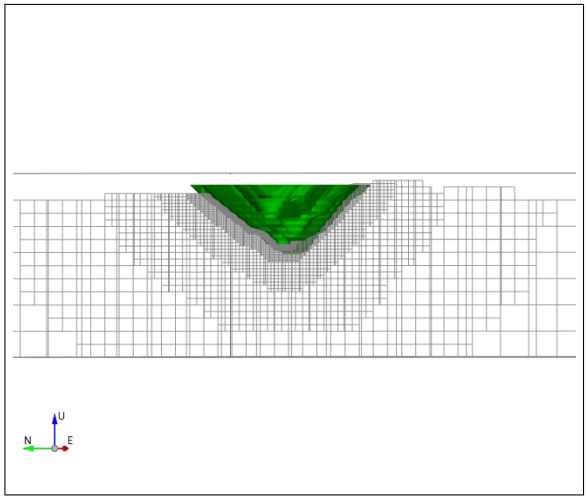
Mesh and Boundary Conditions

A structured cubic (octree) mesh was used to build the model. The zones adjacent to the pit walls have edges of 2.5 m, and the zones gradually increase in size to reach 80 m edges at a distance of more than 310 m from the pit. Figure 32 shows a section view where the mesh is visible for the pit area.

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(a) Section location intersecting the pit.



(b) Mesh refinements on the section plane.

Figure 32. View of the open pit numerical model structured (octree) mesh.

The in-situ stresses used in the numerical model are those presented in Table 39 (Section 5.2). Boundary conditions were applied by fixing all model boundaries except for the positive Z axis, which was assigned a free (unrestrained) condition.

Material Properties

Material properties used in the numerical model were derived from the geomechanical characterization, with the exception of the host rock. Mohr-Coulomb is modelled as a perfectly-plastic material, while Hoek-Brown is modelled using the IMASS constitutive model, an elasto-plastic model that accounts for cohesion and tension weakening, frictional strengthening, post-peak brittleness, and dilational behaviour. The overall process, known as "strain softening," results from strain-dependent material properties.

An example of a material exhibiting strain-softening behaviour is illustrated in Figure 33. Finally, the parameter known as the "critical plastic strain" (*critical plastic shear strain* in English) – that is, the amount of inelastic deformation required for the material to transition from peak to residual strength – is evaluated as follows (Lorig and Varona, 2013):

$$\epsilon^{p}_{cr_1} = \frac{12.5 - 0.125 \times GSI}{100 \times d} \qquad \textit{(eq. 30)}$$

Where $\epsilon^{p}_{cr_1}$ = critical plastic shear strain.
d = dimension of a numerical zone (m).

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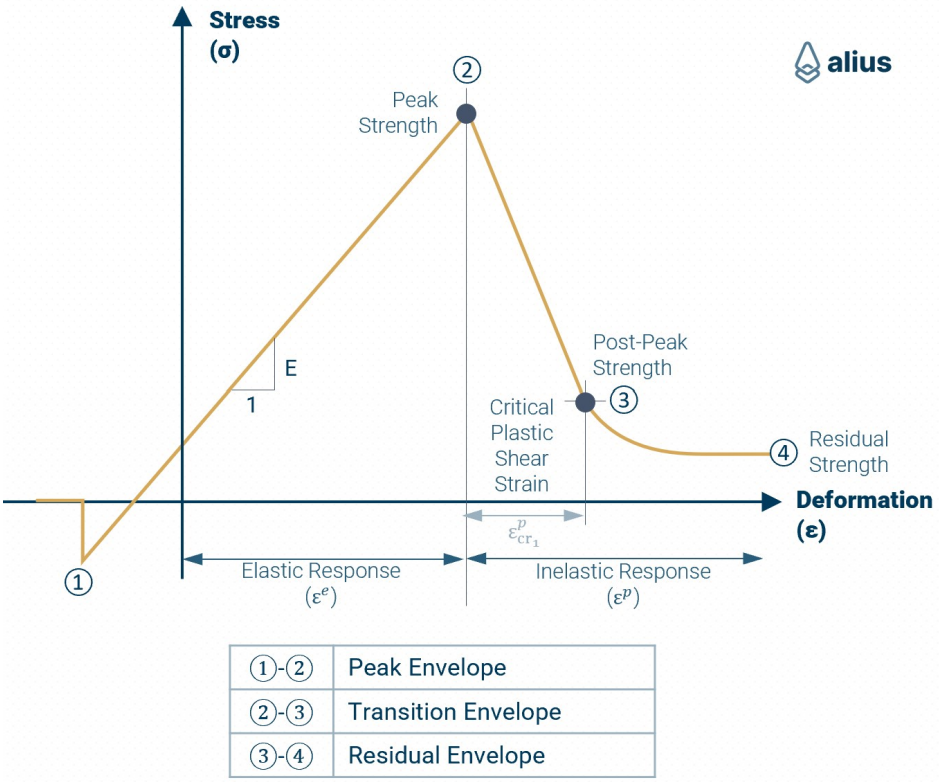


Figure 33. Terms and definitions associated with a strain-softening material.

The numerical model parameters are summarized in Table 48.

Table 48. Material properties of the open pit numerical model.

Unit	Constitutive Model	Density (kg/m^3)	E_i (GPa)	ν (–)	σ_i (MPa)	m_i (–)	GSI (–)
AMP	Hoek-Brown	2 984	89.2	0.22	198.2	24.2	71
Host rock	Elastic	2 700	70.0	0.20	–	–	–
META	Hoek-Brown	2 876	38.3	0.22	153.0	15.1	64
OVB	Elastic	2 321	1.0	0.20	–	–	–
PEG	Hoek-Brown	2 636	51.2	0.20	120.4	24.7	64
UMU	Hoek-Brown	2 874	15.9	0.22	57.7	12.3	64

Notes Hoek-Brown is modelled using the IMASS constitutive model.
Codes E_i Young's modulus of the rock mass, ν Poisson's ratio, σ_i unconfined compressive strength of intact rock,m_i Hoek-Brown intact rock constant, **GSI** Geological Strength Index, φ friction angle, **c** cohesion.

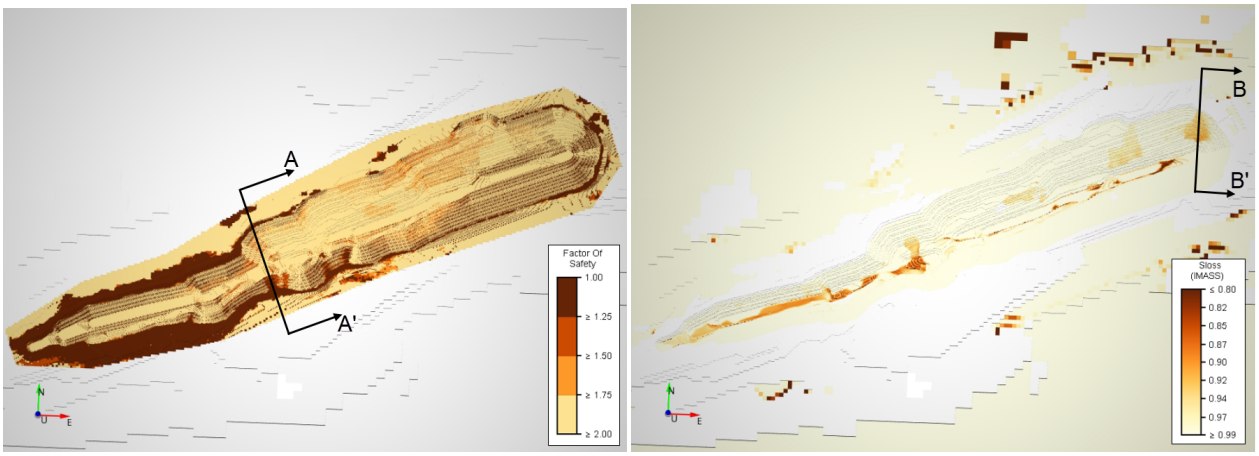
Sequence

Given the shallow depth of the planned pit, the mining sequence was modelled as a single stage corresponding to the full excavation of the pit.

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Modelling Results – Open Pit

The final open pit design was evaluated using numerical modelling to verify that the resulting factors of safety meet the target criteria. Figure 34 presents the distribution of static Factors of Safety (FoS) projected onto the pit walls, together with the associated damage indicator (*SLOSS*, for strength loss).



(a) Factor of safety. *(b) Damage (SLOSS).*

Figure 34. Projection of factor of safety and damage (SLOSS) on planned open pit.

All global FoS values exceed 1.5, with two exceptions. The first occurs in pit sectors where overburden is exposed, mainly in the southwest pit area. As the overburden was modelled as a perfectly elastic material for simplification, a FoS of 1.0 was obtained in these areas. The second exception is observed in tight pit corners, such as along section A–A', visible in Figure 34a. This area was analyzed in more detail to assess the extent of the potential instability. Figure 35 shows the total displacement along section A–A'. Displacement is mainly concentrated near the surface, with a maximum magnitude of less than 0.03 m within the fresh rock. These small movements are interpreted as minor deformations resulting from local stress relaxation associated with sharp changes in wall orientation and do not constitute a stability issue. This topic is further discussed in Section **??**.

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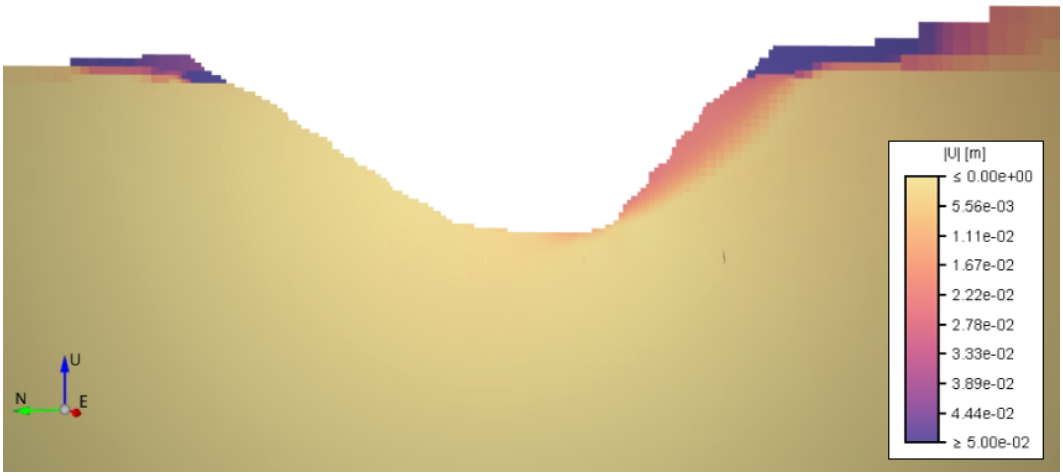


Figure 35. Total displacement across section A-A'.

Figure 34b highlights areas with lower SLOSS values, such as along section B–B', indicating localized damage along the pit wall. This section was similarly reviewed to evaluate the significance of the response. Figure 36 shows the total displacement along section B–B', overlaid with the UMU lithological solid. Displacements are again shallow and limited in magnitude (less than 0.02 m), indicating minor deformation. The key observation in this case is the correlation between zones of higher damage indicators and the presence of the UMU unit on the north wall. Although all FoS values remain above 1.5, these areas may be more susceptible to localized degradation and should be prioritized for monitoring. Additional recommendations related to these zones are provided in Section **??**.

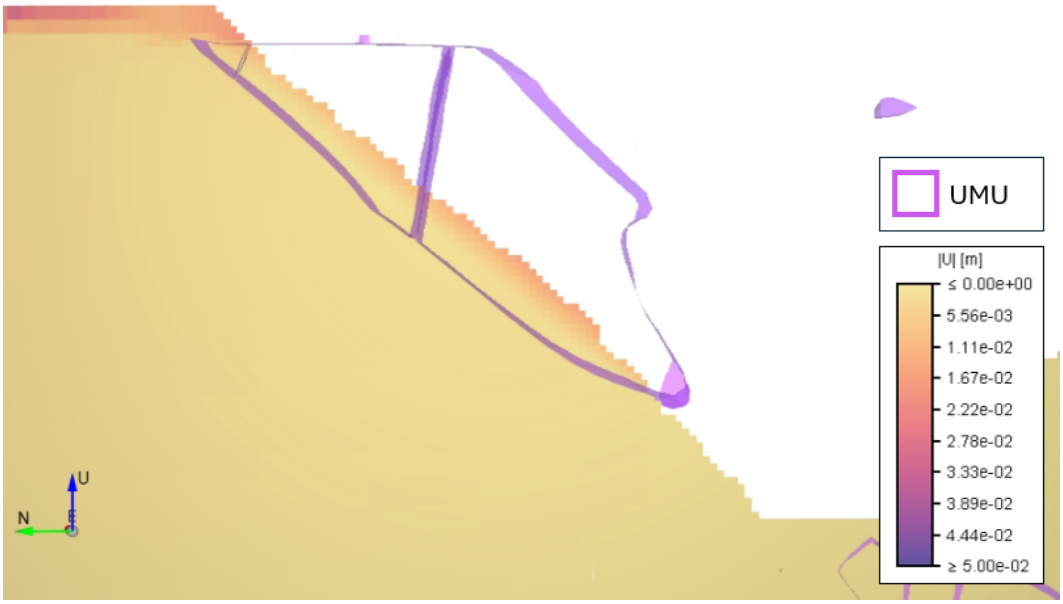


Figure 36. Total displacement across section B-B' with nearby ultramafic unit.

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7.6.3 Results

This section summarizes the guidelines provided at the multi-bench scale while integrating bench-scale recommendations.

01. **Employing Double Benching Practices**
    → Double benching can be implemented to increase the IRA and OSA.
02. **Recommendations on Inter-Ramp Angle**
    → Considering double benching, bench widths ranging from 8.5 to 12.0 m and bench face angles between 40 to 75°, the resulting IRAs are in the range of 32 to 55°, depending on the pit sector.
03. **Recommendations on Overall Scale Angle**
    → The integration of geotechnical berms into the design is strongly recommended.
    - A 16 m wide geotechnical berm is recommended around mid-slope for walls exceeding approximately 160 m in height.
    - Incorporating geotechnical berms also ensures that wider catch benches are present to intercept potential large rockfalls.
    - Berms should begin once 80 m of rock is exposed and should terminate approximately 80 m above the pit bottom.
    - Several strategies may be considered for integrating berms into the design:
        - Integrated to an existing bench, e.g., increasing the width from 8.5 m to 16.0 m.
        - Attaching the berm to a ramp, ensuring a minimum clear width of 7.5 m between the pit wall and the haul road. Haul road widths should follow Québec mining standards, which require a width equal to three times the truck width (Éditeur officiel du Québec, 2025).

Major geological structures, including faults and diabase dykes, can influence pit stability at both the IRA and overall scale. These structures have been modelled as sub-vertical planes intersecting the pit. Due to their near-vertical orientation, they are not expected to trigger large-scale failures individually. However, they can create localized zones of reduced rock quality, which may affect stability at a smaller scale. Where multiple modelled faults and dykes intersect, weaker rock conditions should be expected. The potential impact of several major structures occurring in close proximity should be carefully evaluated during the detailed engineering phase.

7.7 Open Pit Design Guidelines

Following the stability analyses, design guidelines were developed for both the overburden and the fresh rock at the relevant design scales. Table 49 presents the recommended guidelines for soils and fresh rock at the bench and inter-ramp scales, as well as the recommendations for geotechnical berms considered at the overall slope scale. The proposed design criteria represent the maximum values to be adopted and do not account for interactions between different sectors. Figure 37 illustrates the position of the geomechanical sectors relative to the pit design.

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Table 49. Summary of open pit design guidelines.

Sector	Area–Wall	Recommended FS Criteria						
		Lithology	BH (m)	BW (m)	BFA (°)	IRA (°)	DD (°)	Geotechnical Berm
1	All	OVB	< 25	0.0	36	36	N/A	10 m between overburden and rock
A	West–North	AMP	20	10.5	68	47	130–200	At mid-wall if H > 160 m. Width = 16 m (total)
B	West–West	AMP, UMU, META	20	8.5	75	55	005–130	
C	West–South	AMP, META	20	11.0	75	51	340–005	
D	Central–North	AMP	20	8.5	59	44	150–165	
E	Central–North	META	20	8.5	50	38	140–155	
F	Central–South	AMP, META	20	11.0	75	51	330–350	
G	East–North	AMP, META	20	8.5	47	36	165–180	
H	East–North	UMU	20	8.5	40	32	165–180	
I	East–East	AMP, PEG, UMU, META	20	8.5	75	55	180–310	
J	East–South	AMP, META	20	12.0	75	49	310–335	

Codes BH bench height, BW bench width, BFA bench face angle, IRA inter-ramp angle, DD dip direction range, H height.

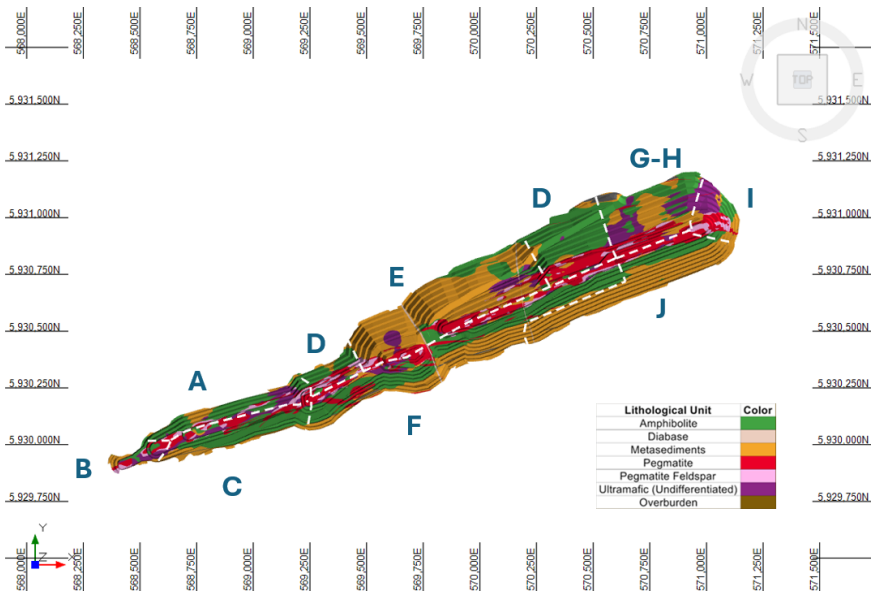


Figure 37. Summary of open pit sector guidelines (based on March 2025 pit design).

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8 Underground Mine Design Guidelines

Design guidelines were developed for multiple aspects of the underground (UG) mine:

01. **Stope Dimensions and Dilution**
02. **Mine Sequence**
03. **Backfill**
04. **Crown Pillar**
05. **Ground Support**
06. **Portal (Box Cut)**
07. **Infrastructures**
08. **Open Pit–Underground Interaction**

These are detailed in the following sections.

8.1 Terminology and Input Parameters

The underground mine stope terminology used in the following sections is illustrated in Figure 38.

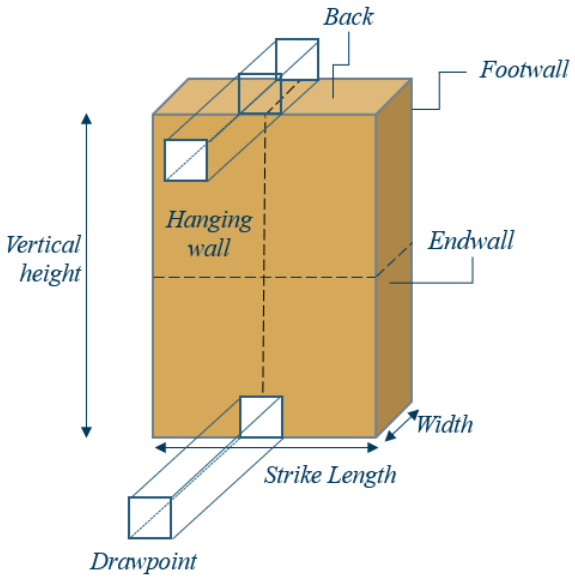


Figure 38. Stope terminology used in the report (not to scale).

As some of the input parameters used for the design of the open pit and the underground mine differ, the parameters adopted for the underground analyses are summarized below:

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01. **Rock Density**
    - → Refer to Section 5.1.1, Table 14.
    - → Densities range from 2.64 to 2.98 t/m^3.
02. **Uniaxial Compressive Strength**
    - → Refer to Section 5.1.1, Table 19.
    - → As a reminder, first and second quartiles of the UCS distribution for amphibolite, metasediments, pegmatite, and ultramafic units are 167–186, 101–130, 62–96 and 48–55 MPa, respectively.
03. **Rock Mass Classification**
    - → Refer to Section 5.1.2, Table 30.
    - → As a reminder, the first and second quartiles of the Q' distribution are presented below:

Lithology	Q'	
	Q1	Q2
Amphibolite	11	18
Metasediments	10	16
Pegmatite	7	18
Ultramafic	5	11

04. **Joint Sets**
    - → Refer to Section 5.1.2, Table 37.
    - → As a reminder, between two and three joint sets were identified per geomechanical domain.
05. **In-Situ Stress State**
    - → Refer to Section 5.2, Table 39.
    - → As a reminder, the major principal stress component is assumed to be oriented at 70°, which is parallel to the orebody.
    - → The depth of the underground mine below the surface ranges from 100 m to 450 m; a general value of 400 m is considered for the analyses.
06. **Stope Design**
    - → Stope configuration: Approximately 91% of the stopes are transverse, while the remaining 9% are longitudinal retreat stopes (based on the intermediary design from the July 2024 block model).
    - → The global dip direction of both the HW and FW is approximately 340°.
    - → The HW and FW dip is variable; for transverse stopes, it generally ranges from 60–80°, whereas for longitudinal stopes, dips typically range from 60–75°. In some cases, the dip was found to be lower than 60°.

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8.2 Numerical Model

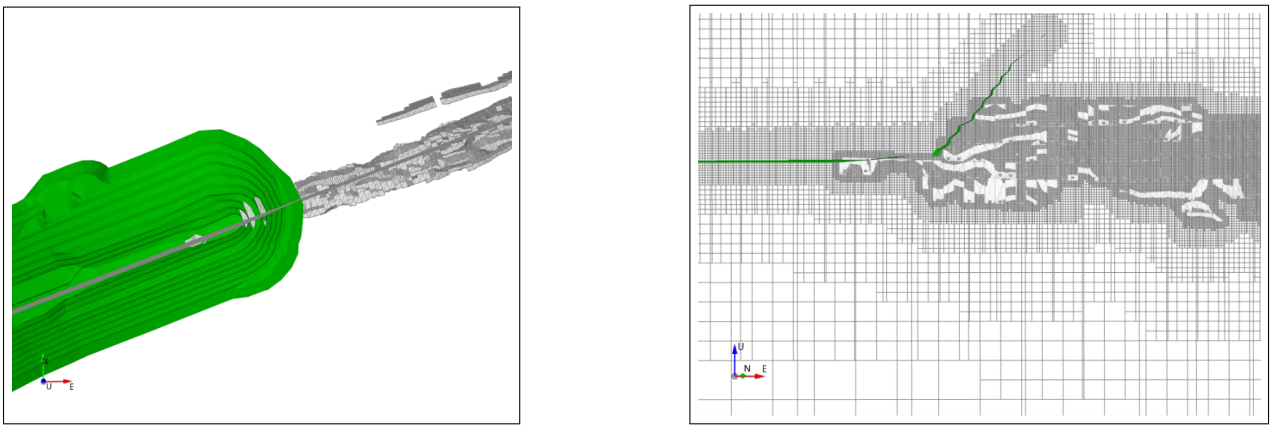
A 3D model was built, and extents along the E-N-Z axes are approximately 6.2 × 3.1 × 0.9 km. This global model incorporates the lithological model, as well as the planned open pit and underground excavations. Additionally, host rock material is added as a filler around the model to ensure that boundary conditions are far away and do not affect the results.

Geometry

The modelled open pit geometry is based on the March 2025 pit design, and the underground excavations are based on the March 2025 underground mine design. Geological solids derived from the lithological model were used to assign appropriate material properties to each zone within the numerical model. Faults identified on site were not incorporated in the current model, as their interpretation in the underground area was still ongoing when the numerical model was created. Moreover, structural characterization of these faults is still in progress. It is recommended that the final interpretation of the fault network be integrated into a subsequent 3D model to assess and validate its potential impact on overall slope and underground stability.

Mesh and Boundary Conditions

A structured cubic (octree) mesh was used to build the model. The zones adjacent to the stopes and pit walls have edges of 2 and 4 m, respectively, and the zones gradually increase in size to reach 64 m edges at a distance of more than 186 m. Figure 32 shows a section view where the mesh is visible for both the pit area and some stopes.



(a) Section location intersecting the stopes and the open pit.

(b) Mesh refinements on a section plane.

Figure 39. View of the global numerical model structured (octree) mesh.

The in-situ stresses used in the numerical model are those presented in Table 39 (Section 5.2). Boundary conditions were applied by fixing all model boundaries except for the positive Z axis, which was assigned a free (unrestrained) condition.

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Material Properties

Material properties used in the numerical model were derived from the geomechanical characterization, with the exception of the host rock. The global numerical model parameters are summarized in Table 50.

Table 50. Material properties of the global numerical model.

Unit	Constitutive Model	Density (kg/m^3)	E_i (GPa)	ν (–)	σ_i (MPa)	m_i (–)	GSI (–)	φ (°)	c (kPa)
AMP	Hoek-Brown	2 984	89.2	0.22	198.2	24.2	64	–	–
META	Hoek-Brown	2 876	38.3	0.22	153.0	15.1	64	–	–
OVB	Mohr-Coulomb	2 321	5.0	0.20	–	–	–	45	0
PEG	Hoek-Brown	2 636	51.2	0.20	120.4	24.7	78	–	–
UMU	Hoek-Brown	2 874	15.9	0.22	57.7	12.3	64	–	–
Backfill	Mohr-Coulomb	1 800	1.5	0.30		–	–	30	156
Host rock	Elastic	2 700	70.0	0.20	–	–	–	–	–

Notes Hoek-Brown is modelled using the IMASS constitutive model.
Codes E_i Young's modulus of the rock mass, ν Poisson's ratio, σ_i unconfined compressive strength of intact rock,m_i Hoek-Brown intact rock constant, GSI Geological Strength Index, φ friction angle, c cohesion.

Sequence

The mining sequence implemented in the model begins in 2031, incorporating the open pit mine plan up to that year. From 2031 to 2049, the model includes annual mining steps representing both open pit production and underground stope extraction.

8.3 Stope Dimensions and Dilution

8.3.1 Design Methods

Stope dimensions and dilution were assessed using empirical open stope stability charts, including Potvin (1988), as well as the Equivalent Linear Overbreak/Slough (ELOS) chart (Clark, 1998) for dilution estimates. For the ELOS method, a probabilistic approach was applied. The methodologies used for both assessments are summarized below.

No explicit stope-scale continuum or discontinuum numerical modelling was conducted.

8.3.2 Open Stope Stability Charts

The empirical charts consider rock mass classification, induced stress conditions, intact rock strength, joint orientations, and stope geometry (i.e., hydraulic radius) and orientation to predict stope surface stability. The input parameters required for the analyses, as well as the outputs, are presented in this section. All open stope stability charts prepared for the Project are presented in Appendix F.

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Theory

The open stope stability charts were used to assess the stability of individual surfaces – i.e., back, hanging wall, footwall and endwalls. The charts rely on the two following inputs:

01. **Hydraulic Radius (i.e., dimension)**
02. **Stability Number, N'**

An example of the Potvin open stope stability chart is shown in Figure 40.

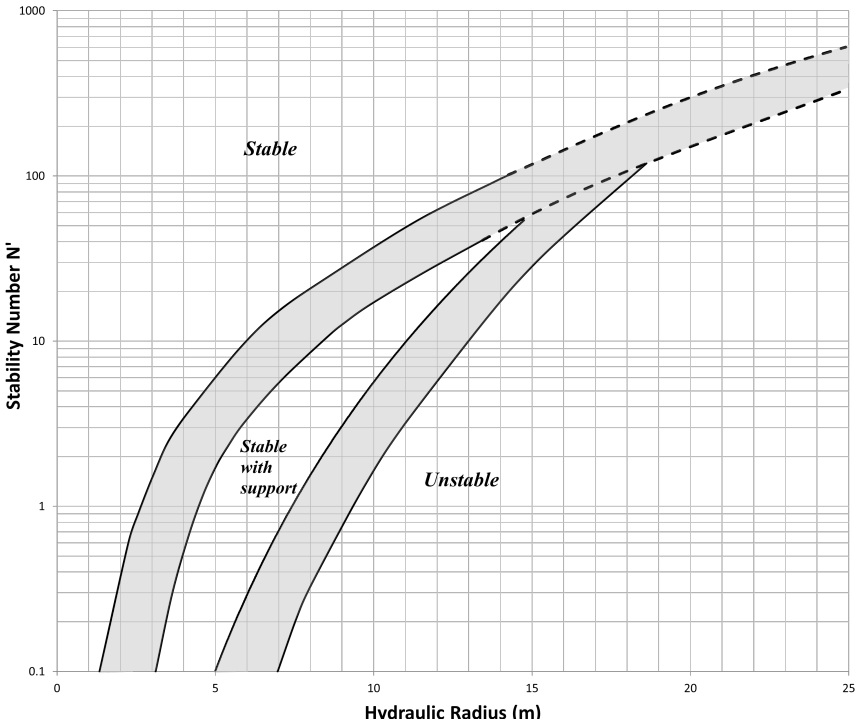


Figure 40. Potvin open stope stability chart (reproduced from Potvin, 1988).

The hydraulic radius (HR) of a stope surface is defined as follows:

$$HR = \frac{Area}{Perimeter} = \frac{W \times H}{2(W + H)} \qquad \textit{(eq. 31)}$$

Where HR = Hydraulic radius (m)
W = Surface width (m)
H = Surface height (m)

While the stability number (N') is defined as follows:

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$$N' = Q' \times A \times B \times C \quad \textit{(eq. 32)}$$

Where Q' = rock mass classification
A = rock stress factor
B = joint orientation factor
C = gravity adjustment factor

With factors A, B and C defined accordingly:

$$A = \begin{cases} 0.1 & \text{if } SR < 2 \\ 0.1125 \times SR - 0.125 & \text{if } 2 < Ratio < 10 \\ 1 & \text{if } SR > 10 \end{cases} \quad \textit{(eq. 33)}$$

Where SR = $\frac{UCS}{\sigma_{max}}$
UCS = uniaxial compressive strength (MPa)
σ_{max} = maximum induced tangential stress acting on the stope surface, based on the Stewart and Forsyth (1995) approach (MPa)

$$B = \begin{cases} 0.3 - 0.01\alpha & \text{if } \alpha < 10° \\ 0.2 & \text{if } 10° < \alpha < 30° \\ -0.4 + 0.02\alpha & \text{if } 30° < \alpha < 60° \\ 0.4 + 0.0066\alpha & \text{if } \alpha > 60° \end{cases} \quad \textit{(eq. 34)}$$

Where α = angle between stope surface and joint (°)

$$C_{Gravity\ Fall\ \&\ Slabbing} = 8 - 6\cos(dip_{surface}) \quad \textit{(eq. 35)}$$

$$C_{Sliding} = \begin{cases} 8 & \text{if } dip_{joint} < 30° \\ 11 - 0.1 \times dip_{joint} & \text{if } dip_{joint} > 30° \end{cases} \quad \textit{(eq. 36)}$$

Generally, stope surface dimensions can be maximized as long as they remain within the *stable zone* of the Potvin stability chart. The back surfaces can lie within the *stable with support zone* as these can be cablebolted from the overcut.

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Inputs

When applicable, the analyses were conducted using Low–High sets of values, generally consisting of the 25th and 50th percentiles – or the first quartile (Q1) and the median (Q2) – of the cumulative distribution functions (CDF). Stopes were differentiated between transverse and longitudinal mining methods, and the planned exposed lithologies were considered.

Assuming that a retreating sequence from the footwall to the hanging wall will be adopted (refer to Section 8.4), the lithologies exposed in the primary and secondary stopes of transversal mining areas can differ. Due to the orebody's significant thickness, blasted stopes may expose different lithologies. As summarized in Table 51, transverse stope endwalls and backs are always assumed to be in pegmatite. The footwall of the first stope of every drawpoint (the footwall side of the orebody) exposes rock. Once the first stopes are backfilled, subsequent stopes within a given drawpoint will expose backfill on the footwall (refer to Section 8.5). Hanging walls can expose pegmatite if the orebody is sufficiently thick to include multiple stopes. If this is not the case, typical waste rock lithologies are exposed. Again, this information is summarized in Table 51, and shown in Figure 41. Surfaces composed of pastefill are not evaluated directly using the Potvin charts.

A "case" is defined as a specific combination of lithologies exposed on the stope surfaces. For transverse stopes, ten cases with a frequency greater than 2% were identified, representing the relative occurrence of lithology combinations on the exposed surfaces, and their stability was assessed. For longitudinal stopes, four cases were identified; however, only two required direct evaluation using the Potvin chart, as the remaining two corresponded to previously assessed configurations.

Table 51. Details of exposed rock and material type for transverse and longitudinal stopes.

Code	Stope Type & Sequence	Exposed Material or Rock Type			
		Back	EW	FW	HW
P_1	Transverse – Initial primary	PEG	PEG	PEG/AMP/ META/UMU	PEG/AMP/ META/UMU
P_i	Transverse – Subsequent prim.	PEG	PEG	CPB	PEG/AMP/ META/UMU
P_n	Transverse – Last primary	PEG	PEG	CPB	PEG/AMP/ META/UMU
S_1	Transverse – Initial secondary	PEG	CPB	PEG/AMP/ META/UMU	PEG/AMP/ META/UMU
S_i	Transverse – Subsequent sec.	PEG	CPB	CPB	PEG/AMP/ META/UMU
S_n	Transverse – Last secondary	PEG	CPB	CPB	PEG/AMP/ META/UMU
P_{1-i}	Longitudinal – All	PEG	PEG/CPB	AMP/META/UMU	AMP/ META/UMU

Codes AMP amphibolite, CPB cemented paste backfill, META metasediments, PEG pegmatite, UMU ultramafic, HW hanging wall, FW footwall, EW endwalls.

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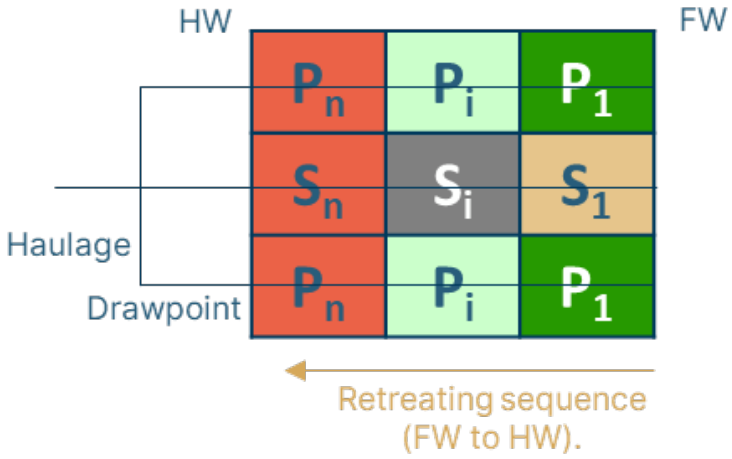


Figure 41. Schematic plan view of the transversal stope sequence.

A summary of the input assumptions is detailed below, assessed individually for each stope surface (and corresponding lithology):

01. **Q'**
    - → Low First quartile (25th percentile) of the Q' cumulative distribution function (CDF)
    - → High Median (50th percentile) of the Q' cumulative distribution function (CDF)
02. **A Factor**
    - → Low Ratio of the first quartile of the UCS CDF with respect to the induced stress (σ_{max})
    - → High Ratio of the median of the UCS CDF with respect to the induced stress (σ_{max})
03. **B Factor**
    - → Comparing the stope surface orientations with regard to all the joint sets identified for the exposed lithology (refer to Section 5.1.2). The lowest B factor is used to identify the critical joint.
    - → The surfaces dip and dip directions used for the analyses are presented in Table 52:

Table 52. Stope surface orientations.

Stope Surface	Transverse Stope		Longitudinal Stope	
	Dip (°)	Dip Direction (°)	Dip (°)	Dip Direction (°)
Back	00	000	00	000
HW	60–80	340	60–75	340
FW	60–80	340	60–75	340
EW	90	070	90	070

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04. **C Factor**
    - → Orientation (dip/dip direction) of the critical joint identified previously.
05. **Hydraulic Radius**
    - → Based on stope surface dimensions (i.e., length and height). This parameter is an input in the method, and also corresponds to a parameter to be optimized through an iterative process, while ensuring that the stope surfaces remain stable.
    - → Stope height of 30–36 m was used, as this value was initially provided to the mine engineering team, for preliminary designs, based on the PEA available information. Alius deemed acceptable to maintain this assumption moving forward.
    - → As a result of applying gradients to mine development, approximately 85% of stopes have heights varying from 30 to 36 m.

For simplification purposes, and given their comparable strength characteristics and structural orientations, the amphibolite and metasediments were grouped for the open stope stability analyses. The metasediment properties were selected to represent the group, as they exhibit slightly lower Q' values and UCS, providing a conservative basis for design. Conversely, the joint set orientations from the amphibolite were adopted, as their dip angles are marginally lower, representing the more critical structural configuration.

Results

The outputs of the methods consist of stope stability predictions. Again, stope surface dimensions were maximized while ensuring that they remain within the *stable zone* of the Potvin stability chart. The back surfaces can lie within the *stable with support zone*, as they can be supported easily (i.e., cablebolted) from the overcut.

An example of the Potvin open stope stability chart process is presented in Figure 42. The upper portion of the figure presents the assessment inputs, including stope dimensions and in-situ stresses. The bottom table presents the rock mass classifications, factors A, B, and C, the hydraulic radii, and the stability numbers resulting from the assessment. Footwall stability is not assessed in this example, as it is considered engineered backfill; only the hanging wall, endwalls, and back are included. It can be seen that the endwalls are the limiting surface as they lie on the edge of the stable zone. The stope back is located at the edge of the *stable with support zone*, indicating the need for cablebolting.

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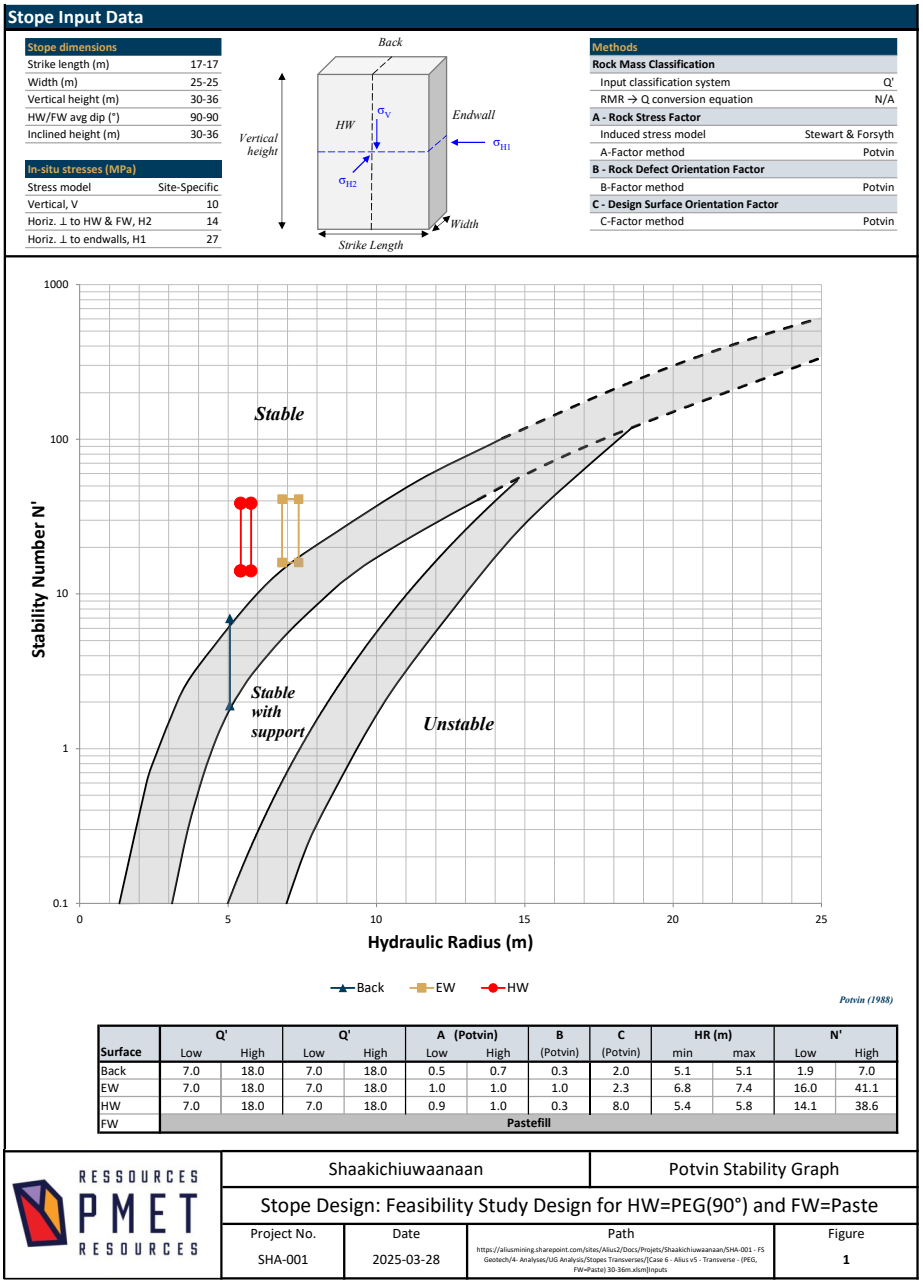


Figure 42. Example of a Potvin open stope stability chart assessment. In this example, the HW and FW are considered to be pegmatite and backfill, respectively.

The optimized dimensions for the different cases are presented in Table 53 for transverse stopes and in Table 54 for longitudinal stopes. Each table row represents a case. Backfilled endwalls are not evaluated as they are not critical. Therefore, no secondary stope types are listed in the table below, as endwalls are always backfilled.

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Table 53. Optimized transverse stope sizes and limitations.

Type	Freq.[1]	Lithology			Max Stope Size (mL × mW × mH)	Limitation
		HW	FW	Back & EW		
P_i	26%	PEG (90°)	CPB	PEG	17 × 25 × 30–36	Back & EW
P_n	19%	AMP/META	CPB	PEG	17 × 25 × 30–36	HW & Back & EW
P_1	18%	PEG (90°)	AMP/META	PEG	17 × 25 × 30–36	Back & EW
P_1	10%	PEG (90°)	PEG	PEG	10 × 22 × 30–36	FW
P_n	8%	PEG	CPB	PEG	15 × 22 × 30–36	HW
P_1	6%	PEG	AMP/META	PEG	15 × 22 × 30–36	HW
P_1	5%	AMP/META	PEG	PEG	10 × 22 × 30–36	FW
P_1	3%	PEG	PEG	PEG	10 × 22 × 30–36	FW
P_n	3%	UMU	CPB	PEG	17 × 25 × 30–36	Back & EW
P_1	2%	PEG (90°)	UMU	PEG	17 × 25 × 30–36	Back & EW

Codes **Freq.** frequency, **HW** hanging wall, **FW** footwall, **EW** endwalls, **CPB** cemented paste backfill.
Notes **[1]** Frequency was established based on the analyzed zone of the PEA stope design (Figure 4a).

Table 54. Optimized longitudinal stope sizes and limitations.

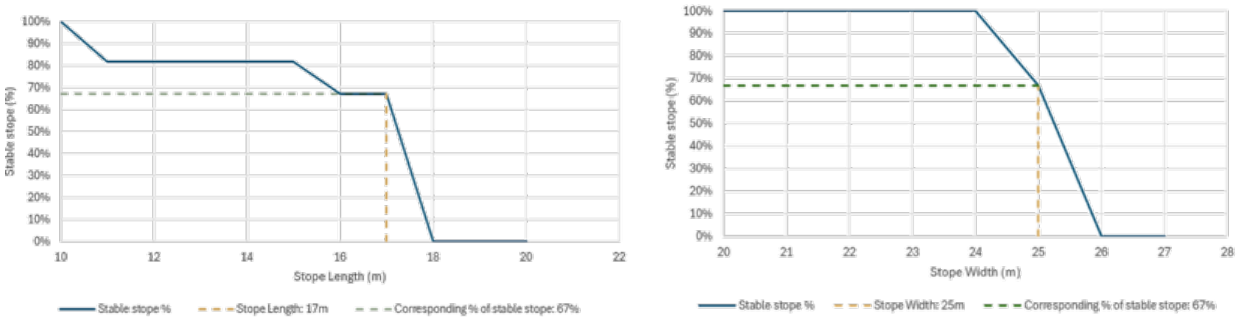
Type	Freq.[1]	Lithology			Maximum Stope Size (mL × mW × mH)	Limitation
		HW	FW	Back & EW		
P_{1-i}	45%	UMU	UMU	PEG or CPB	25 × 7 × 30–36	FW & Back
P_{1-i}	39%	AMP/META	AMP/META	PEG or CPB	17 × 7 × 30–36	HW & Back
P_{1-i}	9%	AMP/META	UMU	PEG or CPB	17 × 7 × 30–36	HW & Back
P_{1-i}	7%	UMU	AMP/META	PEG or CPB	22 × 7 × 30–36	FW & Back

Codes **Freq.** frequency, **HW** hanging wall, **FW** footwall, **EW** endwalls, **CPB** cemented paste backfill.
Notes **[1]** Frequency was established based on the intermediary DSO from the July 2024 block model (Figure 4b).

To facilitate the stope-size design process, a uniform stope size was evaluated for both transverse and longitudinal stopes. For the longitudinal stopes, a stope size of 17 mL × 7 mW × 30–36 mH was recommended.

As shown in Figure 43, for a stope length of 17 m, approximately 67% of the stopes are stable, and for a stope width of 25 m, approximately 67% of the stopes are stable. This includes the primary cases (those with a frequency exceeding 11%). After standardizing the stope size to 17 mL × 25 mW × 30–36 mH, several cases fall within the *stable with support* zone; however, no case falls within the *unstable* zone. The Potvin stability charts corresponding to this standardized stope size are presented in Appendix F.

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(a) Variation in stope length. *(b) Variation in stope width.*

Figure 43. Percentage of stable stopes with respect to stope dimensions.

8.3.3 Dilution (ELOS) Chart

The methodology underlying the ELOS chart (Clark, 1998) follows principles similar to those of the Potvin (1988) stability chart and is briefly introduced below.

Theory

In underground mining, quantifying overbreak and assessing stope stability are essential for operational planning, safety, and cost control. One of the most widely used measures to evaluate overbreak is the Equivalent Linear Overbreak/Slough (ELOS) parameter (Clark, 1998). ELOS represents the average linear extent of overbreak per unit area of a stope surface. For any given surface (e.g., footwall, hanging wall, end wall or back), it is calculated as:

$$ELOS\ (m) = \frac{Volume\ of\ overbreak\ (m^3)}{Surface\ area\ of\ stope\ (m^2)} \qquad \textit{(eq. 37)}$$

A widely used approach for correlating stope stability with overbreak is the stability design chart overlaid with ELOS curves developed by Clark (1998). These charts are based on the original stability chart by Mathews (1980) and have been calibrated or modified by several authors, including Potvin (1988). Therefore, following the same approach as the Potvin chart calculations presented in Section 8.3.2, the stability number (eq. 32) and the hydraulic radius (eq. 31) are plotted on the chart:

The position of each surface relative to the ELOS curves serves as a predictive tool for estimating overbreak and support design. An empty ELOS stability chart is shown in Figure 44.

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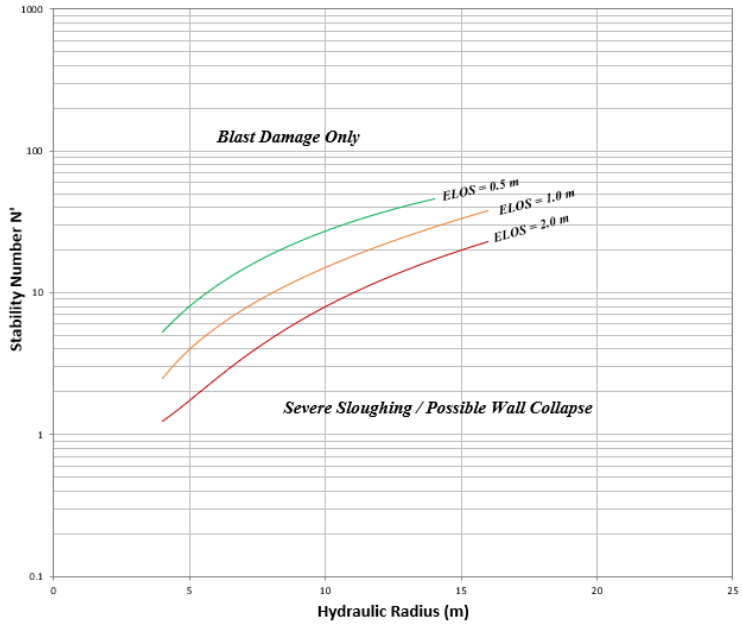


Figure 44. ELOS open stope stability chart (reproduced from Potvin, 1988 and Clark, 1998).

Recognizing that using only 0.5 m (green curve), 1.0 m (orange curve), and 2.0 m (red curve) values can be overly conservative, the adopted approach considers intermediate values between the curves.

Inputs

The inputs are the same as those presented in Section 8.3.2.

Results

The outputs of the method are ELOS predictions. ELOS values are assessed after determining the maximized stope surface dimensions using the Potvin charts. For certain stope surfaces, ELOS values are assumed without reference to the ELOS chart, as some concepts are not captured by this method. For instance:

- **Backfill surfaces** – All backfilled surfaces are assumed to have an ELOS of 0.5 m.
- **Endwalls and Back** – Since mineralized material is expected on these surfaces, and support is recommended for the back, the recommended ELOS for mine planning purposes are 0 m for EW and the back.

An example of the resulting ELOS chart is presented in Figure 45. The upper portion of the figure shows the assessment inputs, including the stope dimensions and in-situ stresses. The bottom table presents the rock mass classifications; the A, B, and C factors; the hydraulic radii; and the stability numbers obtained from the assessment. Footwall stability is not evaluated in this example, as it is represented by engineered backfill; therefore, only the hanging wall, endwalls, and back are

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assessed.

In this example, the endwalls plot on the green curve, corresponding to an ELOS of 0.5 m. The hanging wall plots close to the orange curve, yielding an interpreted ELOS of 1.0 m. The footwall, consisting of backfill, is assigned an ELOS of 0.5 m. The back plots on the red curve, which corresponds to an interpreted ELOS of 2.0 m; however, as previously noted, the ELOS for the back is ultimately assumed to be 0.0 m (i.e., ore).

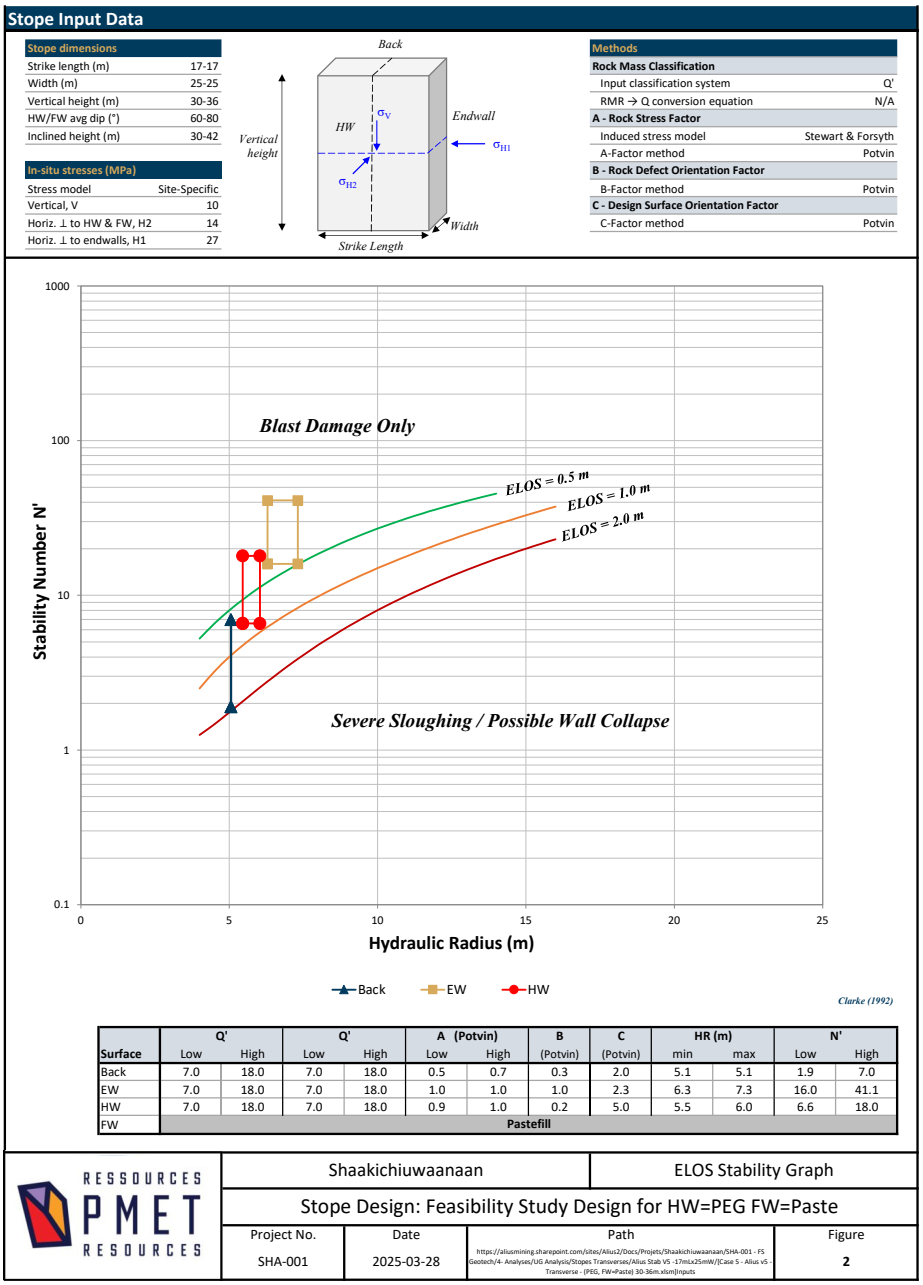


Figure 45. Example of an ELOS assessment. In this example, the HW and FW are considered to be pegmatite and backfill, respectively.

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The expected dilution estimates associated with the stope design are presented in Table 55 for transverse stopes and in Table 56 for longitudinal stopes.

Table 55. Dilution (ELOS) evaluation for transverse stopes.

Type	Freq.[1]	Lithology			Stope Size (mL x mW x mH)	ELOS			
		HW	FW	Back & EW		HW	FW	EW	Back[2]
P_i	26%	PEG (90°)	CPB	PEG	17 × 25 × 30–36	0.5	–	0.5	2.0
P_n	19%	AMP/META	CPB	PEG		0.5	–	0.5	2.0
P_1	18%	PEG (90°)	AMP/META	PEG		0.5	0.5	0.5	2.0
P_1	10%	PEG (90°)	PEG	PEG		1.0	2.0	0.5	2.0
P_n	8%	PEG	CPB	PEG		1.0	–	0.5	2.0
P_1	6%	PEG	AMP/META	PEG		1.0	0.5	0.5	2.0
P_1	5%	AMP/META	PEG	PEG		0.5	2.0	0.5	2.0
P_1	3%	PEG	PEG	PEG		1.0	2.0	0.5	2.0
P_n	3%	UMU	CPB	PEG		0.5	–	0.5	1.0
P_1	2%	PEG (90°)	UMU	PEG		0.5	0.5	0.5	2.0

Codes N/A Not available or indicated, **Freq.** Frequency, **HW** Hanging wall, **FW** Footwall, **EW** Endwalls, **CPB** Cemented CPB backfill.
Notes [1] The frequency was established based on the analyzed zone of the PEA stope design (Figure 4a).
[2] The back ELOS values presented are direct readings from the ELOS charts. However, for planning purposes, and considering the recommended support and expected mineralization on this surface, the final ELOS is taken as 0.0 m.

Table 56. Dilution (ELOS) evaluation for longitudinal stopes.

Type	Freq.[1]	Lithology			Stope Size (mL x mW x mH)	ELOS (m)			
		HW	FW	Back & EW		HW	FW	EW	Back[2]
P_{1-i}	45%	UMU	UMU	PEG or CPB	17 × 7 × 30–36	0.5	0.5	0.5	2.0
P_{1-i}	39%	AMP/META	AMP/META	PEG or CPB		1.0	0.5	0.5	2.0
P_{1-i}	9%	AMP/META	UMU	PEG or CPB		1.0	0.5	0.5	2.0
P_{1-i}	7%	UMU	AMP/META	PEG or CPB		0.5	0.5	0.5	2.0

Codes N/A Not available or indicated, **Freq.** Frequency, **HW** Hanging wall, **FW** Footwall, **EW** Endwalls, **CPB** Cemented CPB backfill.
Notes [1] The frequency was established based on the intermediary DSO from July 2024 block model, (Figure 4b).
[2]The back ELOS values presented are direct readings from the ELOS charts. However, for planning purposes, and considering the recommended support and expected mineralization on this surface, the final ELOS is taken as 0.0 m.

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8.3.4 Guidelines

The stope design guidelines are presented in Table 57, alongside those from the previous geomechanical study (PEA). This table also introduces the recommended support measures for the back and hanging wall stope surfaces. Note that for longitudinal stopes, the backs are considered stable, as they are supported during development.

Table 57. Stope design guidelines.

Stope Parameter	Transverse Stopes		Longitudinal Stopes	
	PEA	FS	PEA	FS
Vertical height (m)	30	30–36	30	30–36
Strike length (m)	15	17	15	17
Width (m)	20	25	6	7
Back stability	Cablebolted	Cablebolted	Cablebolted	Cablebolted
HW stability	N/A	Cablebolted if dip <60°	N/A	Cablebolted if dip <60°

Codes N/A Not available or indicated.
Notes PEA BBA (2023a). FS Current Feasibility Study.

Table 58 presents the recommended dilution (ELOS), after weighting by the relative frequency of each stope type and after simplification for planning purposes.

Table 58. Summary of simplified ELOS assumptions for planning purposes.

ELOS	Surface	ELOS (m)	
		Transverse Stopes	Longitudinal Stopes
From chart	Endwalls	0.5	0.5
	Footwall	1.25	0.5
	Hanging wall	0.75	0.75
	Back	2.0	2.0
	All surfaces (backfill)	0.5	0.5
Simplified	Endwalls[1]	0.0	0.0
	Endwalls (backfill)	0.5	0.5
	Footwall	1.25	0.5
	Footwall (backfill)	0.5	0.5
	Hanging wall	0.75	0.75
	Back[1]	0.0	0.0

Notes [1] Assuming mineralized material and/or support.

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8.4 Mine Sequence

The selected long-hole open stoping mining method, adopted in both transverse and longitudinal configurations, is a productive and widely used technique in hard rock mining for subvertical and thick ore bodies. This method allows for various mining sequences, particularly in transverse configurations.

Selecting an appropriate mining sequence is critical to ensure safe and efficient underground mine development. The recommended mining sequence for the Project was developed based on the following key objectives:

01. **Simultaneous Mining Fronts** – The sequence must allow mining multiple adjacent zones concurrently (typically two zones), progressing from west to east.
02. **Limit Early Development** – The sequence prioritizes the development of the lower portions of each zone before advancing to the upper portions, minimizing the amount of early development required at higher levels.
03. **Minimize Open Drawpoints** – Mining should be prioritized at already-developed drawpoints (DPs) before initiating new ones, to reduce potential rehabilitation requirements.

8.4.1 Development and Stope Sequence

A typical transverse stope panel, adopting a primary-secondary mine sequence that maintains a pyramidal shape, is presented in Figure 46. Based on preliminary optimization conducted by GMS, each mining zone was defined with a width of 9 stopes (approximately 150 m in strike length), to ensure efficient production. The vertical extent of the zones range between 120 and 280 m.

Contrary to what is illustrated below, no pillar is left between the lower and upper portions of the stopes; the separation is only intended to distinguish two mining fronts. As shown in Figure 46, the proposed sequence focuses primarily on the lower portion of the ore zone to minimize development requirements while maintaining a pyramidal mining sequence. This proposed sequence serves as a general guideline and should be further optimized based on local stope geometries and geotechnical conditions.

For narrower stope panels, adopt a longitudinal retreat sequence. Avoid convergent fronts and diminishing pillars.

Regarding mine development, the following practices are recommended:

- → When possible, minimize early development and adopt a "just-in-time" development strategy to reduce exposure of unused excavations.
- → Prioritize mining in already-developed drawpoints (DPs) before initiating new ones, thereby reducing the risk of deterioration and limiting rehabilitation requirements.
- → Limit the time required to support development headings, and ensure ground support installation keeps pace with advances to maintain excavation stability.

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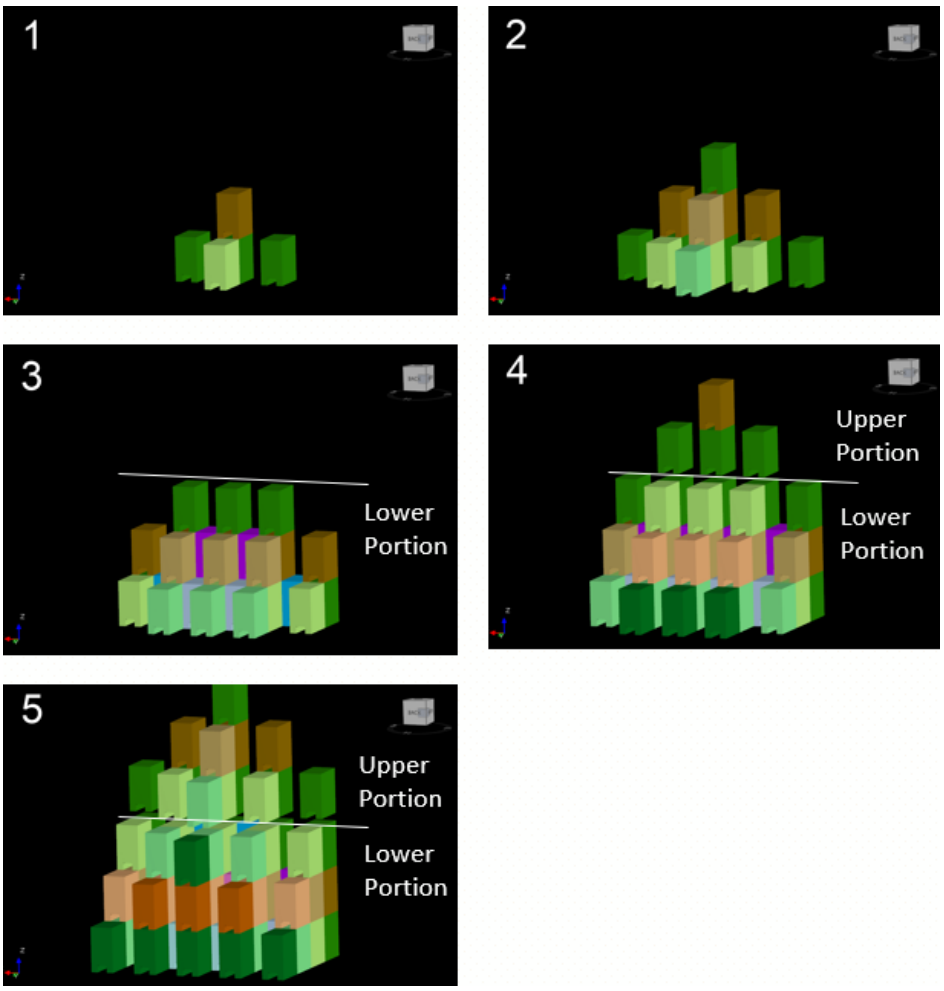


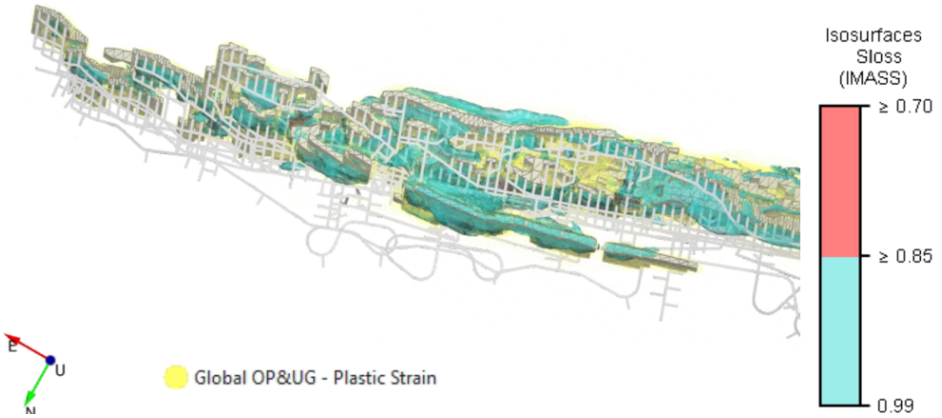
Figure 46. Isometric view of the proposed mining sequence.

