Exhibit 96.2

SEC TECHNICAL REPORT SUMMARY

Pre-Feasibility Study

ON THE

WHABOUCHI MINE

NEMASKA, QUEBEC

UTM Zone 18N 441000 m E; 5725750 m N

Prepared for:
Livent Corporation

Report Date: September 8, 2023
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Amended Date: November 14, 2023

Prepared By:
BBA Inc.
DRA Americas Inc.
SGS Geological Services
WSP Canada Inc.

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1 SUMMARY

1.1 Introduction

The Whabouchi Property is located in the Eeyou Istchee/James Bay area of the Province of Quebec, approximately 30 km east of the community of Nemaska and 300 km north-northwest of the town of Chibougamau, more specifically at km 276 on the Route du Nord.

The Whabouchi Mine is lithium-bearing spodumene deposit controlled by Nemaska Lithium Inc. (Nemaska or NLI), in which the Registrant (Livent Corporation) owns a 50% interest. The Whabouchi Project comprises mining operations as well as the crushing and concentrating of the ore to produce spodumene concentrate. The concentrator is designed to nominally produce 235,000 dry tonnes per year of 5.5% Li_2O of spodumene concentrate, which is used as feedstock to chemical processing plants for various lithium-bearing end products. This pre-feasibility study is focused on mining, mineral processing and sale of spodumene concentrate produced from the Whabouchi Mine.

In 2018, the Securities and Exchange Commission (SEC) adopted amendments to modernize the property disclosure requirements for mining registrants by requiring disclosures concerning mineral resources and reserves. The amendments more closely align the SEC's disclosure requirements and policies for mining properties with current industry and global regulatory practices and standards (i.e., Committee for Reserves and International Reporting Standards [CRIRSCO]). This report was prepared to pre-feasibility standards in accordance with SEC regulations S-K 601(b) (96) on behalf of Livent Corporation.

1.2 Property Description and Location

The Whabouchi Property is located in the James Bay area of the Province of Quebec, approximately 30 km East of the Cree community of Nemaska and 300 km north-northwest of the town of Chibougamau. The center of the Property is situated approximately at UTM 5,725,750 mN, 441,000 mE, NAD83 Zone 18. The Property is accessible by the *Route du Nord*, the main all-season gravel road linking Chibougamau and Nemaska. The Property is also accessible through Matagami by the *Route Billy-Diamond* Highway. The road crosses the Property near its center. The Nemiscau airport is 18 km west of the Property.

The Property is composed of one block containing 35 map-designated claims (MDC) covering a total of 1,632.24 hectares and one Mining Lease by the *Ministère des Ressources naturelles et forêts* (*MRNF*). NLI owns 100% interest in the Property. At the date of this Report, all claims are in good standing. The expiry date of the claims ranges from November 2, 2024 to January 24, 2025.

On October 26, 2017, NLI obtained the Mining Lease number 1022, under the conditions provided for in the Loi sur les mines (Mining Act) and those prescribed by regulation. The surface of the Mining Lease totals 138.106 ha, consisting of lot 4,994,037 of the Quebec cadastre, registration division of Lac-Saint-Jean-Ouest. This lease gives the tenant the right to extract all mineral substances owned by the Crown in the above-named land, but it does not give entitlement to surface mineral substances, petroleum, natural gas, or brine. This lease is for a period of 20 years from the date of the landlord's signature on October 26, 2017 and will end on October 25, 2037. Mining leases can be renewed three times in 10 year increments for a nominal fee.

There are no royalty obligations on any of the claims of the Property.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.3.1 Physiography

The Property is easily accessible via the Route du Nord that crosses the Property near its center. This road links the town of Matagami, via the *Route Billy-Diamond* highway, approximately 390 km to the SSW. The Route du Nord also links the town of Chibougamau, located approximately 300 km to the SSE, and leads to the community of Nemaska.

1.3.2 Physiography and Climate

The Property is characterized by a relatively flat topography, with the exception of the local ridge where the more competent pegmatites outcrop, forming the surface expression of the deposit. The elevation above sea level ranges from 275 m, at the lowest point on the Property, to 325 m at the top of the pegmatite ridge, with an average elevation of 300 m. Lakes and rivers cover approximately 15% of the Property area. The flora in the area is typical of the taiga environment observed in the region with a mix of black spruce forest and peat moss-covered swamps. A vast portion of the Property was devastated by forest fires less than 20 years ago. There is no permafrost at this latitude and the overburden cover ranges in depth from 0 m near the ridge to 25 m in the south part of the Property.

The climate in the region is sub-arctic. This climate zone is characterized by long, cold winters and short, cool summers. Daily average temperature ranges from -20°C in January to +17°C in July. Break-up usually occurs in early June, and freeze-up in early November. Precipitation mostly occurs between May and November with an average monthly rainfall of 135 mm. Annual snowfall is generally of 315 cm of snow, mostly between October and May. Averages are based on data from 2009 to 2022 (https://www.worldweatheronline.com/nemiscau-weather-averages/quebec/ca.aspx).

1.3.3 Local Resources and Infrastructure

The nearest infrastructure with general services is the *Relais Routier Nemiscau* Camp, located 12 km west of the Property, where NLI has access to lodging facilities, if needs exceed the capacity of the camp installed on the property. The community of Nemaska, located 30 km west of the Property, can also provide accommodation and general services. The area is serviced by the Nemiscau airport, serviced by regular Air Creebec flights and charter flights, and by mobile phone network from the main Canadian service providers.

Hydro-Québec owns several infrastructure and facilities in the area including the Poste Albanel and Poste Nemiscau electrical stations located approximately 20 km east and 12 km west from the Property, respectively. Electrical (735 kV) transmission lines connecting both stations run alongside the Route du Nord and cross the Property near its center. Also, a 69 kV power line connecting the Poste Nemiscau electrical station to the mine site has been put in service and is supplying power to the facilities.

1.3.4 Surface Rights

All claims comprising the Property are located on Crown Lands. NLI secured in October 2017 all surface rights to construct and operate the projected infrastructure.

1.4 **History**

Numerous geological surveys and geoscientific studies have been conducted by the Quebec Government in the James Bay area. Geological surveys in the 1960s (Valiquette, 1964, 1965 and 1975) cover the entire property area. In 1998, the *MRNF* released the results of a regional lake bottom sediment survey completed in 1997.

The first exploration work reported in the area dates back to 1962 by Canico and included the discovery of a lithium-bearing pegmatite by the geologists of the Québec Bureau of Mines. That same year, Canico drilled two (2) packsack drill holes on the pegmatite, followed by three (3) diamond drill holes on the same pegmatite ridge in 1963.

No exploration was reported for the next ten (10) years. In 1973, James Bay Nickel Ventures (Canex Placer) performed a large-scale geological reconnaissance that covered the property (Burns, 1973).

From 1974 to 1982, the exploration work was exclusively reported by the *Société de Développement de la Baie James* (*SDBJ*), which mainly executed large scale geochemical surveys, followed by geological reconnaissance of the anomalies (Pride, 1974, Gleeson, 1975 and 1976).

Two (2) exploration programs, one in 1978 and the other in 1980 were aimed at lithium exploration, with the evaluation of the Whabouchi spodumene-bearing pegmatite (Goyer, et al. 1978, Bertrand, 1978, Otis, 1980, Fortin, 1981, and Charbonneau, 1982). No channel sampling or drill holes are reported. No work was conducted from 1982 to 1987.

In 1987, Westmin Resources completed an airborne Dighem III survey. A part of this survey was located immediately east of the property (McConnell 1987). In 1987-1988, Muscocho Exploration also completed ground magnetic and VLF surveys that covered a major part of the property. The spodumene-bearing pegmatite gave a weak magnetic and VLF response. The Muscocho Exploration efforts were oriented towards the search for massive sulphides. A program of 14 holes, 11 of them located on the southern part of the Whabouchi Property, was completed.

In 2002, while exploring for tantalum, Inco re-sampled the spodumene-bearing pegmatite, taking 11 channel samples and seven (7) grab samples. The best value obtained by Inco was 0.026% Ta, and Li_2O values ranging from 0.30% to 3.72% (Babineau, 2002).

1.5 Geological Setting and Mineralization

The Whabouchi Property is located in the northeast part of the Superior Province of the Canadian Shield craton, in the Lac des Montagnes volcano-sedimentary formation, which comprises metasediments and amphibolites (mafic and ultramafic metavolcanics). At the local scale, the metavolcano-sedimentary sequence is intruded by different bodies of granites and pegmatites with varying composition. At the Property, a spodumene-bearing pegmatite dyke swarm occurs and is composed of interconnecting dykes and plug shaped intrusions. Most of the dykes are steeply dipping towards the southeast. In cross section, some of the dykes have different dip orientation and potentially connect to other dykes at depth. The corridor occupied by the dyke swarm has been recognized on a strike length of 1,340 m with a width ranging from 60 m to 330 m.

The mineralization at Whabouchi is found in spodumene, rare metal-bearing pegmatites. Spodumene is lithium-bearing mineral, which contains 8% Li_2O when pure. Assays for spodumenes normally range between 7.6% and 8% Li_2O depending on the degree of replacement by Na_2O. Typically, the Whabouchi pegmatite sampled from drill core averages 1.42% Li_2O with values up to 5.19% Li_2O. Recent mineralogical assessment shows minor amount of other Li-bearing minerals, such as petalite, muscovite, and holmquistite.

Two distinct phases are observed in the Whabouchi pegmatites: a spodumene-bearing phase comprising most of the pegmatite material and a lesser, white to pink barren quartz-feldspar pegmatite. The lithium mineralization occurs mainly in medium to large spodumene crystals (up to 30 cm in size) but petalite also occurs, averaging approximately 2.3% in the deposit (petalite contains approximately 4.5% Li_2O). Muscovite also contains minor lithium and averages less than 2% in the deposit. Petalite and muscovite are not recoverable by the mineral processing method discussed in the present Report.

1.6 Exploration

Nemaska Lithium Inc. (formerly Nemaska Exploration Inc.), at that time and as part of the Qualifying NI 43-101 Technical Report dated July 14, 2010, initiated its exploration work on the Property during the fall of 2009. During the site visit, several outcrops of spodumene-bearing pegmatite were observed and nine (9) samples were collected and analyzed for Li_2O ranging from 6.3% Li_2O to 1.18% Li_2O (Théberge, 2009).

Following that and during the fall 2009 exploration program, mechanical stripping successfully exposed the spodumene-bearing pegmatites.

In 2010 and in addition to drilling, 14 line-km of ground magnetic surveying covering the main mineralized occurrence and 670 line-km of helicopter-borne magnetic surveying covering the Property were completed. Later in May 2010, 2,780 m of mechanical stripping of the south contact of the Main Zone was completed and allowed the mapping of the surface geology. In May 2011, a 50-tonne bulk sample was collected at surface for metallurgical testing purposes.

1.7 Drilling

A total of 277 diamond drill holes were completed by NLI to define the mineral deposit, for exploration, as well as for geotechnical and metallurgical tests. In addition to the drilling, extensive mechanical stripping at surface permitted the completion of 108 channels. Table 1-1 and Table 1-2 summarize the drilling and channel sampling completed by NLI to define the mineralized pegmatite intrusion and for exploration Northwest of the deposit.

Table 1-1 Drilling Completed by NLI at Whabouchi

Year	Count	Meters Drilled	Number of Lithium Assays
2009	8	999	456
2010	82	15,670	6,088
2011	41	9,264	1,869
2013	14	1,815	350
2013 (exploration)	10	1,308	150
2016	51	17,424	4,038
2017	48	4,361	1,819
2018	14	2,099	818
2018, 2021 (geotech.)	9	1,610	0
Total	**277**	**54,550**	**15,588**

Table 1-2 Channel Sampling done by NLI at Whabouchi

Year	Channels	Total Samples
2009	37	295
2010	71	649
Total	**108**	**944**

1.8 Sample Preparation, Analysis and Security

NLI implemented an internal QA/QC protocol by inserting reference material (non-certified and certified), blanks, core duplicates, pulp duplicates and umpire duplicates. SGS reviewed the QA/QC results of the various campaigns and is of the opinion that the results are adequate for utilization in a mineral resource estimate.

Throughout the years of drilling, sodium-peroxide and 4-acid digestion methods were used to evaluate the lithium content of the samples. In the future, a sodium peroxide fusion analysis method should be used instead of the 4-acid digest method to ensure a full analysis of refractory minerals. The QP suggests continuing its internal QA/QC protocol for blanks, duplicates (core and pulp) and certified reference material.

In 2021 and 2022, SGS was mandated by NLI to conduct an assessment of the bias observed between the main lithium digestion methods used for sample assays (i.e., peroxide fusion and 4-acid). The result of the study shows that 4-acid based digestion underestimates lithium grades by 4%. Thus, it is recommended to only use peroxide fusion for all lithium analysis. Furthermore, it was evaluated that the global impact on the resource may be an underestimating of lithium grades by 1.6% (Camus and Dupéré, 2022).

NLI completed 274 pycnometers tests to better assess the density of rock units in the deposit.

Based on field observations, NLI initiated a mineralogical identification program to identify the sources of lithium mineralization. The mineral information is still incomplete and fragmentary, and do not yet cover a sufficient area to be included in the block model. As part of the validation process, results were compared against other mineral identification technologies such as TIMA and XRD. Furthermore, the theoretical grade of the intervals was calculated and correlate well against the database Li grades. Theoretical grades are calculated based on the sum of Li-bearing minerals phases multiplied by its measured lithium content.

In the opinion of the QP, the sample preparation, security and analytical procedures used by NLI are consistent with generally accepted industry best practices and are, therefore, adequate. In the future, a sodium peroxide fusion analysis method should be used instead of the 4-acid digest method to ensure a full analysis of refractory minerals. The QP suggests continuing its internal QA/QC protocol for blanks, duplicates (core and pulp) and certified reference material.

1.9 Data Verification

The data validation process was undertaken by SGS with various tasks, such as database validation, core inspection, drill hole collar location, outcrop inspection and geological model ground truthing.

The database validation consisted of inspecting the drill hole collars, deviation surveys, hole length, assays, and lithology. The assay database was compared with the original laboratory certificates. No errors or issues were found during this validation.

Mr. Marc-Antoine Laporte, P.Geo., M.Sc. independent QP of SGS Geological Services completed a site visit at the Whabouchi Project on July 11, 2022. During his visit, the QP visited the mine infrastructures, core logging facilities, offices, rejects and pulps storage, outcrops (including the bulk sample area and channel sampling) and the stockpiles that served for the pilot process plant. The site visit permitted to validate the lithium mineralization of the pegmatite dykes, the location of drill hole collars and the precision of some geological contacts. It also permitted to gain an appreciation of the type of spodumene mineralization.

The database validation process, drill core inspection, outcrop inspection, diamond drill hole and channel sample verification, and geological model ground truthing confirmed the validity of the drilling database and supporting information using the mineral resource estimate. No major issues were found during data validation, with neither the digital nor the field data.

1.10 Mineral Processing and Metallurgical Testing

Mineral processing testing was performed on spodumene concentrate production and lithium hydroxide monohydrate (LHM) production separately. A summary of spodumene concentrate production testwork is presented in this section. The LHM production testwork, related to offsite processing of spodumene concentrate is beyond the scope of this report.
Between 2010 and 2017, multiple testwork programs were performed to develop the Whabouchi concentrator flowsheet. This involved crushing, ore sorting, hydro classification, dense media separation (DMS) and flotation methods. The testwork was performed in order to support a flowsheet with target of producing a 6.25% Li_2O spodumene concentrate product to feed to the flash calciner in the conversion plant. The test program includes screening, settling, filtration, freezing, drying, and magnetic separation tests. These tests were performed by various laboratories and suppliers. A summary of these tests is provided in Section 1.10.1.

In 2021, NLI performed a detailed evaluation to mitigate the process risks associated with flash calcining leading to the decision to revert to a rotary kiln for spodumene calcination. This decision allowed for a reduced concentrate specification entering the conversion plant to 5.5% Li_2O. Following a review of the data by NLI and modification to the product specification, some additional testwork was recommended and carried out. A summary of the additional work is described in Section 1.10.2.

1.10.1 Summary of 2010-2017 Testwork

Ore Sorting was tested at full scale by two (2) suppliers to evaluate the ore amenability to coarse size sorting. The ore can be effectively separated into rejects and accepted with minimal lithium losses. This was implemented in the flowsheet to reduce contamination with amphibolite.

Hydraulic separation has been tested to remove muscovite before the two (2) main separation processes (DMS and flotation). It has been used in pilot plant campaigns. It was also tested in a manufacturer laboratory. A single test has been performed at 8 mm top size to support the coarse muscovite removal.

Multiple DMS testing programs, at bench scale with Heavy Liquid Separation tests and in pilot plant tests, have been done since the beginning of the flowsheet development. DMS performs well with particles of less than 9.5 mm and improves as the top size is reduced to 6.3 mm. DMS can produce a final concentrate, a final reject and a middlings stream which will be reprocessed in flotation.

Multiple test programs involved flotation. Both bench scale and pilot plant work were performed since 2010. Testing between 2014 and 2019 aimed at taking advantage of the coarse liberation of the material and coarse flotation with coarse particle flotation, or hydroflotation. In addition, column flotation shows a better selectivity against muscovite and other contaminant in the fine flotation concentrate by using wash water addition. Final design tests were performed at Eriez which supplies the hydroflotation technology. The grade and recovery of these tests were good. The reagent consumption was reduced through optimization.

Thickening, filtration, and freezing tests have been done to size various equipment.

The Whabouchi DMS pilot plant operated in 2016, and while not piloting the current flowsheet, the pilot plant did demonstrate that the amphibole minerals (dilution material) behaved similarly to spodumene and proved the necessity to include ore sorting and dry magnetic separation into the commercial flowsheet. The Whabouchi DMS pilot plant was an important de-risking activity for the Project.

DMS concentrate drying tests have been done to evaluate conditions required to have a good concentrate for dry magnetic separation of the coarse DMS concentrate. This last operation in the upgrading of the ore was tested at two (2) supplier's facilities with good results.

1.10.2 Summary of Recent Metallurgical Testing (2019-2022)

Variability work was performed at SGS using the Whabouchi flowsheet on five (5) samples which were representative of the first five years of the mine plan. The goal of the program was to produce five representative 5.5% Li_2O concentrates for downstream conversion testing and to evaluate the distribution and concentration behaviour of gangue minerals throughout the beneficiation process. These tests achieved the objective by reaching concentrate grades ranging from 5.29% to 5.64% Li_2O.
Further variability work is planned in 2023. Previous large-scale work carried out prior to 2019 has been based on bulk samples taken from outcrops, with limited work performed on variability samples taken spatially throughout the deposit.

Additional coarse particle flotation testing was carried out at Eriez with the HydroFloat. The purpose of this campaign was to investigate whether improved lithium grade and recovery performance could be achieved by treating a single composite as two (2) distinct split feeds ('coarse' and 'ultra coarse', screened at 500 μm) instead of a single feed. Based on these tests, the results demonstrated that higher grades could be achieved with the split-feed method.

Coagulant testing was performed at two (2) vendor labs in order to investigate the impact of adding coagulant on the quality of water expected at the thickeners overflow. Based on the results, the addition of coagulant in the thickening process of the concentrator was recommended by both laboratories to improve the quality of recirculated process water.

Large-scale saponification testing was conducted at SGS to confirm the performance of spodumene collector dilution in cold water. Visual observations and flotation test results confirmed the effectiveness of high shear mixing in properly saponifying collector even in cold water.

The main risks identified by the QP on testing include the lack of a full pilot plant on the current flowsheet, variability testing, and the presence of petalite in the deposit, which is not recoverable with the current flowsheet.

1.11 Mineral Resource Estimate

SGS was tasked by NLI to complete a Mineral Resource Estimate (MRE) on the Whabouchi deposit. The resource database was supplied by NLI in an Access^{TM} format (.accdb) on February 17, 2022, with a closing date of January 21, 2022. The database used in the MRE comprises drill holes and channel sampling from 2009 to 2018.

The Mineral Resources reported herein have been interpolated into a sub-block model using the modelled spodumene bearing pegmatites. The resource estimate methodology is summarized by the following procedures:

- Drillhole database validations and selection of the drillholes and channels for the Mineral Resource estimation database;
- 3D modelling of spodumene-bearing pegmatite wireframes, based on lithology and lithium content (%Li_2O);
- Geostatistical analysis for data conditioning: density assignment, capping, compositing and variography;
- Block modelling and grade estimation;
- Resource classification and grade interpolation validations;
- Grade and tonnage sensitivities to spodumene concentrate selling prices;
- Gains and losses analysis with previous resource estimate (SGS, 2019).

No new drilling or channel sampling has been completed since 2018. The drilling database used for the mineral resource estimate comprises 258 diamond drillholes and 108 channels. Assay values were also replaced to 0.00% Li_2O for all samples contained within waste material, such as basalt, amphibolite, or diorite. It has been demonstrated that lithium in these waste units is generally hosted within minerals other than spodumene such as holmquistite and are assumed not to be recoverable.

Drillhole and channel data was imported in a Leapfrog Geo project by NLI personnel and validated by SGS. The geological model was completed jointly by NLI and SGS personnel. The model involved wireframing the following units: spodumene-bearing pegmatite dykes, barren pegmatite dykes and internal amphibolite (combined with diorite and/or basal) units within pegmatite dykes. Leapfrog Geo software was used to model these units. Based on core drilling data and outcrop channel sampling, a three-dimensional model was created for the pegmatite dykes. The geological model honours the lithological logging data. The dykes were modelled from logged pegmatite intervals with a lower cut-off of 0.30% Li_2O and a minimum thickness of 2.0 m as implicitly derived vein contact surfaces in Leapfrog Geo software (version 2021.2.4).

Specific gravity was assigned based on the lithology and the percentage of waste inside the pegmatite. Values of 2.76 g/cm^3 for spodumene-bearing pegmatite, 2.67 g/cm^3 for barren pegmatite and 3.04 g/cm^3 for waste material were used in the block model.

The MRE was estimated from a block model where grades were interpolated using Ordinary Kriging (OK). Assay data was composited to 2 m length prior to interpolation in a sub-blocked block model with parent block sizes of 6 m x 4 m x 6 m and sub-block size of 3 m x 1 m x 3 m. Interpolation parameters were chosen based on Kriging Neighbourhood Analysis and by common agreement between SGS, BBA and NLI. Background blocks, or all blocks not labelled as spodumene pegmatite, were interpolated using the Inverse Distance Square method. Background blocks include the interval waste, the dilution skin around mineralized dykes and barren pegmatite dykes.

Various steps of grade estimation were undertaken, including the following: visual checks comparing composite grades against block grades, global statistical checks, local statistical validation (swath plots) and a peer review by BBA.

Block model grades estimated for the Whabouchi project were classified according to the CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and adhere to the CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (2019). As defined by the CIM, all classified material must be within a potentially mineralized wireframe and within the "reasonable prospects of eventual economic extraction" shapes. The mineral resources at the project were classified as Measured, Indicated, and Inferred mineral resources. The resource classifications provided herein align with the SEC's standards resource classification. Inferred mineral resources were not considered in the economic analysis.

SGS considered variogram ranges, drill hole spacing, confidence in the geological interpretation and presence of channel samples to determine parameters that will define the resource categories. The final mineral resource classification is mostly based on average drill hole spacing, the number of samples used in the interpolation, specific geological units, and manual editing to avoid isolated blocks. Final categories of al domains were manually edited to remove isolated clusters of blocks that did not show Reasonable Prospect for Eventual Economic Extraction (RPEEE).

To report a Mineral Resource that responds to a RPEEE, open pit optimisations were generated. To report an underground mineral resource assuming a RPEE, SGS reviewed the material below the pit optimization used to report the open pit mineral resources, with the following constraints: continuity of grades, continuity thickness, distance between mineralized zones, and overall size of mineralized zones.

Mineral Resources are reported exclusive of reserves, on a total basis for the property and on an attributable basis consistent with Livent's ownership interest in the property.

To calculate the resource exclusive of reserves, the proven reserves were deducted from the measured resources and probable reserves were deducted from indicated resources. The total and attributable Mineral Resource, exclusive of reserves, for the Whabouchi Project, combining resources potentially amenable to open pit and underground mining are listed in Table 1-3.

Table 1-3 Mineral Resources, Exclusive of Reserves

Category	Total Tonnes (Mt)	Grade (%Li_2O)	Attibutable Tonnes (Mt)	Total Lithium Oxide (Mt Li_2O)	Attributable Lithium Oxide (Mt Li_2O)
Measured					
Indicated	7.8	1.61	3.9	0.126	0.063
M&I	**7.8**	**1.61**	**3.9**	**0.126**	**0.063**
Inferred	8.3	1.31	4.1	0.108	0.054

Notes:

1. Livent's attributable portion of the property's total mineral resources is 50%.
2. The Mineral Resource described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow the Best Practices outline by the CIM (2019).
3. Mineral Resources point of reference is in-situ and undiluted.
4. The QP for this Mineral Resource Estimate is SGS Geological Services mining QP.
5. The effective date of the Mineral Resource Estimate is December 31, 2022.

6. Density is applied by rock type and the proportion of waste inside each block. Density of 2.77 was used for mineralized pegmatites.

For the Open Pit Mineral Resources:

7. The cut-off grade used to report the Open Pit Mineral Resources is 0.30% Li_2O.
8. Pit optimization parameters are described as follows:
   i. Spodumene concentrate of 5.5% Li_2O price: C$1,264 /t.
   ii. Metallurgical recoveries of 85%
   iii. Ore based costs of C$57.97 /t.
   iv. Northern wall angle of 55°
   v. Southern wall angle of 52°

For the Underground Mineral Resources:

7. The cut-off grade used to report Underground Mineral Resources is 0.60% Li_2O.
8. The classification has been adjusted to remove blocks that do not satisfy RPEEE requirements.
9. No crown-pillar was assumed below the pit optimization.

These Mineral Resources are not Mineral Reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred Mineral Resources in this report are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Mr. Marc-Antoine Laporte, P.Geo., is not aware of any factors or issues that materially affect the Mineral Resource Estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors and additional risk factors regarding Indicated and Inferred Resources.

1.12 Mineral Reserve Estimate

The Whabouchi deposit will be mined using conventional open pit mining for the first 24 years of operation, followed by 10 years of underground mining. The Project life of mine (LOM) plan and subsequent Mineral Reserves are based on a spodumene concentrate selling price of $1,264CAD/t. The effective date of the Mineral Reserve estimate is December 31, 2022.

Development of the LOM plan included pit optimization, pit design, mine scheduling and the application of modifying factors to the Measured and Indicated Mineral Resources. These tasks allowed for the conversion of a certain portion of the Measured Mineral Resources into Proven Mineral Reserves and a certain portion of the Indicated Mineral Resources into Probable Mineral Reserves. The reference point for the Mineral Reserves is the feed to the primary crusher. The tonnages and grades reported are inclusive of mining dilution, geological losses, and operational mining losses. Mineral Reserves are reported on a total basis for the property and on an attributable basis consistent with Livent's ownership interest in the property.

Table 1-4 presents the total and attributable Mineral Reserves as of December 31, 2022 for the Whabouchi deposit.

Table 1-4 Mineral Reserves

Category	Total Tonnes (Mt)	Attributable Tonnes (Mt)	Li_2O Grade (%)
Proven	10.5	5.2	1.40
Probable	27.7	13.8	1.28
Proven & Probable	**38.2**	**19.1**	**1.31**

Notes:

1. Totals may not add due to rounding.
2. Mineral Reserves are based on a Spodumene concentrate selling price of $1,264/t CAD at 5.50% Li_2O.
3. A metallurgical recovery of 85% was used.
4. The Mineral Reserves are inclusive of mining dilution and ore loss.
5. The reference point for the Mineral Reserves is the primary crusher.
6. The economic viability of the Mineral Reserve has been demonstrated.

7. The reserves estimate has an effective date of December 31, 2022.

For the Open Pit Mineral Reserves:

8. The Open Pit Mineral Reserves were prepared by BBA's mining QP.
9. A cut-off grade of 0.40% Li_2O was used.
10. Estimated variable mining costs (CAD) of $2.25/MT for overburden and $3.46/MT for rock, variable processing and tailings management costs of $11/MT milled, transportation costs of $159 MT of concentrate, and $46.7M/yr of fixed costs.
11. The stripping ratio for the open pit is 2.8 to 1.

For the Underground Mineral Reserves:

12. The underground Mineral Reserves were prepared by DRA's mining QP
13. A variable cut-off grade was used (0.5-0.72%), depending on mining method.
14. Li_2O content (tonnes) are estimated as reported (include dilution and mining recovery) values.
15. The Mineral Reserve is estimated with a stope mining recovery of 90%.
16. The Mineral Reserve includes both internal and external dilution.
17. External dilution included a mining dilution of 0.5 m on the hanging and footwalls for the long-hole mining method.
18. A minimum true mining width of 4 m was used.

1.13 Mining Methods

The Whabouchi deposit characteristics make open pit mining more favorable from a technical and economic standpoint because of its proximity to surface. Open pit mining will, therefore, be favored for the upper portions of the deposit. However, open pit mining is commonly associated with more significant environmental and social impacts than underground mining, essentially because of the associated larger surface footprint. To mitigate environmental and social effects of the projected mine, where geological characteristics and economic factors made it feasible to switch to underground mining, the latter was favored.

Consequently, from Year 25, the mine will be operating from underground, thus not only limiting the surface footprint of the ultimate open pit, but also minimizing the amount of waste rock to be managed and stockpiled at surface. Such an approach also enables a longer mine life without significantly increasing the surface area impacted by mining activities, which extends the duration and cumulative importance of the Project's economic spin offs for local, regional, and provincial stakeholders.

1.13.1 Open Pit Rock Slope Design

The feasibility level open pit rock slope engineering and design for the Whabouchi open pit were completed by WSP Canada Inc., (WSP). Assumptions have been provided for the overburden slopes.

The rock slope geotechnical investigation completed in 2021–2022 included logging of 3 geotechnical boreholes totalling 653 m, performing 34 laboratory strength tests and 8 hydraulic conductivity tests in 3 boreholes within the Whabouchi open pit area. This built on previous work completed by Journeaux (2012, 2019).

Major structures in the current geological interpretation include faults and dykes. Faults do not appear to have a strong control on slope designs due to their favourable orientations. The bench mapping of faults will be important to verify this. Dykes are oriented parallel to foliation and may act as conduits for groundwater. The dykes were considered in the stability models for potential deep-seated toppling.

Industry experience in hardrock open pits located in northern Quebec suggest that footwall bench faces typically break back to the mean foliation dip. For the Whabouchi north wall design, the mean foliation dip from oriented core was used to define the footwall bench face angle. Careful perimeter blasting and scaling will be required to successfully apply this design and reduce backbreak. The design bench face angles are defined by the kinematic assessment of discontinuity populations from oriented core.

Kinematic stability analyses results indicate potential for significant structurally controlled combined toppling and planar failures involving continuous foliation planes and a moderately in-pit dipping set on the south wall of the Whabouchi open pit. An effective bench face angle of 75° after all losses is assumed to manage bench-scale toppling. This will require an inclined pre-shear.

To manage potential deep-seated toppling the footwall overall slope angle is constrained to 52° for the proposed ultimate open pit depth of 200 m, including ramps and geotechnical benches. The south wall bench geometries will require a high level of skill for perimeter blasting and scaling. Bench and multi-bench stability in this sector will be sensitive to the presence of groundwater pressures along structures where slope stability is marginal, and horizontal drains will be required where slopes do not adequately depressurize on their own. Vibrating wire piezometers will be required to verify slope water pressure conditions as the open pit deepens to assess the degree to which horizontal drains are needed.

In the east and west end walls that are oblique to potential toppling and planar failures along the foliation, steeper bench face angles should generally be achievable with careful blasting, excavation, and scaling. While wedge or toppling failures may occur locally, available structural data does not indicate these failure mechanisms to be a control on the end wall bench design.

1.13.2 Open Pit

The Whabouchi pit will be mined using conventional open pit mining methods consisting of drilling, blasting, loading, and hauling. Vegetation, topsoil, and overburden will be stripped and stockpiled for future reclamation use. The ore and waste rock will be drilled and blasted with 12 m high benches and loaded into haul trucks using mining backhoes which will mine 6 m high flitches. Overburden will be hauled to an overburden stockpile and waste rock will be hauled to the Co-disposal Storage Facility (CSF). Ore will be dumped on the Run-of-Mine (ROM) pad in several stockpiles which will be rehandled and trammed to the primary crusher by a front-end wheel loader. The purpose of this rehandling is to provide a properly blended ore feed to the mill. The mine will operate on two 12-hour shifts, seven days per week, 50 weeks per year.

Phases, also referred to as pushbacks, have been designed to access ore quicker and to defer waste stripping. The phase design process was guided by the smaller revenue factor pit shells from the open pit optimization analysis as a guide. A minimum working width of 40 m between phases was considered acceptable based on the size of the mining equipment and the proposed scale of mining operations. The phase designs also attempt to avoid mining the petalite zones in the initial years of the mining operation. A total of four (4) phases were designed for the LOM.

The mine production plan has been prepared using the MinePlan Schedule Optimizer (MPSO) tool in the Hexagon MinePlan 3D software. Provided with economic input parameters and operational constraints such as phase sequencing, maximum bench sink rates, and mining and milling capacities, the software determines the optimal mining sequence which maximizes the Net Present Value (NPV) of the mine production plan.

The mine plan has been prepared quarterly for the first three years of production, annually for the following seven (7) years, and in three (3) year increments thereafter. The mine plan also includes a three (3) month period of pre-production. The purpose of the pre-production period is for the mine to provide waste rock for construction material and to prepare the pit for mining operations. During pre-production, a total of 529 kt of material is planned to be mined, including 97 kt of overburden, 300 kt of waste rock, and 130 kt of ore.

During the mining operation, the total material mined from the open pit peaks at 5.2 Mt in Year 5 and averages 4.5 Mt/y from Years 2 to 20. The average Li_2O grade ranges from 1.26% to 1.50% over the life of the open pit mine. The mine plan is expected to produce an average of 226,400 tonnes of concentrate per year, with a high of 238,500 tonnes in Years 20 to 23.

The mining equipment fleet will be owner operated. In full production, the open pit mining operation will have two (2) mining backhoes equipped with 6.7 m^3 buckets, five (5) 64-tonne haul trucks, two (2) wheel loaders as well as a fleet of support and service equipment.

Drilling and blasting will be done using bulk emulsion which will be transported to site by an explosives supplier using 20,000 kg tankers. The explosives supplier will provide down the hole service.

BBA also calculated the fleet of equipment required to load, transport, and place the tailings and ore sorter rejects at the CSF. This fleet includes a front end wheel loader, two (2) 60-tonne haul trucks, as well as a track dozer and a hydraulic excavator.

The total workforce for the open pit operation as well as for the transport and placement of material at the CSF including supervision, mine equipment maintenance, and mine technical services is expected to reach a peak of 148 employees.

1.13.3 Underground

Transverse long-hole mining method was selected as the main underground mining method due to its production nature and the geometry of the deposit. Longitudinal long-hole and AVOCA mining methods were also selected for some remote areas where the lodes are thinner and smaller. The underground ore production rate will reach 3,361 t/d at the concentrator feed and will generate 11.7 million tonnes of ore at an average diluted grade of 1.29% Li_2O over a period of ten (10) years. The backfill method selected is cemented rock fill (CRF) for transverse stopes and rockfill (RF) for longitudinal and AVOCA stopes.

To access the deposit, a single decline will be driven from a single portal, located at surface near the exit of the pit at elevation 275 m. The main level and infrastructure will be excavated south of the deposit. The main underground mine plan consists of 14 levels set at 30 m intervals. The pit bottom will also be used to recover the crown pillar from the underground. The crown pillar will be mined at the end of the underground LOM. No work has been performed for the geotechnical and hydrogeology aspects for the underground mine or to assess the potential changes to the open pit slope design to accommodate underground mining.

The mine will be ventilated using a push-pull system with two (2) systems of air intake and two (2) exhaust systems. The intake will be located near the level access to be centralized and minimized the smoking time of the production blast. Exhaust will be located at each end of the level.

1.14 **Recovery Methods**

Section 10 of this Report described the metallurgical testwork and how the results were used to derive the Process Flow Diagrams (PFD) and mass balance. The process design has been split in two (2) locations. The Concentrator will be located 675 m northeast of the Whabouchi mine open pit while the Conversion Plant will be located in Bécancour.

The Whabouchi concentrator process is composed of crushing, particle ore sorting, dense media separation, grinding, classification, froth flotation, and dewatering. The Whabouchi concentrator was originally designed in 2014 to produce 216,485 dry tonnes per year of 6.0% Li_2O spodumene concentrate. The concentrator building was erected in 2016 based on the 2014 flowsheet. The flowsheet was modified in 2018 in order to increase spodumene concentrate grade to 6.25% Li_2O at 215,000 dry tonnes per year of production, which included the switch from mechanical flotation cells to a combination of coarse particle flotation and column flotation. The increase in concentrate grade was implemented to mitigate the process risks associated with flash calcining of the spodumene concentrate at the conversion plant. In 2019, this design was refined based on the expected mine plan and, due to a lower feed grade, the production target was reduced to 205,000 dry tonnes per year of 6.25% Li_2O spodumene concentrate, and this design was maintained for equipment procurement and the start construction of the plant prior to halting of the project at the end of 2019. In 2021, NLI performed a detailed evaluation to mitigate the process risks associated with flash calcining leading to the decision to revert to a rotary kiln for spodumene calcination. This decision allowed for a reduced concentrate specification with increased lithium recovery. Based on additional metallurgical testing and a revised metallurgical balance, the plant design was updated for a target spodumene concentrate of 5.5% Li_2O.

Due to the complexity of the flowsheet and the constraints associated with the existing procured equipment, constructed building and half constructed plant, and current overall design, it is the opinion of the QP that the selected availability of 91.5% is optimistic and a significant effort will be required to achieve it. A review of the process data and modifications is in progress and will continue as the Project moves forward to flag and mitigate identified risks and establish contingency plans for dealing with process upsets. A list of high-level recommendations and modifications have been prepared and are listed in Section 23. An additional detailed list of risks, proposed modifications, and verifications to complete has been prepared by DRA and will be actioned in this phase of the project with the goal of achieving targeted plant performance. However, there is still a significant risk that ramping up to the stated production values will take longer than indicated in this report and will require additional sustaining capital projects. In the QP's opinion, the upper limit of plant availability may be lower than 91.5% and may be as low as 75 to 85%, or recovery may decrease to maintain concentrate production. It should be recognized that reductions in plant capacity will lead to reduction in tonnes of concentrate produced by the process plant.

The Whabouchi concentrator is located at 675 m northeast of the open pit mine. The concentrator design has been updated to produce an average of 220,846 tonnes per year for Years 1-4, 227,021 tonnes per year for Years 5-24 and finally 238,841 tonnes per for Years 25-34 of spodumene concentrate per year at 5.5% Li_2O. The overall Life-of-Mine concentrate production averages 229,797 tonnes per year. The Run-of-Mine (ROM) mineralized material will be fed into the primary jaw crusher by wheel loader and then screened to feed a coarse and fine ore sorter, as well as a fines by-pass. The sorted material will combine with the fines and get crushed with the secondary and tertiary cone crushers. The final crushed product is sent to the concentrator feed hopper or fine ore stockpile.

The crushed material will be screened on the fine ore screen and the oversize will be upgraded in a dense media circuit after a stage of mica hydroseparation removal to produce a coarse spodumene concentrate, a tailings product, and a middlings product. The DMS coarse concentrate will then be dried in a rotary dryer before treatment by a dry magnetic separation system. The magnetic product will be discarded with the tailings and the non-magnetic product will be the first portion of the final spodumene concentrate.

The DMS middlings product will be ground to less than 0.85 mm with a ball mill in closed circuit with a screen and combined with the fines fraction that bypassed DMS. This product feeds a fine stage of mica hydroseparation removal and then goes to flotation circuit. The flotation circuit consists of de-sliming, wet magnetic separation, attrition scrubbing, and three (3) stages of spodumene flotation. Coarse particle flotation, or hydro-flotation, is performed at two (2) coarse sizes (850 µm to 500 µm, and 500 µm to 212 µm) in two (2) hydro-flotation cells and at fine size (212 µm to 20 µm) by flotation columns circuit.

Tailings from DMS concentration, dry magnetic separation, mica hydro-separation, desliming, wet magnetic separation and flotation will be dewatered by a combination of screen dewatering, as well as thickening and filtration (fines only) before storage into a tailings bin. The dewatered tailings will be transported by haul truck to the CSF.

The spodumene flotation concentrate will be thickened and filtered by a belt vacuum filter to less than eight percent (8%) moisture. The flotation concentrate will be combined with the dry DMS concentrate for transport in containers by road trucks to Matagami. The shipped concentrate will have moisture of less than five percent (5%). In Matagami, the concentrate containers will be transloaded onto railcars for transport to a conversion plant for further processing. Figure 1-1 illustrates the crushing area with ore sorting. Figure 1-2 depicts the concentrator simplified flow sheet.

Figure 1-1 Simplified Flow Sheet of Crushing and Ore Sorting

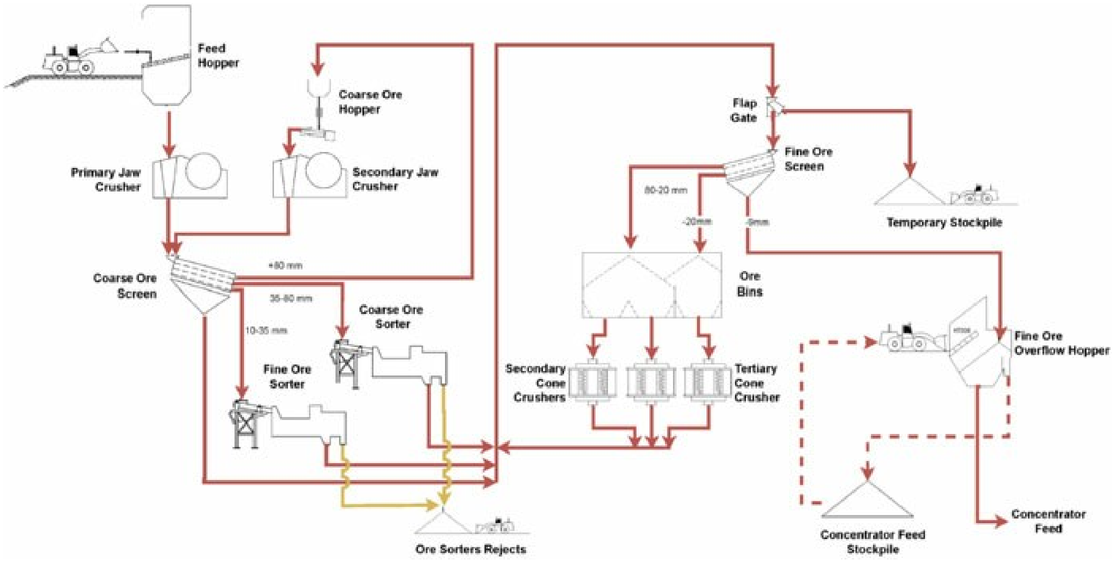
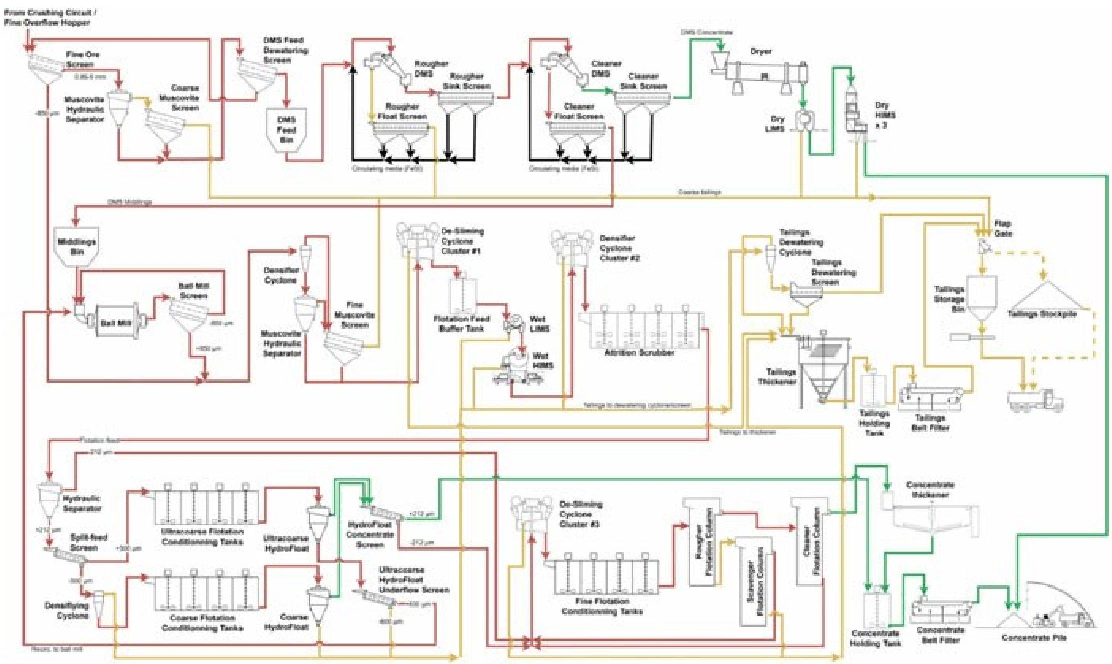


Figure 1-2 Simplified Flow Sheet of Concentrator



1.15 Project Infrastructure

1.15.1 Whabouchi Project Infrastructure

Two (2) different scale plans were produced for the Project. Drawing No. W000-EN-DWL-DRA1-0001 (Figure 1-3) illustrates the overall general site plan and Drawing No. W000-EN-DWL-DRA1-0002 (Figure 1-4) shows the Whabouchi Village area and related infrastructure.

1.15.1.1 W000 – Whabouchi Mine Site

The Whabouchi mine site is located at km 276 on the Route du Nord. The infrastructure required to service a project in a remote location and fulfil the needs of the workers is significant and comprises the following main facilities with their current status (in parenthesis):

- W131 Core Shack and Core Storage
- W151 Mine Garage and W152 Wash Bay (partially completed with some work required to complete);
- W153 Fuel Tank Farm (currently being designed);
- W428 Effluent Treatment Plant (new facility to be designed);
- W442 Sewage Treatment (temporary facilities in place and functional);
- W443 Fire Protection (Fire water network installed around concentrator, pump station installed and commissioned some upgrades required for the next phase);
- W444 Sewage Treatment (new facility to be designed);
- W488 Electrical sub-station, power supply and distribution (completed and commissioned with modifications required for the next phase);
- W492 Laboratories (metallurgical laboratory partially completed);
- W493 Project access and Gate House (to be demolished and replaced);
- W496 Temporary Camp Facilities (completed and operational for approximate 40-person peak capacity, to be expanded for the construction phase).

1.15.1.2 W450 - Whabouchi Village

The existing Administration Building will be demolished and the new Whabouchi Village will be built comprising the following facilities to accommodate 250 people:

- W441 Potable Water Treatment Plant;
- W451 Heated Corridors;
- W452 Kitchen and Dining;
- W453 Dormitories;
- W454 Welcome and Medical Centre;
- W455 Laundry and Services;
- W456 Recreation Centre;

- W457 Arctic Corridors;
- W459 Mine Dry;
- W491 Administration Office (to be demolished and replaced);
- W495 Emergency Services Building.

Figure 1-3 General Site Plan

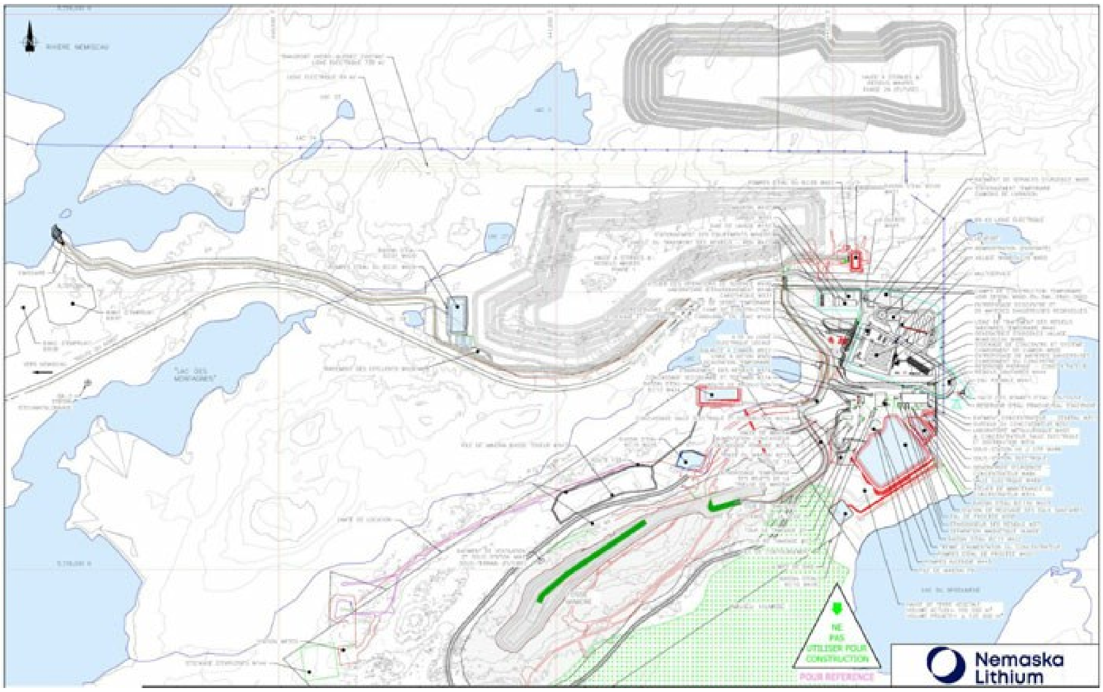
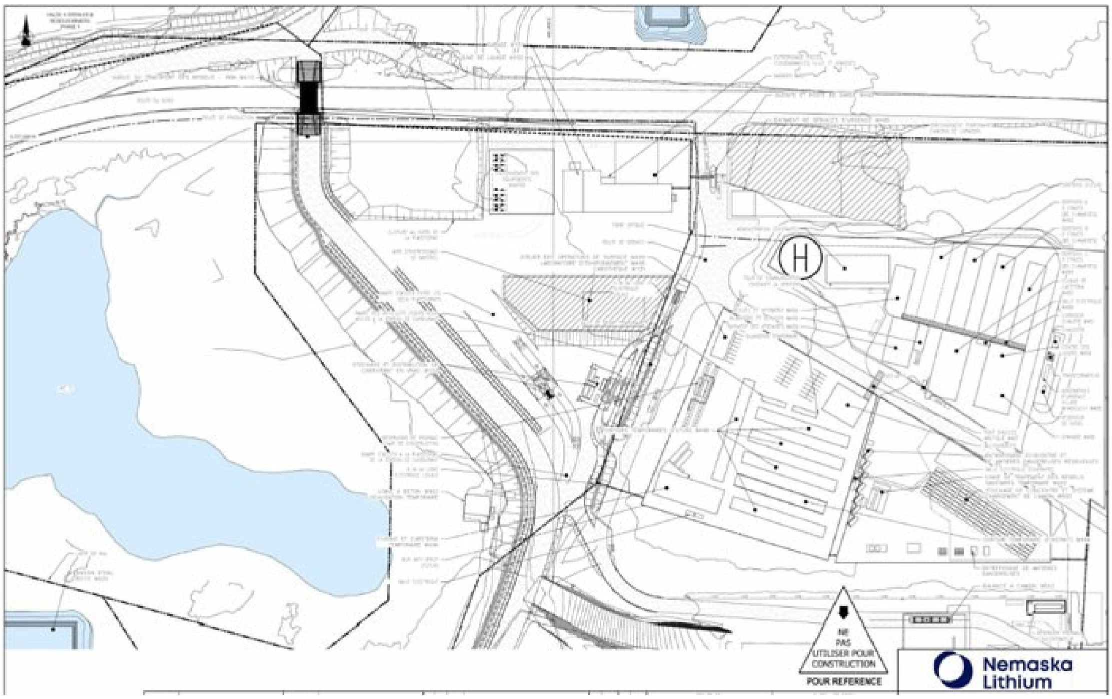


Figure 1-4 General Site Plan – Whabouchi Village Area View



1.15.2 Water and Mine Waste Management at the Whabouchi Mine

Water management strategy and infrastructure design for Whabouchi mine has been completed in accordance with recommendations from Quebec's Directive 019 (MELCCFP, 2012) and Canada's Environmental Code of Practice for metal mines (ECCC, 2009), and in compliance with Quebec's Environmental Discharge Objectives determined for the Whabouchi Mine by the MELCCFP. All seepage and runoff water generated on areas impacted by mining activities is considered "contact water." Contact water and water from pit dewatering activities will be collected and retained for the settlement of sediment and treatment prior to being released to the environment.

The development of the water management infrastructure (i.e., ponds, ditches, and pumping requirements) is sized based on the required volume of surface runoff to manage, which varies according to the catchment area of the co-disposal storage facility (CSF).

For the mine waste management, two provincial guidelines, Directive 019 (MELCCFP, 2012) and the MERN Closure Guidelines (2017), support the CSF design. The design criteria adopted is based on co-disposal storage of mine waste rock; this strategy uses waste rock to construct the perimeter berm and roads around the CSF, and store filtered tailings in the middle.

The planned CSF is located to the north of Route du Nord. All phases of the CSF were planned to meet the anticipated duration of the Project. The co-disposal piles have a total capacity of approximately 52.8 million cubic metres (Mm^3). All waste rock, filtered tailings, and ore sorter rejects would be contained in the co-disposition piles. The CSF phase 1 has an approximate storage capacity of 13.1 Mm^3.

1.16 **Market Studies and Contracts**

In a structural LCE supply (refined and mined) deficit scenario, it is generally difficult to split LCE supply between lithium carbonate and lithium hydroxide. Benchmark Mineral Intelligence estimate a mined supply deficit of just under 0.1 million t LCE in 2022. They expect the mined supply to fluctuate between deficit and surplus from 2023 until 2028 and then a return to widening mined supply deficits starting in 2029 into the next decade.

With the ongoing overall market tightness, battery-grade products are likely to experience a more pronounced supply deficit than technical-grade products. The lithium hydroxide market is expected to experience tighter conditions than carbonate as supplies are constrained by producers' struggle to expand production in line with demand while maintaining increasingly strict quality standards. In addition, lithium hydroxide shelf-life challenges (e.g., clumping over extended periods of storage) will hamper prolonged stockpile growth in times of subdued demand. As such, producers will need to proactively schedule production in line with contractual needs and market conditions as part of efforts to mitigate production volume versus product sale risk. This will naturally tighten market supply and provide additional upward pressure on pricing.

Lithium raw materials is unlike traditional commodities that can be physically traded on transparent metal exchanges, such as the London Metal Exchange (LME). Lithium companies increasingly refer to market pricing from price reporting agencies such as Fastmarkets and Wood Mackenzie, as a single reference index for pricing does not yet exist. While each market research firm's pricing forecast shows a degree of variation (due to differing respective supply-demand models), following recent extreme price increases, due to rapidly improving demand for lithium chemicals, analysts expect prices to show only moderate improvement in the short-term before prices stabilise or show a slight decline as supply from new conversion plants increases over the medium-to-long-term.

1.16.1 Contracts

As of the date of this Technical Report, NLI has entered into agreements with Livent or its subsidiaries, as described below.