8.4.2 Numerical Modelling Results

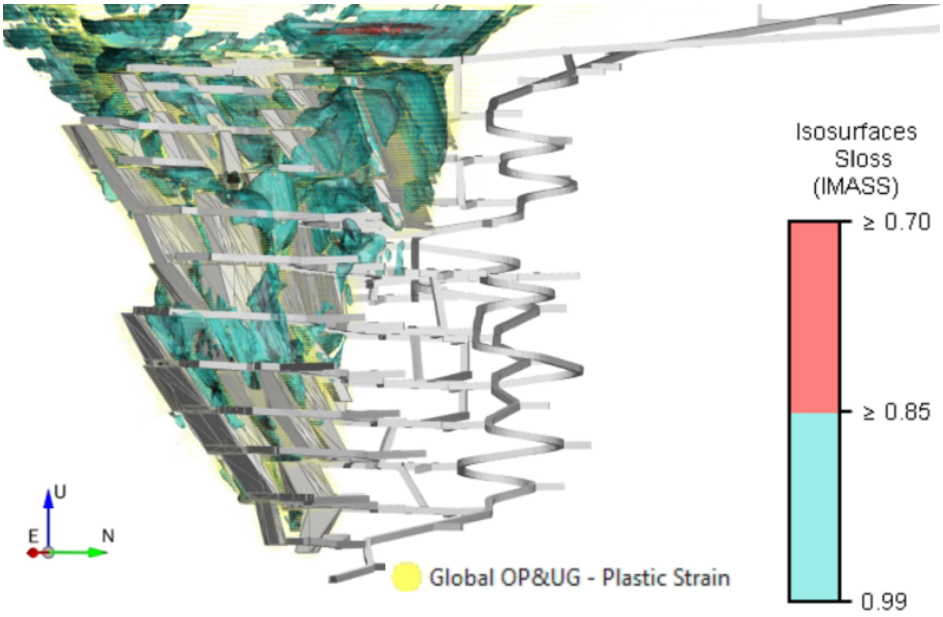
To assess the stability of the underground excavations, the mining sequence was incorporated into the numerical model to simulate the progressive extraction of stopes and the associated stress redistribution throughout the planned mine schedule.

Stability was evaluated using several key indicators derived from the model results. Figure 47 presents plan and section views showing the degree of rock mass damage (SLOSS, for strength loss) and the distribution of plastic strain after Year 2049. These outputs provide insight into localized instability around excavations and the overall stability of underground workings throughout the mining sequence. Based on the modelled response, only minor damage is anticipated, primarily confined to the immediate vicinity of the stoping areas.

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(a) Plan view.



(b) Section view.

Figure 47. Plastic strain and rock mass damage (Sloss) using two isosurfaces.

Figure 48 presents the major principal stress (S1) and the isosurfaces of rock mass damage in plan view. Localized zones of elevated plastic strain are observed within the orebody, with yielding occurring around the stopes, within the drawpoints, and in several sections of the haulage drifts.

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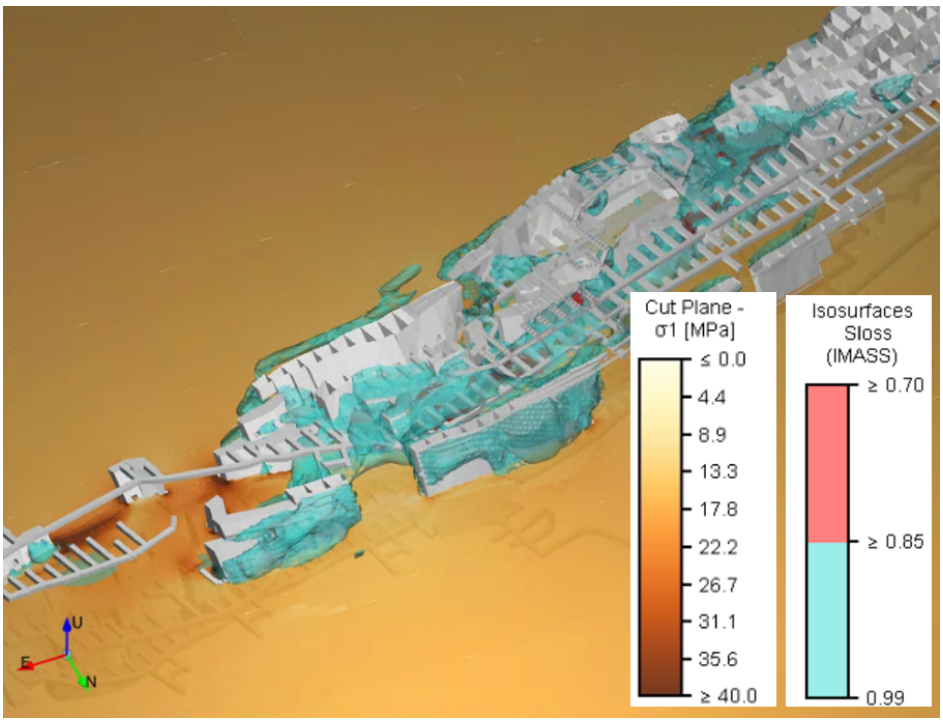


Figure 48. Major principal stress presented on a plan view and isosurfaces of rock mass damage.

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8.5 Backfill

Stopes will be backfilled using cemented paste backfill (CPB) to avoid leaving rib pillars. The methodology for assessing the required cemented paste backfill strength combines empirical approaches with numerical model validation. The recommended backfill strengths are presented in this section.

8.5.1 Design Methods

Empirical methods were used to define the minimum backfill strength requirements based on stope geometry, assumed backfill properties, and the planned mining sequence. In addition, for stopes mined beneath previously placed backfill (i.e., underhand), a numerical model was used to confirm that the cemented paste backfill is stable, thereby ensuring overall stope stability.

The empirical assessment is based on a comparison of three established methods: the approach of Mitchell et al. (1982), the modified exposed-backfill method proposed by Li and Aubertin (2012), and the generalized solution developed by Belem et al. (2022). Figure 49 presents the geometric parameters used in these methods. The Mitchell et al. (1982) approach is described below, whereas the other methods are detailed in Appendix G.

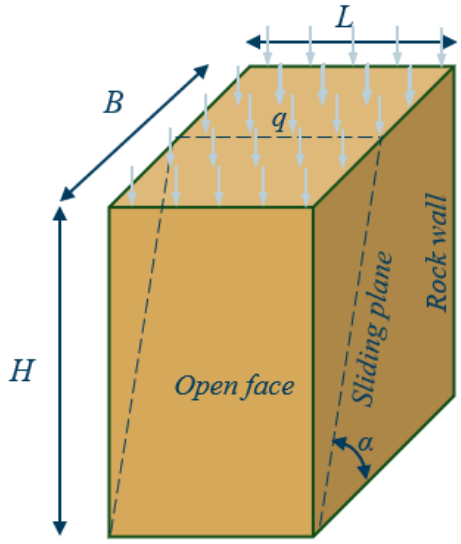


Figure 49. Stope configuration adopted for the backfill strength requirement assessment (reproduced from Li and Aubertin, 2012).

The Mitchell et al. (1982) approach relies on a validity criterion defined as a function of the stope height-to-width ratio and the backfill friction angle:

$$Criterion: \frac{H}{B} \geq \tan\alpha \qquad \textit{(eq. 38)}$$

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Where H = Stope height (m)
B = Stope width (m)
α = 45 + $\frac{\phi}{2}$ (°)
ϕ = Internal friction angle of backfill (°)

If this criterion is satisfied, the required backfill strength can be calculated using the following equation:

$$\sigma_F = \frac{\gamma L H}{L + H} \qquad \textit{(eq. 39)}$$

Where σ_F = Compressive strength of backfill (kPa)
γ = Unit weight of backfill (kN/m^3)
L = Stope length (m)

For each stope type (i.e., transverse and longitudinal), all three methods were applied, and the final recommended CPB strength requirements were established by comparing the values derived from each approach.

To evaluate the CPB strength requirements for the underhand mining sequence, a simplified 3D numerical model was developed in *FLAC3D* for both transverse and longitudinal stopes. The CPB strength was incrementally reduced in the model until noticeable backfill deformation was observed. A target factor of safety of 2.0 was used for the assessment.

The empirical and numerical analyses were conducted using the following inputs:

01. **Cemented Paste Backfill**
    - → Target factor of safety: 1.5 – 2.0
    - → Density: 20 kN/m^3
    - → Internal friction angle: 30°
02. **Stope Dimensions**
    - → Transverse: 17 mL × 25 mW × 30–36 m
    - → Longitudinal: 17 mL × 7 mW × 30–36 m

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8.5.2 Results

The empirical analyses results are presented in Table 59 for FoS ranging from 1.5 to 2.0.

Table 59. Comparison of CPB strength requirements obtained from empirical analyses.

Stope Type – Surface	Method	Factor of Safety (–)			
		2.0	1.85	1.7	1.5
Transverse – EW	Mitchell et al. (1982)	590	546	502	443
	Li and Aubertin (2012)	537	511	483	441
	Belem et al. (2022)	533	506	476	432
Transverse – HW/FW	Mitchell et al. (1982)	462	427	393	346
	Li and Aubertin (2012)	405	388	369	339
	Belem et al. (2022)	464	434	402	357
Longitudinal	Mitchell et al. (1982)	234	217	199	176
	Li and Aubertin (2012)	236	231	225	215
	Belem et al. (2022)	227	221	214	202

Codes **CPB** Cemented paste backfill, **EW** Endwalls, **HW–FW** Hanging wall and footwall.
Notes The criteria for Mitchell et al. (1982) method was not valid for the Transverse – HW/FW.

The numerical model results for both transverse and longitudinal stopes were likely optimistic for the study. Based on experience and comparisons with other mine sites, more conservative values were deemed appropriate for design purposes.

Figure 50 presents the predicted displacement within the backfilled transverse stope. In this model, a 7 mH plug composed of pastefill with a strength of 800 kPa is represented at the base of the stope, while the remaining backfill consists of cemented paste backfill (CPB) with an assigned strength of 590 kPa.

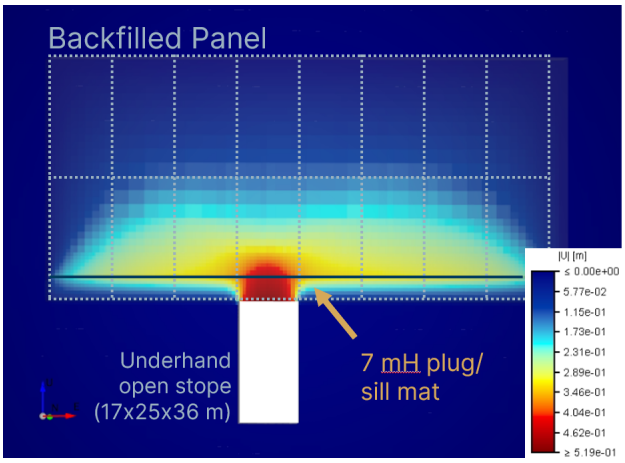


Figure 50. Longitudinal view of numerical modelling results for underhand mining beneath a backfilled stope or panel.

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8.5.3 Guidelines

Table 60 presents the recommended strength requirements for cemented paste backfill (CPB). CPB is expected to be used for all stopes, with the exception of isolated stopes, which may be filled with uncemented rockfill (URF). If URF cannot be used and CPB is required instead, a minimum strength of 100 kPa should be adopted.

For the transverse primary and secondary stopes, and the longitudinal stopes, Table 60 provides a range of CPB strength requirements. This reflects the decision to adopt a factor of safety (FoS) of 2.0 at the start of the mine life, gradually reducing it to 1.5 after three years of operation, in line with conventional engineering optimization practices. Table 61 summarizes the corresponding annual strength requirements for both stope types.

For underhand mining, conservative CPB strengths are recommended. While these values may be optimized during operations, doing so would require monitoring and additional analysis.

Different CPB strength requirements apply in areas where the underground stopes interact with the open pit. This topic is discussed in Section 8.10; however, the associated CPB strength recommendations are still included in Table 60 for completeness.

Table 60. Summary of cemented paste backfill strength requirements.

Stope Type	**Strength Requirement** (kPa)	**Curing Period to Reach Required Strength** (days)
Longitudinal retreated	210–240 [1,2]	28 [1]
Longitudinal retreated – last stope (minimum capacity if URF not possible)	100	14
Transverse primary and secondary	440–540 [1,2]	28 [1]
Transverse secondary – last stope (minimum capacity if URF not possible)	100	14
Pit–stope interaction zone	2 000	56
Underhand mining	5–15 mW stope: 1 200 15–25 mW stope: 1 600	56
Upper mining [3]	100	14

Codes **CPB** Cemented paste backfill, **URF** Uncemented rockfill.
Notes [1] Strength of 200 kPa required after 3 days of curing.
[2] Refer to the Table 61 for detailed values.
[3] Upper mining: when stopes are mined from a single drift beneath the stope.

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Table 61. Cemented paste backfill strength requirement over mine life.

Year	**Strength Requirement** (kPa)	
	Longitudinal	**Transverse**
0	240	540
1	230	510
2	220	480
3+	210	440

At this stage, fill barricades are considered to consist either of shotcrete walls or muck barricades. For muck barricades, a trapezoidal geometry is assumed following (Yang et al., 2017):

- Back length: 5.5 m
- Base length: 17.0 m
- Volume: 310 m^3

No cemented paste backfill (CPB) testwork results were available at the time these analyses were completed. For budgeting purposes, it is proposed to assume a binder content of 3% in the slurry for longitudinal stopes and 5% for transverse stopes. A curing period of 14 days should be assumed prior to blasting above a backfilled stope, and 28 days prior to exposing CPB walls. These guidelines should be reviewed once laboratory test results for the paste backfill become available.

Additional strength requirements must also be considered. A plug strength greater than 150 kPa is required before placing the residual fill. Furthermore, a residual strength exceeding 75 kPa is required before placing a muck layer over the CPB, and a residual strength greater than 100 kPa is required before blasting within a 25 m radius of the stopes.

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8.6 Crown Pillar

The following sections present the analysis of the surface crown pillar. This section addresses only the vertical pillar.

8.6.1 Design Methods

The Shaakichiuwaanaan CV5 underground mine is in a particular setting, as it is intended to be developed beneath a lake. Following Quebec's Regulation respecting occupational health and safety in mines (Éditeur officiel du Québec, 2025):

> ***Article 76***
>
> *For the purposes of this Subdivision:*
> - *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 open pit mine is under the influence of a body of water where the width of the protective wall is less than 100 m (328.1 ft) from the body of water at its highest point.*

> ***Article 77***
>
> *An underground excavation under the influence of a body of water may not be begun without there being obtained from an engineer plans and specifications 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.*
>
> *The plans and specifications and the accompanying studies shall be kept at the mine site and shall be available at all times.*

Based on discussions with PMET, a preliminary stability assessment of a surface crown pillar less than 100 m thick was undertaken to provide increased design flexibility at this stage of the Project. This assessment was completed despite some supporting studies not yet being available to fully satisfy regulatory requirements. By comparison with similar sites in Québec, an assumed pillar thickness of 85 m was selected for evaluation. It should be noted that this value does not necessarily represent an optimized design scenario.

The crown pillar design was evaluated using two complementary approaches:

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01. **Scaled Span Empirical Method (Carter, 2014)**
02. **Three-Dimensional (3D) Finite Difference Numerical Modelling**

The methods and their results are presented below.

First, the Scaled span method was used to complete the crown pillar thickness stability assessment. The required inputs are the rock mass classification index Q and the effective span C_s, which together allow an estimate of the probability of failure (PoF) of the pillar. The relevant equations are defined below:

$$Q = Q' \times Jw/SRF \qquad \textit{(eq. 40)}$$

where Jw = joint water reduction factor
SRF = stress reduction factor

$$C_s = S \times \sqrt{\frac{\gamma}{T \times (1 + S_r) \times (1 - 0.4\cos\theta)}} \qquad \textit{(eq. 41)}$$

where S = crown pillar span (or underground excavation span) (m)
T = pillar thickness (m)
L = length of crown (m)
γ = rock mass density (t/m^3)
S_r = S/L = span ratio (S/L)
θ = dip of pillar/foliation/mineralisation (°)

$$PoF\ (\%) = \frac{100}{1 + (440 \times \exp(-1.7 \times \frac{C_s}{Q^{0.44}}))} \qquad \textit{(eq. 42)}$$

The main rock units composing the surface crown pillar are amphibolite, metasediments, and pegmatite. Since all three units are present within the pillar, it is assumed that the lower-quality rock type is most likely to govern crown pillar stability. Accordingly, the input parameters for the analysis are based on the pegmatite unit, as it exhibits the lowest Q' values (i.e., first quartile, Q1). The minimum and maximum values adopted for the various inputs to the analysis are detailed below:

01. **Geomechanical Classification Q**
    - → Q' – the values correspond respectively to the 25th and 50th percentiles of the runs analyzed: 7 and 18, respectively.
    - → J_w – the minimum value is established as jet inflow or high pressure in competent rock with unfilled joints, while the maximum value considers a medium inflow and occasional outwash of joints. Values ranging from 0.50 to 0.66 were assumed.

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→ SRF – the stress reduction factor corresponds to a medium stress level for a competent rock mass: 1.0.

02. **Pillar Span**
    → The excavation corresponds to a stope size of 17 mL × 25 mW × 30-36 mH. Therefore, a minimum and maximum width of 17 m was assumed.
03. **Length of Crown**
    → Similarly to the pillar span, since the excavation corresponds to a stope of size 17 mL × 25 mW × 30-36 mH, a minimum and maximum width of 25 m is used.
04. **Rock Mass Density**
    → The rock mass density is assumed to be 2 800 kg/m^3.
05. **Pillar Dip**
    → Values between 45° and 55° are used and take into account the foliation dip in this area.
06. **Pillar Thickness (vertical height)**
    → A thickness between 80 and 90 m is used.

The numerical model introduced in Section 8.2 was used to evaluate the stability of the surface crown pillar. In this model, the open pit, the stopes, and the overburden were included, whereas the dam and the water body were not explicitly represented. In addition, the model did not incorporate development openings or pore pressure.

The model results were examined to determine whether plastic strain –indicative of permanent deformation (e.g., fracturing) – develops within the crown pillar during the period 2031–2049.

8.6.2 Results

The empirical analysis, incorporating both lower- and upper-bound input parameters, indicates that a pillar thickness of 85 m would be acceptable from a stability perspective. This assessment is based on the conservative assumption that no ground support will be installed, thereby providing an adequate safety margin. Under these assumptions, the surface pillar exhibits an average probability of failure (PoF) of 1.5%, with a minimum–maximum range of 1.0–3.7%. Based on these results, the pillar corresponds to a “Class F” stability category.

According to the risk exposure acceptance guidelines presented in Table 62, a "Class F" pillar is recommended given the length of time for which the span will remain exposed. This class corresponds to a long-term structure with a service life of 50 to 100 years. The pillar thickness adopted in the design is therefore considered appropriate for this classification.

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Table 62. Guidelines for acceptable risk exposure. Reproduced from Carter and Miller (1995).

Class	Probability of Failure		FS min.	Max. Scaled Span	Design Guidelines for Pillar Acceptability / Serviceable Life of Crown Pillar				
	Min %	Max %			Expectancy	Years	Public Access	Regulatory Position on Closure	Operating Surveillance Required
A	50	100	< 1	$11.31 \times Q^{0.44}$	Effectively zero.	< 0.5	Forbidden	Totally unacceptable	Ineffective
B	20	50	1.0	$3.58 \times Q^{0.45}$	Very short-term [1]	1.0	Forcibly prevented	Not acceptable	Continous sophisticated monitoring
C	10	20	1.2	$2.74 \times Q^{0.46}$	Very short-term[2]	2-5	Actively prevented	High level of concern	Continous monitoring with instruments
D	5	10	1.5	$2.33 \times Q^{0.47}$	Short-term[3]	5-10	Prevented	Moderate level of concern	Continous simple monitoring
E	1.5	5	1.8	$1.84 \times Q^{0.48}$	Medium-term[4]	15-20	Discouraged	Low to moderate level of concern	Conscious superficial monitoring
F	0.5	1.5	2	$1.12 \times Q^{0.49}$	Long-term[5]	50-100	Allowed	Of limited concern	Incidental superficial monitoring
G	0.0	0.5	> 2	$0.69 \times Q^{0.50}$	Very long-term[6]	>100	Free	Of no concern	None required

[1] Temporary mining purposes only.
[2] Undesirable for temporary civil works.
[3] Semi-temporary crowns, e.g., under non-sensitive mine infrastructure.
[4] Semi-permanent crowns.
[5] Quasi-permanent crowns, civil portals.
[6] Permanent crowns over civil tunnels.

The stability chart incorporating the parameters described above is presented in Figure 51. The orange rectangle illustrates the following:

→ This is the range of possible values for the crown pillar.

→ The top-left corner of the rectangle corresponds to the scenario that incorporates all the most conservative parameter values used in the analysis.

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→ The bottom-right corner of the rectangle corresponds to the scenario that incorporates all the least conservative parameter values.

The red diamond represents the most realistic estimate of the proposed surface crown pillar, based on the selected parameters and assumptions.

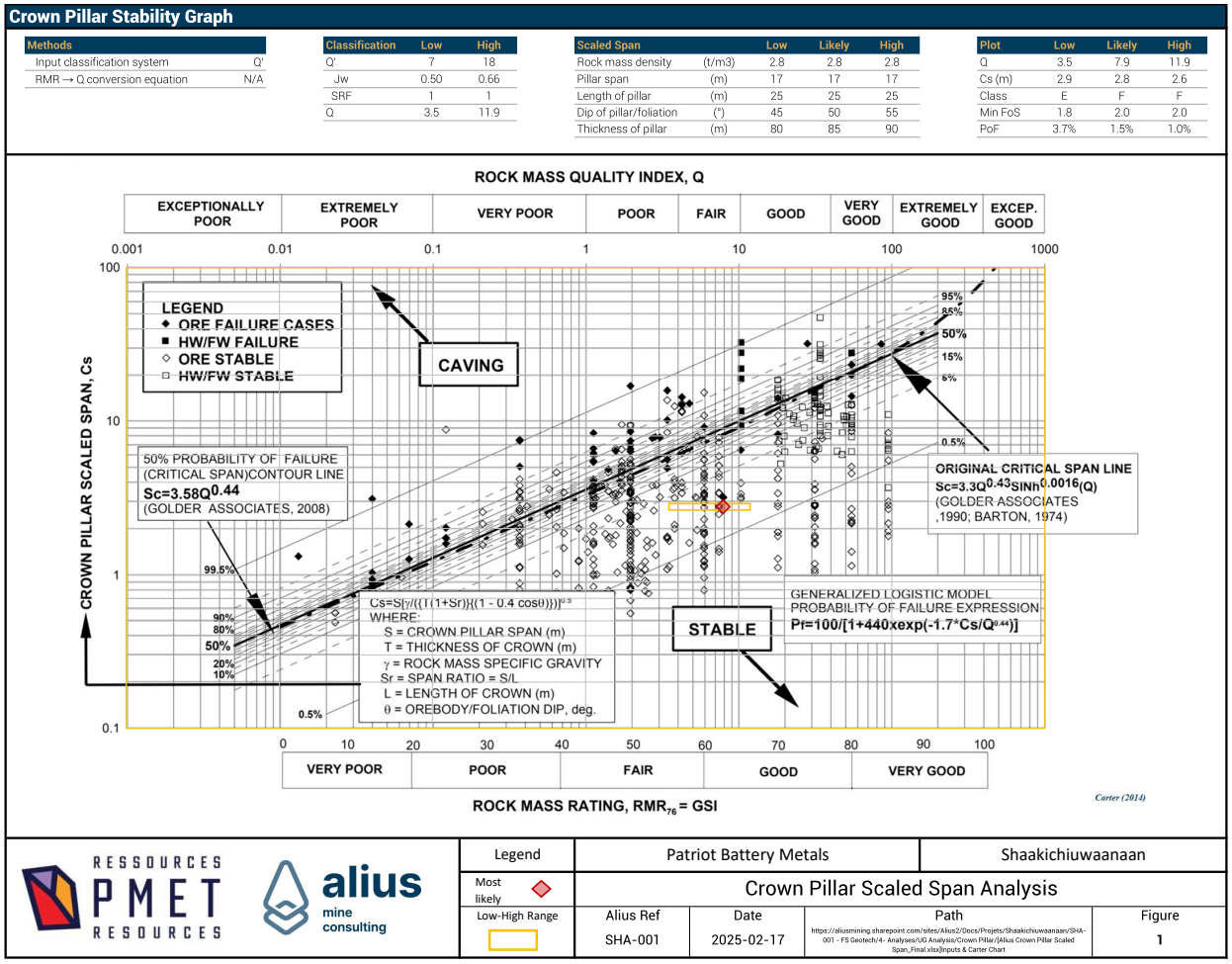


Figure 51. Scaled span stability chart with proposed inputs for the crown pillar.

The modelling results are presented in Figure 52. The numerical model indicates that plastic strain begins to develop in the crown pillar very late in the mining sequence (Years 2046–2049). This deformation occurs following the extraction of the uppermost stopes located directly beneath the water body. Further investigations and refinements – including potential adjustments to the mining sequence – will be required to mitigate this risk.

More specifically, in Year 2047, deconfinement is observed around the stopes and within the pillar. At this stage, the associated risk remains low and acceptable. By Year 2049, plastic strain is clearly present in the crown pillar. This will require additional analysis to ensure safe operating conditions when mining beneath the lake.

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Year 2031 – No plastic strain



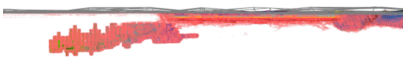
Year 2033 – No plastic strain



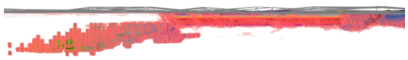
Year 2035 – No plastic strain



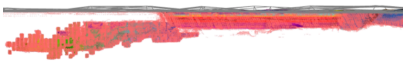
Year 2037 – No plastic strain



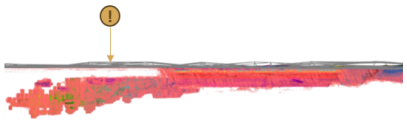
Year 2039 – No plastic strain



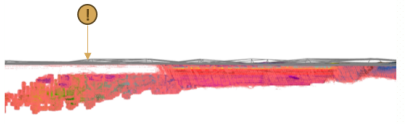
Year 2041 – No plastic strain



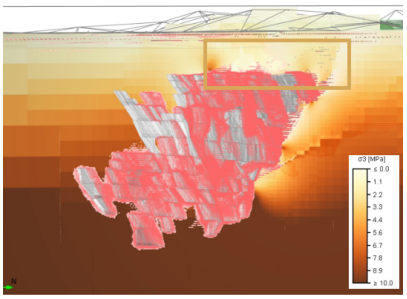
Year 2043 – No plastic strain



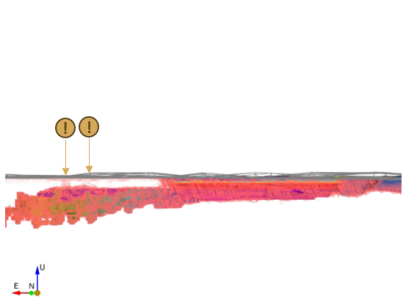
Year 2045 – Slightly visible plastic strain



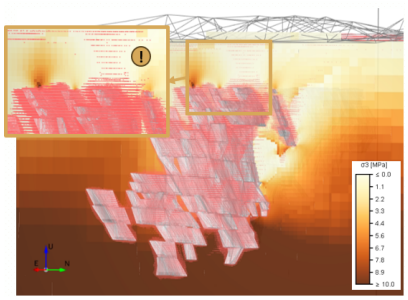
Year 2047 – Some visible plastic strain



Year 2047 – Minor principal stress (S3) & plastic strain



Year 2049 – Visible plastic strain



Year 2049 – Minor principal stress (S3) & plastic strain

Figure 52. Plastic strain development in the crown pillar for the period 2031–2049. Years 2047 and 2049 include minor principal stress (S3).

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8.6.3 Guidelines

The crown pillar design was evaluated using an empirical method combined with 3D numerical modelling. The empirical analysis, incorporating lower- and upper-bound input parameters, indicates that a minimum pillar thickness of 85 m is required. If the pillar thickness is reduced below 85 m, additional mitigation measures will be needed to manage operational risk. These may include remote drilling techniques and other engineering controls.

The numerical modelling shows that plastic strain begins to develop in the crown pillar late in the mining sequence (years 2047–2049). This deformation occurs after the extraction of stopes located near the top of the sequence, directly beneath the water body. Additional investigations and refinements—including optimization of the upper stope mining sequence—will be required to mitigate this risk.

Based on observations from the final modelling steps, it is recommended that the mining sequence of the upper stopes be optimized to reduce loading and deformation effects on the crown pillar.

Recommendations for using an 85 m crown pillar are based on the currently available data and analyses. Additional engineering studies—including those related to geotechnical soils, rock mass strength, structural design, and hydrogeology—will be required to fully satisfy the requirements of the Regulation respecting occupational health and safety in mines, Article 77, as the Project advances toward execution. Installation of monitoring instruments is recommended to track deformations and stress changes in the pillar, and to validate ongoing crown pillar stability. Operational awareness tools – such as live monitoring and a Trigger Action Response Plan (TARP) – are further measures that can help ensure safe and sustainable operations, particularly when mining within 100 m of the water body.

To reduce the risk of a plug failure propagating upward to the surface, rockfill should be placed in the overcut drive to limit the vertical extension of any localized failure. During stope development, two stopes may be opened simultaneously, provided a minimum spacing of at least one stope width (17 m) is maintained between them. During the overcut phase, muck should be placed to within 2–3 m of the overcut back. Assuming a 3 m void and a 25% swell factor, the maximum potential cave height is estimated at 12 m. For an 85 m crown pillar, this represents approximately 15% of the total pillar thickness.

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8.7 Ground Support Design

The ground support guidelines for the underground development were assessed to ensure safe and reliable conditions. Design methods and guidelines are presented below.

8.7.1 Design Methods

Initial support recommendations were developed using empirical guidelines, practical experience, and engineering judgment. The stability of the proposed support systems was evaluated using in-house scripts and tools, targeting a factor of safety of 1.5. For intersections, the analyses were verified using the *National Institute for Occupational Safety and Health* (NIOSH) *Ground Support Factor of Safety Calculator*, which applies the three-dimensional (cone) method (Pakalnis, 2015). As an example, a 10 m three-way intersection was assessed using the software to determine the factor of safety for the recommended support configuration (Figure 53), resulting in a FoS of 1.51. Where required, support configurations were iteratively refined to achieve an acceptable factor of safety while balancing safety, constructability, and operational efficiency.

The assessment of ground support for the portal development has not yet been carried out in detail (Section 8.8); however, this will be fully addressed during the Project's detailed engineering phase.

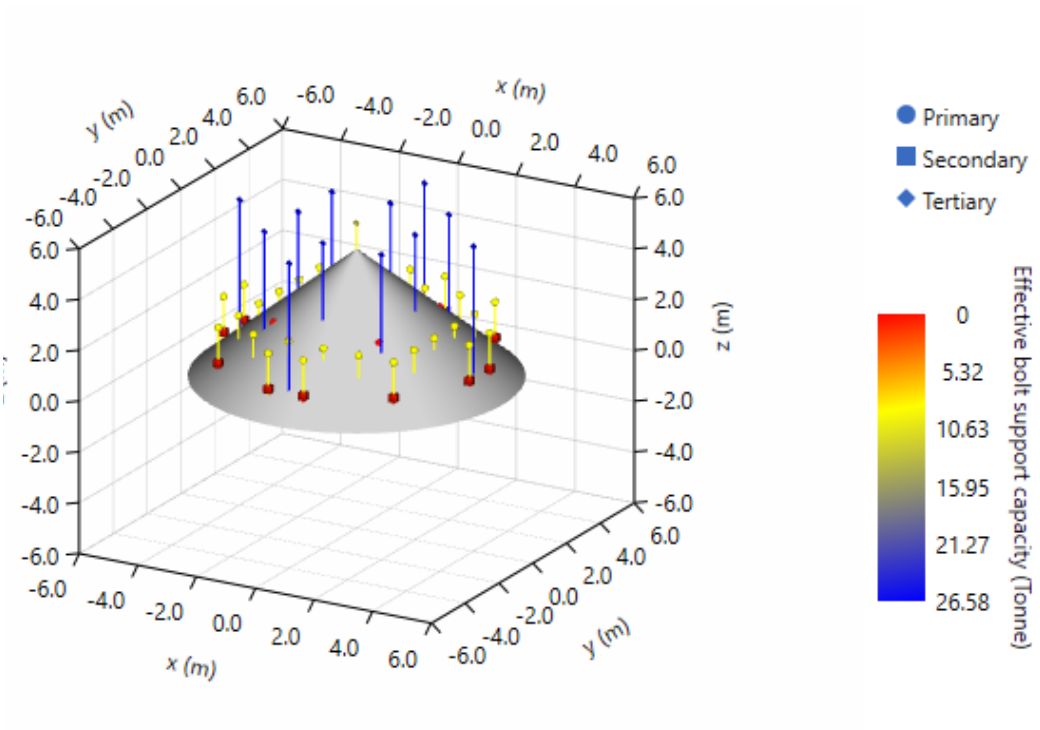


Figure 53. Example of the ground support for a typical 10 m intersection (3-way).

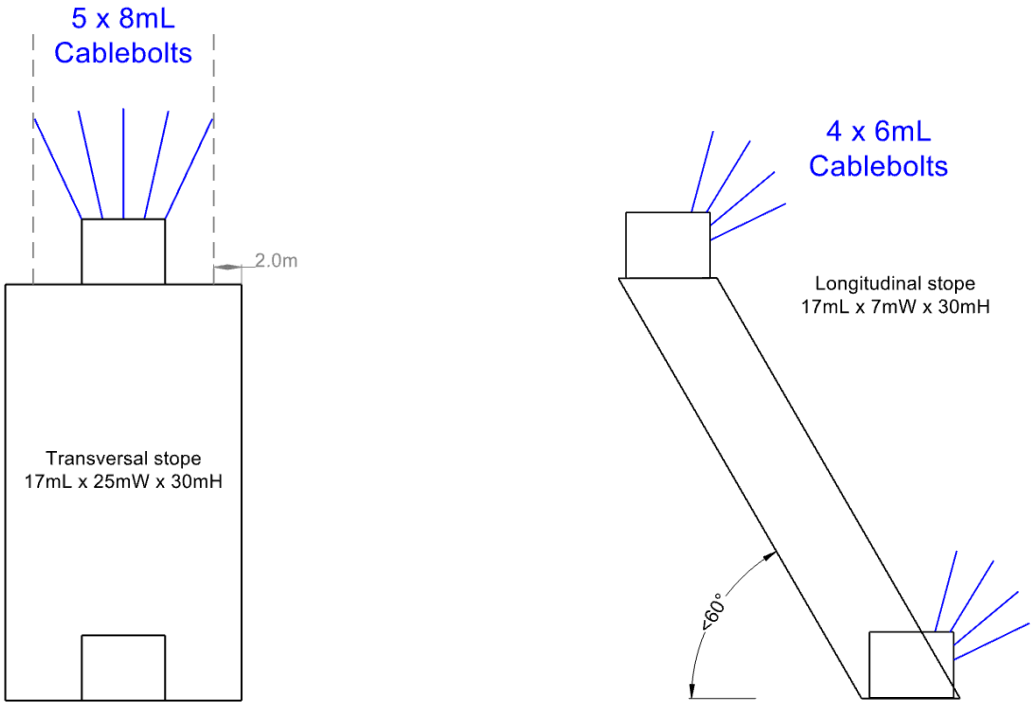
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8.7.2 Guidelines

Permanent development is planned to be supported using 2.4 m resin rebars (back) and 1.5 m friction bolts (walls), on a 1.2 × 1.2 m pattern, with 4” × 4” 6-gauge mesh. 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 63.

Support intersections require 6 m cable bolts. It is recommended to avoid planning four-way intersections as the factor of safety (1.39) is below the 1.5 target with the current design recommendations.

Back support for transverse stopes was evaluated for various widths. 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 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 63, while Figure 54 shows a schematic representation of the ground support for the back, from the overcut drift.



(a) Example of back cablebolting in a 25 mW transverse stope.

(b) Example of HW cablebolting in a 7 mW longitudinal stope (dip<60°).

Figure 54. Schematic representation of ground support for stopes (not to scale).

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Table 63. Summary of ground support guidelines.

Excavation Type	Details	Typical Dimensions (m)		Max Span	Back Support		Wall Support	
		Height	Width	(m)	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
Lateral	Wide Dev.	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
Lateral	Backfill Development	5.5	5.5	6.0	Primary	SC; 2 × 2” + WWM + SX 2.1 m; 1.2 × 1.2 m	Primary	SC; 2 × 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	–	–
Intersections	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
Vertical	Raise	–	<5.0	5.0	–	–	Primary	RB 2.4 m; 1.2 × 1.2 m
Vertical	Raise	–	<5.0	5.0	–	–	Secondary	SS 1.0 m; 1.2 × 1.2 m
Stopes	Brow	–	–	RSG	Secondary	9 × CB 5 m	–	–
Stopes	Back	–	–	RSG	Secondary	RSG	–	–

Codes **CB** 15 mm Cablebolt, **RB** Rebar 20 mm (#6), **SC** Two passes of 2-inch shotcrete, **SS** Split Set 35 mm, **SX** 12 t Swellex, **WWM** Welded Wire Mesh, **RSG** Refer to stope guidelines.

Notes If secondary support is recommended, primary support must first be added (using the appropriate excavation type recommendation).
E.g., 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).
Lateral development: assumes wall support to 1.5 m from the floor, unless otherwise specified.
Lateral development: assumes 1.5 × 2.7 m galvanised WWM, #6 gauge.
Vertical development: assumes chain-link or WWM.
Use 10-ton dome plates, except for cables (use 17-ton plates).
Raises that are inaccessible to personnel during and after development do not require support (e.g., drop raises or temporary ventilation raises).

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8.8 Portal (Box cut)

This section presents the recommendations for the portal (box cut), including the access ramp and wall slopes, the portal face, and the surface pillar. The GMS team has determined the location of the main portal, along with its dimensions and the design of the access ramp. These parameters were used as inputs for the box cut assessment. The terminology used in this section is illustrated in Figure 55.

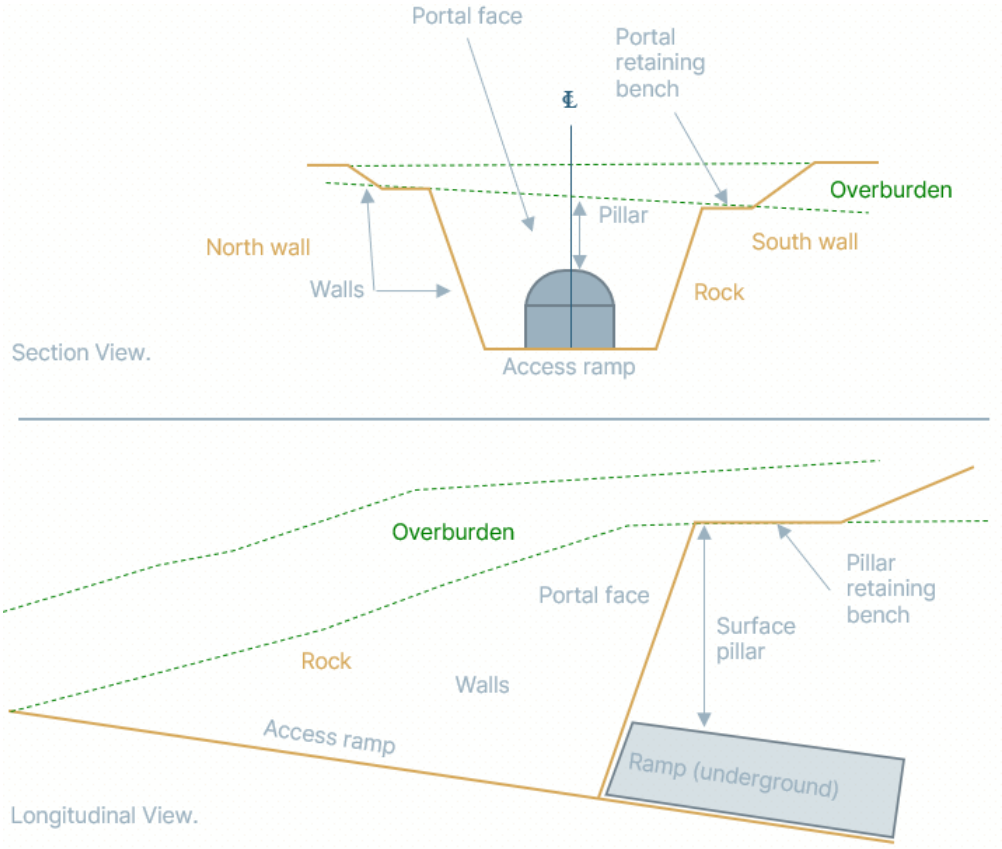


Figure 55. Terminology related to the general infrastructure of a portal (not to scale).

The location of the portal is planned to be approximately 400 m north of the pit and even further from the main area of underground stopes, as shown in Figure 56.

Design parameters for the box cut excavation slopes (in both overburden and rock), and the associated ground support recommendations have been developed based on limited available data.

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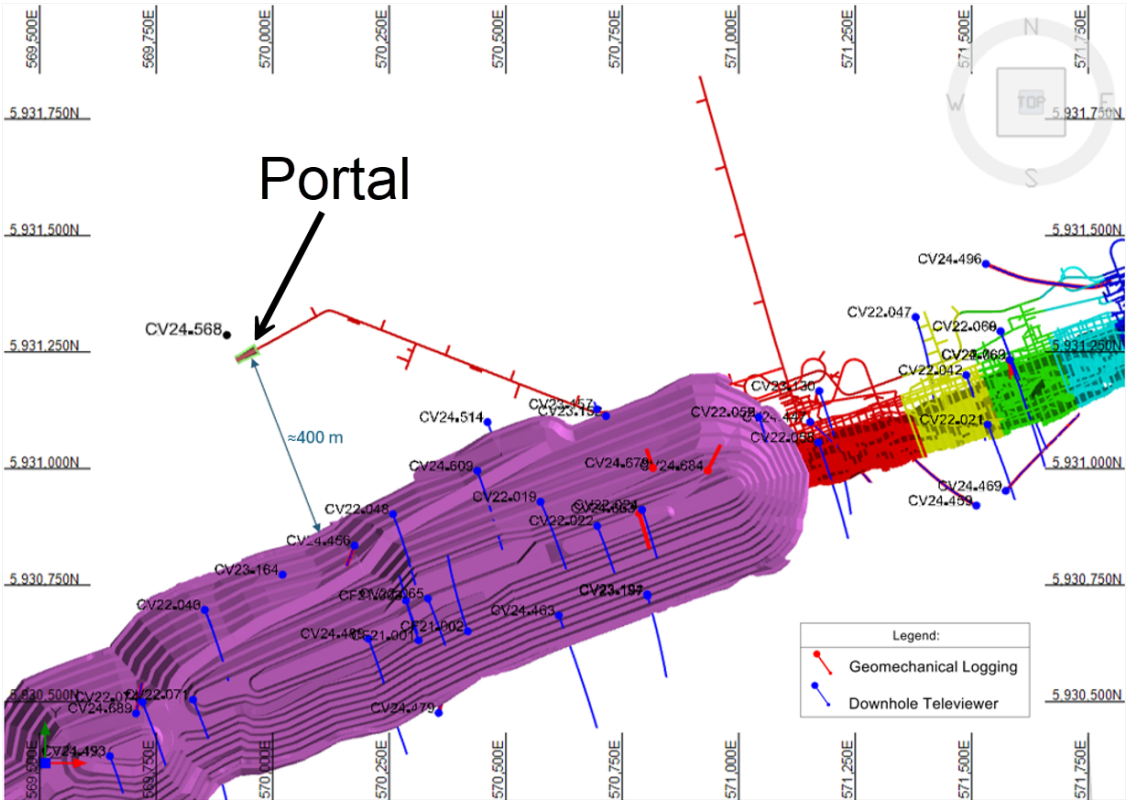


Figure 56. Planned portal location.

Overburden Slopes

Regarding the determination of the recommended overburden slopes, no site-specific data are currently available. The following assumptions have therefore been made: based on the lithological model and surface topography, the overburden thickness at the portal location is estimated at approximately 5 m. It is further assumed that the overburden material is similar in composition to that characterized elsewhere on site, consisting predominantly of till. At this stage, a 2H:1V slope is recommended for the overburden surrounding the portal area.

Rock Slopes

Only one drillhole (CV24-568) is located within a 50 m radius of the proposed portal site. No geomechanical core logging or televiewer surveys are available for this drillhole, nor for any drillhole within approximately 400 m of the portal location. Consequently, data from drillholes situated farther from the portal were used to support the analysis. Table 64 summarizes the assumptions and input parameters adopted for the portal box-cut design. Figure 57 illustrates the main components of the portal layout.

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Table 64. Assumptions and inputs for portal box cut design.

Parameter	Value
Rock lithology	Metasediments (from CV24-568 and lithological model)
Ramp height (m)	5.0
Ramp aperture width (m)	5.0–6.0
Ramp azimuth (°)	61
Portal face dip direction (°)	241
Left (north) portal wall dip direction (°)	151
Right (south) portal wall dip direction (°)	331
Televiewer surveys considered	**(CV22-)** 019, 048 **(CV23-)** 152, 157, 164 **(CV24-)** 456, 514, 609

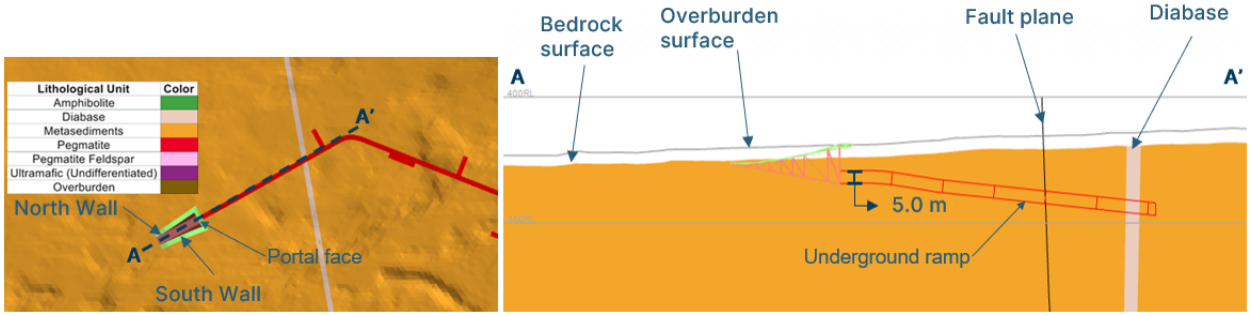


Figure 57. Plan view (left) and section view (right) of the main portal.

From the televiewer surveys considered, a stereonet of the discontinuities in the metasediments was prepared in *Dips* (Rocscience, 2024a) to determine the major joint sets. The resulting joint sets are presented in Table 65. One joint set (JS1) is considered major, whereas the other (js2) is considered to be a minor joint set due to its high joint spacing. Figure 58 shows the stereographic projections of discontinuities and interpreted joint sets. Note that the Terzaghi weighting correction is applied in this figure.

From the televiewer surveys considered, a stereonet of the discontinuities in the metasediments was prepared to identify the major joint sets. The resulting joint sets are presented in Table 65. One joint set (JS1) is interpreted as a major set, while the other (js2) is classified as minor due to its large joint spacing. Figure 58 illustrates the stereographic projections of the discontinuities and the interpreted joint sets (including Terzaghi weighting correction).

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Table 65. Interpreted joint sets in the metasediments – Main portal sector.

Lithology	Joint Set	Dip (°)	Dip Direction (°)	Mean Joint Spacing (m)	Number of Poles
Metasediments	JS1	50	165	1.72	133
	js2	49	342	4.66	43

Notes Capitalized JS: major joint set. Lowercase js: minor joint set.

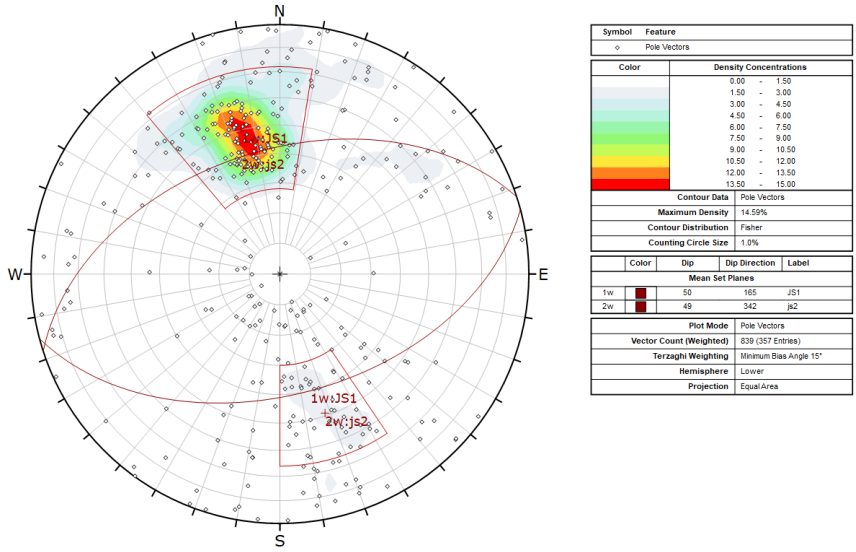


Figure 58. Stereographic projections of discontinuities in the metasediments considered in the portal analysis and interpreted joint sets.

Limit equilibrium stability analyses were conducted to evaluate the potential for planar and wedge failures using *RocPlane* (Rocscience, 2025a) and *SWedge* (Rocscience, 2024b), respectively. The joint properties used in the analyses correspond to those of the metasediments (ϕ = 23.5°; see Section 5.1.2). A cohesion value of zero was assumed, reflecting the shallow bedrock conditions in the portal area, where the rock mass is expected to be more fractured and subject to surface weathering. A rock density of 2 876 kg/m^3 was adopted. In the analyses presented below, the slope dip was evaluated at an angle of 80°.

The following conclusions are drawn:

01. **Planar Sliding**

- → The north wall slope is controlled by the presence of JS1. The recommended slope for the north wall is therefore set equal to the average dip of JS1, which is 50°.
- → For the south wall, no limitation is required: js2 is considered a minor joint set with a large joint spacing, and thus planar sliding does not govern the stability of either the portal face or the south wall.
- → Table 66 summarizes the results of the planar sliding assessment:

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Table 66. Planar sliding assessment of portal slopes.

Slope	Dip Direction (°)	Critical Joint Set	Factor of Safety	Notes
North wall	151	JS1	0.37	Difference in dip direction with JS1 of 14°.
South wall	331	js2	0.38	Difference in dip direction with js2 of 11°; high joint spacing measured.
Portal face	241	None	—	No critical joint set identified.

02. **Wedge Sliding**

→ Wedge sliding does not limit the slope of any of the faces.

→ Table 67 summarizes the results of the wedge sliding assessment:

Table 67. Wedge sliding assessment of portal slopes.

Slope	Dip Direction (°)	Critical Joint Sets	Factor of Safety	Notes
North wall	151	JS1 & js2	—	No wedge formed.
South wall	331	JS1 & js2	21.8	—
Portal face	241	JS1 & js2	21.8	—

Surface Pillar

The surface pillar assessment was performed using the Scaled Span method of Carter (2014), as introduced in Section 8.6. The input parameters adopted for the analysis are summarized below, with metasedimentary rocks expected to be present within the pillar area:

01. **Geomechanical Classification Q**

→ Q' – In the absence of data close to the portal, it is assumed that the rock quality at the portal location is equivalent to the quality of the metasediments in the underground area. However, when designing portals, it is recommended to multiply the J_n parameter by 2, which has a direct effect on the determined Q value. The adjusted values correspond, respectively, to the 25th and 50th percentiles of the analyzed runs (i.e., 5 and 8).

→ J_w – the minimum and the maximum value consider minor water inflow. A value of 1 is assumed.

→ SRF – The stress reduction factor set at the lowest possible for a competent rock mass, i.e. 2.5. As the pillar is at the surface, very low stress components are expected.

→ Q – As per equation eq. 6, the Q classification is obtained from the previous inputs, resulting in values of 2.0 and 3.2. Once the crown pillar is considered supported, the Q values of 7 and 9 are obtained.

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02. **Pillar Span**

→ A minimum width of 5 m is used as it is the expected underground ramp width. A maximum width of 7 m is used to account for a conservative scenario (e.g., overbreak).

03. **Length of Crown**

→ An influence length ranging from 50 to 100 m is used. This input has very little impact on the results.

04. **Rock Mass Density**

→ The rock mass density is assumed to be 2 800 kg/m^3.

05. **Pillar Dip**

→ Values between 45 and 55° are used and take into account the foliation dip in this sector.

06. **Pillar Thickness (vertical height)**

→ A thickness of between 8 and 12 m is used.

The stability chart incorporating the parameters mentioned above is shown in Figure 59. The signification of the orange rectangle and the red diamond is presented in section 8.6.2.

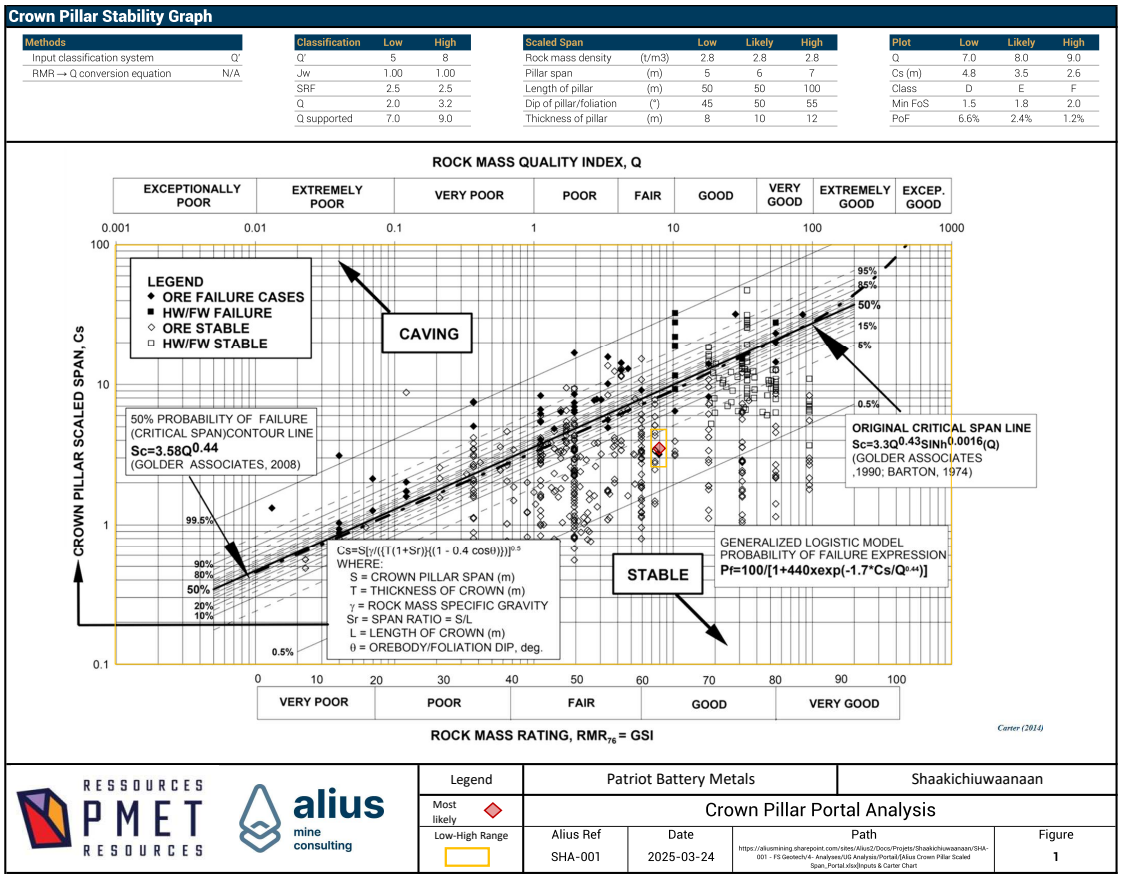


Figure 59. Scaled span stability chart with proposed inputs for the portal surface pillar.

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With support, the surface pillar has an average probability of failure (PoF) of 2.4%, with a minimum–maximum range of 1.2–6.6%. Based on these results, the pillar corresponds to a “Class E” stability category.

According to the risk-exposure acceptance guidelines presented in Table 62, a "Class E" pillar is recommended given the long-term duration for which the span will remain exposed. The pillar thickness adopted in the design is therefore considered appropriate for this classification.

8.8.1 Guidelines

Table 68 summarizes the dimensions and configuration of the portal box cut design. General recommendations are provided for the portal ground support (pillar and faces).

Table 68. Portal box cut design dimensions and configuration.

Parameter	Value
Access ramp width (m)	10
Pillar thickness (m)	10
Pillar support	Either cablebolted from surface (drilled downward) or from ramp (drilled upward) using 6 m cables on a 2 × 2 m pattern (ideally plated, if drilled from ramp)
Pillar retaining bench (m)	20
Portal retaining bench (m)	2 (RSSTM art. 41-1a)
Rock slope – north wall (°)	50
Rock slope – south wall (°)	80
Rock slope – portal face (°)	80 or 90[1]
Temporary face and wall support	Split-Set (FS35) 1.8 m and chainlink
Overburden slope	2H:1V

Notes [1] The portal face slope depends on excavation conditions.

The presence of both the fault FLT_1525 and the diabase dyke may be associated with locally reduced rock quality. Due to the absence of drillholes in this area, the rock quality remains uncertain, and the exact positions of these two major structures have not been confirmed. Based on the current modelling, these structures are expected to intersect the underground ramp approximately 90–120 m from the portal face. Given the limited data available for the portal box cut design, the input parameters and resulting recommendations should be reviewed and refined during the detailed engineering phase.

Ground support requirements for the portal development have not yet been evaluated in detail; this assessment will need to be completed during the Project’s detailed engineering phase.

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8.9 Infrastructures

The standoff distances of underground infrastructures were evaluated using the global numerical model. Key infrastructures include ramps, raises, and haulage drifts.

In the development layout proposed by GMS in March 2025, the main ramp is consistently located more than 100 m from the stopes. This distance should be maintained as a minimum requirement to prevent potential excavation damage. For haulage drifts, a minimum distance of 20 m from the stopes is recommended to ensure stability. Finally, raises should be positioned at least 40 m away from the stopes to maintain structural integrity and minimize interaction with mined-out zones.

Standoff distances from the open pit excavation are also required for some underground excavations. The main ramp and the ventilation raise L305 have been investigated as they are the two (2) closest infrastructure to the open pit. Figures 60 and 61 present the minor principal stress (S3) and rock mass damage around a ventilation raise and the main ramp where these openings are closest to the open pit.

Figure 60 shows that no plastic strain or rock damage is observed around the raise at its location. Similarly, Figure 61 indicates that although some deconfinement occurs near the pit area, no plastic damage is detected in the vicinity of the ramp. As this section represents the shortest distance between the open pit and the main underground ramp, it is recommended to install instrumentation such as MPBX in this area to monitor potential ground response over time.

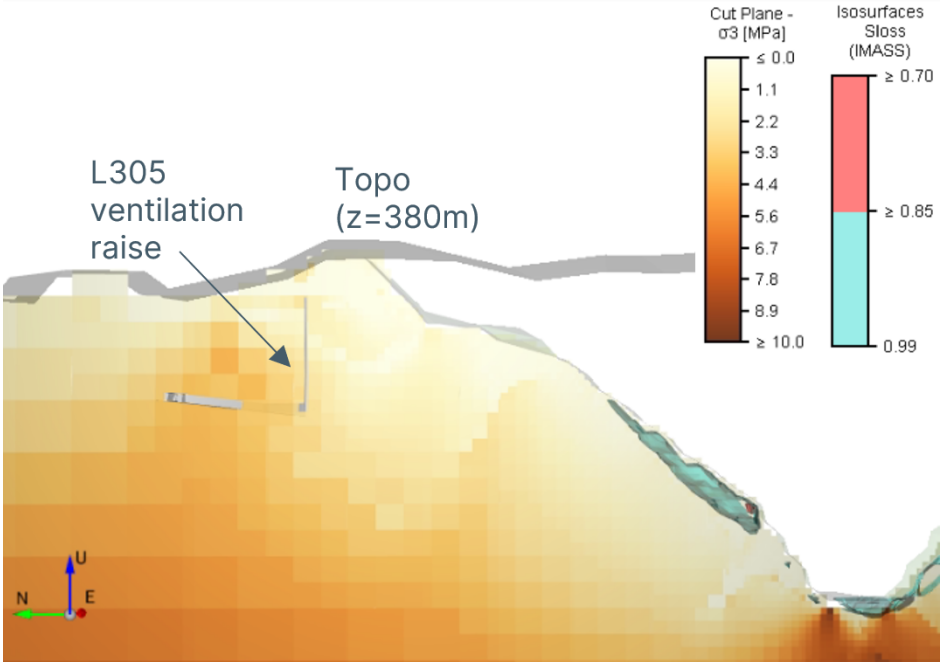


Figure 60. Minor principal stress (S3) and rock mass damage around L305 ventilation raise.

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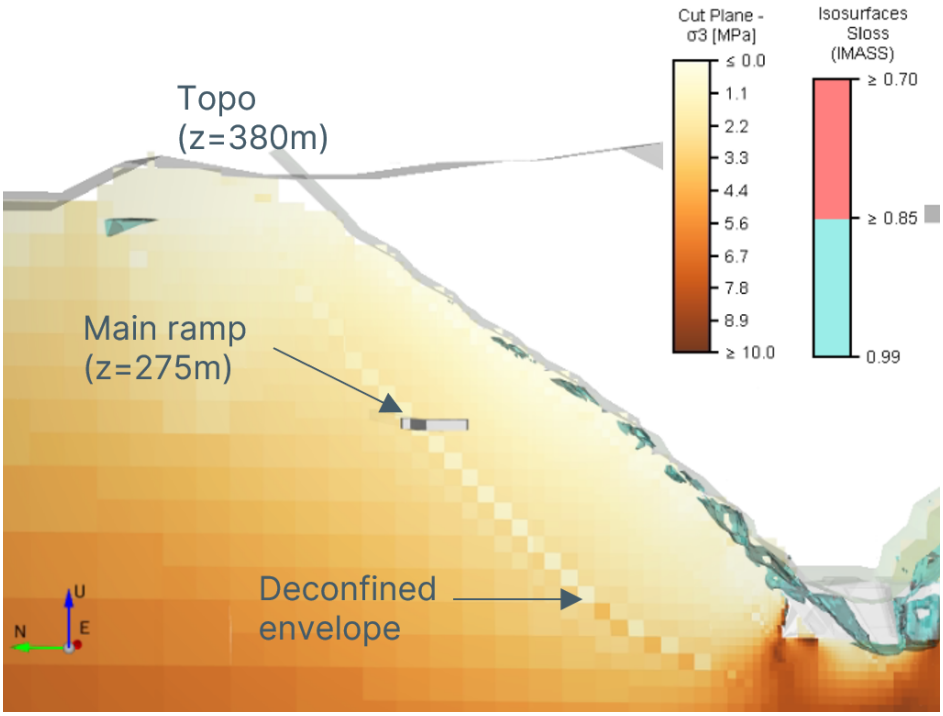


Figure 61. Minor principal stress (S3) and rock mass damage around the main ramp located nearest to the pit.

8.10 Open Pit–Underground Interaction

The underground mine design extends toward the north-eastern end of the open pit, where several stopes are located immediately adjacent to the planned pit wall. As a result, it is necessary to evaluate the potential interactions between the open pit slope and the underground excavations. In particular, an assessment of the required backfill strength and of the global stability of the combined system is essential to ensure that underground stopes can be safely exposed on the pit wall. This analysis supports the objective of enabling early ore recovery in this part of the deposit while maintaining the stability of both the open pit and underground workings. In addition to establishing backfill strength requirements, the assessment also examined the minimum distance behind the pit wall at which strengthened backfilled stopes must be implemented.

To complete this assessment, two complementary approaches were adopted:

01. **Two-Dimensional (2D) Limit Equilibrium Modelling**
    → Built to assess local slope stability by determining both the backfill strength requirements and the pattern of stopes that should be reinforced.
02. **Three-Dimensional (3D) Finite Difference Numerical Modelling**
    → Built to validate the conclusions of the 2D model by applying the recommended guidelines to a representative 3D geometry. This approach accounts for the confinement effects associated with the actual three-dimensional pit configuration.

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The 2D limit equilibrium analysis was developed using the software *Slide2* (Rocscience, 2025b). It includes the optimized pit shell, the DSO stope layout (dated February 13, 2025), and the lithological model (dated January 13, 2025). A typical section was modelled to assess stability, assuming entirely backfilled stopes. This typical 2D section has been modified from the actual stope design to represent the worst section, taking into account minor modifications to the present lithologies and the extent of the stopes. Indeed, this typical section corresponds to a critical geometry, where approximately three (3) daylighting stopes, equivalent to about 90 m in height, are exposed on the pit wall.

Considering a backfill density of 1 960 kg/m^3 and a friction angle of 30°, a minimum uniaxial compressive strength of 2.0 MPa is required for the backfill to maintain a factor of safgety (FoS) > 1.3. A FoS value of 1.3 is targeted due to the conservative geometry assessed. Also, the required thickness of this high-strength pastefill behind the wall to ensure slope stability is 100 m (equivalent to six stopes). Figure 62 presents the factor of safety obtained on the typical section analyzed by 2D limit equilibrium analysis.

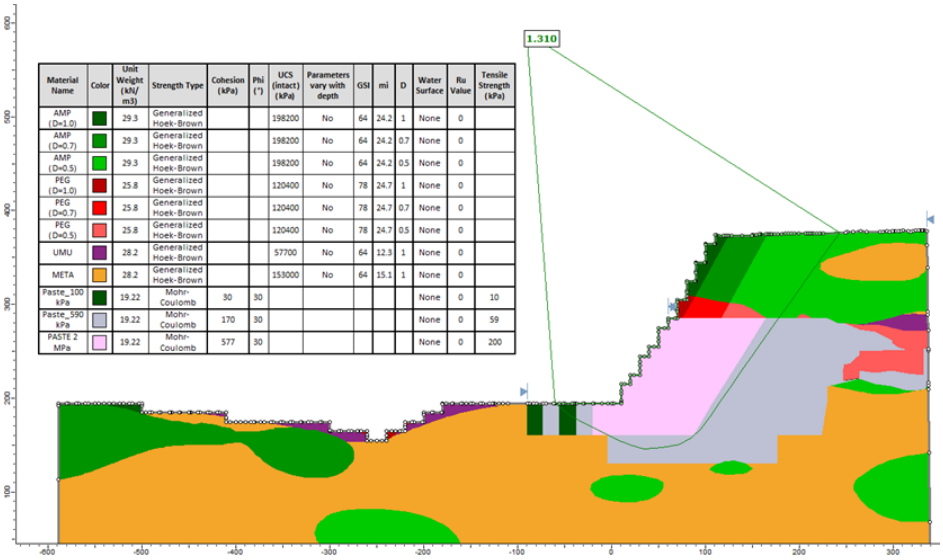


Figure 62. OP-UG interaction 2D modelling – non-circular failure in rock mass and backfill.

More precisely, the stopes included in the 2.0 MPa backfill strength requirement are:

- Stopes within 100 m horizontally from the open pit wall, including those on Level 170.
- Stopes under the pit floor within approximately 35 m horizontally (about two stopes) from the pit toe.

At this stage, the 2D model presents several limitations. The analysis includes three daylighting stopes. Should additional stopes be expected to daylight, the results would require revision. The model assumes purely two-dimensional conditions and therefore does not capture three-dimensional confinement effects, which would have a stabilizing influence given the pit curvature. The analyzed section represents a worst-case geometry, slightly adjusted from the current design

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to reflect conservative assumptions. This analysis should be validated and refined during the detailed engineering phase.

To validate the 2D modelling conclusions, a 3D numerical model was built in *FLAC3D* using the same input parameters as the global model, with a few exceptions. The main difference between both models is that only the stopes located within 100 m of the open pit walls were included. This approach reduced computational time while focusing the analysis on the pit–underground interaction. The objective of this model was to validate the factor of safety associated with a potential large-scale failure at the interface between the open pit and the underground stopes. The factor of safety was assessed by progressively reducing the strength of the cemented paste backfill.

As shown in Figure 63, less than 0.9 cm of displacement is anticipated, with no evidence of rock mass damage, when a backfill strength of 2.0 MPa is employed. The backfill properties were subsequently downgraded to 1.0 MPa. Given that overall stability was maintained at this lower strength, the factor of safety is considered to exceed 2. Although further optimization is possible, the recommended backfill strength of 2.0 MPa is confirmed to be appropriate.

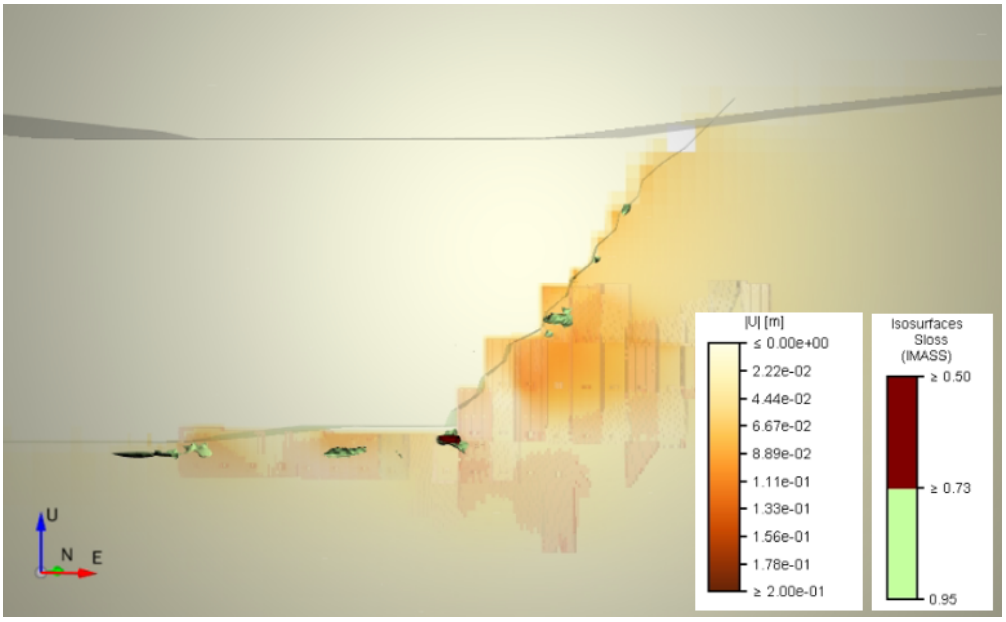


Figure 63. OP-UG interaction 3D modelling – Isosurfaces of rock mass damage and displacement.

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9 Operational Considerations

This section summarizes key operational practices to consider during mine planning, design, and execution. These considerations are intended to support operational safety, maintain slope performance, and allow for optimization of the mine plan as new information becomes available. The section is divided into three (3) subsections:

01. **Overburden Slopes**
02. **Open Pit Mining**
03. **Underground Mining**

9.1 Overburden Slopes

The following considerations apply to the construction, operation, and long-term performance of overburden slopes:

01. **Design Updates**
    - → Design criteria have been proposed for both local and global slopes. Should changes occur in material strength, stratigraphy, groundwater conditions, or excavation geometry, design criteria should be reviewed.
02. **Groundwater**
    - → Current stability analyses assume that the till has relatively high hydraulic conductivity and that excavation induces local drawdown, lowering the water table. Since groundwater is expected to emerge at the till toe, the toe must be secured, and a drainage channel should be excavated to ensure proper water drainage. The implementation of proactive water management measures to limit water pressure above the till layer is then required.
    - → Installation of vibrating wire piezometers behind overburden slopes is recommended to confirm porewater pressure assumptions and support early detection of changes in hydrogeological conditions.
    - → Seasonal variations (e.g., freeze–thaw cycles and rainfall infiltration) should be monitored, as they may impact pore pressures and soil strength.
    - → Any new occurrence of water inflow or any change in an existing seepage point on the wall surfaces should be monitored and documented.
03. **Construction Considerations**
    - → Overburden excavation should preferably be carried out during the winter months, when frozen ground conditions generally improve soil strength and workability. Although winter conditions often facilitate operations, excavation can still be performed during summer if the material allows.
    - → If stripping occurs during thawed conditions, equipment-induced disturbance should be minimized to reduce softening of the upper soils.

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04. Slope Monitoring

- When soil thickness is considerable (i.e., greater than 10 m), a proactive monitoring program is recommended throughout all stages of excavation. Monitoring should, at a minimum, rely on regular visual inspections, and should also consider quantitative measurements such as:
  - Prism monitoring
  - Slope stability radar (SSR or LiDAR-based systems)
  - Piezometric measurements
- Sufficient staffing resources should be assigned to collect, process, and interpret geotechnical data. Timely identification of movement trends or tension cracks is essential for maintaining slope stability.
- Findings should be communicated regularly with operations and engineering teams to assist in optimizing slope geometry and validating design assumptions.

9.2 Open Pit Mining

The proposed open pit slopes are sensitive to operational practices. The following considerations are recommended to maintain stable wall conditions:

01. Engineering Practices & Mine Design

- Geomechanical and geological mapping should be completed as benches are exposed to document rock mass quality and structures, lithological contacts, and groundwater conditions.
- Information from mapping should be incorporated into a living geomechanical model to evaluate whether the exposed conditions remain consistent with design assumptions.
- Water inflows observed in drillholes should be recorded (date, depth, and approximate rate) and compiled in a georeferenced database. Surface drillholes above the underground stopes should be cemented or sealed to prevent hydraulic connectivity with underground workings.
- Restrictions and limitations related to slope angle transitions should be considered when adjusting wall geometries. Abrupt changes in angle or wall orientation (e.g., bullnoses or convex wall profiles) may impact rock mass confinement and local stability.

02. Drilling & Blasting

- Blasting through highly fractured upper pit rock may generate significant flyrock; blast design should be adjusted as needed to ensure safety.
- Controlled blasting is required to minimize overbreak, protect bench crests, and reduce the development of local overhangs or key-block instabilities.
- Pre-shear blasting is recommended along final walls to produce smooth and continuous faces, and to minimize rock mass disturbance.

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- → Damage behind the design line should be minimized through optimized charge weights, reduced burden, and appropriate delay timing.
- → Interim slopes may tolerate less stringent controlled blasting, but still require attention to minimize unnecessary overbreak.
- → Phase walls and initial benches should be used to refine blast designs through iterative testing and performance review.
- → Where mining occurs near the underground mine or other surface infrastructure, blasting should be monitored for vibration impacts using geophones or accelerometers.

03. **Pit Wall Stability**

- → Monitoring the evolution of joint sets that may influence the design criteria is essential. The recommended design criteria represent average conditions for the wall, and ongoing verification is therefore required to ensure their validity as mining progresses.
- → Effective scaling must remove loose material from bench faces and crests before initiating drilling on the underlying bench.
- → Scaling should focus on removing hazardous loose material while preserving intact rock and minimizing unnecessary disturbance.
- → Undercutting during mucking operations should be avoided to limit the development of ravelling and shallow-dipping plane/wedge failures.
- → Benches should remain clear and functional, including maintenance of drainage channels and access for inspection. They should always be cleaned before losing access.
- → Sinking cuts should be avoided near final walls to reduce disturbance to the rock mass.

04. **Groundwater**

- → Surface run-off must be diverted away from pit crests using ditches or berms to reduce infiltration.
- → In-pit sumps and pumping capacity should be established early in mine development.
- → Accumulated water on benches increases rockfall risk and freeze–thaw damage; drainage must be maintained.

05. **Slope Monitoring**

- → Continuous monitoring is recommended, including:
  - Visual inspections
  - Prism monitoring
  - Slope stability radar (SSR or LiDAR-based systems)
  - Crack meters or extensometers where tension cracks may develop
- → Geotechnical monitoring data should be reviewed daily, with formal reports issued at a frequency appropriate to the risk level.

06. **Quality Control**

- → Quality control programs and task observations should be implemented for key operational and mine engineering processes, including:

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- Drilling and blasting practices
- Pit wall scaling
- Comparison of design pit walls with as-built geometries

9.3 Underground Mining

Operational considerations for underground mining are presented below. These recommendations focus on excavation stability, sequencing, and interaction with the open pit.

01. Engineering Practices & Mine Design

→ Water inflows encountered during development or drilling should be systematically logged, quantified, and stored in a 3D database shared with mine engineering.

→ Surface exploration drillholes intersecting underground developments should be cemented or sealed to prevent uncontrolled inflows or blast-related risks.

→ Underground excavations beneath or adjacent to the planned open pit design must be sequenced to avoid undermining slopes.

→ Backfill sequencing should support wall and pillar stability and minimize exposure of large unsupported spans.

02. Drilling & Blasting

→ Underground blast designs should limit vibration transmission toward open pit slopes and critical infrastructure.

→ Perimeter blasting can be considered for development headings near weak lithologies, major structures, or critical excavations to maintain stability.

→ Where mining occurs near the pit or other surface infrastructure, blasting should be monitored for vibration impacts using geophones and/or accelerometers.

03. Excavation Stability and Ground Support

→ Ground support selection should be based on rock mass quality, structure, in-situ stress conditions, and expected exposure time; support may include bolts, cables, mesh, and shotcrete.

→ Rehabilitation of older development headings should be considered if they remain within influence zones of new excavations or production areas.

→ Backfilling should generally follow the stoping sequence to minimize the number of unfilled stopes and control stress redistribution.

→ Quality control of ground support is critical and should be conducted at early stages, including task observations and pull-testing of representative support tendons.

04. Groundwater and Water Management

→ A mine-wide dewatering strategy should be implemented, including sumps on multiple levels, staged pumping, and monitoring, based on anticipated inflows.

→ Inflow sources (e.g., faults, fracture zones, and drillholes) should be grouted or sealed as necessary to maintain safe working conditions.

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05. Monitoring and Instrumentation

→ A monitoring system appropriate for underground conditions should be implemented, which may include:
- Convergence monitoring (e.g., convergence stations in high-stress areas – although unlikely)
- MPBX installations in critical infrastructure or large spans

→ Data should be reviewed regularly, with thresholds and trigger action response plans (TARPs) guiding operational responses.

06. Quality Control

→ Quality control programs and task observations should be implemented for key underground activities, including:
- Ground support installation (e.g., bolts, cables, shotcrete)
- Cemented paste backfill production and placement
- Construction of backfill barricades (e.g., shotcrete fences or muck barricades)

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10 Conclusion

The geomechanical assessment completed for the Shaakichiuwaanaan Project provides a comprehensive foundation for FS-level open pit and underground mine design. Through integrated geological, structural, laboratory, and various analyses, the study defines geomechanical domains, characterizes rock mass behaviour, and establishes design guidelines for slopes, stopes, backfill, and ground support.

For the **open pit**, a risk-based bench design methodology was applied, resulting in sector-specific bench face angles, bench widths, and inter-ramp angles that achieve acceptable levels of probability of failure and spill containment. Three-dimensional numerical modelling confirms that, with the recommended geometries, the overall pit slopes meet the project stability criteria under static conditions. Localized deformation zones – primarily associated with the ultramafic unit and sharp wall transitions—do not pose significant stability concerns but should be monitored during operations.

Underground mine design was supported by stope stability assessments using empirical methods, leading to standardized stope dimensions that balance dilution control and operational practicality. Backfill strength requirements were defined using empirical equations and verified using numerical modelling, with conservative factors of safety applied early in the mine life. Numerical analysis of the mining sequence indicates that the underground development and stoping strategy can be implemented safely, with only localized areas of damage expected near active stopes.

The **interaction between the open pit and underground** workings was evaluated using two-dimensional limit equilibrium analysis and 3D numerical modelling, confirming that a minimum cemented paste backfill strength of 2.0 MPa and a 100 m extent of strengthened backfill behind the pit wall are required to maintain slope stability in pastefilled stopes.

A preliminary assessment of the surface **crown pillar** indicates that an 85 m pillar may satisfy stability requirements; however, this thickness does not meet the minimum regulatory threshold established under Québec's RSSTM (Article 77). Additional geotechnical, hydrogeological, and geomechanical investigations will therefore be required in future project phases to develop a fully compliant pillar design.

With respect to the **main portal and boxcut**, the analyses indicate that a focused site characterization program will be essential during the next project stage. This investigation should specifically target the depth and composition of overburden materials in the portal area, including along the planned ramp alignment, to refine slope design parameters and ground support requirements.

Overall, the analyses demonstrate that the proposed open pit and underground mine designs are technically feasible and meet their respective safety criteria, provided that the recommendations presented in this study are implemented and supported by ongoing monitoring, refinement of the geomechanical and structural model, and continued investigation as the Project advances toward execution.

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11 Recommendations and Limitations

11.1 Feasibility-Level Design Recommendations

Design guidelines to support the Feasibility Study were provided across several key areas. A summary is provided below.

01. **Open Pit – Design Guidelines**

Apply the guidelines presented in Section 7.7, summarized as follows:

- Inter-ramp angles of 36° are recommended in soils.
- Rock slope design assumes double benching (20 m bench height).
- Resulting inter-ramp angles vary between 32 and 55°, depending on the geomechanical domain:
  - Bench face angles between 40 and 75°
  - Bench widths between 8.5 and 12 m to retain more than 70% of potential spills in general
- Geotechnical berms (16 m wide) are recommended for walls exceeding 160 m in height, starting 80 m below the surface and ending 80 m above the pit bottom.

02. **Underground – Stope Design and Dilution Estimates**

Apply the guidelines presented in Section 8.3.4, summarized as follows:

- Adopt the recommended stope dimensions:
  - Transverse stopes 17 mL × 25 mW × 30–36 mH
  - Longitudinal stopes 17 mL × 7 mW × 30–36 mH
- Stope backs should be cablebolted to limit instability and dilution.
- Apply the following dilution (ELOS) values for transverse and longitudinal stope surfaces:
  - Hanging wall 0.75 m
  - Footwall 1.25 m / 0.50 m
  - Back None (cablebolted)
  - Endwalls None (ore)
  - Backfill 0.50 m

03. **Underground – Mine Sequence**

Apply the guidelines presented in Section 8.4, summarized as follows:

- Minimize early development and adopt a "just-in-time" approach where possible.
- Prioritize mining in already-developed drawpoints before initiating new ones to reduce rehabilitation needs.
- For transverse stope panels, adopt a primary–secondary sequence that maintains a pyramidal extraction shape (refer to Figure 46).
- For narrower stope panels, adopt a longitudinal retreat sequence and avoid converging fronts or diminishing pillars.

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04. **Underground – Backfill**

Apply the design guidelines presented in Section 8.5.3, summarized as follows:

- → Residual CPB strengths between 210–540 kPa are generally required, depending on stope type and sequencing.
- → A higher factor of safety (FoS = 2.0) is recommended early in mine life, reducing to 1.5 as experience is gained.
- → Underhand stoping requires more conservative CPB strengths (1.2–1.6 MPa) and should be minimized when possible.

05. **Underground – Crown Pillar**

Apply the design guidelines presented in Section 8.6.3, summarized as follows:

- → Use a minimum crown pillar thickness of 85 m.
- → Treat this thickness as **preliminary** pending further studies.
- → Apply enhanced monitoring (instrumentation and TARP) if mining proceeds beneath a pillar less than 100 m thick under a water body.

06. **Underground – Ground Support Design**

Apply the design guidelines presented in Section 8.7.2, summarized as follows:

- → Proposed support types are adequate for FS-level budgeting.
- → Typical lateral development (<7.5 m span) requires 2.4 m rebar (20 mm) on a 1.2 × 1.2 m pattern, with 1.5 m split-sets in the secondary rows, supplemented by 6-gauge, galvanized welded-wire mesh (4 inch aperture).
- → Typical intersections require 6 m cablebolts on a 2 × 2 m pattern.
- → Stope backs and brows require cablebolting (6 m, 2 × 2 m pattern).
- → Ground support requirements for the portal and some infrastructure excavations remain to be defined.

07. **Underground – Portal (Box Cut)**

Apply the design guidelines presented in Section 8.8.1, summarized as follows:

- → Overburden slopes of 2H:1V are recommended.
- → North wall boxcut slope: 50°.
- → South wall and portal face boxcut slopes: 80°.
- → A surface pillar thickness of 10 m is appropriate at this stage.

08. **Underground – Infrastructures**

Apply the design guidelines presented in Section 8.9, summarized as follows:

- → Maintain ramp location at least 100 m from stopes.
- → Plan haulage drifts at 20 m from stopes.
- → Locate raises at a minimum of 40 m from stopes.

09. **Open Pit–Underground Interaction**

Apply the design guidelines presented in Section 8.10, summarized as follows:

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- → Mine stopes before excavating the pit wall in their vicinity to avoid ore loss.
- → Use CPB with a compressive strength of 2.0 MPa for all stopes located within 100 m of the open pit walls.

11.2 Design Guideline Limitations

Based on the completed analyses and the current level of available information, the following limitations should be recognized when applying the design guidelines.

01. Litho-Structural Model

- → Lithological contacts were modelled based on current interpretations and drilling density; revisions should be expected as mining advances.
- → The location and continuity of major structures (e.g., FLT_1525, diabase dyke) remain uncertain in areas with sparse drilling, particularly near the portal box cut.
- → The structural model covers only the open pit area; structural conditions in the underground mine have not been modelled.
- → The UMU unit is comparatively under-characterized, reducing confidence in design parameters where this lithology is exposed.
- → Foliation orientation varies along the pit and may differ locally from the assumed averaged sets. Although this was accounted for by defining multiple geomechanical domains, real variability may be greater and should be confirmed as exposures develop.

02. Geotechnical Characterization

- → No laboratory testing of soils was performed; strength envelopes for overburden rely solely on SPT results and empirical correlations.
- → Limited drilling exists around the open pit, especially near the portal and pit boundaries, increasing uncertainty in soil thickness and properties.

03. Geomechanical Characterization

- → Joint persistence values are assumed to be 15 m until validated by field measurements (e.g., pit wall mapping, LiDAR).
- → Strength parameters for foliation planes remain uncertain.
- → Joint sets observed on the south wall are considered less restrictive with respect to the design criteria than those on the north wall, due to their wider spacing.
- → Excavations were assumed dry or with minor inflow; groundwater pressures were not explicitly incorporated in classification or modelling.

04. Open Pit Design

- → Rock mass disturbance factors (blast damage, stress relief) were not incorporated into the stability envelopes.
- → Achieving the recommended bench widths requires controlled drilling and blasting; excessive backbreak may reduce catch bench capacity.

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- → Large-scale structures were not included in the numerical models; potential instabilities associated with these features are therefore not captured.
- → Hydrogeological conditions were assumed to have a limited impact based on current data; pore pressures were not included in the numerical analyses.
- → Guidelines apply only to the geometries and lithologies evaluated; deviations may require reassessment.

05. **Underground Mine Design**

- → Large-scale structures were not included in the continuum numerical models, limiting assessment of structurally controlled failure mechanisms.
- → In-situ stress assumptions introduce uncertainty; variations in the major principal stress orientation or magnitude may affect stope stability and crown pillar performance.
- → Hydrogeological effects were not explicitly modelled; water pressures may affect excavation stability.
- → Seismic hazard was not assessed due to the relatively shallow mine depth, but localized stress concentrations may still develop near stopes or in pillars.
- → The portal area lacks geotechnical drilling and requires additional investigation before final design and execution.
- → Stability of large underground infrastructure was not evaluated.
- → The current 85 m crown pillar thickness does not meet the regulatory requirements of Québec's RSSTM Article 77 and must be re-evaluated through the required engineering studies before mining (or development) beneath the water body can commence.

06. **Operational Variability**

- → Slope performance is highly dependent on drilling accuracy, blasting quality, and scaling practices. Good practices were assumed throughout the assessments.
- → Excessive crest loss or overbreak may reduce effective catch bench widths below design assumptions.
- → Operational deviations may reduce stability and require design modifications.

11.3 Recommendations for Future Work

Based on the analyses completed in this study, the following work is recommended to advance the Project toward detailed engineering and execution.

01. **Litho-Structural Model**

- → Improve the definition of lithological contacts, particularly those with unfavourable orientations.
- → Acquire additional structural data (surface mapping, oriented core) in key pit wall sectors.
- → Update the structural model to better define large-scale structures, especially in areas with limited drill coverage (e.g., underground mine footprint).

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→ Conduct a focused site characterization program for the portal area. This investigation should confirm overburden depth near the box cut and along the ramp axis, as well as the soil and rock slopes to be used in the design.

02. **Geotechnical Characterization**

→ Develop a site-specific geotechnical model for overburden in the portal and pit-periphery areas.

→ Plan a geotechnical investigation campaign to increase data density and improve the understanding of soil stratigraphy around the open pit.

→ Perform laboratory testing on representative soils to establish appropriate shear strength envelopes.

03. **Geomechanical Characterization**

→ Prioritize data collection for the UMU unit, which currently has the greatest uncertainty.

→ Expand laboratory testwork (intact strength, shear strength) for UMU, pegmatite, altered zones and foliation planes.

04. **Open Pit Design**

→ Update the 3D numerical models as litho-structural interpretations are refined.

→ Reassess structural persistence assumptions (currently 15 m) once exposed pit walls allow direct measurement (e.g., LiDAR scanning).

→ For Sector F-2, evaluate the observed faults and open joints against the updated structural model to confirm their potential impact on design parameters.

→ As mining exposes south walls, further assess the actual impact of the north-dipping joint sets on bench-scale wall integrity, and review the assumption that these joints do not significantly influence the expected IRA.

05. **Underground Mine Design**

→ Update the 3D numerical models as new litho-structural information becomes available.

→ Incorporate major structures into the underground stability assessments once their geometries are confirmed.

→ The current 85 m crown pillar thickness does **not** comply with Québec RSSTM Article 77 for mining beneath a water body. Additional engineering studies (hydrogeology, soil mechanics, geomechanics) are legally required before approval. These studies must be submitted to the CNESST before any excavation under the influence of the water body.

→ For the portal design, drill at least three (3) additional drillholes (approx. 90 m total) to confirm the rock mass conditions at the portal location.

→ Consider performing in-situ stress measurements (e.g., Doorstopper testing) early in development to validate the far-field stress tensor.

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06. Operational Monitoring and Controls

- → Develop a site-specific Ground Control Management Plan (GCMP) that formalizes geotechnical monitoring requirements, data review frequency, and decision-making protocols.
- → Develop Trigger Action Response Plans (TARPs) for both open pit and underground operations, defining action levels and required responses for changes in displacement, pore pressure, or observed ground conditions.
- → Update the GCMP and TARPs according to operational observations and monitoring results.
- → Implement real-time monitoring systems (prisms, radar, LiDAR) in sectors influenced by major structures or sensitive pit geometries.
- → Implement a hydrogeological monitoring program to track groundwater levels and pore pressures in both the pit and underground mines:
  - Install vibrating-wire piezometers (VWP) in key locations, including pit walls, the crown pillar, and areas where groundwater inflow could influence stability.
  - Establish automated data logging and alarm thresholds to support early detection of abnormal pore-pressure trends.

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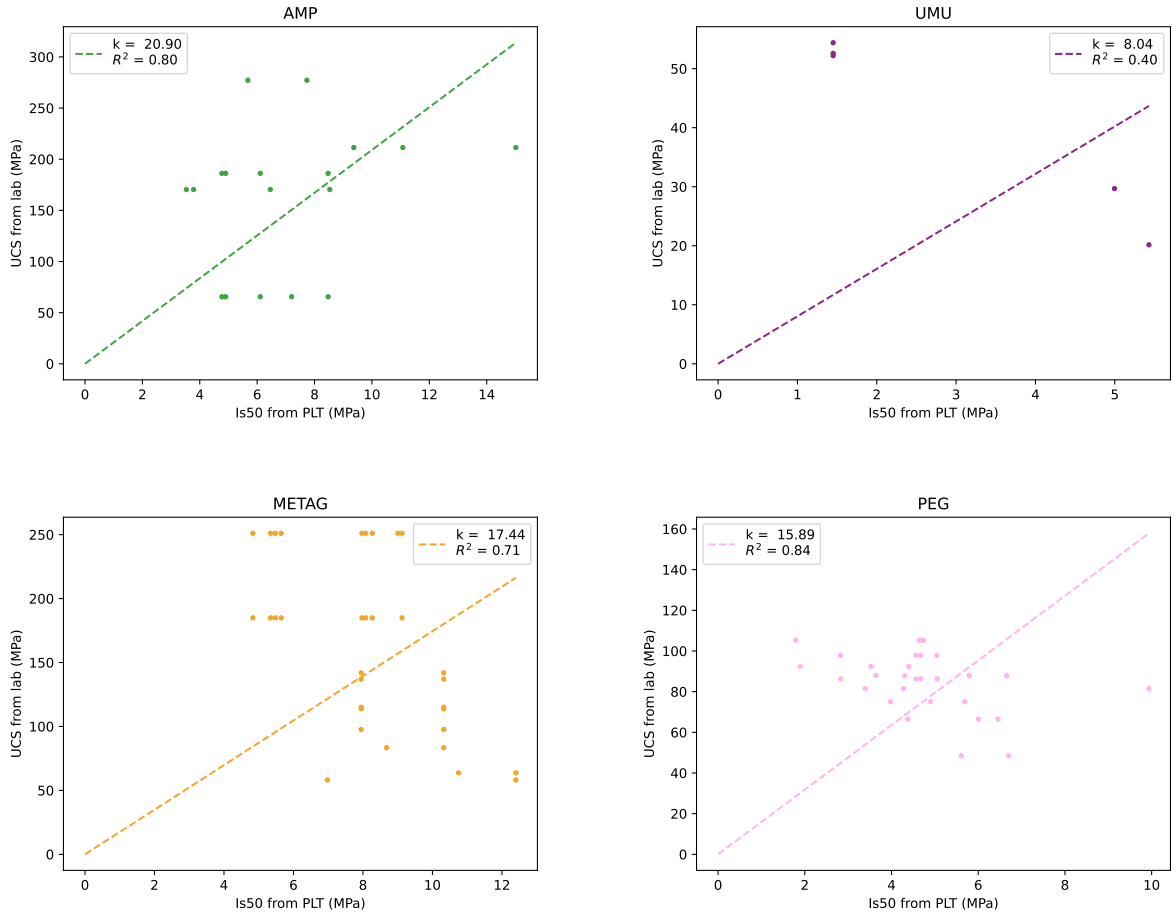
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A PLT to UCS conversion factors

The data used to determine the UCS-to-PLT conversion factors (k) are presented in the figures below, along with the resulting linear regression parameters, where k corresponds to the slope.



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B Stereographic Projections

Figures 64 to 77 present the stereographic projections (lower hemisphere) of the poles of the surveyed discontinuities and the interpreted joint sets for each of the geomechanical domains within fresh rock of the open pit. Note that the Terzaghi weighting correction is applied in all the figures.

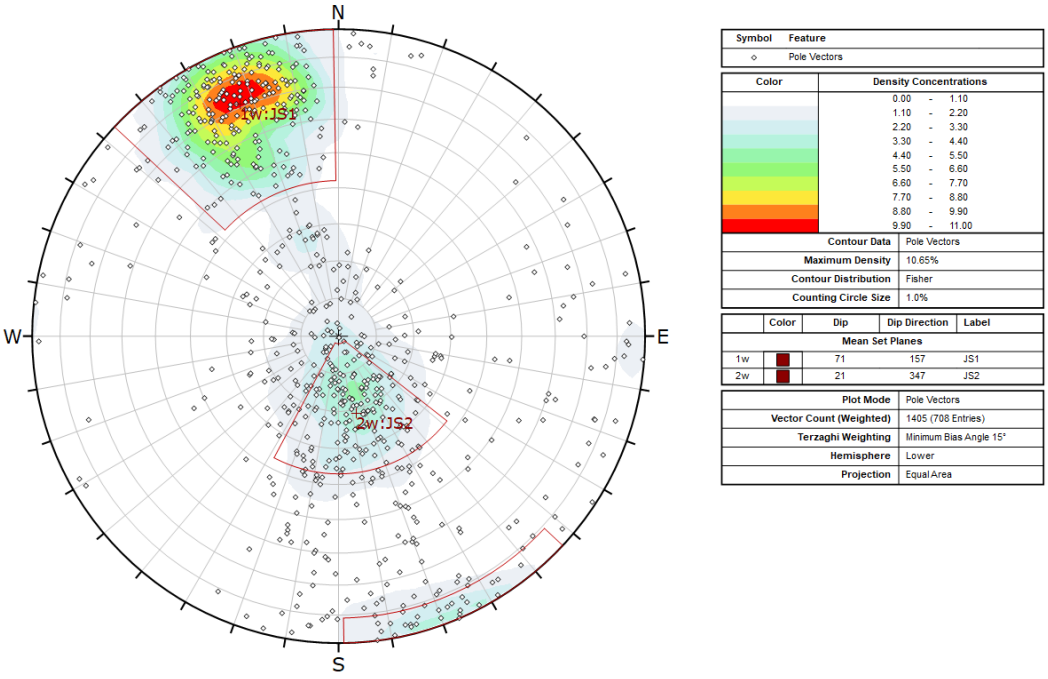
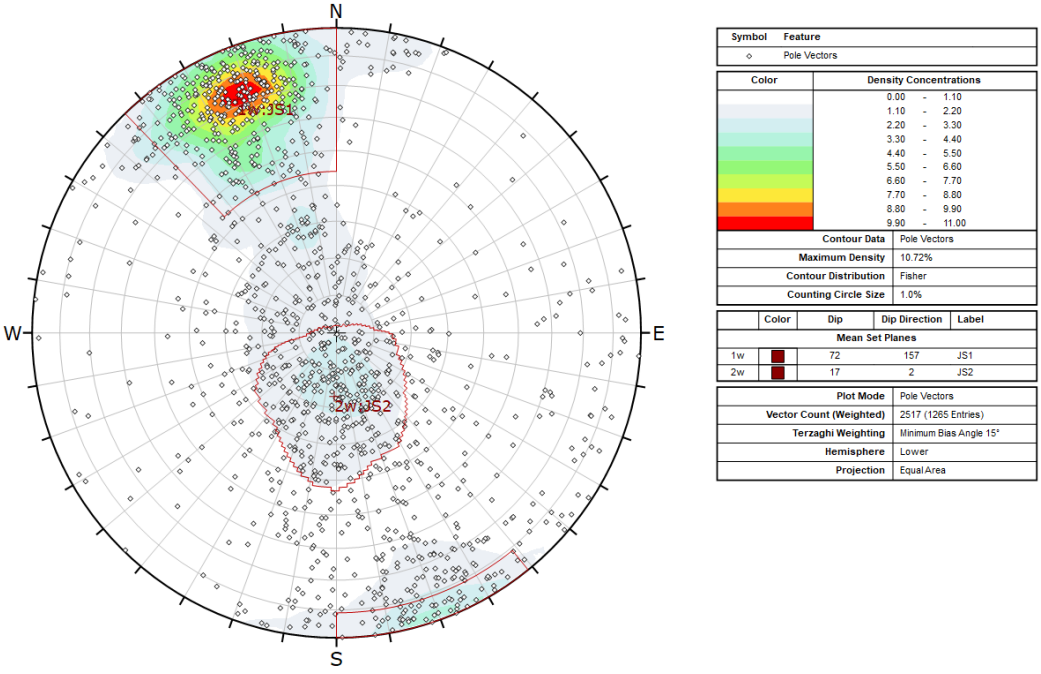


Figure 64. Stereographic projections of GMD WAll-AMP discontinuities in amphibolite and interpreted joint sets for the west area (north wall and south wall).

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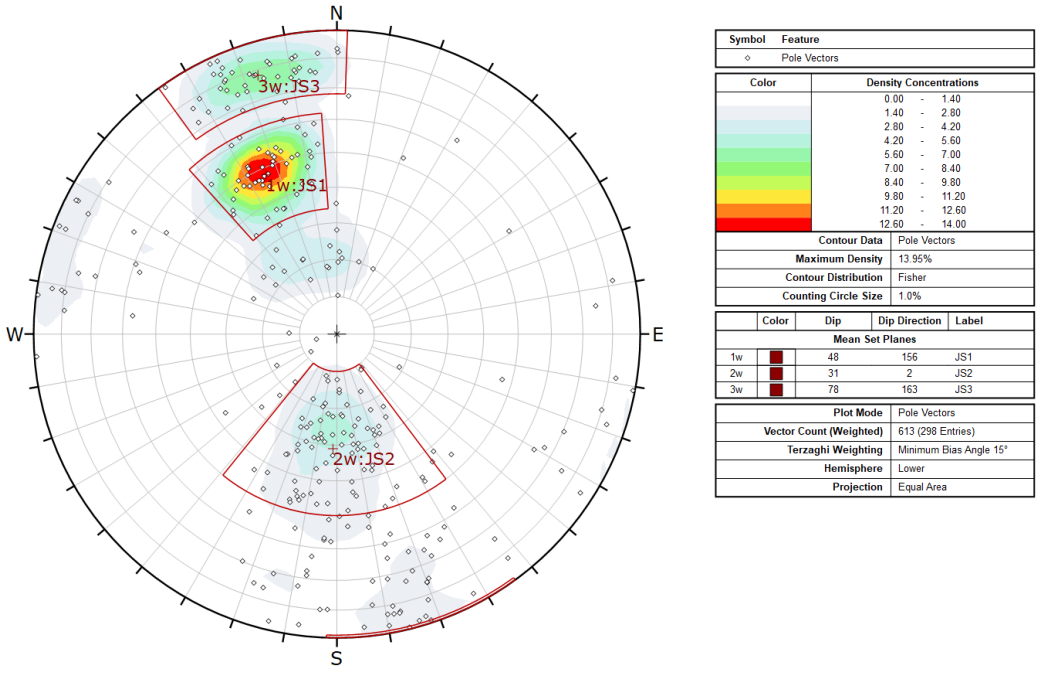
Symbol	Feature
◇	Pole Vectors

Color	Density Concentrations
	0.00 - 1.10
	1.10 - 2.20
	2.20 - 3.30
	3.30 - 4.40
	4.40 - 5.50
	5.50 - 6.60
	6.60 - 7.70
	7.70 - 8.80
	8.80 - 9.90
	9.90 - 11.00
Contour Data	Pole Vectors
Maximum Density	10.72%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		72	157	JS1
2w		17	2	JS2

Plot Mode	Pole Vectors
Vector Count (Weighted)	2517 (1265 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Area

Figure 65. Stereographic projections of GMD WW-APU discontinuities in all lithologies for the west area (west wall).



Symbol	Feature
◇	Pole Vectors

Color	Density Concentrations
	0.00 - 1.40
	1.40 - 2.80
	2.80 - 4.20
	4.20 - 5.60
	5.60 - 7.00
	7.00 - 8.40
	8.40 - 9.80
	9.80 - 11.20
	11.20 - 12.60
	12.60 - 14.00
Contour Data	Pole Vectors
Maximum Density	13.95%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		48	156	JS1
2w		31	2	JS2
3w		78	163	JS3

Plot Mode	Pole Vectors
Vector Count (Weighted)	613 (298 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Area

Figure 66. Stereographic projections of GMD WS-META discontinuities in metasediments and interpreted joint sets for the west area (south wall).

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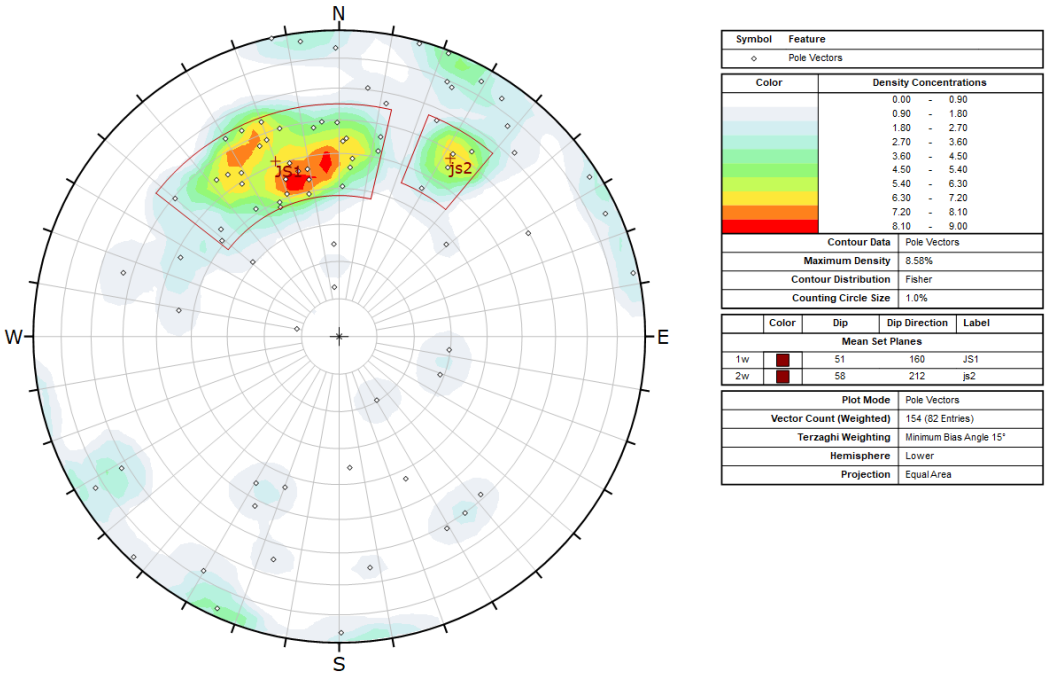


Figure 67. Stereographic projections of GMD CN-UMU discontinuities in ultramafic rocks and interpreted joint sets for the central area (north wall).

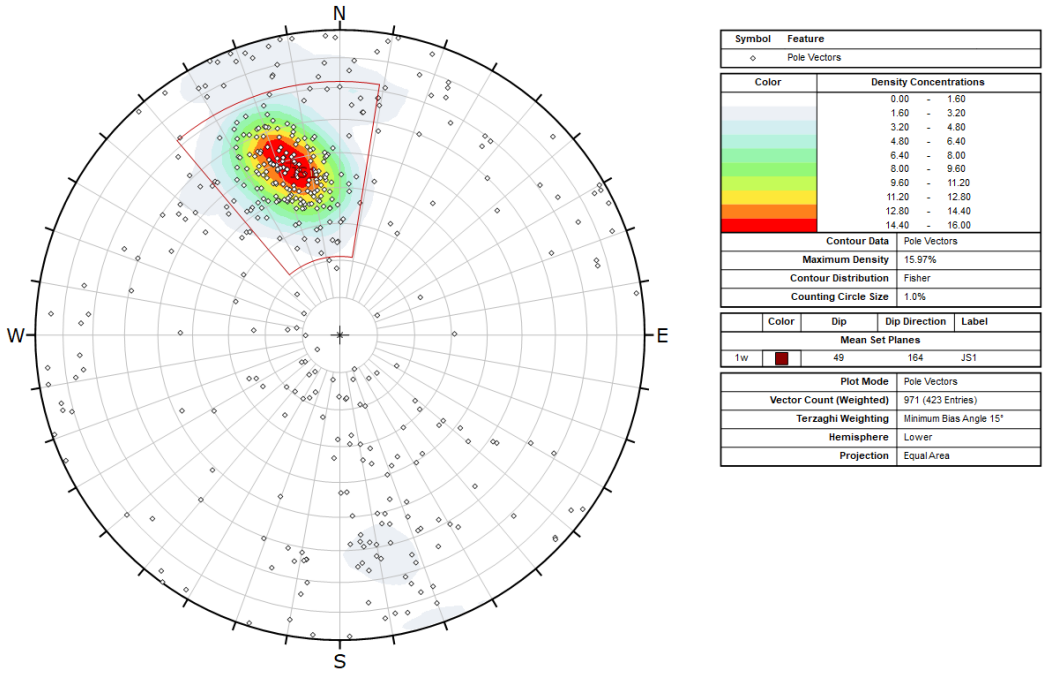


Figure 68. Stereographic projections of GMD CN-META discontinuities in metasediments and interpreted joint sets for the central area (north wall).

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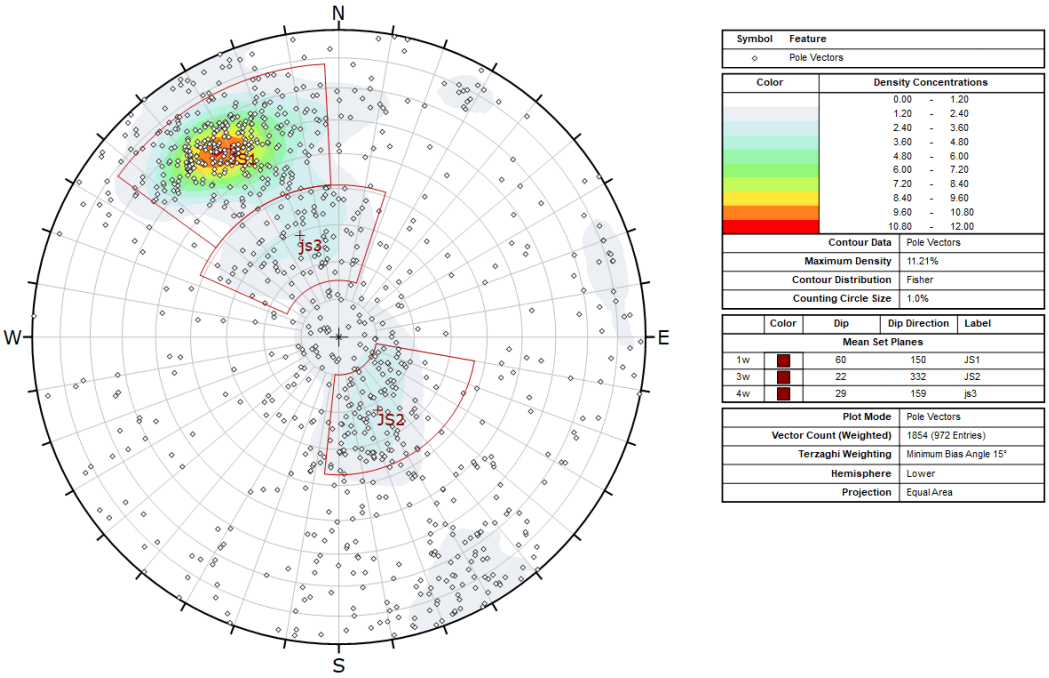


Figure 69. Stereographic projections of GMD CN-AMP discontinuities in amphibolite and interpreted joint sets for the central area (north wall).

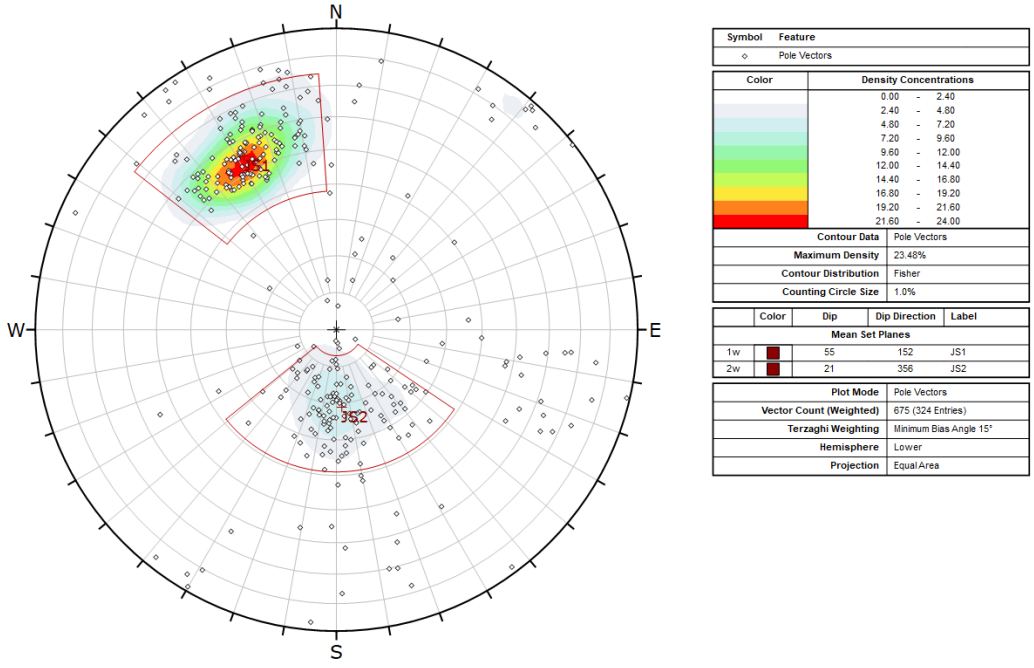
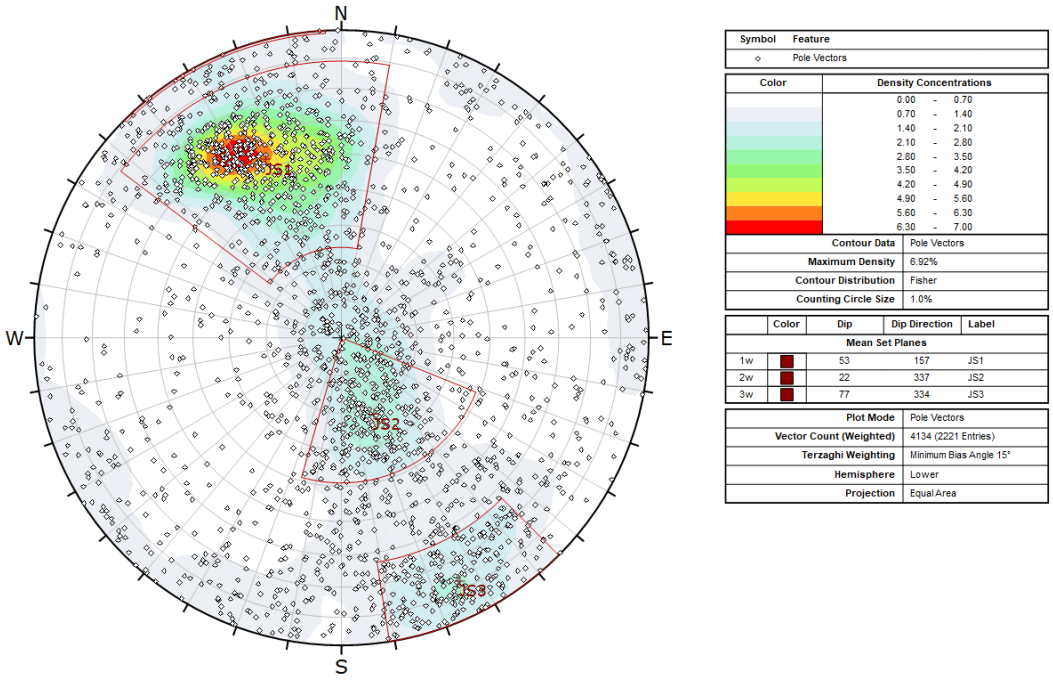


Figure 70. Stereographic projections of GMD CS-META discontinuities in metasediments and interpreted joint sets for the central area (south wall).

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Symbol	Feature
◇	Pole Vectors

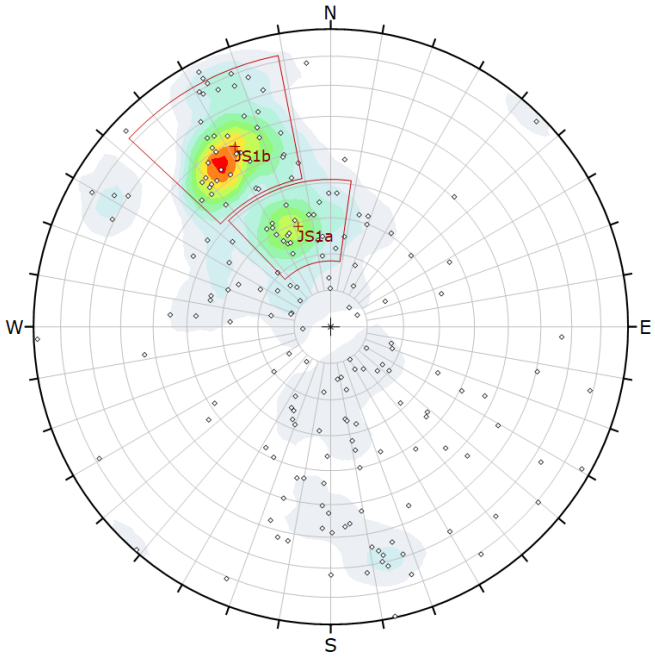
Color	Density Concentrations
	0.00 - 0.70
	0.70 - 1.40
	1.40 - 2.10
	2.10 - 2.80
	2.80 - 3.50
	3.50 - 4.20
	4.20 - 4.90
	4.90 - 5.60
	5.60 - 6.30
	6.30 - 7.00

Contour Data	Pole Vectors
Maximum Density	6.92%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		53	157	JS1
2w		22	337	JS2
3w		77	334	JS3

Plot Mode	Pole Vectors
Vector Count (Weighted)	4134 (2221 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Area

Figure 71. Stereographic projections of GMD CS-AMP discontinuities in amphibolite and interpreted joint sets for the central area (south wall).



Symbol	Feature
◇	Pole Vectors

Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00

Contour Data	Pole Vectors
Maximum Density	11.40%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		29	162	JS1a
2w		58	152	JS1b

Plot Mode	Pole Vectors
Vector Count (Weighted)	383 (191 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Area

Figure 72. Stereographic projections of GMD EN-UMU discontinuities in ultramafic rocks and interpreted joint sets for the east area (north wall).

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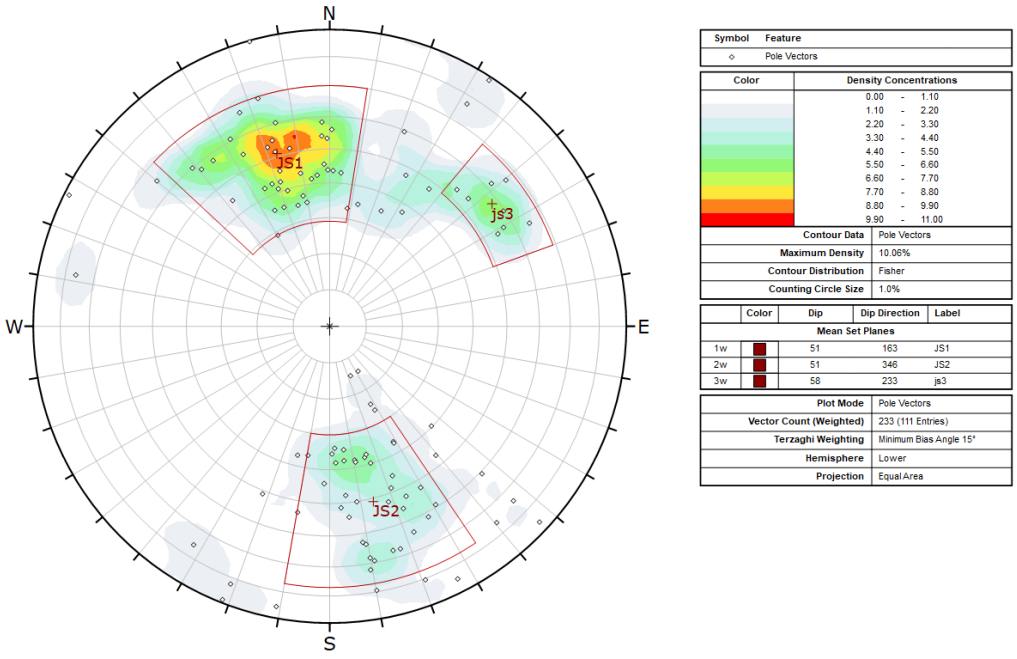


Figure 73. Stereographic projections of GMD EN-META discontinuities in metasediments and interpreted joint sets for the east area (north wall).

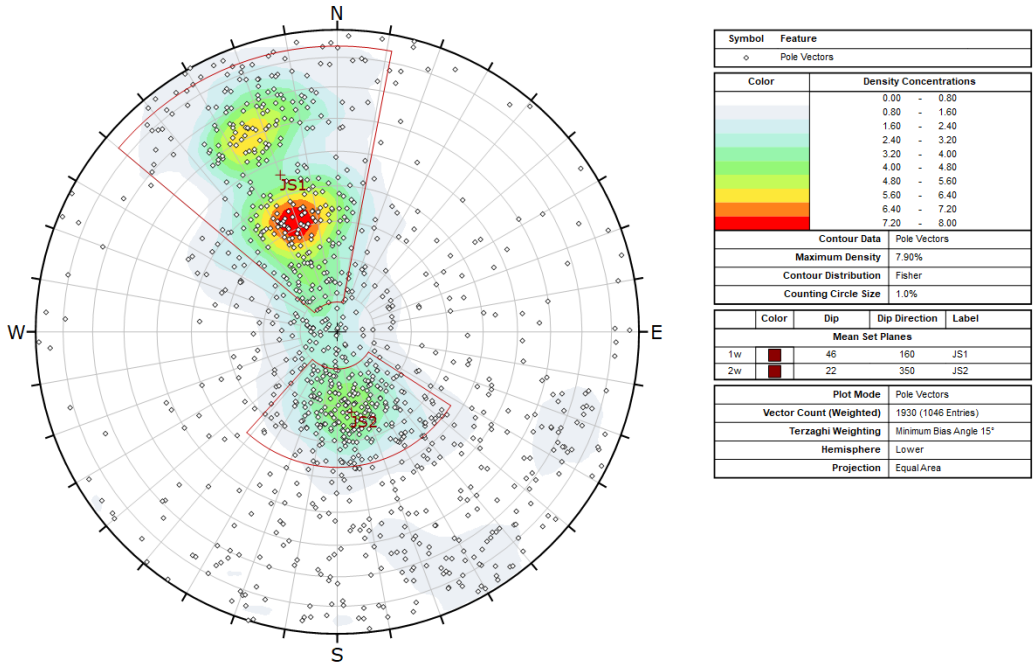


Figure 74. Stereographic projections of GMD EE-AMPU discontinuities in all lithologies for the east area (east wall).

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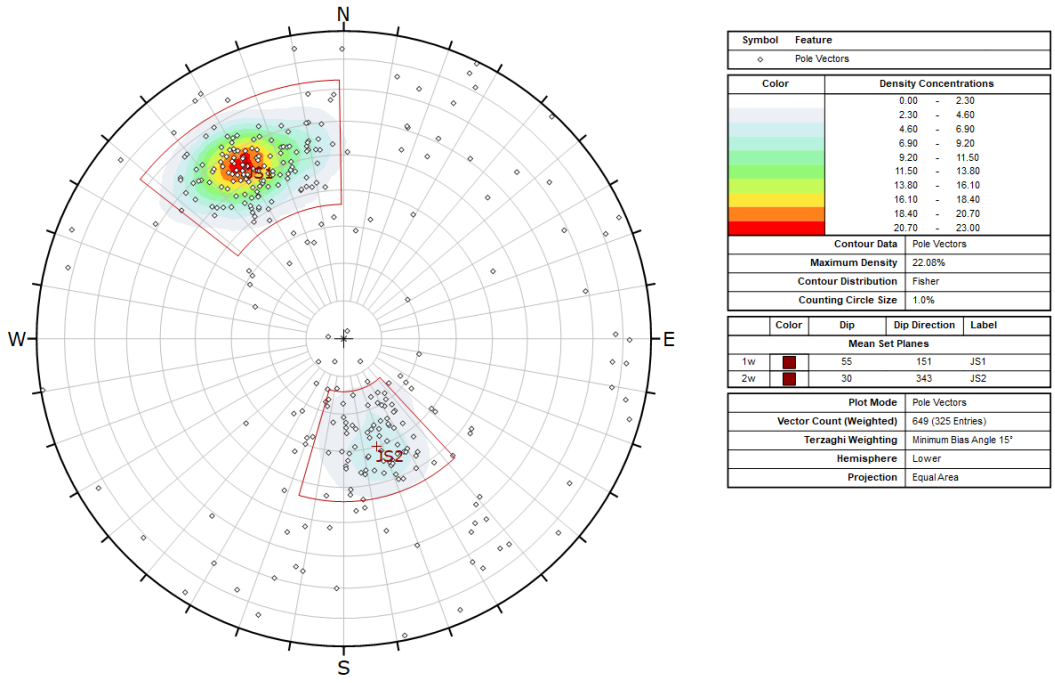


Figure 75. Stereographic projections of GMD ES-META discontinuities in metasediments and interpreted joint sets for the east area (south wall).

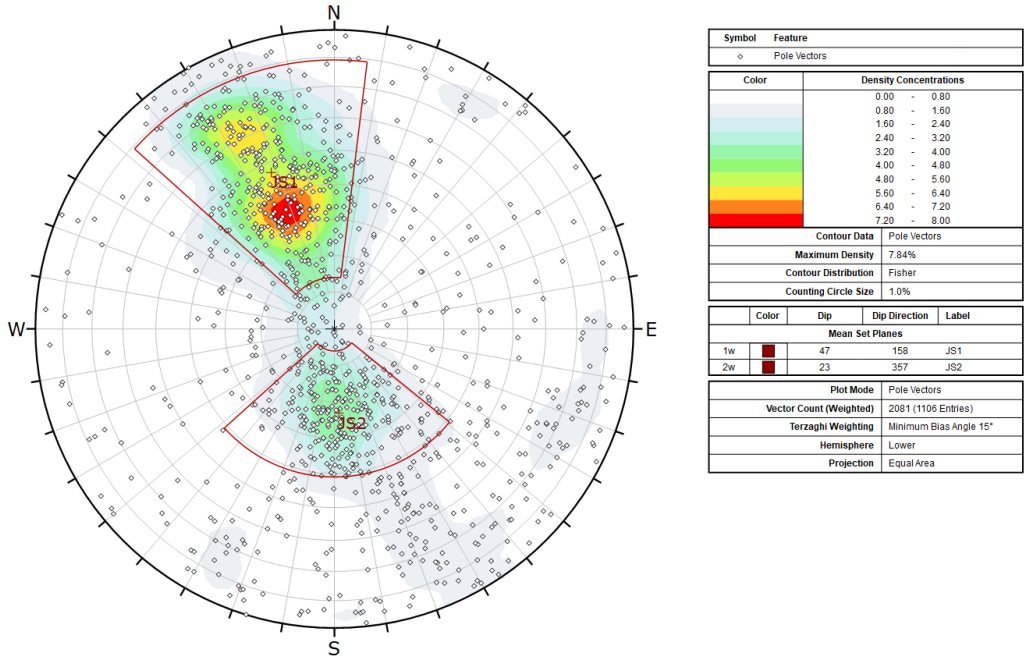


Figure 76. Stereographic projections of GMD EAll-AMP discontinuities in amphibolite and interpreted joint sets for the east area (north wall and south wall).

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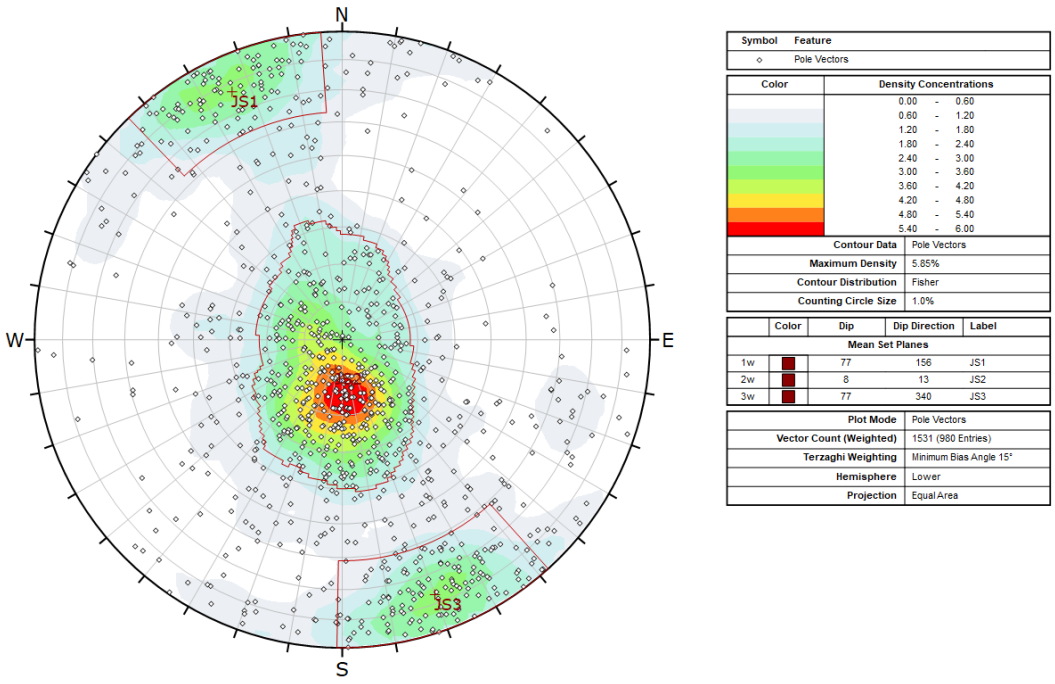
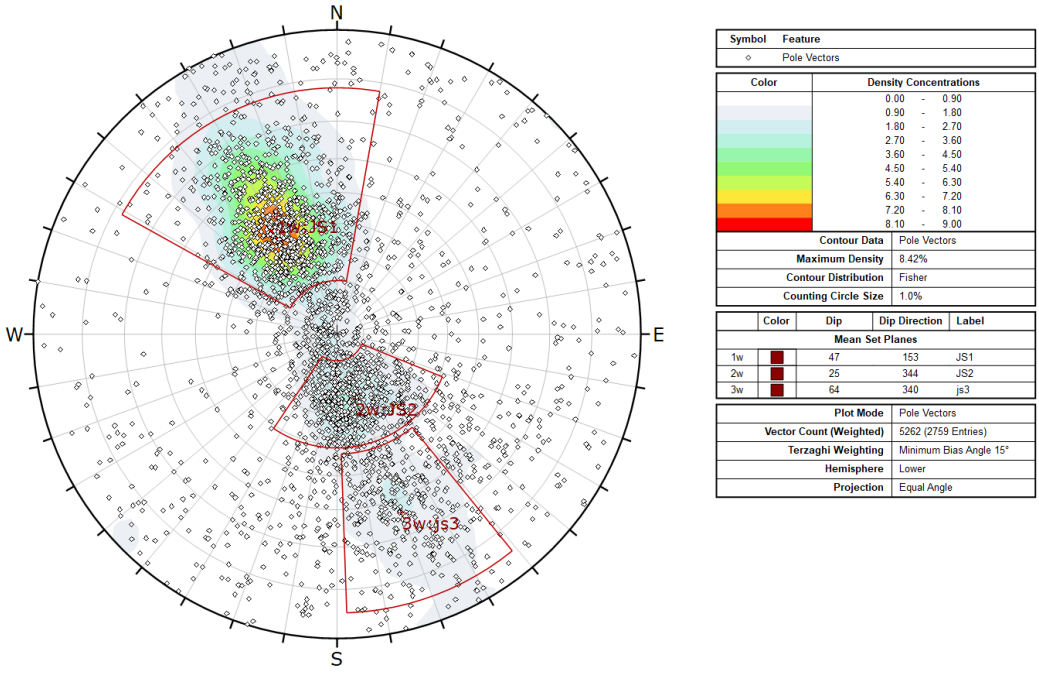


Figure 77. Stereographic projections of GMD WCE-PEG discontinuities in pegmatite and interpreted joint sets for the west, central and east areas.