In May 2022, NLI and Livent USA Corp., a subsidiary of Livent, entered into agreement in connection with NLI's development of facilities and operations for the conversion of lithium sulfate to lithium hydroxide. The agreement is effective for five years unless earlier terminated. NLI and Livent are also parties to a marketing services agreement under which Livent provides exclusive marketing and sales services to NLI, and a project management services agreement.

In June 2023, NLI, Investissement Québec and Québec Lithium Partners (UK) Limited, a subsidiary of Livent, entered into an amended and restated unanimous shareholder agreement, which establishes terms for, among other things, the governance, funding and operations of NLI.

In May 2023, Ford and NLI entered into a long-term supply agreement (over 11 years), beginning in 2025. The agreement initially calls for spodumene concentrate and later transitions to delivery of battery-grade lithium hydroxide. Spodumene concentrate produced exclusively from the Whabouchi mine will provide the feedstock for lithium hydroxide at volumes up to 13,000 metric tons per year, which is equivalent to approximately 100,000 dry metric tons per year of spodumene concentrate.

Additional supply agreements will be negotiated on a case-by-case basis and with prices linked to an internationally recognized and industry-accepted price index.

1.17 **Environmental Studies, Permitting and Social or Community Impact**

The environmental and social impact assessment (ESIA) was submitted to both federal (Impact Assessment Agency of Canada (IAAC)) and Québec (Review Committee of the James Bay and Northern Québec Agreement (COMEX)) authorities in April 2013.

The COMEX held public hearings in March to April 2015 in Eeyou Ischtee James Bay territory. Other forms of consultation were organised by Nemaska Lithium and/or the Cree Nation of Nemaska. On September 4, 2015, following a positive recommendation by the COMEX, the Provincial Administrator of the James Bay and Northern Québec Agreement granted authorisation for the Project and, therefore, confirmed that NLI received the general CA for the Whabouchi Project from the MELCCFP.

On 29 July 2015, following a comprehensive assessment of the Whabouchi project by IAAC, the federal Minister of Environment and Climate Change issued a positive Decision Statement (DS), declaring that the Project was not likely to cause any significant adverse environmental effects. This DS also set out the conditions to be respected by NLI in terms of mitigation measures and monitoring programs. The IAAC issued its final EA report on that same day.

Even though the Project underwent an environmental impact assessment and was authorized by the Government pursuant to the EQA and CEAA, it is still subject to other sections of the EQA, the Mining Act and to other applicable provincial and federal laws and regulations. Indeed, in addition to the authorizations required under Section 22 of the EQA, the proponent must obtain the permits, authorizations, approvals, certificates and leases required from the appropriate authorities.

The issuance of certificates of authorization is still required under Section 22 of the EQA. The authorizations applications and permitting process is expected to last the full construction phase and has started in Q1 2016. Applications were filed in a timely manner with the construction works and had therefore no impact on Project schedule before the project was halted by the CCAA procedure in 2019.

Most of the required permits were obtained starting in 2016 during the construction period before the project was halted by the CCAA procedure in 2019. This includes, among many others, the general certificate of authorization as well as the authorization for the mine and co-disposal operation.

Several minor changes to the general CA were also requested by NLI and authorized by the COMEX and the MELCCFP. The IAAC continues to monitor the acceptability of changes and operations with reference to the Decision Statement. The last request for change was suspended in 2019 when NLI sought protection under the CCAA.

The Whabouchi Mine project now has some minor differences from the previous one. Some of those changes require modification request to the already granted permits. Meetings with MELCCFP and IAAC helped identify which changes are sufficient to warrant a permit modification request to the initial General CA and other regional authorizations that were obtained to cover the Project as it was described prior to the current Feasibility Study.

NLI is continuing to fulfill the provisions included in the General CA as well as the DS for Whabouchi, since those apply to all project phases from construction to site restoration. Whabouchi regional construction permits have now expired and will need to be renewed once plans to restart construction are confirmed.

As described earlier in this Report, there will be a need to increase the area dedicated to waste rock and tailings management as part of the normal operations over the 34-year mine life. Such modifications (as provided for under section 30 of the new EQA) can be obtained following a new ESIA and COMEX and IAAC blessing.

Prior to the environmental permitting process NLI was already engaged in a continuous relationship with the Cree Nation of Nemaska as well as the Cree Regional government. This culminated in the signing of the Chinuchi Agreement. This Impact Benefit Agreement established the collaboration between the Cree and NLI to jointly develop the community and the project.

During the period of corporate restructuring, NLI continued to respect the commitments of the Chinuchi Agreement. The Band Council of the Cree Nation of Nemaska was met with to explain the restructuring and introduce the new partners.

With the resumption of the Project, meetings with the community have multiplied. The statutory committees of the Chinuchi Agreement have been reformed and have resumed regular meetings. In addition, to facilitate the organization of the committees and the implementation of the proposed actions, a coordination committee made up of representatives of the administrative staff of each stakeholder meets every two weeks.

NLI also maintain privileged relations with the non-Cree communities of Jamésie. The two (2) communities of interest are Matagami and Chibougamau.

1.18 Capital and Operating Costs

The Capex and Opex values were provided by a number of qualified firms, each with their scope of work and then blended by NLI into one Capex and Opex for each Project site. The Owner’s costs were provided by NLI. A working capital equal to two (2) months trade payables and trade receivables has been included as well.

The Capex are reported in Canadian Dollars (CAD). The Capex includes costs pre-production capital cost, working capital and the initial required costs to complete the initial construction.

Table 1-5 presents the Capex for the Whabouchi Project, including related transporation costs to bring mine materials to market.

Table 1-5 Capex – ($ 000's CAD)

Description	Whabouchi
Direct Costs	283,565
Indirect Costs (incl. Owner's Cost)	147,589
Contingencies	42,000
Total Capex	**473,154**

The Capex consists of direct and indirect capital costs as well as contingency which includes escalation. An estimate for the sustaining capital has been prepared for use in the financial model. A provision for contingency of 9.7% was selected to cover the remaining work to be completed.

The Capex associated to the remaining scope of work at Whabouchi is $473.2 M CAD for the initial capital costs, and $198.4 M CAD for sustaining costs over the life of the mine. The closure costs are not included in the Capex nor sustaining capital.

Operating costs were estimated for the Whabouchi Mine operation and concentrate transport and to cover the costs related ore extraction, spodumene concentration, management of tailings, waste, and water, General and Administration (G&A) costs including site services, transport and lodging of workers and operation expenses and concentrate shipping.

The unit operating costs were based on a typical steady state spodumene concentrate production of 221,400 t/y (dry).

The sources of information used to develop the operating costs include in-house databases and outside sources.

The average operating cost estimate for an example year (2028) is summarized in Table 1-6.

Table 1-6 Example Annual Opex for Whabouchi

Unit Cost	Estimate $CAD/t Concentrate
Mining (open pit and underground)	$91
Processing	$394
Tailings and water management	$11
Concentrate transport	$263
General and administration	$210
Unit Cost	**$970**

1.19 Economic Analysis

The economic assessment of the Project is based on price projections in U.S. currency and cost estimates in Canadian currency. An exchange rate of 1.31 CAD per USD was assumed to convert USD market price projections and particular components of the cost estimates into CAD. No provision was made for the effects of inflation. The base-case evaluation was carried out on a 100%-equity basis.

Current Canadian tax regulations were applied to assess the corporate tax liabilities and the regulations adopted in 2013 were applied to assess the Quebec mining tax liabilities. Historical tax losses carried forward were considered. This assessment is based on the fact that the Project is emerging from a care and maintenance phase. Consequently, all funds invested up until April 2023 are considered sunk and are omitted from the capital expenses in the present economic analysis. Only that part of the capital expenditure that remains to be incurred to bring the Project to the production phase is considered.

The total revenue (life of mine) derived from the sale of concentrate from the mine was estimated at $23,154.9 M CAD. The total operating costs were estimated at $7,950.1 M CAD.

The financial results indicate a pre-tax NPV of $3,588.4 M CAD at a discount rate of 8.0%. The pre-tax Internal Rate of Return (IRR) is 50.2% and the payback period is 1.0 year.

The after-tax NPV is $2,080.1 M CAD at a discount rate of 8.0%. The after-tax IRR is 36.9% and the payback period is 1.4 years.

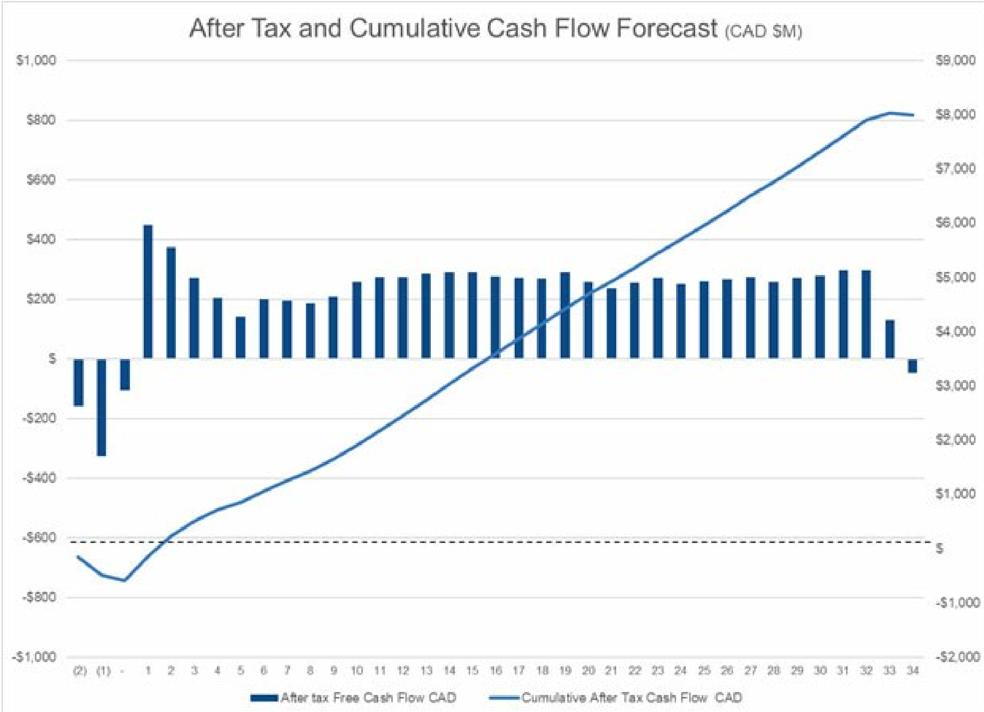
Table 1-7 presents the financial indicators under base case conditions.

Table 1-7 Base Case Scenario Results

Base Case Financial Results	Unit	Value
Pre-Tax (P-T) NPV @ 8%	M CAD	3,588.4
After-Tax (A-T) NPV @ 8%	M CAD	2,080.1
P-T IRR	%	50.2
A-T IRR	%	36.9
P-T Payback Period	Years	1.0
A-T Payback Period	Years	1.4

Figure 1-5 illustrates the after-tax cash flow and cumulative cash flow profiles of the Project for base case conditions. The intersection of the after-tax cumulative cash flow curve with the horizontal dashed line represents the payback period at the start of production.

Figure 1-5 After-Tax Cash Flow and Cumulative Cash Flow Profiles



A sensitivity analysis has been carried out, with the base case after-tax NPV described above as a starting point, to assess the impact of changes in total pre-production Capex, Opex, product prices and the USD/CAD exchange rate (F/X) on the Project's NPV @ 8.0% and IRR. Each variable was examined one-at-a-time (all product prices are varied together). An interval of ±25% with an increment at 10% from the base case was used for the first three (3) variables (assuming a static USD/CAD F/X rate). It is to be noted that the margin of error for cost estimates at the feasibility study level is typically ±15%. Table 1-8 presents the base case sensitivity analysis results.

Table 1-8 Base Case Sensitivity Analysis ($ CAD)

Cost Variable	Sensitivity Factor				
	-25%	-10%	0	10%	25%
Spodumene Price $ /Mt	$1,121 M	$1,699 M	$2,080 M	$2,458 M	$3,023 M
Initial Capital	$2,219 M	$2,136 M	$2,080 M	$2,024 M	$1,941 M
Operating Expense	$2,423 M	$2,217 M	$2,080 M	$1,940 M	$1,727 M

M = million dollars (CAD)
Mt = metric ton

1.20 Other Relevant Data and Information

1.20.1 Project Schedule

The planned production start-up is March 2025.

The schedule for the Whabouchi Area is based on the start of detailed engineering as work on this study concludes, mobilization to site in the second quarter of 2023 and full start of construction by October 2023 with construction completion in February 2025.

1.20.2 Key Milestones

The Project key milestones are outlined in Table 1-9.

Table 1-9 Project Key Milestones

Milestone	Plan
Submit Detailed Feasibility Study	Mar 2023
Approval of Basic Permits & Start of Construction	Mar 2023
Temporary Camp Available for Use	May 2023
Start of Crushing Plant Commissioning	Sep 2024
Start of Concentrator Commissioning	Dec 2024
Declare Commercial Production	Mar 2025

1.21 Interpretation and Conclusion

The Nemaska Integrated Lithium Project consists of the development of the Whabouchi spodumene mine and concentrator located approximately 300 km North of Chibougamau and a lithium hydroxide conversion plant to be built in Bécancour, mid-way between Montreal and Quebec City.

During the 24-year life of the open pit mine, a total of 33.9 Mm^3 of waste rock and 13.0 Mm^3 of tailings will be generated for a total of 46.9 Mm^3. The underground mine will generate an additional 0.7 Mm^3 of waste rock and 5.9 Mm^3 of tailings. In total, the Project will generate 53.4 Mm^3 of waste materials. All the waste rock and filtered tailings will be contained in the designed storage facilities, except 4.0 Mm^3 of waste rock that will be used as backfill material for the underground operation. Approximately 6.0 Mm^3 of waste rock could also potentially be disposed in the open pit mine.

The underground project will take three (3) years and two (2) months to develop the required underground infrastructure to start commercial production.

The site-wide water management strategy and the water balance study will be updated during future engineering design phases and mine operations; accounting, whenever possible, for the construction sequence of the water management infrastructure based on the planned site development.

The results indicate that the site runoff and the groundwater inflows to the pit can supply the plant process water demand during the life of mine, and that no uncontrolled overflow of contact water to the environment is expected under the analyzed climate conditions.

The final effluent will release water to the Nemiscau River with regular monitoring of flow and water quality. If required, a water treatment plant will be implemented to ensure full compliance with all applicable quality criteria. Tests indicate that the ore and waste are non-acid generating, and no elements are leachable. The water management system has been designed to allow for sufficient settling time.

During the preparation of this Report, extensive additional laboratory testwork, vendor testwork and orebody characterization were complete to further reduce technological risks of the project.

A computed cash flow analysis was developed by NLI from the technical aspects and based on SC6 price projections provided from a reputable market study firm. This Technical Report resulted in a Mineral Reserve Estimate that contains 38.2 Mt of Mineral Reserves averaging 1.31% Li_2O.

Consequently, the QPs conclude that the Project is technically feasible as well as economically viable and the authors of this Technical Report consider the Project to be sufficiently robust to warrant pursuing the implementation phase.

1.22 **Opportunities**

The Whabouchi deposit has an opportunity to increase its mineral resources at depth by confirming the extension of known spodumene-bearing dykes. There are also some areas within the current mineral resource pit footprint that could see an increase in mineral resources by connecting interpreted dykes in different directions.

Drill core analytical data at Whabouchi varied during exploration campaigns. The sodium peroxide and the 4-acid fusion were the most used lithium digestion methods. Following a study by SGS (Camus and Dupéré, 2022), it was found that samples analyzed with 4-acid fusion underestimate grades by 4%. The impact of this bias on the mineral resource was evaluated at an underestimating of lithium grades globally by 1.6%.

1.23 **Recommendation**

Based on the Project's modelled economic returns, it is recommended to proceed to the implementation phase of the Project. Based on the comments received to date, there are no issues that would materially affect the ability of NLI to develop and put the Project into production. However, NLI is still in consultation with regulators and stakeholders, and potential future conditions of approval could require refinements to Project components or additional mitigation measures to be implemented.

While the detailed design and procurement activities are on-going, it is recommended to monitor or to complete the specific activities listed below.

1.23.1 Mineral Resource Estimate

Diamond drilling and channel sampling is recommended for the following opportunities or to mitigate the risks:

- Channel sampling and near-surface diamond drilling targeting the indicated mineral resource area in the first five years of mining operations to convert to measured mineral resources,
- Diamond drilling at moderate depth (first 150 m) to convert indicated mineral resources to measured mineral resources. This conversion would require a minimal amount of drilling,
- Diamond drilling in areas of low drilling density to convert inferred mineral resources to indicated mineral resources,
- Targeted diamond drilling to confirm connection between dykes of different orientations,
- Exploration diamond drilling at depth to confirm the continuity of spodumene-bearing dykes.

NLI should pursue all chemical analysis with the peroxide fusion method, while maintaining a robust QA/QC protocol.

It is recommended that NLI continue the analysis on lithium deportment. Once results are satisfactory and well distributed in the deposit, it is recommended to integrate these results into a mineralogical block model. It should also be investigated if other geometallurgical factors can be integrated, such as mineral liberation and grain size.

1.23.2 Open Pit Mining

For the open pit mine, BBA proposes that NLI:

- Evaluate the possibility of incorporating a fleet of battery powered equipment;
- Complete a trade-off study to assess the optimal bench height and excavator size.

1.23.3 Geotechnical – Open Pit

It is recommended that operations restrict production blasts to within 50 m of an unblasted presplit line. Once presplit is shot, production blasts will be taken to within 10 m of the presplit and then a trim shot used to clean the face. Given that larger production shots may be more likely to damage the final walls, all blasts shall be monitored, and blast designs shall be adjusted to avoid this.

It is recommended that the mine planners keep the ramp on the south wall to keep the overall slope angle at 52° or lower to manage the potential for deep seated toppling. Alternatively, the south wall should be interrupted with geotechnical benches to achieve the same overall slope angle. There is a potential for steepening if deep seated toppling can be shown not to be a concern based on slope performance and additional information.

Complete an overburden investigation program for slope and waste dump design and to identify localized areas of thicker overburden (>5 m). Plan and assign budget for:

- Routine bench mapping to document:
  - Achieved bench face angles vs. slope design;
  - Map fault and dyke exposures for predictive analyses;
  - Map orientation variability and persistence in foliation and jointing;
  - Seepage, which may preferentially occur along the foliation, dykes, or at the top of bedrock.
  - Vibrating wire piezometer installations with emphasis on the south wall because of the potential for deep-seated toppling.
  - Drain holes along the south wall ramp.
  - Prism installations and resources to monitor the prisms.

Any updated pit designs developed using these recommendations should be reviewed by a geotechnical engineer to validate that the slope designs have been applied correctly.

1.23.4 Underground Mining

For the next phase of the Project for the underground mine, DRA proposes the following recommendations to be performed:

- Trade-off Study to determine the ultimate open pit vs underground economic limit;
- Trade-off Study to combine the underground ore production with open pit production prior to Year 25;
- Geotechnical (hard rock) study for the underground mine design including the interaction (i.e., crown pillar extraction) between the open pit and underground mine;

- Hydrogeology study for the underground mine design;
- Backfill study including a trade-off study on paste backfill.

Prior to the Project detailed engineering phase for the underground mine, DRA proposes the following work as optional to be performed:

- A fully electric underground equipment fleet study; and
- Autonomous underground equipment fleet study.

1.23.5 Process

Based on the work performed and the test results, additional testwork and design modifications can be performed to both optimise and de-risk the process design and equipment selection. It is recommended to perform certain work for the next stage of the Project:

- To confirm closed circuit performance, recirculation performance, and overall plant recovery, it is recommended that an independent laboratory test the flowsheet at pilot scale as one continuous process.
- In order to control the grinding product, NLI should consider an automated media feeder. This would ensure near constant ball loading in the grinding circuit and reduce somewhat the slime production.
- An on-stream analyser is planned for installation in Year 2. In order to simplify the integration of the analyser, it is recommended to install it with the rest of plant and commission it once the plant has achieved stability.
- It is recommended to evaluate the material flow properties for the plant feed and tailings areas material to ensure proper bin design.
- It is recommended to carry out variability testing on samples representing the mine plan for Years 6-24. The geometallurgical methods for this testing will be prepared in 2023 for subsequent testing.
- A review of the plant feed bin (fine ore overflow hopper), fine ore screen feed chute, and tailings loadout designs is recommended to be performed in the next phase to ensure smooth operation.
- A complete review of all coarse pumping applications is required in the next phase. The current pumping arrangements will cause decreased availability. Larger pumps, ceramic lined pipes, fluidizers, and proper top-size protection is recommended.
- A complete review of all pumps, pipeline sizes, and equipment feed boxes is recommended following the water balance review.
- A review of the ore sorter feed arrangement is recommended, as the vibrating feeders are not designed to be used for feed control, only feed presentation.
- Due to the complexity of the flowsheet, it is recommended for NLI to be proactive with their operational readiness, development of operator training manuals, and on-boarding of skilled labor and operators. This will reduce commissioning and start-up risks.

It is recommended to investigate the use of dozer traps for front-end loader applications or the use of de-lumpers to make it easier to feed/operate the plan.

1.23.6 Mine Waste and Water Management

It is recommended to account for the following points when reviewing the mine waste management and the water management strategy for the site and completing the engineering design of CSF infrastructure:

Mine Waste Management

The CSF structure is designed to store filtered waste rock and tailings. Inadequate filtration over a long period of time can lead to significant operational problems. The design of the CSF will require a prescribed low water content to achieve the required level of compaction. In this regard, compaction and water content control protocol should be implemented.

The placement of properly filtered tailings is not simple in a cold and wet climate. It will require some learning on the part of the operator. NLI must plan for the possibility of lost time, particularly early in the mine ramp-up.

Geotechnical investigations (boreholes and test pits) will be required as part of the detailed design of the CSF to specify the nature of the soils in the footprint and to specify the extent and costs of preparatory work.

It is assumed that the tailings do not generate acid and do not leach metals. If these conditions change, it could significantly alter the CSF foundation. Nemaska shall inform the designer if a change in tailings or waste rock geochemistry is observed.

The advancement of the mine closure plan by Nemaska is recommended.

1.23.7 Water Management

The side-wide water management strategy and the water balance study are to be updated during future engineering design phases and mine operations; accounting, whenever possible, for the construction sequence of the water management infrastructure based on the planned site development.

For the current water management strategy, WSP recommended evaluating to have a polishing pond within the ETP in order to increase the robustness of the water management system.

The current water management strategy considers that all collected contact water is pumped towards the BC-11 pond before the ETP. It is possible that contact water from different sources have different contaminant concentrations. It may be advantageous to have a ETP water intake at BC-01 and provide a connection between ponds BC-12 and BC-01 to treat the water without passing through BC-11. This should be considered when water quality analysis become available.

Considering that water will need to be managed during the winter period, Nemaska must ensure that the system does not freeze and remains operational all year long.

Ponds and pump systems should be equipped with flow and level monitoring instruments, necessary for water management during operations.

Climate and runoff monitoring should be undertaken during mine operations to reduce uncertainties in the water balance predictions.

Given the limitation of the discharge flow into the Nemiscau River, at the beginnings of the operation, Nemaska will have to evaluate the possibility of increasing the effluent capacity.

The design of the water management infrastructure must be based on improved topographic survey data.

1.23.8 Engineering Activities

It is recommended to continue with the engineering activities prior to implementation of the Project. The activities would include the following so as not to delay the construction activities and complete the Project on time:

- Complete the contracting strategy for the Project.
- Preparation of equipment specifications for long delivery items, proceed to tender, recommendation of bidders and prepare purchase orders.
- Preparation of contractual documents for civil and concrete works, proceed to tender and recommendation of contractors and prepare contracts.

2 INTRODUCTION

This report provides exploration results and mineral resource and reserve estimates for spodumene concentrate production from the Whabouchi Mine. This report was prepared to pre-feasibility standards in accordance with SEC regulations S-K 601(b) (96) on behalf of Nemaska Lithium Incorporated (Nemaska or NLI; the Registrant) for its lithium-bearing spodumene mining operation, Whabouchi Project. Other downstream facilities, including a transshipment site at Matagami, Quebec and a chemical processing facility designed to convert spodumene concentrate to lithium hydroxide located in Bécancour, Quebec are not part of the Whabouchi Project and are outside the scope of this report. The Whabouchi Property is located in the Eeyou Istchee/James Bay area of the Province of Quebec, approximately 30 km east of the community of Nemaska and 300 km north-northwest of the town of Chibougamau, more specifically at km 276 on the Route du Nord.

The effective date of this report is December 31, 2022. The Qualified Persons determined that no updates to this report were necessary to make it materially current as of December 31, 2022 or the date hereof.

2.1 Source of Information

DRA Americas Inc. (DRA) were the overall lead consultants in their respective areas and collaborated with other consultants (for scope that fell outside of DRA's areas of expertise or assigned by Livent), specifically SGS, BBA and WSP. Sections of this report prepared by DRA, SGS, BBA, or WSP refer to the respective company and not to any specific individual employee.

SGS's Qualified Persons are responsible for Sections 3-9, 11, 20 and related contributions to Sections 1, 23, and 24. BBA's Qualified Persons are responsible for Sections 12 (except 12.4.2),13 and related contributions to Sections 1, 23, and 24. WSP's Qualified Persons are responsible for Sections 12.4.2, 17.5, 17.6, and related contributions to Sections 1, 23, and 24. DRA prepared all sections of the report that are not identified above as being prepared by another entity.

The Qualified Persons reviewed technical reports and data provided by NLI on the general setting, geology, project history, mine exploration, sampling results, quality assurance, metallurgy, economic and market analyses to complete this report. Additional sources of information are provided in Section 24 References. The Qualified Persons' reliance on information provided by the registrant is set forth in Section 25.

DRA, SGS, BBA, WSP and other third-party collaborators are not insiders, associates, or affiliates of Livent nor have any of them acted as advisor to Livent, its subsidiaries or its affiliates, in connection with this Project.

2.2 Site Visit

This Section provides details of the personal inspection on the Property by the Contributing Authors (Table 2-1).

Table 2-1 Site Visit

Company	Date of Site Visit
BBA	May 20 and 21, 2019
DRA	December 01, 2021 and March 28, 2022
SGS	July 11, 2022
WSP	October 17 and 18, 2022

2.3 Units and Currency

In this Report, all currency amounts are Canadian Dollars (CAD, $) unless otherwise stated, with commodity prices typically expressed in US Dollars (USD). Quantities are generally stated in *Système international d'unités* (SI) metric units, the standard Canadian and international practices, including metric tons (tonnes, t) for weight, and kilometres (km) or metres (m) for distance. The abbreviations provided in Table 2-2 are used in this Report.

Table 2-2 Abbreviations

Abbreviation	Description
µm	Microns, Micrometre
' or ft	Feet
" or in	Inch
$	Dollar Sign
$US or USD	United States Dollar
$/t	Dollar per Metric Tonne
%	Percent Sign
% w/w	Percent Solid by Weight
wt%	Percent by Weight
°	Degree
°C	Degree Celsius
2D	Two-Dimensional
3D	Three-Dimensional
>	Superior to
<	Inferior to
AACE	Association for the Advancement of Cost Engineering
AAS-FP	Atomic Absorption Spectrometry – Flame Photometry
ABA	Acid-Base Accounting
Ag	Silver
Agency	Canadian Environmental Assessment Agency
Agreement	Chinuchi Agreement
ARD	Acid Rock Drainage
Au	Gold
Ba	Barium
BBA	BBA Inc.
BC	British Columbia
Be	Beryl/Beryllium
BFA	Bench Face Angle
BMI	Benchmark Minerals Intelligence
BWi	Bond Ball Mill
C or CAD	Canadian Dollar
$CaCO_3$	Calcite
CaO	Calcium Oxite
CA	Certificate of Authorization
Capex	Capital Expenditures
CBW	Catch Bench Width
CCAA	Compagnies' Creditors Arrangement Act
Cd	Cadmium
CEAA	Canadian Environmental Assessment Act
CEHQ	Centre d'Expertise Hydrique du Québec
cfm	Cubic Feet per Minute
CIM	Canadian Institute of Mining, Metallurgy and Petroleum

Abbreviation	Description
cm	Centimetre
CM	Construction Management
CN	Canadian National Railroad
CNG	Cree Nation Government
COG	Cut-off Grade
COMEX	Review Committee of the JBNQA
COV	Coefficient of Variation
CPP	Cumulative Probability Plot
Cr	Chromium
Cr_2O_3	Chromic Oxide
CRF	Cemented Rock Fill
Critical Elements	Critical Elements Lithium Corporation
CRM	Certified Reference Material
CRS	Cemented Rock Fill
CSF	Co-Disposal Storage Facility
Cu	Copper
d	Day
DC	Bicarbonate Decomposition
DCS	Distributed Control System
DFS	Definitive Feasibility Study
DHIMS	Dry High Intensity Magnetic Separator
DMS	Dense Media Separation
DSO	Deswik Software
DSO	Direct Shipping Ore
DTH	Down-the-Hole
EA	Environmental Assessment
EC	Environment Committee
ECCC	Environment and Climate Change Canada
EDC	Economic Development Committee
EDO	Environmental Discharge Objective
EEMP	Environmental Effects Monitoring Program
EFD	Eriez Flotation Division
EGL	Effective Grinding Length
Eh	Redox Potential
EIS	Environmental Impact Statement
EP	Engineering and Procurement
EPCM	Engineering, Procurement and Construction Management
EPS	Enhanced Production Scheduler
EQA	Environmental Quality Act
ES	Engineering Sampling
ESIA	Environmental and Social Impact Assessment
ESS	Energy Storage System
ETP	Effluent Treatment Plant
EUR	Euro
EV	Electric Vehicle
EW	End Wall
Fe	Iron
Fe_2O_3	Ferric Oxide
FI/FO	Fly In/Fly Out
FoS	Factor of Safety
FS	Feasibility Study
FW	Footwall

Abbreviation	Description
F/X Rate	Exchange Rate
g	Grams
G&A	General and Administration
g/cm^3	Gram per Cubic Metre
g/t	Gram per tonne
GARD Guide	Global Acid Rock Drainage Guide
GCCEI	Grand Council of the Crees of Eeyou Istchee
GCRMM	Guide de caractérisation des résidus miniers et du minerai
GHG	Green House Gas
GISTM	Global Industry Standard on Tailing Management
GMS	G Mining Services
GNSS	Global Navigation Satellite System
GOH	Gross Operating Hour
GPS	Global Positioning System
H or hr	Hour
h/d	Hours per Day
h/y	Hours per Year
H_2SO_4	Sulfuric Acid
ha	Hectare
HARD	Half Absolute Relative Difference
HART	Highway Addressable Remote Transducer
HG	High Grade
HIMS	High Intensity Magnetic Separator
HLS	Heavy Liquid Separation
HMI	Human Machine Interface
HP	Horsepower
HQ	Drill Core Size (6.4 cm Diameter)
HVAC	Heating Ventilation and Air Conditioning
I/O	Input / Output
IAA	Impact Assessment Act
IAAC	Impact Assessment Agency of Canada
IATF	International Automotive Task Force
ICMM	International Council on Mining and Metals
ICP-AES	Inductively Coupled Plasma – Atomic Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma – Mass Spectroscopy
ICP-OES	Inductively Coupled Plasma – Optical Emission Spectroscopy
ID^2	Inverse Distance Square
INAP	International Network for Acid Prevention
IRA	Inter-ramp
IRR	Internal Rate of Return
ISAQ	Inventaire des Sites Archéologiques du Québec
ISP	Internet Service Provider
JBNQA	James Bay and Northern Quebec Agreement
JMBM	Johnson Matthey Battery Material
K_2O	Potassium Oxide
K-40	Potassium-40
$KAl_2(Al\text{-}Si_3O_{10})(OH)_2$	Muscovite
$KAl\text{-}Si_3O_8$	Microline
kg	Kilogram
$kg/m^2/h$	Kilogram per square metre per hour

Abbreviation	Description
km	Kilometre
km/h	Kilometre per hour
km^2	Square kilometre
KNA	Kriging Neighbourhood Analysis
KPI	Key Performance Indicator
kt	Kilotonne
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per Metric Tonne
L	Litre
lb	Pound
lb/h*in	Pound per Hour per Inch
LCE	Lithium Carbonate Equivalent
LCO	Lithium Cobalt Oxide
LFMP	Lithium Iron Manganese Phosphate
LFP	Lithium-Iron-Phosphate
LHD	Load-Haul Dumb
LG	Low Grade
LHM	Lithium Hydroxide Monohydrate
Li	Lithium
$LiAlSi_4O_{10}$	Petalite
LiDAR	Laser Imaging Detection and Ranging Survey
Li_2CO_3	Lithium Carbonate
Li_2O	Lithium Oxide
Li_2SO_4	Lithium Sulfate
LiOH	Lithium Hydroxide
$LiOH\text{-}H_2O$	Lithium Hydroxide Monohydrate
LIMS	Low Intensity Magnetic Separator
LME	London Metal Exchange
LMO	Lithium Manganese Oxide
LOI	Loss on Ignition
LOM	Life of Mine
LQE	Loi sur la qualité de l’environnement
LTE	Long Term Evolution
LV	Light Vehicle
m	Metre
M	Million
M&I	Measured and Indicated
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/s	Cubic Metre per Second
m^3/y	Cubic Metre per Year
mA	Milliampere
MAC	Mining Association of Canada
Mb/s	Mega Bits per Second
Mm^3	Million Cubic Metres
MCC	Motor Control Center
MCS	Master Composite Sample
MDC	Map-Designated Claim
MDMER	Metal and Diamond Mining Effluent Regulations
MELCC	Ministère de l’environnement et de la lutte contre les changements climatiques

Abbreviation	Description
MELCCPF	Ministère de l'environnement, de la lutte contre les changements climatiques, de la faune et des parcs
MERN	Ministère de l'Énergie et des Ressources Naturelles
MDMER	Metal and Diamond Mining Effluent Regulation
MFFP	Ministère des Forêts, de la Faune et des Parcs
Mg	Magnesium
MgO	Magnesium Oxide
mg	Microgram
mg/L	Microgram per Litre
MIMS	Medium Intensity Magnetic Separation
ML	Metal Leaching
mm	Millimetre
Mm^3	Million Cubic Metre
MMU	Mobile Manufacturing Unit
Mn	Manganese
MnO	Manganese(II) Oxide
MOU	Memorandum of Understanding
MPa	MegaPascal
MPSO	MinePlan Schedule Optimizer
MRE	Mineral Resource Estimation
MRNF	Ministère des ressources naturelles et des Forêts
MSDC	Multiservice Day Carte Centre
MSO	Mineable Stope Optimizer
Mt	Million Metric Tonne
Mt/y	Million Metric Tonne per year
MTO	Material Take-Off
MVA	Mega Volt-Ampere
MW	Megawatts
n/a	Non-Applicable
Nº or #	Number
Na	Sodium
Na_2O	Sodium Oxide
Na_2SiO_3	Sodium Metasilicate
Na_2SiO_4	Sodium Sulfate
$NaAl\text{-}Si_3O_8$	Albite
NAG	Non-Acid Generating
NaOH	Sodium Hydroxide
Nb	Niobium
NCA	Nickel-Cobalt-Aluminium
NE	Northeast
NEBA	National Energy Board Act
Ni	Nickel
NI	National Instrument
NLI	Nemaska Lithium Inc.
NNP	Net Neutralization Potential
NOH	Net Operating Hour
NPAG	Non-Potentially Acid Generating
NPR	Neutralisation Potential Ratio
NPV	Net Present Value
NQ	Drill Core Size (4.8 cm diameter)
NRCan	Natural Resources Canada
NSR	Net Smelter Return
NTU	Nephelometric Turbidity Units

Abbreviation	Description
O	Oxygen
OK	Ordinary Kriging
OP	Open Pit
Opex	Operating Expenditures
ORP	Operational Readiness Plan
P	Phosphate
P&ID	Piping & Instrumentisation Diagram
P&P	Proven and Probable
P1P	Phase 1 Plant
P2P	Phase 2 Plant
P_2O_5	Phosphorus Pentoxide
P_{80}	80% Passing
PAG	Potentially Acid Generating
PCS	Process Control System
PC	Personal Computer
PEA	Preliminary Economic Assessment
PF	Process Flow
PF	Productivity Factor
pH	Potential Hydrogen
PIR	Primary Impurity Removal
PLC	Programmable Logic Controllers
PPE	Personal Protective Equipment
ppm	Parts per Million
PPMOF	Pre-Assembly, Modularisation and Off-Site Fabrication
PPSRTC	Politique de protection des sols et de réhabilitation des terrains contaminés
PTAC	Packaged Terminal Air Conditioner
PZ	Petalite Zone
Q1, 2, 3, and 4	First, Second, Third, and Fourth Quarter
QA/QC	Quality Assurance/Quality Control
QC	Province of Quebec
QoS	Quality of Service
QP	Qualified Person
RDPA	Resource Development Partnership Agreement
REACH	Registration, Evaluation and Authorization of Chemicals
RF	Revenue Factor
RF	Rock Fill
RFID	Radio Frequency Identification
RoHS	Restriction of Hazardous Substances
ROM	Run-of-Mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
RSC	Reference Spodumene Concentrate
RTD	Resistance Temperature Detector
RVO	Reverse Vesting Order
s	Second
S	Sulfur
SARA	Canadian Species at Risk Act
SDBJ	Société de Développement de la Baie James
SEDAR	System for Electronic Document Analysis and Retrieval
SFE	Shake Flask Extraction
SG	Specific Gravity

Abbreviation	Description
SGS	SGS – Canada Inc. or SGS – Geostat or SGS – Lakefield or SGS – Mineral Services
Si	Silicium
SiO_2	Quartz
SI	Système international d'unités
SIR	Secondary Impurity Removal
SP WDS	Salient Pole Rare Earth Magnetic Wet Drum Separator
SW	Southwest
t	Metric Tonne
t/d	Metric Tonne per Day
t/h	Metric Tonne per Hour
$t/m^2/h$	Tonne per Square Metre per Hour
t/m^3	Metric Tonne per Cubic Metre
t/y	Metric Tonne per Year
Ta	Tantalum
Ta_2O_5	Tantalum Pentoxide
TCLP	Toxicity Characteristic Leaching Procedure
Th-232	Thorium-232
Ti	Titanium
TiO_2	Titanium Dioxide
TIMA-X	Tescan Integrated Mineral Analyzer
TIR	Tertiary Impurity Removal
TJMC	Table Jamésienne de Concertation Minière
TSCA	Toxic Substances Control Act
TSS	Total Suspended Solids
U	Uranium
U/G or UG	Underground
UPS	Uninterruptible Power Supply
UQAT	Université du Québec en Abitibi-Témiscaninque
URSTM	Unité de recherche et de service en technologie minière
USA	United States of America
UTM	Universal Transverse Mercator
V	Volt
V_2O_5	Vanadium(V) Oxide
VFD	Variable Frequency Drive
VLF	Very Low Frequency
W	Watt
WBS	Work Breakdown Structure
WHIMS	Wet High Intensity Magnetic Separation
WIC	Whabouchi Implementation Committee
WTP	Water Treatment Plant
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
XRT	X-Ray Transmission
y	Year
Zn	Zinc

2.4 Report Version Update

This report marks the first mineral resource and reserve disclosure submitted by Livent for its lithium brine mining operation at Whabouchi in accordance with 17 Code of Federal Regulations (CFR) § SEC 229.1300. This report was amended to include additional clarifying information in November 2023. The basis of the report is unchanged from its original filing date (September 8, 2023). Modifications to this version are summarized below:

- Stratigraphic column added as Table 6-1.
- QP opinion added to Section 11.15.
- Annual information for life-of-mine production disclosed in Section 13.1.
- QP opinion added to Section 17.7.
- Annual economic analysis of reserves added to Section 19.6.
- Typos corrected.

As this report may be updated later following acquisition of new and relevant data or information, the user should confirm that this is the latest filed version of the report.

3 PROPERTY DESCRIPTION

This section includes summary information on the Whabouchi Mine site location, property ownership, and environmental permits.

3.1 Location

The Whabouchi Property is located in the James Bay area of the Province of Quebec, approximately 30 km East of the Cree community of Nemaska and 300 km north-northwest of the town of Chibougamau. The center of the Property is situated approximately at UTM 5,725,750 mN, 441,000 mE, NAD83 Zone 18 (Figure 3-1). The Property is accessible by the Route du Nord, the main all-season gravel road linking Chibougamau and Nemaska. The Property is also accessible through Matagami by the Route Billy-Diamond Highway. The road crosses the Property near its center. The Nemiscau airport is 18 km west of the Property (Figure 3-2).

3.2 Property Ownership and Agreements

The Property is composed of one block containing 35 map-designated claims (MDC) covering a total of 1,632.24 hectares and one (1) Mining Lease by the Ministère de l'Énergie et des Ressources naturelles (MERN). Nemaska Lithium Inc. (NLI or Nemaska Lithium) owns 100% interest in the Property (Figure 3-3andTable 3-1). Livent owns a 50% interest in NLI with Investment Quebec (IQ) owning the remaining 50%.

On October 26, 2017, NLI obtained the Mining Lease number 1022, under the conditions provided for in the Mining Act and those prescribed by regulation. The surface of the Mining Lease totals 138.106 ha, consisting of lot 4,994,037 of the Quebec cadastre, registration division of Lac-Saint-Jean-Ouest. This lease gives the tenant the right to extract all mineral substances owned by the Crown in the above-named land, but it does not give entitlement to surface mineral substances, petroleum, natural gas, or brine. This lease is for a period of 20 years from the date of the landlord's signature on October 26, 2017 and will end on October 25, 2037.

As of the effective date of this Report, all 35 claims are in good standing. The mining lease expires on October 25, 2037. The expiry dates for the claims range from November 2, 2024 to January 24, 2025. Several mining titles affect the mining lease: 101251, 101252, 2203108, 2519870, 2519871, 2137247, 2137248, 2137249, 2141920, 2137250, 2137251, 2137252. Mining leases can be renewed three times in 10-year increments for a nominal fee.

In September 2009, NLI acquired a 100% interest in certain mining claims included in the Whabouchi property in exchange for a consideration which included a 3% Net Smelter Return (NSR) royalty on such claims and on four (4) other claims later acquired by map designation payable to the vendor.

For an amount of C$1,000,000, a 1% NSR royalty may be repurchased by the Corporation once it has officially declared that it is in commercial production.

On October 15, 2020, in connection with NLI's CCAA proceedings, the Superior Court of Quebec issued a reverse vesting order (an RVO) pursuant to which NLI was acquired and declared free and clear of the claims of creditors. A holder of a pre-proceeding royalty asserted that such royalty is a "sui generis real right or royalty right in and to the assets and properties of the Nemaska Entities" which the RVO cannot purge. NLI contested the claim, and the Monitor appointed in connection with the CCAA proceedings supported NLI's view. On September 11, 2023, the Quebec Superior Court dismissed the holder's claim that he acquired a real right and declared that NLI owns and holds the property free and clear of any right of the claimant.

The following mining titles (2137259, 2137260, 2137261 and 2137262) are on Category II land. Category II lands are areas in which Native Community have the exclusive rights to hunt and fish (CBJNQ - 5.2.6 b).

The Category II land affected claims fall under the Chinuchi Agreement, signed in 2014 and available on SEDAR and NLI website.

The following is a brief description of the objectives of The Chinuchi Agreement (Agreement). The Agreement was signed to provide for the establishment and maintenance of a long-term working relationship between NLI and the Cree nation of Nemaska; the Grand Council of the Crees (Eeyou Istchee) and the Cree Nation Government based on mutual trust and respect; to adopt and maintain a sustainable development approach during all phases of the Whabouchi Project; to provide for a framework through which communication and cooperation can take place between the Parties in the performance of their respective obligations under this Agreement.

Figure 3-1 Property General Location

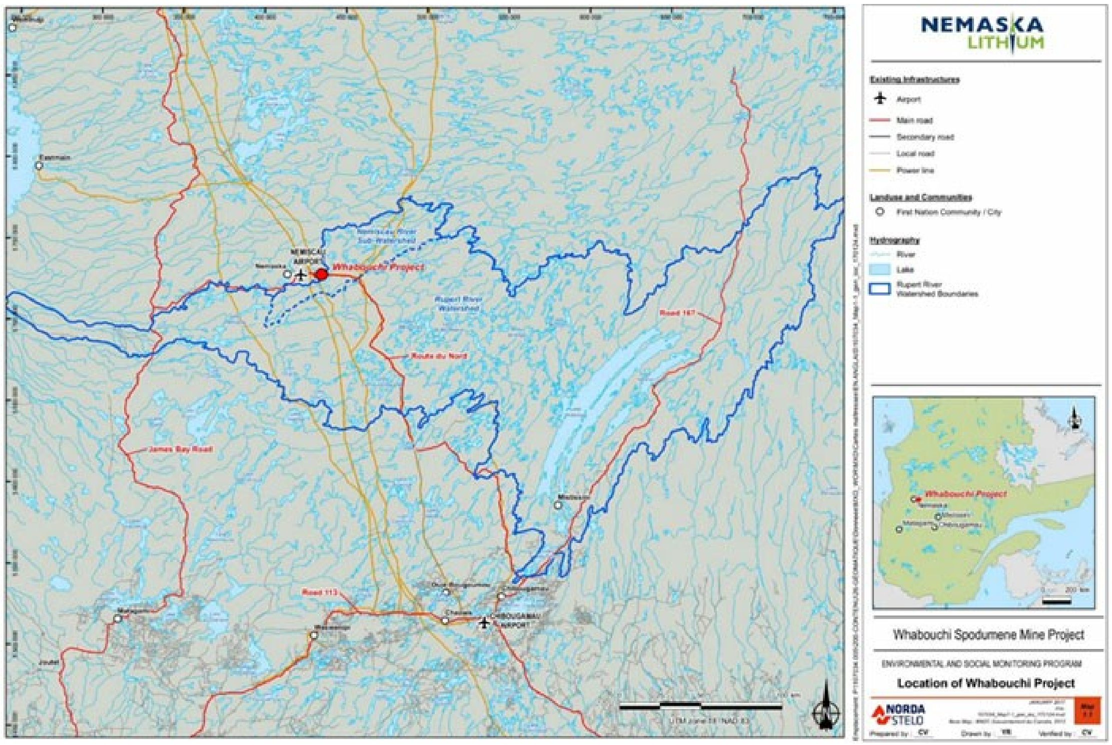


Figure 3-2 Property Location with Nearby Infrastructure

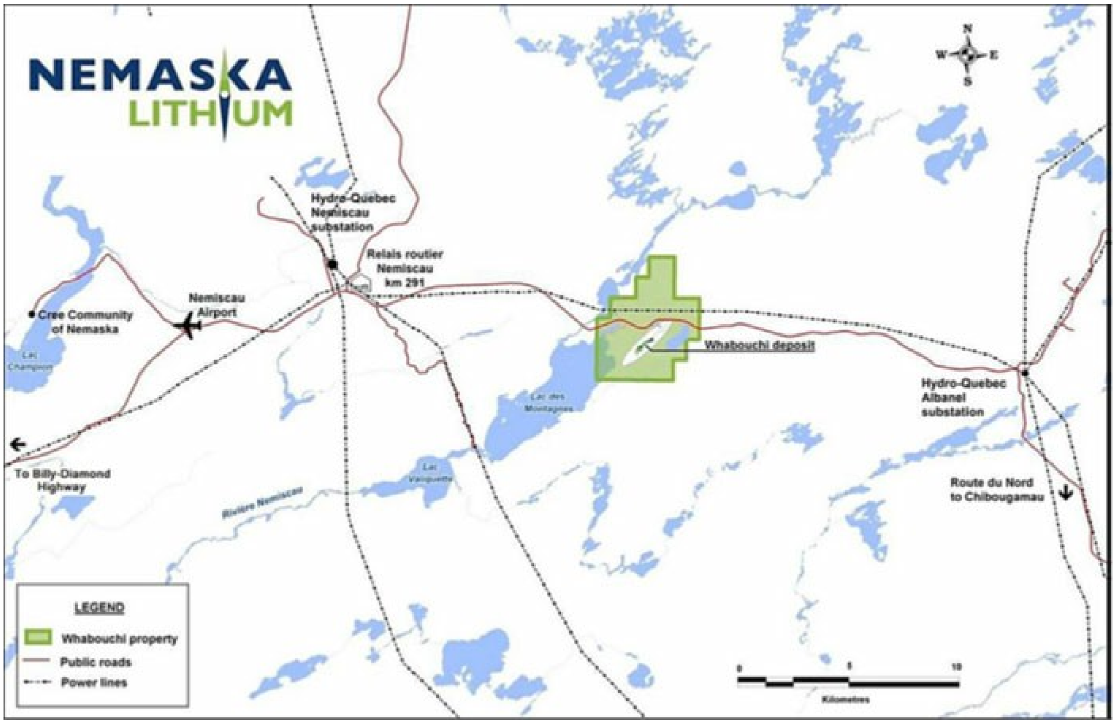
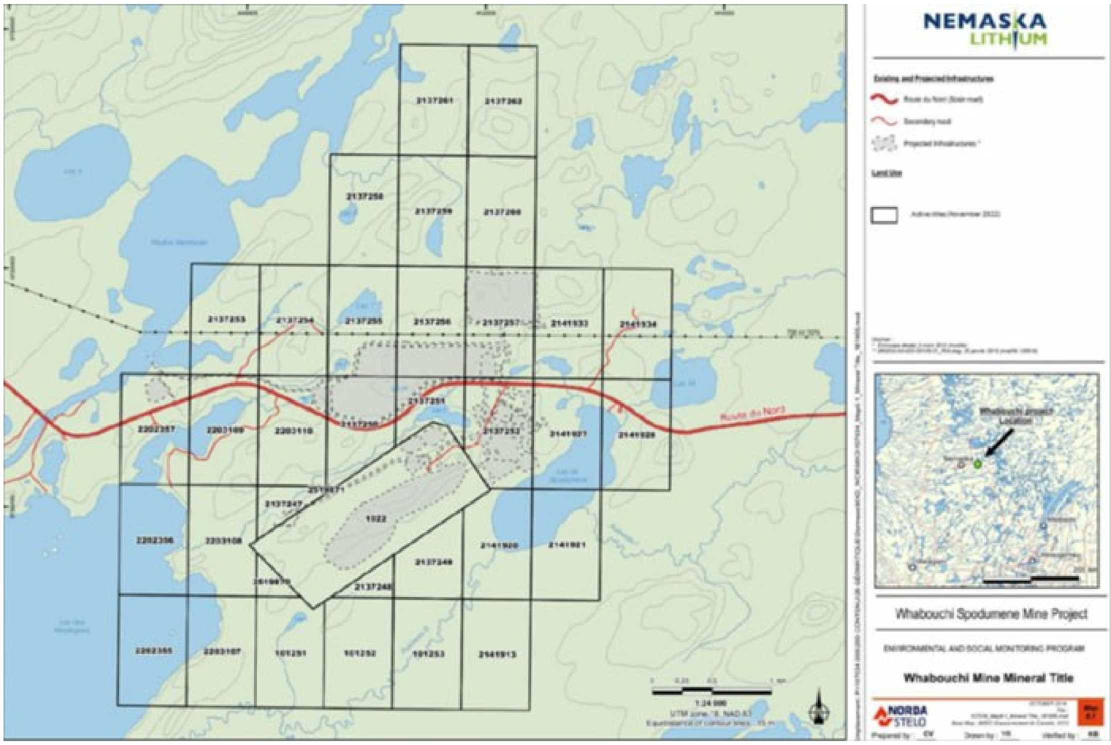


Table 3-1 List of the Property Mineral Titles (information collected on 2022-11-04)

Title Number	Type of Title	SNRC Sheet	Area (ha)	Status	Registration Date	Expiration Date	Renewals Done	Title Holder
1022	Mining Lease	32O12	138.1	Active	2017-10-26	2037-10-25	0	100% Nemaska Lithium inc.
101251	MDC	32O12	52.3	Active	2005-11-03	2024-11-02	8	100% Nemaska Lithium inc.
101252	MDC	32O12	53.4	Active	2005-11-03	2024-11-02	8	100% Nemaska Lithium inc.
101253	MDC	32O12	53.4	Active	2005-11-03	2024-11-02	8	100% Nemaska Lithium inc.
2137247	MDC	32O12	14.7	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137248	MDC	32O12	9.6	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137249	MDC	32O12	32.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137250	MDC	32O12	45.1	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137251	MDC	32O12	27.1	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137252	MDC	32O12	58.1	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137253	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137254	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137255	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137256	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137257	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137258	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137259	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137260	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137261	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2137262	MDC	32O12	53.4	Active	2007-11-26	2024-11-25	7	100% Nemaska Lithium inc.
2141913	MDC	32O12	53.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2141920	MDC	32O12	51.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2141921	MDC	32O12	53.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2141927	MDC	32O12	53.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2141928	MDC	32O12	53.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2141933	MDC	32O12	53.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2141934	MDC	32O12	53.4	Active	2008-01-24	2025-01-23	7	100% Nemaska Lithium inc.
2202355	MDC	32O12	53.4	Active	2010-01-21	2025-01-20	6	100% Nemaska Lithium inc.
2202356	MDC	32O12	53.4	Active	2010-01-21	2025-01-20	6	100% Nemaska Lithium inc.
2202357	MDC	32O12	53.4	Active	2010-01-21	2025-01-20	6	100% Nemaska Lithium inc.
2203107	MDC	32O12	53.4	Active	2010-01-25	2025-01-24	6	100% Nemaska Lithium inc.
2203108	MDC	32O12	53.1	Active	2010-01-25	2025-01-24	6	100% Nemaska Lithium inc.
2203109	MDC	32O12	53.4	Active	2010-01-25	2025-01-24	6	100% Nemaska Lithium inc.
2203110	MDC	32O12	53.4	Active	2010-01-25	2025-01-24	6	100% Nemaska Lithium inc.
2519870	MDC	32O12	6.9	Active	2018-06-21	2024-11-25	2	100% Nemaska Lithium inc.
2519871	MDC	32O12	0.2	Active	2018-06-21	2024-11-25	2	100% Nemaska Lithium inc.

Figure 3-3 Whabouchi Property Mineral Titles



3.3 Permits and Environmental Liabilities

The main permits required to conduct exploration work on the Property are the forest management permit delivered by the provincial *Ministère des Ressources naturelles et des Forêts (MRNF)* along with owning active mining rights. A Certificate of Authorization (CA) from the *Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP)* may also be necessary to conduct specific advanced exploration works such as, for example, the mechanical stripping of more than 1,000 m^3 of overburden.

As of the time of writing this Report, NLI's management confirmed having valid environmental permits and authorizations. As the Project engineering has progressed since the original permits were granted, discussions are ongoing with *MELCCFP* to clarify the permit modification request that will be required. *MELCCFP* has already confirmed that construction can proceed on some key Project infrastructure per the already granted permits. Most notably a modification will be required for the relocation of the camp infrastructure, but management does not anticipate any difficulties in obtaining this. To the knowledge of the author, there are no environmental liabilities pertaining to the Property.

For the Whabouchi mine Project, a first version of the Environmental and Social Impact Assessment (ESIA) document was submitted to both federal (Canadian Environmental Assessment Agency) and provincial (Review Committee of the James Bay and Northern Quebec Agreement, or COMEX) authorities for review in April 2013. Following questions and comments as well as public hearings, NLI announced September 4, 2015 that it has received the General CA for the Whabouchi Project from the then *Ministère de l'Environnement, de la Lutte contre les changements climatiques (MELCC, now MELCCFP)*.

On July 29, 2015, the Canadian Minister of Environment decided that the Project is not likely to cause any significant adverse environmental effects and set out in her positive decision statement the conditions relative to the mitigation measures and monitoring program to be respected by NLI. NLI has already begun and is continuing to fulfill the provisions included in the General CA for Whabouchi. Any required permit modification application request will continue to be filed in a timely manner with the construction works and are therefore not anticipated to impact on Project schedule.