Figures 79 to 81 show the stereographic projections of the surveyed discontinuities and interpreted joint sets. Note that the Terzaghi weighting correction is applied in all the figures.

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Symbol	Feature
◇	Pole Vectors

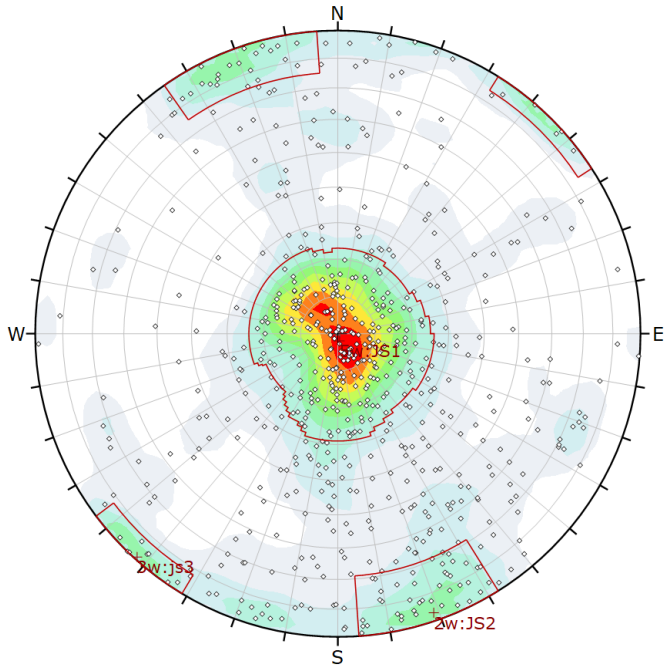
Color	Density Concentrations
	0.00 - 0.90
	0.90 - 1.80
	1.80 - 2.70
	2.70 - 3.60
	3.60 - 4.50
	4.50 - 5.40
	5.40 - 6.30
	6.30 - 7.20
	7.20 - 8.10
	8.10 - 9.00

Contour Data	Pole Vectors
Maximum Density	8.42%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		47	153	JS1
2w		25	344	JS2
3w		64	340	js3

Plot Mode	Pole Vectors
Vector Count (Weighted)	5262 (2759 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Angle

Figure 78. Stereographic projections of discontinuities in the amphibolite in the underground mine area and interpreted joint sets.



Symbol	Feature
◇	Pole Vectors

Color	Density Concentrations
	0.00 - 0.60
	0.60 - 1.20
	1.20 - 1.80
	1.80 - 2.40
	2.40 - 3.00
	3.00 - 3.60
	3.60 - 4.20
	4.20 - 4.80
	4.80 - 5.40
	5.40 - 6.00

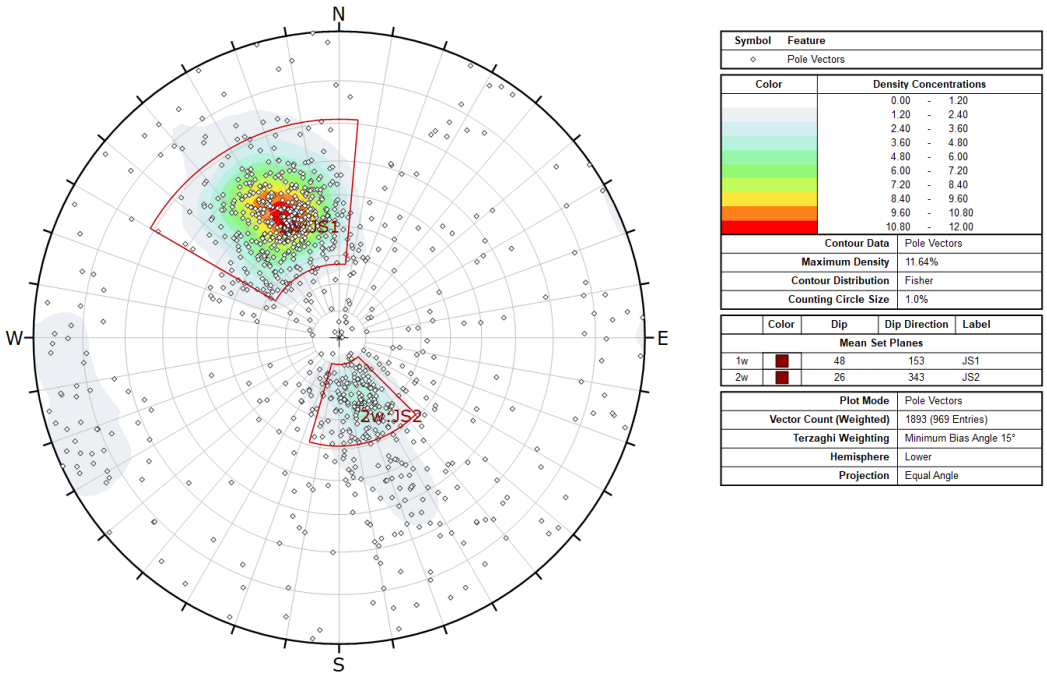
Contour Data	Pole Vectors
Maximum Density	5.85%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
2w		87	341	JS2
3w		89	42	js3
5w		2	346	JS1

Plot Mode	Pole Vectors
Vector Count (Weighted)	1014 (656 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Area

Figure 79. Stereographic projections of discontinuities in the pegmatite in the underground mine area and interpreted joint sets.

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Symbol	Feature
◇	Pole Vectors

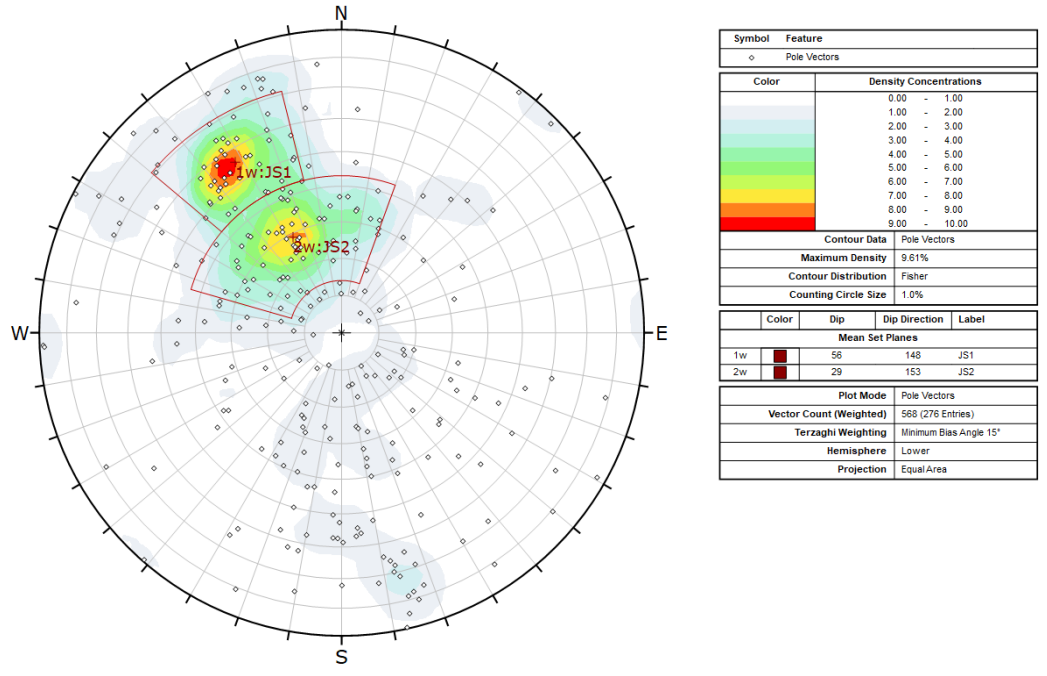
Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00

Contour Data	Pole Vectors
Maximum Density	11.64%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		48	153	JS1
2w		26	343	JS2

Plot Mode	Pole Vectors
Vector Count (Weighted)	1893 (969 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Angle

Figure 80. Stereographic projections of discontinuities in the metasediments in the underground mine area and interpreted joint sets.



Symbol	Feature
◇	Pole Vectors

Color	Density Concentrations
	0.00 - 1.00
	1.00 - 2.00
	2.00 - 3.00
	3.00 - 4.00
	4.00 - 5.00
	5.00 - 6.00
	6.00 - 7.00
	7.00 - 8.00
	8.00 - 9.00
	9.00 - 10.00

Contour Data	Pole Vectors
Maximum Density	9.61%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1w		56	148	JS1
2w		29	153	JS2

Plot Mode	Pole Vectors
Vector Count (Weighted)	568 (276 Entries)
Terzaghi Weighting	Minimum Bias Angle 15°
Hemisphere	Lower
Projection	Equal Area

Figure 81. Stereographic projections of discontinuities in the ultramafic in the underground mine area and interpreted joint sets.

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C Open Pit Detailed Structural characterization

This appendix presents the stereographic projection (lower hemisphere) of the poles of the surveyed discontinuities per lithology for each one of the eight (8) zones defined along the pit length for characterization purposes. Note that some zones do not have a stereonet due to the lack of data collected in those areas or the absence of the lithological unit in the zone.

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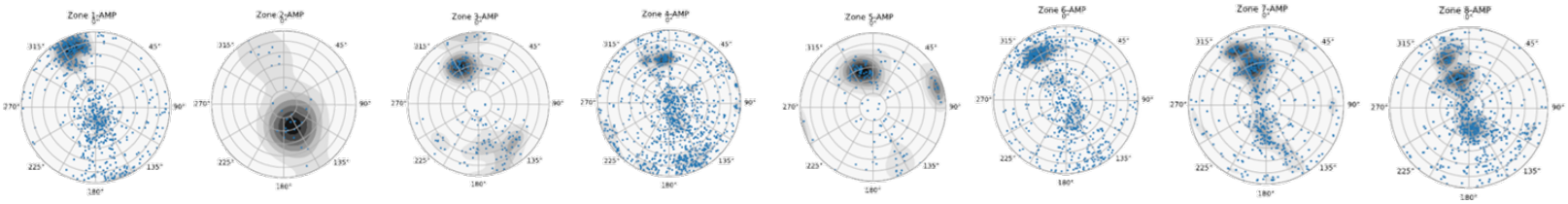


Figure 82. Stereonet for Zones 1-8 of the open pit - Amphibolite

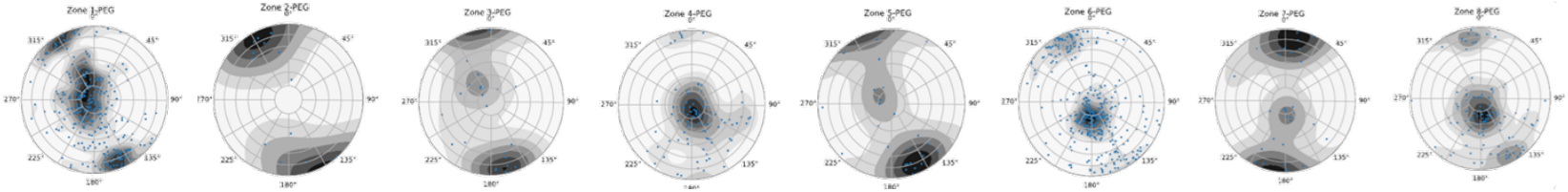


Figure 83. Stereonet for Zones 1-8 of the open pit - Pegmatite

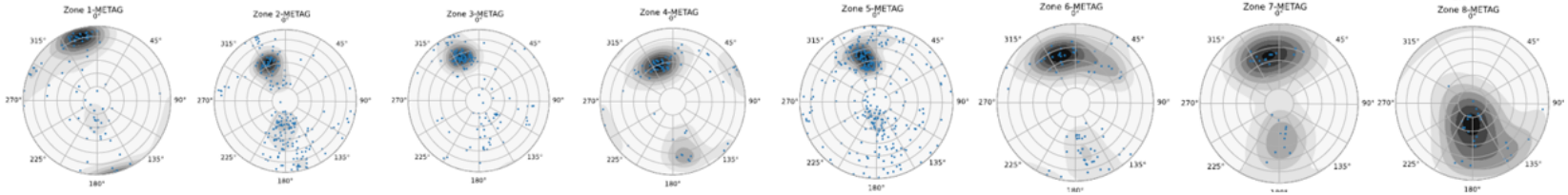


Figure 84. Stereonet for Zones 1-8 of the open pit - Metasediments (Guyer)

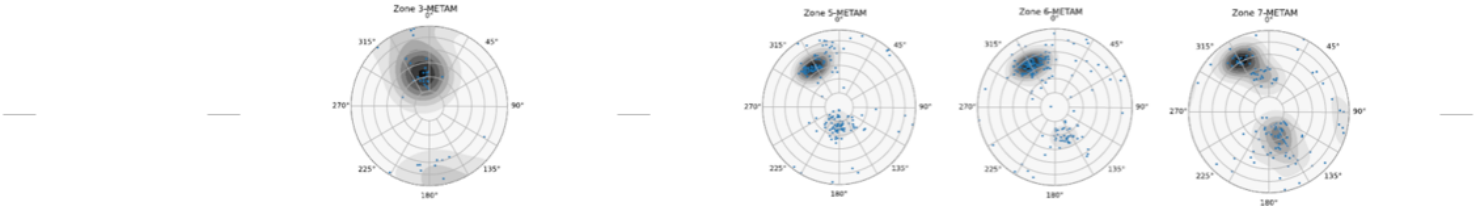


Figure 85. Stereonet for Zones 1-8 of the open pit - Metasediments (Marbot)

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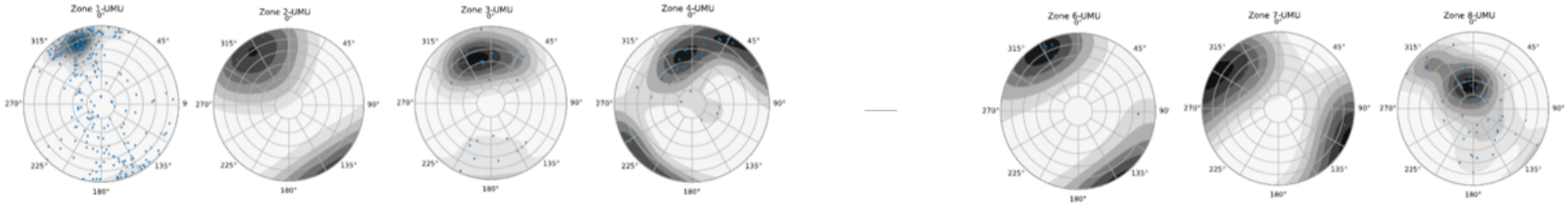


Figure 86. Stereonet for Zones 1-8 of the open pit - Ultramafic

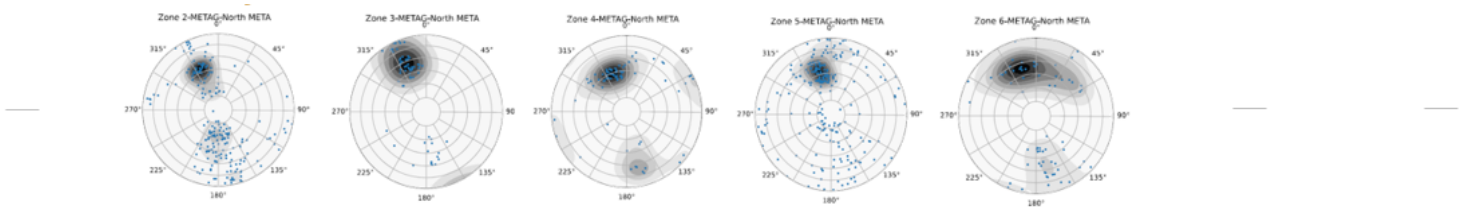


Figure 87. Stereonet for Zones 1-8 of the open pit - Metasediments (Guyer) - North

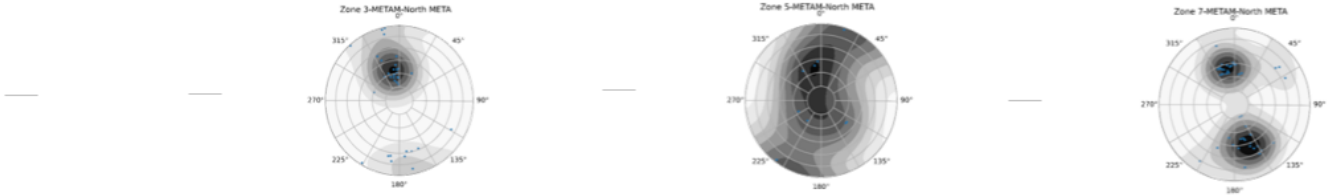


Figure 88. Stereonet for Zones 1-8 of the open pit - Metasediments (Marbot) - North

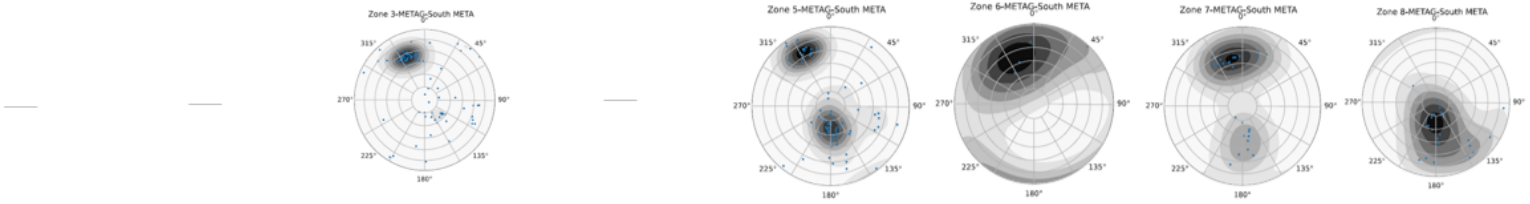


Figure 89. Stereonet for Zones 1-8 of the open pit - Metasediments (Guyer) - South

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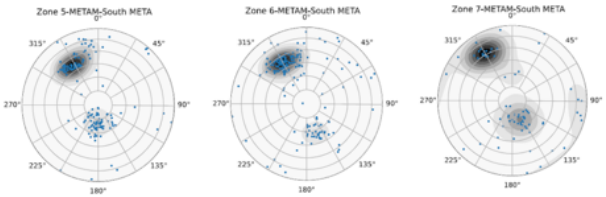


Figure 90. Stereonet for Zones 1-8 of the open pit - Metasediments (Marbot) - South

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D Open Pit Guidelines – 2025 Update

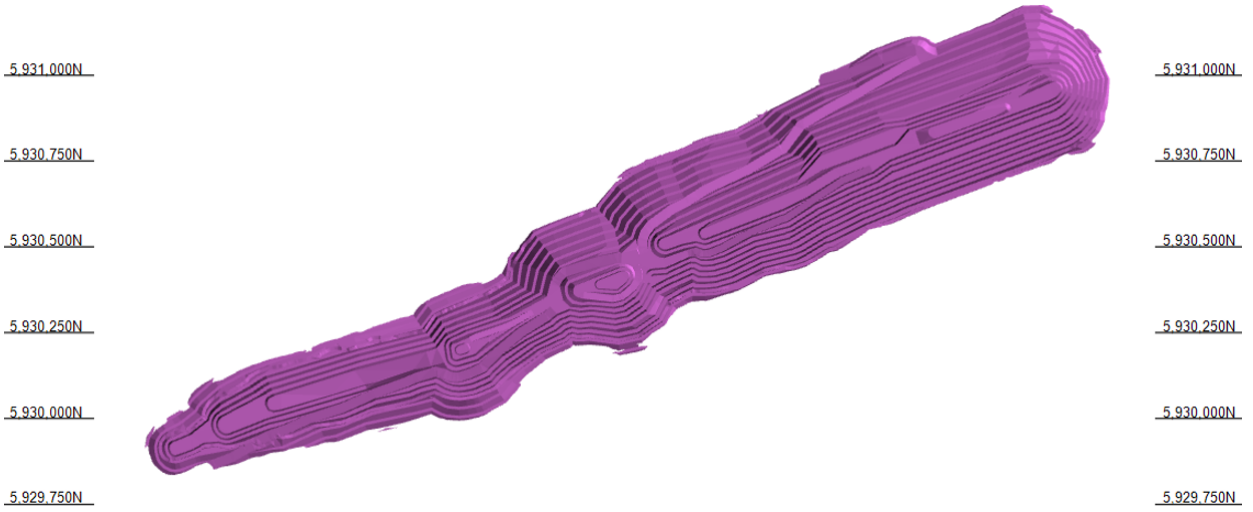
In March 2025, the open pit design was optimized from an economic perspective, as shown in Figure 91, resulting in modified lithological exposures along the final pit walls (Figure 92). In this updated configuration, the western portion of the pit is narrower and now comprises approximately four to five benches in height. The wall geometry has also evolved from a previously straight configuration to a more optimized, undulating profile.

This undulating geometry introduces local convex segments along the pit wall. Such convexities may reduce rock-mass confinement and increase the potential for instability relative to planar or concave geometries. Furthermore, these new wall orientations locally expose lithologies in directions that have not been previously analyzed.

Despite these changes, the overall impact on wall performance is expected to remain limited because the curvature is localized and does not extend continuously along the pit perimeter. From an operational standpoint, it is anticipated that the as-mined wall profile will be smoother and less undulating than the design representation, facilitating equipment operations and reducing localized geometric effects.

Given these considerations, the proposed wall geometry should be reassessed during detailed engineering to confirm its geotechnical feasibility and to ensure that operational constraints and stability requirements are adequately addressed.

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(a) Plan view.



(b) Side view from South-East.

Figure 91. Open pit FS design as provided by GMS in March 2025.

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Design guidelines have been updated for the pit sectors, as shown in Figure 92 and Table 69, which presents the slope design guidelines for each pit sector. These guidelines include one (1) sector for the overburden domain and ten (10) sectors for the fresh rock domain.

Due to the overall good quality and strength of the fresh rock, slope stability is expected to be controlled primarily by structural features rather than rock mass strength. However, particular attention should be given to the pit areas where UMU is exposed, as the current FS design results in localized exposure of this rock type. Future designs should aim to minimize UMU exposure by either mining out these zones or avoiding UMU sectors altogether. Additionally, 16 m-wide geotechnical berms are recommended around the mid-slope in rock slopes at least 160 m high to reduce overall slope angles and provide additional design contingency.

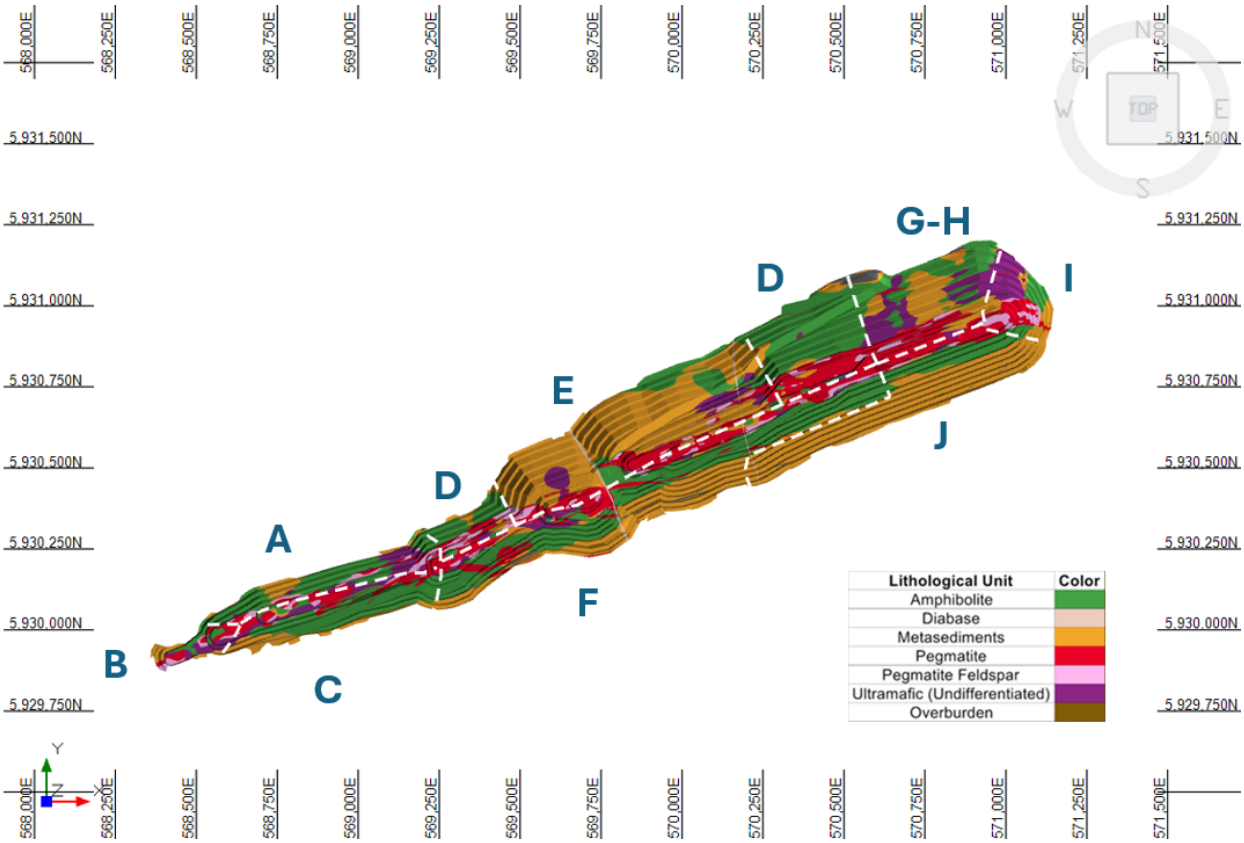


Figure 92. Open pit sectors based on the March 2025 pit design.

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Table 69. Open pit slope design guidelines.

Sector	GTD/GMD	Recommended FS Criteria					
		BH (m)	**BW** (m)	**BFA** (°)	**IRA** (°)	**DD** (°)	**Geotechnical Berm**
1	OVB	< 25	0.0	36	36	N/A	10 m between overburden and rock
A	WAll-AMP	20	10.5	68	47	130–200	At mid-wall if H > 160 m. Width = 16 m (total)
B	WW-APU	20	8.5	75	55	005–130	
C	WAll-AMP, WS-META	20	11.0	75	51	340–005	
D	CN-AMP	20	8.5	59	44	150–165	
E	CN-META, CN-UMU	20	8.5	50	38	140–155	
F	CS-AMP, CS-META	20	11.0	75	51	330–350	
G	EAll-AMP, EN-META	20	8.5	47	36	165–180	
H	EN-UMU	20	8.5	40	32	165–180	
I	EE-AMPU	20	8.5	75	55	180–310	
J	EAll-AMP, ES-AMP	20	12.0	75	49	310–335	

Codes **BH** Bench height, **BW** Bench width, **BFA** Bench face angle, **IRA** Inter-ramp angle, **DD** Dip direction range, **H** Height, **GTD** Geotechnical domain, **GMD** Geomechanical domain.

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E Detailed Bench Scale Assessments

The bench scale analyses results obtained with the geomechanical risk-based approach are presented for each pit sector in the series of Figures 93 to 107. optimized design parameters are presented in Tables 70 to 84. The final adopted bench configuration can differ from the optimized results due to standardization. It should be noted that the design guidelines are also based on engineering judgment, assumptions, and experience.

E.1 Sector A – AMP, North Wall–West Area

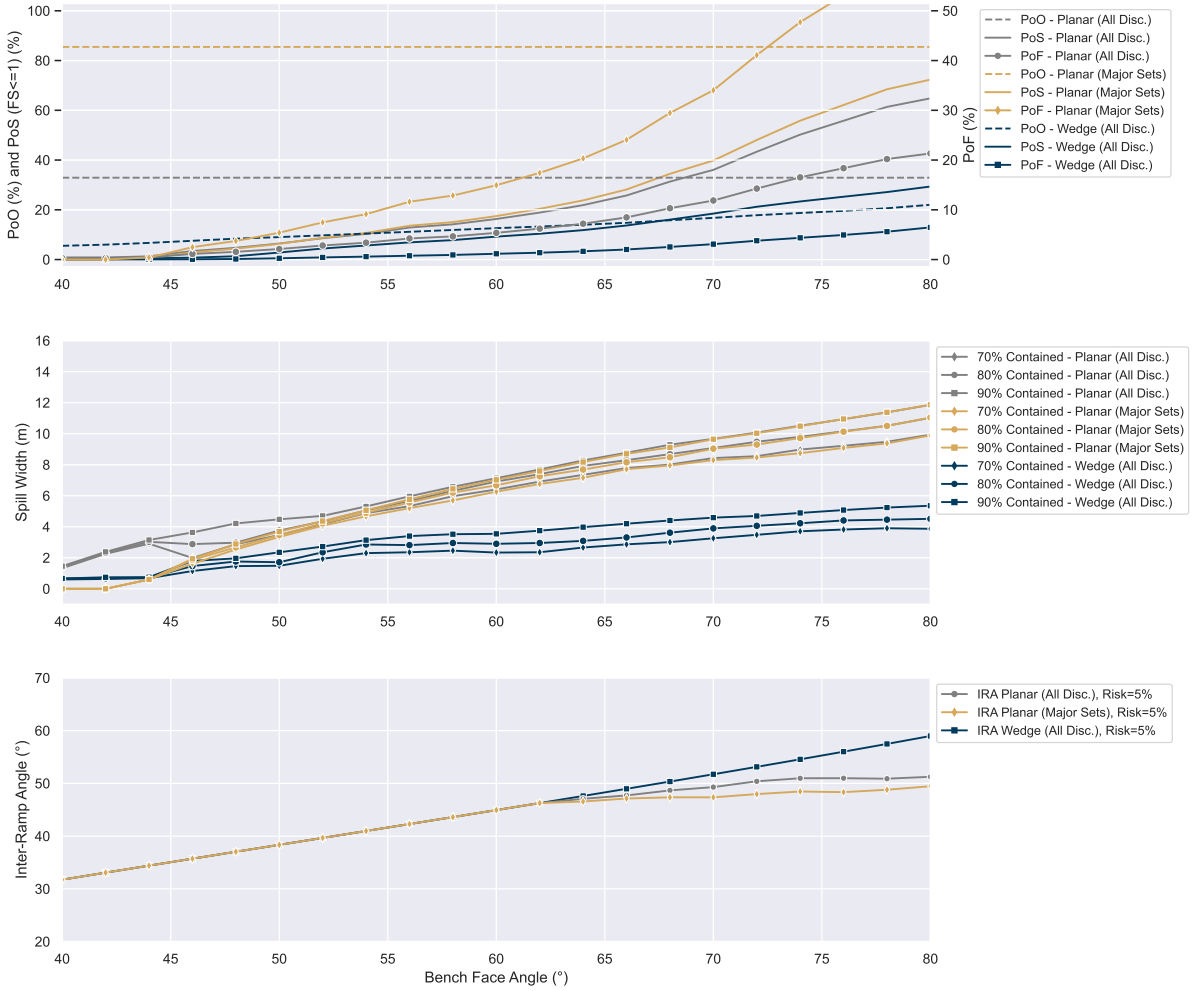


Figure 93. Bench scale stability analyses results – Sector A (AMP).

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Table 70. Bench design optimization parameters – Sector A (AMP)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	85.5%
	PoS	31.3%
	PoF	29.5%
Wedge	PoO	15.8%
	PoS	16.0%
	PoF	2.5%
Design Constraints		
Limitation		Planar sliding & Minimum bench width
Design Outputs		
Bench Face Angle (BFA)		68°
Spill Contained (% Contained, Width) [3]		85%, 9.0 m
Bench Width (BW)		10.5 m *from max(8.5 ; 9.0 + 1.5)*
Inter-Ramp Angle (IRA)		47°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector A in the amphibolite, a single major discontinuity set (JS1) can cause issues from a planar-sliding perspective. The JS1 joint set is mainly composed of foliation, open and closed joints and has an average joint spacing of 2.3 m. Wedge sliding was not identified as limiting for this sector and lithology. The BFA selection is therefore based on the planar PoF, and the catch bench was adjusted to retain 85 % of the spill length. With those parameters, the evaluated risk (eq. 28) is 4.4%.

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E.2 Sector B – AMP, PEG, UMU, West Wall – West Area.

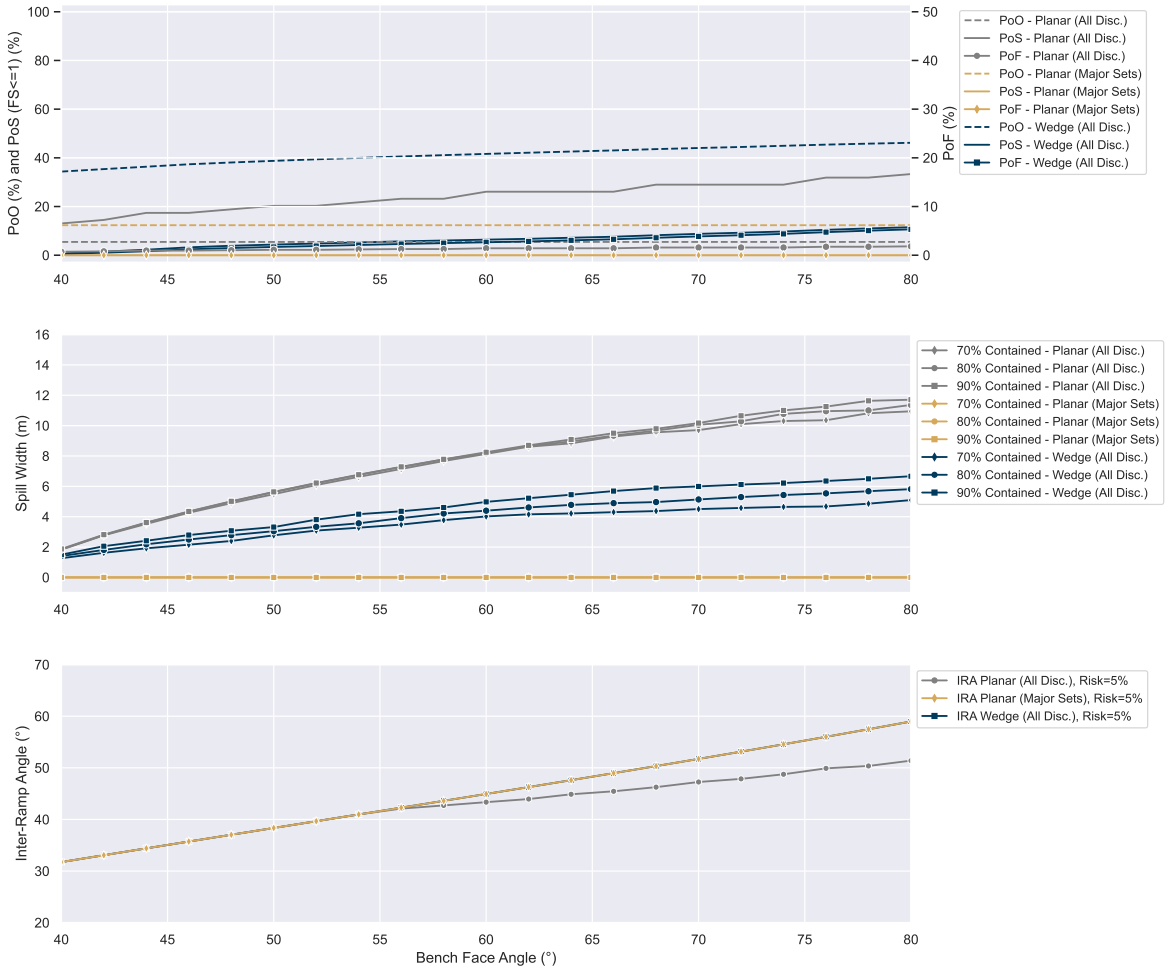


Figure 94. Bench scale stability analyses results – Sector B (AMP, PEG, UMU).

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Table 71. Bench design optimization parameters – Sector B (AMP, PEG, UMU)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	12.3%
	PoS	0.0%
	PoF	0.0%
Wedge	PoO	45.1%
	PoS	10.1%
	PoF	4.6%
Design Constraints		
Limitation		Max. BFA
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		70%, 4.7 m
Bench Width (BW)		8.5 m *from max(8.5 ; 4.7 + 1.5)*
Inter-Ramp Angle (IRA)		55°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector B, in the interlayered lithologies exposed on the wall, namely the amphibolite, the pegmatite, and the ultramafic, no surveyed discontinuity set is expected to cause issues from a planar sliding perspective. Since wedge sliding was also not identified as a limiting factor in this sector, the recommended bench face angle equals the maximum allowed, 75°.

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E.3 Sector C-1 – AMP, South Wall – West Area.

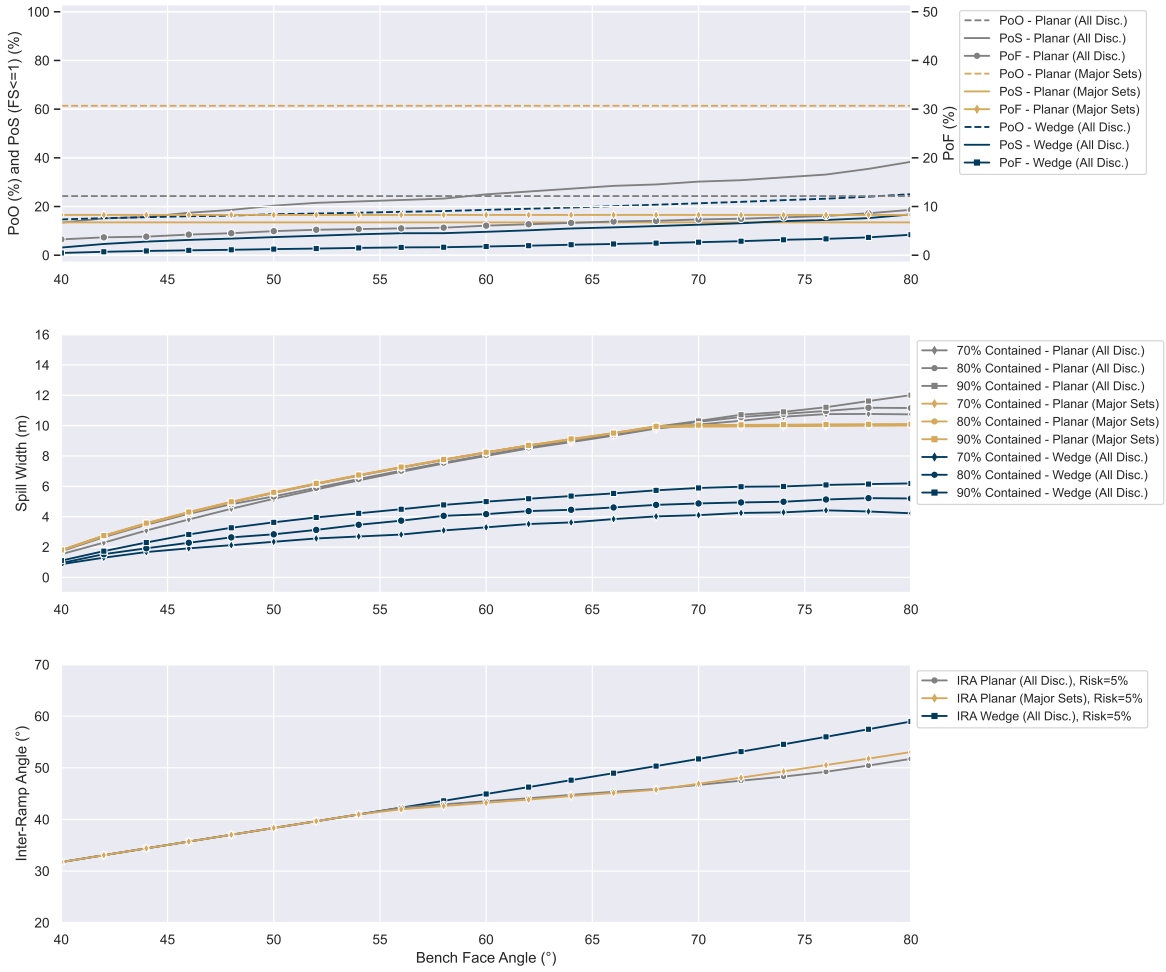


Figure 95. Bench scale stability analyses results – Sector C-1 (AMP).

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Table 72. Bench design optimization parameters – Sector C-1 (AMP)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	61.4%
	PoS	13.5%
	PoF	8.3%
Wedge	PoO	22.9%
	PoS	14.2%
	PoF	3.3%
Design Constraints		
Limitation		Max. BFA & Minimum bench width
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		70%, 10.0 m
Bench Width (BW)		11.5 m[4] *from max(8.5 ; 10.0 + 1.5)*
Inter-Ramp Angle (IRA)		50°[4]

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.
[4] Prior to standardization.

For Sector C-1, in the amphibolite, one discontinuity set (JS2) can lead to issues from a planar sliding perspective. The JS2 joint set is mainly composed of open and closed joints and has an average joint spacing of 4.86 m. Wedge sliding was not identified as limiting for this sector and lithology. Due to the relatively high joint spacing and to standardize implementation of the guidelines, a bench width of 11 m is recommended for this sector instead of 11.5 m, resulting in an IRA of 51° for this sector.

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E.4 Sector C-2 – META, South Wall – West Area.

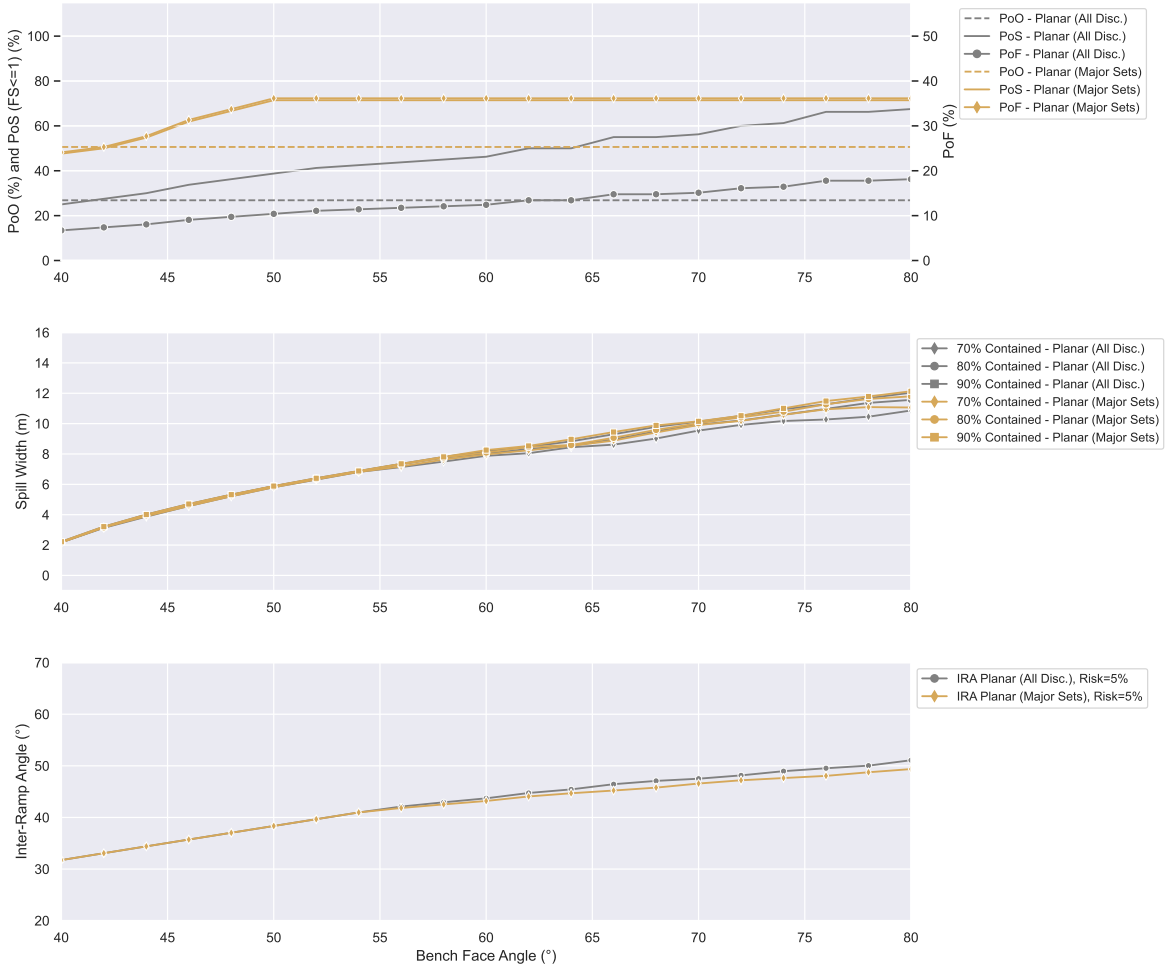


Figure 96. Bench scale stability analyses results – Sector C-2 (META).

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Table 73. Bench design optimization parameters – Sector C-2 (META)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	50.6%
	PoS	71.4%
	PoF	36.1%
Wedge	PoO	–%
	PoS	–%
	PoF	–%
Design Constraints		
Limitation		Planar sliding & Minimum bench width
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		90%, 11.0 m
Bench Width (BW)		12.5 m[4] *from max(8.5 ; 11.0 + 1.5)*
Inter-Ramp Angle (IRA)		48°[4]

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.
[4] Prior to standardization.

For Sector C-2, in the metasediments, one discontinuity set (JS2) can lead to issues from a planar sliding perspective. The JS2 joint set is mainly composed of open and closed joints and has an average joint spacing of 3.90 m. A PoF of 36% is reached at a BFA of 75°, which is slightly over the 30% target. However, as shown in the Figure 96, the PoF reaches 30% at 45° and remains at 36% for BFA higher than 50°. Furthermore, only one to three benches are expected to expose metasediments on the pit wall. Therefore, due to the relatively high joint spacing of JS2, the expected limited exposure of this lithology on the pit wall and for standardization to facilitate implementation of the guidelines, it was deemed acceptable to recommended a BFA of 75° and a bench width of 11 m for this sector (instead of 12.5 m) which results to an IRA of 51° for this sector. This way, the design is consistent throughout design sector C. At a BFA of 75°, the evaluated risk (eq. 28) is over 10%, which is only acceptable due to the limited exposure of this lithology on the pit wall in sector C.

Detailed wedge sliding was not evaluated for this sector and lithology, as it was not deemed limiting for the design of this sector based on the analysis of the stereonet projection.

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E.5 Sector D – AMP, North Wall – Central Area.

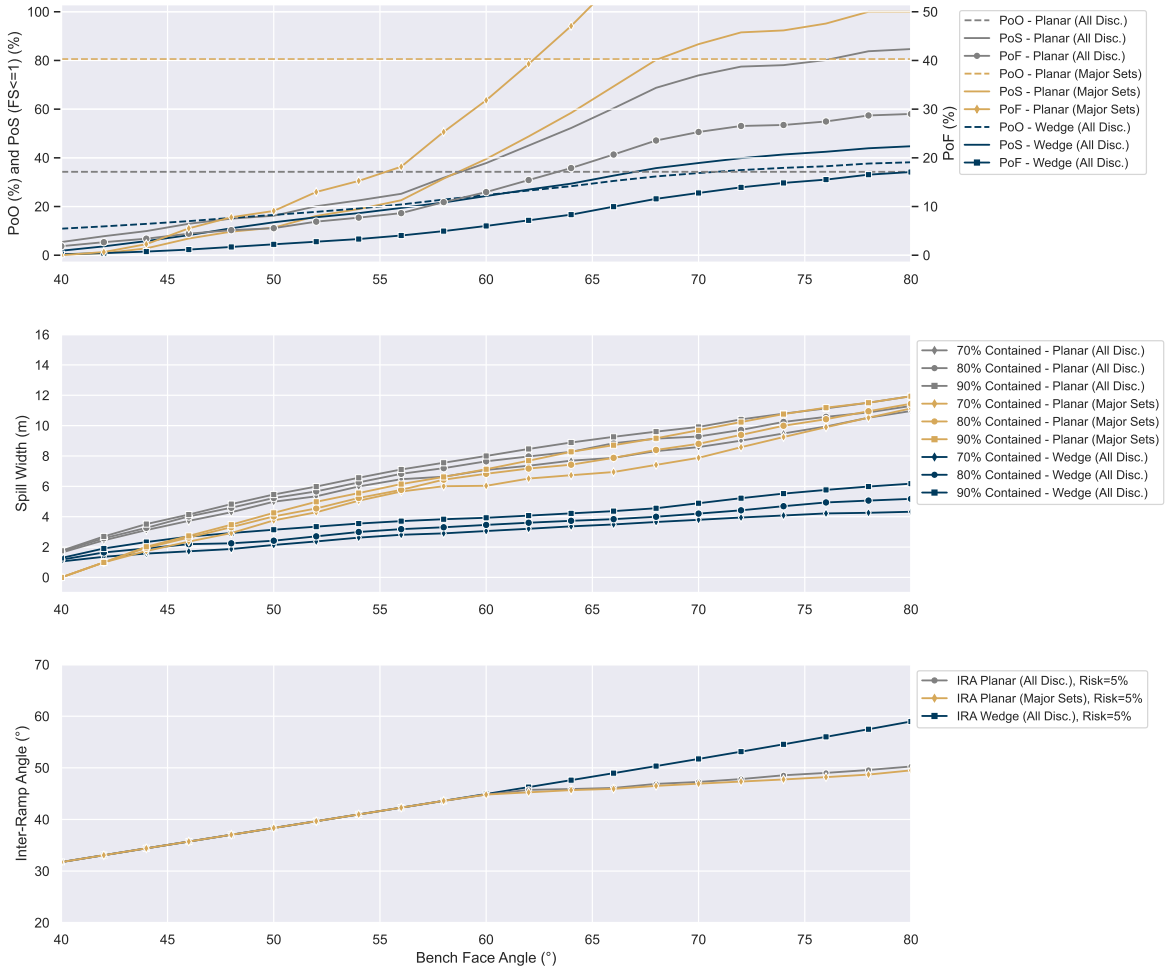


Figure 97. Bench scale stability analyses results – Sector D (AMP).

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Table 74. Bench design optimization parameters – Sector D (AMP)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	80.5%
	PoS	35.5%
	PoF	29.8%
Wedge	PoO	23.8%
	PoS	23.0%
	PoF	5.5%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		59°
Spill Contained (% Contained, Width) [3]		85%, 6.8 m
Bench Width (BW)		8.5 m *from max(8.5 ; 6.8 + 1.5)*
Inter-Ramp Angle (IRA)		44°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector D, in the amphibolites, two discontinuity sets (JS1 and js3) can lead to issues from a planar sliding perspective. However, due to the higher joint spacing in js3 (3.44 m) and its average pole dip (29°) being below the amphibolite friction angle (32°), the analysis considered only the set JS1. The JS1 joint set is mainly composed of foliation, open and closed joints and has an average joint spacing of 1.58 m. Wedge sliding was not identified as limiting for this sector and lithology.

Although not deemed limiting for the design, the js3 joint set should be monitored due to its relatively low dip angle. A degree of sampling bias is suspected, as seven out of twelve poles in this set may create the impression of a double joint set with JS1. Nevertheless, two strategically located boreholes did not record any structural measurements corresponding to JS3, suggesting that this feature is of limited significance. However, continued monitoring is recommended.

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E.6 Sector E-1 – META, North Wall – Central Area.

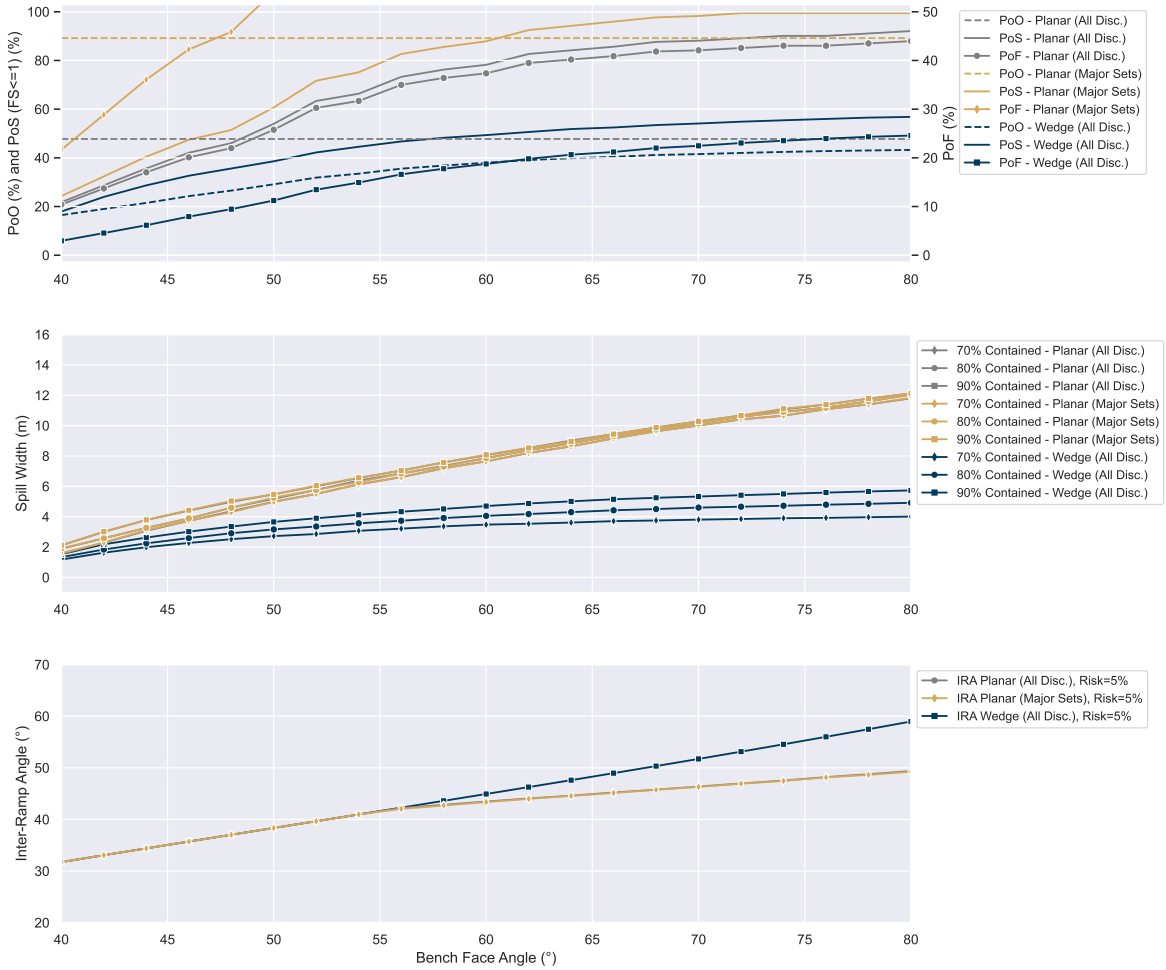


Figure 98. Bench scale stability analyses results – Sector E-1 (META).

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Table 75. Bench design optimization parameters – Sector E-1 (META)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	89.2%
	PoS	60.7%
	PoF	54.1%
Wedge	PoO	29.1%
	PoS	38.6%
	PoF	11.2%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		50°
Spill Contained (% Contained, Width) [3]		95%, 5.8 m
Bench Width (BW)		8.5 m *from max(8.5 ; 5.8 + 1.5)*
Inter-Ramp Angle (IRA)		38°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector E-1 in the metasediments, a major discontinuity set (JS1) can cause issues from a planar-sliding perspective. The JS1 joint set is mainly composed of foliation, open and closed joints and has an average joint spacing of 1.02 m.

A PoF of 54% is reached at a BFA of 50°, which is over the 30% target. However, as shown in the Figure 98, the PoF reaches 30% at 42°, which would be very limiting from a design perspective. Since joint spacing is low (1.02 m), caution is required with this set. Regarding the composing of this set, fault and open joints mostly have a higher dip than the average pole of the set. On the stereonet projection, foliation and closed joints poles are located around the average pole of the set or below (lower dip). Taking that into consideration, it is considered appropriate to design this pit sector with the average pole of the JS1 set (dip of about 50°). It is also important to mention that in both cases (BFA of 42° and 50°), the evaluated risk (eq. 28) remains below the 5% target.

Wedge sliding was not identified as limiting for this sector and lithology.

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E.7 Sector E-2 – UMU, North Wall – Central Area.

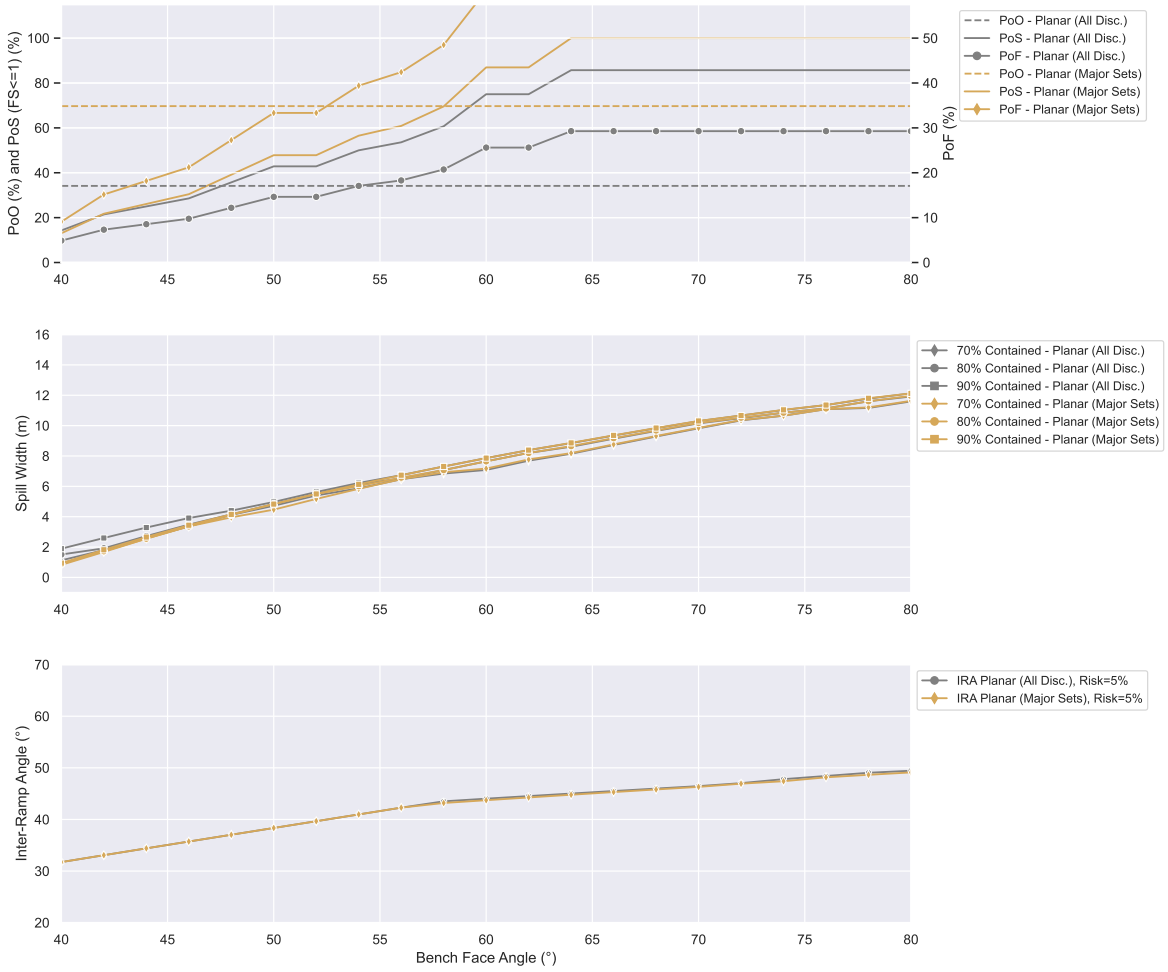


Figure 99. Bench scale stability analyses results – Sector E-2 (UMU).

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Table 76. Bench design optimization parameters – Sector E-2 (UMU)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	70.0%
	PoS	48.0%
	PoF	33.0%
Wedge	PoO	–%
	PoS	–%
	PoF	–%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		50°
Spill Contained (% Contained, Width) [3]		85%, 4.2 m
Bench Width (BW)		8.5 m *from max(8.5 ; 4.2 + 1.5)*
Inter-Ramp Angle (IRA)		38°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

The ultramafic undifferentiated unit is not expected to be exposed over a large portion of the pit wall; however, a brief analysis was conducted for the UMU pocket exposed in Sector E. If a larger exposure of the UMU occurs, this analysis should be re-evaluated.

For Sector E-2, in the ultramafic undifferentiated, a single discontinuity set (JS1) can lead to issues from a planar-sliding perspective. The JS1 joint set is mainly composed of foliation, faults, open and closed joints and has an average joint spacing of 0.53 m. A PoF of 33% is reached at a BFA of 50°, which is slightly over the 30% target. As shown in the Figure 99, the PoF reaches 30% at 48°. Only one to three benches are expected to be exposed very locally on the pit wall. Therefore, due to the expected limited exposure of this lithology on the pit wall and to standardize for the implementation of the guidelines, it was deemed acceptable to recommend a BFA of 50° for this sector. This way, the design is consistent throughout design sector E. At a BFA of 50°, the evaluated risk (eq. 28) remains about 5%.

Detailed wedge sliding was not evaluated for this sector and lithology as it was not deemed limiting for the design of this sector based on the analysis of the stereonet projection.

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E.8 Sector F-1 – AMP, South Wall – Central Area.

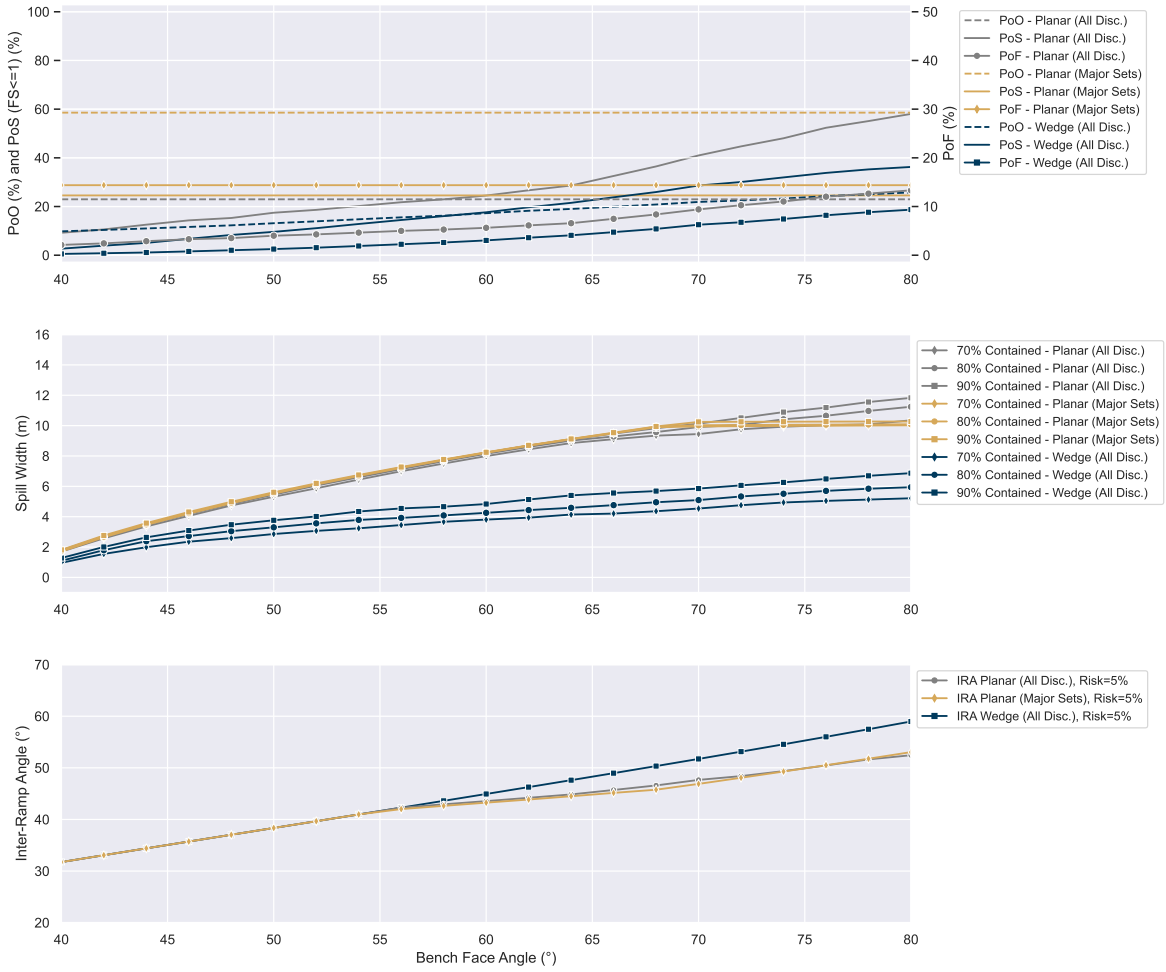


Figure 100. Bench scale stability analyses results – Sector F-1 (AMP).

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Table 77. Bench design optimization parameters – Sector F-1 (AMP)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	58.6%
	PoS	24.6%
	PoF	14.4%
Wedge	PoO	23.8%
	PoS	32.9%
	PoF	7.8%
Design Constraints		
Limitation		Max. BFA
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		70%, 10.0 m
Bench Width (BW)		11.5 m *from max(8.5 ; 10.0 + 1.5)*
Inter-Ramp Angle (IRA)		50°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector F-1, in the amphibolites, two discontinuity sets (JS2 and js3) can lead to issues from a planar sliding perspective. However, due to the fact that the average pole dip of js3 (77°) is higher than the maximum BFA allowed (75°), the analysis considered only the set JS2. The JS2 joint set is mainly composed of open and closed joints and has an average joint spacing of 5.49 m.

Due to the high joint spacing of JS2 (5.49) and for standardization to facilitate implementation of the guidelines, it was deemed acceptable to recommend a bench width of 11 m for this sector (instead of 11.5 m), which results in an IRA of 51° for this sector. This way the design is consistent throughout design sector F and the south walls in sector C. With this suggested BW, at a BFA of 75°, the evaluated risk (eq. 28) is slightly above 5%, which is deemed acceptable due to high joint spacing and low PoF (14.4%) results for this lithology on the pit wall in sector F-1. Wedge sliding was not identified as limiting for this sector and lithology.

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E.9 Sector F-2 – META, South Wall – Central Area.

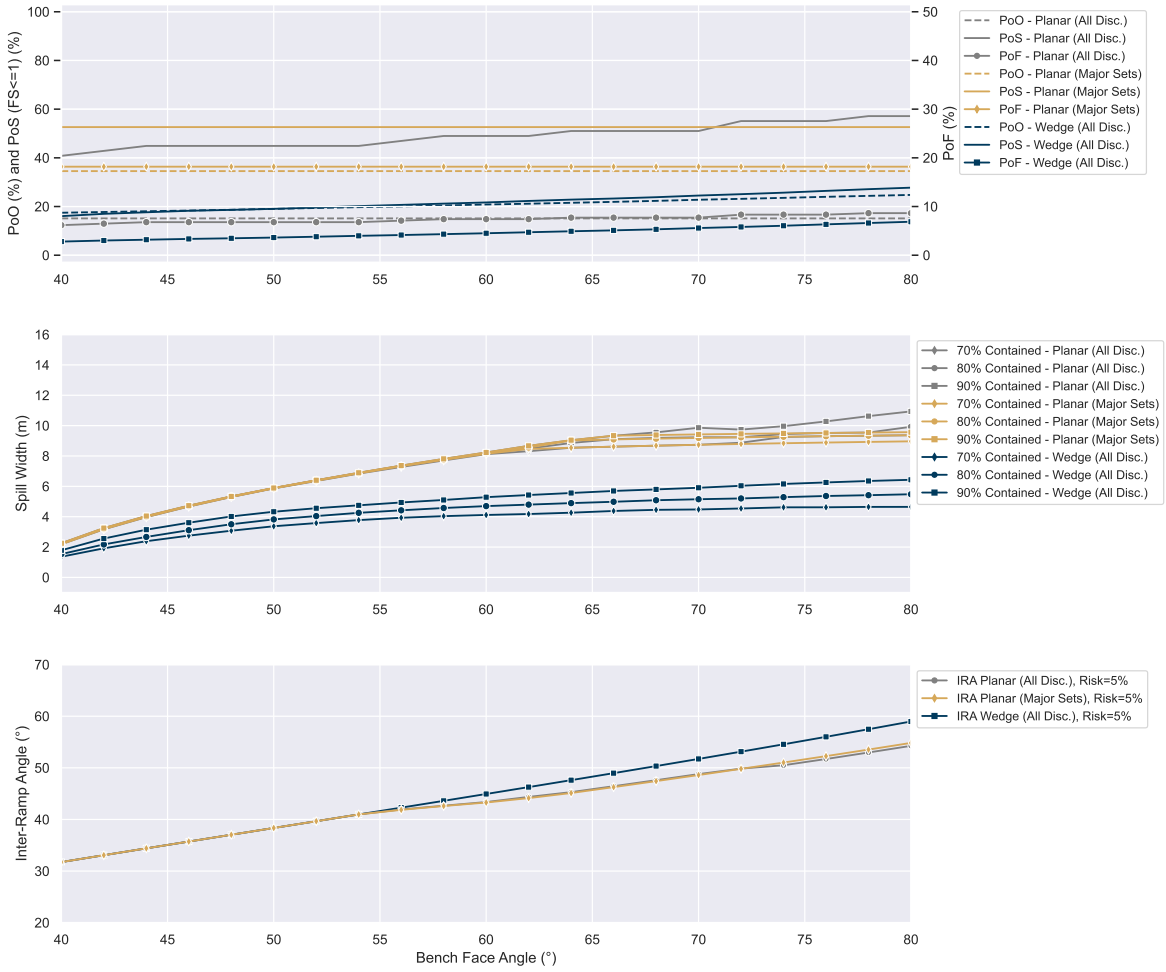


Figure 101. Bench scale stability analyses results – Sector F-2 (META).

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Table 78. Bench design optimization parameters – Sector F-2 (META)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	34.6%
	PoS	52.6%
	PoF	18.2%
Wedge	PoO	23.8%
	PoS	26.1%
	PoF	6.2%
Design Constraints		
Limitation		Max. BFA
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		75%, 9.0 m
Bench Width (BW)		10.5 m *from max(8.5 ; 9.0 + 1.5)*
Inter-Ramp Angle (IRA)		51.6 °

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector F-2 in the metasediments, a single discontinuity set (JS2) can cause issues from a planar-sliding perspective. The JS2 joint set is mainly composed of faults, open and closed joints and has an average joint spacing of 0.98 m.

For standardization, it was deemed acceptable to recommend a bench width of 11 m for this sector (instead of 10.5 m), which results in an IRA of 51° for this sector. This way, the design is consistent throughout design sector F and the south walls in sector C. Wedge sliding was not identified as limiting for this sector and lithology.

Note that the JS2 contains several faults and open joints; it is recommended to compare it with the final structural model to evaluate potential impacts on design assumptions.

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E.10 Sector G-1 – AMP, North Wall – East Area.

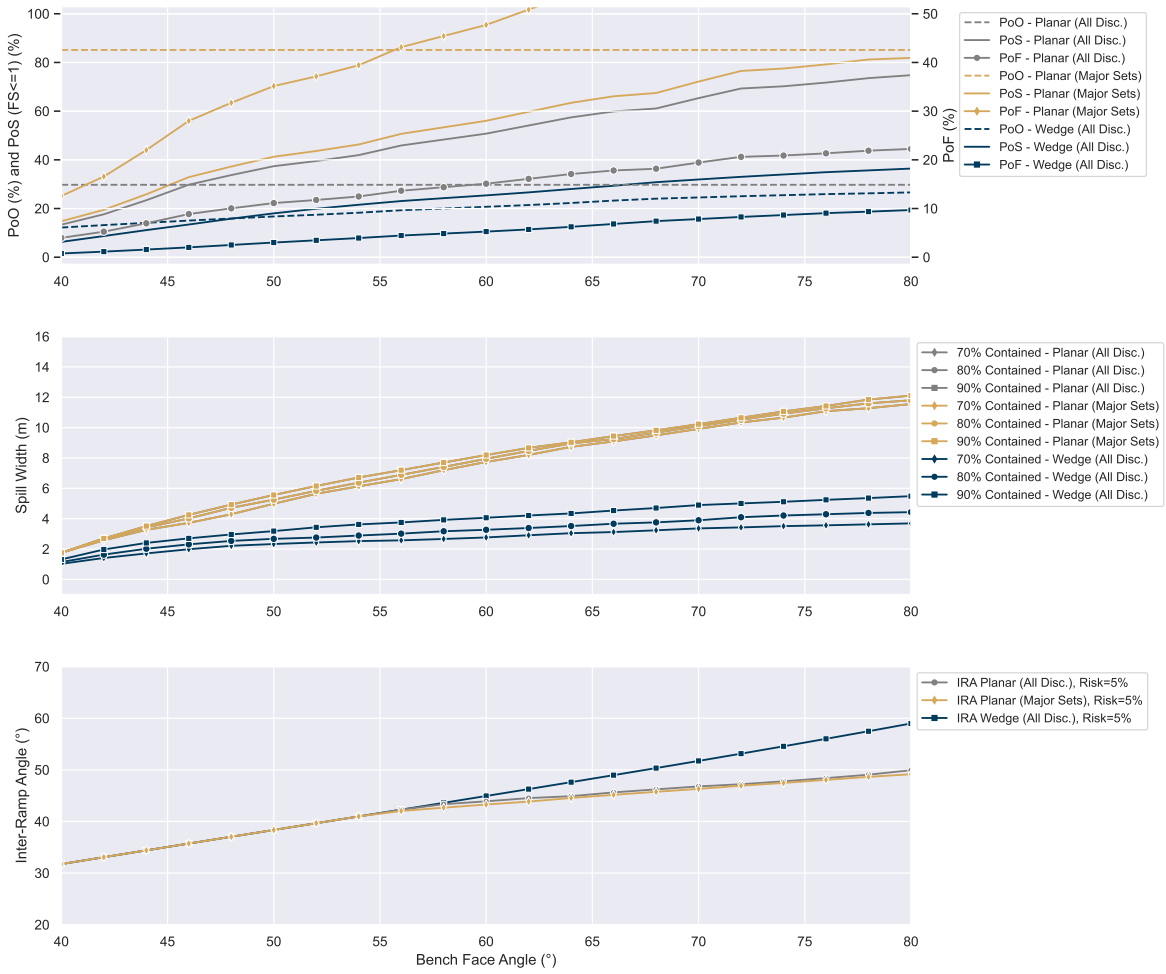


Figure 102. Bench scale stability analyses results – Sector G-1 (AMP).

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Table 79. Bench design optimization parameters – Sector G-1 (AMP)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	29.7%
	PoS	35.0%
	PoF	29.8%
Wedge	PoO	15.5%
	PoS	14.6%
	PoF	2.3%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		47°
Spill Contained (% Contained, Width) [3]		85%, 4.5 m
Bench Width (BW)		8.5 m *from max(8.5 ; 4.5 + 1.5)*
Inter-Ramp Angle (IRA)		36°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector G-1, in the amphibolites, a single discontinuity set (JS1) can cause issues from a planar-sliding perspective. It tends to be divided into two groups of poles with slightly different dips, but was treated as a single set for the analysis. The JS1 joint set comprises foliation, fault, open, and closed joints. If only open joints are considered, the division into two groups is more apparent (62°/153°and 37°/164°). This set constrains the design due to the planar sliding issue; the recommended BFA equals the average pole dip of JS1. Wedge sliding was not identified as limiting for this sector and lithology.

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E.11 Sector G-2 – META, North Wall – East Area.

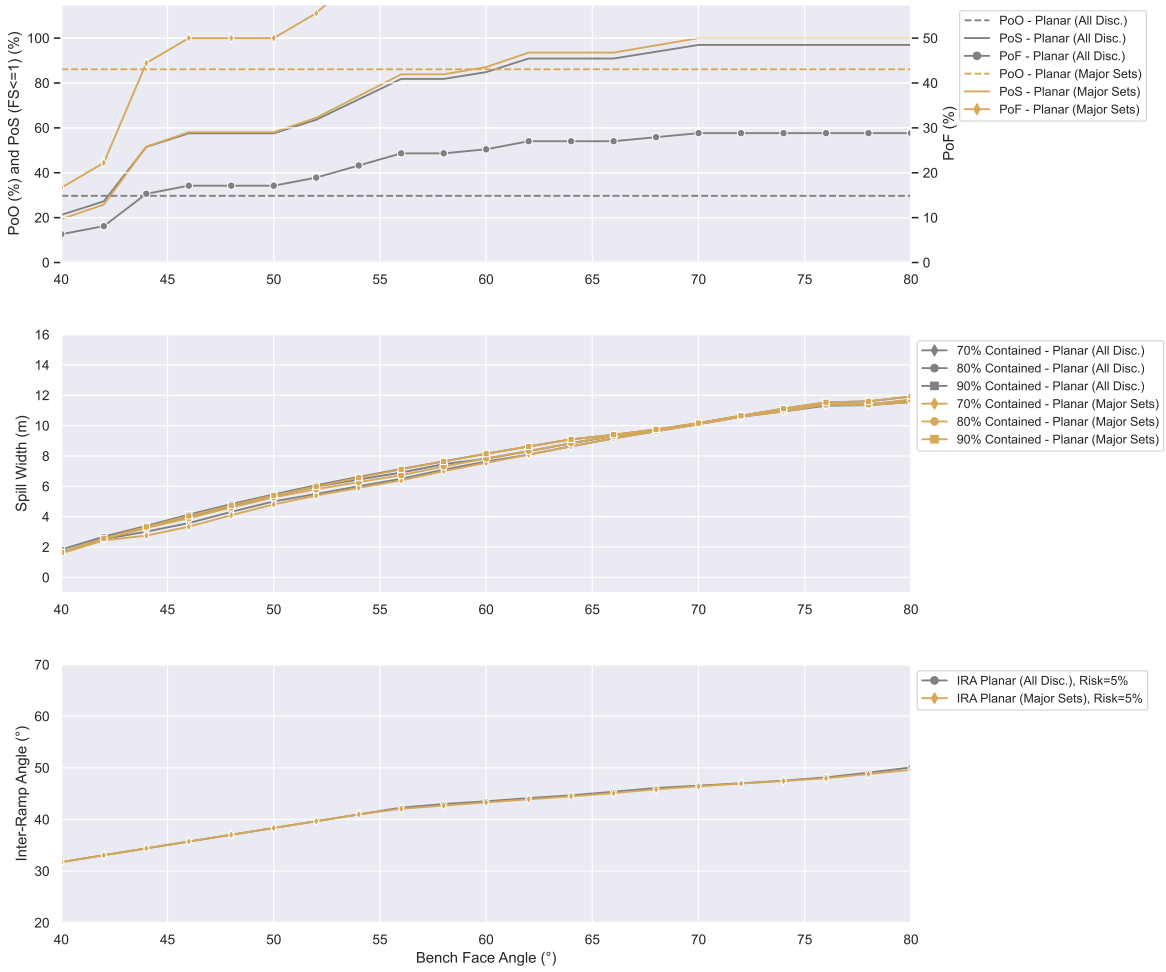


Figure 103. Bench scale stability analyses results – Sector G-2 (META).

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Table 80. Bench design optimization parameters – Sector G-2 (META)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	86.1%
	PoS	58.6%
	PoF	50%
Wedge	PoO	–%
	PoS	–%
	PoF	–%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		47°
Spill Contained (% Contained, Width) [3]		90%, 4.4 m
Bench Width (BW)		8.5 m *from max(8.5 ; 4.4 + 1.5)*
Inter-Ramp Angle (IRA)		36°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector G-2 in the metasediments, a single discontinuity set (JS1) can cause issues from a planar-sliding perspective. The JS1 joint set is primarily foliation-controlled and has an average joint spacing of 2.36 m.

A PoF of 50% is reached at a BFA of 47°, which is over the 30% target. As shown in the Figure 103, the PoF reaches 30% a little over 42°. It is deemed acceptable to go over the PoF of 30% since the 47° BFA remains lower in dip than the average pole of JS1 (51°) (because the main joint type is foliation). This will lead to a more natural design with slope following approximately the average pole of foliation. Therefore, for standardization to facilitate implementation of the guidelines, it was deemed acceptable to recommend a BFA of 47° for this sector. This way the design is consistent throughout design sector G. At a BFA of 47°, the evaluated risk (eq. 28) remains about 5%.

Detailed wedge sliding was not evaluated for this sector and lithology, as it was not deemed limiting for the design of this sector based on the analysis of the stereonet projection.

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E.12 Sector H – UMU, North Wall – East Area.

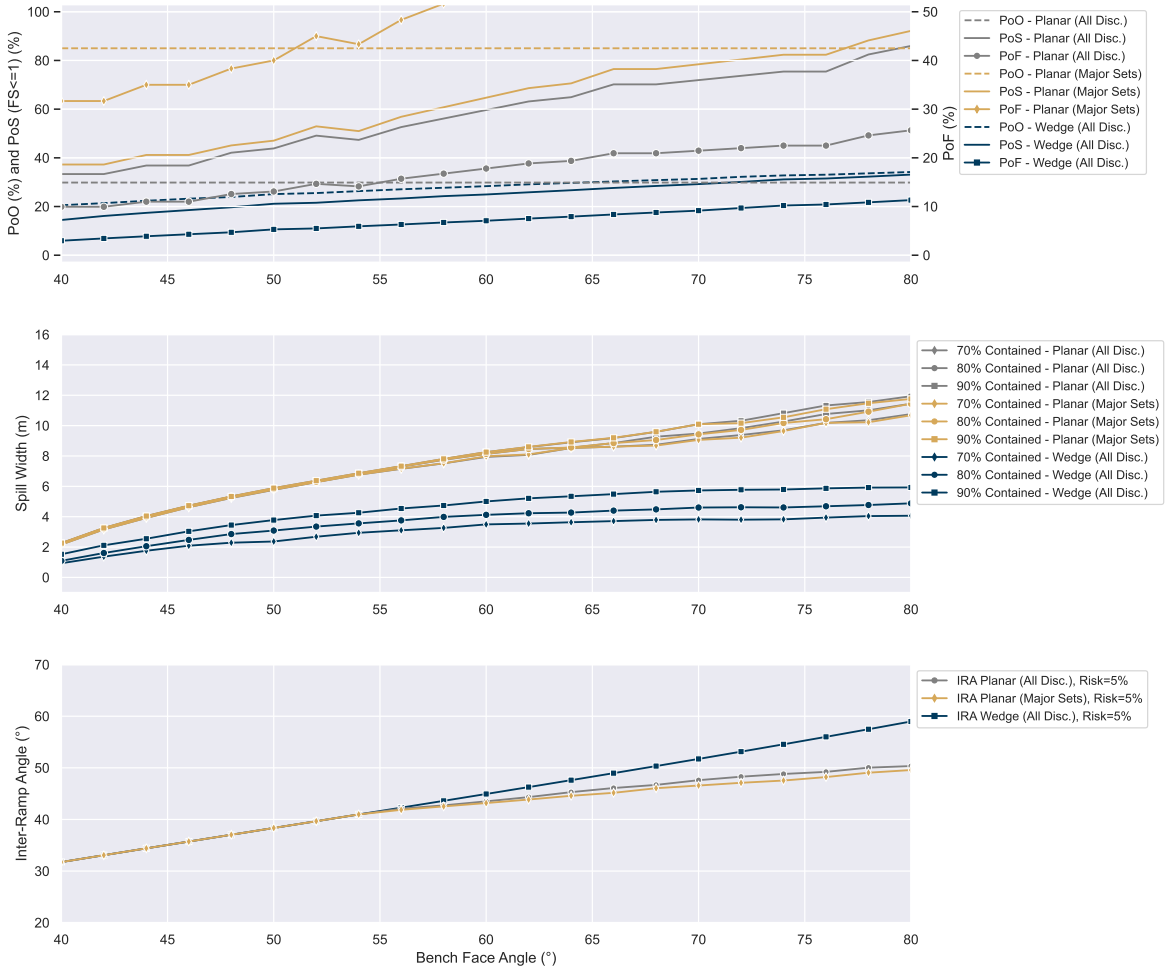


Figure 104. Bench scale stability analyses results – Sector H (UMU).

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Table 81. Bench design optimization parameters – Sector H (UMU)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	85%
	PoS	37.2%
	PoF	31.6%
Wedge	PoO	20.9%
	PoS	15.3%
	PoF	3.2%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		40°
Spill Contained (% Contained, Width) [3]		85%, 2.5 m
Bench Width (BW)		8.5 m *from max(8.5 ; 2.5 + 1.5)*
Inter-Ramp Angle (IRA)		32°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector H, in the ultramafic undifferentiated, a single discontinuity set (JS1a/JS1b) can lead to issues from a planar-sliding perspective. This set is divided into two groups of poles with different dips, both of which were considered for analysis. The sets consist of fault, open, and closed joints and have joint spacings of 1.23 m (JS1a) and 0.42 m (JS1b). If only open joints are considered, they present an average pole dip of 29°, which is slightly above the friction angle for the ultramafic (22°). This set constrains the design due to the planar sliding issue. The recommended BFA is 40°, although the associated PoF is slightly over the 30% target, since the evaluated risk (eq. 28) remains below 5%. Wedge sliding was not identified as limiting for this sector and lithology.

Note that little information is available for this sector and lithology, and the available information indicates potential issues with the discontinuities. In addition, the contact with the amphibolite dips towards the pit (44.3°).

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E.13 Sector I – AMP, PEG, UMU, META, East Wall – East Area.

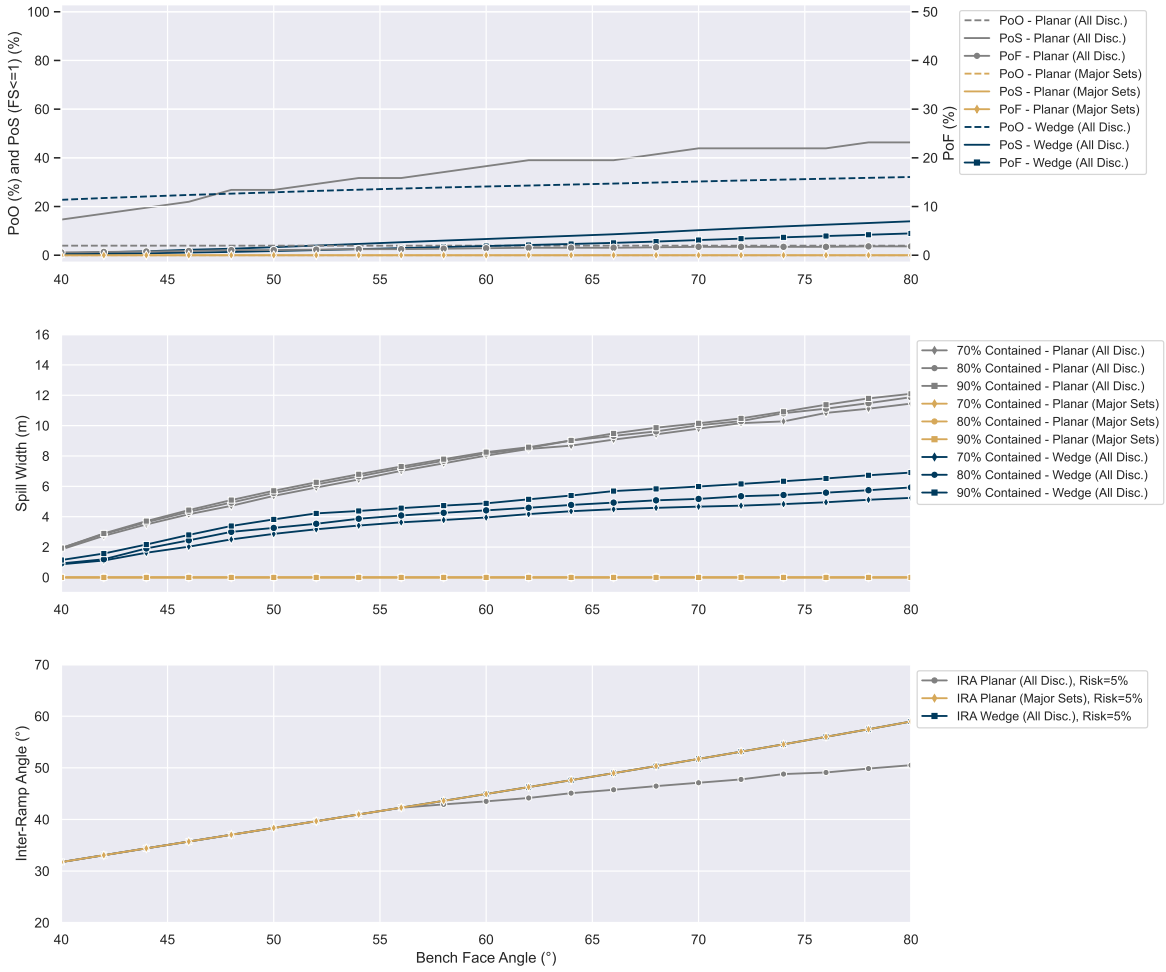


Figure 105. Bench scale stability analyses results – Sector I (AMP, PEG, UMU, META).

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Table 82. Bench design optimization parameters – Sector I (AMP, PEG, UMU, META)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	0.0%
	PoS	0.0%
	PoF	0.0%
Wedge	PoO	31.2%
	PoS	12.2%
	PoF	3.8%
Design Constraints		
Limitation		Max. BFA
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		70%, 4.9 m
Bench Width (BW)		8.5 m *from max(8.5 ; 4.9 + 1.5)*
Inter-Ramp Angle (IRA)		55°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector I, in the interlayered lithologies exposed on the wall, namely the amphibolite, the pegmatite, the metasediments, and the ultramafic undifferentiated, no surveyed discontinuity set is expected to pose issues from a planar sliding perspective. Since wedge sliding was also not identified as limiting in this sector, the recommended bench face angle equals the maximum allowed, 75°.

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E.14 Sector J-1 – AMP, South Wall – East Area.

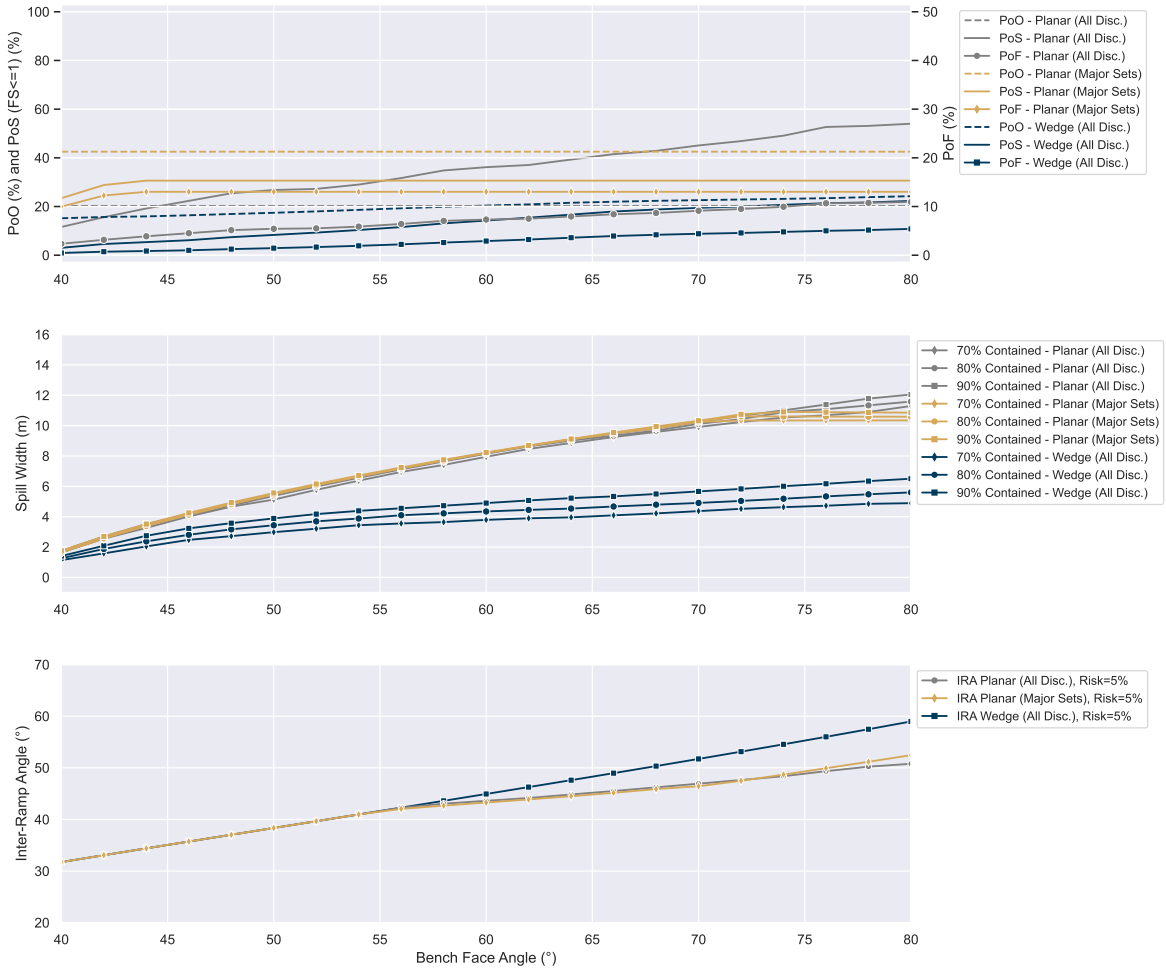


Figure 106. Bench scale stability analyses results – Sector J-1 (AMP).

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Table 83. Bench design optimization parameters – Sector J-1 (AMP)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	42.5%
	PoS	30.6%
	PoF	13.0%
Wedge	PoO	23.3%
	PoS	21.0%
	PoF	4.9%
Design Constraints		
Limitation		Max. BFA
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		70%, 10.5 m
Bench Width (BW)		12 m *from max(8.5 ; 10.5 + 1.5)*
Inter-Ramp Angle (IRA)		49°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector J-1 in the amphibolites, a single discontinuity set (JS1) can cause issues from a planar-sliding perspective. The JS1 joint set is mainly composed of faults, open and closed joints and has an average joint spacing of 3.74 m. However, the planar PoF does not limit the design, as it remains below the 30% target. Since wedge sliding was also not identified as limiting in this sector, the recommended bench face angle equals the maximum allowed, 75°.

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E.15 Sector J-2 – META, South Wall – East Area.

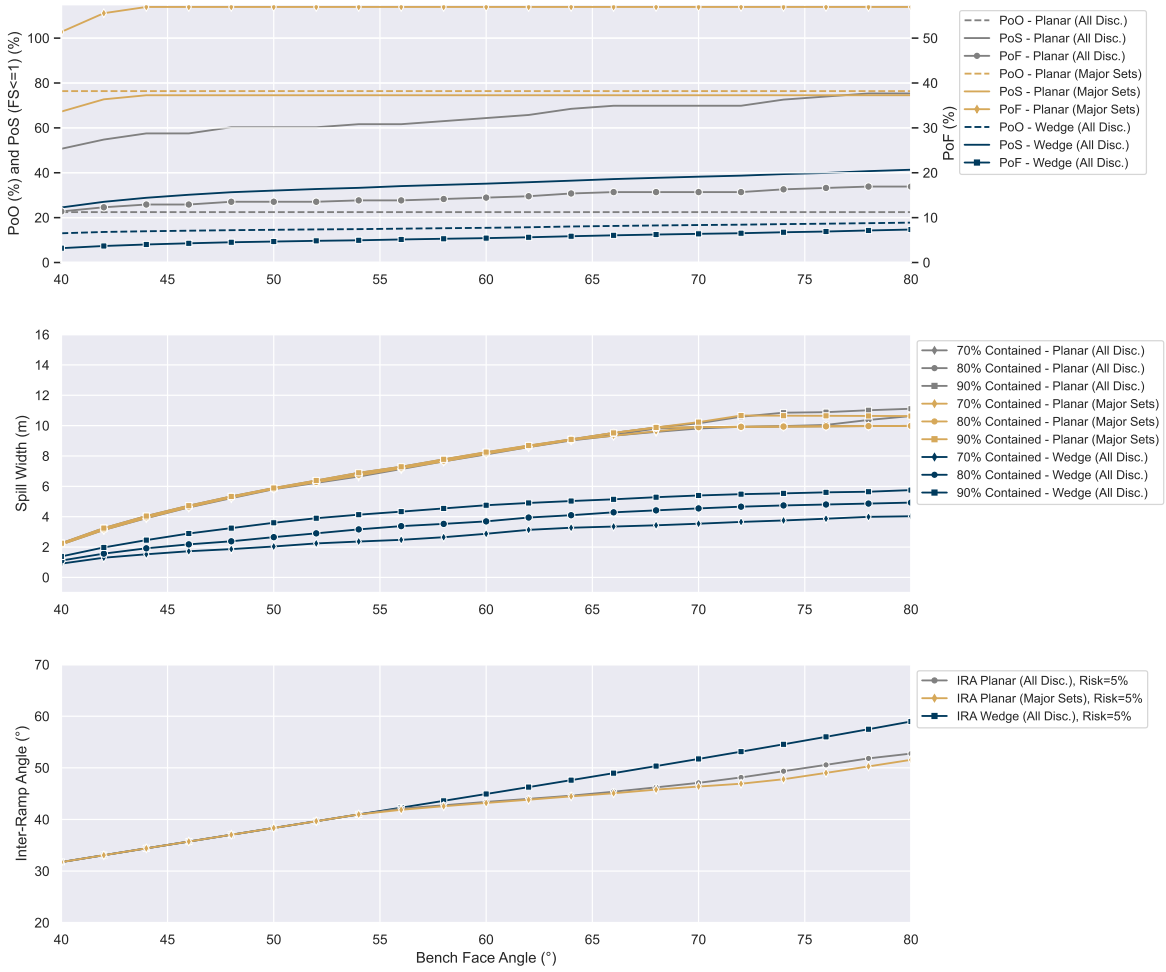


Figure 107. Bench scale stability analyses results – Sector J-2 (META).

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Table 84. Bench design optimization parameters – Sector J-2 (META)

Parameter		Value
Probabilities [1]		
Planar [2]	PoO	76.4%
	PoS	74.5%
	PoF	56.9%
Wedge	PoO	17.2%
	PoS	39.7%
	PoF	6.8%
Design Constraints		
Limitation		Planar sliding
Design Outputs		
Bench Face Angle (BFA)		75°
Spill Contained (% Contained, Width) [3]		95%, 11.0 m
Bench Width (BW)		12.5 m *from max(8.5 ; 11.0 + 1.5)*
Inter-Ramp Angle (IRA)		48°

Notes [1] Probabilities given for selected BFA.
[2] Probabilities given for the major sets analysis.
[3] Based on the limiting spill length at the selected BFA either from planar or wedge sliding.

For Sector J-2 in the metasediments, a single discontinuity set (JS2) can cause issues from a planar-sliding perspective. The JS2 joint set is mainly composed of open and closed joints and has an average joint spacing of 5.75 m.

A PoF of 56.6% is reached at a BFA of 47°, which is over the 30% target. As shown in Figure 107, even at a BFA of 40°, the PoF reaches over 50%. Given the high joint spacing, it is appropriate to design the pit with a BFA of 75°. In addition, at a BFA of 75°, the evaluated risk (eq. 28) remains below 5%.

Good monitoring of slope performance is recommended in this area, but no limitation on the BFA is included in the design guidelines based on the analyses. For standardization, it was deemed acceptable to recommend a bench width of 12 m for this sector (instead of 12.5 m), which results in an IRA of 49° for this sector. This way, the design is consistent throughout the design sector J. Wedge sliding was not identified as limiting for this sector and lithology.

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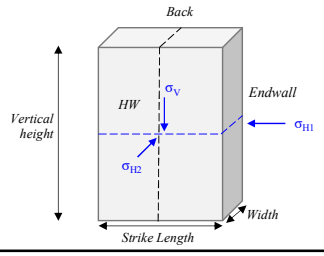
F Open Stope Stability and Dilution (ELOS) Analyses

Figures 108 to 117 present the Potvin open stope stability charts for the transverse stopes.

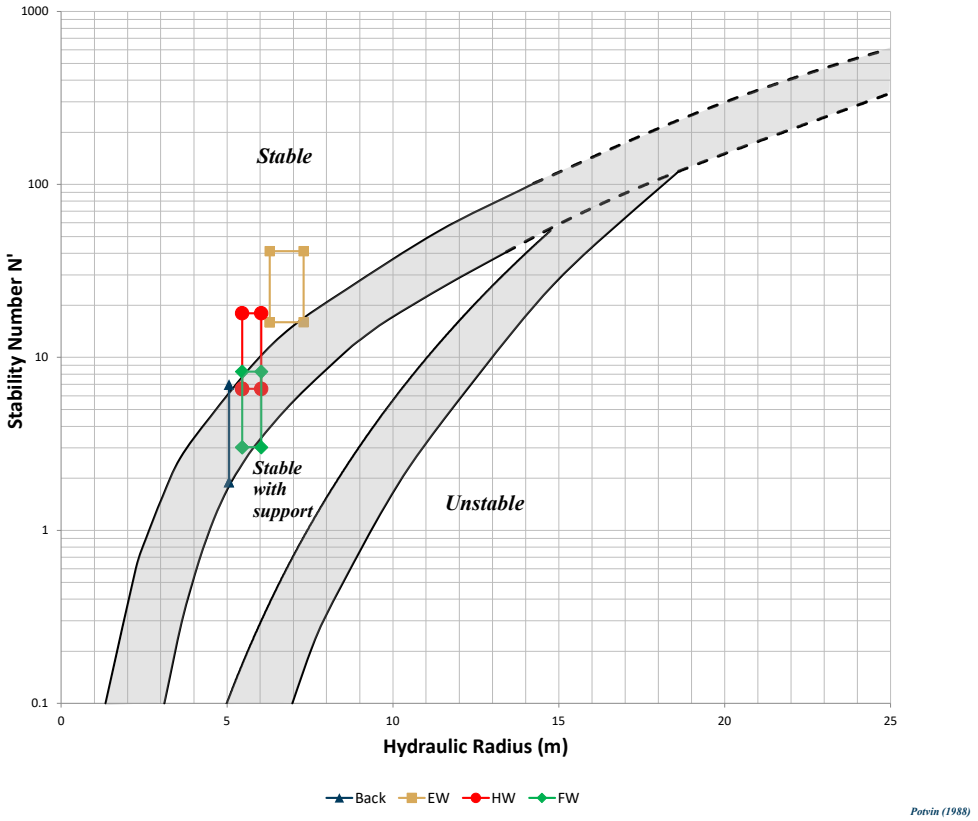
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	5.0	5.5	6.0	6.6	18.0
FW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	2.3	5.5	6.0	3.0	8.3

RESSOURCES PMET RESSOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Stope Design: Feasibility Study Design for HW=FW=PEG	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab V5 -17mLx25mW/[Case 1 - Alius v5 - Transverse - (PEG) 30-36m.xlsm]Inputs	1

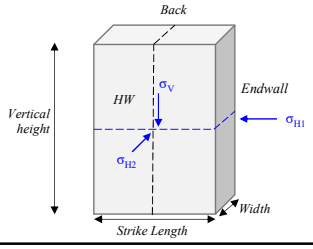
Figure 108. Potvin open stope stability chart – HW: pegmatite; FW: pegmatite.

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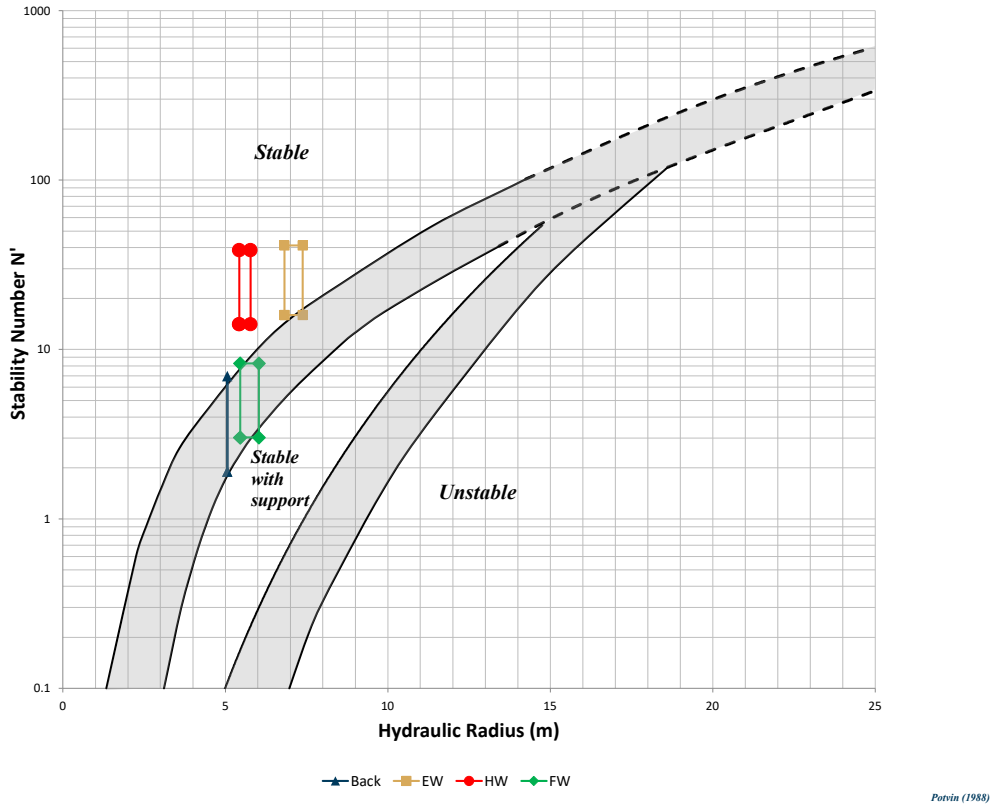
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	HW: 90°; FW:60-80°
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.3	8.0	5.4	5.8	14.1	38.6
FW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	2.3	5.5	6.0	3.0	8.3

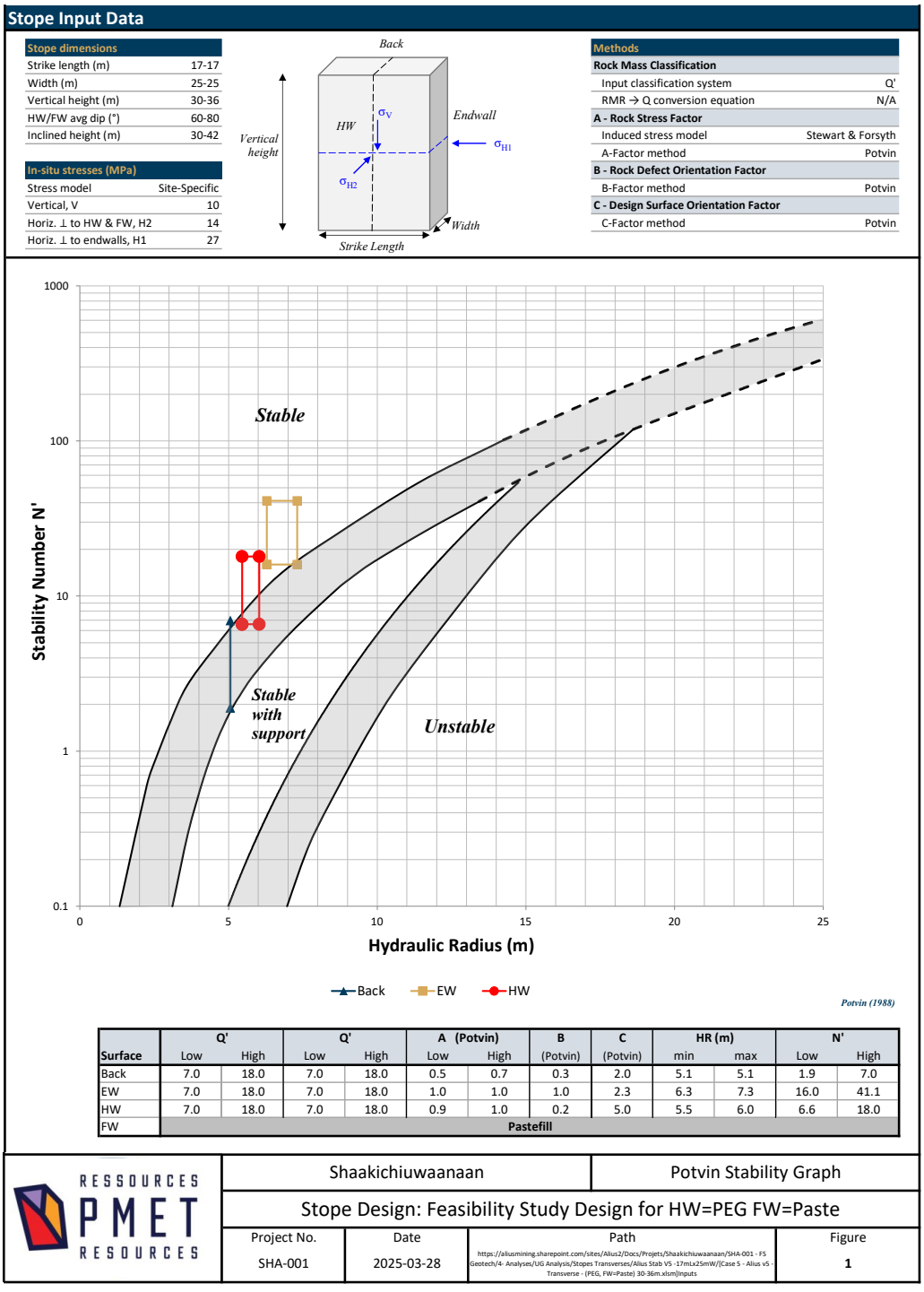
RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°), FW=PEG	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab V5 -17mLx25mW/[Case 3 - Alius v5 - Transverse - (PEG) 30-36m.xlsm]Mawdesley (Final)	1

Figure 109. Potvin open stope stability chart – HW: pegmatite (90°); FW: pegmatite.

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Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	5.0	5.5	6.0	6.6	18.0
FW	Pastefill											

Figure 110. Potvin open stope stability chart – HW: pegmatite; FW: backfill.

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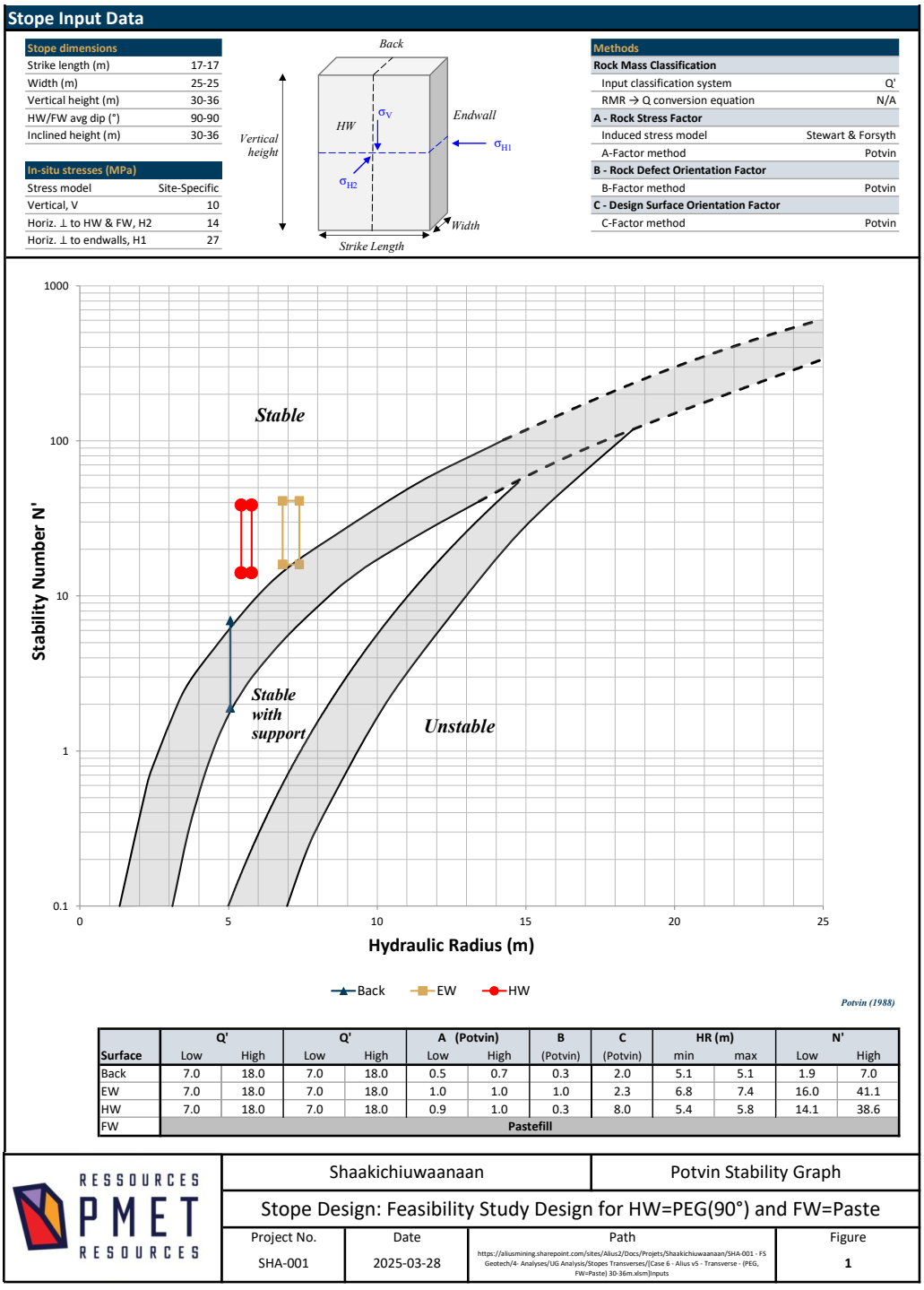


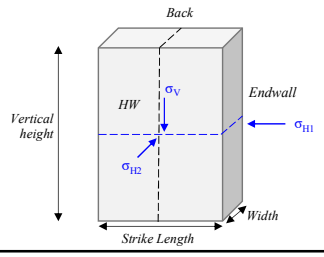
Figure 111. Potvin open stope stability chart – HW: pegmatite (90°); FW: backfill.

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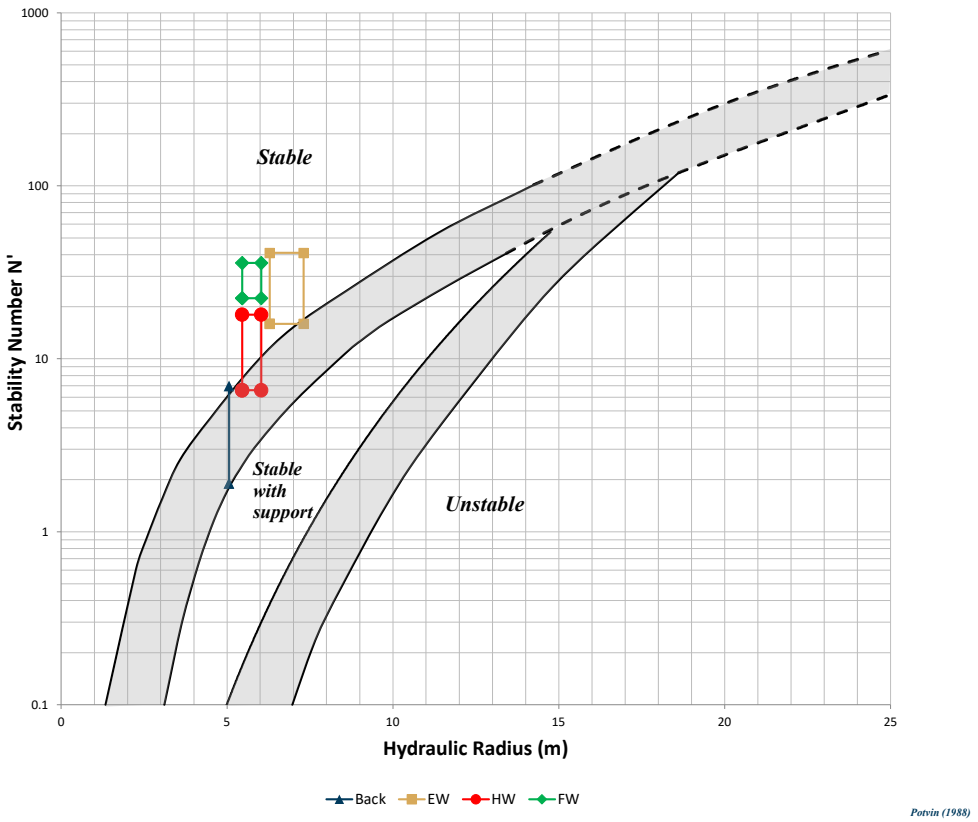
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	15.9	41.0
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	5.0	5.5	6.0	6.6	18.0
FW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	8.0	5.5	6.0	22.4	35.8

RESSOURCES PMET RESSOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Transverse Stope: Feasibility Study Design for HW=PEG FW=AMPorMETA	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab VS -17mLx25mW/[Case 7 - Alius vS - Transverse - (PEG, FW = AMP&META) 30-36m.xlsm]Inputs	1

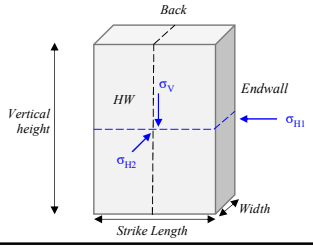
Figure 112. Potvin open stope stability chart – HW: pegmatite; FW: amphibolite or metasediments.

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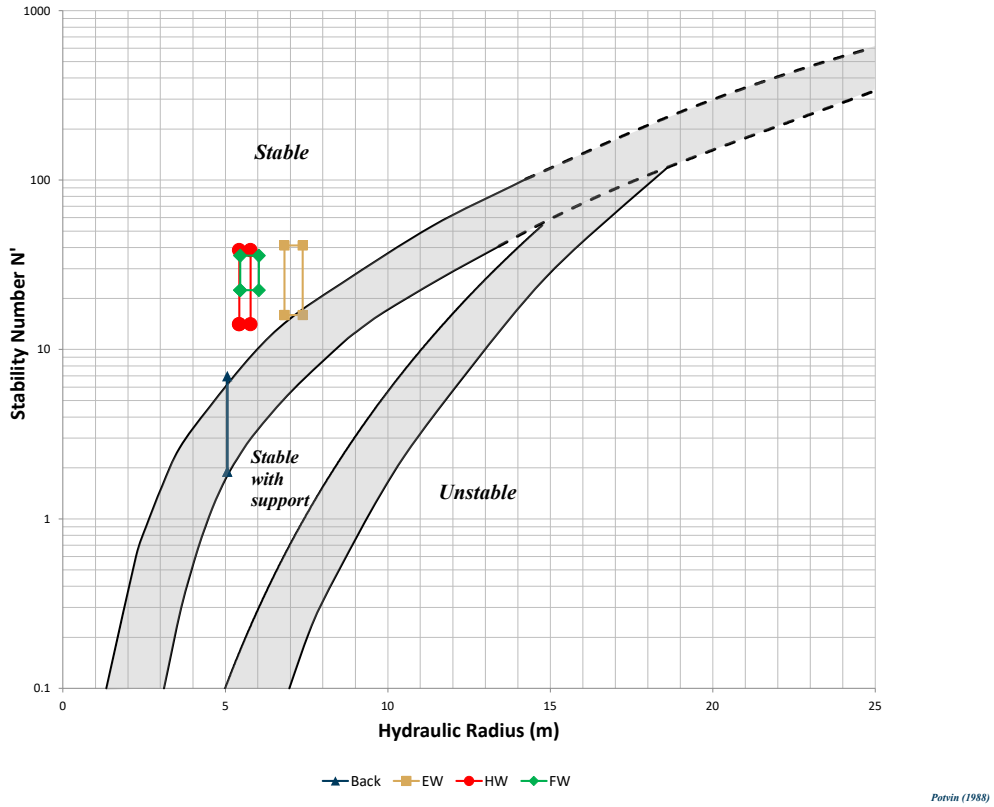
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	HW: 90°; FW:60-80°
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.3	8.0	5.4	5.8	14.1	38.6
FW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	8.0	5.5	6.0	22.4	35.8

RESSOURCES PMET RESSOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°), FW=AMPorMETA	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 8 - Alius v5 - Transverse - (PEG;FW=AMP) 30-36m.xlsm]Mawdesley (Final)	1

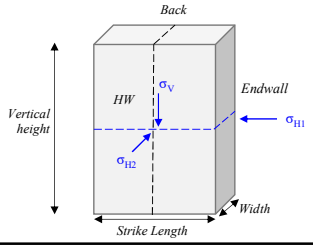
Figure 113. Potvin open stope stability chart – HW: pegmatite (90°); FW: amphibolite or metasediments.

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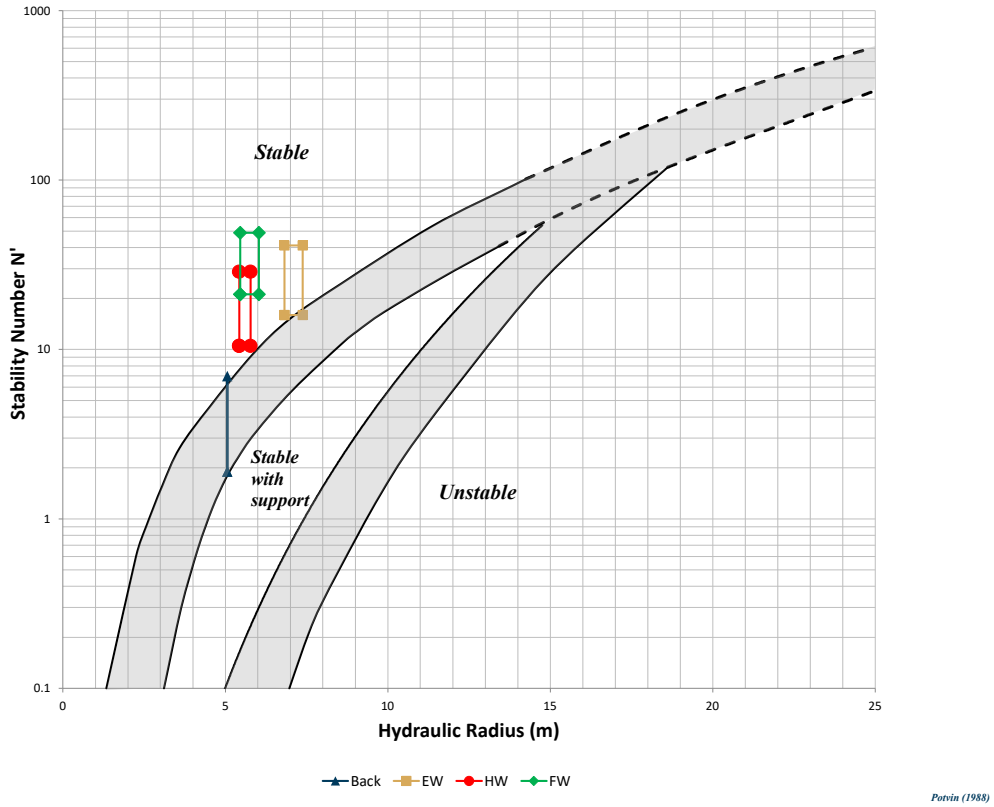
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	HW: 90°; FW:60-80°
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	8.0	5.4	5.8	10.5	28.8
FW	5.0	11.0	5.0	11.0	0.9	0.9	1.0	5.0	5.5	6.0	21.2	48.9

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°), FW=UMU	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 10 - Alius v5 - Transverse - (PEG90;FW=UMU) 30-36m.xlsm]Potvin (Final)	1

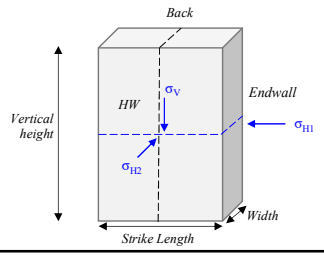
Figure 114. Potvin open stope stability chart – HW: pegmatite (90°); FW: ultramafic undifferentiated.

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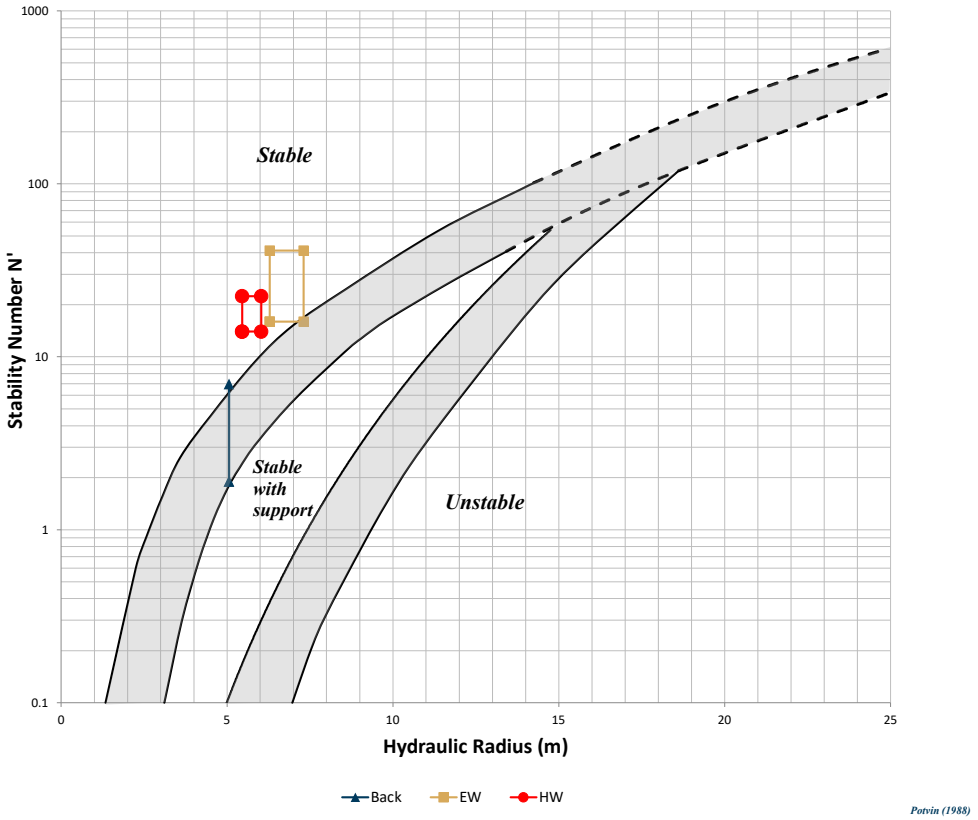
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	5.0	5.5	6.0	14.0	22.4
FW	Pastefill											

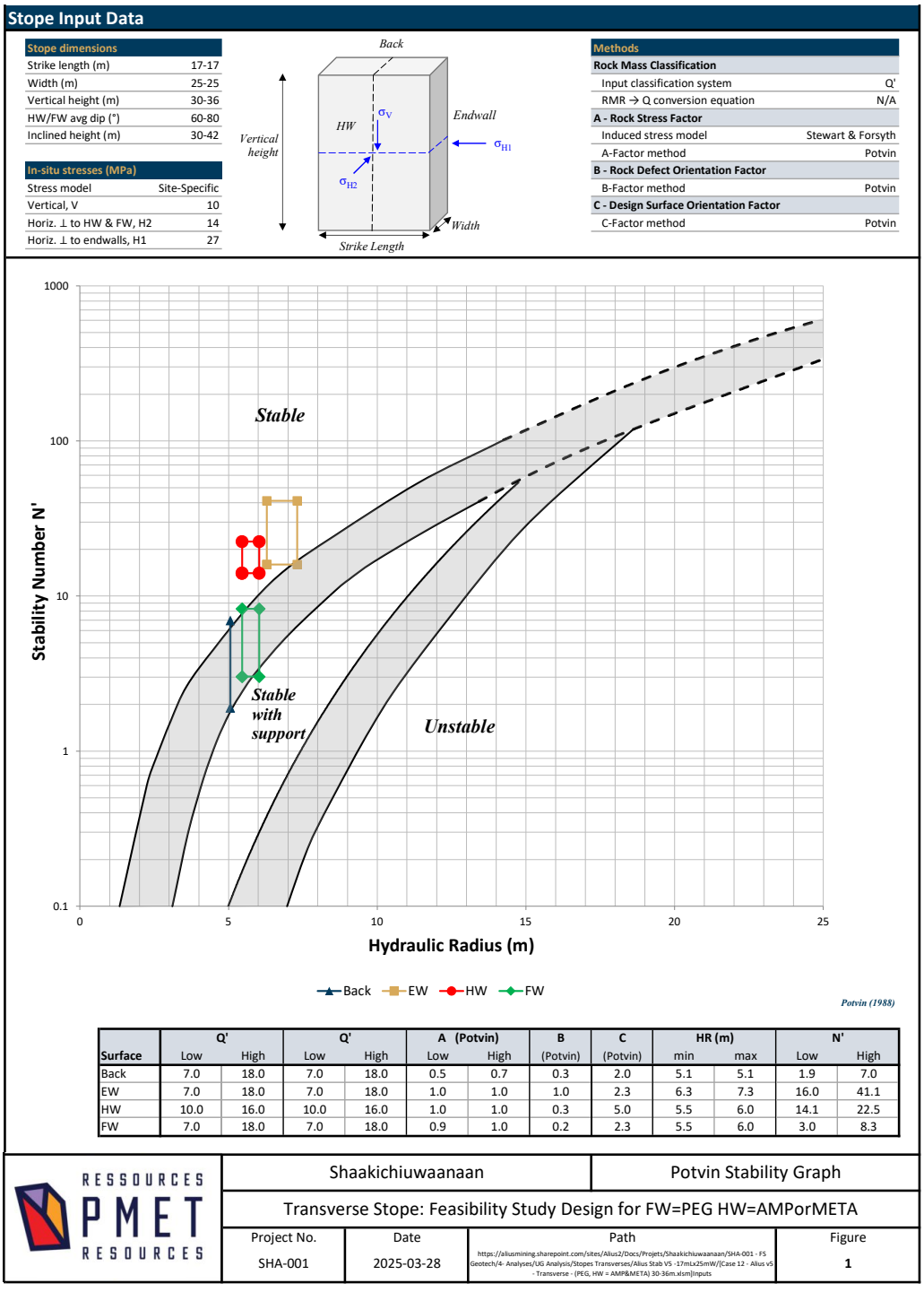
RESSOURCES PMET RESSOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Stope Design: Feasibility Study Stope Design for HW=AMP&META FW=Paste	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 11 - Alius v5 - Transverse - (PEG, HW=AMP&META, FW=Paste) 30-36m.xlsm]Inputs	1

Figure 115. Potvin open stope stability chart – HW: amphibolite or metasediments; FW: backfill.