Section 17 provides further information on environmental impacts and permitting.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

4.1 Whabouchi Site

Information on the Whahbouchi site accessibility, physiography, climate, resources and infrastructure, and surface rights are provided in the following Sections.

4.1.1 Accessibility

The Property is easily accessible via the Route du Nord that crosses the Property near its center. This road links the town of Matagami, via the Route Billy-Diamond highway, approximately 390 km to the SSW. The Route du Nord also links the town of Chibougamau, located approximately 300 km to the SSE, and leads to the community of Nemaska.

4.1.2 Physiography

The Property is characterized by a relatively flat topography, with the exception of the local ridge where the more competent pegmatites outcrop, forming the surface expression of the deposit. The elevation above sea level ranges from 275 m, at the lowest point on the Property, to 325 m at the top of the pegmatite ridge, with an average elevation of 300 m. Lakes and rivers cover approximately 15% of the Property area. The flora in the area is typical of the taiga environment observed in the region with a mix of black spruce forest and peat moss-covered swamps. A vast portion of the Property was devastated by forest fires less than 20 years ago. There is no permafrost at this latitude and the overburden cover ranges in depth from 0 m near the ridge to 25 m in the south part of the Property.

4.1.3 Climate

The climate in the region is sub-arctic. This climate zone is characterized by long, cold winters and short, cool summers. Daily average temperature ranges from -20°C in January to +17°C in July. Break-up usually occurs in early June, and freeze-up in early November. Precipitation mostly occurs between May and November with an average monthly rainfall of 135 mm. Annual snowfall is generally of 315 cm of snow, mostly between October and May. Averages are based on data from 2009 to 2022. (https://www.worldweatheronline.com/nemiscau-weather-averages/quebec/ca.aspx)

4.1.4 Local Resources and Infrastructure

The nearest infrastructure with general services is the Relais Routier Nemiscau Camp, located 12 km west of the Property, where Nemaska Lithium has access to lodging facilities, if needs exceed the capacity of the camp installed on the property. The community of Nemaska, located 30 km west of the Property, can also provide accommodation and general services. The area is serviced by the Nemiscau airport, serviced by regular Air Creebec flights and charter flights, and by mobile phone network from the main Canadian service providers.

Hydro-Québec owns several infrastructure and facilities in the area including the Poste Albanel and Poste Nemiscau electrical stations located approximately 20 km east and 12 km west from the Property, respectively. Electrical (735 kV) transmission lines connecting both stations run alongside the Route du Nord and cross the Property near its center. Also, a 69 kV power line connecting the Poste Nemiscau electrical station to the mine site has been put in service and is supplying power to the facilities.

4.1.5 Surface Rights

All claims comprising the Property are located on Crown Lands. NLI secured in October 2017 all surface rights to construct and operate the projected infrastructure.

5 HISTORY

The history of the Whabouchi Site including a discussion on regional surveys, mineral exploration work and technical studies performed by previous owners are provided in the following Sections.

5.1 Regional Government Surveys

Numerous geological surveys and geoscientific studies have been conducted by the Quebec Government in the James Bay area. Geological surveys in the 1960s (Valiquette, 1964, 1965 and 1975) cover the entire property area. In 1998, the *MRNF* released the results of a regional lake bottom sediment survey completed in 1997.

5.2 Mineral Exploration Work by Previous Owners

The first exploration work reported in the area dates back to 1962 by Canico and included the discovery of a lithium-bearing pegmatite by the geologists of the Québec Bureau of Mines. That same year, Canico drilled two (2) packsack drill holes on the pegmatite, followed by three (3) diamond drill holes on the same pegmatite ridge in 1963. A total of 462.99 m was drilled. The best result obtained was 1.44% Li_2O over 83.2 m (Elgring, 1962).

No exploration was reported for the next ten (10) years. In 1973, James Bay Nickel Ventures (Canex Placer) performed a large-scale geological reconnaissance that covered the property (Burns, 1973).

From 1974 to 1982, the exploration work was exclusively reported by the Société de Développement de la Baie James (SDBJ), which mainly executed large scale geochemical surveys, followed by geological reconnaissance of the anomalies (Pride, 1974, Gleeson, 1975 and 1976).

Two (2) exploration programs, one in 1978 and the other in 1980 were aimed at lithium exploration, with the evaluation of the Whabouchi spodumene-bearing pegmatite (Goyer, et al. 1978, Bertrand, 1978, Otis, 1980, Fortin, 1981, and Charbonneau, 1982). No channel sampling or drill holes are reported. No work was conducted from 1982 to 1987.

In 1987, Westmin Resources completed an airborne Dighem III survey. A part of this survey was located immediately east of the property (McConnell 1987). In 1987-1988, Muscocho Exploration also completed ground magnetic and VLF surveys that covered a major part of the property. The spodumene-bearing pegmatite gave a weak magnetic and VLF response. The Muscocho Exploration efforts were oriented towards the search for massive sulfides. A program of 14 holes, 11 of them located on the southern part of the Whabouchi Property, was completed. Several arsenic anomalies were obtained, with a maximum of 3,750 ppm in Hole ML-88-8 (Brunelle, 1987, Gilliatt, 1987 and Zuiderveen, 1988).

In 2002, while exploring for tantalum, Inco re-sampled the spodumene-bearing pegmatite, taking 11 channel samples and seven (7) grab samples. The best value obtained by Inco was 0.026% Ta, and Li_2O values ranging from 0.30% to 3.72% (Babineau, 2002).

In 2008, Golden Goose Resources visited and sampled the Valiquette (Ni) and chromite showings south of the Whabouchi Property (Beaupré, 2008).

5.3 Previous Technical Studies on the Property

An initial Mineral Resource was estimated in May 2010 by SGS Geostat and was followed by an initial Preliminary Economic Assessment (PEA) of the Project completed in March 2011 by Equapolar, in collaboration with BBA. The initial Mineral Resource Estimate of the Whabouchi Property, effective May 28, 2010, totaled 9.78 Mt grading 1.63% Li_2O in the Measured and Indicated Resources categories, with an additional 15.40 Mt grading 1.57% Li_2O in the Inferred Resources category.

Following further drilling in 2011, SGS Geostat provided NLI with an Updated Mineral Resource (effective June 6, 2011) to be included in the Preliminary Economic Assessment (Prepared by Met-Chem and dated October 2, 2012). This Updated Mineral Resource comprised 11.294 Mt of Measured Mineral Resources with an average grade of 1.58% Li_2O, 13.785 Mt of Indicated Mineral Resources with an average grade of 1.50% Li_2O and 4.401 Mt of Inferred Mineral Resources with an average grade of 1.54% Li_2O. The Mineral Resources were reported within an optimized pit shell and a cut-off grade of 0.43% Li_2O.

Following drilling campaigns in 2013, 2016, 2017 and 2018, SGS Geostat provided NLI with an Updated Mineral Resource (with an effective date of June 26, 2019). This updated open pit Mineral Resource comprised 17.734 Mt of Measured Mineral Resources with an average grade of 1.60% Li_2O, 20.532 Mt of Indicated Mineral Resources with an average grade of 1.33% Li_2O and 11.745 Mt of Inferred Mineral Resources with an average grade of 1.27% Li_2O (NI 43-101 Technical Report, August 9, 2019). This update also comprised an underground Mineral Resource of 0.274 Mt of Indicated Mineral Resource with an average grade of 1.13% Li_2O and 5.413 Mt of Inferred Mineral Resource with an average grade of 1.32% Li_2O. The open pit Mineral Resources were reported within an optimized pit shell and a cut-off grade of 0.30% Li_2O. The underground Mineral Resources were reported below the open pit shell with a cut-off grade of 0.60% Li_2O.

6 GEOLOGICAL SETTING, MINERALIZATION AND DEPOSIT

A summary of the Whabouchi geologic setting, mineralization and deposit characteristics are provide in this Section.

6.1 **Regional Geology**

The Whabouchi Property is located in the northeast part of the Superior Province of the Canadian Shield craton. The Superior Province extends from Manitoba to Quebec and is mainly composed of Archean-age rocks. The general metamorphism is of greenschist facies, except in the vicinity of intrusive bodies, where it reaches the amphibolite-to-granulite facies.

In Quebec, the eastern extremity of the Superior Province has been classified into nine (9) subprovinces, from south to north: 1) Pontiac, 2) Abitibi, 3) Opatica, 4) Nemiscau, 5) Opinaca, 6) La Grande, 7) Ashuanipi, 8) Bienville and 9) Minto (Hocq, 1994). According to Card and Ciesielski (1986), the area covered by the Property is located in the Opinaca or Nemiscau sub-province. Figure 6-1 shows the position of the Property in the Superior Province.

6.2 **Property Geology**

The Whabouchi Property is located in the Lac des Montagnes volcano-sedimentary formation and sits between the Champion Lake granitoids and orthogneiss and the Opatica Northeast, which comprises orthogneiss and undifferentiated granitoids. From the northwest to the southeast, the Property is underlain by the Champion Lake granitoids, a grey oligoclase gneiss and then by the Lac des Montagnes formation.

The Lac des Montagnes belt is approximately 7 km wide in the area, oriented northeast, and is principally composed of metasediments (quartz-rich paragneiss, biotite-sillimanite-staurolite schist and garnet-bearing schist) and amphibolites (mafic and ultramafic metavolcanics). These rocks are strongly deformed and cut by late granitoids (leucogranites and biotite-bearing white pegmatites) (Valiquette, 1975). Figure 6-2 shows the location of the Property relative to the Lac des Montagnes, the Champion Lake and Opatica NE formations. Table 6-1 summarizes the different lithologies occurring in the area.

Figure 6-1 Regional Geological Map

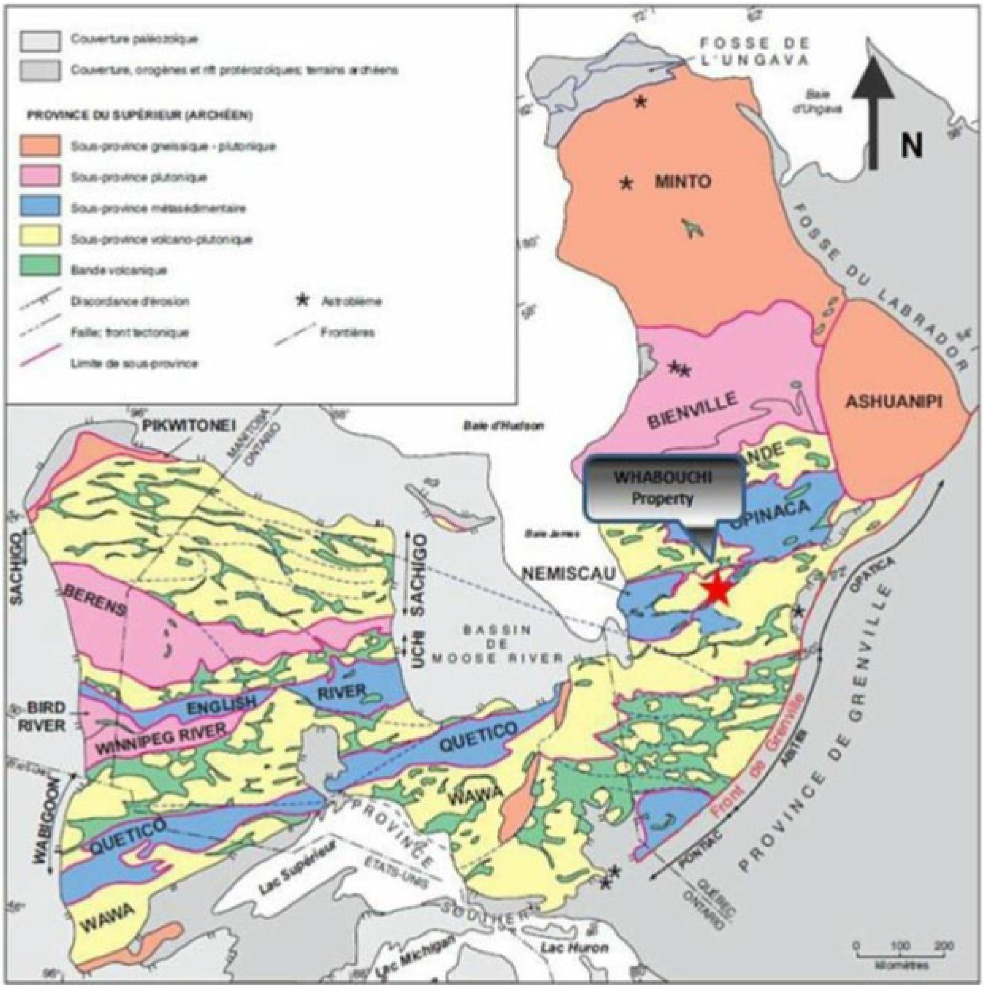


Figure 6-2 Local Geological Map

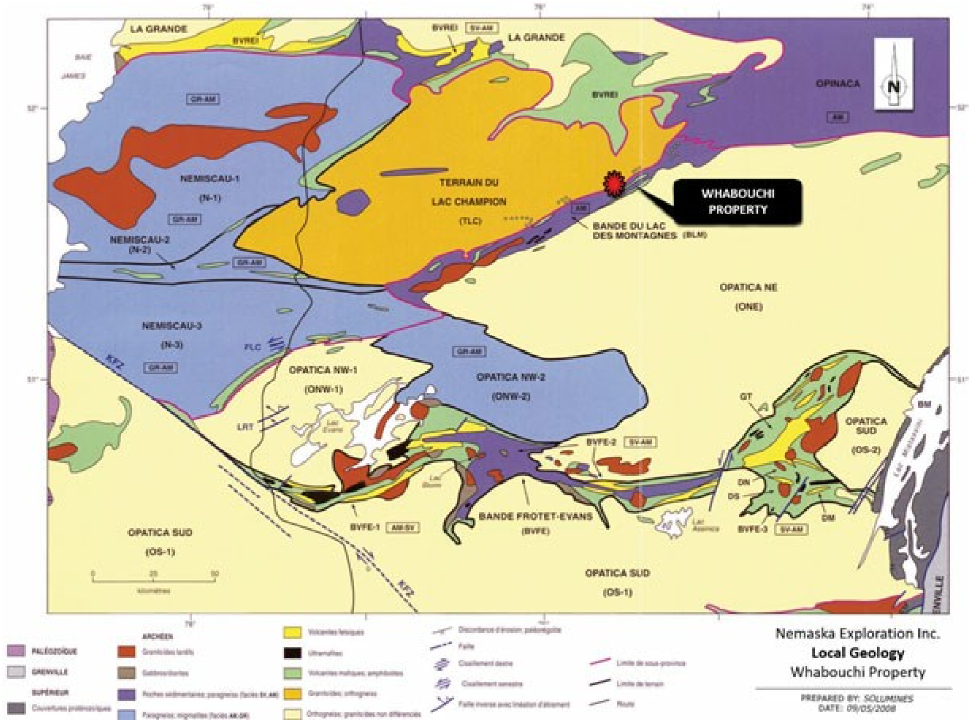


Figure 6-2.1 Generalized Geological Cross-Section Through the Whabouchi Deposit (West to Southwest)

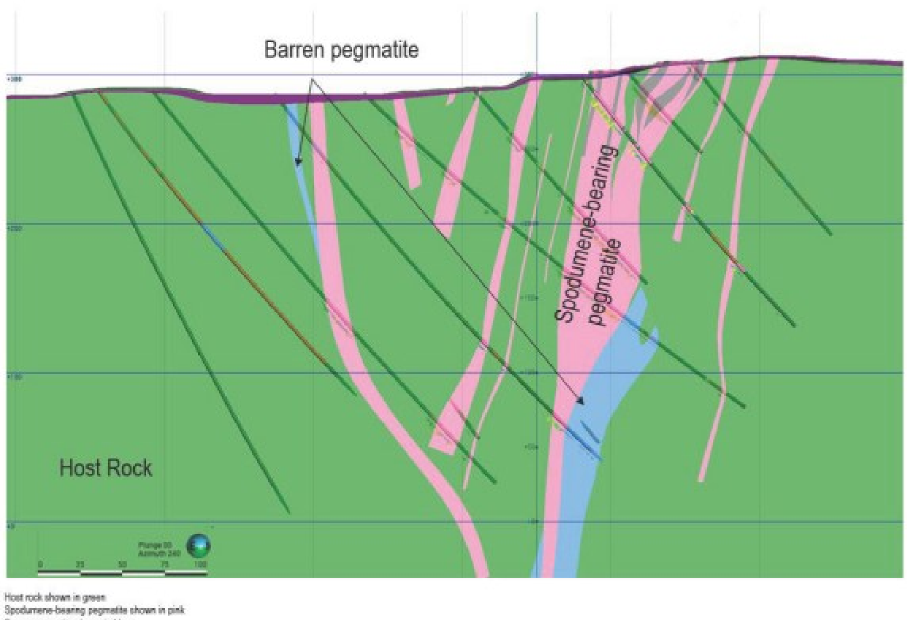


Table 6-1 Summary of the Different Lithologies Occurring in the Area

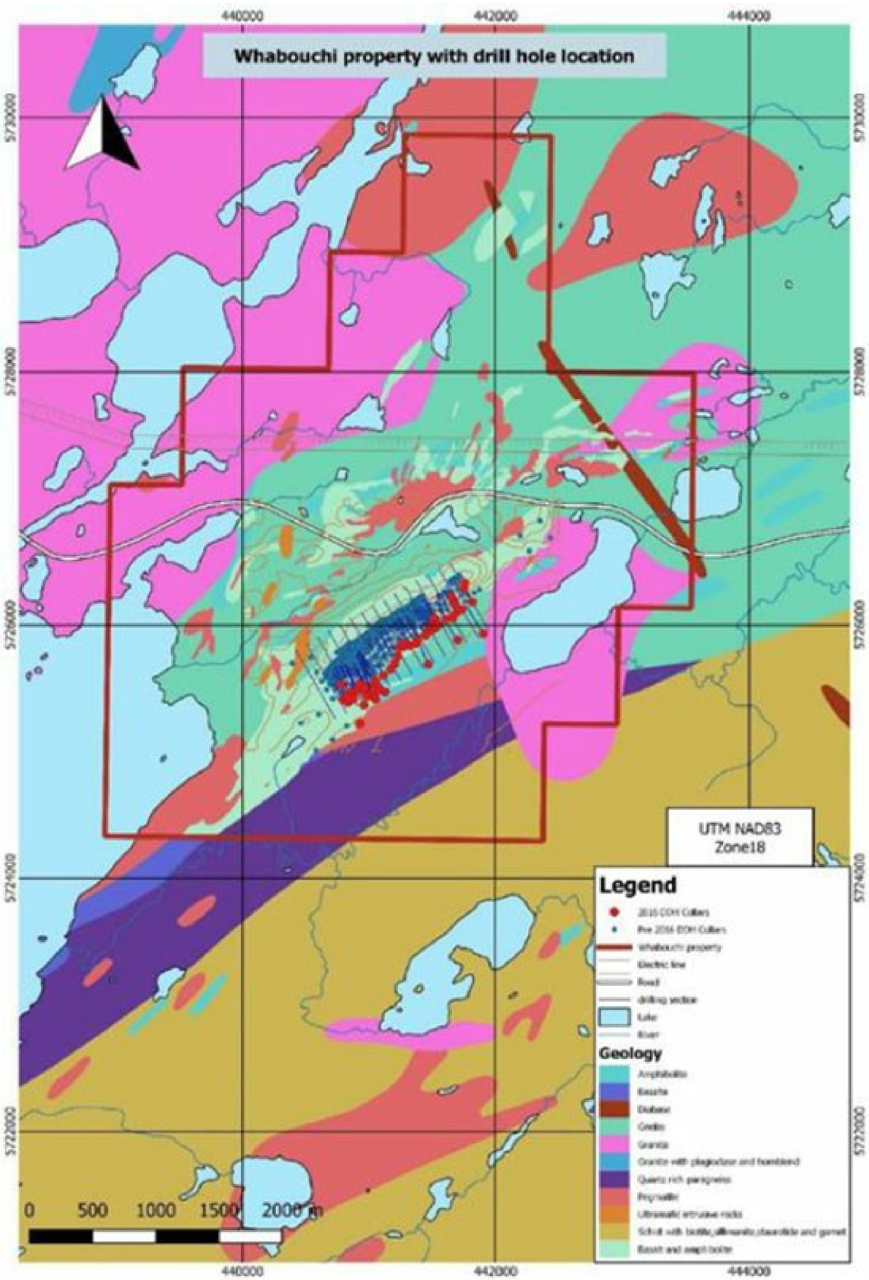
Eon or Series	Lithology	
Pleistocene and Holocene		Moraines, eskers, alluvial deposits, reticulated peat-bogs, morainic belts-
PRECAMBRIAN		11: Diabase
		10: Pegmatites "
		a) White with muscovite, tourmaline, garnet and magnetite
		b) Pink, with microcline
		9: White and pink granite
		8: Grey hornblende-oligoclase granite with phenocryst of pink microcline
		7: Ultramafic-rocks: Serpentinites, tremolite rocks
		6:-Homblende-plagioclase gneiss
		5: Metasomatic anthophyllite-cordierite rocks (mineralization susceptible)
		4: Paragneiss or biotite schists; gamet-biotite-schists; porphyroblastic-schist:
		a) Garnet, sillimanite, biotite
		b) Garnet, cordierite, biotite
		c)-Garnet, andalousite, biotite
		d)-Staurotide,-sillimanite, andalousite, biotite
		e)-Sillimanite, cordierite, andalousite, biotite
		f)-Amphibole paragneiss
		3: Quartz-rich-paragneiss; sillimanite, sericite, and quartz schist; -impure quartzite
		2: Pillowed-metavolcanic amphibolites
		1: Oligoclase gneiss

6.3 Local Geology

At the property level, the geology consists of a volcano-sedimentary assemblage metamorphosed to the amphibolite level (Figure 6-3). The volcanic rocks mostly comprise basalt-andesite rocks and gabbro formation. The primary textures are not identifiable, and no geochemistry data enables to correctly identify the rock types. The sedimentary units range from meta-conglomerates with elongated clasts to fine grained sedimentary units.

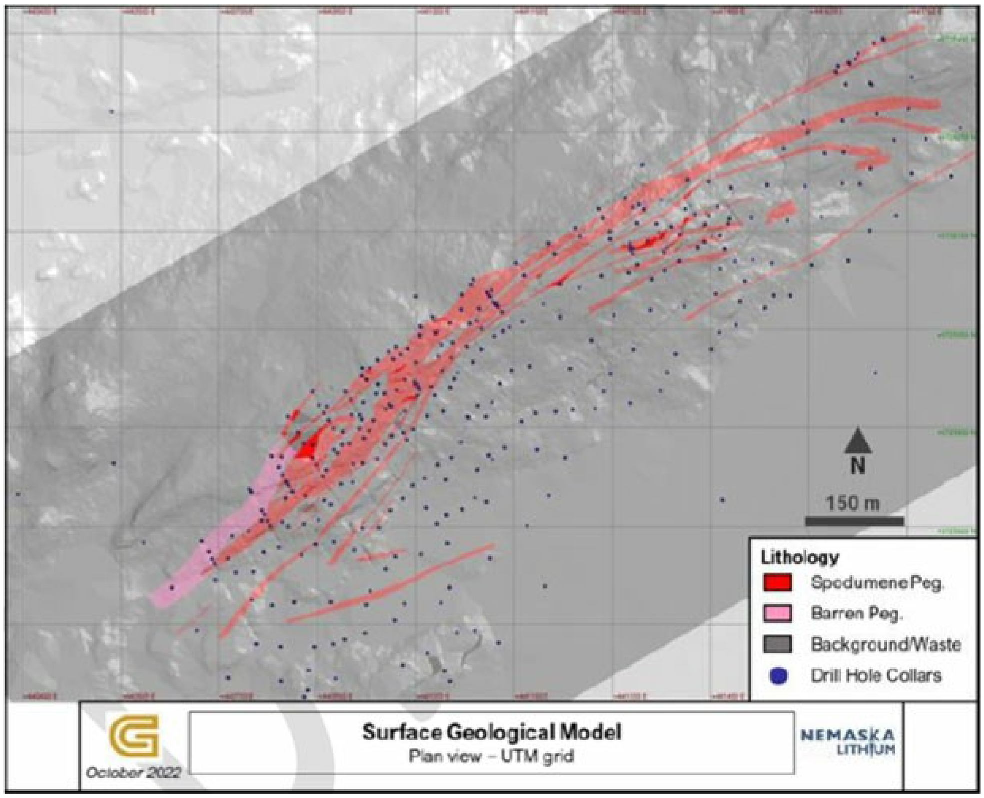
The volcano-sedimentary sequence is intruded by different bodies of granites and pegmatites with varying composition and probably age (no age constraints are available on the local intrusive bodies). The granites vary in texture and composition, from white and pink granite fine-grained granites to grey hornblende-oligoclase granite with phenocryst of pink microcline.

Figure 6-3 Map of the Property Geology



The pegmatite bodies form a swarm of interconnecting dykes and plug shaped intrusions. The Whabouchi dyke swarm comprises the Main dyke (Figure 6-4) and a series of subsidiary dykes, like the Doris zone. The dykes vary in orientation from N055° to N070° and are steeply dipping towards the southeast (Main zone) and northeast (Doris). In cross sections, some of the dykes have different dip orientation and potentially connect to other dykes at depth. The corridor occupied by this dyke swarms as been recognized on a strike length of 1,340 m with a width ranging from 60 m to 330 m.

Figure 6-4 Local Map of the Pegmatite Dykes Interception with Drill Holes



6.4 Mineralization

The regional prospecting done in the region over the years highlighted a potential for precious and base metal deposits. Cu, Zn, and Au lithogeochemical anomalies are found in the region, which is consistent with the volcano-sedimentary setting of this region.

The mineralization of economic interest at the Whabouchi site is found in spodumene-bearing rare metal bearing pegmatite dyke complexes. Spodumene is a lithium-bearing mineral, which contains 8% Li_2O when pure. Spodumene also contains minor amounts of niobium and tantalum. Assays for spodumene normally range between 7.6% and 8.0% Li_2O depending on the degree of replacement by Na_2O. Typically, the Whabouchi pegmatite sampled from drill core averages 1.42% Li_2O with values up to 5.19% Li_2O. Recent mineralogical assessment shows minor amount of other Li-bearing minerals, such as petalite, muscovite, ferrisicklerite, cookeite, and holmquistite.

Rare metal bearing pegmatites are normally found in moderately metamorphosed terranes near vast granitic plutons: a possible parental source for the pegmatitic magmas. Pegmatites are associated with granitic intrusions and are generally zoned around these intrusive centers. Pegmatites tend to be more enriched in volatile elements further away from the intrusive centers. Pegmatites are thought to be derived from primary crystallization of highly differentiated volatile enriched granitic magmas. The host rocks of the intrusion also play a significant role in the final composition of the pegmatites due to the incorporation of host rock in the magma during the intrusive process.

Pegmatite complexes can vary from a few meters to a hundred meters in length with the same variation in widths. Typically, pegmatite intrusions are zoned and show the following structures from the exterior to the interior:

- The rim zone is usually very narrow and fine-grained,
- The wall zone is normally composed of quartz, feldspar and muscovite and marks the development of larger crystals typical of pegmatites,
- The intermediate zone, when present, comprises a more complex mineralogy with varying amounts of economic minerals such as micas, beryl (Be), spodumene (Li), amblygonite (Li), lepidolite (Li-Rb), colombite-tantalite (Nb-Ta) and cassiterite (Sn). Crystals in this zone can extend up to metric lengths, and
- The central zone is mainly composed of quartz in pods or automorph crystals.

Two distinct phases are observed in the Whabouchi pegmatites: a spodumene-bearing phase comprising most of the pegmatite material and a lesser, white to pink barren quartz-feldspar pegmatite. The lithium mineralization occurs mainly in medium to large spodumene crystals (up to 30 cm in size) but petalite also occurs, averaging approximately 2.3% in the deposit (petalite contains approximately 4.5% Li_2O). Muscovite also contains minor lithium and averages less than 2% in the deposit. Petalite and muscovite are not recoverable by the mineral processing method discussed in the present Report.

7 EXPLORATION

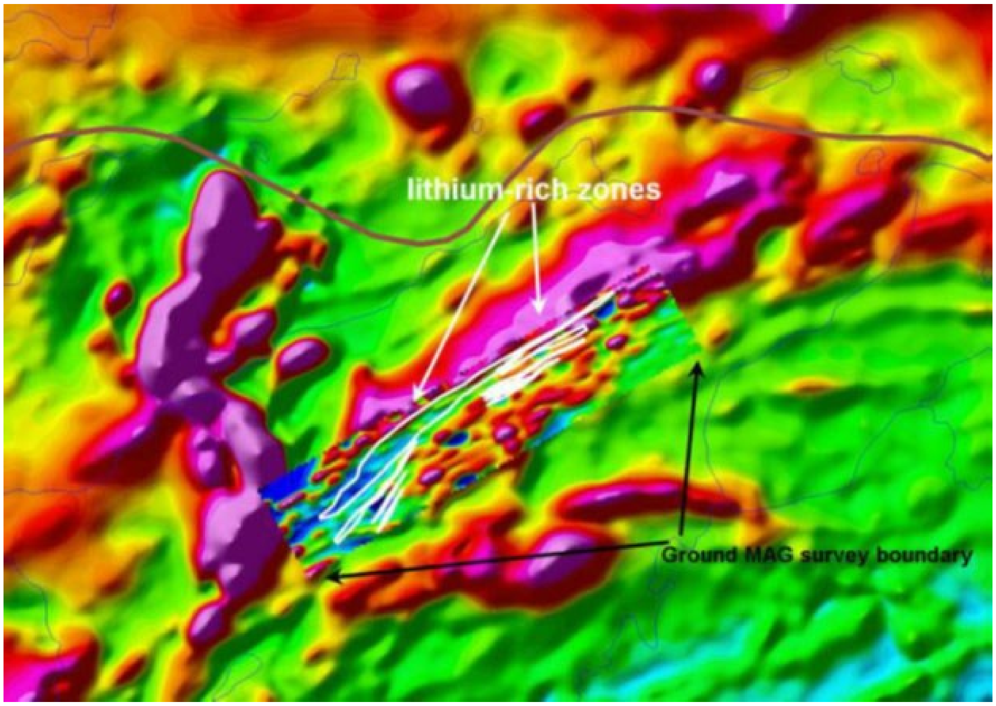
7.1 Exploration

Exploration Nemaska Inc., at that time and as part of the Qualifying NI 43-101 Technical Report dated July 14, 2010, initiated its exploration work on the Property during the fall of 2009. During the site visit, several outcrops of spodumene-bearing pegmatite were observed and nine samples were collected and analyzed for Li_2O. The highest and lowest results obtained during the site visit are the grab sample #946511, with a value of 6.3% Li_2O, and grab sample #946508 at 1.18% Li_2O (Théberge, 2009). Following that and during the fall 2009 exploration program, mechanical stripping successfully exposed the spodumene-bearing pegmatites in 16 trenches spaced between 50 and 100 m apart and covering 1,000 m in strike length. From these trenches, 37 channels were cut and a total of 295 samples were collected for analysis. In addition to the trenching work, eight diamond drill holes were completed; all successful drill holes have intersected pegmatites zones.

In 2010 and in addition to drilling, 14 line-km of ground magnetic surveying covering the main mineralized occurrence and 670 line-km of helicopter-borne magnetic surveying covering the Property were completed (Figure 7-1). Later in May 2010, 2,780 m of mechanical stripping of the south contact of the Main Zone was completed and allowed the mapping of the surface geology. A 1.2 km access road from the Route du Nord main road was constructed in 2010.

In May 2011, a 50-tonne bulk sample was collected at surface for metallurgical testing purposes.

Figure 7-1 Ground Magnetic and Aeromagnetic Image with Lithium-rich Zones



Source: Geophysics GPR International Inc., 2010 (MB10849)

7.2 Drilling

As part of a pulp resample program held in 2021, several analyses were conducted to better understand the variability of the deposit. None of the chemical assays were integrated in the mineral resource database; only the specific gravity measurements were used to update the density model (refer to herein Section 8.11).

A total of 277 diamond drill holes were completed by NLI to define the mineral deposit, for exploration, as well as for geotechnical and metallurgical tests. In addition to the drilling, extensive mechanical stripping at surface permitted the completion of 108 channels. Table 7-1 and Table 7-2 summarize the drilling and channel sampling completed by NLI to define the mineralized pegmatite intrusion and for exploration Northwest of the deposit.

Figure 7-2 illustrates the 2009 to 2018 Diamond Drill Holes as well as the Channel Sampling Location.

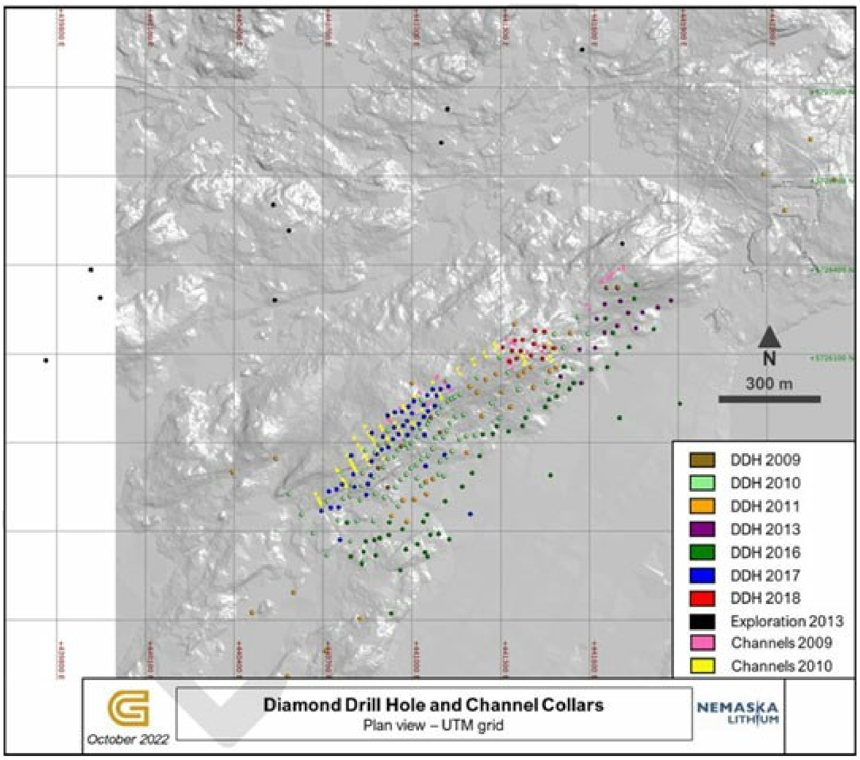
Table 7-1 Drilling Completed by NLI at Whabouchi

Year	Count	Meters Drilled	Number of Lithium Assays
2009	8	999	456
2010	82	15,670	6,088
2011	41	9,264	1,869
2013	14	1,815	350
2013 (exploration)	10	1,308	150
2016	51	17,424	4,038
2017	48	4,361	1,819
2018	14	2,099	818
2018, 2021 (geotech.)	9	1,610	0
Total	**277**	**54,550**	**15,588**

Table 7-2 Channel Sampling done by NLI at Whabouchi

Year	Channels	Total Samples
2009	37	295
2010	71	649
Total	**108**	**944**

Figure 7-2 Plan View of Diamond Drill Hole and Channel Collars



The diamond drilling completed by NLI on the Whabouchi property was done exclusively with NQ and HQ drill size. HQ size was used to collect material for metallurgical testing in 2011. The samples collected for analysis represent approximately 30% of the drill core material and 98% of the channel material. The drill holes are generally spaced 25 m to 50 m apart with azimuth ranging between N312° and N340°, with an average direction of N330°. The dips range from 43° to 75° with most of drill holes drilled at 45° or 50° dips. The longest drill hole reaches a length of 640 m down hole, while the deepest hole reaches 510 m of vertical depth. The mineralized drill intersection ranges from near true thickness to 70% of the true thickness of the dykes.

The geometry of spodumene-bearing pegmatites is defined as a series of stacked dyke-shaped intrusions which include a thicker principal intrusion. Some pegmatite contains local rafts or xenoliths of the host rock which can be a few meters thick and hundreds of meters in length.

Based on the information gathered from drilling, the pegmatite intrusion is more than 1,300 meters in length and can be up to 90 meters thick. The intrusions are generally oriented N050° with dips varying from the southeast to the northwest at an angle ranging between 70° and 85° and are reaching depths of up to 470 meters below surface. Please refer to Section 11 for the interpretation of the drill results.

Figure 7-2 to Figure 7-5 show the diamond drill holes in plan view, on longitudinal sections and on representative cross sections. Holes are colored by year of drilling and channel locations can be seen in plan view.

7.2.1 2009 to 2013 Diamond Drilling and Channels

During the fall 2009 exploration program, 37 channels were cut from mechanical stripping from which 295 samples were collected for lithium analysis. Eight (8) diamond drill holes were completed, including one hole abandoned for technical reasons (WHA-09-001 redrilled as WHA-09-001A). All successful drill holes have intersected pegmatite zones.

Following the successful drilling and channel campaign results from 2009, a second exploration program was conducted from January to April 2010. During that program, 59 drill holes, totaling 11,600 m were completed. In May of the same year, NLI completed 2,780 m of mechanical stripping of the south contact of the main mineralized zone. The trenching allowed to cut 71 channels and to collect 649 samples for lithium analysis. Later in 2010, an additional 23 drill holes were completed for a total of 4,070 m.

In 2011, 41 diamond drill holes were completed, which included 26 holes for infill drilling, three for metallurgical testwork. A total of 9,264 m was drilling that year.

In 2013, 14 drill holes totaling 1,815 m were added to better define the mineralization towards the Eastern boundary and to increase the confidence of the 2011 in-pit mineral resources. NLI also completed 10 diamond drill holes, for a total of 1,308 m, as exploration targeting spodumene-bearing pegmatites approximately 750 m Northwest of the Whabouchi deposit.

7.2.2 2016 Diamond Drilling

In 2016, 51 diamond drill holes were completed. The main objectives of this drilling campaign were to:

- Convert the in-pit inferred resources to indicated resources;
- Increase the confidence level of mineral resources from 0 m to 200 m depth; and
- Extend the mineral potential at depth.

A total of 17,424 m of drilling was completed and 4,038 samples were sent for lithium analysis. A new zone named Doris was discovered to the Southeast of the known Whabouchi deposit. The drilling campaign was conducted by SGS Canada Inc. under the supervision of Jean-Philippe Paiement, P.Geo., M.Sc. The drilling contractor retained for that campaign was Forage Rouillier, a division of Groupe Rouillier (Forage Rouillier). The drilling took place from July 7 to September 16, 2016.

7.2.3 2017 Diamond Drilling

In 2017, NLI commissioned ASDR and its representative Louis Caron, P.Geo., to oversee a drilling campaign on 48 drill holes totaling 4,361 m on the Whabouchi Property. This campaign aimed at verifying the extension of pegmatite dykes from the Doris Zone and to better define the geological continuity and lithium content in the Main Zone, targeted to be mined during the first five years of mining operations. Drilling was carried by Forage Rouillier between April 4 and June 6, 2017. A total of 1,819 samples were sent for lithium analysis.

7.2.4 2018 Diamond Drilling

In 2018, NLI team geologists supervised a drilling campaign consisting of 14 drill holes totaling 2,099 m on the eastern area of what was classified as measured resources between sections 900E and 1100E. This program aimed to verify the extension of mineralization and to better define the geological continuity and lithium content of the Main Zone, targeted to be mined during the first five years of mining operations. An additional six (6) oriented geotechnical holes were added, totalling 960 m.

The drilling campaign was done under the supervision of Patrick Laforest, P. Geo. and Clémence Maltais-Hardy, Geologists-in-Training, of NLI. Drilling was surveyed by the land-surveying firm MYS. Drill holes of more than 100 m were surveyed by the REFLEX multishot method and those of less than 100 m by single-shot REFLEX method. REFLEX results were corrected (-15°) to the REFLEX readings for correction to the geographic north (UTM NAD83). Core drilling was planned by SGS and carried out by Forage Rouillier between September 13 and October 14, 2018.

7.2.5 2021 Diamond Drilling

In 2021, NLI conducted a program consisting of three (3) geomechanical drill holes totalling 650 m. This program aimed at gaining additional information and measurements for pit slope design.

7.2.6 Conclusion

The author completed a verification of the drill hole and analytical data (Sections 8 and 9). The author considers that there are no known drilling, sampling or recovery factors that could materially impact the accuracy, reliability, and integrity of the results. A series of three (3) historical drill holes could not be validated with assay certificates nor with geological logs and therefore, they were not considered in the current Mineral Resource Estimate.

The author completed verification programs of the Project's analytical data and considers that there are no known drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results. The data from the few historical drill holes reported on the Project could not be validated and were not considered as part of the current Mineral Resource Estimates.

Figure 7-3 Longitudinal View of Diamond Drilling

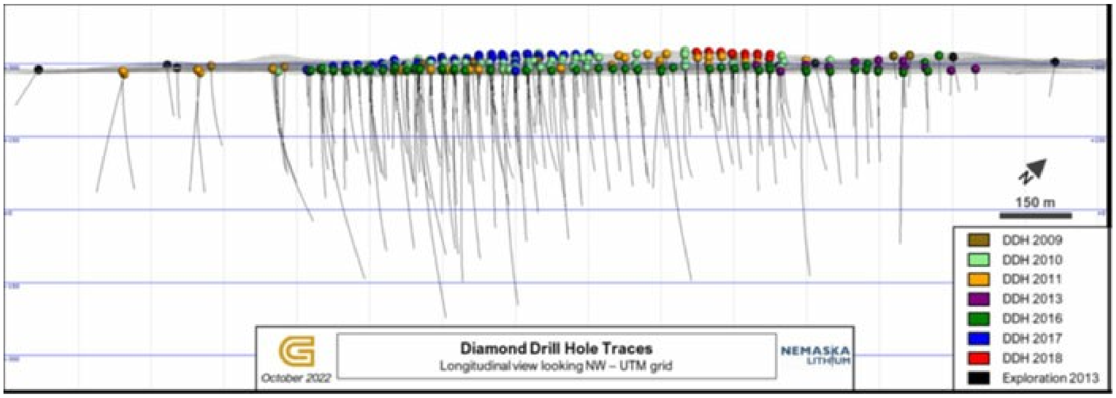


Figure 7-4 Section 375E Showing Diamond Drill Holes, Channels and Mineralized Envelopes – Looking Southwest

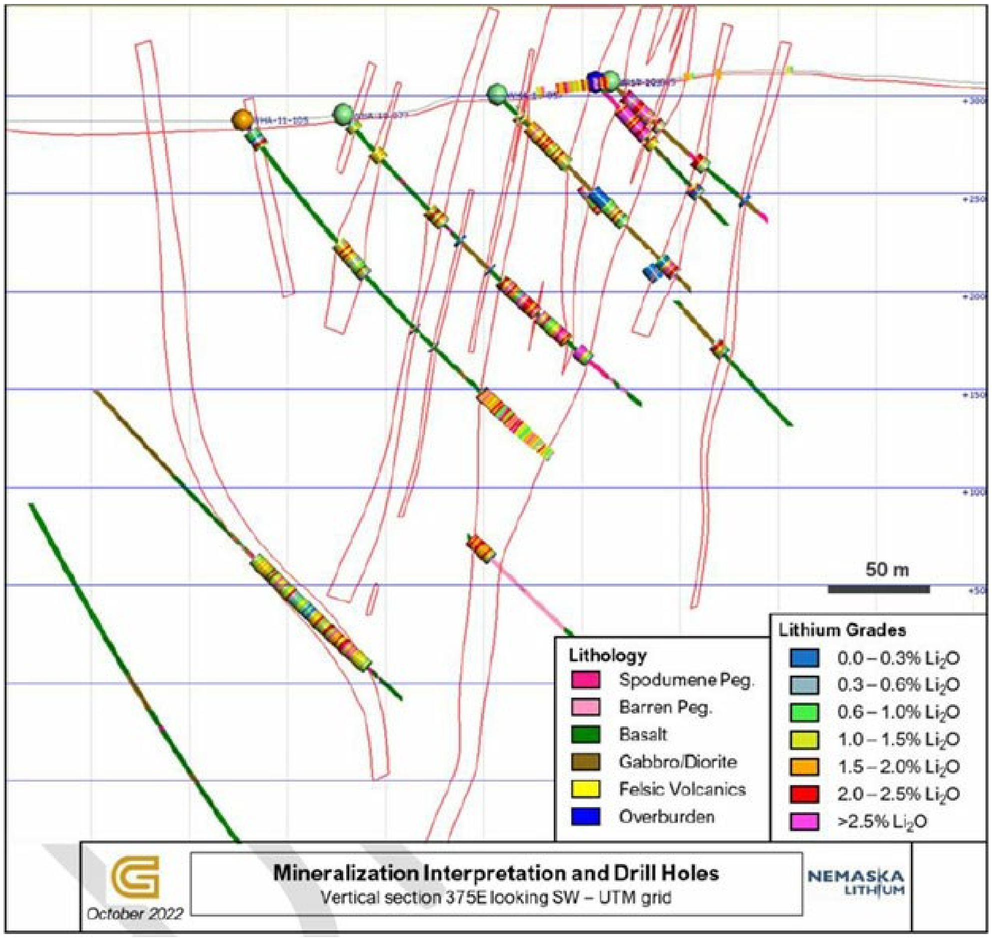
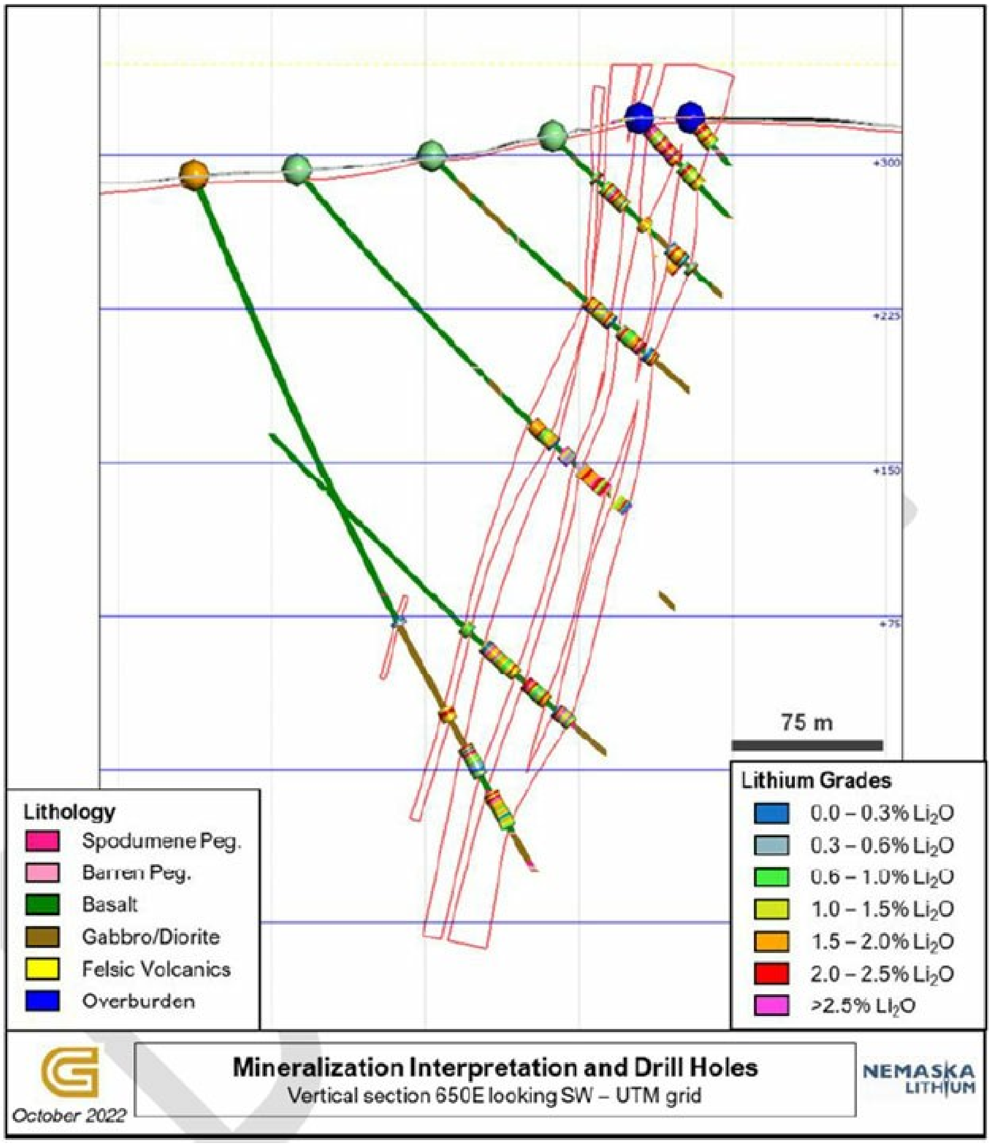


Figure 7-5 Section 650E Showing Diamond Drill Holes and Mineralized Envelopes – Looking Southwest



8 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The graphs and tables present an overview of all QA/QC data available at the effective date of this Report. Information presented herein was supplied by NLI and is based on independent verification programs conducted by SGS on November 27, 2013 and during the drilling programs of summer 2016. The drilling campaign and QA/QC results from 2017 and 2018 were also verified by SGS.

8.1 Sample Procedure and Sample Security

The Whabouchi Project is located less than 12 km east of the Hydro-Québec Nemiscau Camp. The evaluation of the geological setting and mineralization on the Property is based on observations and sampling from surface (geological mapping, grab and channel sampling) and diamond drilling. The channel and drill core logging and sampling were conducted at the Property or the nearby Project facilities. All remaining drill core is stored at the Property site in covered metal core racks.

All channel samples and drill core handling were undertaken on-site with logging and sampling procedures conducted by employees and contractors of NLI. The observations on lithology, structure, mineralization, sample number, and location were noted by the geologists and technicians on hardcopy and then recorded in a Microsoft Access digital database. Backups of the database are stored on external hard drives for security purposes.

Channel samples were collected from two (2) diamond saw cuts, typically 4 cm in width and 4 cm in depth. Each sample is generally one (1) m long and broken directly from the outcrop, identified, and numbered then placed in a new plastic bag.

Drill core of NQ and HQ size was placed in wooden core boxes and delivered twice daily by the drill contractor to the core logging facilities at the Nemiscau Camp. The drill core was first aligned and measured by a technician for core recovery and RQD measurements. After a summary review, the core was logged and sampling intervals were defined by a geologist. Before sampling, the core was photographed using a digital camera and the core boxes were properly identified with aluminum tags (box number, Hole-ID, “From” and “To”). Due to the hardness of the pegmatite units, the recovery of the channel samples and the drill core was generally very good, averaging more than 95%.

Sampling intervals were determined by the geologist, marked, and tagged based on observations of the lithology and mineralization. The typical sampling length is one (1) m but can vary depending on lithological contacts between the mineralized pegmatite and the host rock. In general, one (1) host rock sample was collected from the footwall and hanging wall of each pegmatite units.

The NQ drill core samples were split into two (2) halves with one (1) half placed in a new plastic bag along with the sample tag; the other half was placed back in the core box with the second sample tag for reference. The third sample tag was archived on site. The HQ size drill core, limited to a portion of the 2011 drilling program, was obtained for metallurgical purposes. The first half of the HQ drill core was selected for metallurgical testing. The second half was split in two (2) quarters, one (1) quarter placed in a new plastic bag along with the sample tag and the remaining quarter was placed back in the core box with the second sample tag for reference. The samples plastic bags were then catalogued and placed in rice bags or pails for shipping. The sample shipment forms were prepared on site with one (1) copy inserted with the shipment, one (1) copy sent by email to the Table Jamésienne de Concertation Minière (TJCM), and one (1) copy kept for reference.

The samples were transported on a regular basis by NLI’s employees or contractors by pickup truck directly to the TJCM facilities in Chibougamau, QC. At the TJCM laboratory, the sample shipment was verified, and a confirmation of shipment reception and content was emailed to NLI’s project manager.

In 2011 as part of an independent verification program, SGS Geological Services validated the exploration processes and core sampling procedures used by NLI. The author concluded that the drill core handling, logging, and sampling protocols were at conventional industry standards and conform to generally accepted best practices. The author considered that the sample quality is good, and the samples are generally representative. The current QP (Marc-Antoine Laporte of SGS Geological Services Inc.) is of the opinion that the sample procedures and sample security are suitable, and appropriate for the estimation of a Mineral Resource.

Coarse rejects and pulp samples are currently stored on-site in a secured facility, a dome located near the current infrastructure of the future concentrator. Coarse rejects and pulp samples are generally organised by laboratory batches in rice bags, stored in wooded boxes on pallets (Figure 8-1). Bags are generally well identified, and boxes have a number assigned to each of them. It was observed during the site visit that some wooden boxes are in poor condition. It is recommended that NLI reorganize the sample storage with proper identification on wooden boxes. All missing sample bags should be noted in a simple database, mapping all samples location per boxes. At the time of writing this report, NLI already started this reorganization and inventory project. Twenty-Five (25) wooden boxes have been delivered to the Project site and inventory is set to begin in early 2023.

Figure 8-1 Coarse Rejects and Pulp Samples Storage

8.2 Sample Preparation and Analysis

Sample preparation and analysis was performed by various laboratories throughout the years. Table 8-1 shows a summary of analyses done by year and Table 8-2 presents a summary of the main analysis methodology used for lithium (only the main analysis methods were compiled for simplicity). The following sub-section explains in detail the preparation and lithium analysis methodologies.

The 2009 and 2010 sample pulps were shipped for analysis to SGS Canada Inc. – Mineral Services (SGS Minerals) laboratory in Don Mills, ON. The remaining of 2010 sample pulps, as well as 2011 and 2013 sample pulps were sent for analysis to ALS Canada – Chemex Laboratory (ALS Chemex) in North Vancouver, BC and Val-d'Or, QC. The 2016, 2017 and 2018 samples were shipped to SGS Canada Inc. – Mineral Services laboratory in Quebec City, QC for preparation and to Lakefield, ON for analysis. The remaining drill core is stored at the Property site in covered metal core racks.

Table 8-1 Summary of Analyses by Laboratory and Year

Drilling Year	Laboratory		Sample Preparation
	SGS	ALS	
2009	827	0	TJCM
2010	5,565	1,096	TJCM
2011	0	1,869	TJCM
2013	0	500	TJCM
2016	4,038	0	SGS Quebec
2017	1,819	0	SGS Quebec
2018	818	0	SGS Val-d'Or
Total	**13,067**	**3,465**	**16,532**

Table 8-2 Analysis Method by Year

Lab. Code	GC ICP93A	GE ICP91A	ICM90A	ICP90Q	GO ICP41Q	Li-OG63
Laboratory	SGS					ALS
Dissolution	Na-Peroxide				4-acid	
2009	730	0	14	76	0	0
2010	0	0	4,410	1,041	0	1,092
2011	0	0	0	0	0	1,396
2013	0	0	0	0	0	496
2016	0	0	0	0	4,005	0
2017	200	0	0	0	1,618	0
2018	0	637	0	0	0	0
Total	**930**	**637**	**4,424**	**1,117**	**5,623**	**2,984**

Channel and drill core samples collected during the 2009, 2010, 2011, and 2013 exploration programs were transported directly by NLI personnel to the *TJCM* laboratory facilities in Chibougamau, QC for sample preparation. The submitted samples were pulverized at the *TJCM* laboratory to respect the specifications of the analytical protocol then shipped to SGS Minerals or ALS Chemex for analysis.