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Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	5.0	5.5	6.0	14.1	22.5
FW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	2.3	5.5	6.0	3.0	8.3

Figure 116. Potvin open stope stability chart – HW: amphibolite or metasediments; FW: pegmatite.

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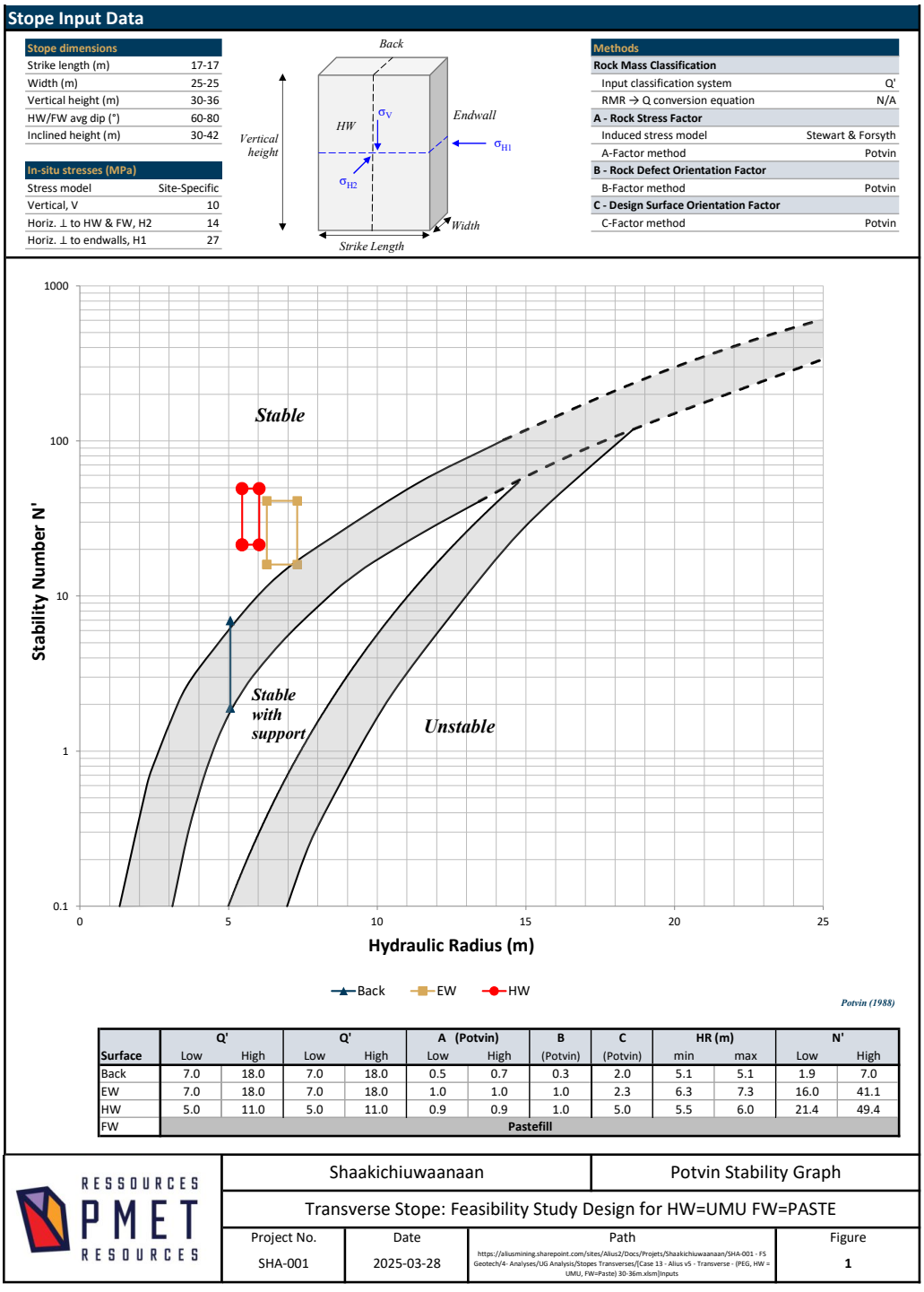


Figure 117. Potvin open stope stability chart – HW: ultramafic undifferentiated; FW: backfill.

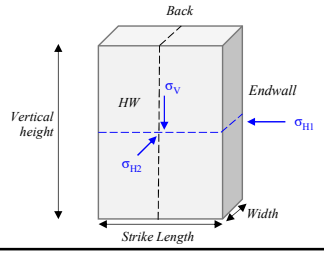
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The Figures 118 to 119 presents the open chart stability for the longitudinal stopes.

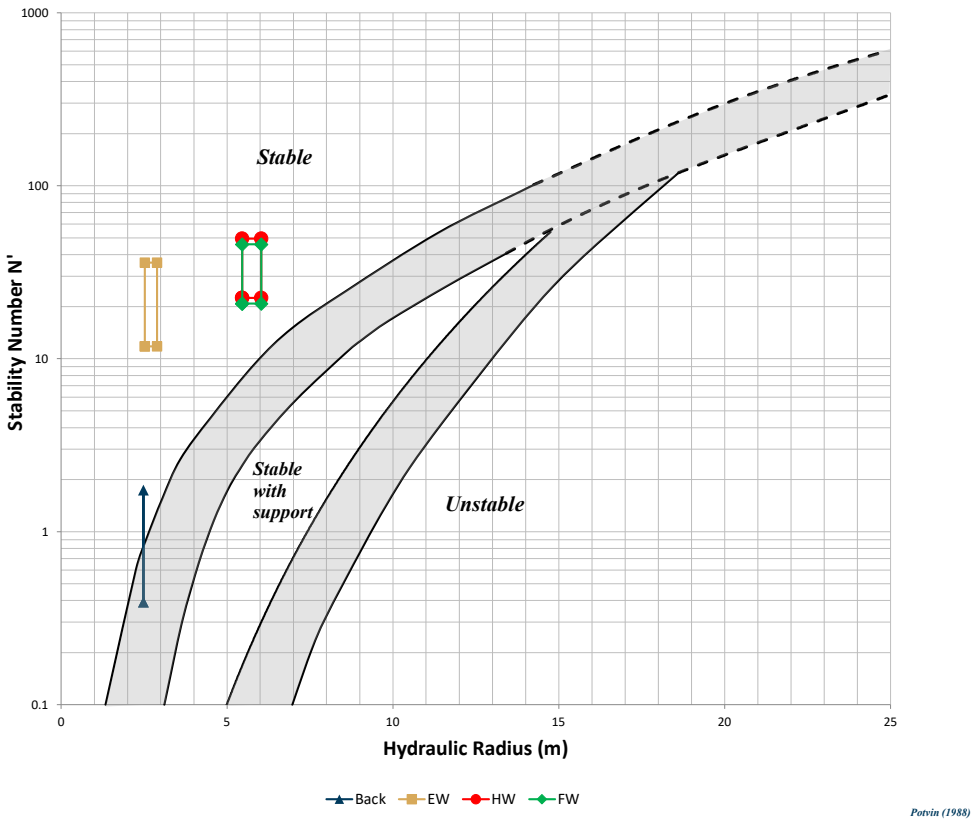
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	7-7
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	11
Horiz. ⊥ to HW & FW, H2	15
Horiz. ⊥ to endwalls, H1	29



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.1	0.2	0.3	2.0	2.5	2.5	0.4	1.7
EW	7.0	18.0	7.0	18.0	0.7	0.9	1.0	2.3	2.5	2.9	11.8	35.9
HW	5.0	11.0	5.0	11.0	1.0	1.0	0.8	5.4	5.5	6.0	22.5	49.5
FW	5.0	11.0	5.0	11.0	1.0	1.0	0.8	5.0	5.5	6.0	20.8	45.8

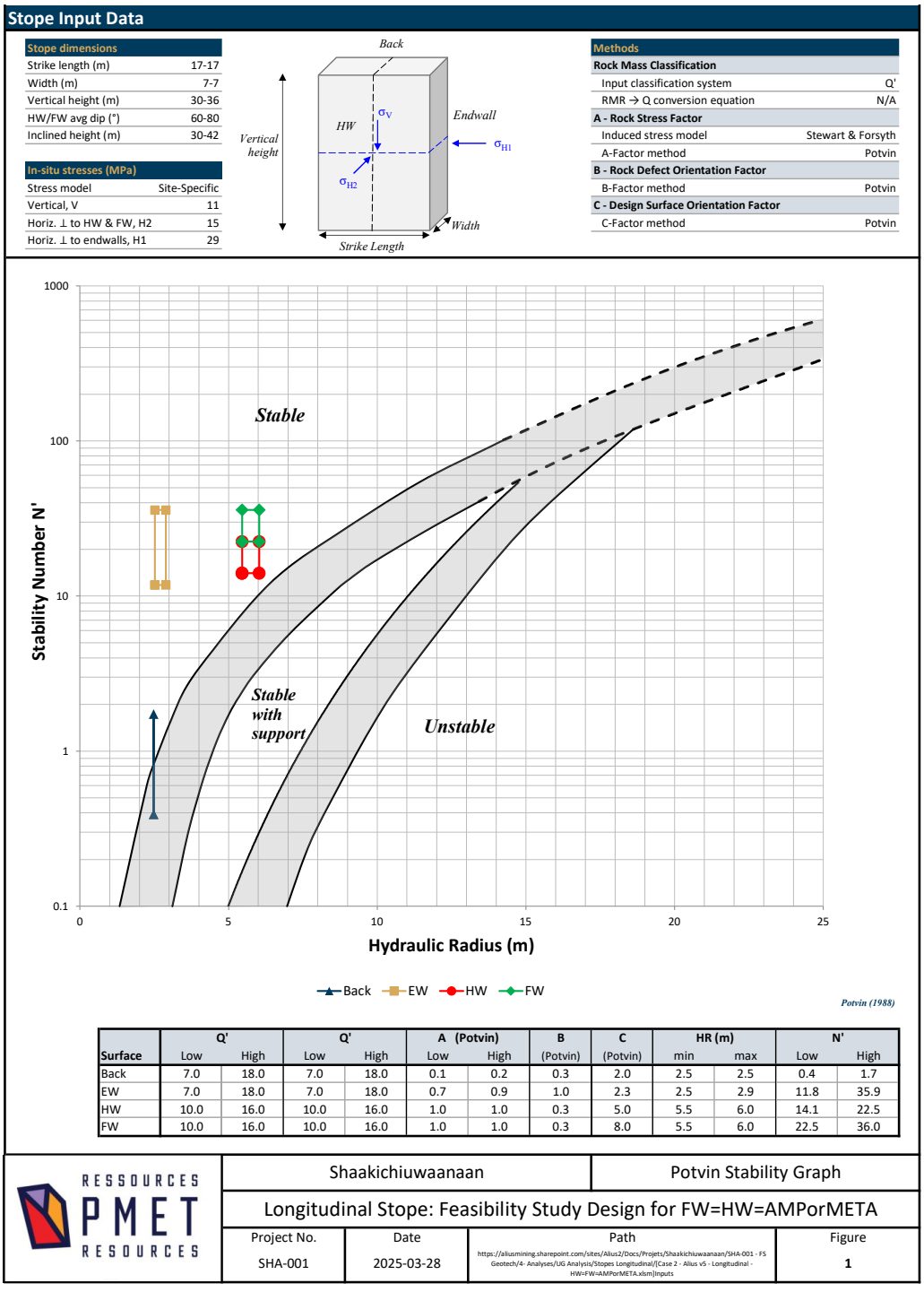
RESSOURCES PMET RESSOURCES

Shaakichiuwaanaan	Potvin Stability Graph
Longitudinal Stope: Feasibility Study Design for FS FW=HW=UMU	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Longitudinal/[Case 1 - Alius v5 - Longitudinal - HW=FW=UMU.xlsm]Inputs	1

Figure 118. Potvin open stope stability chart – HW: ultramafic undifferentiated; FW: ultramafic undifferentiated.

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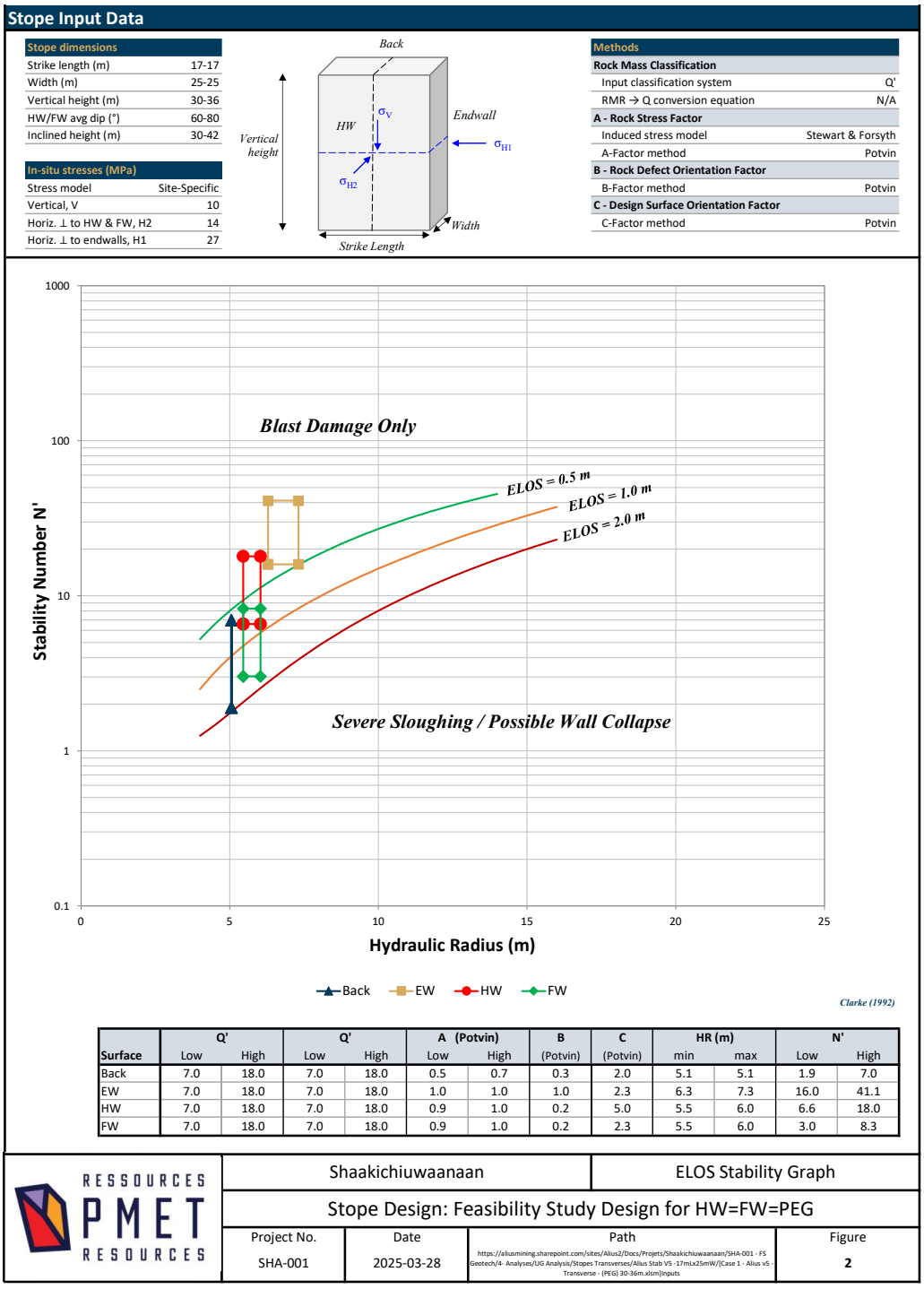


Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.1	0.2	0.3	2.0	2.5	2.5	0.4	1.7
EW	7.0	18.0	7.0	18.0	0.7	0.9	1.0	2.3	2.5	2.9	11.8	35.9
HW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	5.0	5.5	6.0	14.1	22.5
FW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	8.0	5.5	6.0	22.5	36.0

Figure 119. Potvin open stope stability chart – HW: amphibolite or metasediments; FW: amphibolite or metasediments.

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The Figures 120 to 129 present the ELOS chart for the transverse stopes.



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	5.0	5.5	6.0	6.6	18.0
FW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	2.3	5.5	6.0	3.0	8.3

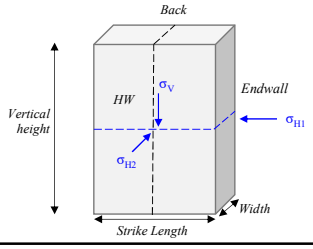
Figure 120. ELOS chart – HW: pegmatite; FW: pegmatite.

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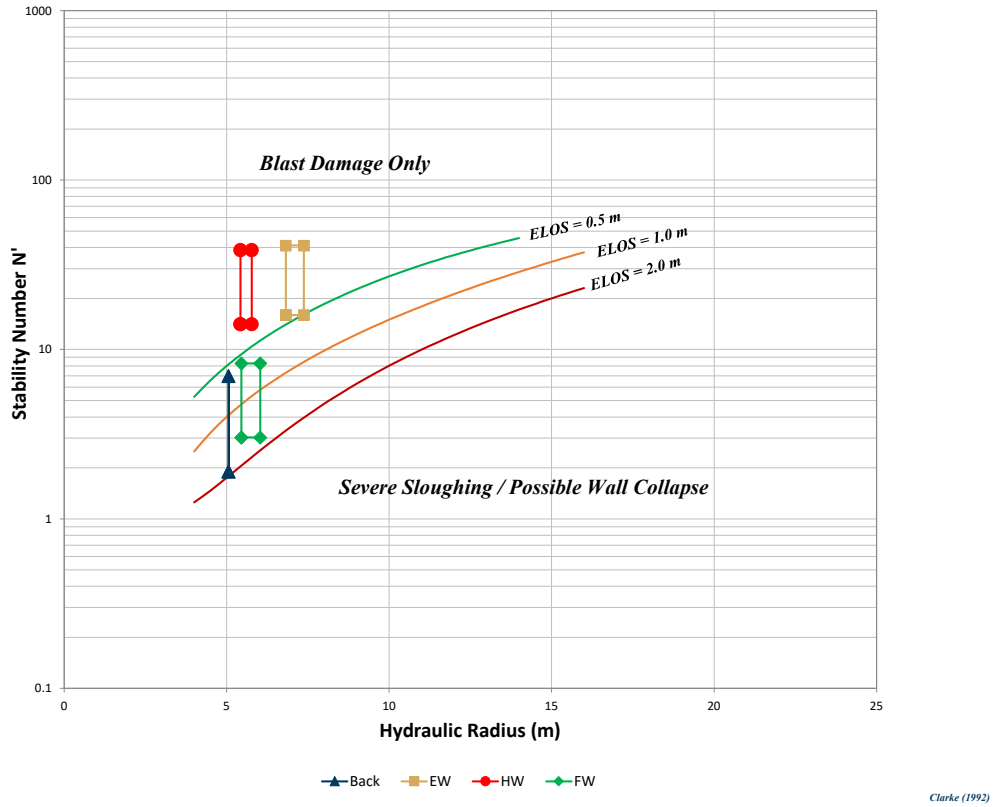
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	HW: 90°; FW:60-80°
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.3	8.0	5.4	5.8	14.1	38.6
FW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	2.3	5.5	6.0	3.0	8.3

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°), FW=PEG	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab V5 -17mLx25mW/[Case 3 - Alius v5 - Transverse - (PEG) 30-36m.xlsm]Mawdesley (Final)	2

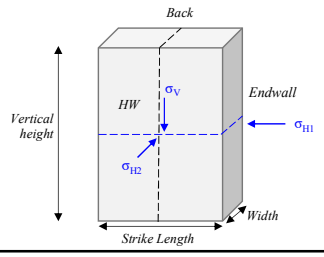
Figure 121. ELOS chart – HW: pegmatite (90°); FW: pegmatite.

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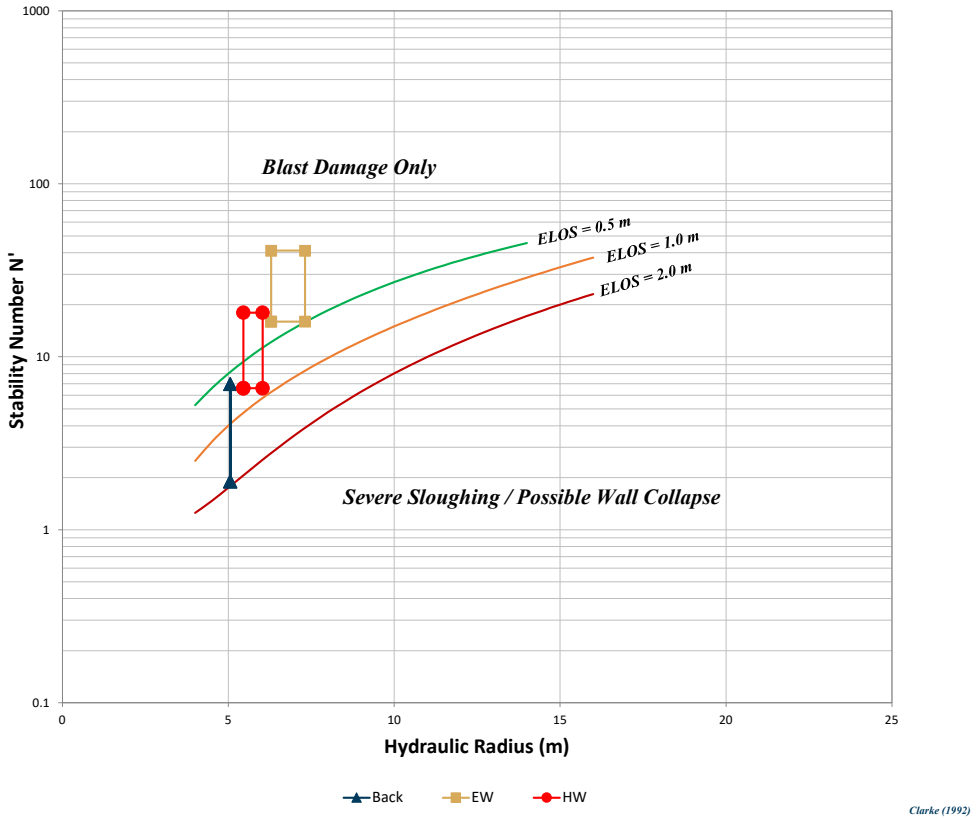
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	5.0	5.5	6.0	6.6	18.0
FW	Pastefill											

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Stope Design: Feasibility Study Design for HW=PEG FW=Paste	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab V5 -17mLx25mW/[Case 5 - Alius v5 - Transverse - (PEG, FW=Paste) 30-36m.xlsm]Inputs	2

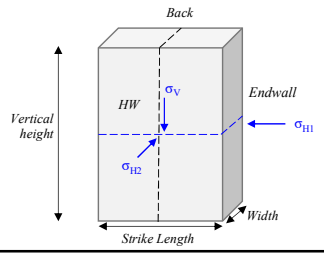
Figure 122. ELOS chart – HW: pegmatite; FW: backfill.

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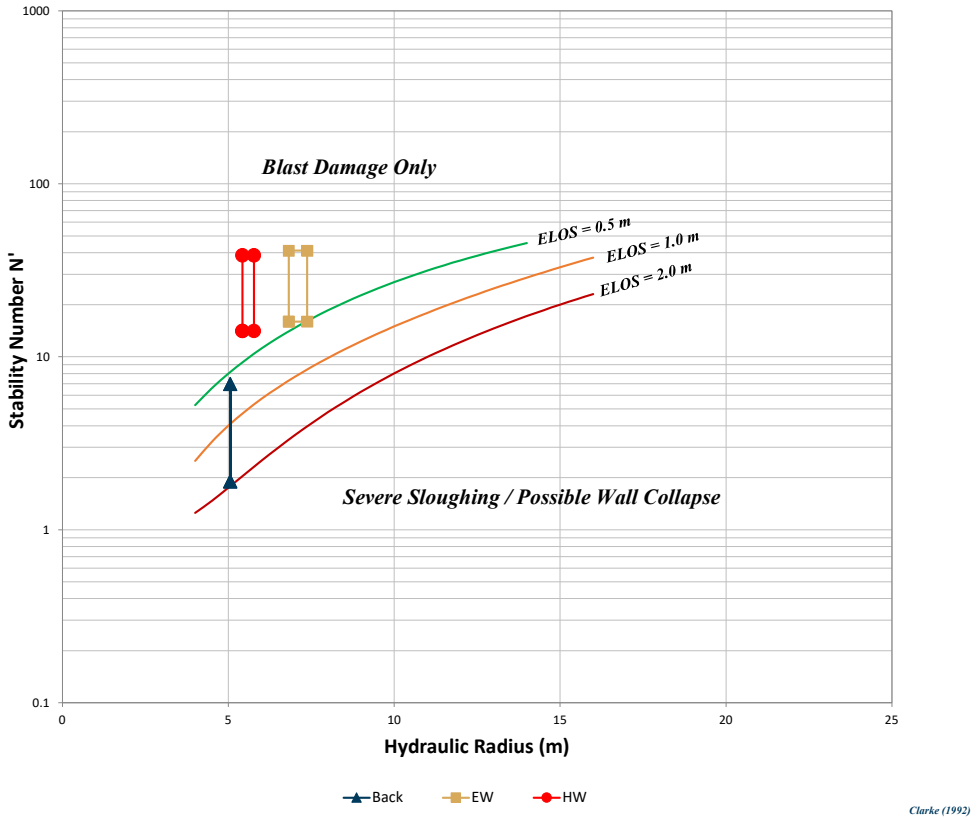
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	90-90
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.3	8.0	5.4	5.8	14.1	38.6
FW	Pastefill											

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°) and FW=Paste	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 6 - Alius v5 - Transverse - (PEG, FW=Paste) 30-36m.xlsm]Inputs	2

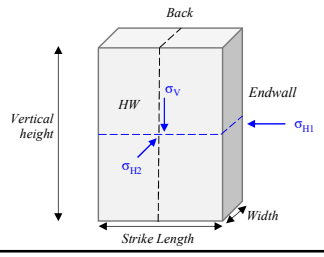
Figure 123. ELOS chart – HW: pegmatite (90°); FW: backfill.

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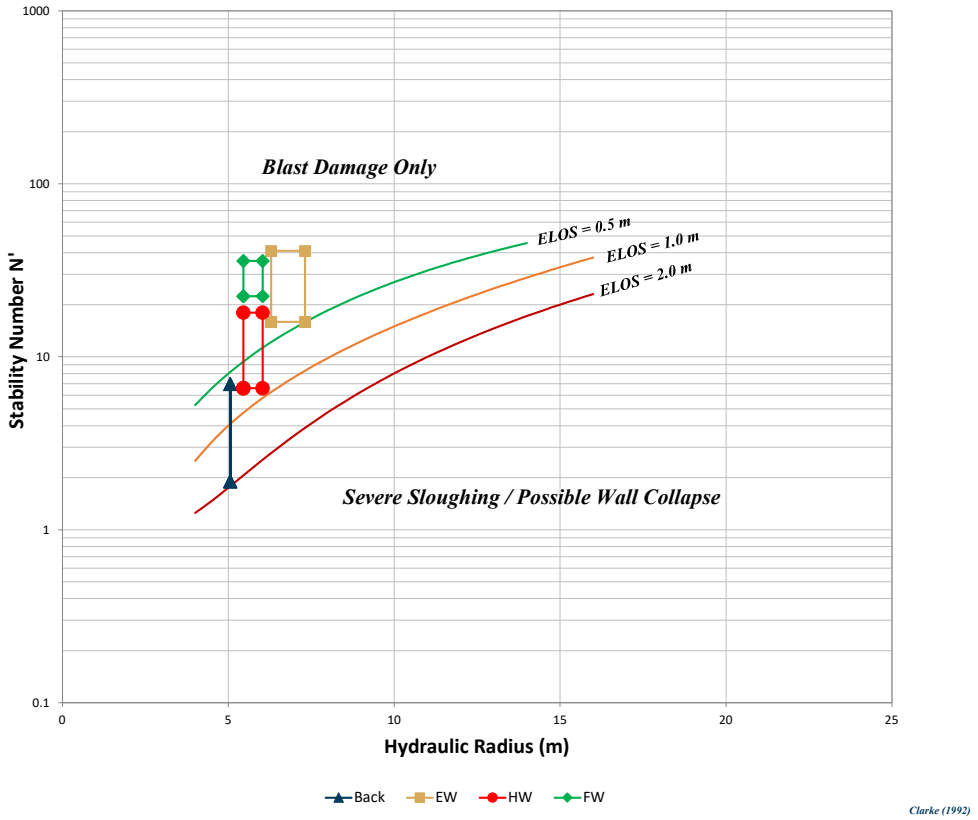
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	15.9	41.0
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	5.0	5.5	6.0	6.6	18.0
FW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	8.0	5.5	6.0	22.4	35.8

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Transverse Stope: Feasibility Study Design for HW=PEG FW=AMPorMETA	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab VS -17mLx25mW/[Case 7 - Alius vS - Transverse - (PEG, FW = AMP&META) 30-36m.xlsm]Inputs	2

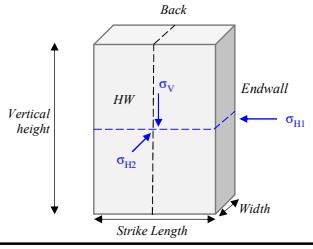
Figure 124. ELOS chart – HW: pegmatite; FW: amphibolite or metasediments.

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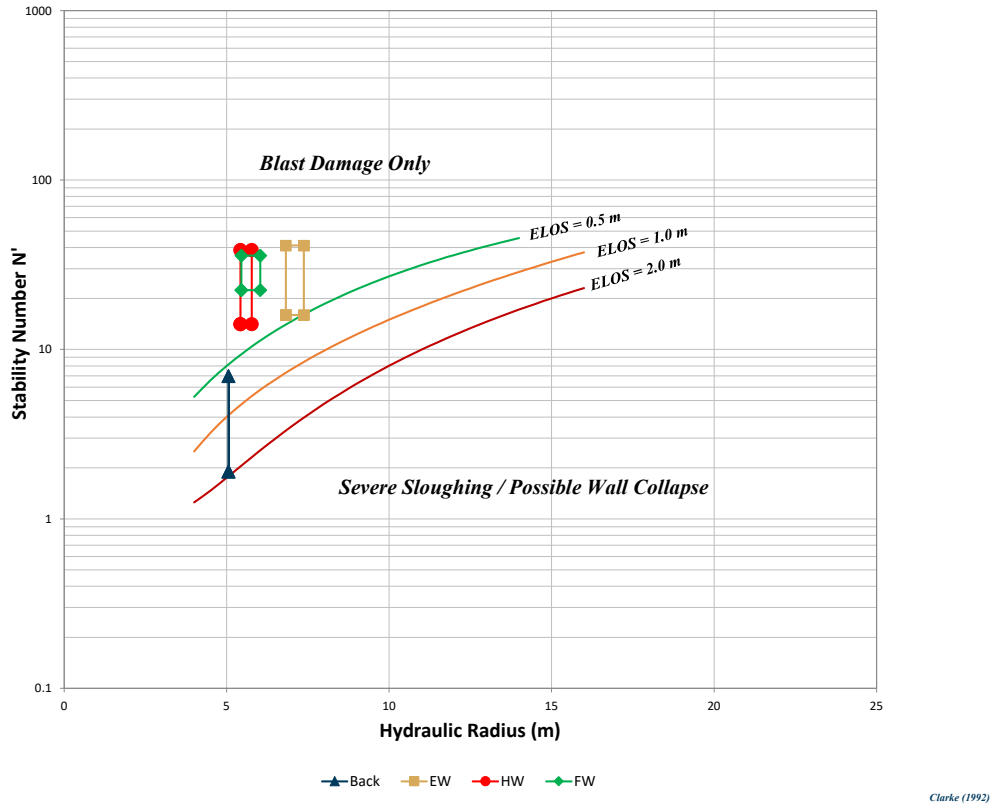
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	HW: 90°; FW:60-80°
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.3	8.0	5.4	5.8	14.1	38.6
FW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	8.0	5.5	6.0	22.4	35.8

RESSOURCES PMET RESSOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°), FW=AMPorMETA	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 8 - Alius v5 - Transverse - (PEG;FW=AMP) 30-36m.xlsm]Mawdesley (Final)	2

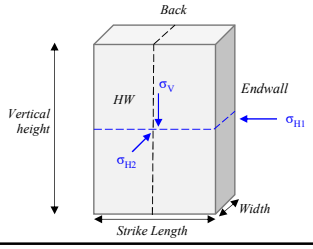
Figure 125. ELOS chart – HW: pegmatite (90°); FW: amphibolite or metasediments.

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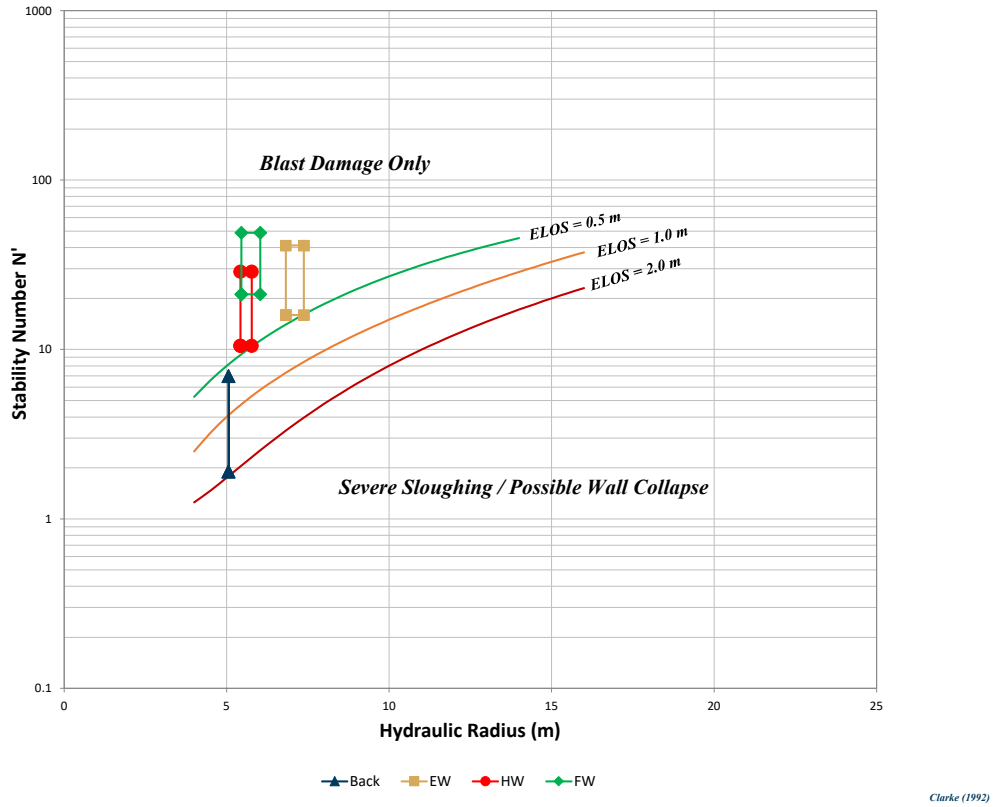
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	HW: 90°; FW:60-80°
Inclined height (m)	30-36

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.8	7.4	16.0	41.1
HW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	8.0	5.4	5.8	10.5	28.8
FW	5.0	11.0	5.0	11.0	0.9	0.9	1.0	5.0	5.5	6.0	21.2	48.9

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Stope Design: Feasibility Study Design for HW=PEG(90°), FW=UMU	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 10 - Alius v5 - Transverse - (PEG90;FW=UMU) 30-36m.xlsm]Potvin (Final)	2

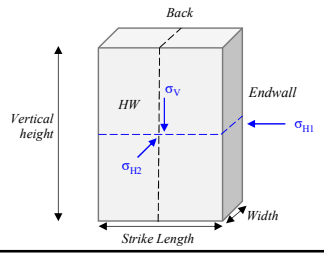
Figure 126. ELOS chart – HW: pegmatite (90°); FW: ultramafic undifferentiated.

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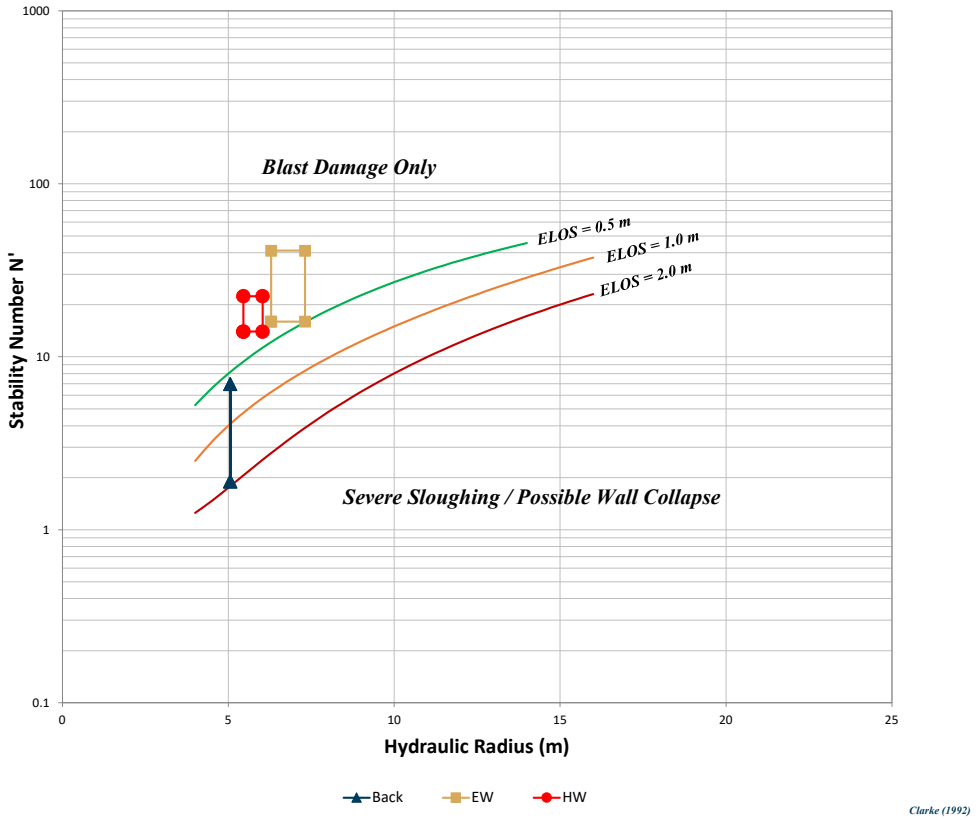
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	5.0	5.5	6.0	14.0	22.4
FW	Pastefill											

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Stope Design: Feasibility Study Stope Design for HW=AMP&META FW=Paste	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 11 - Alius v5 - Transverse - (PEG, HW=AMP&META, FW=Paste) 30-36m.xlsm]Inputs	2

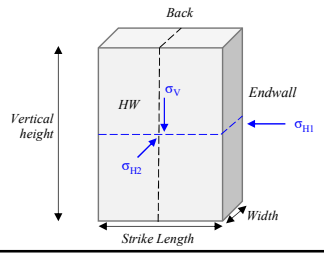
Figure 127. ELOS chart – HW: amphibolite or metasediments; FW: backfill.

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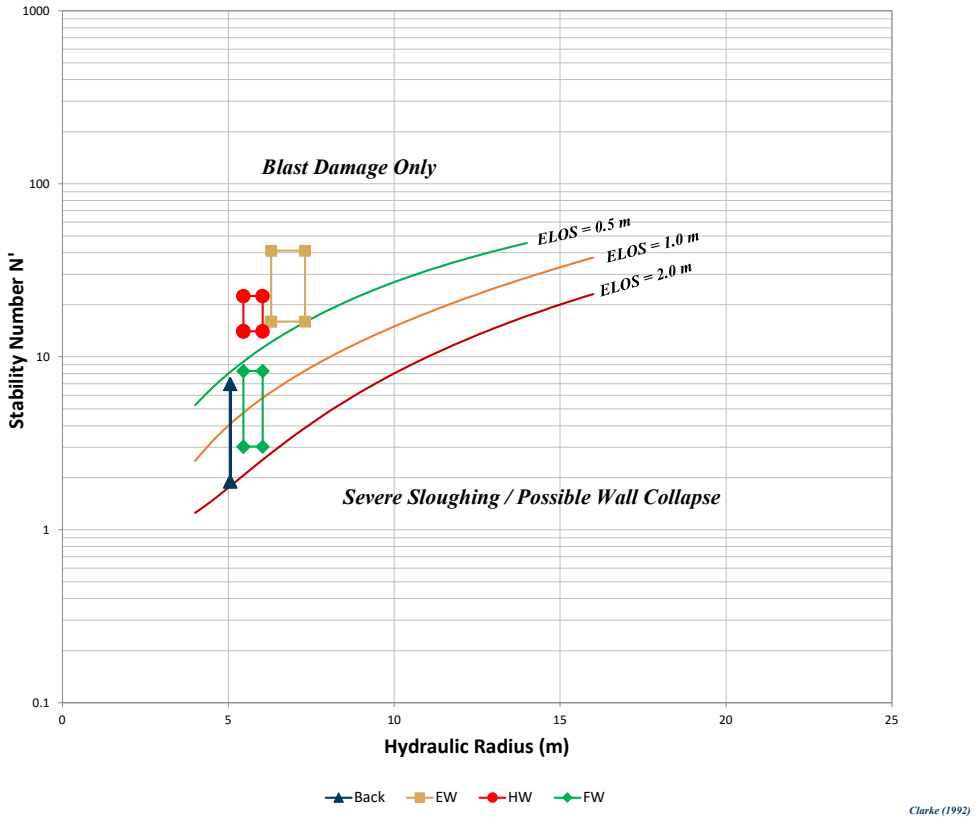
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	5.0	5.5	6.0	14.1	22.5
FW	7.0	18.0	7.0	18.0	0.9	1.0	0.2	2.3	5.5	6.0	3.0	8.3

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Transverse Stope: Feasibility Study Design for FW=PEG HW=AMPorMETA	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/Alius Stab V5 -17mLx25mW/[Case 12 - Alius v5 - Transverse - (PEG, HW = AMP&META) 30-36m.stsm]Inputs	2

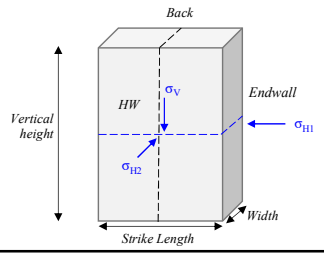
Figure 128. ELOS chart – HW: amphibolite or metasediments; FW: pegmatite.

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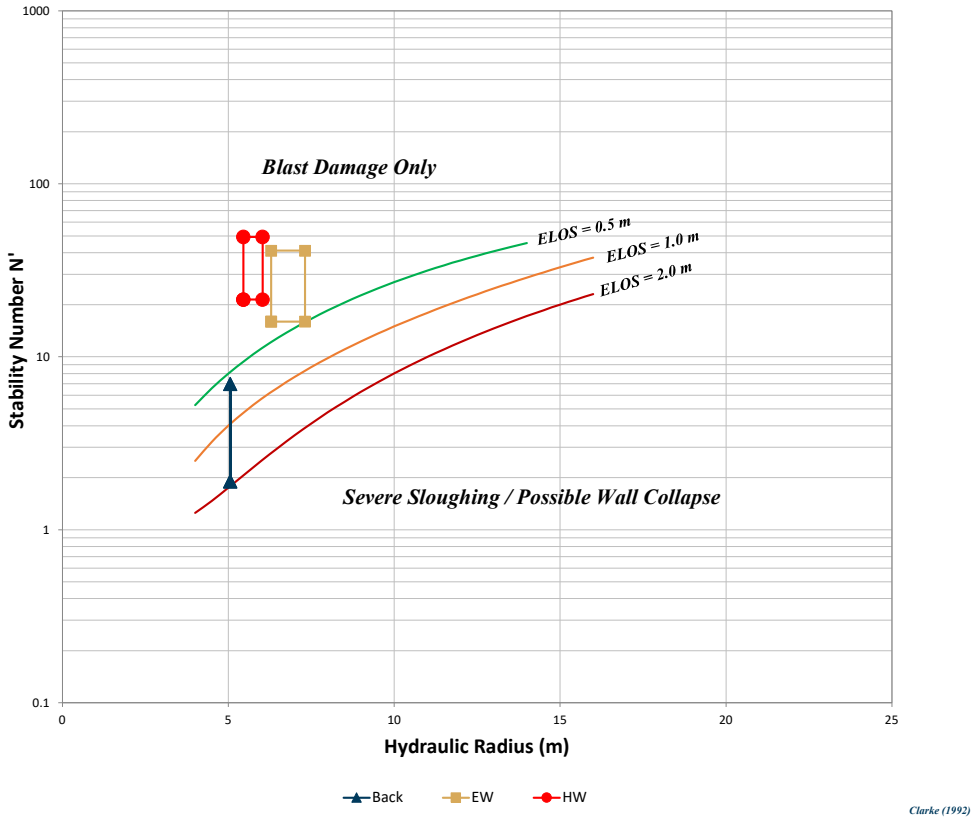
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	25-25
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	10
Horiz. ⊥ to HW & FW, H2	14
Horiz. ⊥ to endwalls, H1	27



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.5	0.7	0.3	2.0	5.1	5.1	1.9	7.0
EW	7.0	18.0	7.0	18.0	1.0	1.0	1.0	2.3	6.3	7.3	16.0	41.1
HW	5.0	11.0	5.0	11.0	0.9	0.9	1.0	5.0	5.5	6.0	21.4	49.4
FW	Pastefill											

RESSOURCES PMET RESSOURCES

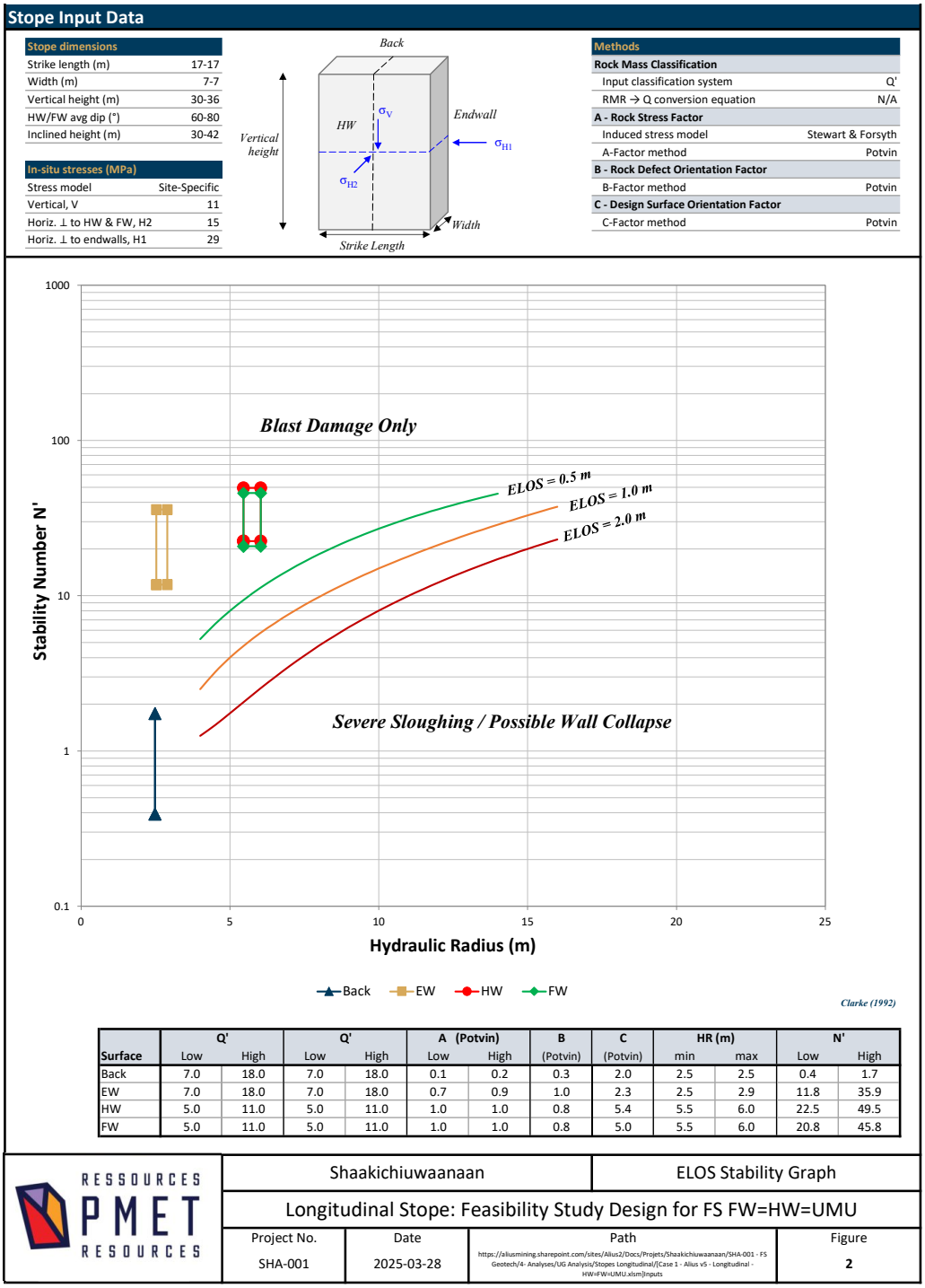
Shaakichiuwaanaan	ELOS Stability Graph
Transverse Stope: Feasibility Study Design for HW=UMU FW=PASTE	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Transverses/[Case 13 - Alius v5 - Transverse - (PEG, HW = UMU, FW=Paste) 30-36m.xlsm]Inputs	2

Figure 129. ELOS chart – HW: ultramafic undifferentiated; FW: backfill.

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The Figures 130 to 131 present the ELOS chart for the longitudinal stopes.



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.1	0.2	0.3	2.0	2.5	2.5	0.4	1.7
EW	7.0	18.0	7.0	18.0	0.7	0.9	1.0	2.3	2.5	2.9	11.8	35.9
HW	5.0	11.0	5.0	11.0	1.0	1.0	0.8	5.4	5.5	6.0	22.5	49.5
FW	5.0	11.0	5.0	11.0	1.0	1.0	0.8	5.0	5.5	6.0	20.8	45.8

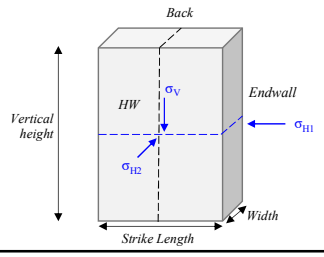
Figure 130. ELOS chart – HW: ultramafic undifferentiated; FW: ultramafic undifferentiated.

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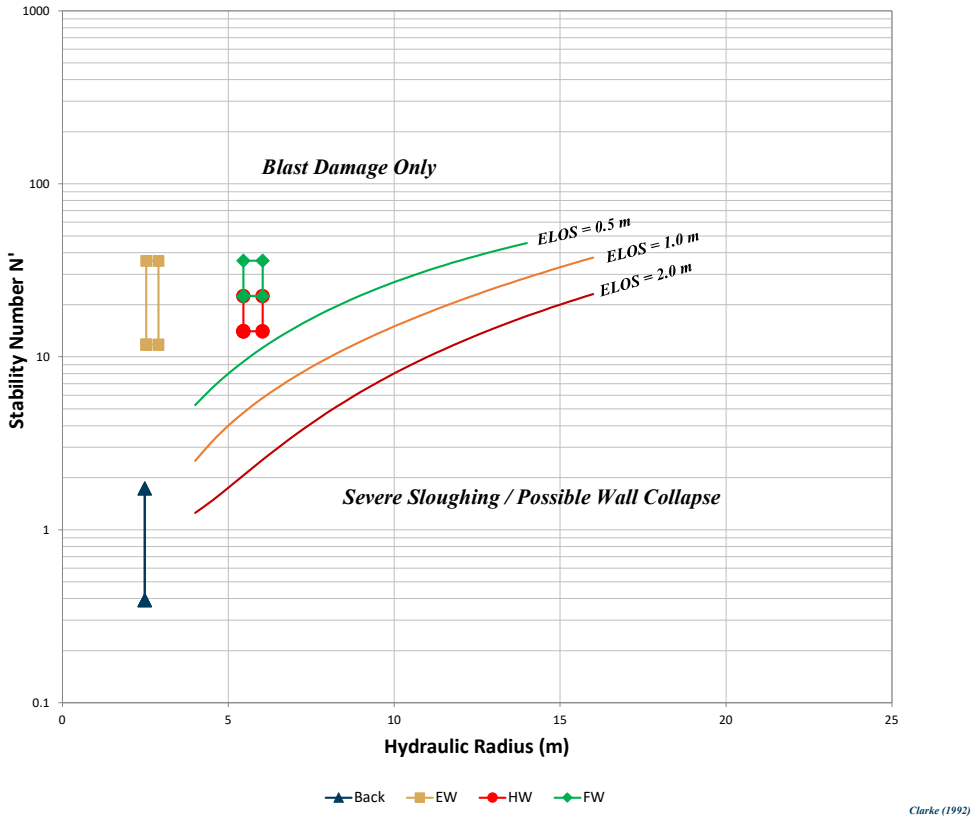
Stope Input Data

Stope dimensions	
Strike length (m)	17-17
Width (m)	7-7
Vertical height (m)	30-36
HW/FW avg dip (°)	60-80
Inclined height (m)	30-42

In-situ stresses (MPa)	
Stress model	Site-Specific
Vertical, V	11
Horiz. ⊥ to HW & FW, H2	15
Horiz. ⊥ to endwalls, H1	29



Methods	
Rock Mass Classification	
Input classification system	Q'
RMR → Q conversion equation	N/A
A - Rock Stress Factor	
Induced stress model	Stewart & Forsyth
A-Factor method	Potvin
B - Rock Defect Orientation Factor	
B-Factor method	Potvin
C - Design Surface Orientation Factor	
C-Factor method	Potvin



Surface	Q' Low	Q' High	Q' Low	Q' High	A (Potvin) Low	A (Potvin) High	B (Potvin)	C (Potvin)	HR (m) min	HR (m) max	N' Low	N' High
Back	7.0	18.0	7.0	18.0	0.1	0.2	0.3	2.0	2.5	2.5	0.4	1.7
EW	7.0	18.0	7.0	18.0	0.7	0.9	1.0	2.3	2.5	2.9	11.8	35.9
HW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	5.0	5.5	6.0	14.1	22.5
FW	10.0	16.0	10.0	16.0	1.0	1.0	0.3	8.0	5.5	6.0	22.5	36.0

RESSOURCES PMET RESOURCES

Shaakichiuwaanaan	ELOS Stability Graph
Longitudinal Stope: Feasibility Study Design for FW=HW=AMPorMETA	

Project No.	Date	Path	Figure
SHA-001	2025-03-28	https://aliusmining.sharepoint.com/sites/Alius2/Docs/Projets/Shaakichiuwaanaan/SHA-001 - FS Geotech/4- Analyses/UG Analysis/Stopes Longitudinal/[Case 2 - Alius v5 - Longitudinal - HW=FW=AMPorMETA.xlsm]Inputs	2

Figure 131. ELOS chart – HW: amphibolite or metasediments; FW: amphibolite or metasediments.

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G Cemented Backfill Strength Assessment

The modified exposed-backfill method proposed by Li and Aubertin (2012), and the generalized solution developed by Belem et al. (2022) are presented below based on the terminology used in Figure 132.

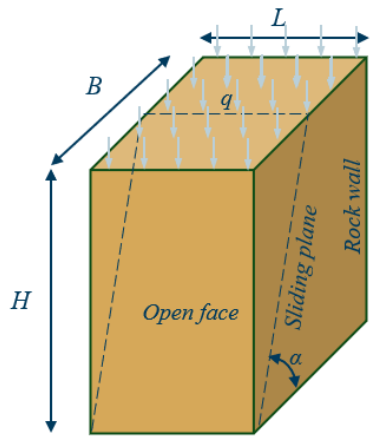


Figure 132. Stope configuration adopted for the backfill strength requirement assessment (reproduced from Li and Aubertin (2012)).

The method proposed by Li and Aubertin (2012) estimates the required cemented paste backfill strength for stopes where the backfill is exposed. The approach depends on two validity criteria based on the stope height to width ratio and the backfill slope:

$$Criterion: \frac{H}{B} \geq \tan\alpha \qquad \textit{(eq. 43)}$$

Where H = Stope height (m)
B = Stope width (m)
α = 45 + $\frac{\phi}{2}$ (°)
ϕ = Internal friction angle of backfill (°)

For this case, the required cohesion of the backfill is calculated using the following equation:

$$c = \frac{(p_0 + \gamma \cdot H^*)/2}{\left[\left(FoS - \frac{\tan\phi}{\tan\alpha}\right) \cdot \sin(2\alpha)\right]^{-1} + \frac{r_b \cdot H^*}{L}} \qquad \textit{(eq. 44)}$$

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Where c = Cohesion of backfill (kPa)
$H^* = H - \frac{B \tan \alpha}{2}$ is the equivalent height of the wedge block (m)
p_0 = Surcharge load on backfill (kN)
γ = Unit weight of backfill (kN/m^3)
FoS = Target factor of safety (–)
r_b = Adherence ratio of the fill–rock interface (–)
L = Stope length (m)

If the above criterion is not respected, the following criterion applies:

$$Criterion: \frac{H}{B} < \tan \alpha \quad \textit{(eq. 45)}$$

For this case, the cohesion is calculated using an alternative expression:

$$c = \frac{p_0 + (\gamma \cdot H)/2}{2\left[\left(FoS - \frac{\tan \phi}{\tan \alpha}\right) \cdot \sin(2\alpha)\right]^{-1} + \frac{r_b \cdot H}{L}} \quad \textit{(eq. 46)}$$

Once the cohesion is determined for either criterion, the required unconfined compressive strength of the backfill can be evaluated as:

$$\text{UCS} = 2\,c \cdot \tan\left(45 + \frac{\phi}{2}\right) \quad \textit{(eq. 47)}$$

Where UCS = Unconfined compressive strength of backfill (kPa)

The generalised solution of Belem et al. (2022) is applicable to all types of stopes. The required CPB strength is obtained with this simplified expression:

$$\text{UCS} = \frac{q \cdot S + \gamma \cdot H^* \cdot W}{F^{-1} + C + T} \quad \textit{(eq. 48)}$$

Where S = Surcharge load factor, eq. 49
W = Block weight factor, eq. 50
F = Safety factor, eq. 51
$C + T$ = Cohesion and tensile factor, eq. 52
q = Surcharge load on backfill (kPa)
γ = Unit weight of backfill (kPa/m)

The variable S, W, F and $C + T$ are defined below:

$$S = 1 - m \cdot f \quad \textit{(eq. 49)}$$

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Where m = Number of sidewall contacts (0, 1, or 2)
f = Coefficient, $f = \frac{H^*}{L \cdot (K \cdot \tan \delta)}$
K = Earth pressure coefficient (–)
δ = Internal friction angle of backfill (°)

$$W = 1 - 0.5\, m \cdot f \qquad \textit{(eq. 50)}$$

$$F = \left(\mathrm{FoS} \cdot \tan \alpha - \tan \phi'\right) \cdot \sin(2\alpha) \qquad \textit{(eq. 51)}$$

Where α = $\frac{\pi}{4} + \frac{\phi'}{2}$ (°)
ϕ' = Effective friction angle of backfill, $\phi' = \frac{\delta}{R_f}$
R_f = Reduction/degradation factor $\frac{\delta}{\phi'}$ (–)

$$C + T = \frac{R_c}{\tan \alpha} \left[0.5\, m \frac{H^*}{L} + \frac{n\, H'}{B} \cdot \tan \beta \right] \qquad \textit{(eq. 52)}$$

Where R_c = Adhesion-cohesion ratio, $R_c = \frac{C_b}{c'}$
C_b = Cohesion at the interface (°)
c' = Internal cohesion of backfill (°)
n = Number of backwall contacts (0 or 1)
$H' = H - (B \cdot \tan \alpha)$
$\beta = \frac{\pi}{4} + \frac{\delta}{2}$ (°)

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APPENDIX

5-2 ANALYSES AND INTERPRETATION OF THE CV5 STATIC AND KINETIC GEOCHEMISTRY CHARACTERIZATION RESULTS

Innova Lithium Inc.

Geochemical Characterization Results

Corvette Property, QC

Technical Report

Analysis and Interpretation of the CV5 Static and Kinetic Geochemistry Characterization Results

BBA Document No. / Rev.: 7535001-000500-4E-ERA-0001 / R00

May 23, 2024

FINAL

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BBA Document No. - Rev.: 7535001-000500-4E-ERA-0001-R00

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APPENDICES

Appendix A: Summary of Composite Sample Details
Appendix B: Lab Certificates
Appendix C: ABA Test Results
Appendix D: Leaching Test Results and Regulations
Appendix E: Kinetic Testing - Laboratory Analysis Results

1. Introduction

BBA has been retained by Innova Lithium Inc. for the initial environmental geochemical assessment of the mining materials, which will support the economic studies and environmental permitting activities for the Corvette Lithium Project.

The objective of this preliminary characterization study is to obtain information regarding the geochemical behaviour of waste rock, ore and tailings for the Corvette Lithium Project. While geochemical testing is a requirement for obtaining a mining permit, it also plays an essential role in planning the management of water, waste rock, ore and tailings. The results of geochemical characterization also have an impact on capital costs (CAPEX), operating costs (OPEX) and reclamation costs.

The following study has been aligned in accordance with the metal leaching and acid rock drainage (ML/ARD) guidelines from the Québec Ministry of the Environment-based Mining Wastes and Ore Characterization Guide (MELCC, 2020) and the Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials – Report 1.20.1 (MEND, 2009).

The geochemical assessment will also evaluate metal leaching due to neutral mine drainage (NMD) for mobile elements under near-neutral conditions, such as arsenic, considering NMD has been identified in similar LCT (Li-Cs-Ta) pegmatite lithium projects in both Ontario and Québec.

It should be noted that the Corvette Lithium Project does not have any defined "ore" at this time. However, the term ore is referred to as spodumene pegmatite throughout the report to separate the classification from waste rock.

2. Materials and Methods

2.1. Program Design

The geochemical characterization assessment was conducted as a two-phase program. This two-phased program was designed to serve as an iterative and evolutionary approach, allowing the Phase 2 sampling program to be adapted according to the results obtained during Phase 1.

- Phase 1 was designed for static geochemical characterization of important mine materials (waste rock, ore, preliminary DMS tailings) to provide initial characterization, understand the general elements of concern, and to understand geochemical variability.

- Phase 2 was designed based on the results of Phase 1, in which kinetic characterization using humidity cell tests (HCT) was implemented to better understand metal leaching, acid rock drainage potential and element mobility characteristics. The results obtained were used as inputs into the waste rock geochemical modelling and will also be used as inputs in future water quality modelling exercises.

2.2. Sampling Methodology

The sampling strategy that guided the Phase 1 program aimed to capture both geochemical variability and spatial variability by targeting the following aspects:

- Weighted sampling of all waste rock units based on approximate abundance of logged lithologies;
- Spatial variance along strike of mineralization as well as at depth;
- Proximal to the ore zone;
- Geochemical variance based on initial review/screening (As, Be, etc.);
- Variability in sulfide concentration.

Samples were selected from the drillhole database (DHDB) to form composite samples through the collection of 2 to 8 subsamples, ideally representing approximately 10 m length, although composite samples ranged from 2 to 12 m based on continuous available intersections. This level of resolution is appropriate for waste rock segregation as it is generally consistent with an approximate bench height in the future mine design for an open pit. It also helps to ensure spatial representation by reducing outlier intervals. A summary of the composite sample details (i.e., drillhole number, from (m), to (m), length (m), sample ID) are provided in Appendix A.

Samples selected for Phase 1 static testing were collected as pre-existing sample rejects located at SGS Labs in Burnaby, BC. The sample rejects were shipped to SGS Labs in Lakefield, ON and compositing was completed in-house by SGS. Samples selected for the Phase 2 kinetic testing were acquired from archived core samples and cut into ¼ core length samples and shipped to SGS for crushing and sieving to 6.3 mm.

2.3. Sample Inventory

A total of 84 samples were selected for the static program, including four generalized waste rock lithologies (namely amphibolite, metasediment, barren pegmatite and ultramafic), as well as one ore material (spodumene pegmatite), and one preliminary tailings material (DMS tailings). All

samples were submitted to SGS laboratories for acid-base accounting (ABA), extractible metals content and leaching tests (SPLP, CTEU-9, TCLP). A summary of the collected samples per material type and sampling zone is shown in Table 1.

Table 1: Summary of static test samples collected per lithology and associated material types

Lithology	Total number of samples	Material type
Amphibolite	30	Waste Rock
Metasediment	25	Waste Rock
Barren pegmatite	13	Waste Rock
Ultramafic	6	Waste Rock
Ore (spodumene pegmatite)	8	Ore
DMS Tailings	2	Tailings

2.4. Acid-base Accounting

Acid-base Accounting (ABA) comprises a set of laboratory test procedures, and data interpretation framework designed to determine if a material can be qualified as potentially acid-generating (PAG) or non-PAG (NPAG) by quantifying and comparing the neutralization potential (NP), and acidification potential (AP).

ABA was performed following the CEAEQ Method *MA.110 ACISOL1.0,* as recommended by the *Ministère de l'Environnement, de la Lutte contre le Changement Climatique, de la Faune et des Parcs* (MELCCFP) in the *Guide de caractérisation des résidus miniers et du minerai* (2020).

2.4.1. Neutralization Potential Assessment

The NP of a material, expressed in kg$CaCO_3$/tonne, generally depends on the content and availability of calcite ($CaCO_3$), and dolomite ($Ca,Mg(CO_3)_2$), as these minerals tend to dissolve rapidly and provide NP when exposed to acidity. In addition, certain silicate minerals may also account for a substantial proportion of the NP, although silicate minerals tend to have lower dissolution rates.

The NP of a material can either be measured directly through a laboratory procedure (referred to here as bulk-NP), or theoretically based only on the carbonate carbon content (CO_3-NP) of the sample.

Bulk-NP is estimated by acidifying a pulverized sample of solid material using an HCl solution in order to exhaust all of its neutralization potential, before titrating it back to a pH of 7 using a Na(OH) solution. The difference between the HCl and Na(OH) employed during the test can be used to derive the bulk-NP of the material, which includes the contribution of all neutralizing minerals (e.g., carbonates, and silicates).

CO_3-NP is a theoretical estimate based on the assumption that the NP of the material essentially relies on the presence of carbonates. The CO_3-NP can either be calculated using the carbonate content or the total carbon content, assuming that most of the carbon content of the sample is in the form of carbonates. The equation used to calculate CO_3-NP is as follows:

$$Carb\text{-}NP\ (kgCaCO_3/tonne) = \%C \times (M_{CaCO_3}/M_C) \times 10$$

Where $\%C$ is the percentage of carbonate carbon or total carbon, M_{CaCO_3} is the molar mass of carbonate (=100.086 g/mol), and M_C is the molar mass of carbon (=12.011 g/mol).

2.4.2. Acidification Potential Assessment

The acidification potential (AP) of samples can be estimated through theoretical calculation with the sulfide sulfur content, or the total sulfur content, assuming that it is mainly present in the form of sulfides similar to pyrite. The equation used for the calculation of AP is as follows:

$$AP\ (kgCaCO_3/tonne) = \%S \times (M_{CaCO_3}/M_S) \times 10$$

Where $\%S$ is the sulfide sulfur content or total sulfur content, and M_S the molar mass of sulfur (=32.065 g/mol).

2.5. Extractible Metals Content

Extractible (or environmentally available) metals content was measured in accordance with CEAEQ Method MA. 200 - Mét. 1.2. Aqua Regia extracts (partial digestion) were analyzed via ICP-MS for major, minor, and trace elements. This extraction method is designed to provide an estimate of the elements that can be leached over a long time period through natural processes such as dissolution and oxidation. Aqua Regia extract contents were compared to average concentrations of the continental crust.

It should be noted that average concentrations of the continental crust refer to total content (total digestion) and extractible contents refer to partial content (partial digestion) and, therefore, direct comparisons should be treated with caution.

However, extractible contents can be compared with soil Criteria A presented in MELCCFP (2021) "*Guide d'intervention – Protection des sols et rehabilitation des terrains contaminés*".

2.6. Leaching Tests

As required in MELCCFP guidelines, samples were subjected to CEAEQ Method MA. 100 – Lix.Com. 1.1 recommendations, a portion of the selected sample was subjected to a series of leaching procedures to assess if the tested materials are susceptible to leaching compounds of potential concern (COPC) into the receiving environment. All samples were subjected to Synthetic Precipitation Leaching Procedure (SPLP) and CTEU-9 leaching procedure. Samples were also subjected to the Toxicity Characteristic Leaching Procedure (TCLP).

The SPLP, CTEU-9 and TCLP procedures are designed to evaluate the concentration of species that could be potentially leached when a material is put in contact with water. The SPLP method employs acidic water at a pH of 4.2 in order to simulate mildly-acidic environmental conditions such as acid rain, whereas CTEU-9 simply involves de-ionized water and the material is crushed to a finer grain size, exposing more surface area. In TCLP procedure, buffered addition of acetic acid is used in order to simulate behavior of materials when disposed in organic containing landfill.

All leaching test extracts were subjected to ICP-MS analysis of major, minor and trace elements. Additionally, TCLP, SPLP and CTEU-9 leachates were subjected to fluoride (F^-), sulfate (SO_4^{2-}), chloride (Cl^-), bromide (Br^-), nitrite (NO_2^-), and nitrate (NO_3^-) measurements compounds characterization through ionic chromatography, as well as acidity/alkalinity titration analysis, and pH and electric conductivity (EC) measurements.

2.7. Classification Criteria

2.7.1. Potentially Acid Generating (PAG) Material Classification

Samples with total sulfur greater than 5% are de facto qualified as PAG, while samples with less than 0.04% total sulfur are classified as NPAG based on the *Guide de caractérisation des résidus miniers et du minerai* (2020). All intermediate samples are qualified as PAG if the net neutralization potential (NNP = NP-AP) is below 20, or if the neutralization potential ratio (NPR = NP/AP) is below 2. However, samples classified as PAG based on static testing can be requalified as NPAG on the basis of kinetic test results. The screening criteria for ABA are summarized in Table 2.

Table 2: Classification criteria of PAG and NPAG samples and required future steps based on total sulfur content, NPR and NNP

Parameter	Criteria	Acid generating potential	Procedure
Total S	<0.04%	NPAG	No further testing
Total S	>0.04%	Further testing required	Static ARD prediction tests
Total S	>5%	Further testing required	▪ De facto PAG; or ▪ Kinetic testing to validate
NNP	<20	PAG	If one of NNP or NPR is below the regulatory value, then sample is PAG
NPR	<2	PAG	

2.7.2. Metal(loid) Leaching Potential

The results of the leaching tests conducted on the selected samples were evaluated with regards to the MELCCFP guidelines.

In order to highlight potential metal(loid) enrichments of the materials, the Aqua Regia leach test results were compared to the average continental crust concentration (Rudnick and Gao, 2014) as well as to the background contents of the Superior geological province provided in Annex 1 of the *Guide d'intervention - protection des sols et réhabilitation des terrains contaminés* of the MELCCFP.

The SPLP and CTEU-9 leach test results were compared to the*Critère de qualité pour la protection de la vie aquatique – effet aigu* (CVAA) of Critère de qualité des eaux de surface (CQES) provided by the MELCCFP. Criteria for certain elements (e.g., cadmium (Cd), manganese (Mn)) depend on the hardness of the receiving water system, whereas for aluminium (Al) it also depends on the dissolved organic carbon (DOC) concentration and on the pH. The criteria were calculated assuming a DOC concentration of 10 mg/L, a pH of 6, and a hardness of 10 mg/L.

The TCLP leach test results were compared to the Directive 019 – Annex 2 on the characteristics of (high risk) mine residues.

2.8. Kinetic Testing

A set of 17 samples (+1 duplicate) were subjected to kinetic testing through a humidity cell test (HCT) procedure (Table 3).

The HCT samples were selected on the basis of static test results in order to include all lithology and material types, and to cover a wide range of behaviours in terms of acid generation and metal leaching potential, primarily to assess relation between arsenic content and leachability.

Table 3: Inventory of the conducted HCTs and corresponding lithology or material types

HCT test ID	Lithology or material type
CV5-WR-HCT-2	Amphibolite
CV5-WR-HCT-5	Amphibolite
CV5-WR-HCT-5 (DUP)	Amphibolite
CV5-WR-HCT-14	Amphibolite
CV5-WR-HCT-22	Amphibolite
CV5-WR-HCT-33	Amphibolite
CV5-WR-HCT-36	Ultramafic
CV5-WR-HCT-39	Metasediment
CV5-WR-HCT-50	Metasediment
CV5-WR-HCT-56	Metasediment
CV5-WR-HCT-59	Ultramafic
CV5-WR-HCT-62	Ultramafic
CV5-WR-HCT-64	Barren pegmatite
CV5-WR-HCT-69	Barren pegmatite
CV5-WR-HCT-74	Ore
CV5-WR-HCT-76	Ore
CV5-DMS-HCT-83	DMS tailings
CV5-DMS-HCT-84	DMS tailings

The testing procedure (ASTM D5744-18) was conducted by SGS, and consisted of placing 1 kg of dried material from each sample inside a humidity cell, and proceeding to weekly flushing cycles using 1 litre of deionized water. The waste rock and ore materials were crushed down to a particle size fraction of 6.3 mm, while DMS tailings were placed without additional processing.

The leachates resulting from the weekly flushing were sampled and subjected to a series of analysis. These included pH and electric conductivity (EC) measurements, acidity/alkalinity titration analysis, ICP-MS analysis of major, minor and trace elements, as well as fluoride (F^-), sulfate (SO_4^{2-}), chloride (Cl^-), bromide (Br^-), nitrite (NO_2^-) and nitrate (NO_3^-) measurements through ionic chromatography.

This report presents the results of the first 40 weeks of HCT; however the tests are still ongoing as the report is being prepared.

3. Results and Interpretation

A static test analysis comprising ABA and leaching test procedures was carried out in order to determine whether the investigated materials may generate acidity, leach COPCs, and potentially pose a change to the water quality of the receiving environment. The certificates of analysis for all tests conducted on the samples are presented in Appendix B.

3.1. Acid-Base Accounting

ABA tests were carried out on the 84 collected samples in order to determine their AP and NP and identify materials that can be identified as PAG. Tables summarizing the ABA results are provided in Appendix C.

3.1.1. Sulfur Speciation Analysis

Figure 1 displays the correlation between total sulfur concentration and sulfide sulfur. The regression analysis between total sulfur and sulfide sulfur has an $R^2 = 1.00$, and a slope of 0.94, indicating a good agreement between sulfide sulfur and total sulfur contents. This result suggests that most of the sulfur in the samples is likely present as sulfide and, consequently, that total sulfur contents are an appropriate approximation of sulfide sulfur.

However, 35 samples of all samples (42%) are below instrument detection limit, and are plotted at their detection limit, thus artificially increasing the sulfide sulfur to total sulfur ratio. Additionally four samples (two amphibolite, and two metasediment samples) stand out with high total sulfur contents (>3.5%) and sulfide to total sulfur ratios close to 1.0. Because of their offset position, these outliers tend to raise the slope closer to 1.

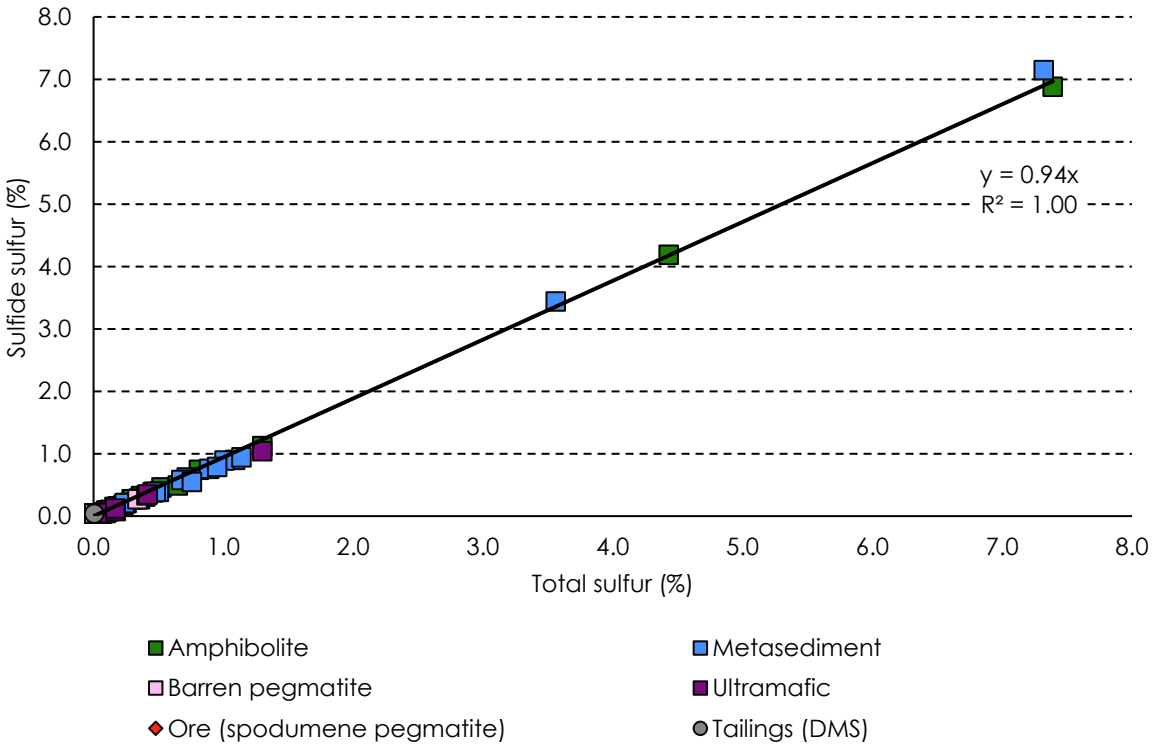


Figure 1: Relationship between sulfide sulfur and total sulfur

The acid leachable sulfur (or sulfate) plotted as a function of total sulfur presents a regression line slope of 0.17, and a correlation of R^2=0.83 when outliers are disregarded. This suggests that for samples with low total sulfur content, approximately 83% of the sulfur is non-leachable (sulfide) and is, therefore, susceptible to participating in AP. For the four samples with high sulfur contents, the relative proportion of acid leachable sulfur (or sulfate) is lower.

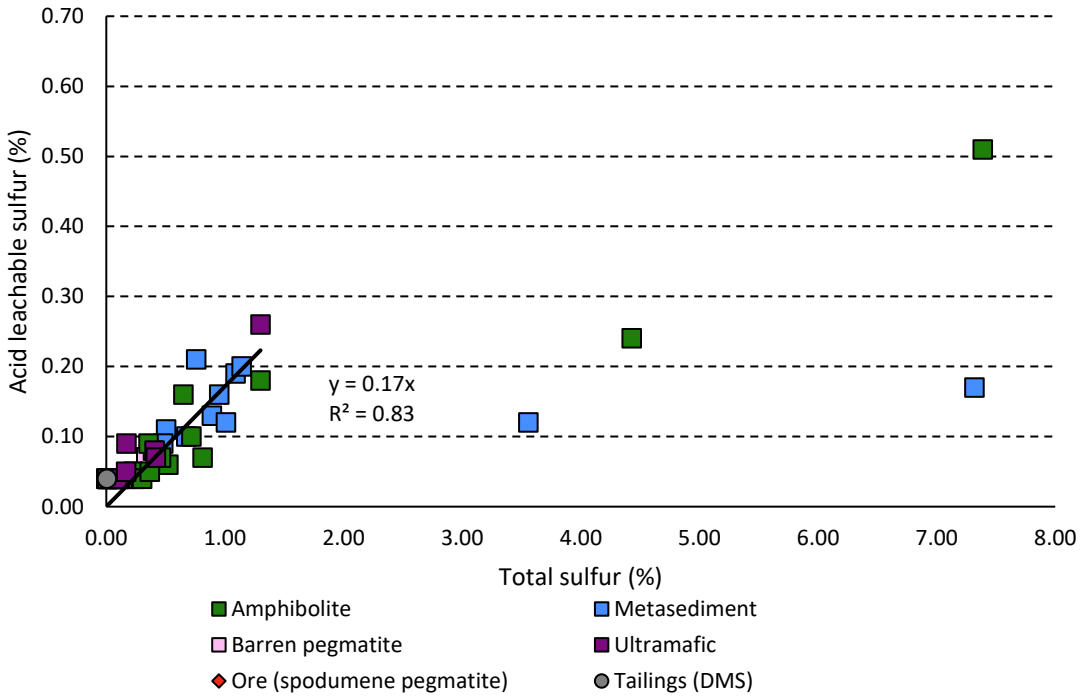


Figure 2: Relationship between acid leachable sulfur and total sulfur

All spodumene pegmatite and DMS tailings samples and 92% of the barren pegmatites present a total sulfur content below 0.04% and are de facto classified as NPAG, whereas solely 24% of the metasediment, 10% of the amphibolites, and none of the ultramafic samples showed total sulfur content below the 0.04% criterion (Table 4).

Table 4: Summary of samples presenting total sulfur concentration below 0.04% and corresponding portion to total

	n	<0.04% total sulfur (%)
All samples	**84**	**31**
Amphibolite	30	3
Metasediment	25	6
Barren pegmatite	13	12
Ultramafic	6	0
Ore (spodumene pegmatite)	8	8
DMS Tailings	2	2

3.1.2. Carbon Speciation Analysis

Relation between the CO_3 carbon and total carbon (Figure 3) is well correlated ($R^2 = 0.95$) and generally well below the 1:1 reference (dotted) line, with regression line slope of 0.66, suggesting that a significant portion of the carbon (34% in average) is not in the form of carbonate and does not participate to the NP of the material. Two ultramafic samples are considered outliers (>2% total C) and were excluded from the regression analysis.

The carbonate analysis by pyrolysis revealed that 25 samples were at or below instrument detection limits (0.025 or 0.04%, depending on the measurement) indicating negligible carbonate content in some of the samples. Samples containing carbonate below instrument detection limits were amphibolite (n=2; 6.7%), metasediment (n=7; 28%) and barren pegmatite (n=10; 77%) waste rocks, spodumene pegmatite ore (n=4; 50%), and tailings (n=2; 100%).

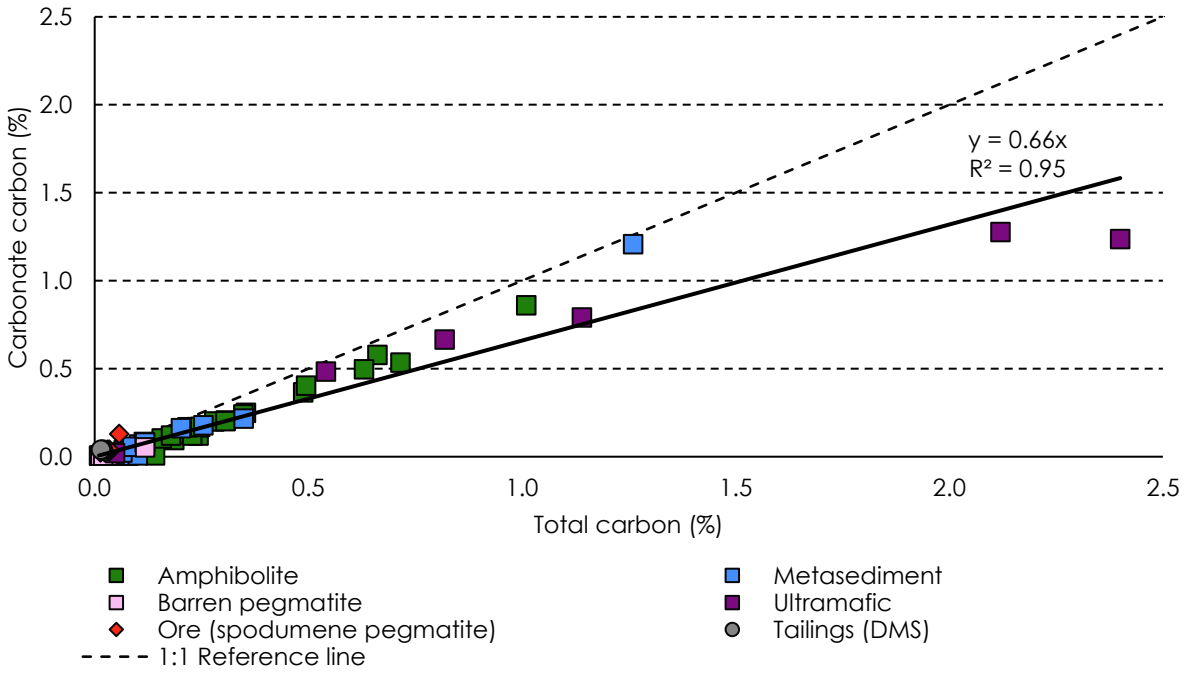


Figure 3: Relationship between carbonate carbon and total carbon

3.1.3. Neutralization and Acidification Potential Assessment

Figure 4 shows that despite some correlation between Bulk-NP, and CO_3-NP (R^2=0.51), most data points remain well below the reference (dotted) line, with a regression line slope of 0.40. This suggests that the neutralization potential of most samples is provided by mineral phases other than carbonates. In this regard the ultramafic samples appear to stand-out with an average CO_3-NP to bulk-NP ratio of 0.75, suggesting that carbonates play a more significant role in the neutralization potential of this lithology.

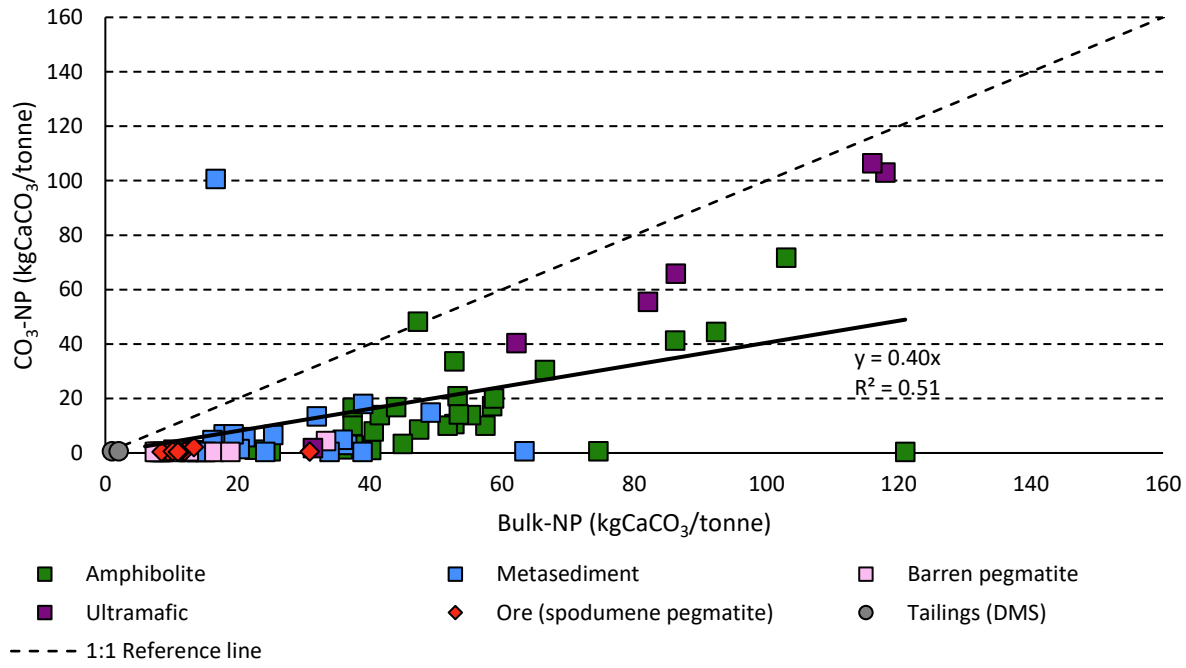


Figure 4: Relationship between bulk-NP and CO_3-NP

The graphic representation of bulk-NNP as a function of total sulfur (Figure 5) shows that all samples exceeding the NNP PAG criterion (NNP<20, green line) also exceed the NRP PAG criterion (NRP<2, orange line). However, numerous samples which respect the NRP criteria do not respect the NNP criteria. The NNP criterion is, therefore, more conservative.

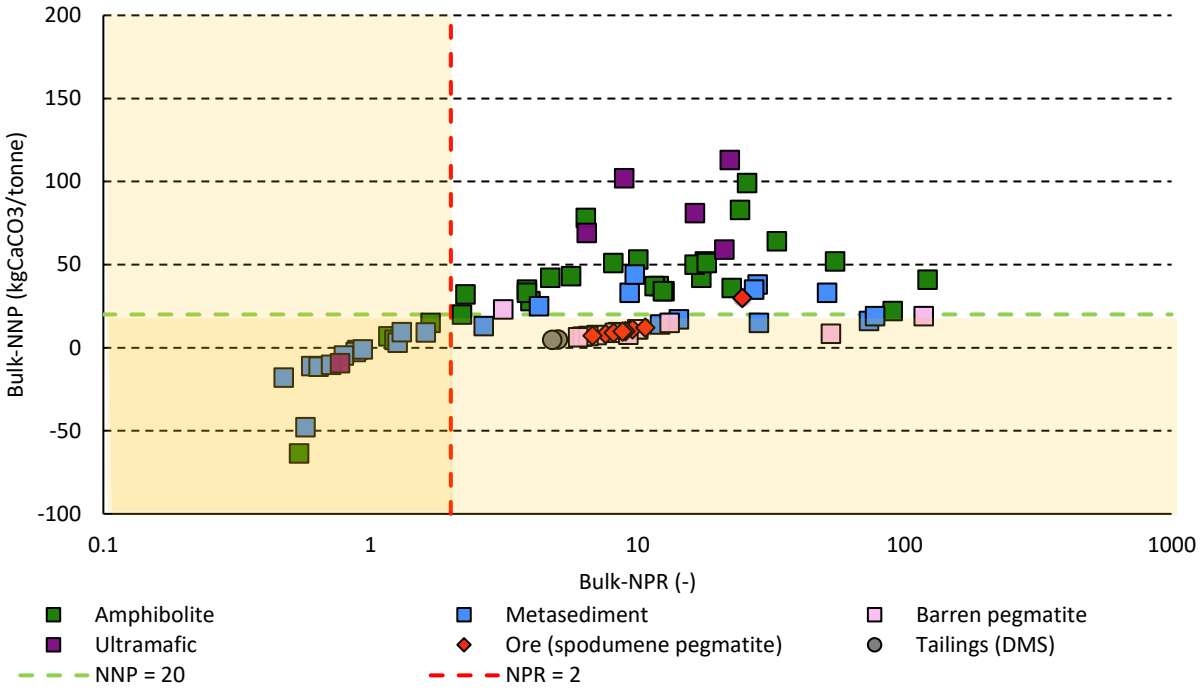


Figure 5: Graphic representation of bulk-NPR as a function of bulk-NNP

The graphic representation of NNP as a function of total sulfur (%) shows that several samples (n=26) were classified as NPAG on the basis of a total sulfur concentration below 0.04%, despite having a NNP below 20 (Figure 6). Moreover, total sulfur content appears to be an important driver as all samples with a content above 0.81% are classified PAG, and all samples with a content below 0.26% are classified as NPAG. Overall there are 20 samples falling within the PAG category "orange box" based on these two criteria.

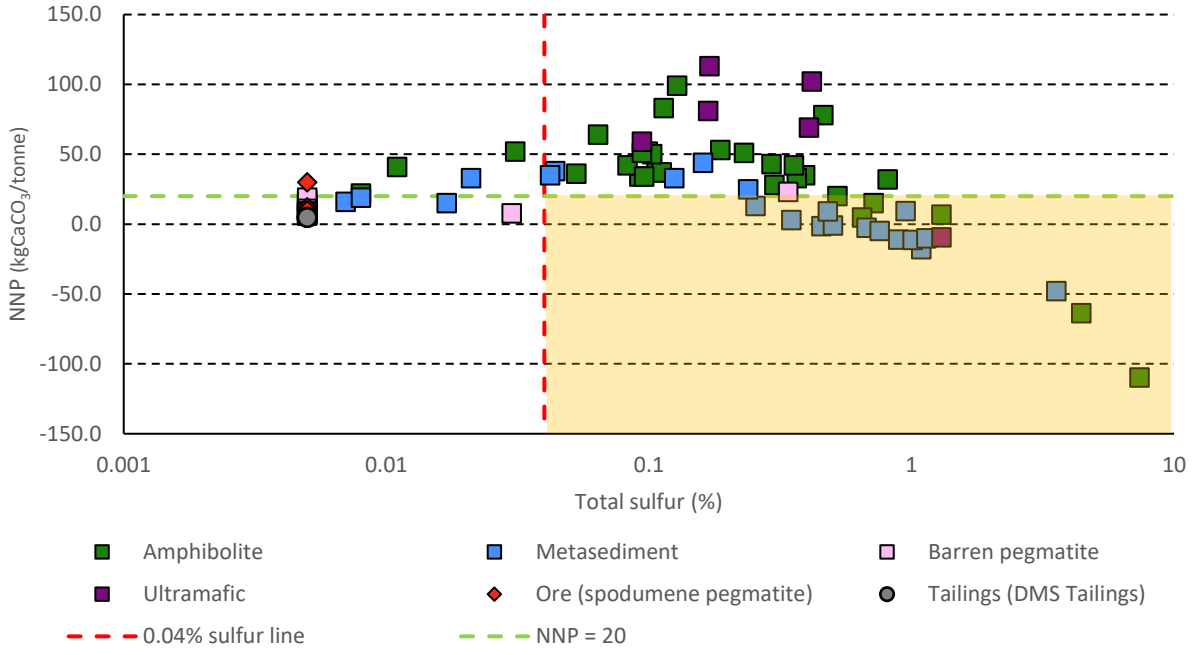


Figure 6: Graphic representation of NNP as a function of total sulfur

3.1.4. Summary and Interpretation

Results showed that total sulfur was a good, yet slightly conservative, predictor of AP with an average 83% of sulfur being in the form of sulfides.

On the other hand, solely 66% of total carbon on average was found to be in the form of carbonates likely to participate to the NP of the material. Moreover, CO_3-NP was found to be poorly correlated with bulk-NP, suggesting that a significant portion of the materials NP is associated with other minerals than carbonates such as silicates (e.g., olivine, feldspar, pyroxene, amphibole) (Lawrence and Scheske, 1997; Jambor et al., 2002). As a result of these observations total carbon as well as carbonate carbon appear to be poor predictors of NP. Should further static testing be carried out, the measurement of bulk-NP should remain the preferred approach.

Among the collected samples (n=84), a total of 20 were classified as PAG (Table 5). The amphibolite (n=6, 19%) and metasediment (n=13, 54%) waste rocks lithologies showed the highest proportion of PAG samples, whereas the ultramafic presented only one PAG sample, and the barren pegmatite did not present any. The ore and tailings samples were all classified as NPAG based because of their negligible total sulfur contents (<0.04%). Table 5 summarizes the number of samples classified as PAG per type of lithology.

Table 5: Summary of the classification of samples as PAG or NPAG and the proportion of samples classified as PAG or NPAG

Lithology	n	Number of PAG samples	Proportion of PAG samples (%)
Amphibolite	30	6	19
Metasediment	25	13	54
Barren pegmatite	13	0	0
Ultramafic	6	1	17
Ore (spodumene pegmatite)	8	0	0
DMS Tailings	2	0	0

3.2. Metal(loid) Leaching Potential

The results of the leaching tests conducted on the selected samples were evaluated with regards to the MELCCFP guidelines.

In order to highlight potential metal(loid) enrichments of the materials, the Aqua Regia environmentally available contents results were compared to the average continental crust concentration (Rudnick and Gao, 2014) as well as to the background contents of the Superior geological province provided in Annex 1 of the *Guide d'intervention - protection des sols et réhabilitation des terrains contaminés* of the MELCCFP.

As stated earlier, it should be noted that terrestrial crust average contents refer to total content (total digestion) and extractible contents refer to partial content (partial digestion) and, therefore, comparisons require precaution. However, extractible contents can be compared with soil Criteria A presented in MELCCFP (2021) « *Guide d'intervention – Protection des sols et rehabilitation des terrains contaminés* ».

The SPLP and CTEU-9 leach test results were compared to the *Critères de qualité pour la protection de la vie aquatique – effet aigu* (CVAA) from the Critères de qualité des eaux de surface (CQES) provided by the MELCCFP. The regulatory values for certain elements (e.g., cadmium (Cd), manganese (Mn)) depend on the hardness of the receiving water system, whereas for aluminium (Al) it also depends on the dissolved organic carbon (DOC concentration and on the pH. The regulation values were calculated assuming a DOC concentration of 10 mg/L, a pH of 6, and a hardness of 10 mg/L.

The TCLP leach tests results were compared to the Directive019 – annexe 2 on the characteristics of (high risk) mine residues.

Tables summarizing the leaching test results and regulations are provided in Appendix D.

3.2.1. Aqua Regia Environmentally Available Contents

A comparison of the elements with respect to the average continental crust content gives insight of metal(loid) enrichments in relation to natural background contents (Table 6 and Figure 7).

The elements most enriched with respect to continental crust are summarized in Table 6. Arsenic (As) was the most enriched (37% of samples) with respect to crustal content, mostly within the amphibolite (n=10) and metasediment (n=10), and ultramafic (n=6) waste rocks. Antimony (Sb) and lithium (Li) were also highly enriched (26% of samples). Sb enrichment was observed in the amphibolite (n=7), barren pegmatite (n=4) and ultramafic (n=5) waste rocks. Lithium was most abundant in the metasediment waste rocks (n=12).

Since the arsenic, antimony and lithium contents measured by Aqua Regia represent only a portion of the total content, exceedances of the terrestrial crust average content are likely to be greater and present in more samples.

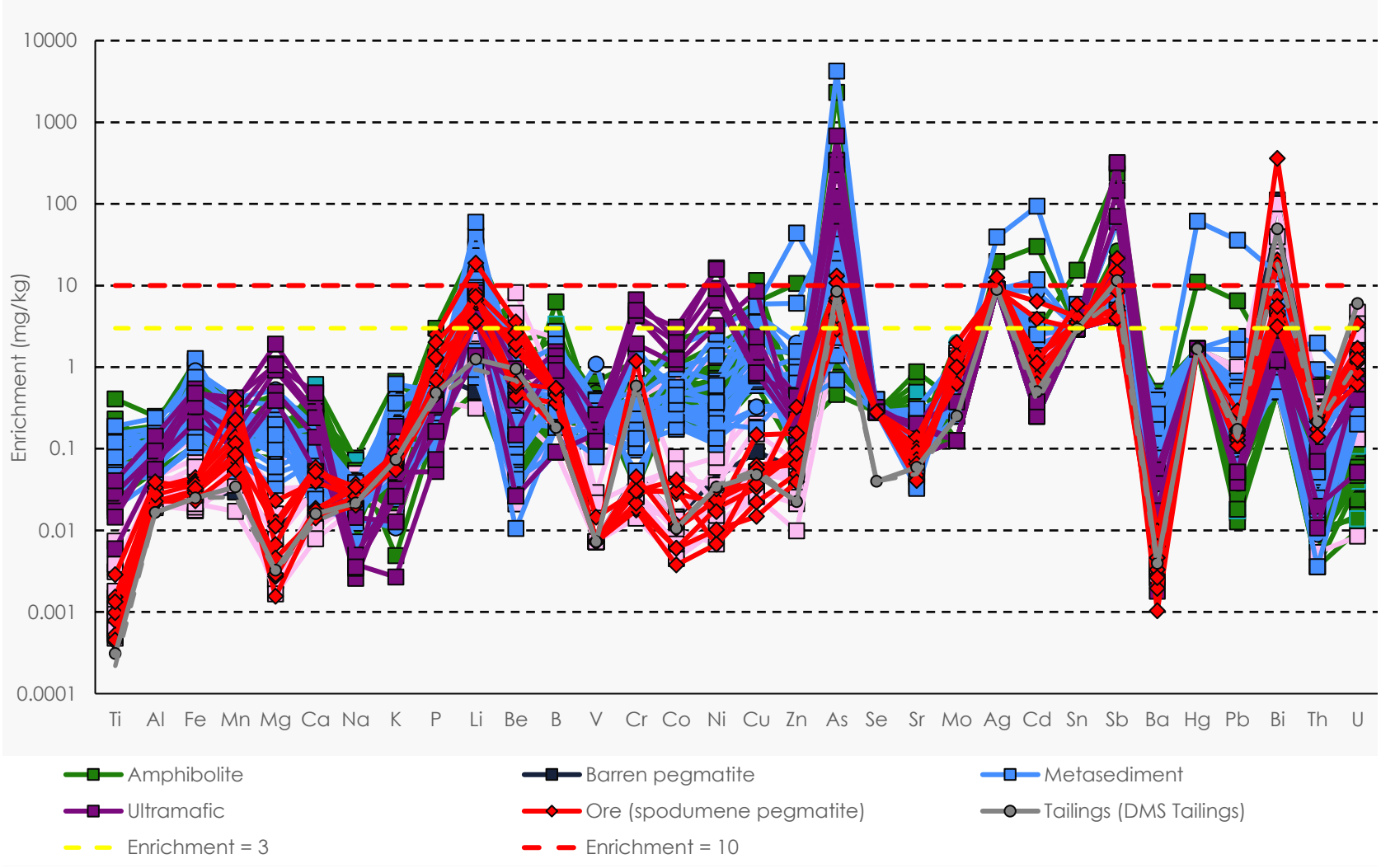


Figure 7: Graphic representation of metal(loid)s relative enrichment with respect to the average continental crust contents

Table 6: Summary of samples exceeding 10 times the average continental crust contents per lithology

		Ag	As	Bi	Cd	Cu	Hg	Li	Ni	Pb	Sb	Sn	Zn
Continental crust content (ppm, Rudnick and Gao. 2014)		**0.056**	**1.3**	**0.18**	**0.08**	**27**	**0.03**	**16**	**59**	**11**	**0.2**	**1.7**	**71**
	n												
Enriched (≥10 times the continental crust)	84	3	31	15	3	1	2	22	2	1	22	1	2
Amphibolite	30	1	10	1	1	1	1	9	0	0	7	1	1
Metasediment	25	1	10	1	2	0	1	12	0	1	3	0	1
Barren pegmatite	13	0	3	8	0	0	0	0	0	0	4	0	0
Ultramafic	6	0	6	0	0	0	0	0	2	0	5	0	0
Ore (spodumene pegmatite)	8	1	2	3	0	0	0	1	0	0	2	0	0
Tailings	2	0	0	2	0	0	0	0	0	0	1	0	0
Total % enriched		**3.6**	**37**	**18**	**3.6**	**1.2**	**2.4**	**26**	**2.4**	**1.2**	**26**	**1.2**	**2.4**

Arsenic (As) was the metal(loid) which presented the most exceedances with regards to the MELCCFP Criteria A for soil quality, with 56 occurrences (67%), followed by copper (Cu; 31%), nickel (Ni; 24%), tin (9.5%), cobalt (Co; 8.3%), silver (Ag; 3.6%), cadmium (Cd; 3.6%), zinc (Zn; 3.6%), mercury (Hg; 2.4%), and lead (Pb; 2.4%). A comprehensive inventory of the number of exceedances per lithology and material types is presented in Table 7.

Table 7: Summary of Aqua Regia results exceeding the Criteria A

		Ag Exceedance	As Exceedance	Cd Exceedance	Co Exceedance	Cu Exceedance	Hg Exceedance	Ni Exceedance	Pb Exceedance	Sn Exceedance	Zn Exceedance
Criteria A (mg/kg); *Critères Génériques pour les sols, Guide d'intervention – protection des sols et réhabilitation des terrains contaminés – annexe 1*)		**0.5**	**5**	**0.9**	**30**	**65**	**0.3**	**50**	**40**	**5**	**150**
	n										
Total exceedance	**84**	**3**	**56**	**3**	**7**	**26**	**2**	**20**	**2**	**8**	**3**
Amphibolite	30	1	18	1	1	18	1	9	1	1	1
Metasediment	25	1	12	2	1	5	1	4	1	3	2
Barren pegmatite	13	0	11	0	0	2	0	1	0	1	0
Ultramafic	6	0	6	0	5	1	0	6	0	0	0
Ore (spodumene pegmatite)	8	1	7	0	0	0	0	0	0	3	0
DMS Tailings	2	0	2	0	0	0	0	0	0	0	0
Total % exceedance		**3.6%**	**67%**	**3.6%**	**8.3%**	**31.0%**	**2.4%**	**24%**	**2.4%**	**9.5%**	**3.6%**

3.2.2. SPLP and CTEU-9 Leach Tests

The pH results measured in leachates from the SPLP leach tests ranged from 6.5 to 9.2, and remained within the surface water quality guidelines for mine effluent (6.0 to 9.5) for chronic exposure of the CQES (MELCCFP). The metal(loid)s that presented exceedances with regards to the resurgence criteria (RES) comprised aluminium (Al; 81%), copper (Cu; 16%), zinc (Zn; 14.3%), cadmium (Cd; 16%) and arsenic (As; 4.8%). The details of the exceedances met by lithology and material type are presented in Table 8.

Table 8: Summary of samples presenting SPLP leach test results exceeding RES criteria and soil quality Criteria A

		As Exceedance	Al* Exceedance	Cd† Exceedance	Cu† Exceedance	Zn† Exceedance
RES criteria (mg/L; MELCCFP Criteria for aquatic life protection - acute effect)		**0.34**	**0.28**	**0.0002**	**0.0016**	**0.017**
	n					
Total rock types	**84**	**4**	**68**	**7**	**13**	**12**
Amphibolite	30	1	29	0	3	3
Metasediment	25	1	23	2	2	2
Barren pegmatite	13	0	10	2	4	3
Ultramafic	6	2	0	0	0	0
Ore (spodumene pegmatite)	8	0	6	1	2	2
Tailings	2	0	0	2	2	2
Total % of Exceedance		**4.8**	**81**	**8.3**	**16**	**14**

* Criteria calculated assuming a DOC concentration of 10 mg/L, a pH of 6, and a hardness of 10 mg$CaCO_3$/L

† Criteria calculated assuming a hardness of 10 mg$CaCO_3$/L

The pH results obtained from CTEU-9 leach tests were comprised between 8.0 to 9.8, with six samples exceeding the upper limit for pH of 9.5. The CTEU-9 leach tests yielded more elements for concern than the SPLP leach test (Table 9).

Table 9: Summary of samples presenting CTEU-9 leach test results exceeding RES criteria and soil quality Criteria A

		Al* Exceedance	As Exceedance	Be† Exceedance	Cu† Exceedance	Li† Exceedance	Ni† Exceedance	Pb† Exceedance	Sb Exceedance	V Exceedance	Zn† Exceedance
RES criteria (mg/L; MELCCFP Criteria for aquatic life protection - acute effect)		**0.28**	**0.34**	**0.0012**	**0.0016**	**0.91**	**0.067**	**0.004**	**1.1**	**0.11**	**0.017**
	n										
Total number of samples	**84**	**64**	**36**	**11**	**23**	**27**	**2**	**1**	**5**	**3**	**4**
Amphibolite	30	23	5	0	2	7	0	0	1	1	0
Metasediment	25	18	11	0	6	8	0	0	0	1	1
barren pegmatite	13	13	5	6	9	4	0	1	0	1	1
Ultramafic	6	0	6	0	2	0	2	0	4	0	0
Ore (spodumene pegmatite)	8	8	7	5	2	8	0	0	0	0	0
Tailings	2	2	2	0	2	0	0	0	0	0	2
Total % exceedance		**76**	**43**	**13**	**27**	**32**	**2.4**	**1.2**	**6.0**	**3.6**	**4.8**

* Criteria calculated assuming a DOC concentration of 10 mg/L, a pH of 6, and a hardness of 10 mg$CaCO_3$/L

† Criteria calculated assuming a hardness of 10 mg$CaCO_3$/L

The greater number of metal(loid)s leached from the CTEU-9 extraction than for the SPLP extraction may be explained by the finer particles grinding and longer extraction time (7 days vs. 18 hours) employed as part of the CTEU-9 extraction procedure compared to the SPLP one. It is also known that arsenic is more labile at higher pH.

3.2.3. TCLP Leach Tests

The TCLP leach test pH measurements ranged from 4.9 to 7.1, and 79 leachates showed a pH below 6. Arsenic (As) was the only element from the TCLP leach tests that was identified as a COPC with regards to the Directive 019 criteria for high-risk mining waste (Table 10), and both of the identified exceedances were from the ultramafic waste rock samples.

Table 10: Summary of TCLP leach test results exceeding Directive 019 criteria for high-risk mining waste

		As Exceedance
High risk criteria (mg/L, *Directive 019 – annexe 2*)		**5.0**
	n	
Total number of samples	**84**	**2**
Amphibolite	30	0
Metasediment	25	0
Barren pegmatite	13	0
Ultramafic	6	2
Ore (spodumene pegmatite)	8	0
Tailings	2	0
Total % exceedance		**2.4**

3.2.4. Summary and Interpretation

A summary of the enriched metal(loid)s and COPC identified from the various leaching tests conducted as part of the static test study is presented in Table 11.

The Aqua Regia extractible content analysis have allowed the identification of a wide range of enriched metal(loid)s with regards to the (10 times) continental crust total contents Criteria A and the Superior geological province background extractible content, and in both cases arsenic (As) was identified as the most enriched COPC.

The SPLP and CTEU-9 leaching tests conducted on the samples partially reflected the enrichments measured with the Aqua Regia extractible content analysis, but also highlighted aluminium (Al) as a main leaching potential concern. Arsenic (As) was especially identified as a concern for the CTEU-9 test with 64 exceedances (76%) as opposed to four exceedances (4.8%) for the SPLP test, thus suggesting that the particle size and pH has an important impact on the mobility of this element. Moreover, arsenic exceedances were observed for all lithologies and material types, but was particularly apparent in the ultramafic samples (100%), the tailings (100%), and the spodumene pegmatite ore material (88%).

Finally arsenic was the only COPC identified from the TCLP leach test with the only two occurrences being from the ultramafic samples (n=6; 33%).

The HCT protocol was designed on the basis of these results in order to establish if the identified exceedances can be confirmed through kinetic analysis.

Table 11: Summary of enriched metal(loid)s and COPCs from the various leaching test procedures

Extraction test procedure	Criterion	Exceeding metal(loid)s ranked by occurrence
Aqua regia	10 times continental crust content	As, Li, Sb, Bi, Ag, Cd, Hg, Ni, Zn, Cu, Pb
	MELCCFP Superior geological province background contents	As, Cu, Ni, Sn, Co, Ag, Cd, Zn, Hg, Pb
SPLP	MELCCFP criteria for aquatic life protection (acute effect) and Criteria A	Al, Cu; Zn, Cd, As
CTEU-9	MELCCFP criteria for aquatic life protection (acute effect) and Criteria A	Al, As, Li, Cu, Be, Sb, Zn, V, Ni, Pb
TCLP	Directive 019 – High risk criteria	As

3.3. Kinetic Testing

Kinetic testing was run on a set of 17 samples (+1 duplicate) for a duration exceeding 40 weeks and are still on-going in May 2024. The results presented in this report apply to the first 40 weeks.

The samples were selected on the basis of the static test results in order to include the various lithologies and material types and cover a wide range of acid generation and metal leaching potential. The chemical parameters presented and interpreted in this section include:

- Physicochemical criteria such as pH, and electric conductivity (conductivity);
- Oxidation and neutralization products concentrations including sulfate, calcium, and magnesium;
- The concentrations of the COPCs identified as part of static testing including aluminium (Al), arsenic (As), beryllium (Be), cadmium (Cd), copper (Cu), lithium (Li), nickel (Ni); lead (Pb), antimony (Sb), vanadium (V), zinc (Zn), and total phosphorus (total P).

Summary tables from laboratory analysis results are presented in Appendix E, and the kinetic testing certificates of analyses are recorded in Appendix B.

3.3.1. Kinetic Testing Results

Figure 8 displays the kinetic variation of the physicochemical parameters namely pH (a), EC (b).

a. The figure shows a progressive decrease of the pH measurements during the first 20 weeks with values ranging from 7.37 to 9.09 at Week 0, and from 6.38 to 7.91at Week 20, and appearance of stabilization during the second half of the monitoring period. Despite this decrease, pH values from all HCT remain within the regulatory range of the MDMER and the Directive 019 (6 to 9.5; orange dotted lines). The DMS tailings appear to stand-out with overall lower pH reading throughout the monitoring.
b. The conductivity readings display rapid decreasing trend of variable amplitudes during the first weeks for some of the tests with, for instance, values going from 164 µS/cm and 154 µs/cm, down to 59 µS/cm and 14 µS/cm at Week 4, for CV5-WR-HCT-50, and CV5-WR-HCT-74, respectively. This initial conductivity peak is believed to be associated with the dissolution and release of elements associated with secondary mineral phases during the first flushes. Starting from Week 5, the conductivity readings remained fairly stable throughout the whole monitoring period. The DMS tailings display overall lower conductivity.

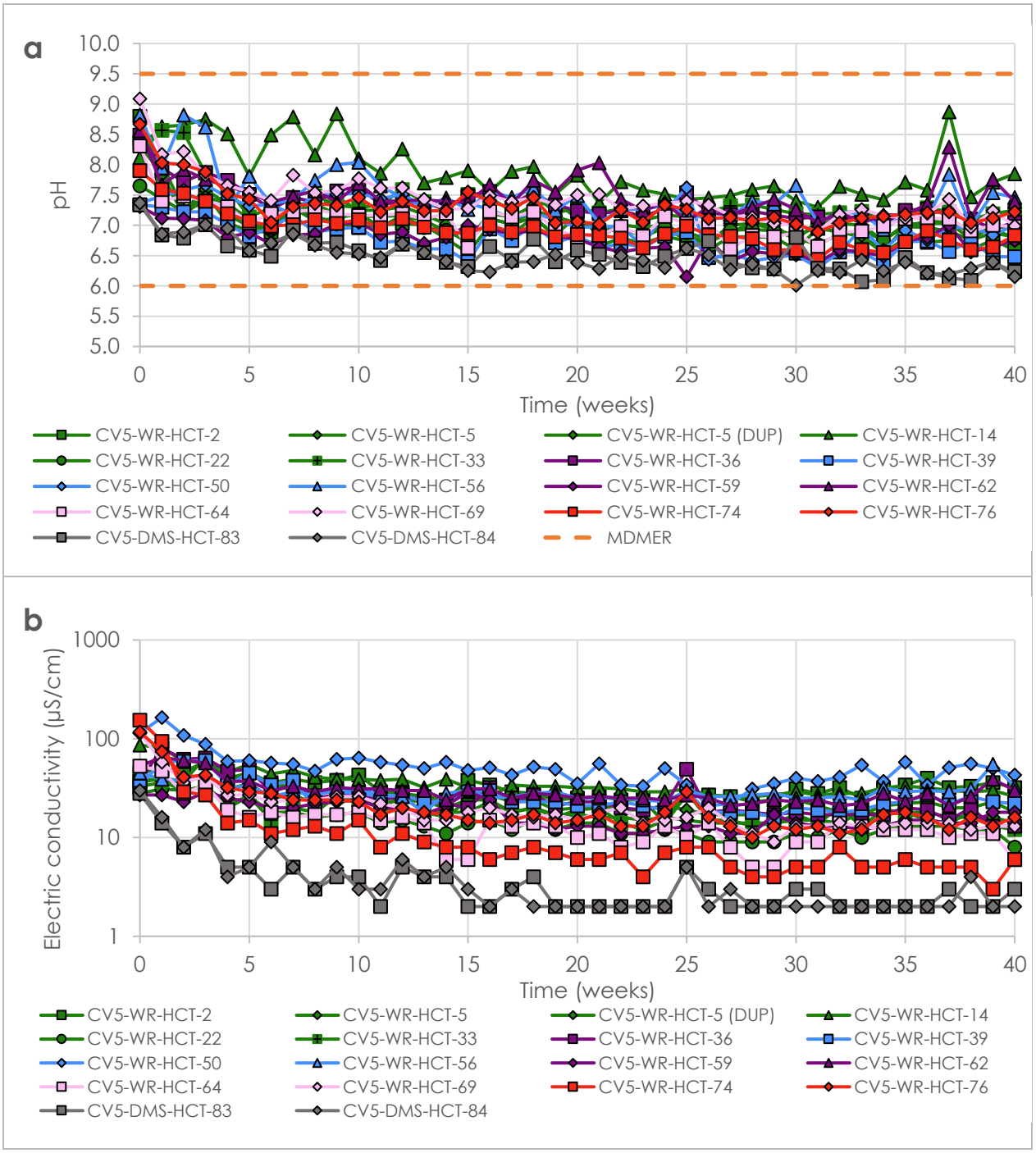


Figure 8: Conductivity and pH evolution in kinetic tests

The graphs displayed in Figure 9 present the alkalinity (a) and acidity (b) measurements monitored throughout the 40 weeks of HCT.

a. The alkalinity readings generally followed a decreasing trend during the first weeks of the test until, eventually, reaching equilibrium at various points of the monitoring period. Both DMS tailings tests (CV5-DMS-HCT-83, and CV5-DMS-HCT-84) decreased from 6 mg/L at Week 0 down to detection limit (2 mg/L) within 4 weeks, while CV5-WR-HCT-14 decreases from 22 mg$CaCO_3$/L at the beginning of the test and reached pseudo-equilibrium in the range of 13 mg/L at Week 20.
b. The acidity measurements essentially remained below detection limit (2 mg$CaCO_3$/L) for all HCTs, despite some concentration variations within the 2 to 10 mg$CaCO_3$/L range throughout the experiment. CV5-WR-HCT-2, and CV5-WR-HCT-50 remained above the detection limit more consistently.

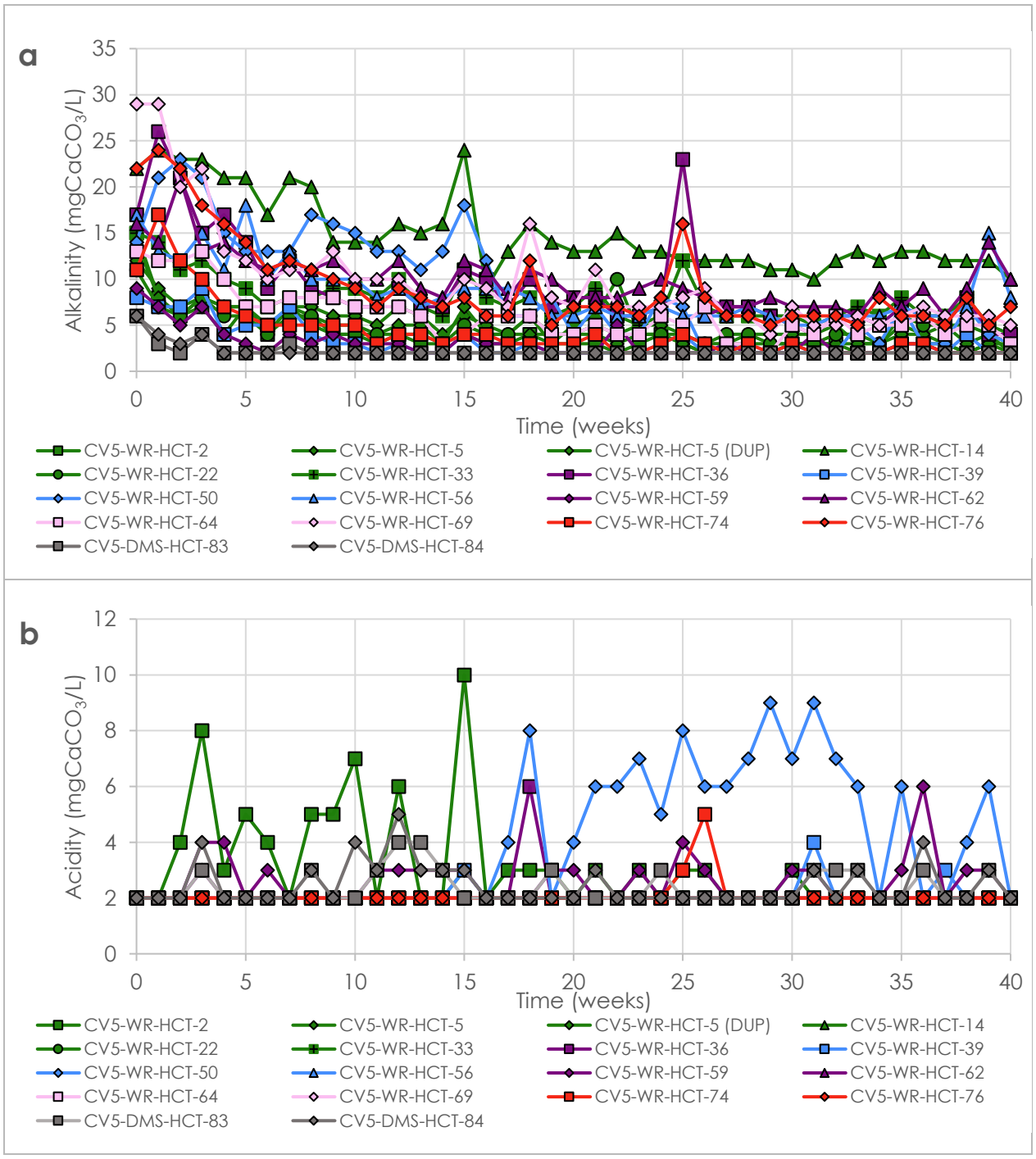


Figure 9: Evolution of alkalinity and acidity in kinetic tests

The graphs presented in Figure 10 display the monitoring of the neutralization and oxidation products concentrations throughout the experiment, namely calcium (a) and magnesium (b) on one hand, and sulfate (c) on the other hand. The trends of these elements were fairly similar between tests, and tended to follow a slow and steady trend decrease after a slight peak observed during the initial flushes. It should be highlighted that the DMS tailings present lower concentrations for all of these elements suggesting that oxidation and neutralization phenomena are taking place at a lower rate in these materials.

a. Calcium concentrations range from 16 mg/L to 0.29 mg/L, depending on the tested sample, and appear relatively highest in metasediments and smallest in the DMS tailings than for other lithologies. Moreover, the calcium concentrations are generally about an order of magnitude above magnesium's, suggesting that calcite is the dominant carbonate specie in terms of neutralization rate.
b. The magnesium concentrations range from 5.73 mg/L to 0.01 mg/L and appear more prominent among the ultramafic samples and smallest in the case of the DMS tailings.
c. The sulfate concentrations of most HCTs remained near or below detection limit (2 mg/kg/L) throughout the whole test, indicating relatively low sulfide oxidate rates, especially in the case of the barren pegmatite, the spodumene pegmatite (ore), and the DMS tailings. On the other hand, The sulfate concentrations appear to be greater in the case of the metasediment, followed by the amphibolite, and the ultramafic lithologies. Low sulfate concentrations indicate that potential presence of PAG material in the static tests is not confirmed in more representative kinetic tests.

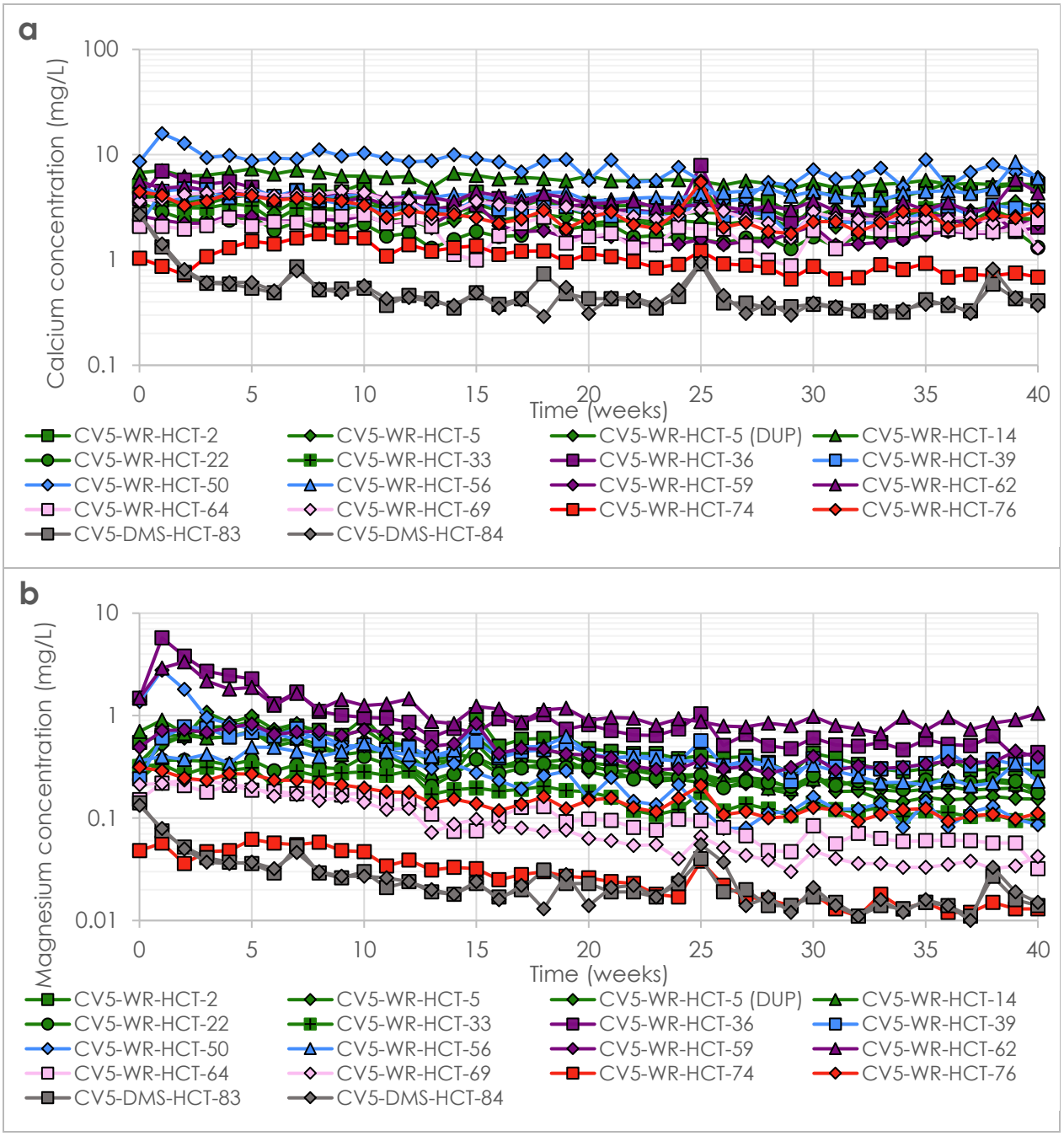


Figure 10: Evolution of oxidation and neutralization products concentration in kinetic tests (1 of 2)

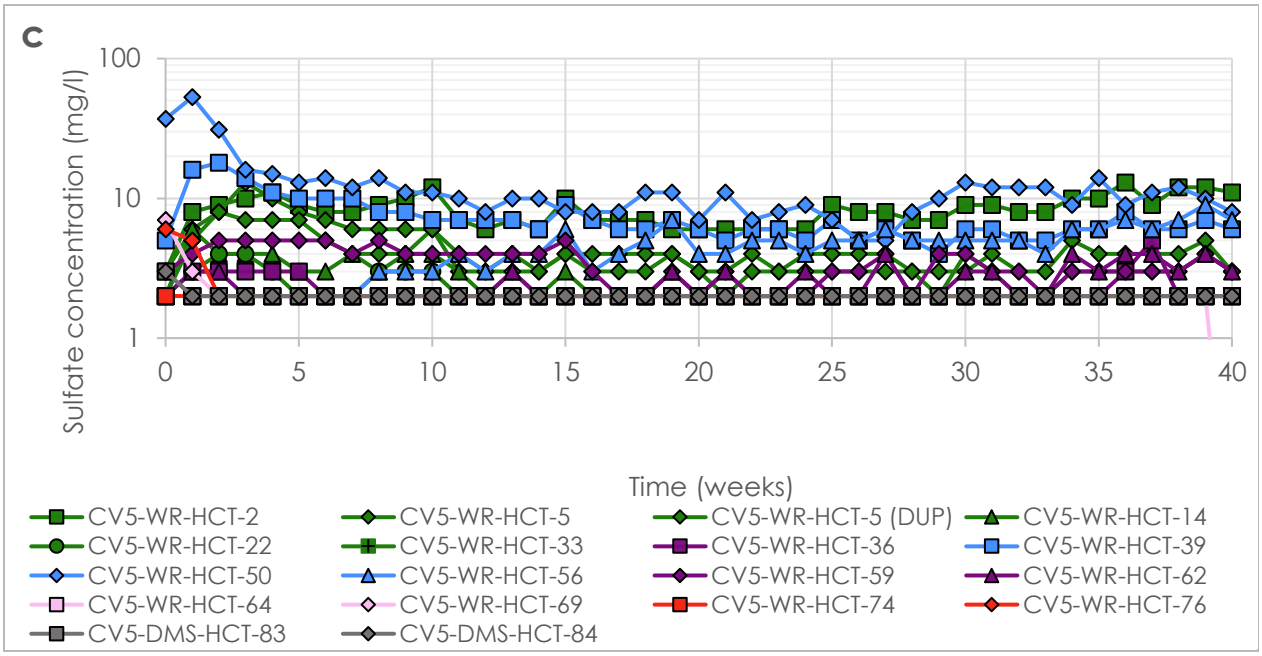


Figure 11: Evolution of oxidation and neutralization products concentration in kinetic tests (2 of 2)

The graphs presented in Figure 12 to Figure 17 present the HCT results of the COPC that were identified as part of the leaching tests. For all COPC the MELCCFP resurgence criterion is identified by a yellow dotted line, and the MDMER effluent criterion (As, Cu, Ni, Pb, Zn) is identified by an orange dotted line. The Directive 019 criteria were not represented on the graphs as they are less conservative than their MDMER counterparts.

a. Aluminium concentrations remained below the MELCCFP resurgence criterion (0.30 mg/L) throughout the whole test, despite some relatively high values, notably in the case of a barren pegmatite sample (CV5-WR-HCT-69) which ranged from 0.12 to 0.28. The aluminium concentrations were lowest in the ultramafic samples HCTs.

b. The arsenic concentrations ranged from 19.6 to 0.0024 mg/L, depending on the tested sample and flush, and have been on an overall decreasing trend after the initial peak. The MELCCFP resurgence criterion (0.34 mg/L) and/or MDMER criterion (0.1 mg/L) were exceeded consistently by leachates from the ultramafic samples (CV5-WR-HCT-36, CV5-WR-HCT-59, and CV5-WR-HCT-62) and to a lesser extent by amphibolite samples (CV5-WR-HCT-5, and CV5-WR-HCT-22). Additionally, some barren pegmatite samples (CV5-WR-HCT-64, and CV5-WR-HCT-69), and spodumene pegmatite samples (ore; CV5-WR-HCT-74) exceeded at least one of these criteria during the first flushes.

c. The beryllium concentrations measured for most tests remain rear or below detection limit (7×10^{-6} mg/L), and remain fairly stable after the peak associated with the first flushes. All

measurements are below the MELCCFP resurgence criterion. Highest values are measured in barren pegmatite samples (CV5-WR-HCT-64, and CV5-WR-HCT-69) and spodumene pegmatite samples (CV5-WR-HCT-74, and CV5-WR-HCT-76).

d. The measured cadmium concentrations are mostly near or below detection limit (3×10^{-6} mg/L) and below MELCCFP resurgence criterion. Measurements remain fairly stable throughout the test. Highest values are measured among metasediment samples (CV5-WR-HCT-39, and CV5-WR-HCT-50).

e. Copper concentration measurements from all lithologies and material types punctually exceed the MELCCFP resurgence criterion (0.0016 mg/Ll) with a total of 20 occurrences, but remain way below the MDMER criterion (0.1 mg/L). It should also be mentioned that all lithologies and material types also present a significant proportion of concentration values near or at detection limit. The importance of oscillation of the measurement and apparent randomness of the observed spikes and exceedances, could potentially be explained by the proximity of the measurements, and MELCCFP criterion to limit of quantification of the method.

f. The lithium concentrations follow a decreasing trend throughout the test with values ranging from 0.918 to 0.0019 mg/L at Week 0 and which seem to converge towards equilibrium near the end of the test, with values ranging from 0.0006 to 0.0078 mg/L at Week 40. One occurrence of exceedance of the MELCCFP resurgence criterion (0.91 mg/L) was recorded at Week 1 for a spodumene pegmatite sample (CV5-WR-HCT-74). Generally speaking, the ore samples followed by the amphibolites, the metasediments and the barren pegmatites appear to leach the most lithium while the ultramafic lithology and the DMS tailings appear to leach the least.

g. The nickel concentrations appear relatively constant after a peak observed for some ultramafic samples during the initial flushes. A total of six exceedances of the nickel MELCCFP resurgence criterion (0.067 mg/L) were noted and concern the ultramafic lithology (CV5-WR-HCT-36, CV5-WR-HCT-62) exclusively. DMS tailings and ore samples' nickel concentrations remain mostly at detection limit (0.0001 mg/L).

h. Most lead concentrations remain at detection limit (9×10^{-6} mg/L), and below the MELCCFP resurgence (0.04 mg/L) and MDMER (0.08 mg/L) criteria throughout the test.

i. The antimony concentrations ranged from 3.14 mg/L to detection limit (0.0009 mg/L) depending on samples, and appeared to follow an overall decreasing trend after the peak observed consecutively to the initial flushes. The MELCCFP groundwater protection criteria (1.1 mg/L) is exceeded consistently, for a total of 25 occurrences, by the leachate concentrations of two ultramafic samples (CV5-WR-HCT-36, CV5-WR-HCT-62) during the first weeks. One metasediment sample (CV5-WR-HCT-56) remained near or at detection limit throughout the test.

j. The vanadium concentrations remained well below the MELCCFP groundwater protection criterion throughout the test, and appear to follow a slow decreasing trend until Week 25,

before reaching a pseudo-equilibrium at values ranging from detection limit (1×10^{-6} mg/L) to 0.00262 mg/L, depending on the considered sample. Overall, the amphibolites, followed by the ultramafic and the metasediment lithologies, appear to leach the most vanadium, while the DMS tailings, and the barren and spodumene pegmatite (ore) materials appear to leach the least.

k. Most zinc concentration measurements remain near or at detection limit (0.002 mg/L). No exceedance of the MELCCFP resurgence criterion (0.017 mg/L) or MDMER criteria (0.4 mg/L) was recorded, although a few near-exceedances can be observed for a few samples, notably one metasediment (CV5-WR-HCT-50), as well as one DMS tailings (CV5-WE-HCT-83), and one barren pegmatite (CV5-WR-HCT-64) sample.

l. Most total phosphorous concentration readings remain near or at detection limit (0.03 mg/L), however a few samples presented higher total phosphorous concentration, notably the ultramafic samples with four noted exceedances (CV5-WR-HCT-36) of the MELCCFP resurgence criterion (3 mg/L) before Week 5. The samples with relatively high total phosphorous concentration, however, follow a decreasing trend with maximum final concentrations of 0.30 mg/L.