All samples received at TJCM were entered into the system and weighted prior to being processed. Drying was undertaken on samples with excess humidity. Sample material was crushed to 80-85% passing 2 mm using jaw crushers. Crushed material was split using a split riffle to obtain a 275-300 g sub-sample. Sub-samples were then pulverized using a 2-component ring mill (ring and puck mill) or a single component ring mill (flying disk mill) to 85-90% passing 200 mesh (75 µm). The balance of the crushed sample (reject) was placed in the original plastic bag. The pulverized samples were sent to SGS Minerals or ALS Chemex using Canada Post secured delivery services.

In 2016, samples were prepared and pulverized at SGS facilities in Quebec City, QC, following the same specification used by TJCM.

In 2018, samples were prepared and pulverized at SGS facilities in Val-d'Or, QC, following the same specification used by TJCM.

The majority of the 2009 and 2010 analyses were conducted at SGS Minerals Laboratory located in Don Mills, On. This laboratory is ISO/IEC 17025 accredited by the Standards Council of Canada. Two (2) types of analytical methods were used for the majority for the pulverized samples.

The first analytical method used by SGS Minerals is the 55-element analysis using sodium peroxide fusion followed by both Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES) and Inductively Coupled Plasma Mass Spectrometry (ICP-MS) finish (SGS code ICM90A). This method uses 10 g of the pulp material and returns different detection limits for each element, with a 10 ppm lower limit detection for Li. The ICM90A analytical method was conducted at the beginning of the 2009-2010 exploration program to verify the content of other elements in the mineralization.

The second method processed 20 g of pulp material and used the mineralization grade sodium peroxide fusion with ICP-OES finish methodology with a lower detection limit of 0.01% Li (SGS code ICP90Q). The ICP90Q analytical method was used at the beginning of the exploration program on samples analysed by ICM90A returning values greater than 0.30% Li. The ICP90Q method for Li was later used on a more systematic basis. Analytical results were sent electronically to NLI and results were compiled in an MS Excel spreadsheet by the project manager.

Drill core samples collected during the 2016, 2017 and 2018 exploration programs were analysed at SGS Minerals Laboratory located in Lakefield, ON. This laboratory is accredited by the Standards Council of Canada. The four-acid digestion with Inductively Coupled Plasma – Atomic Emission Spectrometry (ICP-AES; SGS code GO ICP41Q) methodology was used, verified by a sodium peroxide fusion AAS (SGS code GC AAS93B), for the majority of samples of the 2016 and 2017 exploration campaign. The 2018 exploration campaign used the peroxide fusion with an ICP-OES finish.

Analysis conducted by ALS for the 2010, 2011 and 2013 exploration campaigns used the mineralization grade lithium four-acid digestion with ICP-AES finish (ALS code Li-OG63). The Li-OG63 analytical method used 4 g of pulp material and returned a lower detection limit of 0.01% Li.

The 2018 pulp reanalysis was conducted in Actlabs in Ancaster, ON, using the mineralization grade lithium four-acid digestion with ICP-OES or ICP-MS finish (Actlabs code 8-Li).

8.3 Quality Assurance and Quality Control Procedure

NLI developed an internal Quality Assurance and Quality Control (QA/QC) protocol consisting of the insertion of analytical standards, blanks, and core duplicates on a systematic basis with the samples shipped to the analytical laboratories. In 2010, NLI also sent pulps from a selection of mineralized intersections to ALS Chemex for reanalysis. No pulp reanalysis was performed by NLI in 2011 and 2013. The author did not visit SGS Minerals and the ALS Chemex facilities or conduct an audit of the laboratories.

8.4 Analytical Standard Reference Material

Table 8-3 summarizes the analytical standards used throughout the various exploration campaigns. No standard reference materials were used during the 2009 exploration campaign.

Table 8-3 Summary of Analytical Standard Reference Materials

Drilling Year	Standards	Note
2009	No standards used	
2010	Li-LG, Li-HG	Starting at WHA-10-012, internal uncertified standards
2011	Li-LG, Li-HG	Internal uncertified standards
2013	Li-HG	High grade only
2016	NCS DC 83303, NCS DC 83314	Certified standards
2017	NCS DC 83303, NCS DC 83314	Certified standards
2018	OREAS-148, OREAS-149	Certified standards

8.4.1 Li-LG and Li-HG Reference Material

Expected values of these standards (Li-LG and Li-HG) were determined by repetitive analysis in two separate laboratories: six times for each standard at SGS Minerals (Don Mills, ON facility) and five times for each standard at ALS Chemex (North Vancouver, BC facility). Both facilities are accredited ISO/IEC 17025 laboratories. The analytical protocol used at SGS Minerals is the mineralization grade sodium peroxide fusion with ICP-OES finish (SGS code ICP90Q). The analytical protocol used at ALS Chemex is the mineralization grade lithium four-acid digestion with ICP-AES finish (ALS code Li-OG63).

For the Li-LG standard, the analytical results returned from SGS Minerals for the six (6) samples averaged 0.46% Li versus an average of 0.45% Li for the five (5) samples submitted to ALS Chemex. For the Li-HG standards, the average of the six (6) samples returned 0.72% Li from SGS Minerals versus an average of 0.71% Li for the five (5) samples processed at ALS Chemex.

Each laboratory showed relatively consistent analytical results from one sample to another for each standard analysed. The averages for each standard also showed a good correlation between SGS Minerals and ALS Chemex. The amount of data is not statistically significant to calculate standard deviation (SD) parameters which can be used to determine the success or failure of standards analysis.

The insertion of the analytical standards (Li-LG and Li-HG) started with the drill hole WHA10-012. One standard was inserted in the sample stream at a every 25 regular samples, alternating between Li-LG and Li-HG. A total of 169 Li-LG and 169 Li-HG standards were analysed during the 2010, 2011 and 2013 exploration campaigns, representing approximately 4% of the core samples analysed. To determine the QC warning (±2x SD) and QC failure ((±3x SD) intervals for the Li-LG and Li-HG, the SD values returned for the complete analytical results (169 of each standard) are considered. SD were calculated separately by analytical method (peroxide fusion and 4-acid digestion). A SD of 0.01 and 0.02 was used to determine warning and failure intervals for the Li-LG and Li-HG standards respectively.

Table 8-4 shows the summary of each standard, while Figure 8-2 and Figure 8-3 illustrate the variation of standards over time. From the 169 Li-LG standards analysed, six (6) fall outside the QC Warning interval and one (1) falls outside the QC Failure interval. This single failure is considered acceptable since it represents less than 1% of the assayed Li-LG standards. From the 169 Li-HG standards analyzed, 14 fall outside the QC warning interval and four (4) fall outside the QC Failure interval. The failures are considered acceptable since they are mainly linked to the peroxide fusion digestion method (the reference value is closer to the average of 4-acid reference samples, known for yielding lower% Li values. These results are judged acceptable.

Table 8-4 Summary of Li-LG and Li-HG Standards

Standard	Count	Reference Li (%)		Observed Li (%)	Warnings	Failures
		Average	SD	Average		
Li-LG	169	0.46	0.017	0.47	6	1
Li-HG	169	0.71	0.027	0.73	14	4

Figure 8-2 Internal Reference Material Li-LG

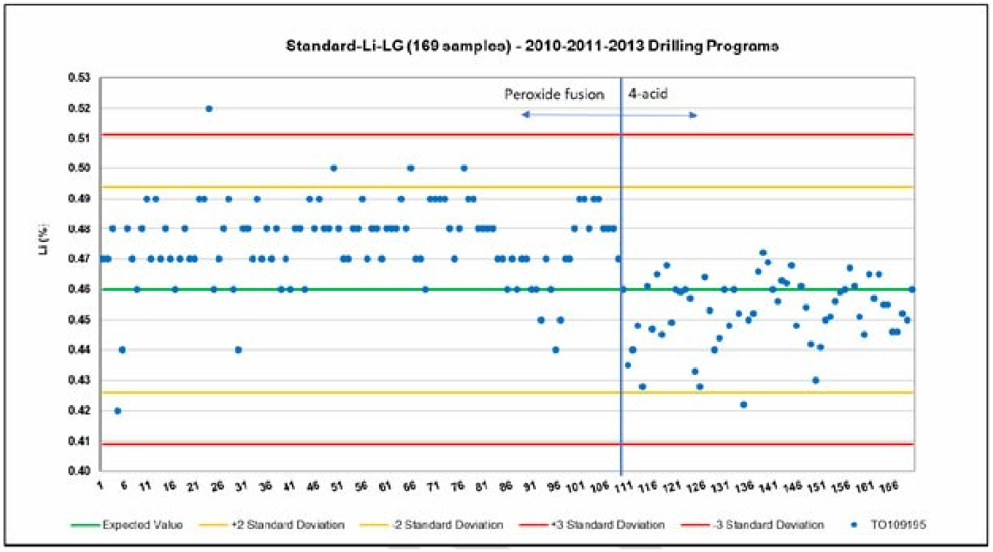
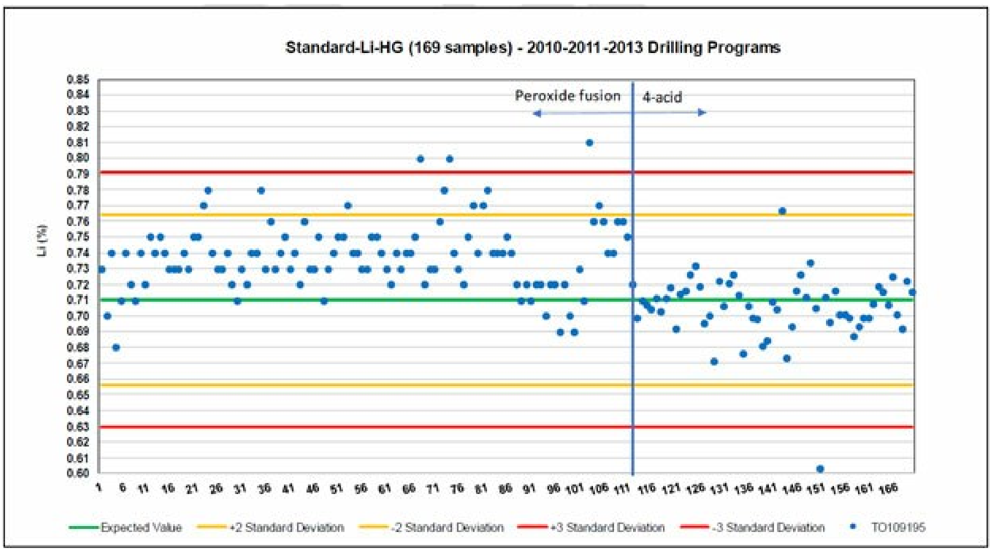


Figure 8-3 Internal Reference Material Li-HG



8.4.2 NCS DC 86303 and DC 86314 Certified Reference Material

During the 2016 and 2017 exploration campaigns, two new standards were used by NLI for the internal QA/QC program: NCS DC 86303, a low-grade standard (0.46% Li_2O) and NCS DC 86314, a high-grade standard (3.89% Li_2O). Both standards were made and certified by the China National Analysis Center for Iron and Steel in Beijing, China. This facility is an accredited ISO 9001 laboratory.

Analytical protocol used by the Chinese Center for the low-grade standard DC 86303 is Flame Atomic Absorption Spectrometry – Flame Photometry (AAS-FP). The analytical protocol used for the high-grade standard DC 86314 is Atomic Absorption Spectrometry (AAS) with Inductively Coupled Plasma – Atomic Emission Spectrometry (ICP-AES) and Inductively Coupled Plasma – Mass Spectrometry (ICP-MS).

Certified Li_2O values for NCS DC 86303 is 0.46% with a SD of 0.01 and for NCS DC 86314 is 3.89% with a SD of 0.14. Certified values are calculated according to the analytical results of 8 or 9 independent laboratories. NLI used the same insertion protocol in 2016 and 2017 of one (1) standard every 25 regular samples, alternating between DC 86303 and DC 86314.

Table 8-5 reports the statistics of the DC 86303 and DC 86314 Certified Reference Material (CRM). Figure 8-4 and Figure 8-5 show plots of the variation of standards over time. From the 76 DC 86303 standards analysed, 43 standards fall outside the QC Warning interval and fifteen (15) standards fall outside the QC Failure interval. The results also show a systematic 5% bias in the assays, likely due to the 4-acid digestion method used in these. From the 79 DC 86314 standards analysed, thirteen (13) standards fall outside the QC Warning interval and three (3) standards fall outside the QC Failure interval. One failure is likely an insertion error. The analytical method bias is also observed is this dataset (systematic 5% bias). These results are sub-optimal, but considered acceptable considering that the bias is conservative.

Table 8-5 Summary of NCS DC 86303 and DC 86314

Standard	Count	Reference Li_2O (%)		Observed Li_2O (%)		Warnings	Failures
		Average	SD	Average	SD		
DC 86303	76	0.46	0.01	0.44	0.024	43	15
DC 86314	79	3.89	0.14	3.73	0.17	13	3

Figure 8-4 Certified Reference Material NCS DC 86303

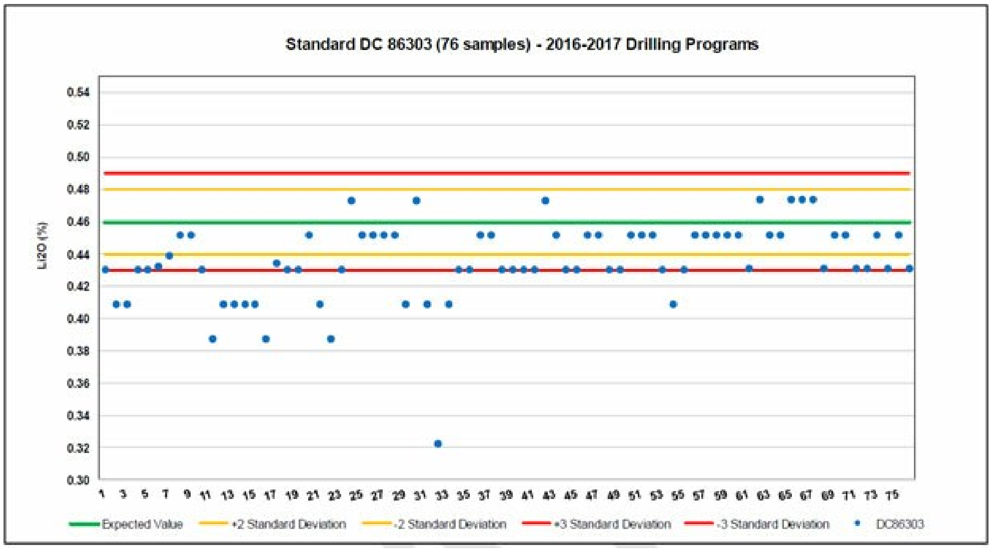
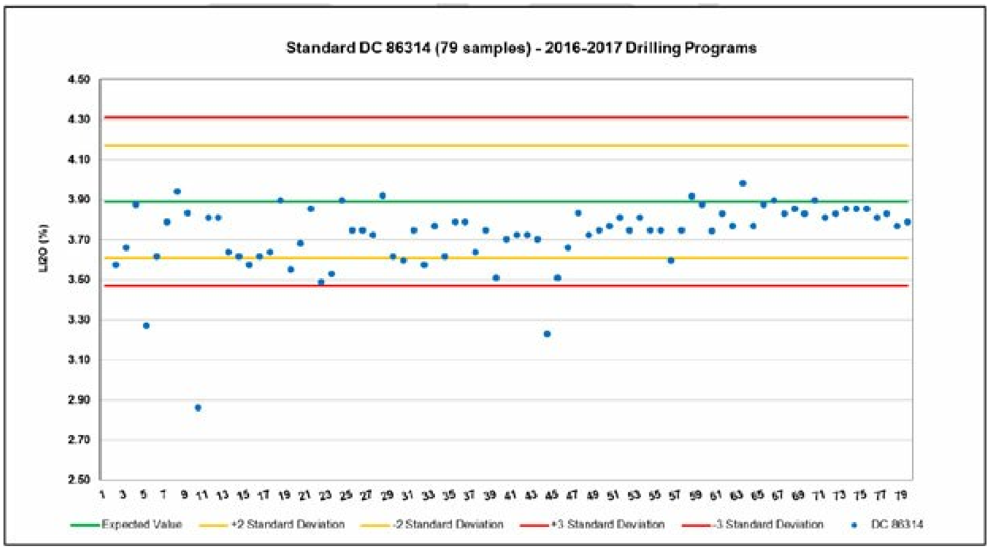


Figure 8-5 Certified Reference Material NCS DC 86314



8.4.3 OREAS-148 and OREAS-149 Certified Reference Material

During the 2018 drilling campaign, two (2) new standards were used by NLI for the internal QA/QC program: OREAS-148, a medium-grade standard (1.03% Li_2O) and OREAS-149, a high-grade standard (2.21% Li_2O). The CRMs have been prepared from spodumene LiAl (Si5O5) rich pegmatite ore blended with granodiorite and with minor additions of Sn-oxide ore and Nb concentrate. Both standards were made and certified by OREAS based in Australia. Twenty-two (22) commercial analytical laboratories participated in the program to certify these standards. The following methods were employed: 4-acid digestion for full ICP-OES and ICP-MS elemental suites (up to 22 laboratories depending on the element; one laboratory used an AAS finish on Li only), peroxide fusion for full ICP-OES and ICP-MS elemental suites (up to 21 laboratories depending on the element), lithium borate fusion with XRF finish for whole rock package including Nb and Ta (up to 22 laboratories depending on the element).

Certified Li_2O values for OREAS-148 is 1.03% with a SD of 0.02 and for OREAS-149 is 2.21% with a SD of 0.06. NLI insertion protocol for 2018 is one (1) standard every 20 regular samples, alternating between OREAS-148 and OREAS-149.

Table 8-6 reports the statistics of the OREAS-148 and OREAS-149 CRM. Figure 8-6 and Figure 8-7 show plots of the variation of standards over time. From the 22 OREAS-148 CRM analysed, nine (9) standards fall outside the QC Warning interval and two (2) fall outside the QC Failure interval. A slight systematic bias is observed for OREAS-148 of approximately 2%, unlikely caused by the digestion method. Results are acceptable considering that the expected bias is slightly conservative and that most results are within the failure interval. Results for OREAS-149 do not show a systematic bias and are judged to be well representative of the CRM expected value.

Table 8-6 Summary of OREAS-148 and OREAS-149

Standard	Count	Reference Li_2O (%)		Observed Li_2O (%)		Warnings	Failures
		Average	SD	Average	SD		
OREAS-148	22	1.03	0.02	1.01	0.04	9	2
OREAS-149	23	2.21	0.06	2.22	0.09	3	0

Figure 8-6 Certified Reference Material OREAS-148

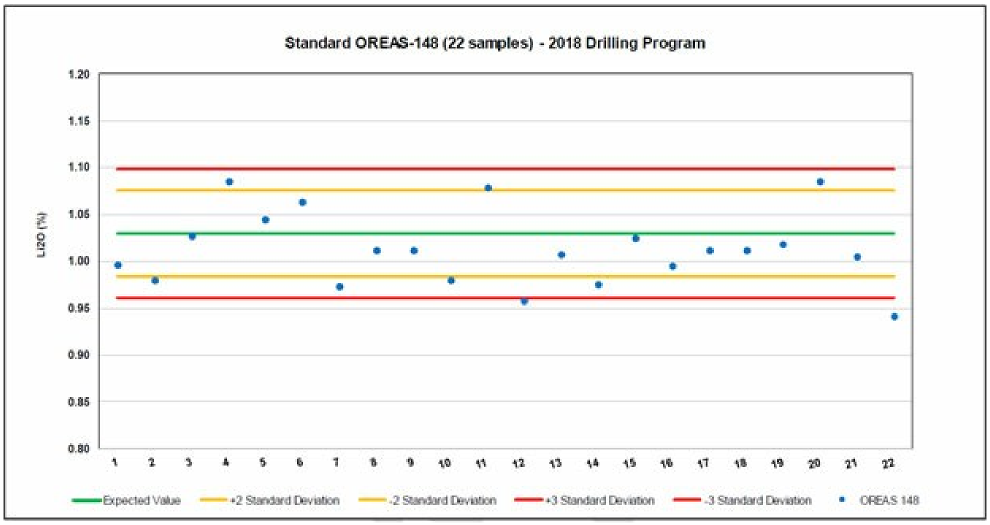
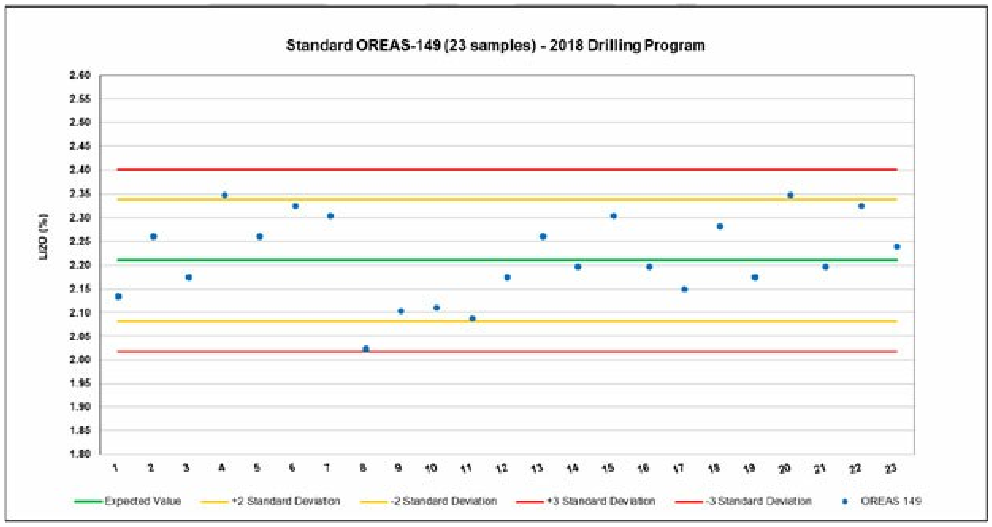


Figure 8-7 Certified Reference Material OREAS-149



8.5 Analytical Blanks

NLI implemented the insertion of analytical blanks in the sample series as part of their internal QA/QC protocol. The blank samples, which are made of coarse silica lumps, are inserted at every 20 samples in the sample series, at the beginning of the sample preparation procedure by TJCM before shipping. The analytical blanks used for 2010 were made of pre-pulverized silica instead of coarse lumps. The 2010 procedure was not considered adequate since the analytical blanks were inserted by TJCM after the sample preparation procedure, and therefore did not test the potential for contamination during sample preparation. The QA/QC procedure was updated by NLI in 2011 and is now considered adequate.

A total of 719 analytical blanks were analysed during the 2009, 2010, 2011, 2013, 2016, 2017, and 2018 drilling programs. From the 719 blanks analyzed, 98.2% of them returned a value less than five (5) times the detection limit and 98.9% of them returned a value less than ten times the detection limit. The detection limit is dependent on the analysis technique used and varies from 0.001% Li, 0.005% Li and 0.01% Li. A summary of samples exceeding the detection limit is presented in Table 8-7.

Table 8-7 Summary of Blanks Analysed

Drill Program/Analysis Method	Total Blanks	Above 5x DT	Above 10x DT
2010-2013 ICP90Q	292	0	0
2010-2013 ICM90A	58	2	0
2010-2013 OG63	145	2	0
2016-2017 ICP41Q	163	8	7
2018 ICP91A	61	1	1
Total	**719**	**13**	**8**
% Failure		1.8%	1.1%

Figure 8-8 to Figure 8-12 below show a time plot of blank analysis, separated by detection limit and drilling campaigns. Some of failures in the 2016-2017 campaigns appear to be an insertion or labelling error (i.e., inserting standards instead of blanks).

Figure 8-8 Blank Analysis from the 2010-2011-2013 Campaigns – ICP90Q

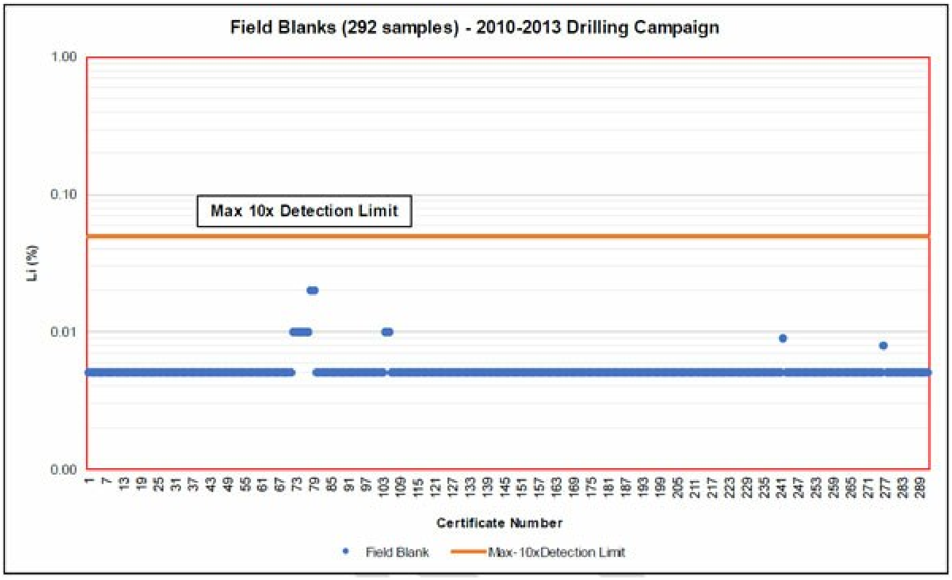


Figure 8-9 Blank Analysis from the 2010-2011-2013 Campaigns – ICM90A

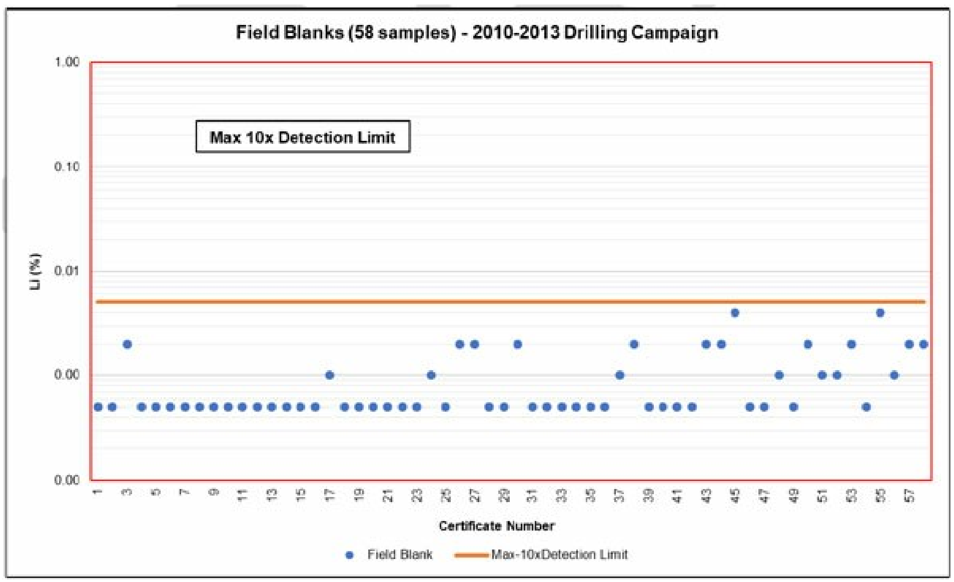


Figure 8-10 Blank Analysis from the 2010-2011-2013 Campaigns – OG63

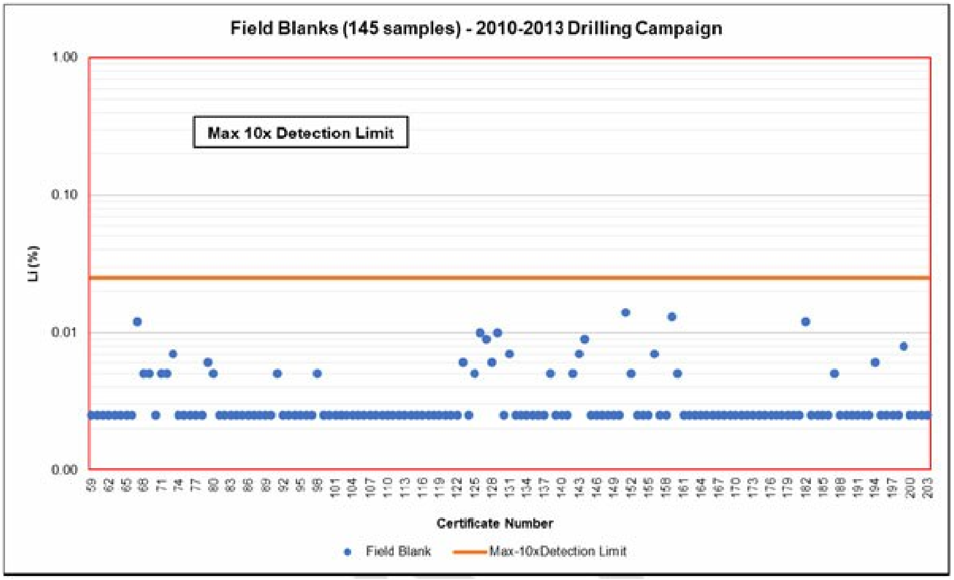


Figure 8-11 Blank Analysis from the 2016-2017 Campaigns – ICP41Q

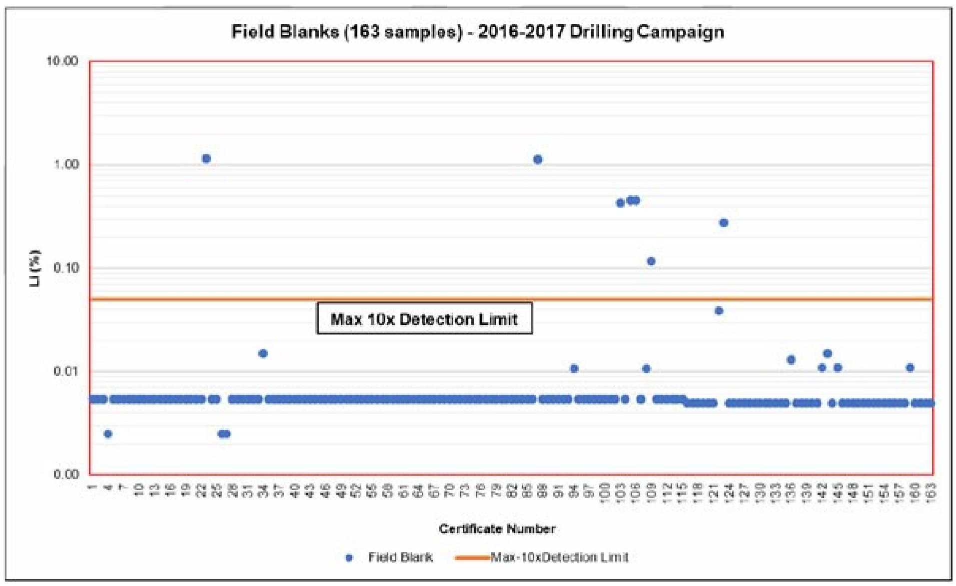
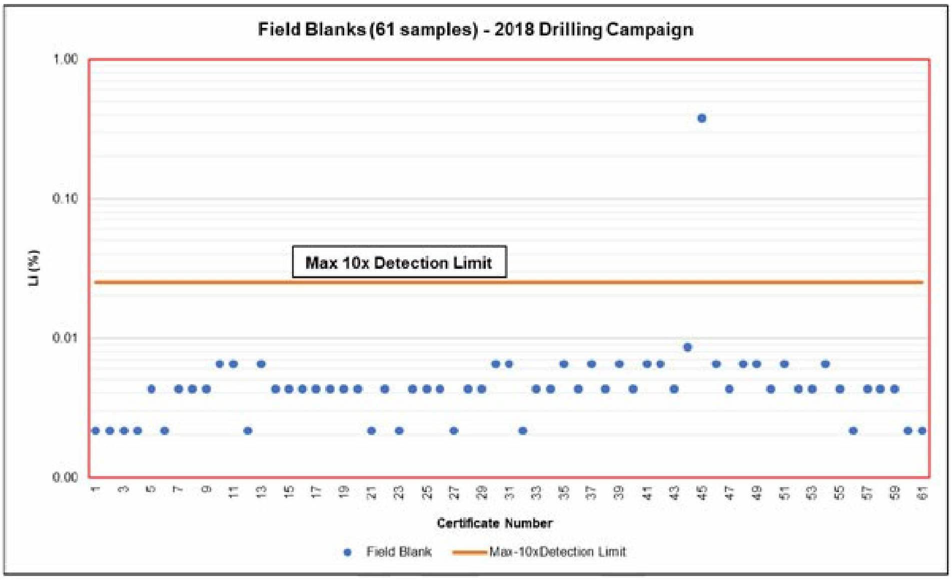


Figure 8-12 Blank Analysis from the 2018 Campaign – ICP91A



8.6 Core Duplicates

Field duplicates were inserted at every 20 samples in the sample stream as part of NLI internal QA/QC protocol. The sample duplicates correspond to a quarter NQ or HQ core from the sample left for reference (half core), or a representative channel sample from the secondary channel cut parallel to the main channel. Figure 8-13 shows a correlation plot for the field core duplicates. To gain further confidence in the reproducibility of the data, the HARD (half absolute relative difference) index plot (Figure 8-14) shows that approximately 90% of the data have a half absolute relative difference below 10%. Sign test for the duplicates does not show any bias.

No core duplicate was done during the 2017 drilling campaign.

Figure 8-13 Field Duplicates (Quarter Core)

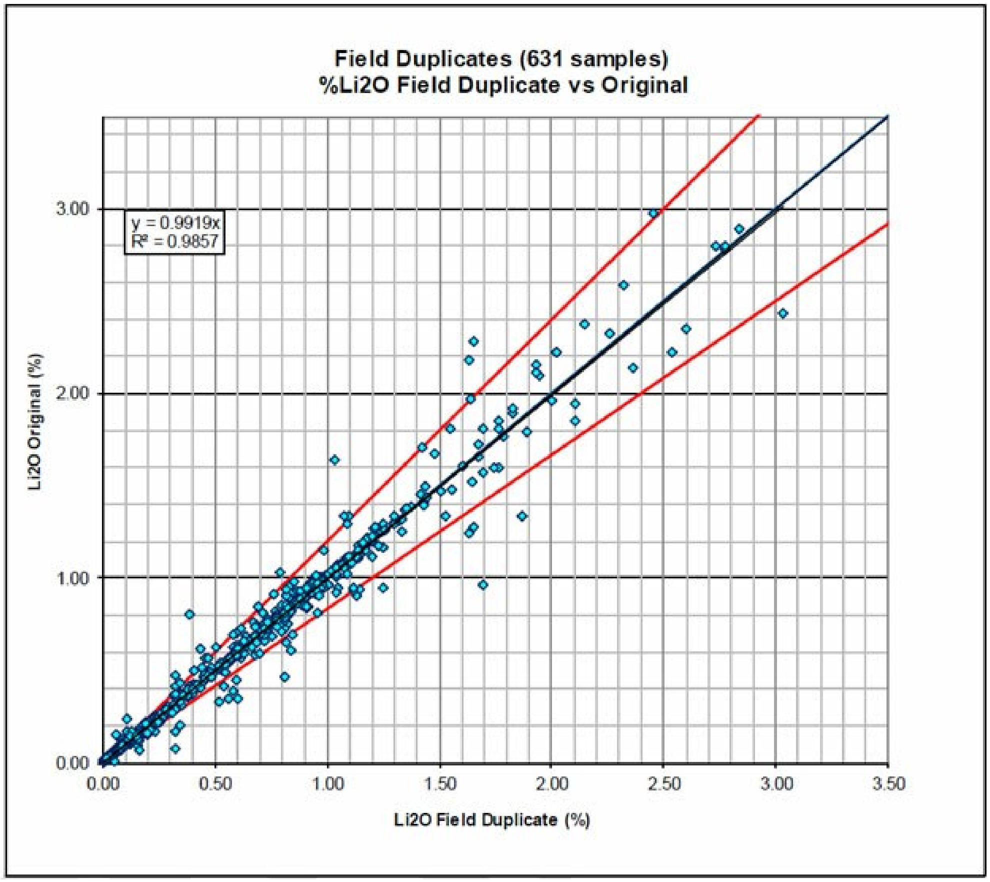
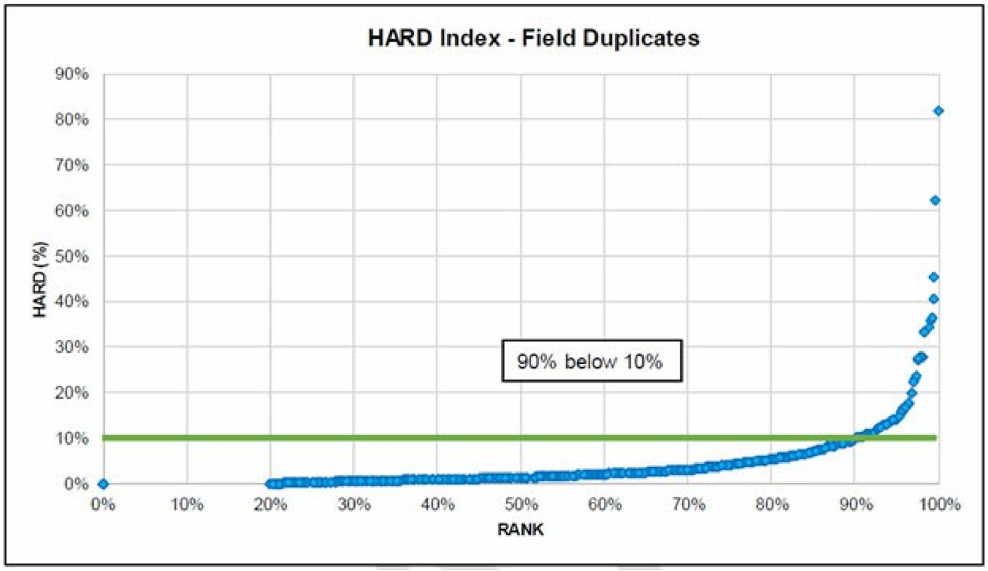


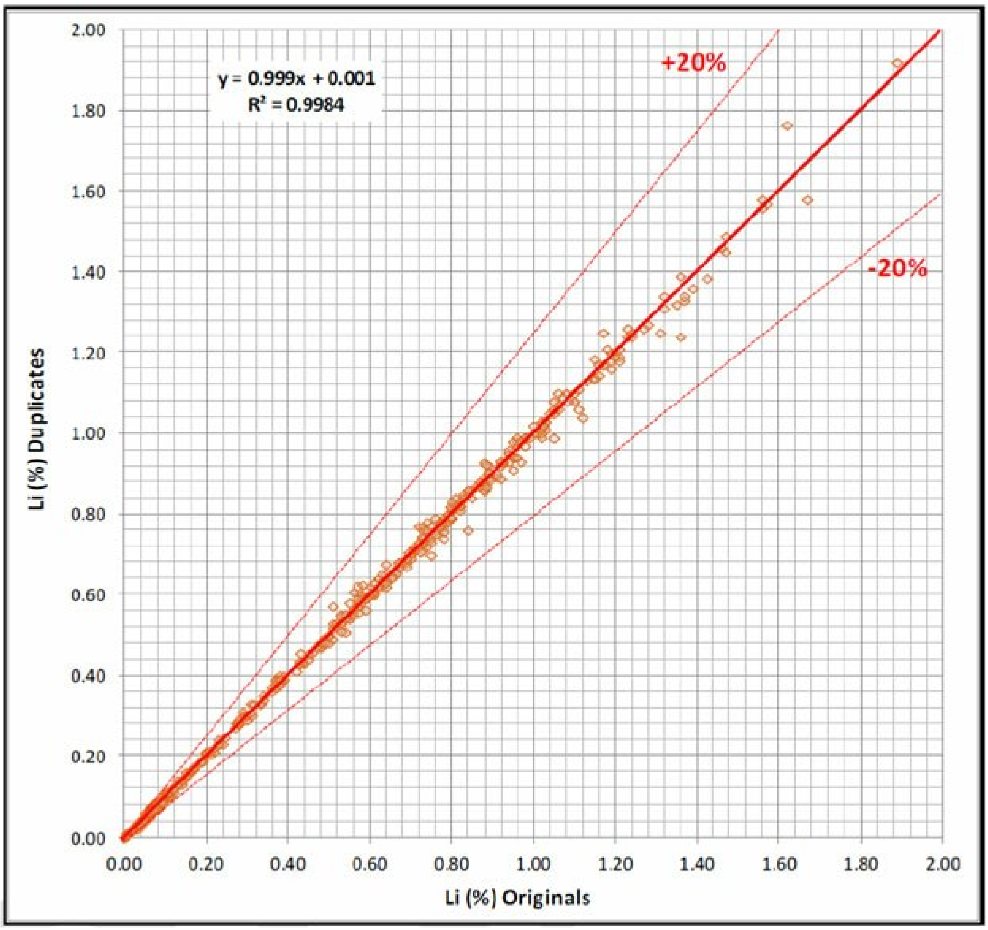
Figure 8-14 HARD Index Plot of Field Duplicates



8.7 Pulp Duplicates

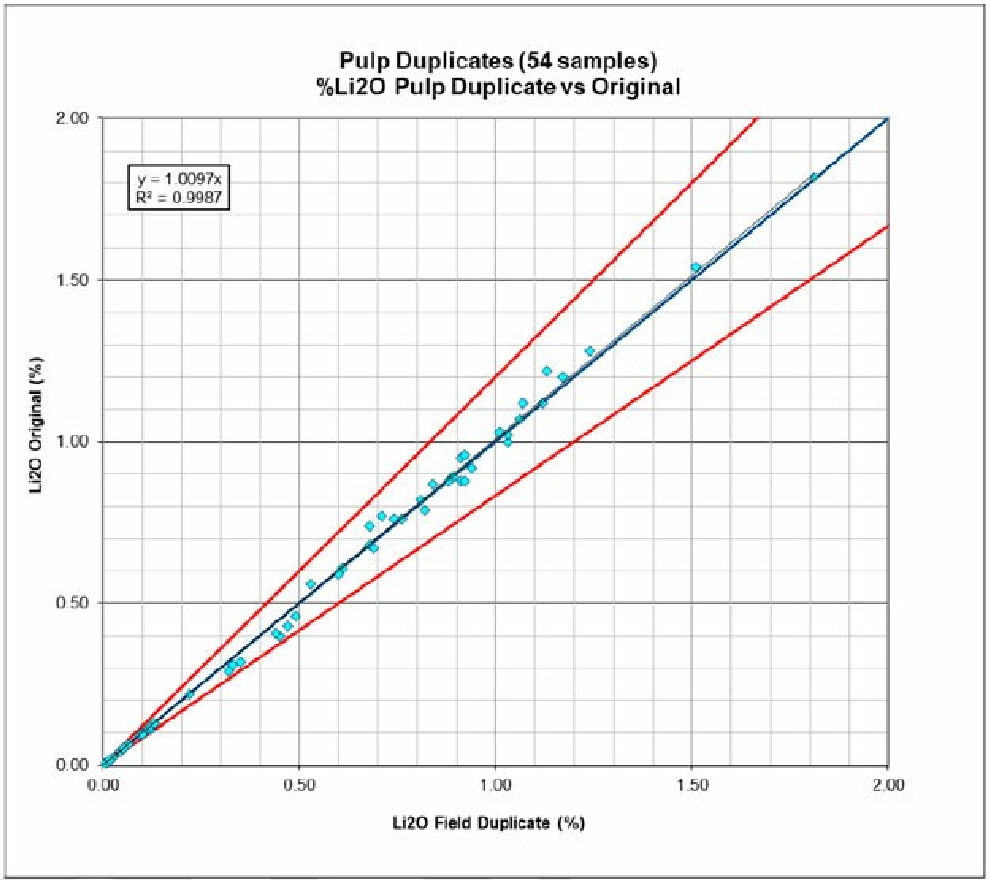
As part of laboratories internal QA/QC during 2010 and 2011 programs, pulp reanalysis was undertaken with the following protocols: 1 pulp re-analysis every 10 samples shipped to SGS, and 1 pulp re-analysis for every 35 samples shipped to ALS Chemex. SGS believes the data provided in Figure 8-15 comes from laboratories routine internal re-analysis but could not verify its origin. Data shows a very good correlation and no bias is observed.

Figure 8-15 Pulp Duplicate (2010-2011)



In 2017, the same exercise was conducted, where 54 pulp samples were sent to SGS for analytical verification. Following the reception of the results, no bias has been detected between the re-assay (Figure 8-16). All pulp duplicates have a HARD index below 10%.

Figure 8-16 Pulp Duplicates (2017)



8.8 Umpire Pulp Duplicate

In 2018, 54 pulp samples were sent to Actlab for analytical verification as inter-laboratory check. Following the reception of the results, no bias has been detected between the re-assay (Figure 8-17). All pulp duplicates have a HARD index below 10%. A graphical representation of HARD index rankings for pulp duplicates is shown in Figure 8-18.

Figure 8-17 Umpire Pulp Duplicates (2018)

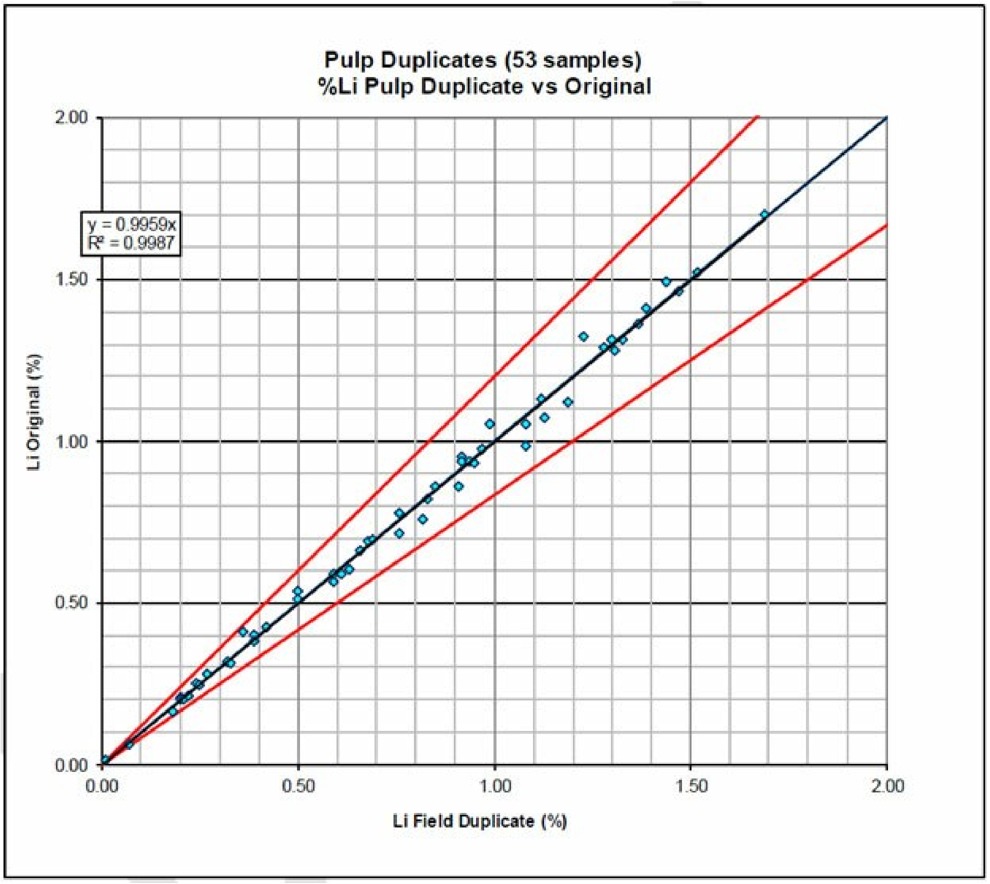
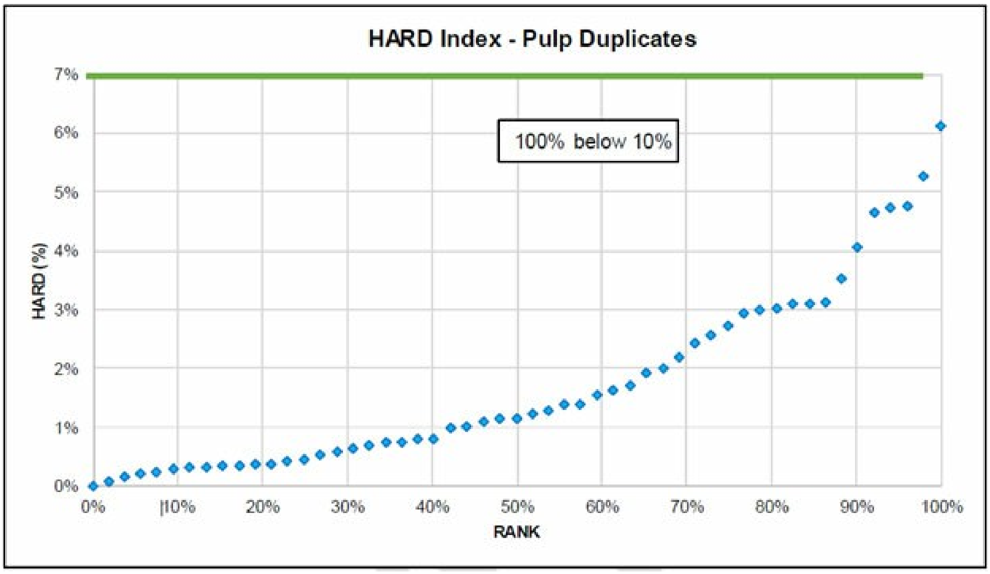


Figure 8-18 HARD Index Plot of Umpire Pulp Duplicates



8.9 2021 Resample Program

As part of a resample program in 2021, a representative selection of samples was selected to cover the first three phases of the mine plan, including 2,159 witness pulp samples to conduct a multi-elemental analysis (GE ICM40Q12). Out of that selection, 551 witness pulp samples were selected to test for halogens (GC ISE05V and GC CLA27E), total carbon and sulfur (GC CSA06) and for XRD mineralogy. Another 101 pulp samples were selected for Li analysis by peroxide fusion (GE ICP92A50). An additional sub-selection of 274 pulps samples and 96 reject material samples were used for specific gravity testing by pycnometer (G_PHY03V). The 96 matching core samples of the reject material was also selection for bulk density measurements for comparison (G_PHY04V). More details are presented in Section 8.11 Specific Gravity.

To compare and validate the pulp analysis, 39 witness reject samples were selected to test for halogens (GC ISE05V and GC CLA27E) and total carbon and sulfur (GC CSA06). To compare the multi-elemental analysis pulp analysis (GE ICM40Q12), 93 witness reject samples were selected.

8.10 Bias Analysis

In 2021-2022, SGS was mandated by NLI to conduct an assessment of the bias observed between the main lithium digestion methods used for sample assays (i.e., peroxide fusion and 4-acid). The result of the study shows that 4-acid based digestion underestimates lithium grades by 4%. Thus, it is recommended to only use peroxide fusion for all lithium analysis. Furthermore, it was evaluated that the global impact on the resource may be an underestimating of lithium grades by 1.6% (Camus and Dupéré, 2022).

8.11 Specific Gravity

An improvement opportunity was raised during technical auditing processes stating that the amount and distribution of density measurements in the deposit was sub-optimal and insufficient. To answer this issue, NLI selected 274 stored pulp samples and mandated SGS to conduct specific gravity measurements (SG) by pycnometer on historical, pulverised samples. To validate this methodology (i.e., SG tests on pulps), a representative sub-selection of 96 samples out of the 274 interval selections was made to assess the following:

1. The presence of a bias between the pulp sample and the unpulverized sample, likely caused by natural voids in the rocks, and

2. A possible degradation of the stored pulp samples over time.

To answer the first potential bias, 96 representative core samples of 10 to 15 cm were selected for bulk density measurement in air and water. To answer the second potential bias on pulp sample degradation, the rejects of the historical source material were tested for SG by pycnometer. Results are shown, in Table 8-8.

Results show that the results of the 274 pycnometers tests are reliable and representative of the main geological units at Whabouchi. Results considering porosity (air and water density) show results 1% lower than their pycnometer on pulps counterparts. Also, SG on reject showed a 1% increase in SG compared to associated pulps.

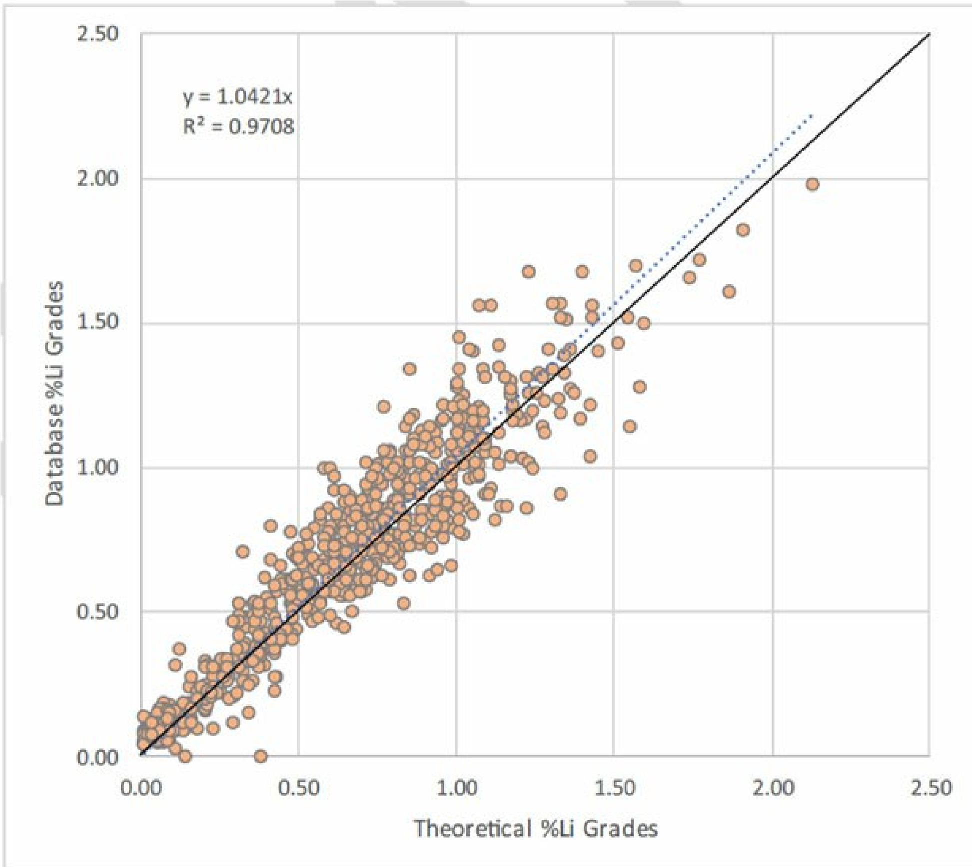
Table 8-8 Summary Statistics of New Density Measurements

Type of Measurement	Spodumene Pegmatite			Baren Pegmatite			Volcanics/Waste		
	Count	Median	Mean	Count	Median	Mean	Count	Median	Mean
SG on pulps	173	2.76	2.77	60	2.67	2.68	41	3.08	3.04
Bulk Density on core	54	2.75	2.75	22	2.64	2.64	20	3.04	3.00
SG on reject material	54	2.80	2.81	22	2.69	2.69	20	3.09	3.04

8.12 Mineralogy Analysis

Based on field observations, NLI initiated in 2022 a mineralogical identification program to identify the sources of lithium mineralization with Elemission Inc., based in Montreal, Quebec. Elemission uses a hyperspectral, laser-induced breakdown spectroscopy technology on drill core with automated mineralogy identification. This information is still incomplete and fragmentary; the mineralogical samples do not yet cover a sufficient area to be included in the block model. NLI implemented an internal quality control of mineralogical results, such as comparison with other mineralogy identification technologies (TIMA, XRD) and lithium deportment compared to original sample grades. After several improvement steps, NLI and SGS are satisfied that the results show a good correlation of mineralogy against the database grades. Figure 8-19 shows a correlation plot of theoretical lithium grades based on mineralogy against the database grades for the same interval. Theoretical grades are a sum of each Li-bearing mineral phases multiplied by its measured lithium content. The correlation observed is judged to be very good and suitable for mineralogical assessment in the Whabouchi deposit. Global preliminary results are presented in Section 11.3.2.

Figure 8-19 Comparison of Theoretical and Database Lithium Grades



8.13 **QP Conclusion**

In the opinion of the QP, the sample preparation, security and analytical procedures used by NLI are consistent with generally accepted industry best practices, therefore, adequate. Based on the bias analysis mentioned herein (SGS, 2021), SGS recommends that going forward all lithium assays should be completed using the peroxide fusion digestion method to ensure a full analysis of refractory minerals. This matter was discussed with and confirmed by NLI personnel and integrated in NLI standard protocols. The QP suggests continuing its internal QA/QC protocol for blanks, duplicates (core and pulp) and certified reference material.

9 DATA VERIFICATION

The drill hole database provided by NLI to SGS was validated by inspecting the following information: drill hole collar, deviation surveys, hole length, assays, and lithology. Drill hole collar and deviations were validated against the annual drilling reports and also checked for lithological consistency in Leapfrog Geo®. No major issues were found during this validation. Minor errors were identified (downhole survey and missing assays) and communicated with NLI representatives. Some drill logs were also compared with the downhole intervals contained within the database.

The assay database was compared with the original laboratory certificates. Approximately 76% of the drilling database was checked against the Excel® format certificates and no errors were found. Approximately 5% of original assay certificates in PDF format were also checked against the Excel® format files and no errors were found. The database used in the Mineral Resource Estimate assumes%Li_2O content, whereas laboratories mostly report lithium as %Li. A ratio of 2.153 was used to convert %Li to%Li_2O and is consistent with the standard atomic weight of Li (6.94) and O (15.999).

The final database used for the Mineral Resource Estimate includes channel samples collected in 2009 and 2010 (prefixed R-###) and drill hole core data collected during the 2009, 2010, 2011, 2013, 2016, 2017 and 2018 drilling campaigns (prefixed WHA-YY-###). Three historical drill holes were removed from the database because discrepancies were found in logging or assaying compared to nearby drill holes. Furthermore, no reference information was provided to GMS relating to these holes (24040, 24041, 24042). The QP is of the opinion that the database is in good standing and can be used for a Mineral Resource Estimate. NLI geologists should continue with the QA/QC protocol already in place and use certified reference material, blanks, coarse duplicate and pulp duplicates.

9.1 Site Visit

SGS Geological Services's QP completed a site visit at the Whabouchi Project on July 11, 2022. During his visit, the QP visited the mine infrastructures, core logging facilities, offices, rejects and pulps storage, outcrops (including the bulk sample area and channel sampling) and the stockpiles that served for the pilot process plant.

9.2 QP Conclusions

The database validation process, drill core inspection, outcrop inspection, diamond drill hole and channel sample verification, and geological model ground truthing confirmed the validity of the drilling database and supporting information using the Mineral Resource Estimate. No major issues were found during data validation, both digitally and in the field.

10 MINERAL PROCESSING AND METALLURGICAL TESTING

Mineral processing testing was performed to evaluate the potential of spodumene concentrate production.

The spodumene concentrate production testwork is presented in Section 10.1.

10.1 Spodumene Concentration Testwork

Preliminary metallurgical investigation of the Whabouchi deposit was first carried out in 2010 and 2011 by SGS at Lakefield, Ontario.

Some of the bench scale and pilot plant testwork results were reported in greater detail in previous Technical Reports in 2012, 2013, 2016, 2018, and 2019. The 2012-2016 testwork was performed to support a flowsheet with target of producing a 6.0 % Li_2O spodumene product to the conversion plant. The 2018-2019 testwork was performed in order to support a flowsheet with target of providing 6.25 % Li_2O spodumene concentrate product to feed to the flash calciner in the conversion plant.

In 2021, under new ownership, a detailed evaluation was performed to mitigate the process risks associated with flash calcining leading to the decision to revert to a rotary kiln for spodumene calcination. This decision allowed for a reduced concentrate specification entering the conversion plant to 5.5 % Li_2O. Following a review of the data by the new owners and the modification to the product specification, some additional testwork was recommended and carried out.

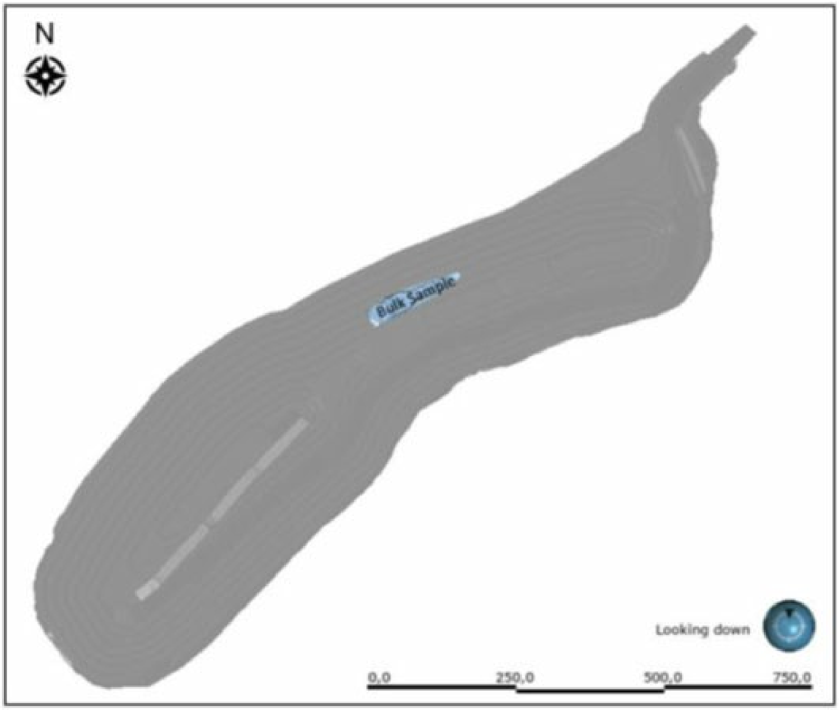
One of the major test programs carried out was some variability work performed using the Whabouchi flowsheet on five (5) samples which were representative of the first five years of the mine plan (based on the 2019 mine plan). The goal of the program was to produce five representative 5.5 % Li_2O concentrates for downstream conversion testing and to evaluate the distribution and concentration behaviour of gangue minerals throughout the beneficiation process. This test program achieved the target grade but did not attain the target spodumene recovery for the concentrator.

This variability work is very important for confidence in the short-term performance of the flowsheet. Previous large-scale work carried out up prior to 2019 has been based on bulk samples taken from outcrops, with limited work performed on variability samples taken spatially throughout the deposit. The main bulk sample location is shown in Figure 10-1.

A summary of the historical testwork performed up until 2019 to develop the Whabouchi concentrator flowsheet is presented in Sections 10.1.1.1 to 10.1.8. These sections include but are not limited to ore sorting, hydraulic separation, dense media separation (DMS), flotation, screening, dewatering, drying, and magnetic separation performed by various laboratories and suppliers.

The additional testing carried out since 2019 is described in Section 10.1.9. This work includes the variability work carried out on samples from the first five (5) years of the mine plan, as well as some additional split-feed coarse particle flotation work, coagulant testing, and saponification testing.

Figure 10-1 Plan View of Bulk Sample Location Relative to the Planned Pit



10.1.1 Ore Sorting Testing

10.1.1.1 Tomra Testwork Program

NLI provided TOMRA with 1.8 tonnes of feed ore ranging in size from 40 mm to 9.5 mm to conduct testing program to remove black rock (amphibolite) from white rocks (pegmatite). The material was sent to the TOMRA test center in Wedel, Germany.

For the first tests of the NLI ore, several TOMRA sorting technologies and several approaches were considered to select the ones that responded best. The data collected according to the characteristics of the rocks present in the ores made it possible to show the potential for sorting of each sensor type for a specific task. Two (2) technologies were targeted for sorting trials: surface color differentiation sorting technology and X-ray transmission (XRT). XRT uses atomic density difference separation technology.

Below, the best results of the two (2) size fractions are discussed.

a. Ore Sorting on - 40 mm + 20 mm, Amphibole Removal

   Using XRT, the accepted material white rocks recovery was 95.8%, with white rock at 98.7% concentration. The photo in Figure 10-2 illustrates the visual results. There is very little displaced material in the accepted stream.

Figure 10-2 TOMRA Ore Sorting Test #1

Source: TOMRA test center in Wedel, Germany March 2017

b. Ore Sorting on - 20 mm + 9.5 mm, Amphibole Removal

Using XRT, the accepted material white rocks recovery was 96.8%, with white rock at 99.2% concentration.

10.1.1.2 <u>Steinert Testwork Program</u>

Steinert US was approached to test NLI Whabouchi ore with sensor-based sorting techniques to beneficiate/upgrade the ore by removing waste rock consisting mainly of amphibolite. The test sample provided consisted of pegmatite and dark amphibolite waste rocks.

The objective was test sorting efficiency of Steinert ore sorting equipment. The sorting sensor selected was X-Ray Transmission or XRT. X-ray sensitive line-scan sensor provides high resolution X-ray absorption images. An X-ray scintillation crystal sensor can capture up to 2,500 lines per second.

NLI had prepared two (2) tonnes of sample. The sample was screened at 10 mm to remove the fines which is not suitable for this sorting application. The first set of three tests were performed on - 50mm + 15mm material. This size range is good ore sorter feed material, about ratio 1:3 sorting size. The next three (3) tests were done on - 15mm + 10mm material. Finally, the last three (3) tests were done on - 50 + 10mm size range, outside the ideal size ratio.

a. Ore Sorting on - 50 mm + 15 mm

The coarse sample - 50 mm + 15 mm yielded excellent results. Test #1 yielded the best result at 99.4% separation efficiency of the white ore and was separated from the dark amphibolite. The photos in Figure 10-3 illustrate the visual results. There is very little misplaced material in the accepted and rejected streams.

Figure 10-3 Steinert Ore Sorting Test #1

Source: Steinert US Test Laboratory April 2017

b. Ore Sorting on - 15 mm + 10 mm

The finer sample - 15 mm + 10 mm was processed through the same equipment and also produced very good results, but noticeably less than the coarse separation. The best separation efficiency obtained was 96.7%.

c. Ore Sorting on - 50 mm + 10 mm

For the combined sample - 50 mm + 10 mm, the best separation efficiency obtained was 94.5%.

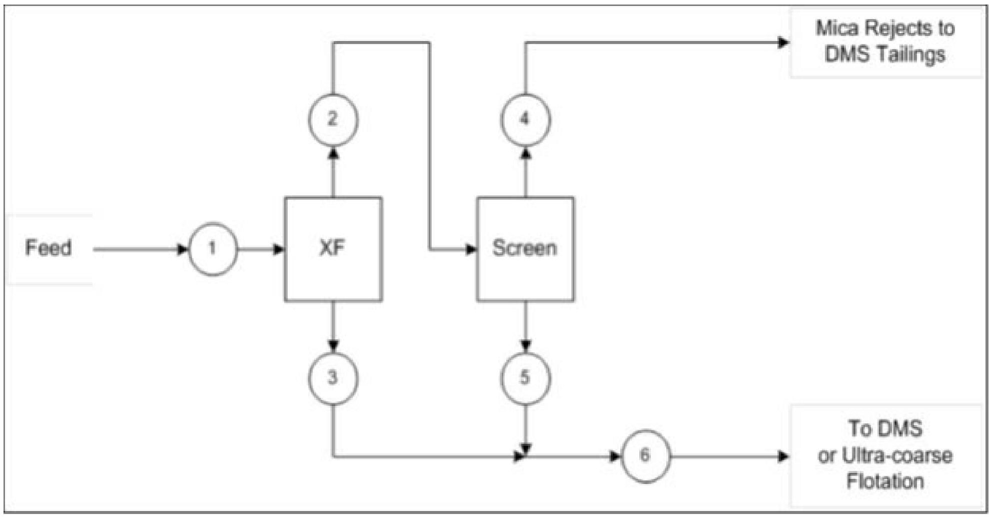
10.1.2 Eriez Hydraulic Separation Testwork

Hydraulic separation was performed at Eriez Central Test Laboratory in Erie, PA. Eriez received spodumene ore from NLI for testing. Hydraulic separation testwork was done with two (2) different size fractions (- 8 mm + 0.85 mm) and (-0.85 mm). The tests were aimed to remove mica from the ore.

10.1.2.1 Hydraulic Separation on - 8 mm + 0.85 mm, Mica Removal

The coarser sample - 8 mm + 0.85 mm was processed using a CrossFlow separation equipment, following the flowsheet shown in Figure 10-4. The material was processed in 9 × 16 in. Eriez CrossFlow Separator. The teeter water up flow was 1.83 cm/s.

Figure 10-4 Flowsheet CrossFlow Test #6



Source: Eriez Flotation Division Test Laboratory 2017

The photos shown in Figure 10-5 illustrate the effectiveness of the CrossFlow separator on coarse muscovite particles.

To a certain extent, the K_2O concentration can be related to the muscovite concentration in the head samples. However, it is not the only potassium bearing mineral. In the case of Whabouchi ore, the presence of K-Feldspar does not allow to evaluate the muscovite recovery based on the K_2O analyses. The CrossFlow separator overflow, which contains mainly coarse muscovite flakes and fines particles entrained with the upward current was screened at 2.0 mm. This allowed the recovery of mainly muscovite and the return of entrained particles into the flotation circuit. The analytical results are presented in Table 10-1.

Pure muscovite is about 11.8% K_2O. It can be seen that the screen oversize contains a very high concentration of muscovite as the other potassium bearing minerals are not retained by the screen because of their size and shape.

Figure 10-5 Photographs Classified CrossFlow Test #6

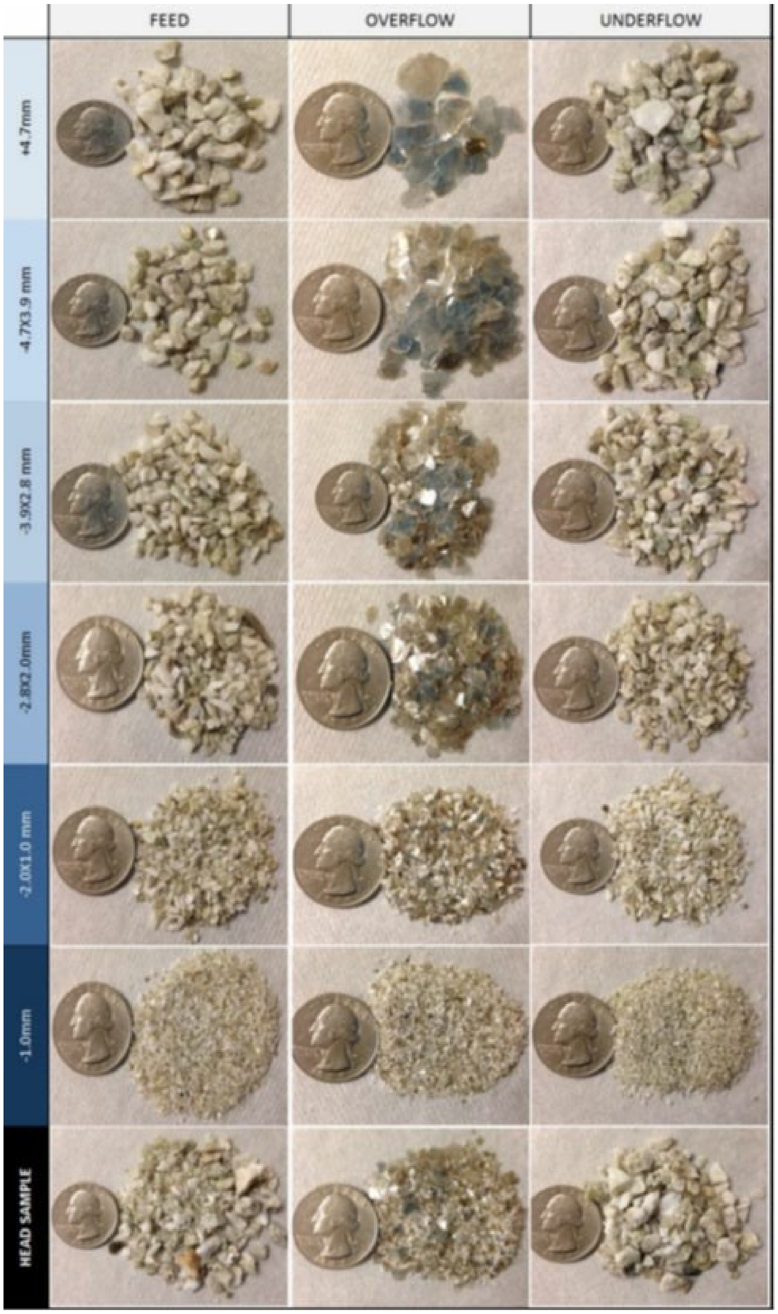


Table 10-1 CrossFlow Separation Test #6

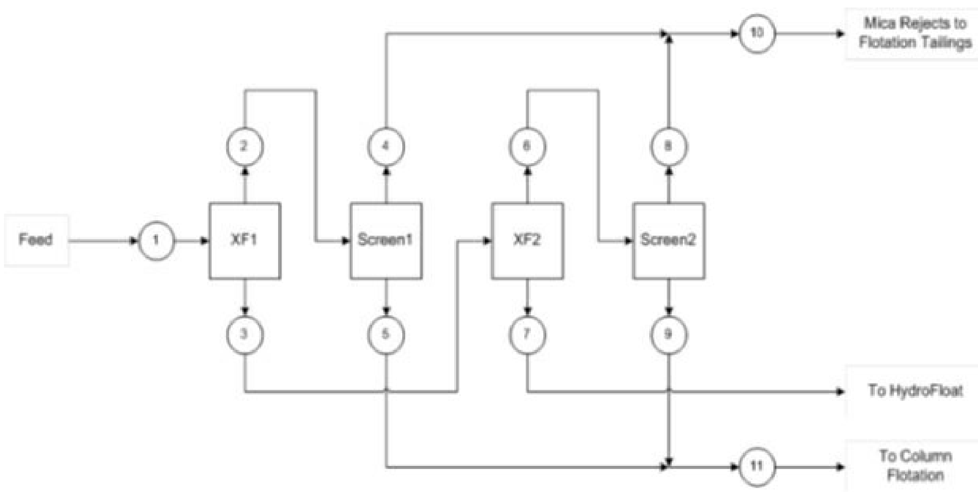
ID	Stream	Weight (%)	Li_2O (%)	Dist. Li_2O (%)	K_2O (%)	Dist. K_2O (%)
1	Feed	100.00	1.70	100.00	2.97	100.00
2	XF6 Overflow	16.88	0.64	6.32	5.28	30.03
3	XF6 Underflow	83.12	1.91	93.68	2.50	69.97
4	Screen Overflow	0.69	0.44	0.18	10.02	2.32
5	Screen Underflow	16.19	0.64	6.14	5.07	27.71
6	DMS Feed	99.31	1.71	99.82	2.92	97.67

In the Table 13.1, the Li_2O concentration in the screen overflow corresponds to the natural content in the muscovite. The loss in spodumene is negligible.

10.1.2.2 Hydraulic Separation on - 0.85 mm Mica Removal

The finer sample - 0.85 mm was processed using CrossFlow separation equipment, following the flowsheet shown in Figure 10-6. Two (2) CrossFlow separators in series with screens were used. The screen oversize is considered mica waste and the remainder will be flotation feed.

Figure 10-6 Flowsheet CrossFlow Test #1 and #2



Both CrossFlow overflows were screened at 212 microns. The analytical results are presented in Table 10-2.

As can be seen, the weight rejection at the second CrossFlow, which used much more water, was very high. This is considered too high and cannot be accepted by the process as a high loss of lithium was observed. The CrossFlow operation will have to be adjusted to make a slightly finer cut which will be sufficient to reject entrained muscovite since its shape factor allows it to be rejected at lower water flows.

Table 10-2 CrossFlow Separation Test #1 and #2

ID	Stream	Weight (%)	Li_2O (%)	Dist. Li_2O (%)	K_2O (%)	Dist. K_2O (%)
1	Feed	100.00	1.39	100.00	2.32	100.00
2	XF 1 Overflow	34.87	0.85	21.24	2.55	38.35
3	XF 1 Underflow	65.13	1.68	78.76	2.20	61.65

4	Screen 1 Overflow	2.10	0.34	0.52	5.72	5.19
5	Screen 1 Underflow	32.76	0.88	20.73	2.35	33.15
6	XF 2 Overflow	18.19	1.06	13.86	2.68	20.98
7	XF 2 Underflow	46.94	1.92	64.90	2.01	40.67
8	Screen 2 Overflow	12.70	0.58	5.27	3.18	17.43
9	Screen 2 Underflow	5.49	2.18	8.59	1.50	3.55
10	Mica Rejects	14.81	0.54	5.78	3.55	22.63
11	Combined Screen U/F	38.25	1.07	29.32	2.23	36.70

10.1.3 DMS Testwork

10.1.3.1 DMS Testwork at SGS – 2011

Four (4) composites were prepared as pilot plant feed. Two (2) composites were prepared for the flotation pilot plant and were labelled as “Outcrop Sample for flotation” and “Mine Representative Sample for Flotation”. Two (2) more composites were prepared for DMS pilot plant testwork and were labelled as “DMS Outcrop Composite” and “DMS-Mine Representative Composite”. The mine representative sample was composed of outcrop material and drill core rejects. A detailed description of these composite is given in SGS report titled: “A pilot plant investigation into The Recovery of Spodumene from the Whabouchi Property, Project 12486-003 – Final Report, April 2, 2012”. The chemical analyses of these composites are presented in Table 10-3.

Table 10-3 Chemical Analysis of the Flotation Pilot Plant Composites

	Composite	Unit	Outcrop	Mine Representative	DMS Outcrop Middlings	DMS Mine Rep. Middlings
Elemental Analysis	Li	%	0.76	0.72	0.68	0.86
	Li_2O	%	1.64	1.55	1.47	1.85
	BE	ppm	178	156	---	---
	Beryl	%	0.27	0.24	---	---
Whole Rock Analysis	SiO_2	%	74.4	74.4	76.1	76
	Al_2O_3	%	15.9	15.9	14.9	15.7
	Fe_2O_3	%	0.78	0.9	0.85	0.88
	MgO	%	0.09	0.18	0.09	0.08
	CaO	%	0.26	0.43	0.3	0.3
	Na_2O	%	3.36	3.34	3.42	3.05
	K_2O	%	2.45	2.63	2.08	2.08
	TiO_2	%	0.01	0.03	0.01	0.01
	P_2O_5	%	0.1	0.12	0.1	0.1
	MnO	%	0.08	0.1	0.1	0.08
	Cr_2O_3	%	0.03	0.03	0.03	0.03
	V_2O_5	%	< 0.01	< 0.01	< 0.01	< 0.01
	LOI	%	0.72	0.84	0.57	0.84
	Sum	%	< 98.5	98.9	98.6	99.1

Semi quantitative XRD analyses of composites are provided in Table 10-4. The results indicate that the outcrop and DMS mine representative samples are very similar in make-up. The Mine Representative sample has less spodumene than the two (2) others. The mineral spodumene content is slightly over 20%. The Mine Representative sample was crushed and screened, and the liberation of Li-minerals was assessed at 91.4%.

Table 10-4 Semi – Quantitative XRD Analyses of the Composite

Mineral	Outcrop Sample (Weight%)	DMS Mine Rep Sample (Weight%)	Mine Rep Sample (Weight%)
Quartz	26.1	26.9	33.4
Spodumene (Monoclinic)	22.6	22.3	20.8
Albite	31.1	31.2	29.3
Microcline	14.9	17.7	11.5
Magnesiohornblende	-	-	1.0
Muscovite	5.3	1.9	4.0
Total	100.0	100.0	100.0

The DMS pilot plant testwork was carried out on two (2) samples, the blasted sample, and the mine representative sample, from the Whabouchi deposit by SGS at Lakefield in 2011. The flowsheet used during the pilot plant, consisting of several unit operations, includes crushing, scrubbing, screening, several dense media separation stages, magnetic separation and dewatering. The DMS test plant was equipped with a 150-mm dense media cyclone. Up to eight (8) DMS stages were used for the blasted sample, whereas only four (4) stages were used in testwork on the mine representative sample. Magnetic separation was conducted by using a high intensity rare earth roll magnetic separator to upgrade DMS sinks on previously dried feed.

The results of the DMS pilot plant were reported in an addendum by SGS (12486-004) entitled “An Investigation into DMS Plant Testing on Material from the Whabouchi Lithium Deposit” issued November 11, 2011 and incorporated in the November 16, 2012 NI 43-101 Technical Report Preliminary Economic Assessment.

The main findings from these results were that for mine representative sample, the 4-stage DMS flowsheet rejects 40% of the feed mass as tailings at a top size of 9.5 mm, at a loss of 10% Li. At a top size of 9.5 mm, approximately 11% of the feed mass is recovered as spodumene concentrate grading 6.4% Li_2O and 45% Li distribution. The combined middlings, 49% of the feed mass, represent the remaining 45% Li distribution.

10.1.3.2 MET-Solve Laboratories Test Program – 2013-2014

In 2013, NLI contracted Met-Solve Laboratories to carry out DMS pilot plant testwork.

DMS testing was investigated as the costs of this form of processing will be considerately lower as compared to flotation.

DMS pilot plant testing was carried out using a single 250 mm separator to simulate multi-stage separations. The primary objective of this test program was to determine the overall grade and recovery of lithium using a DMS system.

The multi-stage separator has been shown to offer better performance compared to single stage, two (2) products and other conventional dense medium cyclone separators. The following section is a summary of the report issued by Met-Solve titled: “Nemaska Lithium Inc., Dense Media Separation, MS 1467 issued August 13, 2013”.

Four (4) drums of pre-crushed sample (- 9.5 mm + 0.5 mm) weighing approximately 900 kg were sent from SGS at Lakefield to Met-Solve laboratories in Langley, BC. Only half of the sample (approximately 450 kg) was used for these DMS tests.

A total of seven (7) DMS tests (FT201 to FT207) were carried out on these pre-crushed samples. Approximately 65 kg of representative sample was used for each test.

The initial five (5) DMS tests (FT201 to FT205) were aimed to simulate a 4-stage DMS concentrator with a re-circulated test on the crushed middlings (Figure 10-7). Each run consisted of passing the floats through the single media separator twice to simulate a 2-stage concentrator. The total sinks (combined Sinks 1, Sinks 3 and Sinks 4) can be considered final concentrate, while Floats 2 is final tailings. Floats 4 and the –0.5 mm rejects could be sent to the flotation circuit to improve the lithium recovery.

Figure 10-7 General Process Flowsheet for Initial DMS Testwork (Tests FT201-FT205)

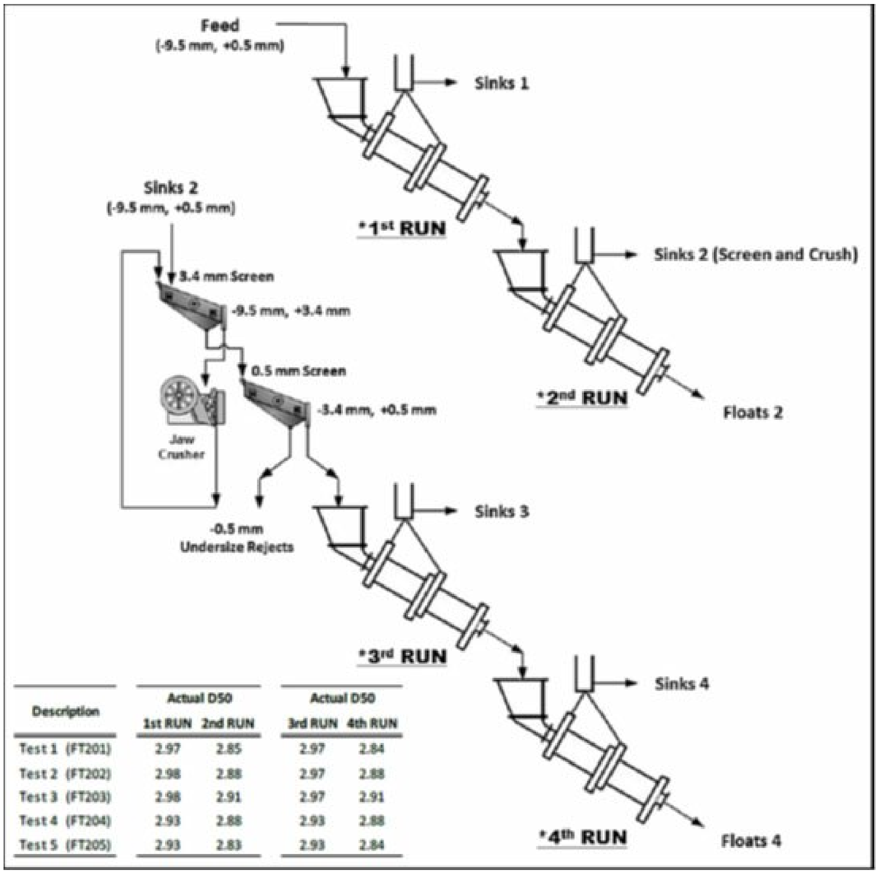


Table 10-5 lists the results of the five (5) tests. In each test, propensity of the spodumene ore to dense media separation was confirmed.

Table 10-5 Process Flowsheet for Initial DMS Testwork Results

Test	Mass (%)			Li_2O Distribution (%)			Li_2O Assay (%)			
	Total Sinks	Floats 2	Floats 4	Total Sinks	Floats 2	Floats 4	Calc. Head	Total Sinks	Floats 2	Floats 4
FT201	17.9	78.9	1.6	59.9	35.6	1.6	1.61	5.39	0.73	1.61
FT202	16.2	79.1	2.4	56.7	35.3	3.6	1.62	5.67	0.72	2.39
FT203	14.9	81.7	1.7	53.3	40.6	2.7	1.59	5.67	0.79	2.52
FT204	17.5	78.2	2.4	59.0	34.2	3.3	1.68	5.56	0.72	2.32
FT205	17.6	77.9	2.0	59.2	34.3	2.1	1.68	5.66	0.74	1.73

Two (2) additional tests, FT206 without and FT207 with crushing middlings, were carried out to simulate a dynamic 3-stage DMS circuit, in order to assess the effectiveness of adding more separation stages. Test FT206 yielded a higher recovery of 66.8%, but with a lower concentrate grade of 5.17% Li_2O. The tailings grade was 0.74% Li_2O, which is comparable to the initial tests. The results of test FT206 are listed in Table 10-6.

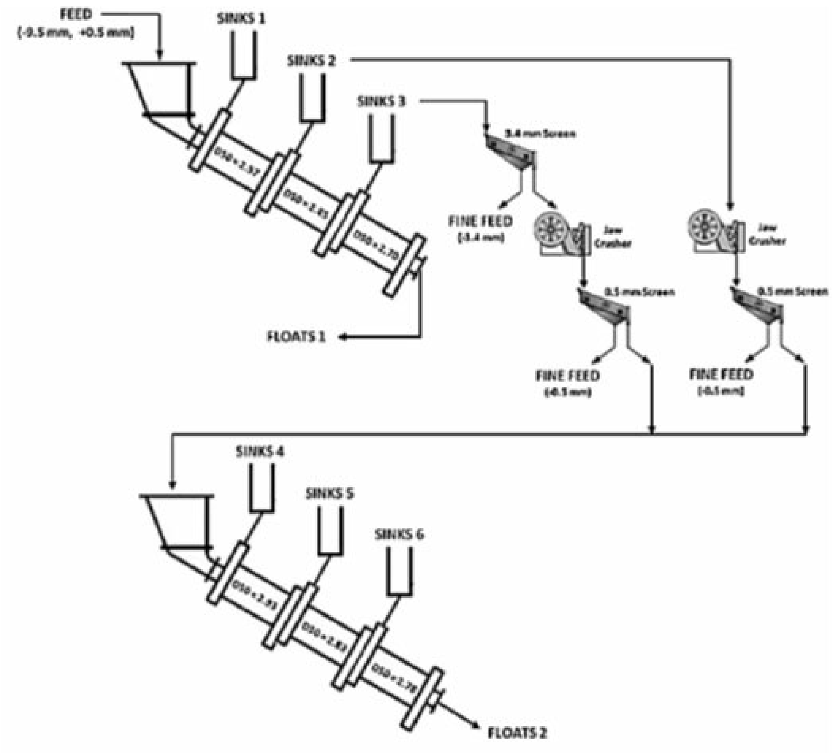
Table 10-6 FT206 3-Stage DMS Test

Description:	FT206 (3-Stage DMS)		
	1st Stage	2nd Stage	3rd Stage
*D50 (Specific Gravity):	3.00	3.00	2.87
Specific Gravity of Dense Media:	2.84	2.84	2.44
Medium Inlet Pressure (psi):	28	28	28
	*Based on tracer tests		
Feed Particle Size:	(- 9.5 mm + 0.5 mm)		

Description	Weight		Assay			Distribution		
	kg	%	Li_2O %	Si %	Al %	Li_2O %	Si %	Al %
Sinks 1	4.27	6.8	6.16	29.6	11.60	24.2	6.1	10.3
Sinks 2	1.67	2.7	5.89	31.0	11.55	9.0	2.5	4.0
Sinks 3	8.07	12.9	4.50	32.8	9.86	33.5	12.7	16.5
Total Sinks	**14.01**	**22.4**	**5.17**	**31.6**	**10.59**	**66.8**	**21.3**	**30.8**
Floats	48.48	77.6	0.74	33.7	6.89	33.2	78.7	69.2
Calc Head	62.48	100.0	1.74	33.2	7.72	100.0	100.0	100.0
Calc Head (SFA)			1.65	33.7	7.81			

The flowsheet of test F207 with the crushing step is shown in Figure 10-8. Test FT207 yielded the best results, based on recovery and final concentrate grade. This indicates that lower density settings with less coarse feed may yield even improved results.

Figure 10-8 General Process Flowsheet for Initial DMS Testwork (Tests FT207)



Description	1st Stage	2nd Stage	3rd Stage	4th Stage	5th Stage	6th Stage
	(Initial 3-Stage DMS)			*(3-Stage Re-circulated on Crushed Middlings)*		
Test 7 (FT207) *Actual D50	2.97	2.85	2.69	2.93	2.83	2.78

The results of test FT207 indicate that using the 3-stage DMS the Floats 1 lithium content can be reduced with a lower dense media pulp specific gravity. The sinks grade can be improved by using some sinks as middlings to be crushed and re-circulated. In conclusion, test FT207 had a low loss of lithium while producing a relatively large quantity of DMS concentrate. The low tailings part is important as the lower grade sinks can be reprocessed as middlings in the flotation circuit. The results from FT207 are listed in Table 10-7.

Table 10-7 Results of Test FT207

Description	Weight		Assay			Distribution		
			Li_2O	Si	Al	Li_2O	Si	Al
	(kg)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Sink 1	7.40	10.9	5.87	29.6	11.55	37.6	9.4	15.7
Sink2 (–0.5 mm)	2.16	3.2	3.52	32.1	9.28	6.6	3.0	3.7
Sink3 (–3.4 mm)	4.76	7.0	1.85	36.1	8.03	7.6	7.4	7.0
Sink3 (–0.5 mm)	2.74	4.1	1.30	32.6	7.27	3.1	3.8	3.7
Sink 4	1.69	2.5	6.32	30.9	11.95	9.3	2.2	3.7
Sink 5	3.37	5.0	5.02	32.7	10.55	14.7	4.7	6.5
Sink 6	4.30	6.3	2.38	34.8	8.26	8.9	6.4	6.5
Total Sinks	26.42	39.0	3.83	32.6	9.65	87.7	36.9	46.7
Floats 1	37.36	55.2	0.31	35.7	7.08	10.0	57.1	48.4
Floats 2	3.91	5.8	0.69	35.5	6.77	2.3	5.9	4.8
Calc. Head	67.69	100.0	1.71	34.4	8.06	100.0	100.0	100.0
Calc. Head (SFA)			1.65	33.7	7.81			

In 2014, test FT700 was carried-out to re-visit the FT207 test, without crushing and re-circulation of the lower grade Sinks. FT700 indicates that when using the 3-stage DMS, the Floats lithium content can be reduced with a low dense media pulp specific gravity, similar to FT207. Test FT700 had the lowest loss of lithium for the standard test; the latter is of prime importance as the lower grade sinks can be reprocessed as middlings in the flotation circuit. The results from FT700 are listed in Table 10-8.

Table 10-8 Results of Test FT700

Description	FT700 (3-Stage DMS)		
	1st Stage	2nd Stage	3rd Stage
*D50 (Specific Gravity):	2.940	2.855	2.700
Specific Gravity of Dense Media:	2.670	2.370	2.700
Medium Inlet Pressure (psi):	25	23	24
Back Pressure (mm):	200		

Description	Weight		Assay			Distribution		
			Li_2O	Si	Al	Li_2O	Si	Al
	(kg)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Sinks 1	33.23	9.67	5.88	33.0	11.80	33.5	8.7	14.2
Sinks 2	46.10	13.42	4.30	35.9	10.16	34.1	13.1	16.9
Sinks 3	68.37	19.91	1.85	37.6	8.03	21.8	20.3	19.8
Total Sinks	147.69	43.00	3.52	36.0	9.54	89.3	42.1	50.9
Floats	195.75	57.00	0.32	37.4	6.95	10.7	57.9	49.1
Calc Head	343.44	100.0	1.70	36.8	8.06	100.0	100.0	100.0
Calc. Head (SFA)			1.65	33.7	7.81			

10.1.3.3 COREM Heavy Liquid Separation Testwork Program

To confirm the cut points for DMS heavy liquid separation tests were conducted on Whabouchi feed, with a grade 1.77% Li_2O. The tested samples were crushed into three size fraction ranges, –12.5 mm 0.85 mm, –9.5 mm 0.85 mm and –6.3 mm 0.85 mm. The separation cut points were 2.96 and 2.70. The results were presented in a Power Point presentation “Résultats préliminaires – Caractérisation minéralogique des roches concassées du dépôt de Whabouchi” or (Preliminary Results - Mineralogical Characterization of Crushed Rocks from the Whabouchi Deposit). The results are presented in Table 10-9.

Table 10-9 HLS Mineralogical Study

Size Fractions	Sinks — 2.96		Sinks — 2.70		Floats — 2.70		–0.85 mm
	Weight (%)	Li_2O (%)	Weight (%)	Li_2O (%)	Weight (%)	Li_2O (%)	Weight (%)
–12.5 +0.85 mm	7.5	6.63	35.2	2.83	43.0	0.33	14.3
–9.5 +0.85 mm	8.6	6.43	26.6	2.84	45.0	0.24	19.8
–6.3 +0.85 mm	12.6	6.47	22.3	2.78	41.1	0.17	24.0

The fraction - 0.85 mm will go to fine ore processing. HLS Floats are final tailings and the - 12.5 mm has the highest tailings grade. The finest fraction (- 6.3 mm) has the lowest tailings indicating that Spodumene was better liberated in the finer fractions. Since NLI elected to use –9.5 mm, these results have been listed in Table 10-10.

Table 10-10 Size Fraction – 9.5 mm HLS

Stream –9.5 mm	Weight (%)	Li_2O (%)	Li Rec. (%)
Sinks — 2.96	10.7	6.43	38.8
Sinks — 2.70	33.2	2.84	53.6
Float — 2.70	56.1	0.24	7.6
Feed (Calc.)	100.0	1.77	100.0

10.1.4 Derrick Testwork Program – Fine Screening Testing

Derrick received samples produced by SGS Minerals during the Pilot plant operation (2017). 200 kg drums of CrossFlow overflow and flotation feed were sent to Derrick's Buffalo facility. Most of the tests were done on the flotation feed to split the material between coarse and fine flotation.

10.1.4.1 Fine Screening at 212 Microns

Tests #1 was performed on overflow from the Fine Muscovite Removal CrossFlow separator and the screening yielded good results with near 94% efficiency.

Flotation feed screening using screen openings of 0.21 mm was done in Test #2, Test #3, and Test #9. There is a significant quantity of fines in the oversize which may interfere with the Hydro-Float separation. The results are listed in Table 10-11.

Table 10-11 Fine Screening Tests at 212 Microns

Test Number	Test Screen Opening (mm)	Feed Rate (t/h)	Feed Solids (%)	Wash Water, (m^3/h)	Cumulative Percentage at 0.212 mm			Screening Efficiency
					Feed	Oversize	Undersize	
1	0.21	34.0	8.75	0.0	5.58	47.0	1.64	93.9
2	0.21	67.3	35.4	0.0	39.9	69.1	6.94	80.4
3	0.21	67.3	35.4	28.4	39.9	74.1	7.45	83.6
9	0.21	68.0	25.3	28.4	40.3	82.9	9.63	87.2

10.1.4.2 Fine Screening at 250 Microns

The flotation feed fine screening tests using 0.25 mm screen openings were done in Test #4, Test #5, and Test #10. Again, there is a significant quantity of fines in the oversize which may interfere with the Hydro-Float separation. The results are listed in Table 10-12.

Table 10-12 Fine Screening Tests at 250 Microns

Test Number	Test Screen Opening (mm)	Feed Rate (t/h)	Feed Solids (%)	Wash Water, (m^3/h)	Cumulative Percentage at 0.212 mm			Screening Efficiency
					Feed	Oversize	Undersize	
4	0.25	59.5	35.4	0.0	39.9	70.8	8.68	81.0
5	0.25	59.5	35.4	28.4	39.9	78.2	9.85	84.9
10	0.25	68.0	25.3	28.4	40.3	85.1	11.94	87.3

10.1.4.3 Fine Screening at 300 Microns

The flotation feed fine screening tests using 0.30 mm screen openings were done in Test #6, Test #7, and Test #8. Even with the larger screen openings, too many fines are present in the oversize and may interfere with the Hydro-Float separation. The results are listed in Table 10-13:

Table 10-13 Fine Screening Tests at 300 Microns

Test Number	Test Screen Opening (mm)	Feed Rate (t/h)	Feed Solids (%)	Wash Water, (m^3/h)	Cumulative Percentage at 0.212 mm			Screening Efficiency
					Feed	Oversize	Undersize	
6	0.30	67.3	35.4	0.0	39.9	72.6	10.9	81.4
7	0.30	67.3	35.4	28.4	39.9	79.0	12.6	83.9
8	0.30	68.0	25.3	28.4	40.3	86.1	14.9	85.5

10.1.5 Flotation Testwork

10.1.5.1 SGS Flotation Pilot Plant Testwork – 2011

In January 2011, NLI contracted SGS to carry out a pilot plant and bench scale testing program as part of a second phase of the Whabouchi Lithium Project.

The objectives of the second phase were:

- To produce two (2) tonnes of spodumene concentrate with a grade of six percent (6%) or higher for electrochemical testwork;
- To confirm and optimize previous bench testwork;
- To generate engineering data for concentrator design.

Findings from the pilot plant program were reported by SGS in the report “Project 12486-003”, April 2, 2012, and are summarized in the following Sections.

Pilot plant flotation tests were performed on four (4) composites, the flotation outcrop (blasted composite), flotation mine representative composite, combined DMS middlings and undersize fractions from outcrop and finally combined DMS middlings and undersize fractions from mine representative composite. Mine representative composite was composited of outcrop material and drill core assay reject samples. The detailed description of these composites is given in the previously mentioned SGS report.

Material from the four (4) composites were processed through the flotation pilot plant in a sequential manner, 21 pilot plant tests were performed.

Various flowsheet configurations were tested in the pilot plant campaigns. The objectives were to find the best and optimal operating conditions for spodumene separation and recovery.

In total, more than 40 tonnes of material were processed through this pilot plant. The final flowsheet of the last run of the pilot plant PP21 is shown in Figure 10-9. Pilot plant test PP21, yielded the most efficient processing results.

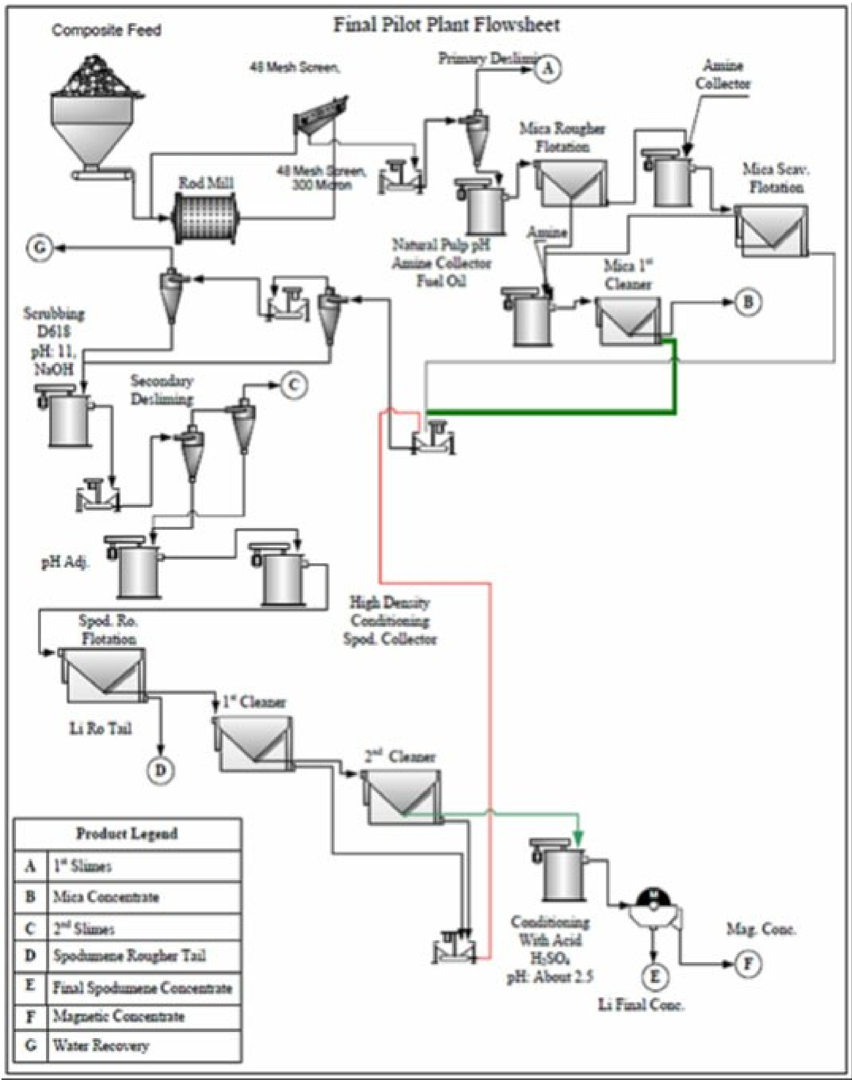
The final flowsheet consists of the following circuits:

- Grinding and screening;
- Primary de-sliming and mica flotation;
- Dewatering, scrubbing and secondary desliming;
- Spodumene flotation;
- Magnetic separation circuit.

The crushed composite feed was ground using a rod mill and screened at 300 µm. The screen oversize (+300 µm) was sent back to rod mill, while the screen undersize (-300 µm) was subjected to primary de-sliming in a cyclone where the overflow exited as slimes. The primary de-sliming cyclone underflow was conditioned with AERO 3030C or Armac collector prior to subjecting to mica rougher flotation stage followed by scavenger stage.

The rougher and scavenger concentrates were combined and transferred to a cleaner stage. Mica cleaner concentrate was collected in 200 L plastic drums, while mica cleaner tailings were dewatered in 2-stage cycloning and then underwent a scrubbing stage where dispersant D618 and NaOH were added. The scrubbed slurry was then subjected to a secondary de-sliming stage before pH adjustment and high-density conditioning with spodumene collector (LR19). The spodumene rougher flotation concentrate was subjected to 2-stage cleaning while spodumene rougher tailings were sent to final tailings.

Figure 10-9 Final Flotation Pilot Plant Flowsheet



The first and second cleaner tailings were recycled back and combined with mica flotation tailings. The final spodumene concentrate was conditioned with acid before magnetic separation to remove iron impurities. Pilot Plant test PP21 yielded the most significant results. These metallurgical results for test PP21 are summarized and presented in Table 10-14.

Table 10-14 Optimal Pilot Plant Performance for PP21 Test

Composite	PP No	Products	Mass	Assay (Adj)		Distribution
			(%)	(% Li)	(% Li_2O)	(% Li)
DMS-Mine Representative Composite	PP21	Feed	100	0.88	1.85	100
		Cyclone O/F	2.65	0.66	1.42	2.03
		Mica 1st Clnr Conc	9.2	0.34	0.73	3.63
		2nd Slimes	2.89	0.9	1.94	3
		Spod Rghr Tail	63	0.19	0.4	13.7
		Spod Mag Conc	0.18	0.23	0.5	0.05
		Spod Final (Non-Mag) Conc	22.1	3.02	6.5	77.5

The conclusions from these pilot plant results were that using flotation a spodumene final concentrate grade of 6.0% Li_2O or higher with more than 77% lithium recovery representing 22.1% weight could be obtained consistently.

These results show also that the lithium losses depend on the nature of the feed composite. For the DMS-Mine representative middling composite (PP21), the majority of lithium losses occurs in the spodumene tailings (13.7%), followed by slime removal (5.0%), mica concentrate (3.6%) and spodumene magnetics (0.05%) for a total of 22.4%. According to the results, the majority of the losses in the rougher tailings occur in the 100 meshes fraction. Coarse grain spodumene can be difficult to float thus the grinding size should be controlled to keep the K80 of the flotation feed to about 200 µm.

These results highlight the importance of mica flotation circuit ahead of spodumene flotation; by eliminating the mica flotation step, significant increase in muscovite grade was observed. The lithium oxide concentration in spodumene concentrate increased from 5.6% in PP19 as compared to 6.5% in PP21, which confirmed that removing mica ahead of spodumene flotation helped to increase final spodumene concentrate grade.

About three (3) tonnes of concentrate grading 6.0% Li_2O was generated by combining concentrates from pilot plant campaigns PP12 to PP21.

Low magnetic intensity separation (about 800 Gauss) was used to separate iron contaminant particles from the flotation concentrate. The iron grade of the lithium concentrate was about 2.11% Fe_2O_3.

10.1.5.2 SGS Minerals Testwork Program – 2017

SGS Minerals Lakefield received an estimated 500 tonnes of pre-screened fines (< 850 µm) from the NLI Whabouchi mini-DMS operation. The aim of this testwork program was to produce flotation concentrate for the electrochemical demonstration plant (P1P) at Shawinigan. Half of this material was processed using the flowsheet as shown in Figure 10-10.

Muscovite (Mica) was removed using Hydraulic separation (CrossFlow) and was screened. The screen oversize was considered mica waste and the undersize was reground to < 430 µm, deslimed and processed in wet magnetic separation. Before flotation, an attrition scrubbing stage was done on the material before a final desliming step to prepare the particles for the conditioning stage. The flotation was split in coarse hydroflotation (for + 0.18 – 0.43 mm) and fine conventional flotation (on – 0.18 mm) with separate conditioning stages for each.

a) Pilot Plant Test #1 to Test #17

This period was mainly commissioning of the equipment and process optimization. After a few extended continuous runs, it became clear that:

- CrossFlow separator feed had to be introduced with less energy;
- Magnetic separation was effective in removing residual iron rich particles;
- Flotation conditioning is critical;

- To achieve no fines in the coarse after fine screening was not possible;
- HydroFloat separation was going to be very sensitive to feed variations;
- Vacuum filtration of the concentrate produced a dry product by touch.

CrossFlow separation required a high percent solids and laminar flow into the separator. This was accomplished with the introduction of cyclones prior to CrossFlow separators. It became clear that the second separator was superfluous.

The magnetic separator mass pull is very low at 2.2%. However, the important aspect that needs to be monitored in the magnetic separation stage is to maximize rejection while preventing spodumene loss. Operating conditions can be adjusted to meet this criterion.

Hydro-Float separation was not successful at SGS, probably due to screening fines in the screen oversize. The finer material soaks up the reagents disproportionally and most finer material are unselectively removed during Hydro-Float separation resulting in poor grade and poor recovery.

The lithium losses in slimes are about 3.5%.

b) Pilot Plant Test #18 to Test #23

The Hydro-Float was replaced by mechanical cells and later a flotation column for cleaning. Very high grades up to 7.1% Li_2O were achieved using the flotation columns, however, the recovery grades were still low.

c) Pilot Plant Test #24 and Test #25

A hydraulic separator was introduced to remove the fines from the rougher flotation tailings of 0.86% Li_2O. The hydraulic separator underflow (1.16% Li_2O) was re-floated using the Hydro-Float separator after reconditioning. The Hydro-Float concentrate grade was only 3.55% Li_2O. The upgrading ratio of the Hydro-Float unit is limited. To produce an acceptable concentrate, it must be provided with a feed of sufficient grade. This test provided an upgrading ratio on 3 to 1.

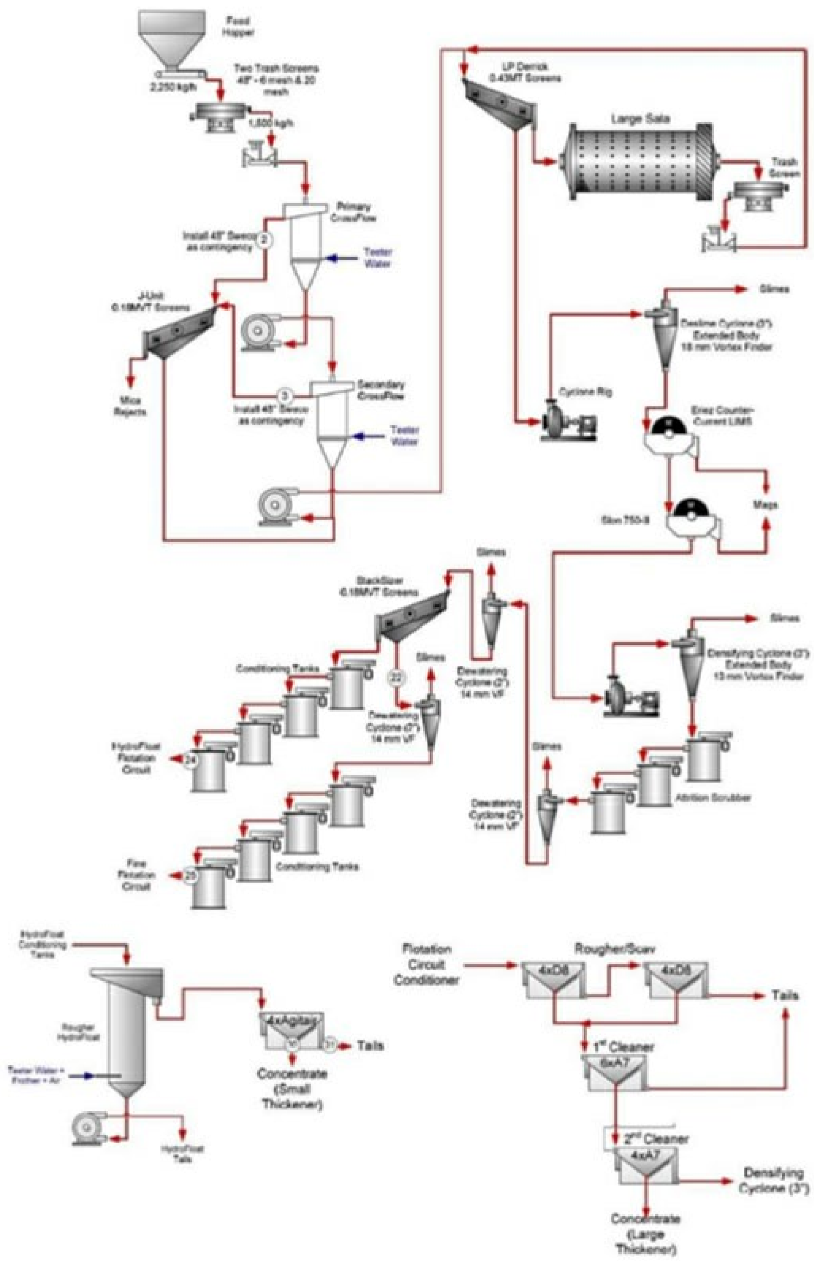
d) Pilot Plant Test #26 and Test #30

These tests involve the successful upgrading of earlier subpar concentrate grade to above 6% Li_2O. The old concentrate was re-conditioned and re-floated using mechanical cells.

e) Pilot Plant Test #31 and Test #40

These tests involve the use of mica flotation between grinding and magnetic separation and spodumene flotation using mechanical cells. A greater than 6% Li_2O concentrate was produced with a recovery above 80%. For these tests, all the ore was ground to less than 300 microns. The Lithium losses due to de-sliming was 6.8% or about double the amount compared when ground to 500 microns.