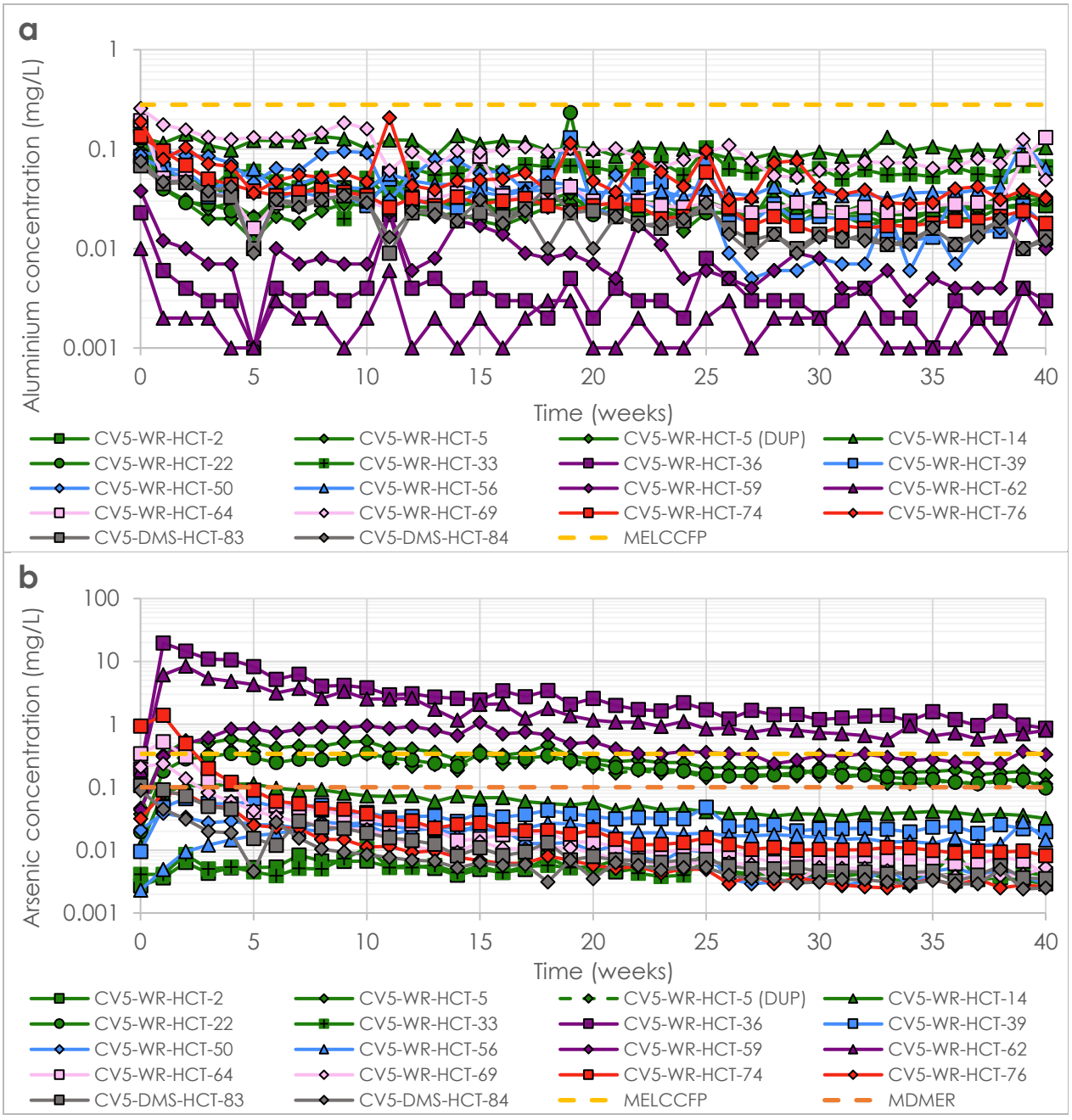


Figure 12: Graphic representation of the COPCs' concentration in kinetic tests (1 of 6)

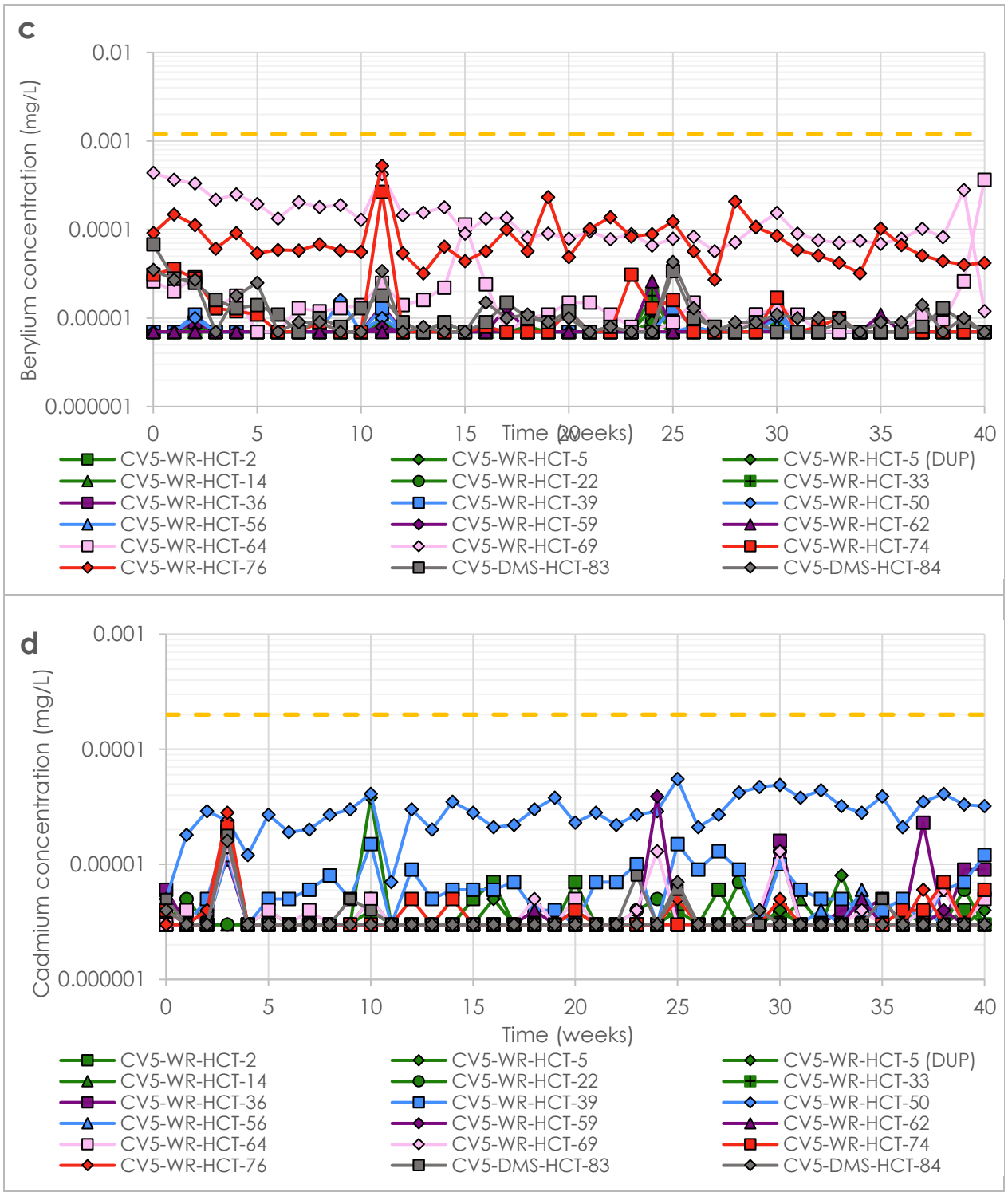


Figure 13: Graphic representation of the COPCs' concentration in kinetic tests (2 of 6)

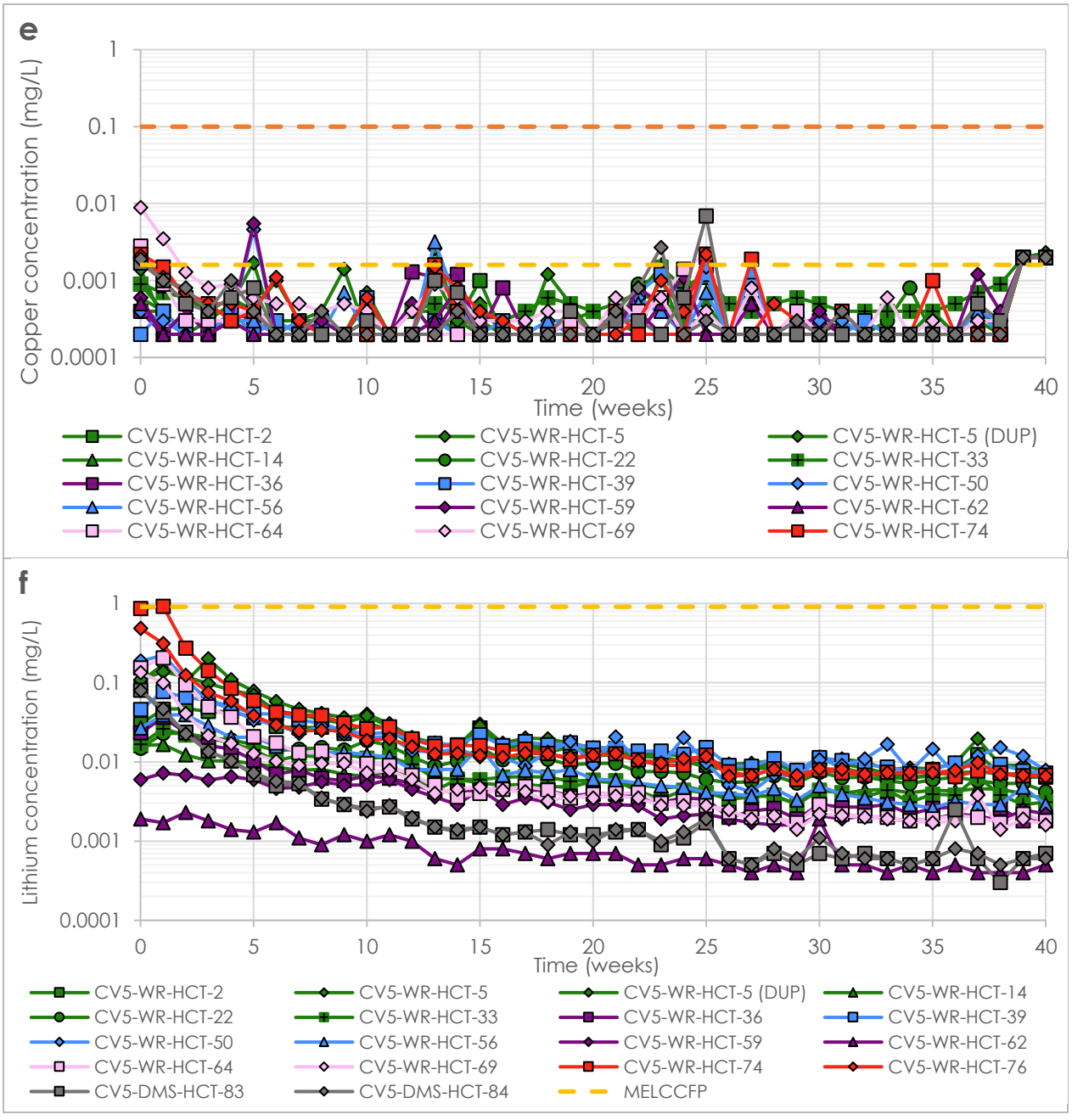


Figure 14: Graphic representation of the COPCs' concentration in kinetic tests (3 of 6)

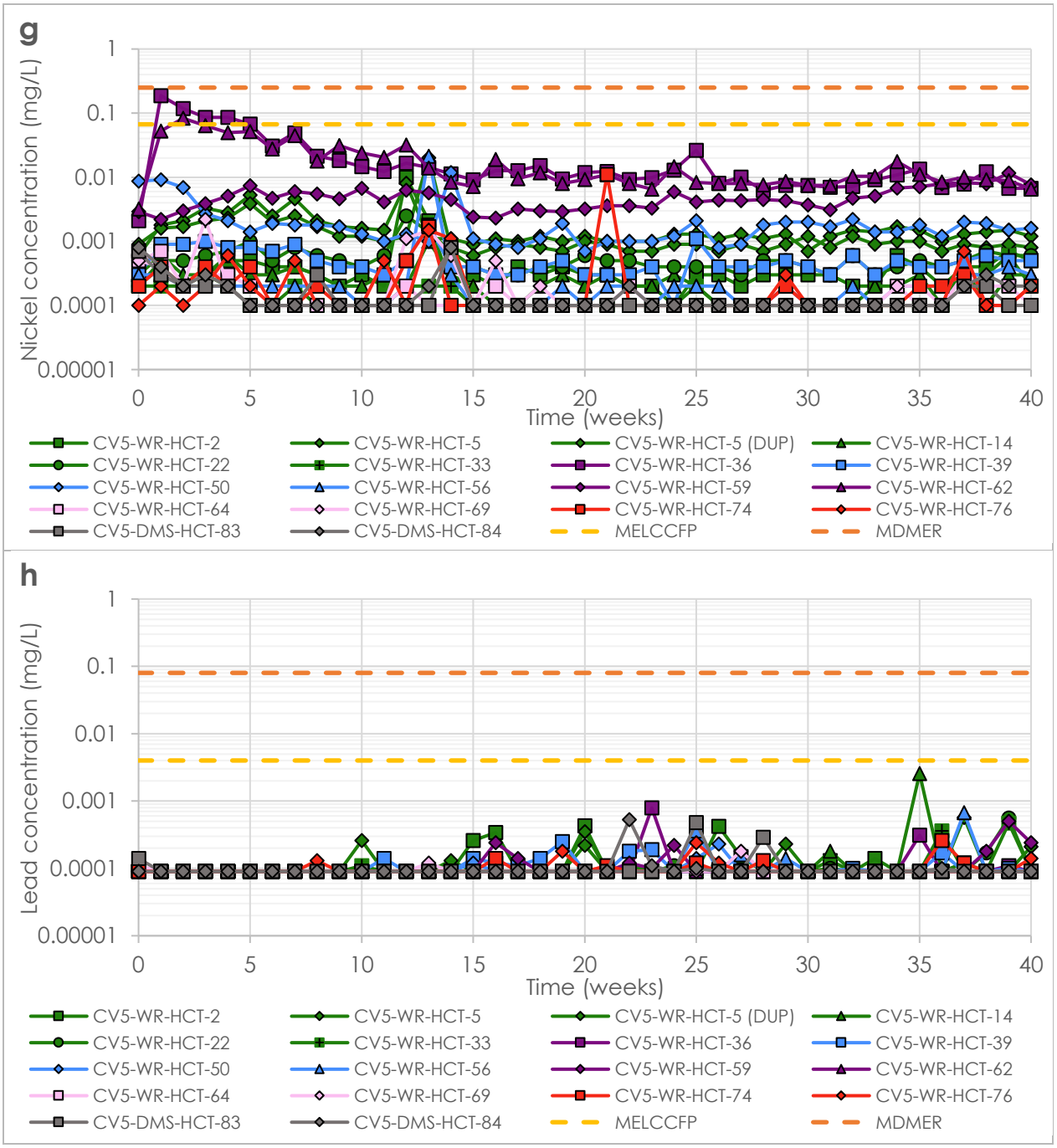


Figure 15: Graphic representation of the COPCs' concentration in kinetic tests (4 of 6)

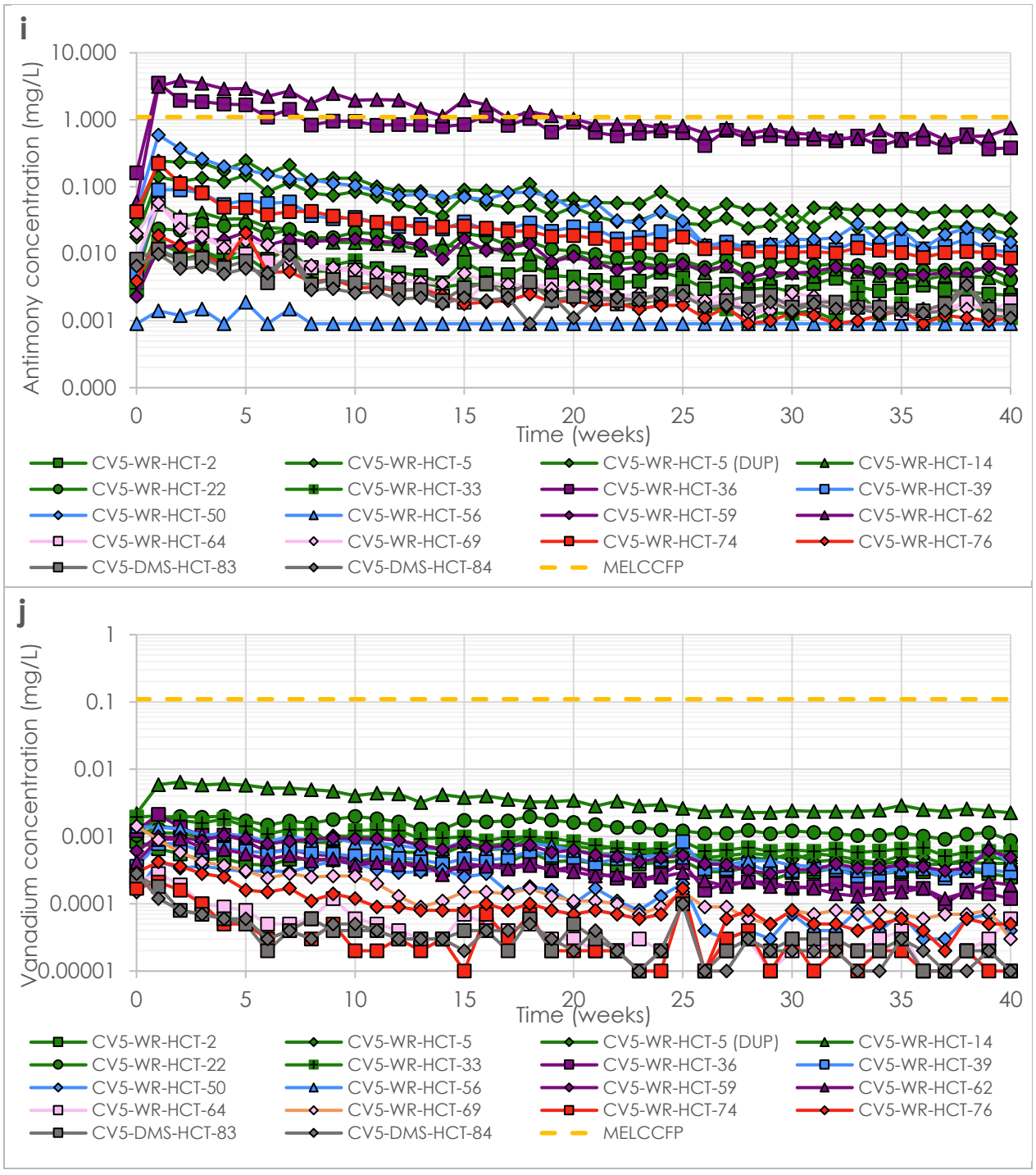


Figure 16: Graphic representation of the COPCs' concentration in kinetic tests (5 of 6)

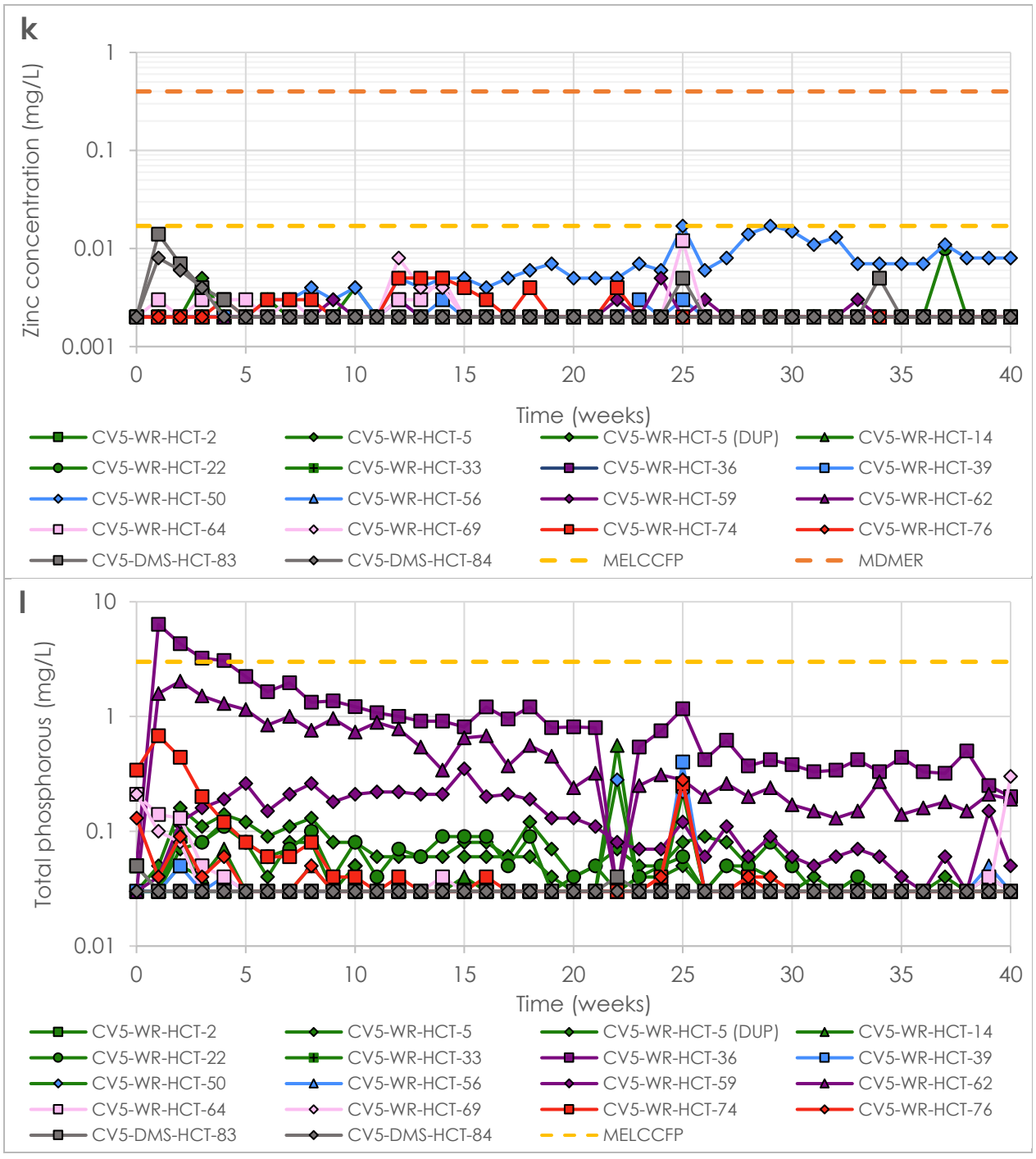


Figure 17: Graphic representation of the COPCs' concentration in kinetic tests (6 of 6)

3.3.2. Kinetic Tests Summary and Interpretation

After 40 weeks, the HCT testing did not confirm whether the samples classified as PAG during Phase 1 static testing would generate net acidity. Sulfate concentrations within the HCT leachates remained small for all HCTs, with values ranging from 2 mg/L(detection limit) to 12 mg/L after the initial flushes, suggesting the existence of small oxidation rates for test samples. However, the PAG characteristics of these samples should not be excluded at this point as net acidity production typically becomes apparent after near-complete exhaustion of the NP of the material. Furthermore, the solid to liquid ratio within the pile will be significantly higher within the pile than the 1:1 ratio employed as part of the HCT procedure. A higher solid to liquid ratio will cause the concentration of oxidation products such as sulfate and acidity to increase.

Among the COPCs identified as part of the leaching test (Table 12), arsenic (As) was found to constitute the principal potential risk of exceedance of the MELCCFP groundwater protection criterion, and the final effluent discharge MDMER criterion with a total of 128, and 259 weekly exceedances, respectively. Antimony (Sb), copper (Cu), and total phosphorous (P), with 25, 20, and four exceedances of the MELCCFP groundwater protection RES criterion, respectively, were also confirmed as COPCs. Lithium (Li), and nickel (Ni) were found to constitute a minor risk with one punctual exceedance each.

It should be highlighted that arsenic (As), antimony (Sb), and total phosphorous (P) leaching potential were especially associated with the ultramafic material, although the amphibolite samples also showed numerous arsenic exceedances especially with regards to the MDMER effluent guidelines. The copper exceedances, on the other hand, were found among all material types and lithologies but were punctual with no apparent trend.

Finally, it should be underlined that the MELCCFP groundwater protection criterion for copper (0.0016 mg/L) is less than an order of magnitude above the detection limit (0.0002 mg/L), and the limit of quantification of the method could, therefore, have been influenced by the number of exceedances. However, given that copper solubility is controlled by pH, this metal could become a more prominent concern should acidic conditions associated with sulfidic minerals oxidation emerge.

Table 12: Summary of weekly COPC exceedances per HCT with regards to MELCCFP classification criteria and MDMER and Directive 019 effluent regulations

HCT ID (at 40 weeks)	Lithology or material type	As Exceedance		Cu Exceedance	Li Exceedance	Ni Exceedance	Sb Exceedance	Total P Exceedance
		MELCCFP	MDMER	MELCCFP	MELCCFP	MELCCFP	MELCCFP	MELCCFP
CV5-WR-HCT-2	Amphibolite	0	0	0	0	0	0	0
CV5-WR-HCT-5	Amphibolite	15	40	1	0	0	0	0
CV5-WR-HCT-5	Amphibolite	2	40	1	0	0	0	0
CV5-WR-HCT-14	Amphibolite	0	4	0	0	0	0	0
CV5-WR-HCT-22	Amphibolite	3	39	1	0	0	0	0
CV5-WR-HCT-33	Amphibolite	0	0	0	0	0	0	0
CV5-WR-HCT-36	Ultramafic	40	41	1	0	5	7	4
CV5-WR-HCT-39	Metasediment	0	0	0	0	0	0	0
CV5-WR-HCT-50	Metasediment	0	0	1	0	0	0	0
CV5-WR-HCT-56	Metasediment	0	0	1	0	0	0	0
CV5-WR-HCT-59	Ultramafic	24	40	1	0	0	0	0
CV5-WR-HCT-62	Ultramafic	40	41	0	0	1	18	0
CV5-WR-HCT-64	Barren pegmatite	1	5	2	0	0	0	0
CV5-WR-HCT-69	Barren pegmatite	0	3	2	0	0	0	0
CV5-WR-HCT-74	Ore	3	5	3	1	0	0	0
CV5-WR-HCT-76	Ore	0	0	2	0	0	0	0
CV5-DMS-HCT-83	DMS tailings	0	1	2	0	0	0	0
CV5-DMS-HCT-84	DMS tailings	0	0	2	0	0	0	0
Total		**128**	**259**	**20**	**1**	**6**	**25**	**4**

4. Conclusion

A total 84 solid samples of tailings (DMS; n=2), ore (spodumene pegmatite; n=8) and waste rock material (n=80) of different lithologies, including amphibolite (n=30), metasediment (n=25), barren pegmatite (n=13), and ultramafic (n=6) were subjected to static testing to establish whether they constitute a risk in terms of acid generation and metal leaching potential.

The ABA results revealed that total sulfur was a good, yet conservative, predictor of AP with an average 83% of sulfur being in the form of sulfides. On the other hand, total carbon provided a poor predictor of NP, as solely 66% of the carbon in average being found in the form of carbonate, and a significant portion of the NP being likely originated from silicate minerals. For future static testing it is, therefore, recommended that bulk-NP remains the preferred approach to measure NP.

Among the collected samples (n=84), a total of 20 were classified as PAG. The amphibolite (n=6, 19%) and metasediment (n=13, 54%) waste rocks had the highest proportion of PAG samples, whereas the ultramafic presented one PAG sample (17%), and the barren pegmatite did not present any. The ore and tailings samples were all classified as non-PAG based because of their negligible total sulfur concentrations (<0.04% total sulfur).

The Aqua Regia extractible content analysis have allowed to identify a wide range of enriched metal(loid)s with regards to the (10 times) continental crust concentrations and the Superior geological province background concentration; however, arsenic (As) was identified as the most enriched COPC in both cases.

The SPLP and CTEU-9 leaching tests partially reflected the enrichments measured with the Aqua Regia extractions, but also highlighted aluminium (Al) as a main leaching potential concern. Arsenic (As) was especially identified as a concern for the CTEU-9 test with 64 exceedances (76%) as opposed to four exceedances (4.8%) for the SPLP test, thus suggesting that the particle size and pH has an important impact on the mobility of this element. Moreover, arsenic exceedances were observed for all lithologies and material types, but was particular apparent in the ultramafic samples (100%), the tailings (100%), and the spodumene pegmatite ore material (88%).

Finally, arsenic was the only COPC identified from the TCLP leach test with the only two leachate concentrations exceeding high-risk criteria being from the ultramafic samples (n=6; 33%).

Kinetic testing was initiated on select samples to confirm the observations and interpretations concluded from Phase 1 static testing. The 17 samples subjected to the HCT were selected on

the basis of static test results in order to cover a wide range of acid generation and metal leaching potential.

After 40 weeks, the HCT results did not allow to confirm whether the samples classified as PAG via ABA indeed generate net acidity. However, the PAG characteristics of these samples should not be excluded, as net acidity production typically becomes apparent after near-complete exhaustion of the NP of the material.

Among the COPCs identified as part of the leaching test, arsenic (As) was found to constitute the principal potential risk of exceedance of the RES groundwater protection criterion, and the final effluent discharge MDMER criterion with a total of 128, and 259 weekly exceedances, respectively, distributed among the 17 HCTs. Antimony (Sb), copper (Cu), and total phosphorous (P), with 25, 20, and four exceedances of the MELCCFP groundwater protection RES criterion, respectively were also confirmed as COPCs. Lithium (Li) and nickel (Ni) were found to constitute a minor risk with one punctual exceedance each.

It should be highlighted that arsenic (As), antimony (Sb), and total phosphorous (P) leaching potential were all associated with the ultramafic material, although some arsenic exceedances were also noted among amphibolite samples, especially with regards to the MDMER criterion.

Finally the copper exceedances were found among all material types and lithologies but were punctual with no apparent trend, and could have partly been influenced by the limit of quantification of the method. However, given that copper solubility is controlled by pH, this chemical could become a more prominent concern should acidic conditions due do sulfidic oxidation emerge.

5. Recommendations

As this report is being issued, the HCT have been operating for 40 weeks. The weekly monitoring of the chemical parameters conducted on the HCT flush water present steady (equilibrated) results for most parameters. pH readings appear to have stabilized over the most recent flushes but some HCTs (e.g., CV5-DMS-HCT-83, and CV5-DMS-HCT-84) have shown recent decreasing trends close to the lower limit of the MDMER criterion (pH=6). Furthermore, arsenic values have been on the decline for most HCTs and the trend appears to be continuing to decrease.

Considering the current state of kinetic testing, continued weekly HCT flushing is thus recommended until further notice, with a reassessment at 60 weeks. If further testing is required beyond 60 weeks, a decrease in the analysis frequency is recommended to optimize data collection versus cost savings. For example, because the results are starting to show signs of the

slow kinetic evolution, decreasing the analytical frequency to one analysis every four weeks is justified, with pH and conductivity monitored on a weekly basis.

The relatively low NP and AP of the material makes the characterization as PAG and NPAG more complex to understand. Although static testing highlighted PAG characteristics in some samples, the kinetic results did not confirm these observations at this point, suggesting that certain material initially considered to be PAG material is actually NPAG based on kinetic testing. The Net Acid Generation (NAG) test is a suitable ARD prediction tool (Miller et al. 1997) that can be employed as a supplement to static tests. We recommend conducting NAG tests on a sub-set of samples to better understand PAG/NPAG characteristics, to therefore provide further clarity and confidence regarding acid generation risk.

To better support future economic studies at the PFS and FS level, while also meeting the requirements for the ESIA, it is recommended that additional static and kinetic testing is completed for CV5. Informed from the current study and gap analysis, a future sampling program design should be focused on a more complete distribution of arsenic concentrations, while also exploring variability in sulfide content.

Furthermore, it is recommended that field barrel testing (FBT) is initiated to simulate CV5 waste rock stockpiles. Similar to HCTs, reaction rates are simulated and quantified under atmospheric weathering conditions. However, FBTs are often conducted directly at the mine development site (or geographically near) with large quantities of material (hundreds of kilograms) per sample, where the natural climate conditions, such as monthly rainfall and winter conditions, help provide more realistic predictions regarding reaction rates and seasonal flushing.

6. References

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Appendix A: Summary of Composite Sample Details

Composite Sample	Hole Number	From	To	Length	Sample_ID
1	CV22-034	87.63	88.5	0.87	CV034-033
	CV22-034	88.5	89.5	1	CV034-034
	CV22-034	89.5	90.51	1.01	CV034-035
	CV22-034	90.51	91.5	0.99	CV034-036
	CV22-034	91.5	92.7	1.2	CV034-038
2	CV22-034	137.5	139	1.5	CV034-060
	CV22-034	139	140	1	CV034-061
	CV22-034	140	141	1	CV034-063
	CV22-034	141	142	1	CV034-064
	CV22-034	142	143.32	1.32	CV034-065
3	CV22-034	191	192.51	1.51	CV034-094
	CV22-034	192.51	194	1.49	CV034-095
	CV22-034	194	195.5	1.5	CV034-096
	CV22-034	195.5	197	1.5	CV034-097
	CV22-034	197	198.5	1.5	CV034-098
	CV22-034	198.5	200	1.5	CV034-100
	CV22-034	200	201.5	1.5	CV034-101
4	CV22-034	294.03	295.5	1.47	CV034-194
	CV22-034	295.5	297	1.5	CV034-195
	CV22-034	297	298.5	1.5	CV034-196
	CV22-034	298.5	300	1.5	CV034-197
	CV22-034	300	301.4	1.4	CV034-198
	CV22-034	301.4	302	0.6	CV034-200
	CV22-034	302	303.5	1.5	CV034-201
5	CV22-034	312.32	313.54	1.22	CV034-213
	CV22-034	313.54	314.7	1.16	CV034-214
	CV22-034	314.7	315.95	1.25	CV034-215
	CV22-034	315.95	317	1.05	CV034-216
	CV22-034	317	318	1	CV034-217
	CV22-034	318	319.12	1.12	CV034-219
	CV22-034	319.12	320.36	1.24	CV034-220
6	CV22-033	5.1	6.6	1.5	CV033-002
	CV22-033	6.6	8.1	1.5	CV033-003
	CV22-033	8.1	9.6	1.5	CV033-004
	CV22-033	9.6	11.1	1.5	CV033-007
	CV22-033	11.1	12.6	1.5	CV033-008
	CV22-033	12.6	14.1	1.5	CV033-009
	CV22-033	14.1	15.6	1.5	CV033-010
7	CV22-033	103.5	105	1.5	CV033-059
	CV22-033	105	106.5	1.5	CV033-060
	CV22-033	106.5	108	1.5	CV033-061
	CV22-033	108	109.5	1.5	CV033-063
	CV22-033	109.5	111	1.5	CV033-064
	CV22-033	111	112.5	1.5	CV033-065
	CV22-033	112.5	114	1.5	CV033-066
	CV22-033	207.5	209	1.5	CV033-170

Composite Sample	Hole Number	From	To	Length	Sample_ID
8	CV22-033	209	210.5	1.5	CV033-171
	CV22-033	210.5	212	1.5	CV033-172
	CV22-033	212	213.5	1.5	CV033-173
	CV22-033	213.5	215	1.5	CV033-175
	CV22-033	215	216.5	1.5	CV033-176
	CV22-033	216.5	218	1.5	CV033-177
9	CV22-033	231.5	233	1.5	CV033-190
	CV22-033	233	234.5	1.5	CV033-191
	CV22-033	234.5	236	1.5	CV033-192
	CV22-033	236	237.5	1.5	CV033-194
	CV22-033	237.5	239	1.5	CV033-195
	CV22-033	239	240.5	1.5	CV033-196
	CV22-033	240.5	242	1.5	CV033-197
10	CV22-031	93.5	95	1.5	CV031-021
	CV22-031	95	96.5	1.5	CV031-022
	CV22-031	96.5	98	1.5	CV031-023
	CV22-031	98	99.5	1.5	CV031-025
	CV22-031	99.5	100.5	1	CV031-026
11	CV22-031	195.17	196.7	1.53	CV031-139
	CV22-031	196.7	197.8	1.1	CV031-140
	CV22-031	197.8	198.96	1.16	CV031-141
	CV22-031	198.96	200.5	1.54	CV031-142
	CV22-031	200.5	202	1.5	CV031-144
	CV22-031	202	203.5	1.5	CV031-145
12	CV22-029	31.5	33	1.5	CV029-011
	CV22-029	33	34.5	1.5	CV029-013
	CV22-029	34.5	36	1.5	CV029-014
	CV22-029	36	37.5	1.5	CV029-015
	CV22-029	37.5	39	1.5	CV029-016
	CV22-029	39	40.5	1.5	CV029-017
	CV22-029	40.5	42	1.5	CV029-019
13	CV22-028	232.93	234.5	1.57	CV028-151
	CV22-028	234.5	236	1.5	CV028-152
	CV22-028	237.05	238.5	1.45	CV028-153
	CV22-028	238.5	240	1.5	CV028-154
	CV22-028	242.38	244	1.62	CV028-157
	CV22-028	244	245.5	1.5	CV028-158
	CV22-028	245.5	247	1.5	CV028-159
14	CV22-028	275	276.5	1.5	CV028-170
	CV22-028	276.5	278	1.5	CV028-171
	CV22-028	278	279.5	1.5	CV028-172
	CV22-028	279.5	281	1.5	CV028-173
	CV22-028	281	282	1	CV028-175
	CV22-028	282	283.5	1.5	CV028-176
	CV22-028	283.5	285	1.5	CV028-177
	CV22-027	9	10	1	CV027-120

Composite Sample	Hole Number	From	To	Length	Sample_ID
15	CV22-027	10	11.5	1.5	CV027-121
	CV22-027	11.5	13	1.5	CV027-122
	CV22-027	13	14.5	1.5	CV027-123
	CV22-027	14.5	16	1.5	CV027-001
	CV22-027	16	17.5	1.5	CV027-125
16	CV22-027	107.5	109	1.5	CV027-096
	CV22-027	109	110.5	1.5	CV027-097
	CV22-027	110.5	112	1.5	CV027-098
	CV22-027	112	113.5	1.5	CV027-100
	CV22-027	113.5	115	1.5	CV027-101
	CV22-027	115	116.5	1.5	CV027-102
	CV22-027	116.5	118	1.5	CV027-103
17	CV22-027	118	119.5	1.5	CV027-104
	CV22-027	119.5	121	1.5	CV027-107
	CV22-027	121	122.5	1.5	CV027-108
	CV22-027	122.5	124	1.5	CV027-109
	CV22-027	124	125.5	1.5	CV027-110
	CV22-027	125.5	127	1.5	CV027-111
18	CV22-025	12.5	14	1.5	CV025-102
	CV22-025	14	15.5	1.5	CV025-103
	CV22-025	15.5	17	1.5	CV025-104
	CV22-025	17	18.5	1.5	CV025-107
	CV22-025	18.5	20	1.5	CV025-108
	CV22-025	20	21.5	1.5	CV025-001
	CV22-025	21.5	22.72	1.22	CV025-002
19	CV22-025	127	128.5	1.5	CV025-119
	CV22-025	128.5	130	1.5	CV025-120
	CV22-025	130	131.5	1.5	CV025-121
	CV22-025	131.5	133	1.5	CV025-122
	CV22-025	133	134.5	1.5	CV025-123
	CV22-025	134.5	136	1.5	CV025-125
	CV22-025	136	137.5	1.5	CV025-126
20	CV22-023	51	52.5	1.5	CV023-003
	CV22-023	52.5	53.3	0.8	CV023-004
	CV22-023	53.3	53.95	0.65	CV023-007
	CV22-023	53.95	55	1.05	CV023-008
21	CV22-020	166.38	167	0.62	CV020-041
	CV22-020	167	168.5	1.5	CV020-042
	CV22-020	168.5	170	1.5	CV020-044
	CV22-020	170	171.5	1.5	CV020-045
	CV22-020	171.5	173	1.5	CV020-046
	CV22-020	174.5	175.8	1.3	CV020-047
22	CV22-019	207.32	209	1.68	CV019-132
	CV22-019	209	210.5	1.5	CV019-133
	CV22-019	210.5	212	1.5	CV019-145
	CV22-019	212	213.5	1.5	CV019-146

Composite Sample	Hole Number	From	To	Length	Sample_ID
	CV22-019	213.5	215	1.5	CV019-147
	CV22-019	215	216.35	1.35	CV019-148
	CV22-019	216.35	217.5	1.15	CV019-150
23	CV22-018	28.15	29.2	1.05	CV018-007
	CV22-018	29.2	30.2	1	CV018-008
	CV22-018	30.2	31.2	1	CV018-009
	CV22-018	31.2	32.15	0.95	CV018-010
	CV22-018	32.15	33.35	1.2	CV018-011
	CV22-018	33.35	34.4	1.05	CV018-013
	CV22-018	34.4	36	1.6	CV018-014
	CV22-018	36	37.5	1.5	CV018-015
24	CV22-018	120.5	122	1.5	CV018-072
	CV22-018	122	123.5	1.5	CV018-073
	CV22-018	123.5	124.96	1.46	CV018-075
	CV22-018	124.96	126.11	1.15	CV018-076
	CV22-018	126.11	127.5	1.39	CV018-077
	CV22-018	127.5	129	1.5	CV018-078
25	CV22-017	83	83.9	0.9	CV017-127
	CV22-017	83.9	84.85	0.95	CV017-128
	CV22-017	84.85	86.17	1.32	CV017-129
	CV22-017	86.17	87.31	1.14	CV017-132
	CV22-017	87.31	88.38	1.07	CV017-133
	CV22-017	88.38	89.5	1.12	CV017-134
26	CV22-017	243	244.5	1.5	CV017-139
	CV22-017	244.5	245.95	1.45	CV017-140
	CV22-017	245.95	247	1.05	CV017-141
	CV22-017	247	248.5	1.5	CV017-142
	CV22-017	248.5	249.95	1.45	CV017-144
	CV22-017	249.95	251.1	1.15	CV017-145
	CV22-017	251.1	252	0.9	CV017-146
27	CV22-015	138.5	140	1.5	CV015-071
	CV22-015	140	141	1	CV015-072
	CV22-015	141	142	1	CV015-073
	CV22-015	142	143	1	CV015-075
	CV22-015	143	144	1	CV015-076
	CV22-015	144	145.5	1.5	CV015-077
	CV22-015	145.5	147	1.5	CV015-078
	CV22-015	147	148	1	CV015-079
28	CF21-003	81.07	82	0.93	CF003-073
	CF21-003	82	83.5	1.5	CF003-074
	CF21-003	83.5	85	1.5	CF003-075
	CF21-003	86	87	1	CF003-076
29	CF21-001	182.84	183.87	1.03	CF001-207
	CF21-001	183.87	184.64	0.77	CF001-208
	CF21-001	184.64	186.34	1.7	CF001-209
	CF21-001	186.34	186.77	0.43	CF001-210

Composite Sample	Hole Number	From	To	Length	Sample_ID
	CF21-001	186.77	188.39	1.62	CF001-211
	CF21-001	188.39	189.99	1.6	CF001-213
	CF21-001	189.99	190.51	0.52	CF001-214
30	CF21-001	219	220.5	1.5	CF001-245
	CF21-001	220.5	222	1.5	CF001-246
	CF21-001	222	223.5	1.5	CF001-247
	CF21-001	223.5	225	1.5	CF001-248
	CF21-001	225	226.5	1.5	CF001-249
	CF21-001	226.5	228	1.5	CF001-250
	CF21-001	228	229.14	1.14	CF001-251
31	CV22-034	178.9	179.9	1	CV034-078
	CV22-034	179.9	180.51	0.61	CV034-079
32	CV22-034	208.54	210	1.46	CV034-109
	CV22-034	210	211.45	1.45	CV034-110
	CV22-034	211.45	213	1.55	CV034-111
	CV22-034	213	214.47	1.47	CV034-113
	CV22-034	214.47	215.22	0.75	CV034-114
33	CV22-034	224.18	225.5	1.32	CV034-123
	CV22-034	225.5	227	1.5	CV034-125
	CV22-034	227	228.5	1.5	CV034-126
	CV22-034	228.5	230	1.5	CV034-127
	CV22-034	230	231.51	1.51	CV034-128
34	CV22-034	266.74	268	1.26	CV034-169
	CV22-034	268	269.5	1.5	CV034-170
	CV22-034	269.5	271	1.5	CV034-171
	CV22-034	271	272.5	1.5	CV034-172
	CV22-034	272.5	273.23	0.73	CV034-173
35	CV22-034	279.5	281	1.5	CV034-182
	CV22-034	281	282.5	1.5	CV034-183
	CV22-034	282.5	284	1.5	CV034-184
	CV22-034	284	285.5	1.5	CV034-185
	CV22-034	285.5	287	1.5	CV034-187
	CV22-034	287	288	1	CV034-188
36	CV22-030	253.95	255.35	1.4	CV030-214
	CV22-030	255.35	257	1.65	CV030-215
37	CV22-027	51.67	53	1.33	CV027-023
	CV22-027	53	54	1	CV027-025
	CV22-027	54	55.07	1.07	CV027-026
38	CV22-026	54.76	55.17	0.41	CV026-024
	CV22-026	55.17	55.47	0.3	CV026-025
	CV22-026	55.47	56.25	0.78	CV026-026
39	CV22-025	85.3	86.5	1.2	CV025-083
	CV22-025	86.5	88	1.5	CV025-084
	CV22-025	88	89.05	1.05	CV025-085
	CV22-025	89.05	90.63	1.58	CV025-086
40	CV22-024	66.38	67.88	1.5	CV024-035

Composite Sample	Hole Number	From	To	Length	Sample_ID
40	CV22-024	67.88	69.38	1.5	CV024-036
41	CV22-022	101.5	103	1.5	CV022-053
	CV22-022	103	104.5	1.5	CV022-054
	CV22-022	104.5	106	1.5	CV022-057
	CV22-022	106	107.5	1.5	CV022-058
	CV22-022	107.5	109	1.5	CV022-059
42	CV22-022	109	110.5	1.5	CV022-060
	CV22-022	110.5	112	1.5	CV022-061
	CV22-022	112	113.5	1.5	CV022-063
	CV22-022	113.5	115	1.5	CV022-064
	CV22-022	115	116.5	1.5	CV022-065
43	CV22-022	116.5	118	1.5	CV022-066
	CV22-022	118	119.5	1.5	CV022-067
	CV22-022	119.5	121	1.5	CV022-069
	CV22-022	121	122.5	1.5	CV022-070
	CV22-022	122.5	124	1.5	CV022-071
44	CV22-021	72	73.5	1.5	CV021-015
	CV22-021	73.5	75	1.5	CV021-016
	CV22-021	75	76	1	CV021-027
	CV22-021	76	76.74	0.74	CV021-028
	CV22-021	76.74	78	1.26	CV021-029
	CV22-021	78	79.05	1.05	CV021-032
	CV22-021	79.05	80	0.95	CV021-033
45	CV22-021	137	138	1	CV021-046
	CV22-021	138	139	1	CV021-047
	CV22-021	139	140	1	CV021-048
	CV22-021	140	141	1	CV021-050
	CV22-021	141	142	1	CV021-051
	CV22-021	142	143	1	CV021-052
	CV22-021	143	144	1	CV021-053
	CV22-021	144	145	1	CV021-054
	CV22-021	145	146	1	CV021-057
46	CV22-021	147	147.93	0.93	CV021-059
	CV22-021	147.93	149.1	1.17	CV021-060
	CV22-021	149.1	150.5	1.4	CV021-061
	CV22-021	150.5	151.55	1.05	CV021-063
	CV22-021	151.55	153	1.45	CV021-064
	CV22-021	153	154	1	CV021-065
47	CV22-019	54	55	1	CV019-140
	CV22-019	55	56	1	CV019-141
	CV22-019	56	57.15	1.15	CV019-142
	CV22-019	57.15	58	0.85	CV019-144
48	CF21-002	233	234	1	CF002-195
	CF21-002	234	235.5	1.5	CF002-196
	CF21-002	235.5	237	1.5	CF002-197
49	CV22-034	326.73	327.9	1.17	CV034-229

Composite Sample	Hole Number	From	To	Length	Sample_ID
49	CV22-034	327.9	328.73	0.83	CV034-232
50	CV22-030	79.4	80.5	1.1	CV030-011
	CV22-030	80.5	82.01	1.51	CV030-013
	CV22-030	82.01	83.5	1.49	CV030-014
	CV22-030	83.5	85	1.5	CV030-015
	CV22-030	85	86.38	1.38	CV030-016
51	CV22-022	80.94	82.5	1.56	CV022-045
	CV22-022	82.5	84	1.5	CV022-046
	CV22-022	84	85.55	1.55	CV022-047
	CV22-022	85.55	87	1.45	CV022-048
	CV22-022	87	88.5	1.5	CV022-050
52	CV22-022	20	21	1	CV022-076
	CV22-022	21	22	1	CV022-077
	CV22-022	22	23	1	CV022-078
	CV22-022	23	24	1	CV022-079
	CV22-022	24	25	1	CV022-082
	CV22-022	25	26	1	CV022-083
53	CV22-018	105	105.83	0.83	CV018-059
	CV22-018	105.83	107.17	1.34	CV018-060
	CV22-018	107.17	108.43	1.26	CV018-061
	CV22-018	108.43	109.5	1.07	CV018-063
54	CF21-001	14.17	15.08	0.91	CF001-014
	CF21-001	15.08	15.99	0.91	CF001-015
	CF21-001	15.99	16.11	0.12	CF001-017
	CF21-001	16.11	17.5	1.39	CF001-018
	CF21-001	17.5	19.01	1.51	CF001-019
	CF21-001	19.01	20.5	1.49	CF001-020
	CF21-001	20.5	22	1.5	CF001-021
	CF21-001	22	22.96	0.96	CF001-022
55	CV22-034	6.58	7.58	1	CV034-001
	CV22-034	7.58	9.2	1.62	CV034-002
	CV22-034	9.2	10.87	1.67	CV034-003
	CV22-034	10.87	12.5	1.63	CV034-004
56	CV22-034	35.5	37	1.5	CV034-009
	CV22-034	37	38.5	1.5	CV034-010
	CV22-034	38.5	40	1.5	CV034-011
	CV22-034	40	41.5	1.5	CV034-013
	CV22-034	41.5	43	1.5	CV034-014
	CV22-034	43	44.5	1.5	CV034-015
	CV22-034	44.5	46	1.5	CV034-016
57	CV22-034	79.82	80.5	0.68	CV034-021
	CV22-034	80.5	81.32	0.82	CV034-022
	CV22-034	81.32	82	0.68	CV034-023
	CV22-034	82	83	1	CV034-025
	CV22-034	83	84	1	CV034-026
	CV22-034	84	85	1	CV034-027

Composite Sample	Hole Number	From	To	Length	Sample_ID
	CV22-034	85	86	1	CV034-028
	CV22-034	86	87	1	CV034-029
	CV22-034	87	87.63	0.63	CV034-032
58	CV22-034	134	135	1	CV034-054
	CV22-034	135	135.96	0.96	CV034-057
	CV22-034	135.96	137	1.04	CV034-058
	CV22-034	137	137.5	0.5	CV034-059
59	CV22-032	106.27	107	0.73	CV032-015
	CV22-032	107	108	1	CV032-016
	CV22-032	108	109	1	CV032-017
	CV22-032	109	110.5	1.5	CV032-019
60	CV22-030	239.15	240.5	1.35	CV030-211
	CV22-030	240.5	242	1.5	CV030-213
61	CV22-028	77.91	78.91	1	CV028-007
	CV22-028	78.91	80	1.09	CV028-008
	CV22-028	80	80.96	0.96	CV028-009
	CV22-028	80.96	81.96	1	CV028-010
62	CV22-016	85.53	85.88	0.35	CV016-004
	CV22-016	86.2	87.67	1.47	CV016-005
	CV22-016	87.67	89.19	1.52	CV016-008
63	CV22-033	128.7	129.6	0.9	CV033-079
	CV22-033	129.6	130.5	0.9	CV033-082
	CV22-033	130.5	131.55	1.05	CV033-083
	CV22-033	131.55	132.5	0.95	CV033-084
	CV22-033	132.5	133.66	1.16	CV033-085
64	CV22-030	146.8	148	1.2	CV030-095
	CV22-030	148	149	1	CV030-096
	CV22-030	149	150	1	CV030-097
	CV22-030	150	151	1	CV030-098
	CV22-030	151	152	1	CV030-100
	CV22-030	152	153	1	CV030-101
	CV22-030	153	154.1	1.1	CV030-102
	CV22-030	154.1	155	0.9	CV030-103
	CV22-030	155	156	1	CV030-104
65	CV22-029	106	107	1	CV029-085
	CV22-029	107	107.6	0.6	CV029-086
	CV22-029	107.6	107.95	0.35	CV029-088
	CV22-029	107.95	109	1.05	CV029-089
	CV22-029	109	110	1	CV029-090
	CV22-029	110	111	1	CV029-091
	CV22-029	111	112	1	CV029-092
	CV22-029	112	113	1	CV029-094
	CV22-029	113	114	1	CV029-095
	CV22-029	114	115	1	CV029-096
	CV22-029	115	116	1	CV029-097
	CV22-026	127.23	128.25	1.02	CV026-108

Composite Sample	Hole Number	From	To	Length	Sample_ID
66	CV22-026	128.25	129.25	1	CV026-109
	CV22-026	129.25	130.3	1.05	CV026-110
	CV22-026	130.3	131.25	0.95	CV026-111
	CV22-026	131.25	132.39	1.14	CV026-113
	CV22-026	132.39	133.43	1.04	CV026-114
	CV22-026	133.43	134.5	1.07	CV026-115
	CV22-026	134.5	135.5	1	CV026-116
	CV22-026	135.5	136.5	1	CV026-117
	CV22-026	136.5	137.5	1	CV026-119
67	CV22-016	125.13	126.03	0.9	CV016-053
	CV22-016	126.03	127	0.97	CV016-054
	CV22-016	127	128	1	CV016-057
	CV22-016	128	129	1	CV016-058
	CV22-016	129	130	1	CV016-059
	CV22-016	130	131	1	CV016-060
	CV22-016	131	132.08	1.08	CV016-061
	CV22-016	132.08	133.26	1.18	CV016-063
	CV22-016	133.26	134.45	1.19	CV016-064
68	CF21-002	89	90	1	CF002-028
	CF21-002	90	91	1	CF002-029
	CF21-002	91	92	1	CF002-030
	CF21-002	92	93	1	CF002-032
	CF21-002	93	94	1	CF002-033
	CF21-002	94	95	1	CF002-034
	CF21-002	95	96	1	CF002-035
	CF21-002	96	97	1	CF002-036
	CF21-002	97	98	1	CF002-038
	CF21-002	98	99	1	CF002-039
69	CV22-033	180	181	1	CV033-144
	CV22-033	181	182	1	CV033-145
	CV22-033	182	183	1	CV033-146
	CV22-033	183	184	1	CV033-147
	CV22-033	184	185	1	CV033-148
	CV22-033	185	186	1	CV033-149
	CV22-033	186	187	1	CV033-151
	CV22-033	187	188	1	CV033-152
	CV22-033	188	189	1	CV033-153
	CV22-033	189	190	1	CV033-154
70	CV22-026	105	106	1	CV026-079
	CV22-026	106	107	1	CV026-082
	CV22-026	107	108	1	CV026-083
	CV22-026	108	109	1	CV026-084
	CV22-026	109	110	1	CV026-085
	CV22-026	110	111	1	CV026-086
	CV22-026	111	112	1	CV026-088
	CV22-026	112	113	1	CV026-089

Composite Sample	Hole Number	From	To	Length	Sample_ID
	CV22-026	113	114	1	CV026-090
	CV22-026	114	115	1	CV026-091
71	CF21-004	72	73	1	CF004-047
	CF21-004	73	74	1	CF004-048
	CF21-004	74	75	1	CF004-049
	CF21-004	75	76	1	CF004-050
	CF21-004	76	77	1	CF004-051
	CF21-004	77	78	1	CF004-052
	CF21-004	78	79	1	CF004-053
	CF21-004	79	80	1	CF004-054
	CF21-004	80	81	1	CF004-056
	CF21-004	81	82	1	CF004-057
72	CF21-001	179.09	180.09	1	CF001-202
	CF21-001	180.09	181	0.91	CF001-203
	CF21-001	181	181.92	0.92	CF001-205
	CF21-001	181.92	182.84	0.92	CF001-206
73	CV22-034	240	241	1	CV034-139
	CV22-034	241	242	1	CV034-140
	CV22-034	242	243	1	CV034-141
	CV22-034	243	244	1	CV034-142
	CV22-034	244	245	1	CV034-144
	CV22-034	245	246	1	CV034-145
	CV22-034	246	247	1	CV034-146
	CV22-034	247	248	1	CV034-147
	CV22-034	248	249	1	CV034-148
	CV22-034	249	250	1	CV034-150
74	CV22-030	172	173	1	CV030-127
	CV22-030	173	174	1	CV030-128
	CV22-030	174	175	1	CV030-129
	CV22-030	175	176	1	CV030-132
	CV22-030	176	177	1	CV030-133
	CV22-030	177	178	1	CV030-134
	CV22-030	178	179	1	CV030-135
	CV22-030	179	180	1	CV030-136
	CV22-030	180	181	1	CV030-138
	CV22-030	181	182	1	CV030-139
75	CV22-028	190	191	1	CV028-097
	CV22-028	191	192	1	CV028-098
	CV22-028	192	193	1	CV028-100
	CV22-028	193	194	1	CV028-101
	CV22-028	194	195	1	CV028-102
	CV22-028	195	196	1	CV028-103
	CV22-028	196	197	1	CV028-104
	CV22-028	197	198	1	CV028-107
	CV22-028	198	199	1	CV028-108
	CV22-028	199	200	1	CV028-109

Composite Sample	Hole Number	From	To	Length	Sample_ID
76	CV22-026	115	116	1	CV026-092
	CV22-026	116	117	1	CV026-094
	CV22-026	117	118	1	CV026-095
	CV22-026	118	119	1	CV026-096
	CV22-026	119	120	1	CV026-097
	CV22-026	120	121	1	CV026-098
	CV22-026	121	122	1	CV026-100
	CV22-026	122	123	1	CV026-101
	CV22-026	123	124	1	CV026-102
	CV22-026	124	125	1	CV026-103
77	CV22-017	195	196	1	CV017-060
	CV22-017	196	197	1	CV017-061
	CV22-017	197	198	1	CV017-063
	CV22-017	198	199	1	CV017-064
	CV22-017	199	200	1	CV017-065
	CV22-017	200	201	1	CV017-066
	CV22-017	201	202	1	CV017-067
	CV22-017	202	203	1	CV017-069
	CV22-017	203	204	1	CV017-070
	CV22-017	204	205	1	CV017-071
78	CF21-001	127.8	128.86	1.06	CF001-145
	CF21-001	128.86	129.87	1.01	CF001-146
	CF21-001	129.87	130.94	1.07	CF001-147
	CF21-001	130.94	132	1.06	CF001-148
	CF21-001	132	133.05	1.05	CF001-149
	CF21-001	133.05	134.13	1.08	CF001-150
	CF21-001	134.13	135.03	0.9	CF001-151
	CF21-001	135.03	136.03	1	CF001-153
	CF21-001	136.03	137	0.97	CF001-154
79	CV22-025	46	47	1	CV025-034
	CV22-025	47	48	1	CV025-035
	CV22-025	48	49	1	CV025-036
	CV22-025	49	50	1	CV025-038
	CV22-025	50	51	1	CV025-039
	CV22-025	51	52	1	CV025-040
	CV22-025	52	52.75	0.75	CV025-041
	CV22-025	52.75	54	1.25	CV025-042
	CV22-025	54	54.85	0.85	CV025-044
80	CV22-015	32.88	34	1.12	CV015-010
	CV22-015	34	35	1	CV015-013
	CV22-015	35	36	1	CV015-014
	CV22-015	36	37	1	CV015-015
	CV22-015	37	38	1	CV015-016
	CV22-015	38	39	1	CV015-017
	CV22-015	39	39.55	0.55	CV015-019
	CF21-002	208	209	1	CF002-165

Composite Sample	Hole Number	From	To	Length	Sample_ID
81	CF21-002	209	210	1	CF002-167
	CF21-002	210	211	1	CF002-168
	CF21-002	211	212	1	CF002-169
	CF21-002	212	213	1	CF002-170
	CF21-002	213	214	1	CF002-172
	CF21-002	214	215	1	CF002-173
	CF21-002	215	216	1	CF002-174
	CF21-002	216	217	1	CF002-175
	CF21-002	217	218	1	CF002-176
82	CV22-033	19.83	20.83	1	CV033-015
	CV22-033	20.83	21.83	1	CV033-016
	CV22-033	21.83	22.86	1.03	CV033-017
	CV22-033	22.86	23.9	1.04	CV033-019
	CV22-033	23.9	24.95	1.05	CV033-020