Figure 10-10 Pilot Plant Flowsheet Test #1 to Test #17



10.1.5.3 COREM Flotation Testwork – 2017

a) Bench Scale Flotation Testwork

COREM performed bench scale flotation test with the aim to determine the most influential flotation parameter. Four main parameters are collector dosage rate, flotation alkalinity, conditioning pulp density, flotation time. The results are presented in Table 10-15.

The results indicated that a minimum five (5) minutes of conditioning time is required. The other parameters are not statistically significant due to interactions. However, Test 22 delivered the best results, and these parameters are, therefore, assumed to be superior.

b) Pilot Plant Flotation Testwork

The pilot scale circuit was based on the proposed flowsheet and included: magnetic separation, attrition scrubbing, de-sliming, conditioning, and flotation. The flotation steps consist of rougher, scavenger and cleaner flotation using 3-inch flotation columns. The flotation feed sample was pre-screened to less than 212 microns.

Table 10-15 Main Flotation Parameters Test

Test	Collector (kg/t)	pH	Cond. Time (min)	Cond. Solids (%)	Weight Rec.[1]	Li_2O (%)	Li_2O Rec.2
18	3.6	9	5.0	50	13.3	6.01	72.8
19	2.4	9	5.0	50	19.7	5.34	92.4
20	1.2	9	5.0	50	9.4	5.77	50.7
21	2.4	10	5.0	50	17.3	5.45	86.9
22	2.4	8	5.0	50	16.9	5.96	91.8
23	2.4	9	5.0	70	18.3	5.42	90.4
24	2.4	9	5.0	60	19.3	5.34	92.2
25	2.4	9	5.0	40	17.2	5.60	87.1
26	2.4	9	2.5	50	13.4	5.70	70.7
27	2.4	9	1.0	50	8.6	5.62	43.5
28	2.4	9	2.5	70	11.5	5.81	56.6
29	2.4	9	1.0	70	9.9	5.92	55.8

[1] Weight recovery = concentrate weight / flotation feed weight × 100%.
[2] Li_2O recovery = weight of Li_2O in concentrate / weight of Li_2O in flotation feed × 100%.

Medium Intensity Magnetic Separation (MIMS) was performed on dry ore and was used to remove the residual ferrosilicon found in the provided sample which comes from the dense media separator circuit (mini-DMS operation 2017). A final magnetic separation was performed on the final concentrate.

In total three (3) pilot plant flotation tests were conducted using the flowsheet shown in Figure 10-11.

Table 10-16 is the pilot plant results. COREM endeavored to simulate the proposed fine flotation part of the flowsheet. The aim was to confirm the column flotation performance in the flowsheet. The rougher flotation time estimate and recoveries are listed in Table 10-16. The performance of the test done at COREM did not meet expectations. However, the flotation feed to these tests was less then 1% Li_2O which is an important factor in the flotation performance. Another important observation is that the reagent scheme differed significantly from the optimized parameters that were recently developed at SGS Lakefield and used by Eriez in the next Section.

Figure 10-11 General Process Flowsheet for COREM Column Flotation Pilot Plant

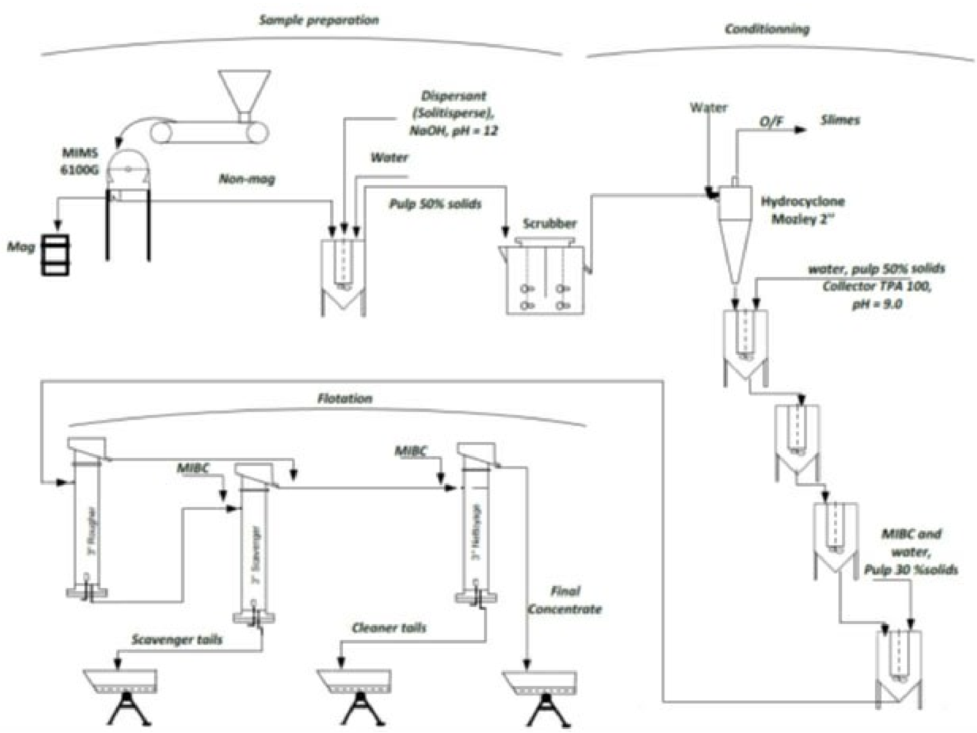


Table 10-16 Pilot Plant Flotation Test

Pilot Plant Test	Collector (kg/t)	pH	Cond. Time (min)	Rougher Float. Time (min)	Rougher Weight Rec.	Cleaner Li_2O (%)	Cleaner Li_2O Rec.
1	4.8	8	25	17.5	10.7	5.33	59.3
2	4.8	8	25	17.1	8.5	5.16	47.1
3	4.8	8	12	7.5	8.3	4.77	27.9

10.1.5.4 Eriez Flotation Testwork – 2017

Eriez Flotation Division (EFD) was provided with one (1) bulk bag of minus 850 microns (µm) low grade ore from SGS testwork program 2017, and one (1) bulk bag of DMS circuit middlings (D80 = 682 µm) from NLI to study the laboratory-scale split feed flotation response. This testwork was a continuation of recent pilot testing efforts performed at SGS-Lakefield. The patented Hydro-Float technology and flotation columns, inclusive of proprietary Eriez Cavitation-Tube sparging technology, were used to treat 497×212 µm and 212×27 µm size fractions, respectively. This split feed flotation approach provides the maximum separation efficiency.

a) Hydro-Float Tests

Coarse flotation tests were conducted on both an independent low-grade ore, as received from SGS, and a blend of DMS circuit middlings and fresh flotation feed. During treatment of the coarse size fraction using Eriez Hydro-Float fluidized bed flotation, optimal upgrade ratios of approximately 1.90-2.0 were achieved at Li_2O recoveries of 92-95%. A 5.8-6.0% Li_2O product was yielded at Li_2O recoveries of nearly 95-97% during treatment of a 3.07% Li_2O feed.

The Hydro-Float feed was nearly 40% passing 300 µm. Size-by-size assays of the Hydro-Float overflow indicate Li_2O grades of the plus 300 µm particle size fractions are greater than 6.3%. The lower grade concentrates are within the minus 300 µm size fractions. Although a coarse 6% Li_2O Hydro-Float concentrate is achievable without classification of the overflow, it is recommended that the circuit be designed such that the Hydro-Float concentrate can be scalped to remove fines floated unselectively and re-process them in a fine flotation circuit to improve global spodumene recovery. This is especially important if the feed grade decreases below 2.4% Li_2O, as demonstrated in preliminary coarse flotation testing.

b) Column Flotation Tests

Sixteen (16) rougher column flotation tests were conducted on a de-slimed 212 × 27 µm blend of SGS Lakefield and DMS samples at varying operating conditions. Optimal upgrade ratios of approximately 2.4 were achieved at Li_2O recoveries of 88.5-92.5%, as a concentrate grade of over 6.1% was achieved in rougher column flotation. A concentrate grade of 6.6% was realized at 88.2% Li_2O recovery and 34.4% concentrate mass yield in a rougher-cleaner open circuit.

Such results were ascertained following a 10-minute scrubbing period using 104 g/t NaOH pH modifier and 250 g/t soltisperse dispersant at 65% solids, by weight. In addition to scrubbing, the rougher and cleaner flotation feed were conditioned for 15 and 2 minutes, respectively, using a cumulative 146 g/t H_2SO_4 (return slurry to neutral pH), 32 g/t Na_2SiO_3, 350 g/t FA-2, and 150 g/t TP-100. Throughout rougher-cleaner column flotation testing, maximum rougher and cleaner carrying capacities of approximately 4.1 tph/m² and 3.9 tph/m², respectively, were achieved at a total circuit retention time of nearly 14.7 minutes (6.3 minutes in rougher and 8.4 minutes in cleaner).

The use of wash water in column flotation significantly improved the spodumene concentrate grade and flotation upgrade ratios. This is because wash water efficiently rids the froth of entrained gangue minerals such as quartz, mica, and feldspar etc., pushing those minerals back into the pulp phase. The use of a minimal sodium silicate dosage (32 g/t) also increased flotation selectivity throughout spodumene flotation testing. However, when added in excessive quantities, sodium silicate can render the froth brittle. As a result, careful monitoring of its addition is necessary to optimize the flotation performance.

10.1.6 Settling, Filtration and Freezing Tests

Various design tests have been done to size dewatering equipment and evaluate freezing of the final concentrate. Settling, filtration and freezing testwork was done, which can be used for the sizing of concentrate and tailings thickeners and filters.

10.1.6.1 Settling Testwork

SGS performed static settling test to determine the optimal feed densities and type of flocculant. Thereafter they performed dynamic thickening testing. The results for thickening tests are listed in Table 10-17. The most important tests involved spodumene concentrate and final tailings. The thickening hydraulic test data was used by suppliers for thickener sizing. The dynamic thickening obtained results with a spodumene concentrate thickener underflow between 61 and 63 weight percent solids. This was as expected. The final tailings underflow density was 61%. Both test samples had low turbidity overflows with total suspended solids (TSS) of ten (10) ppm.

Table 10-17 Dynamic Thickening Testwork Results

Stream Description	Flocculant Type	Flocculant addition rate (kg/t)	Feed Pulp (% w/w)	Underflow Pulp (% w/w)	Dynamic Settling Rate ($t/m^2/h$)
Spodumene Concentrate	Ciba Magnafloc 10	0.010	19	61 to 63	0.62
Combined Final Tailings	Ciba Magnafloc 10	0.028	17.8	61	0.5

10.1.6.2 Filtration Testwork

a) SGS Filtration Testwork on the Spodumene Concentrate and the Final Tailings

The vacuum filtration test had good results for the spodumene concentrate producing a cake of 8.3% moisture by weight at minus 0.4 g bar vacuum, while the lower vacuum of minus 0.7 g bar yielded a higher cake moisture content of 12.2%. The final tailings did not have similar results and ended up with wet cakes of 13.9% and 12.8% at minus 0.4 g bar and minus 0.7 g bar vacuum levels respectively.

The pressure filtration test produced good results for the spodumene concentrate with a cake of 7.7% moisture by weight at a pressure of 4.1 bar, while the higher pressure 6.9 bar yielded a higher cake moisture content of 9.1%. The final tailings had similar results and ended up with dry cakes of 6.2% and 6.6% at 4.1 bar and 6.9 bar respectively as shown in Table 10-18.

Table 10-18 Filtration Testwork Results

Description	Vacuum Filtration			Pressure Filtration		
	Throughput rate ($kg/m^2/h$)	Vacuum Level (barg)	Cake Moisture (% w/w)	Throughput rate ($kg/m^2/h$)	Pressure Level (bar)	Cake Moisture (% w/w)
Spodumene Concentrate	497	–0.4	8.3	561	4.1	7.7
	463	–0.7	12.2	541	6.9	9.1
Combined Final Tailings	425	–0.4	13.9	486	4.1	6.2
	486	–0.7	12.8	670	6.9	6.6

SGS reports that the spodumene concentrate filtration test results were as expected and were considered normal behaviour for fast settling coarse material.

b) Bokela Tailings Filtration Testing

Bokela performed six (6) filtration tests. Three (3) different filtration rates and two (2) different air flows were tested at two (2) differential pressures. The tailings were of fast sedimentation and high filtration speed, a pan filter with a filter area of 25 m² was recommended.

The pan filter is able to handle throughput of 37.8 t/h.

In Table 10-19, two (2) layout cases are shown for a higher moisture content (7,200 m³/h vacuum demand) and for a lower moisture content (13,000 m³/h).

Table 10-19 Tailing Filtration Test at 200 Micron

Item	Filter Area (m^2)	Filter Rate (dry t/h)	Diff. Pressure (bar)	Airflow (m^3/h)	Expected Moisture (%)	Guaranteed Moisture (%)
1	25	26.5	0.55 to 0.6.	13,000	13.5 to 14.0	
2	25	37.8	0.6	13,000	14.0	15.0
3	25	47.2	0.60 to 0.65	13,000	14.0 to 14.5	
4	25	26.5	0.35 to 0.40	7,200	14.5 to 15.2	
5	25	37.8	0.35 to 0.45	7,200	15.2	16.2
6	25	47.2	0.40 to 0.45	7,200	15.2 to 15.7	

10.1.6.3 Freeze Testwork

SGS performed two (2) freezing tests in 2012 on spodumene fine concentrates containing 3% and 5% moisture. SGS placed 750 grams samples on trays and exposed those -18°C temperature for 24 hours. Neither of the two (2) samples indicated major freezing problems as none of the samples froze as one solid block. The frozen 3% moisture sample contained many very small fragile lumps. The frozen 5% moisture sample had larger lumps, but these were also fragile crumbling on slight contact.

10.1.7 DMS Concentrate Drying Testing

ThermoPower received NLI Whabouchi DMS concentrate to perform drying testing in 2016. The aim was to find out the temperature at which the material would be free flowing.

10.1.7.1 Complete Drying of DMS Concentrate

To determine the residence time for completely drying a sample of DMS concentrate in an indirectly heated rotary tube furnace. It took an average of 7 minutes and 2 seconds for the product to reach a temperature of 110°C and thus to complete drying. 110°C is well above the boiling temperature of water (≈ 94°C) of the test location.

10.1.7.2 Drying DMS Concentrate at 80°C

The average residual moisture was 3.4% after drying. Figure 10-12 shows product that was cooled from 80°C to 60°C and then inspected for moisture and adhesion properties before being dried in the Convection Oven. The material was mixed and inspected by hand in the drying tray and can be seen to still stick to contact surfaces (Figure 10-12).

10.1.7.3 Drying DMS Concentrate at 85°C

The average residual moisture was 1.3% after drying at 85°C. This material is reasonably free flowing.

10.1.8 Magnetic Separation Test

10.1.8.1 Eriez Testwork Program Magnetic Separation

Magnetic separation was performed at Eriez in 2017. Eriez Central Test Laboratory received two (2) samples of spodumene concentrate containing hornblende from NLI for testing. Dry magnetic separation on Sample #1 was less than 3 mm, and wet magnetic separation on Sample #2 less than 0.85 mm. The objective of the testwork was to remove iron impurities from the Spodumene concentrate through both dry and wet magnetic separation processes. Later, another sample was sent to Eriez for Dry Magnetic Separation test optimization.

a) Dry Magnetic Separation

The coarser sample less than 3 mm feed was processed using dry magnetic separation equipment, while the other underwent wet magnetic separation.

The dry less than 3 mm feed was processed on a Ceramic Magnetic Drum Separator to remove any residual ferromagnetic debris present. A feed rate of 2 tonnes per hour per foot (t/h•ft–1) was used. This produced a Reject #1 and Non-Magnetic Product #1.

Figure 10-12 ThermoPower Drying Test #2, 80° Discharge Temperature

The Non-Magnetic #1 fraction was reprocessed on the Salient Pole-2 Rare Earth Magnetic Drum Separator The drum surface speed was set to a feed rate of approximately 2 t/h•ft–1, producing Reject #2 and corresponding Non-Magnetic Product.

Again, the non-magnetic fraction was reprocessed on a Rare Earth Magnetic Roll Separator. This part of the sample was processed in three steps; reprocessing the non-magnetic fraction each time. For the first step, the feed rate was set to 100 pounds per hour per inch of feed width (lb/h•in), the other two (2) steps were processed at 75 lb/h•in. The sample fractions of dry magnetic separation were weighed, and the percentages are presented in Table 10-20.

Table 10-20 Magnetic Separation Test

Stream	Dry Magnetic weight (%)	Wet Magnetic 1.0 T- weight (%)	Wet Magnetic 1.3 T- weight (%)
Feed	100.0	100.0	100.0
Reject #1	0.8	7.0	7.0
Non-magnetic #1	99.2	93.0	93.0
Reject #2	13.0	0.3	0.3
Non-magnetic #2	86.2	92.7	92.7
Reject #3, #4 & #5	15.8	32.3	37.9
Final non-magnetic	70.4	60.4	54.8

b) Wet Magnetic Separation

The less than 0.85 mm sample was first processed on the Low-Intensity Magnetic Separator (LIMS) to remove ferromagnetic debris. The sample was mixed with water to produce a slurry of approximately 20 percent solids and processed through the LIMS with a nominal field intensity of 950 Gauss.

The non-magnetic fraction from the LIMS was reprocessed through a Salient Pole Rare Earth Magnetic Wet Drum Separator (SP WDS) to remove additional weakly-ferromagnetic particles. The magnetic fraction was dried for mass yields.

Two (2) batches of the non-magnetic fraction from the SP WDS were reprocessed on the Wet High Intensity Magnetic Separator, (WHIMS). One batch was processed at a background field intensity of 1.0 Tesla and the other was processed at 1.3 Tesla. For both batches, a 2-mm rod matrix was used to collect the magnetic fraction, and low intensity jigging was used to agitate the slurry during processing.

c) Dry Magnetic Separation (second Eriez Test)

A new sample of DMS concentrate was provided to Eriez in early 2018. The objective was to optimize the dry magnetic separation for the coarse DMS material. Four (4) tests were done. The first three were open circuit test where a non-magnetic concentrate, a magnetic reject and a middling product were recovered. For the last test, the middling material was reprocessed, and a non-magnetic concentrate was recovered and added to the first stage non-magnetic product. The second stage magnetic material was rejected with the magnetic tailings of the first stage. Separation test percentages are presented in Table 10-21.

After analyzing the results and discussing with Eriez, it was determined that the best separation arrangement is to perform a first separation stage that produces a final non-magnetic concentrate and an intermediate product that will be reprocessed in a second stage. The second stage non magnetic material is combined with the first stage concentrate and the magnetic product is the final tailings.

Table 10-21 Dry Magnetic Separation Test (Second Eriez Test)

	Test 1			Test 2			Test 3			Test 4		
	Wt%	Li_2O (%)	Rec (%)	Wt%	Li_2O (%)	Rec (%)	Wt%	Li_2O (%)	Rec (%)	Wt%	Li_2O (%)	Rec (%)
Feed	100.0	4.58	100	100.0	4.76	100	100.1	3.92	100	100.0	5.69	100
Concentrate	70.1	5.96	91.3	71.7	5.98	90.0	63.8	5.91	96.2	94.1	5.98	98.9
Middlings	8.0	4.11	7.2	7.5	5.42	8.5	9.3	1.05	2.5	1.6	3.20	0.9
Tailings (magnetics)	21.9	0.32	1.5	20.8	0.34	1.5	27.0	0.19	1.3	4.3	0.28	0.2
							Stage 2 magnetics			0.78	0.82	0.1
							Stage 2 concentrate			0.79	5.68	0.79
							Combined concentrate			94.9	5.98	99.7

10.1.8.2 PhySep Dry Magnetic Separation Testing

Four (4) tests were conducted on August 16, 2017. An impact of the feed temperature, and feed rate on the separator performance were evaluated. To prorate the results, the laboratory vibrating feeder feed rate was then converted to equivalent feed rate for the industrial size unit based on the lab unit feed rate.

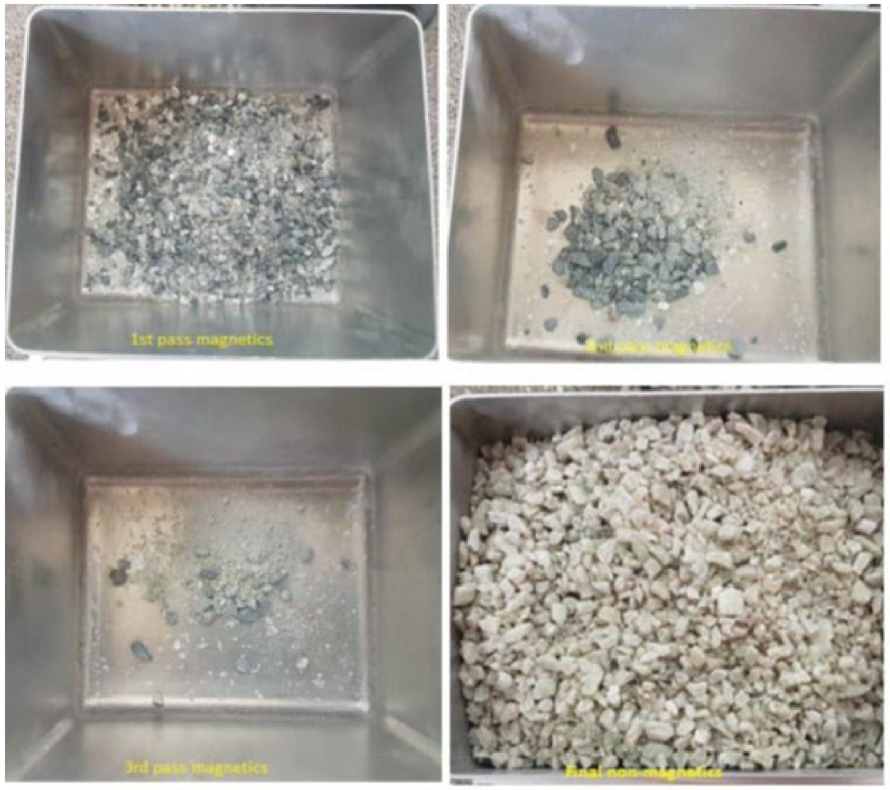
Test #1 was conducted at the ambient temperature of 27°C (in the lab), and Test #2, Test #3, and Test #4 modelled the actual temperatures of the dryer discharge around 100°C. For three (3) latter tests the feed material was preheated to about 120-130°C in the oven, and then fed to the feed hopper of the magnetic separator test unit.

The non-magnetics of the Tests # 2, 3, and 4 were collected and re-heated to about 100°C at magnetic separator feed. After the third (and final) pass magnetic reject and non-magnetics were cooled, weighed, and packaged for assaying elsewhere.

The least rejects occurred in Test #3 with a feed rate of five (5) t/h and a weight recovery of 93.9%. There are visually no dark minerals in the non-magnetic stream (Figure 10-13).

The assay results showed that the higher temperature of material did not negatively impact the separation performance. However, suppliers indicated it could deteriorate the magnets properties and some cooling should be applied to operate under 85°C. The higher processing rate resulted in a lower separation performance (less contaminant rejection), but the separation does not need to be perfect as long as the majority of hornblende is removed to prevent issues in the roasting process. Lithium losses must be minimized.

Figure 10-13 Hot Dry Magnetic Separation Test #3, 99°C Feed Temperature



The technical results are listed in Table 10-22.

Table 10-22 Dry Magnetic Separation Results

Stream	Test #1	Test #2	Test #3	Test #4
Feed temperature, ºC	27	100	99	99
Feed rate, t/h	2.5	2.5	5.0	5.0
1st pass mag rejects	8.1%	6.6%	4.2%	6.9%
2nd pass mag rejects	1.4%	1.1%	1.0%	2.5%
3rd pass mag rejects	0.6%	0.7%	0.8%	2.0%
Total magnet rejects	10.1%	8.5%	6.1%	11.4%
Final non-magnetics	89.9%	91.5%	93.9%	88.6%

10.1.9 Recent Metallurgical Testing (2019-2022)

As part of the process review initiated by the new owners of the Project, several new testwork programs were conducted. This work included a bulk concentrate production run and testing on composites representing the first five years of the mine plan, as well as some additional hydroflotation work, coagulant testing, saponification testing, and concentrate freezing testing.

10.1.9.1 Sample Selection

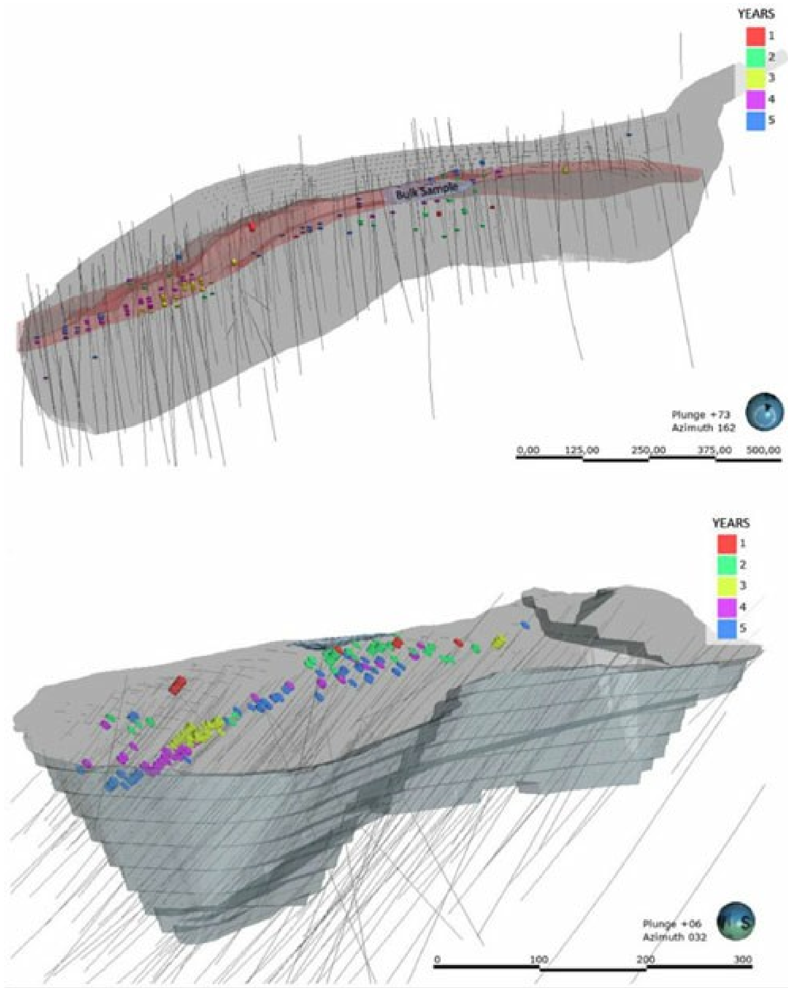
a) Selection of Composite Samples for SGS Testing

The sample selection was performed by SGS Geostat Inc. to generate five (5) composite samples which were statistically representative of each of the first 5 years of operation in the mining plan prepared in 2019. A total of 246 core reject samples were selected to create the five (5) composite samples (CRS1 to CRS5). Individual samples were selected to be statistically representative in terms of Li_2O grade, geological domains, and rock type. Table 10-23 compares the assayed Li_2O grade in each composite sample to the statistically established target grade and Figure 10-14 shows the spatial distribution of selected samples throughout the deposit as well as the location of the bulk sample used for previous testing.

Table 10-23 Composite Samples Representing First 5 Years of Operation

Composite Sample	Sample Name	Target %Li_2O Grade	%Li_2O Grade of the Composite Sample	Number of Samples Selected	Weight of Composite Sample (kg)
Year 1	CRS1	1.50	1.65	22	50.4
Year 2	CRS2	1.67	1.62	49	115.2
Year 3	CRS3	1.72	1.67	56	104.3
Year 4	CRS4	1.73	1.62	53	100.4
Year 5	CRS5	1.56	1.50	66	128.1

Figure 10-14 Spatial Distribution of Selected Samples for SGS Testing



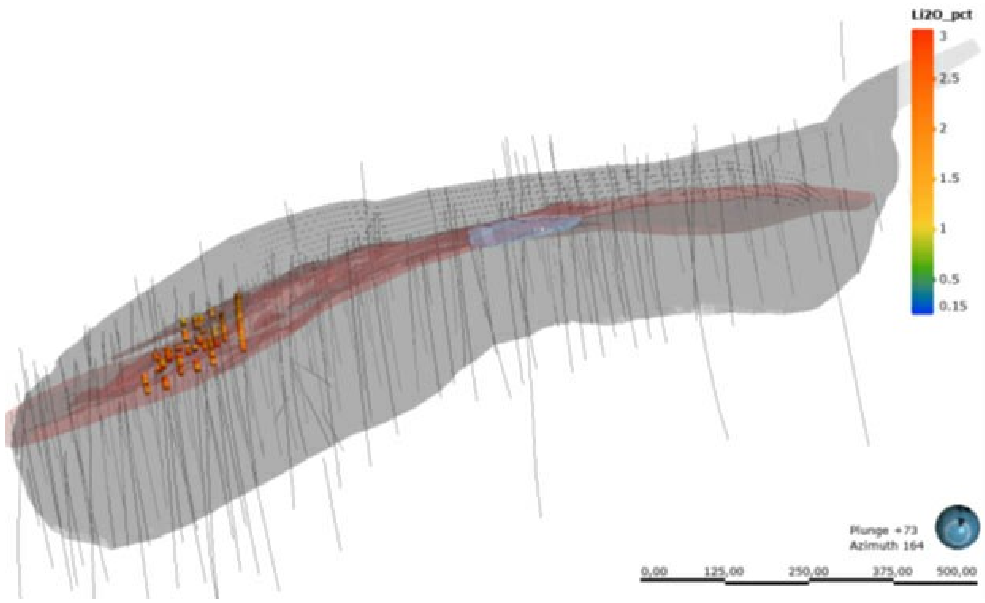
b) Selection of Samples for Eriez HydroFloat Testwork

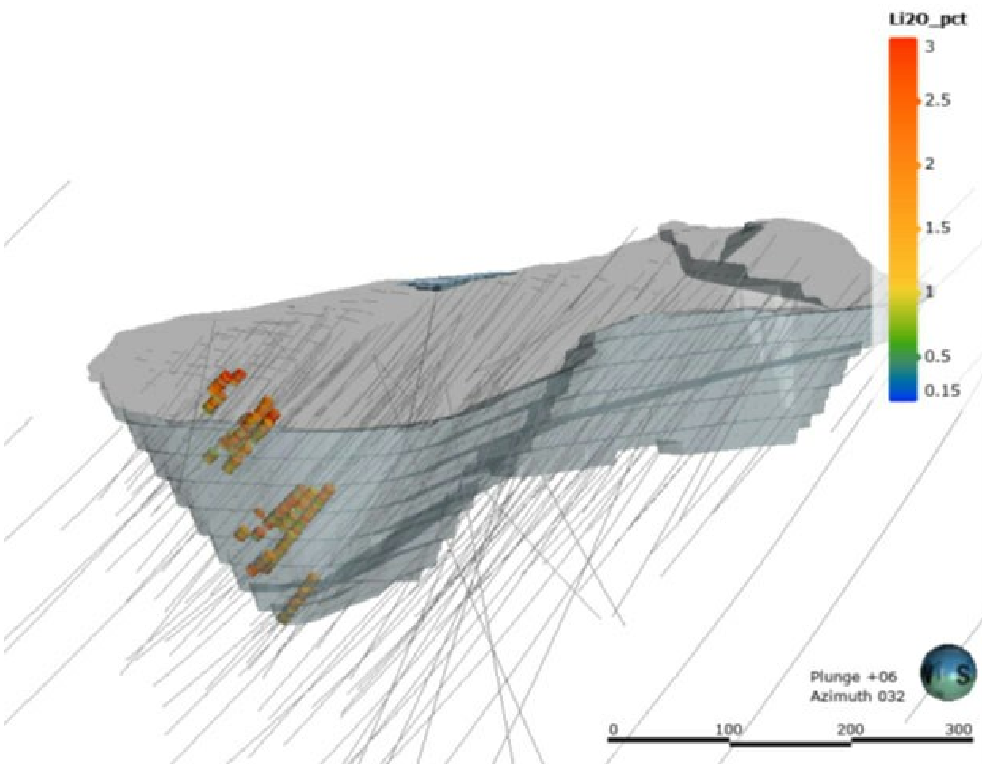
The sample selection was done by the NLI geology team with the main objective being a target Li_2O grade representative of the average expected feed grade at the HydroFloat circuit. The sample selection was also restrained to the West side of the deposit (away from the Bulk Sample). This was done to ensure a better representativity span across the deposit, since most of the previous historic testwork was done on ore from the Bulk Sample. A total of 506 core reject samples were selected to create the composite sample. Table 10-24 compares the obtained Li_2O grade in the composite sample to the established target grade and Figure 10-15 shows the spatial distribution of selected samples throughout the deposit.

Table 10-24 HydroFloat Testwork Composite Sample

Sample	Target %Li_2O Grade	%Li_2O Grade of Composite Sample	Number of Samples Selected	Weight of Composite Sample (kg)
HydroFloat Testwork Composite	1.80	1.77	506	~700

Figure 10-15 Spatial Distribution of Selected Samples for the HydroFloat Testwork Composite





10.1.9.2 SGS Testwork Program (2021-2022) – Production of 5.5 % Li_2O Concentrates

In 2021-2022, SGS' laboratory in Quebec City, Quebec conducted a testwork program to produce five (5) concentrates at 5.5% Li_2O based on samples representing the first five years of the mine plan. The main objective was to produce representative concentrates for downstream hydrometallurgical testing, with the secondary objective of determining the distribution and concentration of spodumene and gangue minerals throughout the process. Li_2O contents in the final concentrates for the five (5) tests ranged from 5.29 % to 5.64 % thus reaching the objective of producing representative concentrates for downstream conversion testwork.

The global lithium recovery for these tests ranged from 52.8% to 78.9%. Since the objective of these tests did not pertain to recovery, the operating parameters were not optimized through iterative testwork to reach target recoveries. These tests did not include the ultracoarse HydroFloat, which was later added to improve flotation recovery (refer to Section 10.1.9.3 for additional details on the ultracoarse HydroFloat testwork).

The tests performed included coarse muscovite removal; heavy liquid separation; dry high intensity magnetic separation; middlings stage grinding; ball mill circuit; fine muscovite removal; dry magnetic separation; attrition scrubbing, desliming and classification; coarse flotation; and fines flotation.

a) Sample Preparation and Characterization

Five (5) samples, each of approximately 100 kg, were prepared for the testwork which were representative of the first five (5) years of the 2019 mine plan / feed material to the Whabouchi plant. The sample selection for this program is described in Section 10.1.9.1.

Samples had an initial size of 100% passing 3-4 mm. Each sample was homogenized, subsampled, sent for analysis, and stage crushed to 100% passing 1.7 mm. The head assay characterization was performed via semi quantitative XRD analysis and Tescan Integrated Mineral Analyzer (TIMA-X) analysis to estimate the chemical compounds present within the assay.

The XRD results showed the samples ranged in mineral content by weight as follows:

- 17.8 - 20.8% for spodumene;
- 34.8 - 41.8% for quartz;
- 20.4 - 25.3% for albite;
- 5.2 – 11.7% for microline;
- 6 - 9.9% for muscovite;
- - 3.3% for magnesiohornblende.

TIMA-X confirmed good liberation of spodumene from the quartz and feldspars. The TIMA-X results showed the following ranges in the samples:

- 18.8 - 24.1% for spodumene;
- 28.1 - 31.0% for quartz;
- 2.53 - 9.97% for muscovite;
- 0.34 - 1.69% for petalite.

Prior to testing, the five (5) crushed samples were screened at 850 μm into two (2) size fractions (+850 and -850 μm). These size fractions were assayed for each sample and the results are presented in Table 10-25 while the distribution is presented in Table 10-26.

Table 10-25 Head Screened Fractions Analysis Summary by Concentration

Description		Weight		Assays															
				Li_2O	F	Cl	Hg	S	C	Al	Ca	Fe	K	Na	Mg	Mn	P	Ti	Zn
Test #	Product	kg	%	%	%	g/t	g/t	%	%	%	%	%	%	%	%	ppm	%	%	ppm
CRS1	+850um	14.3	44.6	1.94	0.02	83	< 0.3	0.01	0.01	8.54	0.40	0.98	2.50	1.82	0.13	676	0.05	0.02	56
	-850um	17.7	55.4	1.31	0.02	40	< 0.3	0.01	0.02	7.98	0.40	0.81	2.60	2.43	0.11	734	0.07	0.02	64
	Head Calc.	32.0	100	1.59	0.02	59	< 0.3	< 0.01	0.02	8.23	0.40	0.89	2.56	2.16	0.12	708	0.06	0.02	60
	Head Direct			1.51	0.02	47	< 0.3	< 0.01	< 0.01	7.78	0.30	0.83	2.40	2.16	0.13	651	0.05	0.02	63
CRS2	+850um	30.1	53.1	1.96	0.03	71	< 0.3	0.01	< 0.01	8.00	0.40	0.85	1.90	2.08	0.12	640	0.06	0.02	71
	-850um	26.6	46.9	1.40	0.02	77	< 0.3	0.01	0.01	7.90	0.40	0.9	1.80	2.72	0.13	914	0.07	0.03	86
	Head Calc.	56.6	100	1.70	0.03	74	< 0.3	0.01	< 0.01	7.95	0.40	0.85	1.85	2.38	0.12	769	0.06	0.02	78
	Head Direct			1.77	0.03	71	< 0.3	0.01	0.01	7.69	0.40	0.83	1.70	2.47	0.14	688	0.07	0.02	93
CRS3	+850um	27.0	40.0	2.00	0.03	64	< 0.3	< 0.01	0.01	8.61	0.40	0.92	2.10	2.18	0.15	639	0.08	0.02	74
	-850um	40.6	60.0	1.40	0.03	33	< 0.3	< 0.01	0.01	7.88	0.40	0.70	2.20	2.51	0.14	727	0.09	0.02	90
	Head Calc.	67.6	100	1.64	0.03	45	< 0.3	0.01	0.01	8.17	0.40	0.79	2.16	2.38	0.14	692	0.09	0.02	84
	Head Direct			1.55	0.04	58	< 0.3	< 0.01	0.02	6.97	0.40	0.64	1.90	2.40	0.12	565	0.09	0.02	65
CRS4	+850um	31.7	44.9	1.94	0.02	46	< 0.3	< 0.01	0.01	8.59	0.40	0.83	2.20	2.29	0.13	661	0.06	0.02	70
	-850um	38.9	55.1	1.36	0.02	42	< 0.3	< 0.01	0.02	7.76	0.40	0.76	2.10	2.60	0.13	756	0.07	0.02	92
	Head Calc.	70.6	100	1.62	0.02	44	< 0.3	< 0.01	0.02	8.13	0.40	0.79	2.14	2.46	0.13	713	0.07	0.02	82
	Head Direct			1.55	0.03	61	< 0.3	< 0.01	0.02	7.95	0.40	0.78	2.30	2.44	0.14	770	0.08	0.02	81
CRS5	+850um	42.9	48.4	1.72	0.03	77	<0.3	0.03	0.01	8.45	0.70	1.14	2.00	2.40	0.25	641	0.05	0.04	80
	-850um	45.6	51.6	1.27	0.03	77	<0.3	0.02	0.02	7.58	0.50	0.89	1.70	2.77	0.18	802	0.07	0.03	87
	Head Calc.	88.5	100	1.49	0.03	77	< 0.3	0.02	0.02	8.00	0.60	1.01	1.85	2.59	0.21	724	0.06	0.03	84
	Head Direct			1.44	0.03	62	<0.3	0.02	0.01	8.06	0.50	0.94	1.90	2.58	0.19	784	0.07	0.03	91

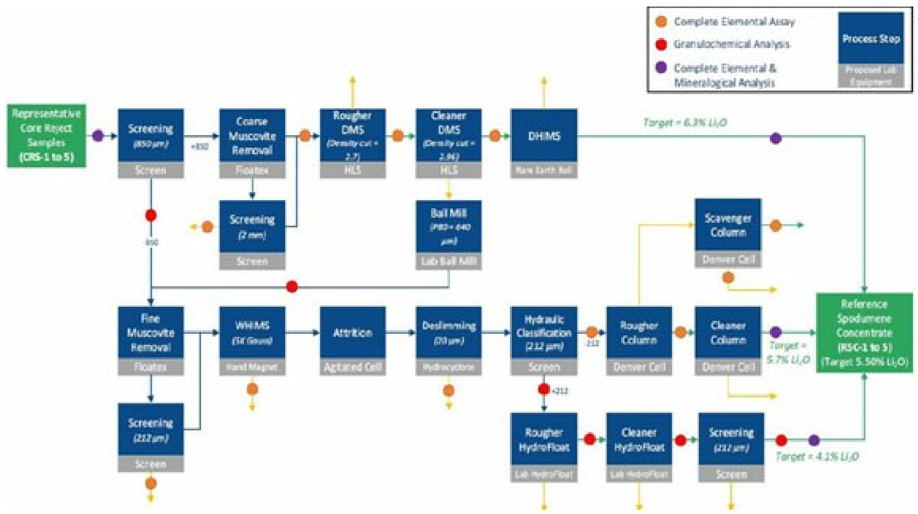
Table 10-26 Head Screened Fractions Analysis Summary by Distribution

Description		Distribution (%)															
Test #	Product	Li_2O	F	Cl	Hg	S	C	Al	Ca	Fe	K	Na	Mg	Mn	P	Ti	Zn
CRS1	+850um	54.3	44.6	62.5	44.6	44.6	28.7	46.3	44.6	49.3	43.6	37.7	48.7	42.6	36.5	44.6	41.3
	-850um	45.7	55.4	37.5	55.4	55.4	71.3	53.7	55.4	50.7	56.4	62.3	51.3	57.4	63.5	55.4	58.7
	Head Calc.	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
CRS2	+850um	61.3	59.4	51.1	53.1	53.1	53.1	53.4	53.1	53.1	54.4	46.4	51.1	44.2	49.2	43.0	48.3
	-850um	38.7	40.6	48.9	46.9	46.9	46.9	46.60	46.9	46.9	45.6	53.6	48.9	55.8	50.8	57.0	51.7
	Head Calc.	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
CRS3	+850um	48.8	40.0	56.4	40.0	40.0	40.0	42.1	40.0	46.7	38.9	36.7	41.6	36.9	37.2	40.0	35.4
	-850um	51.2	60.0	43.6	60.0	60.0	60.0	57.9	60.0	53.3	61.1	63.3	58.4	63.1	62.8	60.0	64.6
	Head Calc.	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
CRS4	+850um	53.8	41.7	47.2	44.9	44.9	29.0	47.5	44.9	47.1	46.1	41.8	44.9	41.6	41.2	44.9	38.3
	-850um	46.2	58.3	52.8	55.1	55.1	71.0	52.5	55.1	52.9	53.9	58.2	55.1	58.4	58.8	55.1	61.7
	Head Calc.	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
CRS5	+850um	56.0	46.1	48.4	48.4	58.5	32.0	51.2	56.8	54.6	52.5	44.9	56.6	42.9	40.2	55.6	46.3
	-850um	44.0	53.9	51.6	51.6	41.5	68.0	48.8	43.2	45.4	47.5	55.1	43.4	57.1	59.8	44.4	53.7
	Head Calc.	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

b) Testwork Program Description

An overview of the testwork program is shown in Figure 10-16. The program included coarse muscovite removal; heavy liquid separation; dry high intensity magnetic separation; middlings stage grinding; ball mill circuit; fine muscovite removal; dry magnetic separation; attrition, desliming and classification; coarse flotation; and fines flotation. The individual unit operations are described in the subsections below.

Figure 10-16 Whabouchi Concentrator Flowsheet



Coarse Muscovite Removal

Coarse muscovite removal testing was done using the Eriez HydroFloat acting as a hydraulic separator. Due to the shape of the muscovite flakes, they are concentrated in the overflow along with some of the fines. The material was passed through the HydroFloat twice to ensure a good separation. The overflow, containing mostly muscovite, was then screened at a particle size of 850 µm and the coarse fraction was sent to tailings. The HydroFloat underflow and screen undersize were combined and used for heavy liquid separation/DMS testing. Overall, the test showed minimal lithium losses ranging from 0.1% and 0.3%, within approximately 1% of the mass that was sent to tailings (muscovite concentrate).

Heavy Liquid Separation

After muscovite removal, the coarse fraction was processed through rougher and cleaner DMS stages using Heavy Liquid Separation (HLS). For the rougher stage, the material was separated at a specific gravity (SG) of 2.7 g/cm³ using methylene iodide diluted with acetone. The sink fraction (heavy fraction) was then processed again in the cleaning stage at a SG of 2.96 g/cm³. The cleaner sinks, or DMS concentrate, were sent to the dry high intensity magnetic separation (DHIMS). This stream reported Li_2O in the range of 5.06% - 5.91%, recovering between 47.3% and 70.2% of the lithium, with iron as the main impurity. The rougher float is mostly silicate gangue material and is sent to tailings, representing lithium losses ranging from 4.3% to 13.8%.

Dry High Intensity Magnetic Separation (DHIMS)

DHIMS testing was performed using a rare earth rolls magnetic separator operating at 5,000 gauss. This stage was used to remove 93.9 % of the iron and 61.4 % of the magnesium content in the feed, while maintaining a 99.1 % lithium recovery. The five (5) DHIMS products produced a lithium grade of >6.4 % Li_2O, thereby meeting the target of 6.3 % Li_2O.

Ball Mill Circuit

The DMS middlings (cleaner floats) were each subjected to a Bond Ball Mill grindability test (BWi), and also were stage ground to 100% passing 1 mm and 80% passing 640 µm in order to minimise the generation of fines.

The BWi grindability test was performed at a closing screen size of 28 mesh with a 2 kg charge. The average results show that the material has a BWi of 10.7 kWh/t and is characterized as soft ore. This value is slightly softer than the design criteria value of 13.2 kWh/t.

Fine Muscovite Removal

The -850 µm feed fraction was combined with the ball mill product and fed to the fine muscovite removal tests. As with the coarse muscovite removal tests, the fine muscovite removal was performed using the Eriez HydroFloat acting as a hydraulic separator. The material was passed though the hydraulic separator twice to ensure a good separation. The overflow, containing mostly muscovite, was screened at 212 µm, and the coarse fraction was sent to tailings. The hydraulic separator underflow and screen undersize were combined and were sent to magnetic separation. Overall, the fine muscovite removal had a stage recovery of 98.5% to 99.8% lithium.

Magnetic Separation

The magnetic separation at this stage of the flowsheet is typically performed with wet magnetic separation. SGS Quebec facilities did not have the proper equipment at their disposal, therefore, the material was dried and treated with a dry magnetic separator at 5,000 gauss. The purpose of this stage was to remove the iron impurities while minimizing lithium losses. This step removed 15.0% to 22.6% of iron and lost 1.3% to 1.8% of lithium.

Attrition, Desliming, and Screening

Attrition and desliming was performed to loosen and remove particles finer than 20 µm. The magnetic separation product was mixed in the attrition scrubber with sodium hydroxide and sodium silicate at high energy intensity. The slimes were removed using syphoning and the deslimed material was screened at 212 µm.

Coarse Flotation

Coarse flotation testing was performed with the Eriez HydroFloat cell and was done with the particles greater than 212 µm screened in the attrition and desliming circuit. A rougher and cleaner stage was performed in order to produce a coarse flotation concentrate. The conditioning and flotation parameters are included in the test report. The coarse flotation concentrate contained 4.03% to 4.54% Li_2O which was near the 4.1% Li_2O target.

Fines Flotation Process

Fines flotation process testing was performed with a rougher, rougher-scavenger, and a cleaner column. This test was done with the particles finer than 212 µm (excluding slimes). The conditioning and flotation parameters are included in the test report. The cleaner flotation concentrate contained 5.30% to 5.75% Li_2O grade which is near the 5.7% Li_2O target.

Results and Recommendations

The final spodumene concentrate is composed of three (3) streams: DMS-DHIMS concentrate, cleaner column flotation concentrated (Flotation Conc.), and cleaner HydroFloat concentrate (Hydrofloat conc.). Chemical analysis on each of the three (3) concentrate streams for all five (5) samples was performed and is summarized in Table 10-27. These results showed the overall Li_2O grades to range from 5.29% to 5.64%, meeting the objective of 5.50%. Semi-quantitative XRD analysis was also performed on the concentrated and the results are shown in Table 10-28.

Mineralogical analysis was also performed on a composite prepared with the tailings streams using TIMA-X. This analysis gave an indication of free and liberated spodumene particles within the tailings. The liberation of spodumene was shown to be high (88.4% to 93.8%) for the fine particles (-212 μm) and low (4.30 to 28.5) in the coarse particles (+850 um). Liberated particles in the -212 μm range could be recovered by improving flotation conditions.

Table 10-27 Final Concentrate Chemical Analyses Summary

Sample ID	Product	Li_2O (%)	SiO_2 (%)	Al_2O_3 (%)	Fe_2O_3 (%)	MgO (%)	CaO (%)	Na_2O (%)	K_2O (%)	TiO_2 (%)	P_2O_5 (%)	MnO (%)	Cr_2O_3 (%)	V_2O_5 (%)	LOI (%)	Sum (%)
CRS1	DMS - DHIMS Conc.	6.41	66.3	24.3	1.43	0.05	0.09	0.38	0.23	<0.01	0.05	0.13	0.03	<0.01	0.36	93.3
	Flotation Conc.	5.43	58.7	25.2	3.53	1.02	2.24	0.59	1.15	0.14	0.80	0.24	<0.01	0.01	1.10	94.7
	Hydrofloat Conc	4.54	67.1	21.9	1.37	0.11	0.42	1.52	1.66	0.02	0.18	0.08	<0.01	<0.01	0.94	95.3
	Total	**5.64**	**64.6**	**23.9**	**1.95**	**0.31**	**0.73**	**0.75**	**0.86**	**0.04**	**0.28**	**0.14**	**0.01**	**<0.01**		
CRS2	DMS - DHIMS Conc.	6.59	66.7	24.1	1.59	0.05	0.13	0.46	0.23	<0.01	0.04	0.13	0.04	<0.01	0.32	93.7
	Flotation Conc.	5.30	58.5	23.7	3.89	1.10	2.81	0.60	0.89	0.17	0.83	0.29	<0.01	<0.01	1.17	94.0
	Hydrofloat Conc	4.03	68.6	21.2	1.55	0.12	0.44	1.99	1.56	0.01	0.15	0.09	0.05	<0.01	0.61	96.3
	Total	**5.51**	**65.1**	**23.2**	**2.18**	**0.34**	**0.92**	**0.94**	**0.79**	**0.05**	**0.28**	**0.16**	**0.03**	**<0.01**	**Total**	
CRS3	DMS - DHIMS Conc.	6.46	67.5	23.5	1.43	0.08	0.20	0.41	0.27	<0.01	0.07	0.14	0.04	<0.01	0.38	93.9
	Flotation Conc.	4.74	58.9	24.5	2.73	0.60	2.46	0.43	1.13	0.07	1.29	0.24	0.03	<0.01	1.56	94.0
	Hydrofloat Conc	4.10	68.7	21.1	1.45	0.06	0.41	1.52	1.60	0.01	0.21	0.08	0.06	<0.01	0.83	96.0
	Total	**5.29**	**65.7**	**23**	**1.76**	**0.21**	**0.84**	**0.76**	**0.9**	**0.02**	**0.42**	**0.15**	**0.04**	**<0.01**		
CRS4	DMS - DHIMS Conc.	6.44	67.1	23.7	1.53	0.04	0.13	0.40	0.25	<0.01	0.04	0.14	0.04	<0.01	0.28	93.6
	Flotation Conc.	5.59	60.5	24.4	2.68	0.68	1.69	0.50	1.02	0.09	0.67	0.27	0.02	<0.01	1.50	94.0
	Hydrofloat Conc	4.24	68.2	21.1	1.79	0.06	0.26	1.56	1.57	<0.01	0.13	0.09	0.07	<0.01	0.55	95.4
	Total	**5.58**	**65.61**	**23.16**	**1.92**	**0.22**	**0.59**	**0.75**	**0.83**	**0.02**	**0.24**	**0.16**	**0.04**	**<0.01**		
CRS5	DMS - DHIMS Conc.	6.52	66.9	23.7	1.39	0.05	0.17	0.41	0.25	<0.01	0.04	0.12	0.04	<0.01	0.32	93.4

Sample ID	Product	Li_2O (%)	SiO_2 (%)	Al_2O_3 (%)	Fe_2O_3 (%)	MgO (%)	CaO (%)	Na_2O (%)	K_2O (%)	TiO_2 (%)	P_2O_5 (%)	MnO (%)	Cr_2O_3 (%)	V_2O_5 (%)	LOI (%)	Sum (%)
	Flotation Conc.	5.75	59.8	23.9	2.79	0.61	2.38	0.54	0.69	0.09	0.93	0.26	0.06	0.01	1.76	93.9
	Hydrofloat Conc	4.12	66.2	22.5	1.49	0.10	0.51	1.49	1.98	0.01	0.20	0.09	0.02	<0.01	1.25	95.9
	Total	**5.52**	**64.7**	**23.36**	**1.81**	**0.22**	**0.89**	**0.8**	**0.94**	**0.03**	**0.34**	**0.15**	**0.04**	**<0.01**		

Table 10-28 Semi-Quantitative XRD Analysis of the Expected Combined Concentrate

Sample ID	Product	Spodumene (wt%)	Quartz (wt%)	Albite (wt%)	Muscovite (wt%)	Magnesio hornblende (wt%)	Microcline (wt%)	Fluorapatite (wt%)	Siderite (wt%)	Holmquistite (wt%)	Chamosite (wt%)	Pyrrhotite (wt%)	Diopside (wt%)
CRS1	DMS-DHIMS Conc.	81.5	13.7	3.4	-	-	1.3	0.1	-	-	-	-	-
	Flotation Conc.	63.9	6.0	5.2	9.8	8.6	0.0	1.8	2.1	1.2	0.8	0.3	0.2
	HydroFloat Conc	53.0	18.4	14.1	13.6	-	0.0	0.4	0.4	-	-	-	-
	Total	**69.1**	**13.1**	**6.8**	**6.3**	**2.2**	**0.6**	**0.6**	**0.6**	**0.3**	**0.2**	**0.1**	**0.1**
CRS2	DMS-DHIMS Conc.	77.3	17	4.2	-	-	1.4	0.1	-	-	-	-	-
	Flotation Conc.	62	7.8	6.1	7.5	10.5	0.0	2.0	1.9	1.0	0.4	0.3	0.1
	HydroFloat Conc	48.1	21	18.1	9.6	-	2.2	0.3	0.2	-	-	-	-
	Total	**64.9**	**15.8**	**8.7**	**4.7**	**2.7**	**1.3**	**0.7**	**0.6**	**0.3**	**0.1**	**0.1**	**0.0**
CRS3	DMS - DHIMS Conc.	76.6	17.5	4.1	-	-	1.6	0.2	-	-	-	-	-
	Flotation Conc.	67.6	7.6	4.0	9.6	4.6	0.0	3.1	2.1	-	1.3	0.1	-
	HydroFloat Conc	49.9	21.5	14	13	-	0.7	0.5	0.3	-	-	-	-
	Total	**66.0**	**16.3**	**7.2**	**6.5**	**1.2**	**0.9**	**1.0**	**0.6**	**0.0**	**0.3**	**0.0**	**0.0**
CRS4	DMS-DHIMS Conc.	76.5	18.2	3.8	-	-	1.5	0.1	-	-	-	-	-
	Flotation Conc.	66	11.1	4.1	8.2	5.3	0.2	1.6	1.8	0.5	0.8	0.1	0.3
	HydroFloat Conc	52.5	21.2	13.4	9.5	-	2.6	0.3	0.2	-	-	0.1	-
	Total	**66.9**	**17.1**	**6.6**	**4.9**	**1.4**	**1.5**	**0.6**	**0.5**	**0.1**	**0.2**	**0.1**	**0.1**
CRS5	DMS-DHIMS Conc.	77	17.4	4	-	-	1.4	0.1	-	-	-	-	-
	Flotation Conc.	69.3	9.2	5.0	5.4	5.9	0.4	2.2	1.4	-	0.7	0.1	-
	HydroFloat Conc	50.7	19.7	13.9	8.9	-	5.7	0.5	0.4	-	-	0.1	-

Sample ID	Product	Spodumene (wt%)	Quartz (wt%)	Albite (wt%)	Muscovite (wt%)	Magnesio hornblende (wt%)	Microcline (wt%)	Fluorapatite (wt%)	Siderite (wt%)	Holmquistite (wt%)	Chamosite (wt%)	Pyrrhotite (wt%)	Diopside (wt%)
	Total	66.3	15.9	7.5	4.4	1.6	2.5	0.8	0.5	0.0	0.2	0.1	0.0

10.1.9.3 Eriez Testwork Program Hydrofloat Flotation for Coarse Flotation Circuit (2022)

In 2022, a split-feed hydroflotation test program was carried out at Eriez Flotation in Erie, PA. The purpose of this campaign was to investigate whether improved lithium grade and recovery performance could be achieved by treating a single composite as two (2) distinct split feeds (coarse and fine) instead of a single feed. Three (3) feedstocks were produced for this testing campaign based on the same composite: 850 x 212 µm, 850 x 500 µm, and 500 x 212 µm. Historically, all hydroflotation testing has been performed on a single size range of 500 x 212 µm and did not evaluate the performance up to the design size of 850 µm. Prior to conducting coarse particle flotation on Eriez' 6" HydroFloat, each feedstock was processed on an Eriez' 2x8" laboratory CrossFlow to reject mica minerals.

a) Sample Preparation

The sample selection for this program is described in Section 10.1.9.1. The samples were received at Eriez, 12.5% was split out through a rotary feeder and returned to NLI, while 87.5% was used for HydroFloat testing. Ore (600 kg) was classified via wet screening at 850 µm, and screen oversize was milled to a P98 of 850 µm. Minus 850 µm material was wet screened at 212 µm. The material with a particle size of minus 212 µm (290 kg) was set aside and not used for this campaign.

A 70 kg sample of 850 x 212 µm was removed for single feed testing. The remaining 850 x 212 µm material was wet screened at 500 µm to produce feed material for split-feed testing: 97.3 kg of 850 x 500 µm material and 156 kg of 500 x 212 µm material.

b) CrossFlow Testing

Prior to HydroFloat testing, each feed material was treated independently using an Eriez CrossFlow hydraulic classifier to remove mica before each HydroFloat feed. Due to its shape, mica concentrates in the overflow along with any fine material, while the bulk of the material reports to the underflow. To feed the CrossFlow units, the 850 x 500 µm and 500 x 212 µm material was fed at approximately 30 solids and the 850 x 212 µm material was fed at approximately 40%.

The CrossFlow products were analysed, and their assays are shown in Table 10-29. These results show the Li_2O head grades of 1.9%, 2.7%, and 2.2%.

Table 10-29 CrossFlow Feed and Product Results

Test #	Feed Size	Stream ID	Balanced Data (%)						Distribution (%)				
			Mass	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	NaO_2	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	NaO_2
1	500x212	Feed	100.0	1.9	78.6	14.9	0.57	2.74	100.0	100.0	100.0	100.0	100.0
		Overflow	1.9	0.8	68.6	24.1	1.91	1.85	0.80	1.6	3.0	6.3	1.3
		Underflow	98.1	1.9	78.8	14.7	0.55	2.76	99.2	98.4	97.0	93.7	98.7
2	850x500	Feed	100.0	2.6	73.0	16.4	0.68	2.11	100.0	100.0	100.0	100.0	100.0
		Overflow	7.5	0.7	69.2	19.2	1.02	2.61	2.1	7.1	8.8	11.2	9.3
		Underflow	92.5	2.7	73.3	16.2	0.66	2.06	97.9	92.9	91.2	88.8	90.7
3	850x212	Feed	100.0	2.1	76.4	15.7	0.59	2.67	100.0	100.0	100.0	100.0	100.0
		Overflow	8.0	0.6	76.6	17.8	0.85	3.07	2.4	8.1	9.1	11.6	9.2
		Underflow	92.0	2.2	76.3	15.5	0.56	2.64	97.6	91.9	90.9	88.4	90.8

c) HydroFloat Flotation Testing

Following the Crossflow testing, a total of 15 HydroFloat tests were performed. This included six (6) for the 500 x 212 µm material, seven (7) for the 850 x 500 µm material, and two (2) for the 850 x 212 µm material. Various dosages of reagents and operating conditions were tested to determine grade and recovery. A rotary drum mixer was used to condition the feed material and was metered into the HydroFloat using a bench-scale vibratory feeder. HydroFloat results are shown in Table 10-30.

Table 10-30 HydroFloat Test Results

Test #	Feed Size	Stream ID	Balanced Data (%)						Distribution (%)				
			Mass	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	NaO_2	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	NaO_2
1	500x212	Feed	100.00	1.62	80.1	14.2	0.50	3.03	100.00	100.00	100.00	100.00	100.00
		Overflow	65.41	2.25	78.2	15.4	0.71	2.53	90.91	63.83	71.32	92.70	54.71
		Underflow	34.59	0.43	83.8	11.7	0.11	3.96	9.09	36.17	28.68	7.30	45.29
2	500x212	Feed	100.00	1.73	78.8	13.9	0.44	2.99	100.00	100.00	100.00	100.00	100.00
		Overflow	35.00	4.33	67.4	19.6	1.12	1.41	87.41	29.93	49.52	88.46	16.55
		Underflow	65.00	0.34	85.0	10.8	0.08	3.83	12.59	70.07	50.48	11.54	83.45
3	500x212	Feed	100.00	1.67	81.0	13.8	0.44	3.04	100.00	100.00	100.00	100.00	100.00
		Overflow	31.31	4.68	72.4	20.4	1.19	1.32	87.67	27.96	46.33	84.98	13.56
		Underflow	68.69	0.30	85.0	10.8	0.10	3.83	12.33	72.04	53.67	15.02	86.44
4	500x212	Feed	100.00	1.92	79.0	14.0	0.47	2.83	100.00	100.00	100.00	100.00	100.00
		Overflow	56.30	3.13	74.6	16.2	0.77	2.08	91.79	53.17	65.37	92.50	41.39
		Underflow	43.70	0.36	84.7	11.1	0.08	3.80	8.21	46.83	34.63	7.50	58.61
7	500x212	Feed	100.00	1.74	77.0	13.5	0.46	2.70	100.00	100.00	100.00	100.00	100.00
		Overflow	33.34	4.58	64.5	19.6	1.21	1.21	87.86	27.91	48.17	88.74	14.99
		Underflow	66.66	0.32	83.3	10.5	0.08	3.44	12.14	72.09	51.83	11.26	85.01
8	500x212	Feed	100.00	1.59	80.3	13.6	0.45	2.69	100.00	100.00	100.00	100.00	100.00
		Overflow	51.52	2.92	75.3	16.5	0.82	1.84	94.70	48.30	62.29	94.28	35.29
		Underflow	48.48	0.17	85.6	10.6	0.05	3.59	5.30	51.70	37.71	5.72	64.71
5	850x500	Feed	100.00	3.17	75.0	16.3	0.65	2.02	100.00	100.00	100.00	100.00	100.00
		Overflow	17.72	5.60	68.8	20.5	1.26	0.77	31.30	16.25	22.31	34.35	6.80
		Underflow	82.28	2.65	76.3	15.4	0.52	2.28	68.70	83.75	77.69	65.65	93.20
6	850x500	Feed	100.00	2.88	74.9	15.2	0.62	2.13	100.00	100.00	100.00	100.00	100.00
		Overflow	28.91	3.33	77.2	13.7	0.69	1.15	33.47	29.80	25.92	32.39	15.58
		Underflow	71.09	2.69	74.0	15.9	0.59	2.53	66.53	70.20	74.08	67.61	84.42
9	850x500	Feed	100.00	2.44	72.8	14.4	0.53	2.06	100.00	100.00	100.00	100.00	100.00
		Overflow	23.03	6.00	62.4	21.5	1.41	0.72	56.58	19.76	34.40	61.69	8.11
		Underflow	76.97	1.38	75.9	12.3	0.26	2.46	43.42	80.24	65.60	38.31	91.89
11	850x500	Feed	100.00	2.67	73.6	15.3	0.58	2.04	100.00	100.00	100.00	100.00	100.00
		Overflow	20.03	5.06	61.5	19.9	1.27	0.60	38.06	16.73	26.11	44.02	5.91
		Underflow	79.97	2.06	76.6	14.1	0.40	2.40	61.94	83.27	73.89	55.98	94.09
13	850x500	Feed	100.00	2.35	78.3	14.8	0.51	2.75	100.00	100.00	100.00	100.00	100.00
		Overflow	67.88	2.74	78.1	14.9	0.60	1.86	79.05	67.72	67.96	79.73	45.93
		Underflow	32.12	1.53	78.7	14.8	0.32	4.64	20.95	32.28	32.04	20.27	54.07
15	850x500	Feed	100.00	2.41	73.1	16.0	0.60	1.87	100.00	100.00	100.00	100.00	100.00

		Overflow	73.80	2.88	71.4	17.0	0.73	1.49	88.20	72.10	78.38	89.39	58.79
		Underflow	26.20	1.08	77.9	13.2	0.24	2.94	11.80	27.90	21.62	10.61	41.21
12	850x212	Feed	100.00	2.19	78.7	14.6	0.51	2.57	100.00	100.00	100.00	100.00	100.00
		Ro OF	54.23	3.35	74.7	16.6	0.78	1.85	82.83	51.47	61.87	83.13	39.09
		Ro UF	45.77	0.82	83.4	12.1	0.19	3.42	17.17	48.53	38.13	16.87	60.91
		Scav OF	5.06	1.02	83.2	12.4	0.36	3.04	2.37	5.35	4.31	3.53	6.00
		Scav UF	40.71	0.80	83.5	12.1	0.17	3.46	14.80	43.18	33.82	13.34	54.91
		Comb. R/S OF	59.29	3.15	75.4	16.3	0.75	1.95	85.20	56.82	66.18	86.66	45.09
14	850x212	Feed	100.00	2.02	78.7	14.8	0.54	2.71	100.00	100.00	100.00	100.00	100.00
		Ro OF	36.78	3.86	71.6	18.4	1.14	1.76	70.27	33.47	45.62	78.18	23.85
		Ro UF	63.22	0.95	82.8	12.8	0.19	3.27	29.73	66.53	54.38	21.82	76.15
		Scav OF	5.36	1.02	83.3	11.8	0.27	3.66	2.72	5.67	4.25	2.70	7.22
		Scav UF	57.86	0.94	82.8	12.9	0.18	3.23	27.01	60.86	50.13	19.12	68.93
		Comb. R/S OF	42.14	3.50	73.1	17.6	1.03	2.00	72.99	39.14	49.87	80.88	31.07

Based on these results, a comparison between split-feed and single feed optimal results have been prepared. Table 10-31 shows optimal global grade, while Table 10-32 shows optimal global recovery.

HydroFloat test results and comparison showed that an improvement in product grade can be achieved by implementing a split-feed coarse particle flotation. When targeting a high-grade concentrate in the 850 x 500 µm material, a Li_2O grade of 6.0% was achieved at a recovery of 56.6% using 200 g/tonne fatty acid. Conversely, when processing for high recovery with the 850 x 500 µm material, a Li_2O grade of 2.9% was achieved at a recovery of 88.2 % using 450 g/tonne fatty acid. The concentrate grade gradient was narrow between these two operating points, meaning selectivity was poor at elevated fatty acid dosage rates. For the 500 x 212 µm material, an optimal Li_2O grade of 4.7% was achieved at a recovery of 87.7% using 300 g/tonne fatty acid. Based on the CrossFlow weight splits, combined global Li_2O recoveries of 75.5% to 87.9% were achieved at concentrate grades of 5.2% and 4.0% Li_2O, respectively.

Single feed HydroFloat testing of 850 x 212 µm material yielded Li_2O grades of 3.2% to 3.5% at recoveries of 85.2% and 73%, respectively. Based on this, HydroFloat testing results demonstrate that higher grade can be achieved with split-feed methods. Figure 10-17 shows optimal grade vs. recovery HydroFloat testing results for single feed vs split feed.

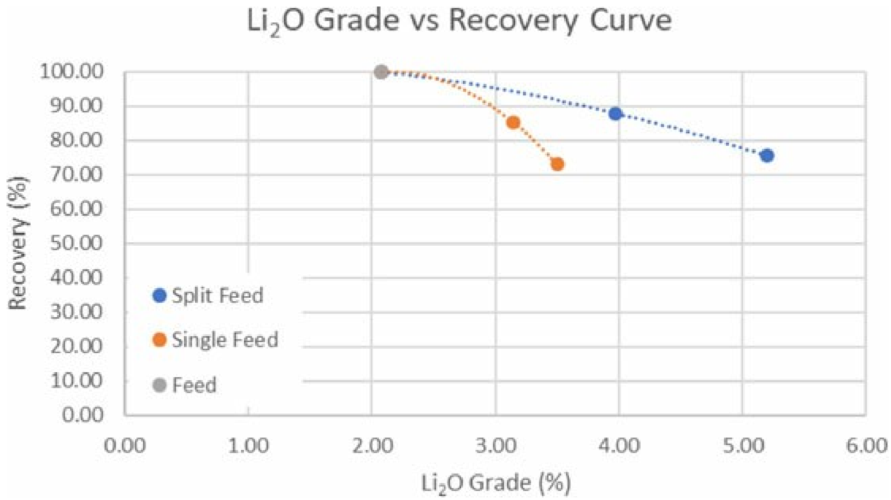
Table 10-31 HydroFloat Test Results Comparison, Global Grade

Split Feed Stocks - Estimated Stream Weight %			
Micron Size	**Stream Wt. %**	**Li_2O Recovery**	**Li_2O Grade**
850x500	39.0	56.58	6.00
500x212	61.0	87.67	4.68
Est. Global +212 Recovery and Grade		75.54	5.20
Test 14 Single Feed Recovery and Grade		72.99	3.50

Table 10-32 HydroFloat Test Results Comparison, Global Recovery

Split Feed Stocks - Estimated Stream Weight %			
Micron Size	**Stream Wt. %**	**Li_2O Recovery**	**Li_2O Grade**
850x500	39.0	88.20	2.88
500x212	61.0	87.67	4.68
Est. Global +212 Recovery and Grade		87.88	3.98
Test 12 Single Feed Recovery and Grade		85.20	3.15

Figure 10-17 Split-Feed vs. Single Feed Grade-Recovery Curves



Based on these results, split feed hydroflotation is recommended for the process to improve hydroflotation performance. In addition, to improve global recovery when targeting a high global grade for the 850x500 µm material, it is recommended that the HydroFloat underflow be ground and reprocessed. However, when targeting a high global recovery for the 850x500 µm material, it is recommended that the overflow be ground and reprocessed to improve the grade. Both cases are acceptable, but the high-grade scenario provides the best results as more than 50% of the Li_2O distribution in the 850x500 µm material is extracted as high-grade concentrate.

10.1.9.4 Coagulant Testing (2019-2021)

Process water treatment testwork was conducted at the Brenntag Canada Inc. facilities in Toronto, Ontario in 2019 and at the SNF Canada Ltd. Laboratory in Quebec City, Quebec in 2021 to investigate the impact of adding coagulant on the quality of water expected at the thickeners overflow.

In the case of the Brenntag testwork, water from the Whabouchi freshwater underground wells was mixed with dry fine tailings & fine concentrate. Multiple cylinder settling tests were then conducted with and without the addition of coagulant (flocculant was added in all tests). Results show that the average turbidity of the supernatants without coagulant reached 29.9 Nephelometric Turbidity Units (NTU), while this was reduced to 1.6 NTU when adding coagulant in the range of 3 to 10 ppm.

The testwork conducted at SNF used tailings water from previous laboratory scale flotation testwork done at SGS laboratory in Quebec City, Quebec. A series of jar mixer testwork were conducted with different coagulant types and dosages (flocculant was added in all tests). Overall, the testwork without coagulant gave on average supernatants with a turbidity of 27.3 NTU while this was reduced on average to 3.2 NTU with the addition of coagulant in the range of 20 to 40 ppm.

Based on these results, the addition of coagulant in the thickening process of the concentrator was recommended by both laboratories to improve the quality of recirculated process water.

10.1.9.5 Saponification Testing (2021-2022)

A series of three (3) large-scale saponification testwork were conducted at SGS Canada Inc. laboratory in Quebec City, Quebec to confirm the performance of spodumene collector dilution in cold water. This testing was performed in 2021-2022 as part of SGS test program described herein 10.1.9.2, however, was not reported on formally by SGS. Mixing of the collector was done directly into 150 L drums using a high-shear mixer plunged into cold water (5-7°C). Water was cooled in a refrigerator prior to testing and the solution was prepared based on the established saponification recipe.

Visual observations confirmed the effectiveness of high shear mixing in properly saponifying collector even in cold water. Adequate saponification and mixing can be assessed visually in the absence of dark lumps (undiluted collector) in the final solution.

The cold water saponified collector was also used in HydroFloat testwork. As seen in Table 10-33, the obtained performances, using the cold-water saponified collector, are comparable to the results when using warm-water saponified collector, which confirms that the high-shear mixing is adequate in properly saponifying collector in cold water. It should be noted that the warm water tests were done with the same collector, but it was saponified in small scale in an agitated beaker filled with warm water (~20°C).

Table 10-33 Collector Saponification Testwork Results

Collector Saponification Process	Number of Tests	Average Lithium Recovery	Average Lithium Upgrade Ratio
Large scale cold-water saponified collector	2	67%	1.9
Small scale warm water saponified collector	3	60%	1.9

10.1.10 QP Conclusions

A number of metallurgical test programs have been carried out over the years. This includes piloting of a dense medium separation circuit (DMS) carried out over a period of 6 months in 2017, and two flotation pilots for mechanical cells and flotation columns in 2011 and 2017, respectively. Many other large scale tests were carried out on individual pieces of equipment that are included in the final flowsheet using ore from a bulk sample from the Whabouchi project, although not as an integrated process.

The QP was not involved in of the selection of samples or metallurgical test work carried out or reported on above. The overall spodumene recovery is estimated as part of the mass balance based on individual stage recoveries noted in this section. The QP has reviewed the data and based on the body of work completed presented above, there is sufficient data to support the design of a spodumene concentrator to produce a 5.5% Li_2O concentrate at a Pre-Feasibility Study level.

There are several risks associated with the process which can impact overall lithium and spodumene recovery. Recommendations to address these risks are outlined in greater detail in Section 22 and include metallurgical risks as well as risks associated with the plant design.

The major metallurgical risks include:

- There is a risk that the overall recovery or the concentrate grade cannot be reached if some unforeseen factor affects the total plant performance. This risk can be mitigated by performing additional pilot-scale testing to help determined scale-up performance and interaction between various unit processes.
- Limited variability testing was carried out with the concentrator flowsheet design. In 2021-2022, variability testing was carried out on composite samples from the first five years of the mine plan. The main objective of these testwork was more focused on providing representative material for downstream conversion testwork than validating the concentrator design. Based on this objective, the produced concentrates successfully reached the target grade of 5.5 % Li_2O. However, since the operating parameters were not optimized through iterative testwork, the target recoveries were not achieved in all cases. Therefore, no variability tests have achieved the target grade and recovery simultaneously. Additionally, no variability work has been performed on samples representing the later years of the mine plan, however, it is currently planned to start sample preparation for this testing in 2023.

- Presence of variable quantities of petalite in the deposit will have a direct impact on lithium recovery. The lithium within petalite is unrecoverable by the designed process. Most testing has been performed with samples that have approximately 1% of total lithium in petalite; however, there are areas in the deposit with higher concentrations. A mitigation method is in place for stockpiling high-petalite material for the first five years of production before being processed in the following years. At the moment the known quantity of petalite has been taken into account in the current recovery assumptions. It is important that the lithium recovery be measured and evaluated based on spodumene during operation, as the recoveries of non-spodumene lithium-bearing minerals is not guaranteed and has not been properly defined by testwork. Note that a development program for the recovery of petalite is anticipated to start in 2024.

QP recommendations to reduce project risks related to mineral processing and metallurgical are provided in Section 23.

11 MINERAL RESOURCE ESTIMATES

11.1 Estimation Methodology

The Mineral Resource reported herein have been interpolated into a sub-block model using the modelled spodumene bearing pegmatites.

The resource estimate methodology is summarized by the following procedures:

- Drillhole database validations and selection of the drillholes and channels for the Mineral Resource estimation database;
- 3D modelling of spodumene-bearing pegmatite wireframes, based on lithology and lithium content (%Li_2O);
- Geostatistical analysis for data conditioning: density assignment, capping, compositing and variography;
- Block modelling and grade estimation;
- Resource classification and grade interpolation validations;
- Grade and tonnage sensitivities to spodumene concentrate selling prices;
- Gains and losses analysis with previous resource estimate (SGS, 2019).

11.2 Resource Database

To complete an updated Mineral Resource Estimation (MRE) for the Whabouchi Project (Whabouchi or the Project), a database in Access™ format (.accdb) was received from Nemaska Lithium (NLI) on February 17, 2022. The final database used for the MRE and geological modelling was compiled by SGS Geological Services (SGS) with a closing date of January 21, 2022. The latest drillhole on the project dates from October 2018. The only information added to the database used in this MRE is the density measurements done part of the 2021 resampling program (refer to herein Sections 8.9 and 8.11).

Fragmental information on mineralogy and lithium deportment is available, but the coverage of data is not sufficient to draw definitive conclusions or to be included in this estimation (refer to herein Sections 8 and 11.3.2 for more detail). All drillholes and channel collar information are recorded in the NAD83 datum, UTM Zone 18 North coordinate system.

The database used for the MRE included the following information:

- Collar information;
- Downhole surveys;
- Assay table (almost exclusively Li_2O);
- Lithology table;
- Density measurement tables;

LiDAR topography and excavation surfaces (pre-open pit excavation and metallurgy bulk sample). The drilling database used for the Mineral Resource Estimate comprises 258 diamond drillholes and 108 channels. Assaying is predominantly within the pegmatite dyke occurrences. Host rock sampling is generally limited to 1 m with the footwalls and hanging walls of each dyke. All holes and channels in the database were used for the MRE, except the following hole-id’s for which no information could be tracked relating to geology and assaying: ’20040’, ’20041’, ’20042’.

Channel hole-id's were standardized to avoid multiple names for a single channel path. This is to avoid over-estimating grades near surface when selecting a maximum number of samples per drillhole (or channel) during block estimation.

Assay values were also replaced to 0.00% Li_2O for all samples contained within waste material, such as basalt, amphibolite, or diorite. It has been demonstrated that lithium in these waste units is generally hosted within minerals other than spodumene such as holmquistite and are assumed to not be recoverable. A total of 3,194 assays were corrected out of the 18,147 assays in the database. Lithium content in these units is generally lower than 0.30% Li_2O (87% of assays, length weighted). An inspection of assays with high lithium content adjusted to 0.00% was undertaken and as expected, they are mostly located near the contact of pegmatite dykes where minor lithium minerals are present (other than spodumene). All assays containing more than 50% of pegmatite were left intact.

Original lithium assays were converted into Li_2O using a 2.1525 conversion ratio and expressed as percentage. Geostatistical analysis, variography and grade estimation considers lithium assays expressed as Li_2O.

The collar position of most drillholes of the database were surveyed by a total station or differential GPS (GNSS) when possible. Channel samples were recorded with handheld GPS and most were validated during the field visit. The elevation of drillholes and channels were adjusted with the high-resolution topographic surface (LiDAR) provided by NLI.

Drillhole and channel data was imported in a Leapfrog Geo project by NLI personnel and validated by SGS. The reader can refer to Section 9 for the in-detail data validation process undertaken by SGS. No errors were found and the QP is satisfied with the database used in the Mineral Resource Estimate.

A summary of the drilling database, including channels, used for the Whabouchi MRE is presented in Table 11-1, with total assays. A summary of assay methodology by year of drilling is presented in Table 11-2.

Table 11-1 Summary of Drillholes, Channels and Assays used for the MRE

Type	Number of Surveys	Total Meters	Number of Assays	Total Assayed Meters	% Assayed
Channel	108	964.0	944	940	98
Diamond Drillhole	258	51,631.5	15,438	15,792	31
Total	**366**	**52,595.5**	**16,382**	**16,732**	**32**

Table 11-2 Summary of Assay Type by Year of Drilling

Drilling Year	Aqua Regia	4-acid fusion	Sodium peroxide fusion	Unknown	Count
Channels (2009-2010)	0%	0%	100%	0%	944
2009	0%	0%	100%	0%	456
2010	0%	18%	82%	0%	6,088
2011	1%	99%	0%	0%	1,869
2013	6%	94%	0%	0%	350
2016	0%	100%	0%	0%	4,038
2017	0%	88%	12%	0%	1,819
2018	0%	0%	99%	1%	818
Total	**0%**	**55%**	**45%**	**0%**	**16,382**

11.3 Geological Modelling

The geological model was completed jointly by NLI and SGS personnel. The model involved wireframing the following units: spodumene-bearing pegmatite dykes, barren pegmatite dykes and internal amphibolite (combined with diorite and/or basal) units within pegmatite dykes. Leapfrog Geo software was used to model these units.

The spodumene-bearing pegmatite dykes are generally well developed along the east-northeast strike-length of the deposit, for approximately 1,350 m, and dipping steeply to the south-east (North 060°/65-70°). Most dykes are sub-parallel to each other representing a dyke swarm for up to 200 to 250 m across strike. Only three interpreted dykes out of the 24 have a discordant azimuth and/or dip direction to the dominant trend.

Based on core drilling data and outcrop channel sampling, a three-dimensional model was created for the pegmatite dykes (Figure 11-1 and Figure 11-2). The geological model honours the lithological logging data. The dykes were modelled from logged pegmatite intervals with a lower cut-off of 0.30% Li_2O and a minimum thickness of 2.0 m as implicitly derived vein contact surfaces in Leapfrog Geo software (version 2021.2.4).

The resulting geological model incorporates 23 pegmatite dykes, with two dykes merged (Main1 with Main1_22 and Main2_2 with Doris_1). Two barren pegmatite dykes (Main1_I1G and Doris_I1G) and one thin, discontinuous dyke (ZoneNord_5_W) were also modelled to better assess specific gravity in the block model: two were modelled with vein contact surfaces and another one with intrusion contacts. For density and grade interpolation purposes, an internal waste model was created inside five (5) of the principal dykes. Most of the internal zones are located inside the main dyke (Main1). Internal waste was created as pinch-out vein contact surfaces using the general trend of its parent dyke.

Finally, a dilution skin of 4 m around each spodumene pegmatite dyke was created. This is to ensure that density and background grades are better defined when reblocking will occur for the Mineral Reserve and assuming the proper dilution grades.

Figure 11-1 Plan View of the Modelled Pegmatite Dykes

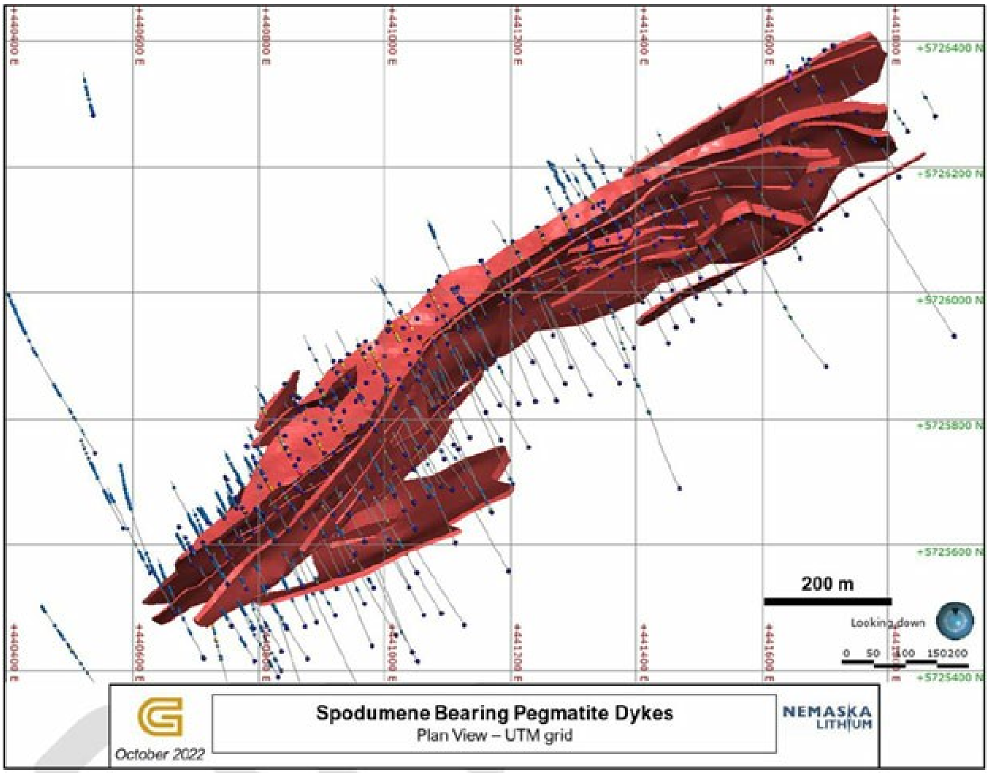
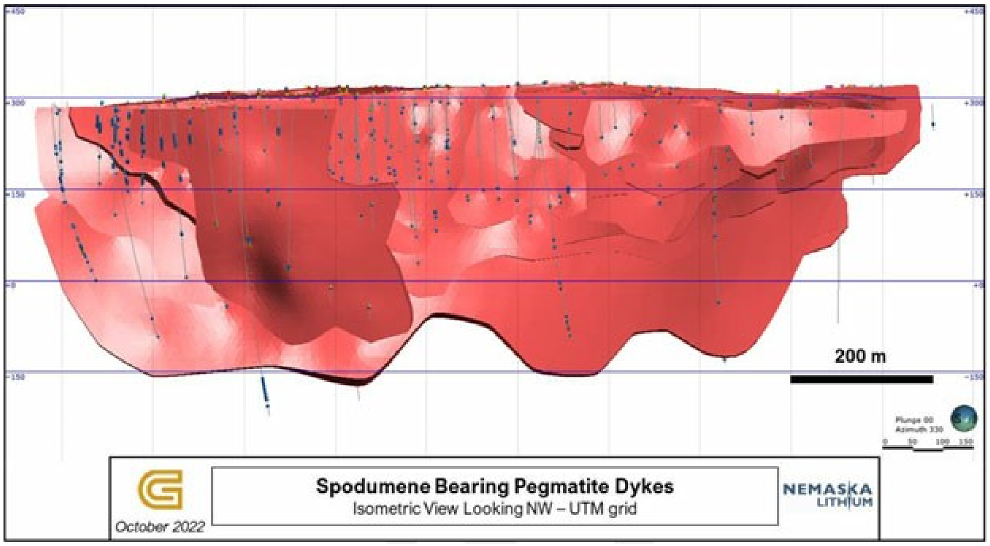


Figure 11-2 Isometric View of the Modelled Pegmatite Dykes – Looking North-West



11.3.1 Topographic and Overburden Models

The topographic surface has been provided by NLI and is a LiDAR survey from 2017. An updated surface was created to account for depletion caused by a pre-open pit excavation and metallurgical bulk sampling by combining the surfaces. The overburden surface was created as an offset from the topographic surface using the logged drill intervals as overburden. This method ensures that the overburden surface does not overlap or exceed the elevation of the topography.

11.3.2 Preliminary Petalite Models and Petalite Mineral Content

As discussed in Section 8, NLI initiated in 2022 a mineralogical identification program to identify the mineralogy of lithium inside the deposit. Results are still fragmentary and preliminary and cannot be used in a block model at the time of writing this report. However, NLI geologists were able to model, with visual core observations, probable petalite bearing sub-domains that preferentially sits inside the hanging wall and footwall of the Main1 domain. To assess this lithium deportment during the Mineral Reserve and mine planning (herein Section 12), the assumptions presented in Table 11-3 were made based on results from approximately 750 scanned samples.

Table 11-3 Preliminary Lithium Deportment in Petalite and Muscovite

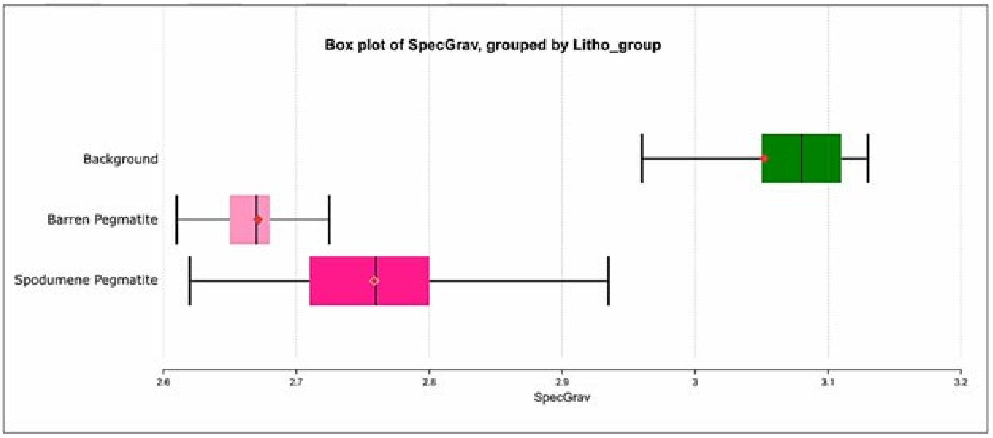
Domain	% of Li from Mineral		Total
	Petalite	Muscovite	
Main1	1.8%	1.6%	3.4%
Petalite Domains	10.3%	0.8%	11.1%
Bulk sample area	0.7%	1.3%	2.0%

11.4 Specific Gravity

Specific gravity measurements were obtained by pycnometer on full sample pulp reject (refer to herein Section 8.11 for detail on methodology). For all pegmatites, the median of all measurements was used: 2.76 g/cm³ and 2.67 g/cm³ for spodumene-bearing and barren pegmatite respectively. In comparison, the previous model had a unique value of 2.71 g/cm³ for mineralized pegmatite, which corresponds to a 2% increase. The average density of the mineralized zones is 2.76 g/cm³ when taking into account internal dilution. Refer to herein Section 11.13 for more detail on changes with the previous block model.

A value of 3.04 g/cm³ was used for waste rock, based on the mean of specific gravity measurements after removing a few outliers. The data distribution of the waste rock shows that geology is probably more complex than a single "background" unit. Density was assigned in the block model based on waste content estimated inside each block. A summary of specific gravity measurements is presented in Figure 11-3, where whiskers represent 1.5 times the interquartile range.

Figure 11-3 Box Plot of Specific Gravity Measurements



11.5 Assays, Capping and Compositing

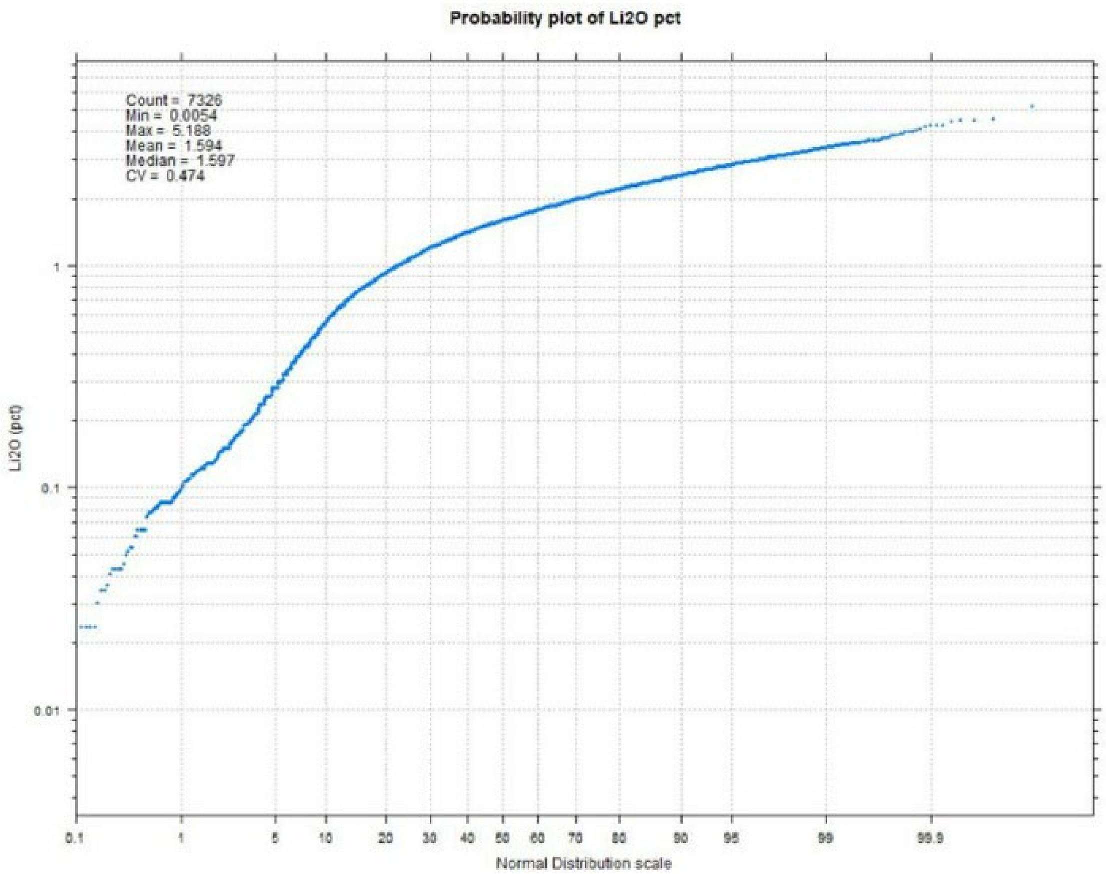
The assays and composites have been examined statistically using histograms, cumulative probability plots (CPP) and boxplots to determine composite lengths and if capping is warranted. Comparison of the assay and composite statistics is also done to ensure compositing has introduced no bias and that the data shows a reasonable distribution for use of linear interpolation methods. Assay statistics are tabulated by domain and are presented in Table 11-4. It is noteworthy that the main dyke (Main1_F) accounts for 72% of the sample lengths, whereas the first three main dykes (Main1_F, Doris_Main2_F and Doris_3) account for 87% of the sample lengths. As seen from the summary statistics presented, medians are generally very close to the mean, indication that data should not be positively or negatively skewed and normally distributed.

The CPP histograms suggest that Lithium grades did not require capping of extremely high-grade outliers. The distribution of lithium grades shows a normal distribution, holds very few outliers, has very low Coefficient of Variation (COV) and no major break in grade distribution is observed (see Main1 dyke example, Figure 11-4). Furthermore, only approximately 22% of the lithium content is in the top decile (top 10% values) and less than 3% of the lithium content is in the top percentile (top 1% values). Inspection of high-grade values (>3% Li_2O) are evenly distributed in the deposit and generally well supported by nearby samples.

Table 11-4 Assay Statistics of Lithium (%Li_2O) by Domain

Domain	Count	Length (m)	Mean	SD	Variance	COV	Minimum	Median	Maximum
Main1_F	7,371	7,429.4	1.59	0.76	0.48	0.58	0.00	1.60	5.19
Doris_Main2_F	997	1,013.6	1.36	0.76	0.56	0.58	0.00	1.34	4.05
Doris_3	490	517.5	1.26	0.73	0.58	0.54	0.00	1.27	3.66
ZoneSud_3	220	220.8	1.57	0.72	0.46	0.52	0.00	1.54	3.66
ZoneNord_6	201	198.0	1.33	0.67	0.50	0.45	0.00	1.36	4.00
ZoneSud_1	187	196.3	1.66	0.82	0.49	0.67	0.00	1.57	4.54
ZoneSud_4	103	111.2	1.23	0.80	0.65	0.64	0.00	1.14	3.55
InterDoris_1	90	87.0	0.84	0.75	0.89	0.56	0.00	0.75	2.61
Doris_2	84	86.4	1.24	0.80	0.64	0.63	0.00	1.25	3.67
ZoneSud_2	76	74.0	1.44	0.76	0.52	0.57	0.00	1.49	3.40
ZoneSud_6	74	72.0	1.58	0.85	0.54	0.73	0.00	1.55	3.70
ZoneNord_4	72	66.9	1.25	1.16	0.93	1.35	0.00	1.01	4.03
Main1_4	49	52.8	1.38	1.06	0.77	1.12	0.00	1.29	4.59
ZoneNord_3	51	52.5	0.77	0.65	0.84	0.42	0.00	0.69	2.35
InterDoris_2	46	42.1	0.50	0.54	1.08	0.30	0.00	0.34	1.59
ZoneSud_5	40	40.8	1.37	0.90	0.66	0.81	0.00	1.42	3.10
ZoneSud_7	43	40.7	1.49	0.84	0.56	0.70	0.00	1.70	2.72
Doris_4	32	31.8	1.39	0.83	0.59	0.68	0.00	1.61	2.71
Doris_8	8	7.6	1.16	0.63	0.54	0.40	0.35	1.03	2.20
InterDoris_3	5	6.1	0.50	0.33	0.66	0.11	0.00	0.45	0.84
ZoneNord_2	6	5.4	0.42	0.38	0.92	0.15	0.00	0.47	0.88

Figure 11-4 Cumulative Probability Plot for Main1 Dyke



11.5.1 Compositing

Compositing was completed for all domains using a 2.0 m composite length to standardize the sample lengths used in interpolation. Composite length was chosen based on the assay length distribution presented below (Figure 11-5); while most of the samples are of 1.0 m length, there is a moderate number of longer samples (9%). With the anticipated block size to be used, based on expected mining bench height, 2.0 m composites are well adapted for 6.0 m block height. The domain boundaries have been honoured during compositing, with any assay length less than 0.50 m added to the previous composite to avoid large numbers of small “remnant” composites at the domain boundaries.

Composite statistics for each domain are presented in Table 11-5 and an example of grade distribution in Main1 is shown in Figure 11-6. Composite means of each domain are very similar to assay means, meaning that no bias has been introduced in the compositing process.

Table 11-5 Composite Statistics of Lithium (% Li_2O) by Domain

Domain	Count	Length (m)	Mean	SD	Variance	COV	Minimum	Median	Maximum
Main1_F	3,850	7440	1.57	0.64	0.41	0.41	0.00	1.62	3.97
Doris_Main2_F	549	1024	1.32	0.64	0.41	0.49	0.00	1.31	3.08
Doris_3	266	518	1.24	0.63	0.40	0.51	0.01	1.25	3.13
ZoneSud_3	118	221	1.55	0.56	0.31	0.36	0.34	1.54	3.21
ZoneNord_6	109	199	1.30	0.56	0.32	0.43	0.00	1.32	3.66
ZoneSud_1	101	197	1.64	0.73	0.54	0.45	0.39	1.57	3.64
ZoneSud_4	62	112	1.20	0.68	0.46	0.57	0.00	1.17	2.96
InterDoris_1	52	93	0.75	0.64	0.41	0.86	0.00	0.53	2.52
Doris_2	53	91	1.13	0.70	0.50	0.62	0.00	1.18	2.83
ZoneSud_2	41	74	1.41	0.66	0.44	0.47	0.21	1.38	2.69
ZoneSud_6	40	74	1.51	0.77	0.59	0.51	0.32	1.49	3.40
ZoneNord_4	36	70	1.29	1.11	1.23	0.86	0.02	1.05	3.82
Main1_4	31	53	1.29	0.92	0.84	0.71	0.01	1.16	3.98
ZoneNord_3	30	53	0.70	0.58	0.33	0.83	0.00	0.61	1.94
InterDoris_2	35	54	0.44	0.45	0.21	1.04	0.00	0.28	1.44
ZoneSud_5	23	41	1.30	0.76	0.58	0.58	0.11	1.24	2.57
ZoneSud_7	24	43	1.34	0.72	0.52	0.54	0.00	1.35	2.44
Doris_4	16	32	1.37	0.63	0.40	0.46	0.10	1.43	2.40
Doris_8	5	8	1.09	0.50	0.25	0.46	0.52	0.91	1.62
InterDoris_3	6	8	0.42	0.31	0.10	0.74	0.03	0.31	0.78
ZoneNord_2	3	6	0.35	0.19	0.04	0.56	0.15	0.36	0.53

Figure 11-5 Histogram of Composite Grade Distribution in Main1

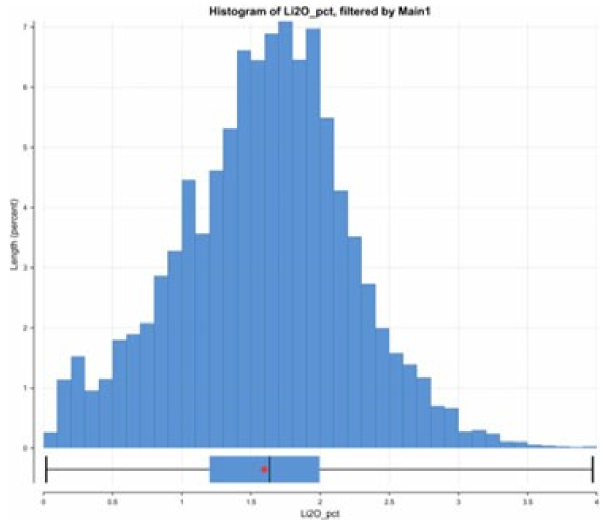
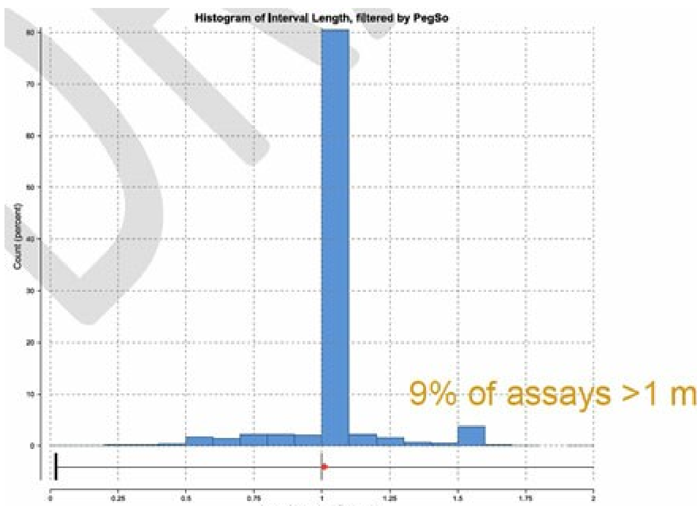


Figure 11-6 Assay Length inside the Pegmatite Dykes



11.6 Variography

Experimental variograms were produced for each domain based on the 2 m composites and were aligned with the clearest angle of continuity. Some domains with few composites and with similar dyke orientation were grouped together. Major dykes spanning over 1 km in strike length and open-folded, such as Main1_F and Doris_Main2_F, were sub-domained to produce experimental variograms. Resulting variograms were then applied to the whole domain. Table 11-6 presents a summary of variogram models used by domain and details the parameters used for each domain for Li_2O interpolation are provided in Table 11-7. An example of the variogram model in shown in Figure 11-7, with clear major ranges around 100 m.

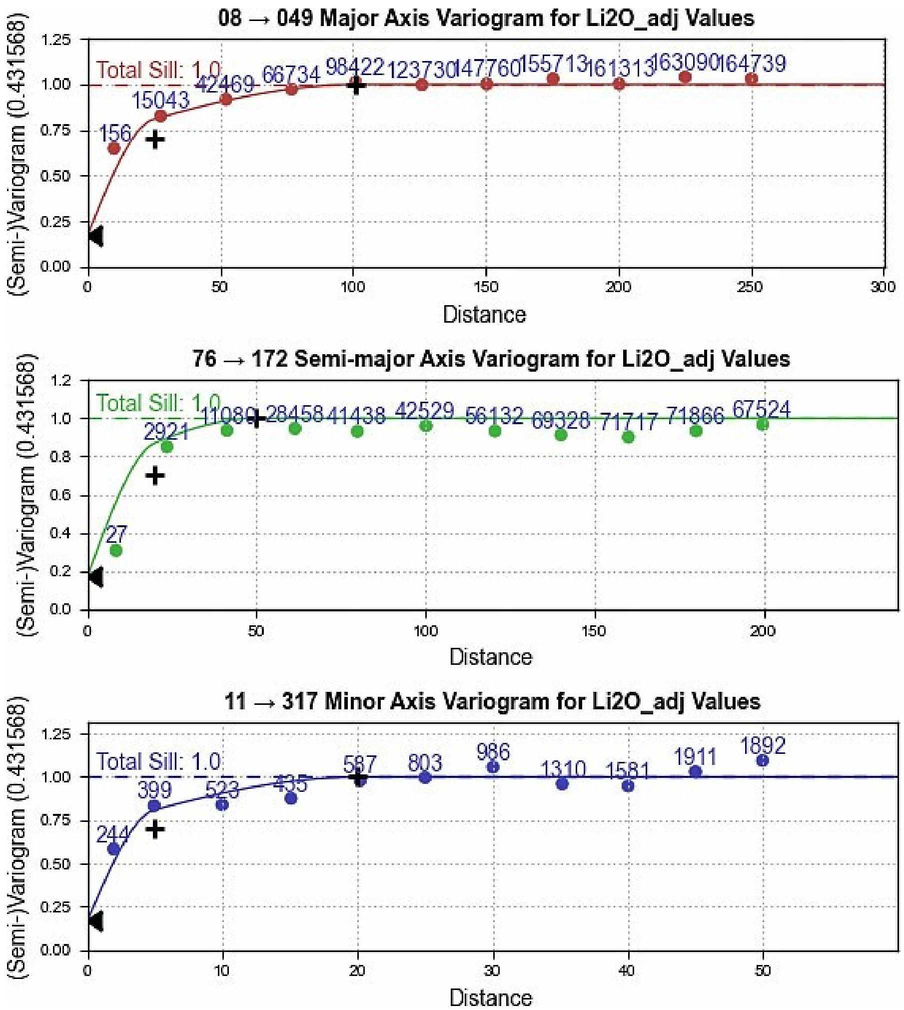
Table 11-6 Experimental Variogram Models per Domain

Domain	Variogram Model Name	Adjusted from
Main1_F	Main1_F_Vario	N/A
Doris_Main2_F	Doris_Main2_Vario_A	N/A
Doris_2	Doris_2_Vario	Doris_Main2_Vario_A
Doris_3	Doris_3_Vario	N/A
Doris_4	Doris_4_Vario	Doris_3_Vario
Doris_8	Doris_2_Vario	N/A
InterDoris_1	Main1_F_Vario	N/A
InterDoris_2	InterDoris_2	Main1_F
InterDoris_3	InterDoris_3	Doris_2_Vario
Main1_4	Main1_4_Vario	Main1_F
ZoneNord_2	ZoneNord_2_Vario	Main1_F
ZoneNord_3	ZoneNord_3_Vario	Main1_F
ZoneNord_4	ZoneNord_4_6_Vario	Main1_F
ZoneNord_6	ZoneNord_4_6_Vario	Main1_F
ZoneSud_1	ZoneSud_1356_Vario	N/A
ZoneSud_2	ZoneSud_2_Vario	N/A
ZoneSud_3	ZoneSud_1356_Vario	N/A
ZoneSud_4	ZoneSud_4_Vario	Doris_Main2_Vario_A
ZoneSud_5	ZoneSud_1356_Vario	N/A
ZoneSud_6	ZoneSud_1356_Vario	N/A
ZoneSud_7	Doris_Main2_Vario_A	N/A

Table 11-7 Variogram Parameters for each Domain

Domain	Direction			Nugget	Structure 1				Structure 2			
	Dip	Dip Azimuth	Pitch		Sill 1	Major	Semi-major	Minor	Sill 2	Major	Semi-major	Minor
Main1_F	77	140	5	0.16	0.58	33	30	4	0.26	101	60	13
Doris_Main2	77	148	12	0.20	3.00	44	37	4	0.14	74	60	6
Doris_2	77.5	127	57	0.20	0.66	44	37	4	0.14	74	60	6
Doris_3	83	336	76	0.20	0.50	50	40	3	0.30	100	77	8
Doris_4	78.5	350	111	0.20	0.50	50	40	3	0.30	100	77	8
InterDoris_2	77	140	129	0.16	0.58	33	30	4	0.26	101	60	13
InterDoris_3	77.5	127	12	0.20	0.66	44	37	4	0.14	74	60	6
Main1_4	61	181	118	0.16	0.58	33	30	4	0.26	101	60	13
ZoneNord_2	64	157	96	0.16	0.58	33	30	4	0.26	101	60	13
ZoneNord_3	86	115	109	0.16	0.58	33	30	4	0.26	101	60	13
ZoneNord_4_6	70	148	101	0.16	0.58	33	30	4	0.26	101	60	13
ZoneSud_1356	70	154	114	0.20	0.52	55	55	8	0.28	100	69	20
ZoneSud_2	86	150	0	0.20	0.37	62	28	2	0.43	80	60	5
ZoneSud_4	61	175	134	0.20	0.66	44	37	4	0.14	74	60	6

Figure 11-7 Example of Variogram Model for Main1_F



11.7 Block Modeling

A single rotated sub-blocked model was created for the Project. Dimensions and parameters are presented in Table 11-8 and a plan view is shown in Figure 11-8.

The block model was created with a parent block size of 6 m x 4 m x 6 m and a sub-block size of 3 m x 1 m x 3 m. The sub-block triggers are the topography, the overburden surface, the geological model (spodumene pegmatite dykes, barren pegmatite dykes, internal waste) and the 4 m dilution skin surrounding all domains. Volumes of each individual domain wireframes, cut by the overburden surface, were compared to the corresponding volume in the block model. It was found that the volumes are well replicated in the block model and no bias, positive or negative, is caused by the block size. All domains are within 1% of difference. Globally there is 0.1% in volume difference for all 21 domains.

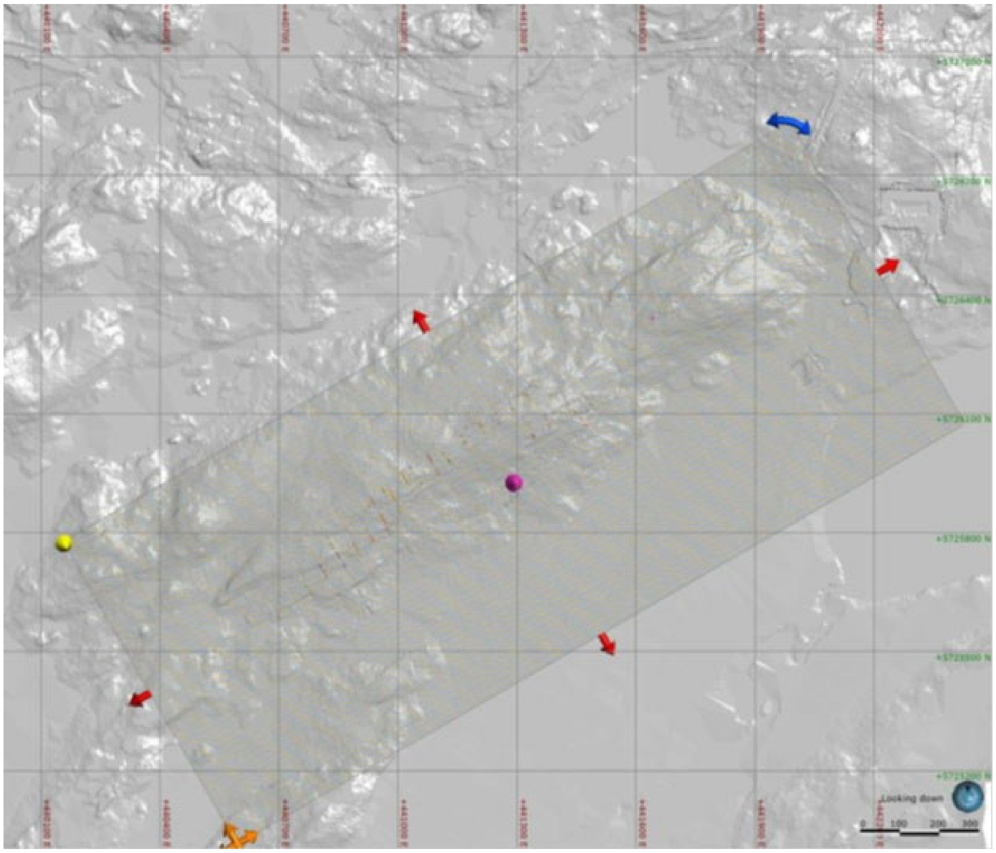
Block size was chosen based on a Kriging Neighbourhood Analysis (KNA) and by common agreement between SGS, BBA, and NLI personnel. The block size height was fixed at 6 m to account for equipment requirements and bench heights of 12 m. Easting and Northing block sizes were tested by KNA. Easting size of 6 m roughly corresponds to 25% of the nominal drill spacing, which is considered the optimal ratio in terms of data spacing (Vasylchuk & Deutsch, 2015). Sub-blocking was chosen to honour geological units and the overburden surface.

The Reader should refer to herein section for more details on the KNA analysis.

Table 11-8 Block Model Parameters and Dimensions

BM Name	Description	Easting (m)	Northing (m)	Elevation (m)
GMS_Final	Origin coordinates	440,595	5,725,015	345
	Parent block size	6	4	6
	Sub-block size	3	1	3
	Number of blocks	353	220	92
	Rotation	330°		
	Sub-block triggers	Topography, overburden, geological model, dilution skin		

Figure 11-8 Whabouchi Deposit Block Model Extent on Topography



11.8 Block Model Interpolation

Interpolations were completed based on the variogram models presented in a previous sub-section using the Ordinary Kriging method (OK). Grades were estimated using a three-pass approach, with increasing ellipsoid size. Search ellipse size and composite search definition are presented in Table 11-9. Maximum sample selection and ellipsoid ranges were selected following the KNA analysis. Table 11-10 presents blocks being interpolated during each pass. Most blocks interpolated during the third pass are extrapolation at depth where no resource is assumed (Section 11.10.1). Dynamic anisotropy based on dyke reference planes was used to locally rotate and align the search ellipse during grade estimation.

Background blocks, or all blocks not labelled as spodumene pegmatite, were interpolated using the Inverse Distance Square method (ID2) with the same interpolation parameters as Pass 3 presented below. Background blocks include the following units: dilution skin, barren pegmatite dykes (Doris_1G, Main1_I1G, ZoneNord_5) and internal waste within the spodumene pegmatite dykes. Blocks not interpolated were assigned a value of 0.00% Li_2O.
Based on the lithological core logging, the percentage of amphibolite (grouped with diorite or basalt) within each composite was calculated and then interpolated in each pegmatite and dilution skin blocks of the block model.

This was done to ensure the following:

- Have an assessment of the percentage of waste already included in each block for the Mineral Reserve model and mine planning; and
- To more accurately calculate the density of each block (amphibolite being more dense than the pegmatite).

Table 11-9 Interpolation Parameters by Passes

Interpolation Pass	Ellipsoid ranges (m)			Orientation	Composites			Minimum Hole
	Maximum	Intermediate	Minimum		Min.	Max.	Max./Hole	
Pass 1	50	30	10	Variable (DA)	7	22	3	3
Pass 2	100	60*	20	Variable (DA)	6	22	3	2
Pass 3	200	200*	30	Variable (DA)	2	22	3	1

*Doris_4: the intermediate range was adjusted to 45 m for Pass 2 and 3 to avoid mixing populations.

Table 11-10 Blocks Interpolation During Each Pass

Interpolation Pass	Volume of Blocks (’000 m3)	% of Total
Pass 1	5,880	27
Pass 2	12,437	56
Pass 3	3,840	17

11.9 Grade Estimation Validation

Various validation steps were taken to ensure that the block model is a robust representation of the composites.

The following validations were undertaken:

- Visual checks on-section comparing composite grades against block grades and validation of the dynamic anisotropy;
- Global statistical checks comparing the various grades of the block model against a Nearest Neighbour (NN) estimate and against the declustered composite data;
- Local statistical validation to identify any over-smoothing are areas of grade over- or under-extrapolation (Swath Plots);
- Peer review by BBA.

11.9.1 Visual Validation

SGS performed a visual validation comparing the composite grades against the block grades in cross-section and plan view. This validation also confirmed that the search ellipsoids were aligned with dyke contacts within the geological model, especially where a dyke changes direction. As seen in Figure 11-9 and Figure 11-10, from the south-western and north-eastern ends of the deposit respectively, block grades are good representation of composite grades. Grades also follow adequately small curvatures in the dyke geometry.

Figure 11-9 Composite vs. Block Grades – South-West, looking SW

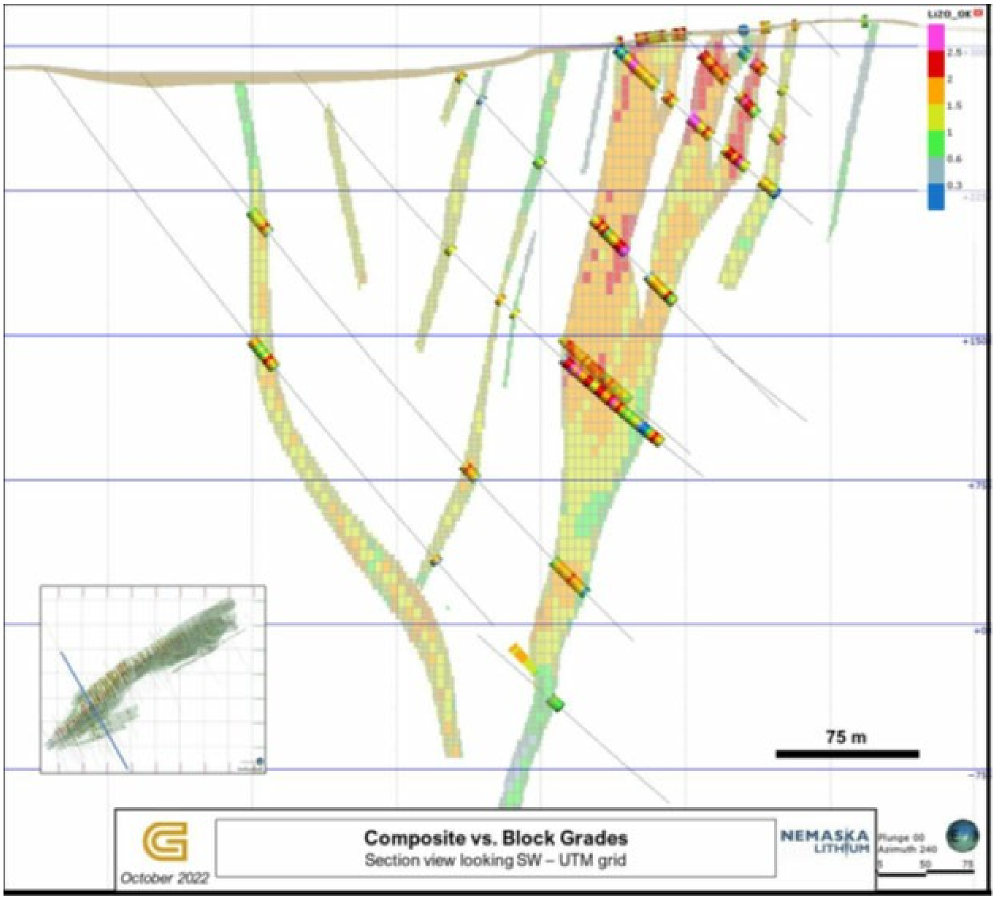
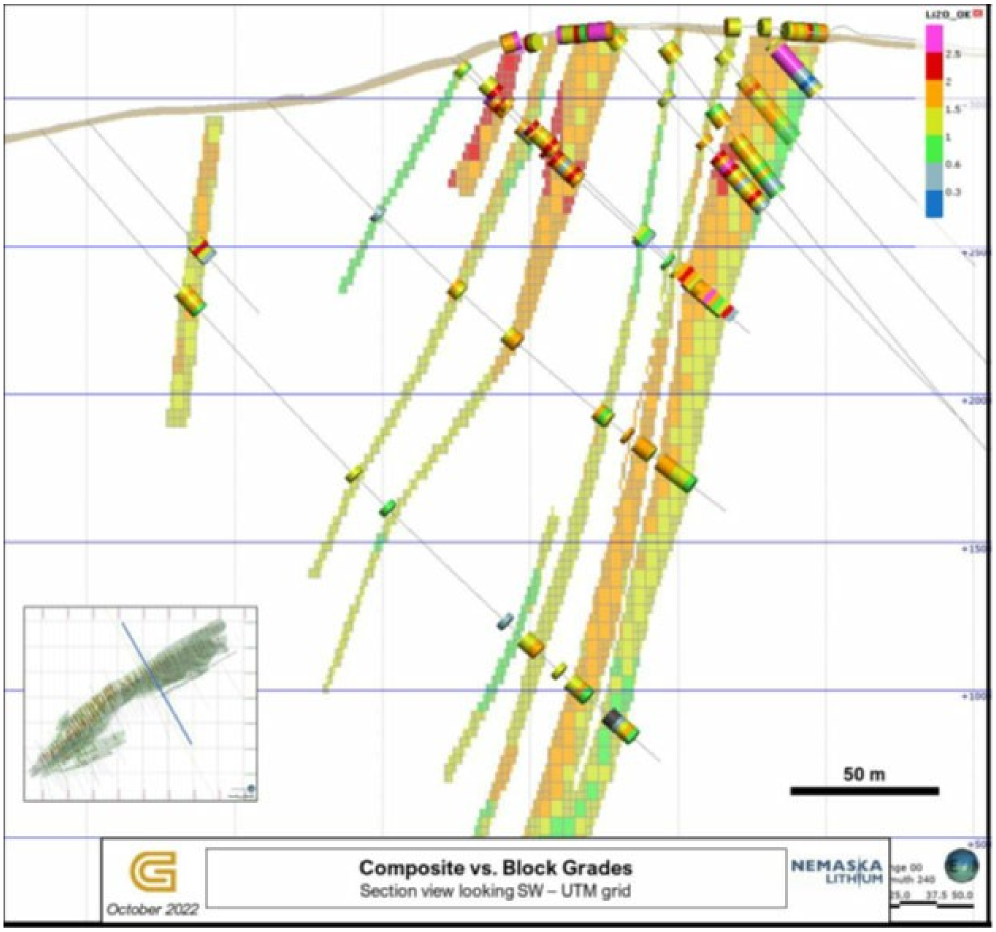


Figure 11-10 Composite vs. Block Grades – North-East, looking SW



11.9.2 Global Statistical Validation

To ensure proper composite representation in each domain, a statistical comparison was made between various attributes: final grades used for the resource model (by OK), grades interpolated using NN, grade interpolated using ID2, assays, composites and declustered composite grades. Table 11-11 shows the comparison between the various data and it is found that for the major dykes (90% of blocks), differences in average grades are less than or equal to 0.01% Li_2O when compared to declustered composites. Comparison between the different interpolation techniques also show that no bias is present in the OK estimate. Globally, blocks using OK are 0.4% (or 0.01% Li_2O) lower than the declustered composites. Based on statistics, the interpolation using OK is judged to be valid and good representation of composite grades.

Table 11-11 Mean Grade Comparison between Assays, Composites and Blocks

Domain	% Mass	Mean Grades (%Li_2O)						Diff. OK - Decl. Comp (%Li_2O)
		Assays	Comp.	Decl. Comp.	OK	ID2	NN	
Main1_F	68	1.59	1.57	1.53	1.52	1.53	1.52	-0.01
Doris_Main2_F	10	1.36	1.32	1.28	1.29	1.31	1.27	0.01
Doris_3	6.7	1.26	1.24	1.24	1.23	1.24	1.21	-0.01
ZoneNord_6	3.0	1.33	1.30	1.26	1.26	1.26	1.27	0.00
ZoneSud_1	1.8	1.66	1.64	1.58	1.57	1.62	1.59	-0.01
ZoneSud_3	1.4	1.57	1.55	1.45	1.47	1.48	1.44	0.02
ZoneSud_4	1.3	1.23	1.20	1.18	1.17	1.17	1.15	-0.01
InterDoris_1	1.1	0.84	0.75	0.66	0.74	0.71	0.63	0.08
ZoneNord_4	1.1	1.25	1.29	0.85	0.87	0.97	0.81	0.02
ZoneSud_2	0.9	1.44	1.41	1.46	1.50	1.50	1.45	0.04
Doris_2	0.8	1.24	1.13	1.13	1.16	1.18	1.15	0.03
Main1_4	0.7	1.38	1.29	1.29	1.43	1.44	1.42	0.14
ZoneNord_3	0.6	0.77	0.70	0.65	0.60	0.59	0.57	-0.05
ZoneSud_6	0.5	1.58	1.51	1.39	1.41	1.45	1.39	0.02
ZoneSud_7	0.4	1.49	1.34	1.26	1.27	1.24	1.21	0.01
InterDoris_2	0.4	0.50	0.44	0.42	0.40	0.40	0.36	-0.02
ZoneSud_5	0.4	1.37	1.30	1.21	1.21	1.29	1.14	0.00
Doris_4	0.2	1.39	1.37	1.37	1.37	1.41	1.43	0.00
ZoneNord_2	0.1	0.42	0.35	0.35	0.35	0.34	0.35	0.00
InterDoris_3	0.0	0.50	0.42	0.42	0.40	0.39	0.35	-0.02
Doris_8	0.0	1.16	1.09	1.09	1.03	1.11	1.11	-0.06
Total	**100**	**1.516**	**1.479**	**1.435**	**1.429**	**1.440**	**1.422**	**-0.01**

11.9.3 Local Statistical Validation – Swath Plots

Swath plots were generated for all domains for Li_2O grades in Eastings, Northings and Elevation. They were investigated for potential over-smoothing of grades (often a side-effect of an OK estimate with inappropriate parameters). It was found that peaks and troughs in composite grades generally follow peaks and troughs in block grades; no significant bias was found, and composite grades are well represented in blocks. Figure 11-11 and Figure 11-12 show example of the Main1 and Doris_Main2 dykes, accounting for approximately 78% of global tonnes. Although some smoothing inherent to OK is observed, general trends are preserved and many of the more significant discrepancies are explained by a very low composite count on the section.

Figure 11-11 Swath Plot by Easting of Main1 (Measured and Indicated blocks only)

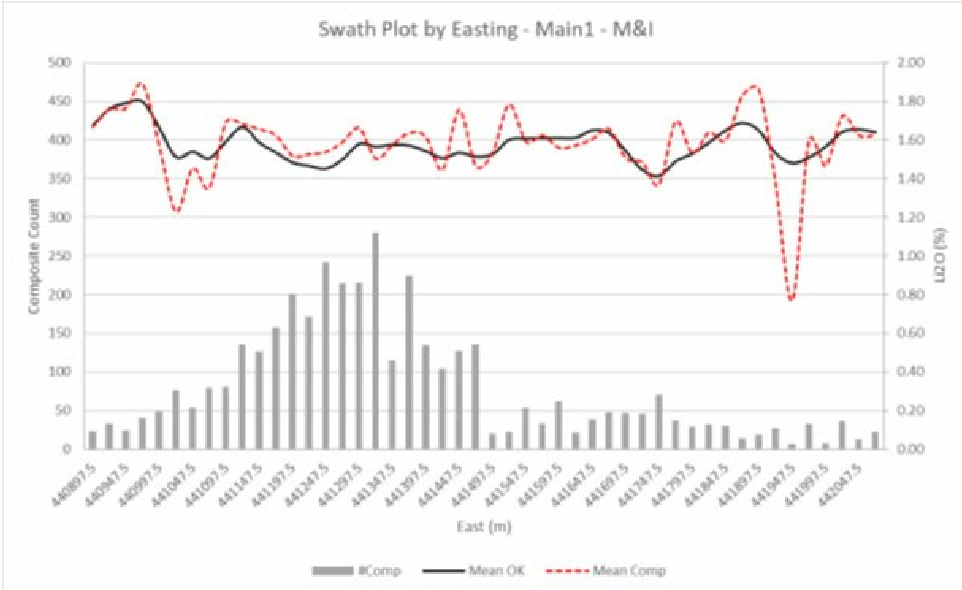
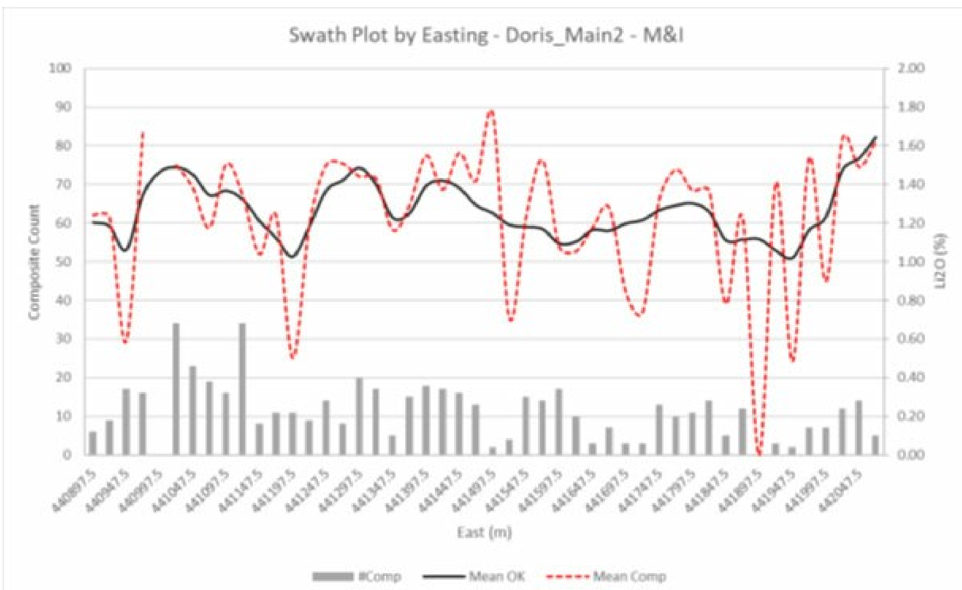


Figure 11-12 Swath Plot by Easting of Main1 (Measured and Indicated blocks only)



11.9.4 Peer Review by BBA

Throughout the Mineral Resource Estimation workflow, BBA Inc. was involved in a peer review of each major step. Recommendations made were mostly integrated in the resource database, geological model, or block model.

Amongst other, the following implemented recommendations were made:

- Single Hole-IDs for channels with the same name and same path;
- Standardize significant digits in Li_2O grades;
- Checks of adjacent assays with identical grade;
- General geological model recommendations, such as combining units and kink removal in a dyke.

Some recommendations with no material impact on the resource will be implemented in future production-oriented block models. No fatal flaws were identified during each independent review.

11.10 **Mineral Resources**

11.10.1 Mineral Resource Classification

Block model grades estimated for the Whabouchi project were classified according to the CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and adhere to the CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (2019). As defined by the CIM, all classified material must be within a potentially mineralized wireframe and within the "reasonable prospects of eventual economic extraction" shapes. The Mineral Resources at the project were classified as Measured, Indicated and Inferred Mineral Resources andare reported on a total basis for the property and on an attributable basis consistent with Livent's ownership interest in the property.

As stated in the CIM's "Definition Standards for Mineral Resources and Mineral Reserves":

"A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit."

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit."

"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity."

SGS considered variogram ranges, drill hole spacing, confidence in the geological interpretation and presence of channel samples to determine parameters that will define the resource categories. The final Mineral Resource classification is mostly based on average drill hole spacing, the number of samples used in the interpolation, specific geological units, and manual editing to avoid isolated blocks.

The principal assumptions undertaken by SGS to classify the Mineral Resources as Measured, Indicated, and Inferred categories are summarized below:

Measured Mineral Resources are generally blocks with an average distance between the three nearest drill holes of less than 30 m. It generally corresponds to the tightest drill spacing (approximately 30 x 30 m). Areas where a vertical distance between drill holes was too high were excluded from the selection and downgraded to Indicated category. The Measured category is limited to the following dykes: Main1, Doris_Main2, ZoneSud_1, and ZoneSud_3.

Indicated Mineral Resources are generally blocks with an average distance between the three nearest drill holes of less than 60 m. It generally corresponds to a drill spacing of approximately 60 m x 60 m.

Inferred Mineral Resources are generally blocks with an average distance between the three nearest drill holes of less than 90 m. Dykes with a lower confidence in the geological interpretation or grade continuity were assigned to the Inferred category, regardless of drill hole spacing. Affected dykes are Doris_4, Doris_8, InterDoris_3, ZoneNord_2 and ZoneNord_3.

Final categories of all domains were manually edited to remove isolated clusters of blocks that did not show Reasonable Prospects for Eventual Economic Extraction (RPEEE), mainly in relation to the underground resource category.

The final classification of Mineral Resources is displayed in Figure 11-13 for the Main1 domain only.

11.10.2 Cut-Off Grade and Open Resource Pit Optimization

The cut-off grade used to report Mineral Resources was calculated with the parameters presented in Table 11-12. The open pit cut-off grade was calculated by BBA at 0.31% Li_2O and rounded down to 0.30% Li_2O for the open pit Mineral Resource (Resource Pit). The same parameters were used to establish an underground cut-off grade, but with total mining costs of C$100/t milled. The underground cut-off grade was established at 0.60% Li_2O.

To report a Mineral Resource that responds to a RPEEE, open pit optimizations were generated, using the parameters tabulated in Table 11-12. Figure 11-13 displays the constraining shell used to report the open pit portion of the Mineral Resource presented in this report.

To report an underground Mineral Resource assuming a RPEEE, SGS reviewed the material below the pit optimization used to report the open pit Mineral Resources, with the following constraints:

- Continuity of grades;
- Continuity thickness;
- Distance between mineralized zones; and
- Overall size of mineralized zones.

With these constraints, it was judged that only the Main1, ZoneNord_4 and ZoneNord_6 domains met all those criteria. A filter was applied to limit the blocks on the southwestern and northeastern edges of the pit to avoid isolated clusters of blocks. Figure 11-14 shows before blocks below the pit optimization above a cut-off grade of 0.60% Li_2O, whereas after illustrates the cleanup undertaken to remove all blocks not meeting RPEEE. No crown-pillar was assumed for the Mineral Resource Estimate.

Underground resources include blocks below the Resource Pit and above underground cutoff grade. It should be noted that the Resource Pit may be larger than the open pit used for the open-pit reserve estimates. Isometric views of the underground resources following cleanout are provided in Figure 11-15.

Table 11-12 Resource Pit Optimization Parameters

Parameters	Unit	Value
Tonnage milled	kt/y	996
In-situ Grade	% Li_2O	1.33%
Plant Head Grade	% Li_2O	1.33%
Sales Revenues		
Concentrate Price (5.5% Li_2O: 2.56% Li)	C$/t concentrate	1,264
Concentrate Transportation	C$/t concentrate	159.00
Operating Costs		
Overburden	C$/t mined	2.25
Mining Cost Ore	C$/t mined	3.46
Mining Cost Waste	C$/t mined	3.46
Reference Bench	m	284
Incremental Haulage Cost (per 6m)	C$/t mined	0.015
Ore Based Cost		
Tailing and rehandling	C$/t mined	3.00
Plant Processing	C$/t milled	7.98
Fixed operating costs	C$/t milled	46.99
Ore Based Cost	C$/t milled	57.97
Metallurgy		
Concentration Recovery	%	85
Concentrate Grade	% Li_2O	5.5
Cut-off Grade	% Li_2O	0.30
Geotechnical Parameters		
Geotechnical Parameters North Wall	deg	55
Geotechnical Parameters South Wall	deg	52
Material Densities		
Mineralized Material	t/m^3	Variable
Waste rock	t/m^3	Variable
Overburden	t/m^3	2.10

Figure 11-13 Isometric View of the Mineral Resource Classification – Main1 Dyke Only

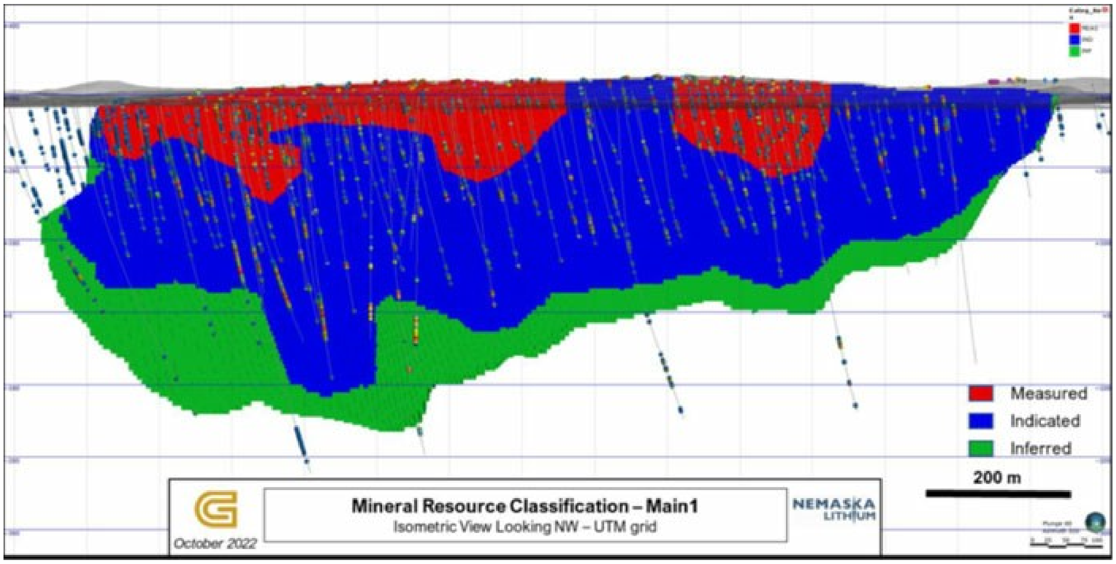


Figure 11-14 Isometric View of the Open Pit Optimization Shell and Mineral Resource Block Model

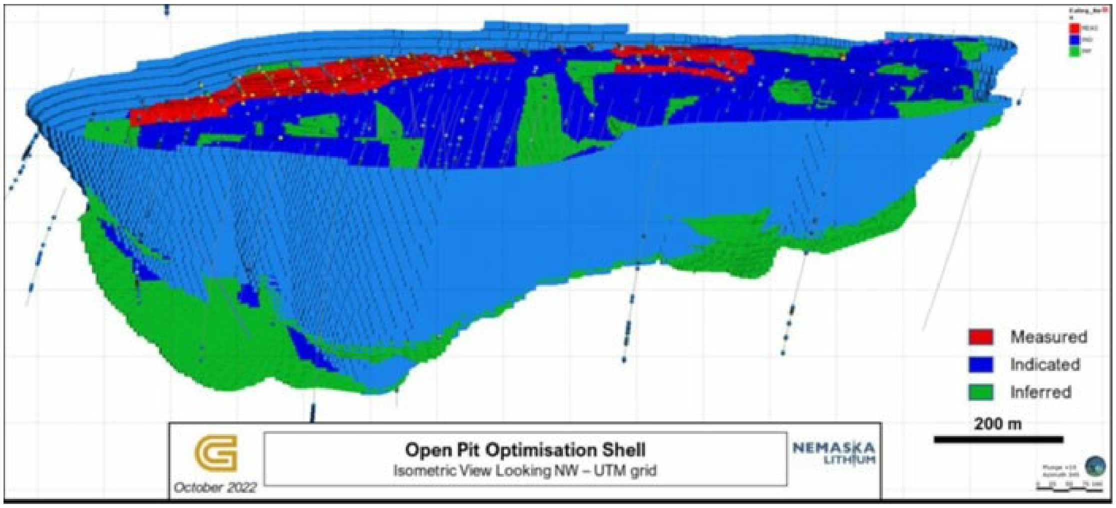
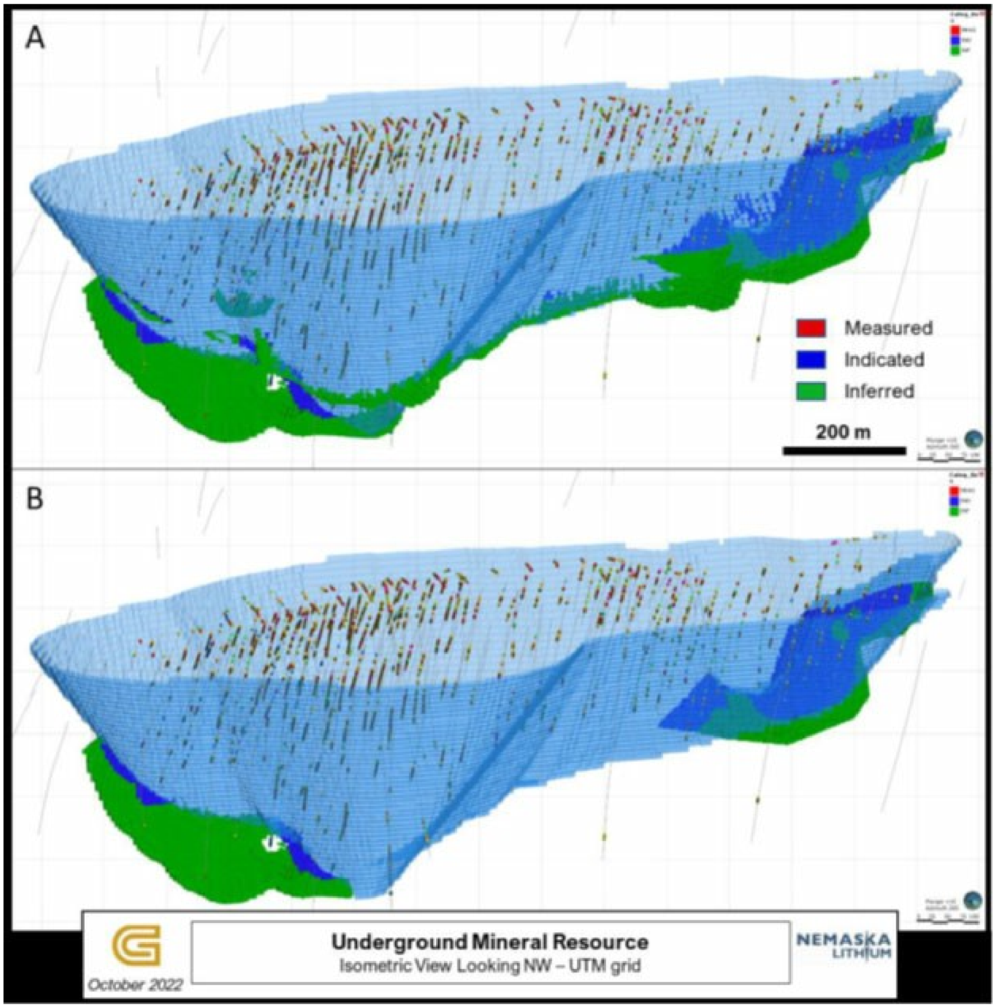


Figure 11-15 Isometric View of Underground Mineral Resource – Before (A) and After (B) the Cleaning Phase



11.10.3 Combined Open Pit and Underground Mineral Resources

The total and attributable Mineral Resource, exclusive of reserves, for the Whabouchi Project, combining resources potentially amenable to open pit and underground mining are listed in Table 11-13, using a lower cut-off grade of 0.30% Li_2O based on parameters presented in Table 11-12. The total Measured and Indicated (M&I) Mineral Resource, exclusive of reserves, is reported at 7.8 Mt @ 1.61% Li_2O for 126 kt of lithium oxide. The total Inferred Mineral Resource, exclusive of reserves is reported at 8.3 Mt @ 1.31% Li_2O for 108 kt of lithium oxide.

In 2017, NLI mined a portion of the deposit for a bulk sample used to feed a demonstration plant in Shawinigan. Approximately 19,000 tonnes of spodumene pegmatite at an average grade of 1.49% Li_2O are currently stockpiled about 600 m from the primary crusher and included as Indicated Mineral Resource.

These Mineral Resources are not Mineral Reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred Mineral Resources in this report are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

The QP is not aware of any factors or issues that materially affect the Mineral Resource Estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors, and additional risk factors regarding Indicated and Inferred Resources.

To calculate the resource exclusive of reserves, the proven reserves were deducted from the measured resources and probable reserves were deducted from indicated resources. The total and attributable Mineral Resource, exclusive of reserves, for the Whabouchi Project, combining resources potentially amenable to open pit and underground mining are listed in Table 11-13.

Table 11-13 Mineral Resources, Exclusive of Reserves

Category	Total Tonnes (Mt)	Grade (%Li_2O)	Attributable Tonnes (Mt)	Total Lithium Oxide (Mt Li_2O)	Attributable Lithium Oxide (Mt Li_2O)
Measured					
Indicated	7.8	1.61	3.9	0.126	0.063
M&I	**7.8**	**1.61**	**3.9**	**0.126**	**0.063**
Inferred	8.3	1.31	4.1	0.108	0.054

Notes:

1. Livent's attributable portion of the property's total mineral resources is 50%.
2. The Mineral Resource described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow the Best Practices outline by the CIM (2019).
3. Mineral Resources point of reference is in-situ and undiluted.
4. The Mineral Resource Estimate was prepared by SGS Geological Services mining QP.
5. The effective date of the Mineral Resource Estimate is December 31, 2022.
6. Density is applied by rock type and the proportion of waste inside each block, resulting in an average density of 2.77 g/cm^3.
7. Underground Resources are significantly different when compared to reserves and are reported combined for consistency.

For the Open Pit Mineral Resources:

8. The cut-off grade used to report the Open Pit Mineral Resources is 0.30% Li_2O.
9. Pit optimization parameters are described as follows:
    i. Spodumene concentrate of 5.5% Li_2O price: C$1,264 /t.
    ii. Metallurgical recoveries of 85%
    iii. Ore based costs of C$57.97 /t.
    iv. Northern wall angle of 55°
    v. Southern wall angle of 52°

For the Underground Mineral Resources:

7. The cut-off grade used to report Underground Mineral Resources is 0.60% Li_2O.
8. The classification has been adjusted to remove blocks that do not satisfy RPEEE requirements.
9. No crown-pillar was assumed below the pit optimization.

The QP is not aware of any factors or issues that materially affect the Mineral Resource Estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors, and additional risk factors regarding Indicated and Inferred Resources.

11.11 Mineral Resource Sensitivities to Cut-Off Grades

The sensitivity of the open pit resource to cut-off grade is summarized in Table 11-14. The tonnages and grade at differing cut-offs shown below are for comparison purposes only and do not continue an official Mineral Resource.

11.12 Mineral Resource Sensitivities to Spodumene Concentrate Selling Price

This section describes the sensitivity to various selling price of a 5.5% Li_2O spodumene concentrate. The following graphs show that the Whabouchi deposit is strongly constrained by the morphology of the orebody, which is well delineated by diamond drilling and locally by channel sampling. Very few tonnes are added when increasing optimization pit shell sizes between a selling price of C$1,264/t and C$2,500/t.

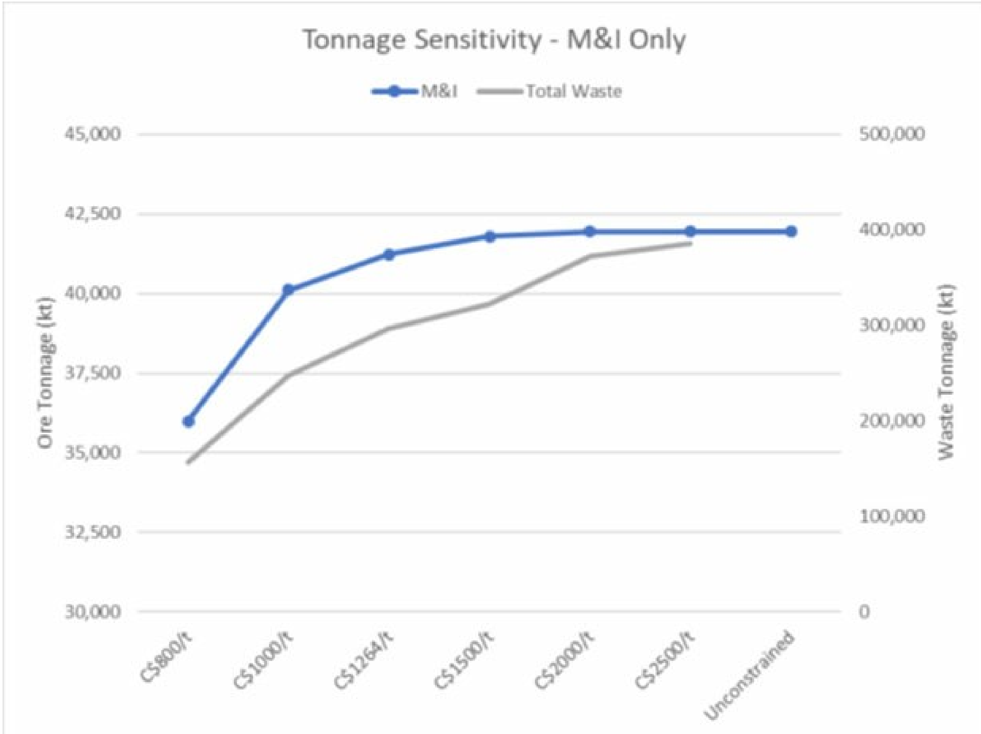
A significant increase of tonnage (4.1 Mt or 11% increase) in Measured and Indicated (M&I) resources is observed between the C$800 and C$1,000 shells (Figure 11-16). For increasing pit shell sizes after C$1,500, M&I tonnages stays virtually the same with an increase of 0.2 Mt (or 0.4% of M&I tonnage) up to the C$2,500 scenario due to limited drilling at depth. While there are very minor additions in tonnage beyond the C$1,264 pit (MRE pit shell), waste tonnage and strip ratios increase therefore increasing mining costs. Rapidly increasing strip ratios are mainly caused by the steeply dipping dykes concentrated in the center of the deposit.

Table 11-14 Sensitivity of the Open Pit Resource to Cut-off Grade – Base Case is Highlighted in Grey

Cut-off (%Li_2O)	Measured			Indicated			*Measured and Indicated*			Inferred		
	Mass (Mt)	Grade (%Li_2O)	Lithium Oxide (kt Li_2O)	Mass (Mt)	Grade (%Li_2O)	Lithium Oxide (kt Li_2O)	*Mass (Mt)*	*Grade (%Li_2O)*	*Lithium Oxide (kt Li_2O)*	Mass (Mt)	Grade (%Li_2O)	Lithium Oxide (kt Li_2O)
0.2	9.8	1.60	156	31.5	1.43	452	41.3	1.47	608	6.7	1.31	88
0.3	9.8	1.60	156	31.5	1.44	452	41.2	1.47	608	6.7	1.31	88
0.4	9.7	1.60	156	31.4	1.44	451	41.1	1.48	607	6.6	1.32	88
0.5	9.7	1.61	156	31.2	1.44	451	40.9	1.48	606	6.5	1.34	87
0.6	9.6	1.61	155	31.0	1.45	449	40.6	1.49	605	6.4	1.35	87
0.7	9.6	1.62	155	30.5	1.46	447	40.1	1.50	602	6.2	1.37	86
0.8	9.5	1.62	154	30.0	1.48	442	39.5	1.51	597	6.0	1.39	84
0.9	9.4	1.63	153	29.2	1.49	436	38.6	1.53	589	5.8	1.41	82
1.0	9.3	1.64	152	28.1	1.51	425	37.3	1.55	577	5.5	1.44	79

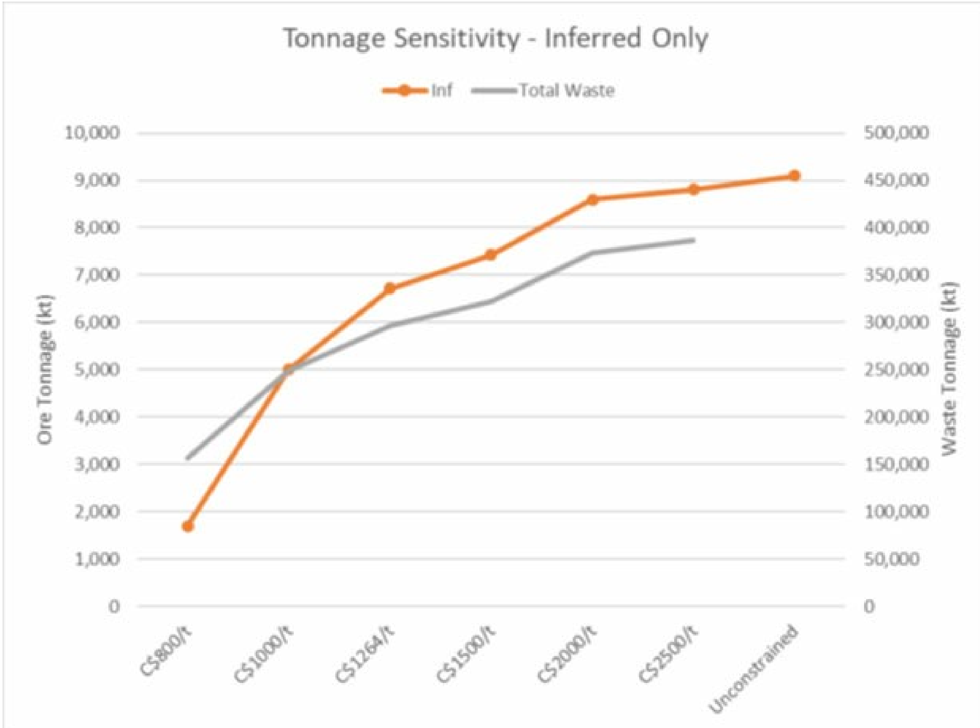
Note: Mineral Resources that are not Mineral Reserves do no have demonstrated economic viability. The tonnages and grade at differing cut-offs shown below are for comparison purposes only and do not continue an official Mineral Resource.

Figure 11-16 Tonnage Sensitivity to Increasing Selling Spodumene Price – Measured and Indicated (Base Case)



For Inferred resources (Figure 11-17), the tonnage sensitivity curve is different. There is no clear plateau, but a decrease in the slope after the C$2,000 scenario is observed. This is caused by the Inferred Mineral Resource being predominantly located at depth and limited by drilling.

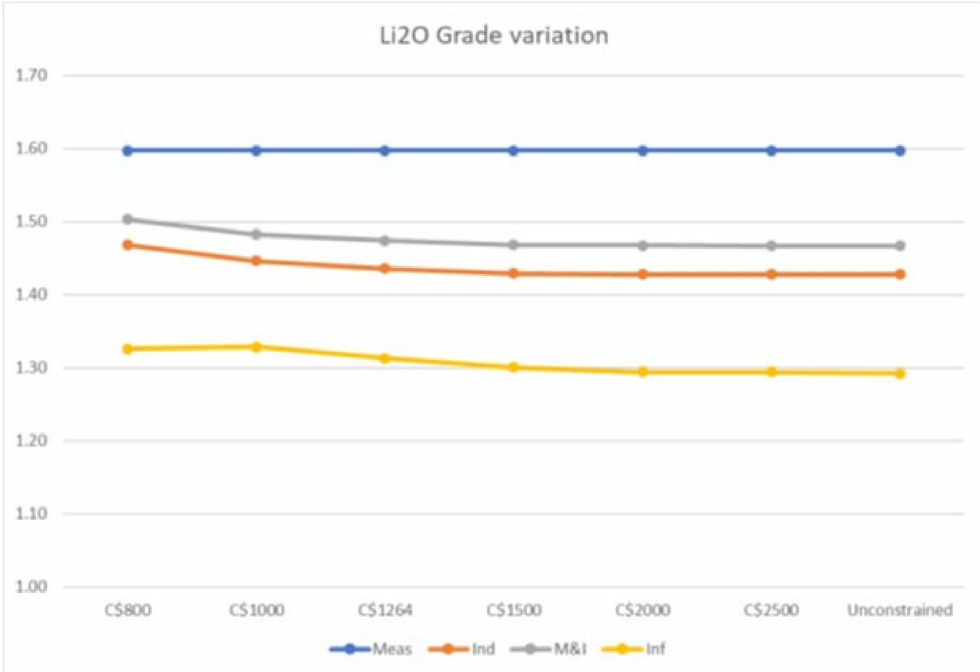
Figure 11-17 Tonnage Sensitivity to Increasing Selling Spodumene Price – Inferred Only



It is noteworthy that stripping ratios are strongly affected by increasing optimization shell size, ranging from stripping ratio of 6.2:1for the preferred C$1,264 scenario to a stripping ratio of 7.6:1 for a C$2,500 pit shell. More specifically, increasing the pit shells size adds significant waste tonnage for a minimal amount of mineralized material.

The impact of using a higher spodumene price is mainly limited to increasing Inferred Mineral Resources by approximately 2.1 Mt compared to the preferred scenario (or 4% of the total resource tonnage), but increasing waste tonnage by 89 Mt, which represents a 30% increase in waste tonnage. Lithium grades are weakly affected by the different selling price scenarios (Figure 14.18). Through further diamond drilling, the author believes there is a strong potential for the extensions of pegmatite dykes at depth.

Figure 11-18 Grade Variation for Increasing Selling Spodumene Price



11.13 Comparison with 2019 Technical Report (SGS)

The main gains and losses with the updated MRE are presented in this section, comparing the final pit optimization with the latest Mineral Resource Estimate of 2019 (SGS). All calculations are based on a "metal" content (kt Li_2O).

The updated model shows an increase in Measured and Indicated (M&I) Mineral Resources of approximately 57 kt Li_2O and a decrease of approximately 112 kt Li_2O in Inferred Mineral Resources. As displayed in the various waterfall graphs below, losses are essentially related to blocks previously categorized as Inferred, now downgraded and not a mineral resource due to insufficient drilling support at depth.

The main factors impacting the gains and losses in the new MRE are summarized here:

- **New Density Model:** new density measurements show that the spodumene pegmatite is slightly denser than what was previously assumed (2.76 g/cm³ vs 2.71 g/cm³, or +2%). The density of each block is calculated with the proportion of waste material estimated within the same block (%Waste x 3.04 g/cm³ +%Pegmatite x 2.76 g/cm³). The mean density of blocks is now at 2.77 g/cm³ compared to the previous 2.71 g/cm³.

- **Grade Increase:** the interpretation of pegmatite dykes in new geological model are more refined than the previous interpretation as an effort was made to better model internal dilution resulting in a global higher mean grade. For example, pit constrained indicated blocks are 8% higher in grade (1.44% Li_2O versus 1.33% Li_2O).

- **New Block Model:** this category includes the new geological model, the new resource classifications (including the transfer of inferred resources to indicated and new limits on the inferred and measured mineral resources), and new block model parameters, but excluding differences in grades and density.

- **Limits on Inferred Mineral Resources:** in the opinion of SGS, the inferred resources in the previous resource model were extended excessively at depth and were weakly supported by drill hole data. The new resource category partly reflects the revised limitation on inferred resources, especially at depth.

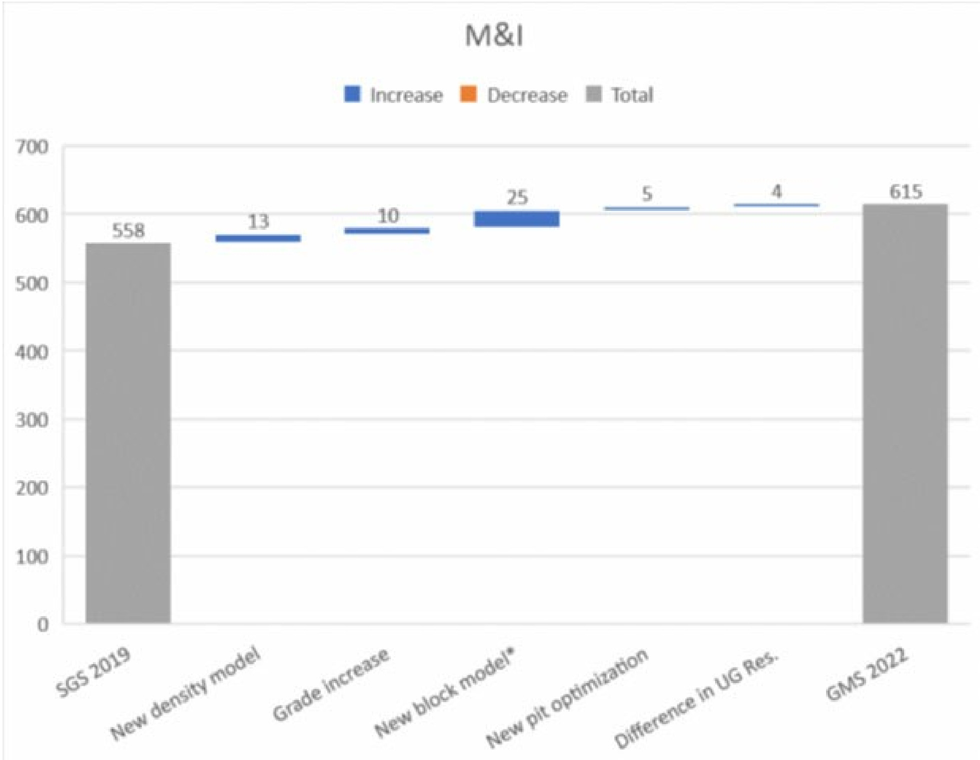
- **New Pit Optimization:** the resource constraining shells are different compared to the legacy model, including a significant change in pit slope angles (modified following additional geotechnical drilling), different mining and selling costs and an increased spodumene concentrate selling price.

- **Difference in underground resources:** this category essentially demonstrates the impact of limiting the inferred mineral resources at depth. To a lesser extent, it also includes a portion of blocks below the pit that do not satisfy RPEEE.

The following graphs show the main gains and losses pertaining to the factors described above. The M&I mineral resource has increased mainly due to the reclassification of some Inferred resources to Indicated, the global lithium grade increase and the new density model (Figure 11-19). Decreases in Inferred mineral resources are caused by the transfer of blocks to the Indicated category, and the decrease in Inferred category at depth (Figure 11-20). As discussed herein, the main factor of the global change in the total resource is the limitations on Inferred resources at depth, a loss mainly attributed to underground resources.

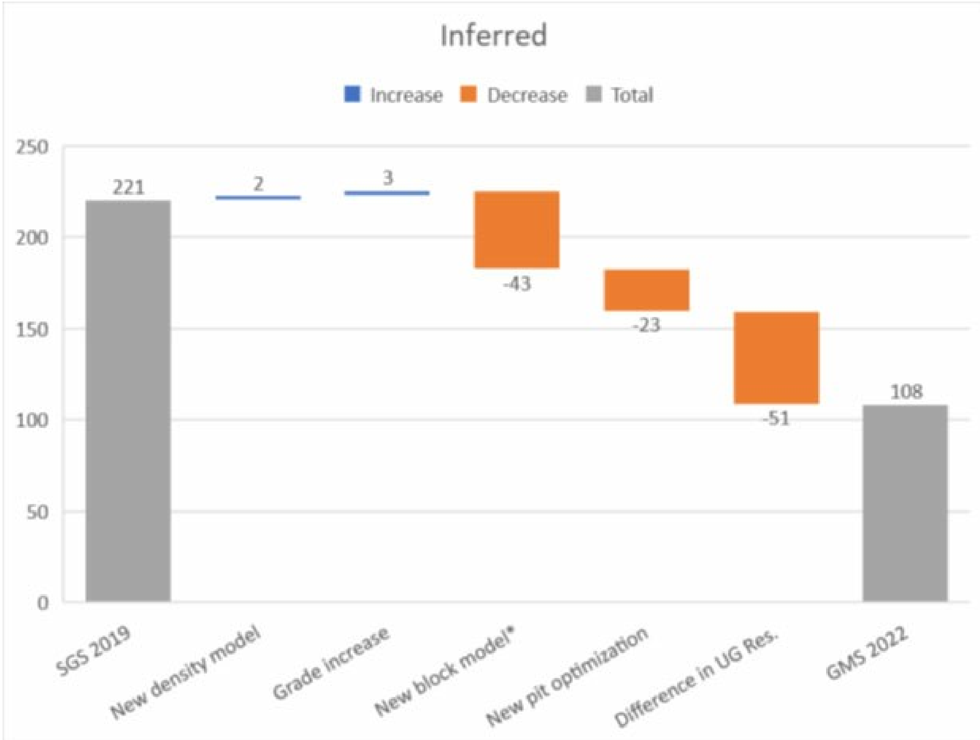
Figure 11-21 illustrates the portion of the Inferred mineral resources that was removed in the updated model, mainly due to the limitations on over-extrapolation at depth, which is weakly supported by drill hole data. The previous resource model classification of Inferred mineral resources is shown in transparent pink, whereas the new Inferred classification is shown in green: differences in limits applied to the model are easily recognizable when comparing both models.

Figure 11-19 Waterfall Chart of metal content of the Updated 2022 MRE – Measured and Indicated only (open pit and underground combined)



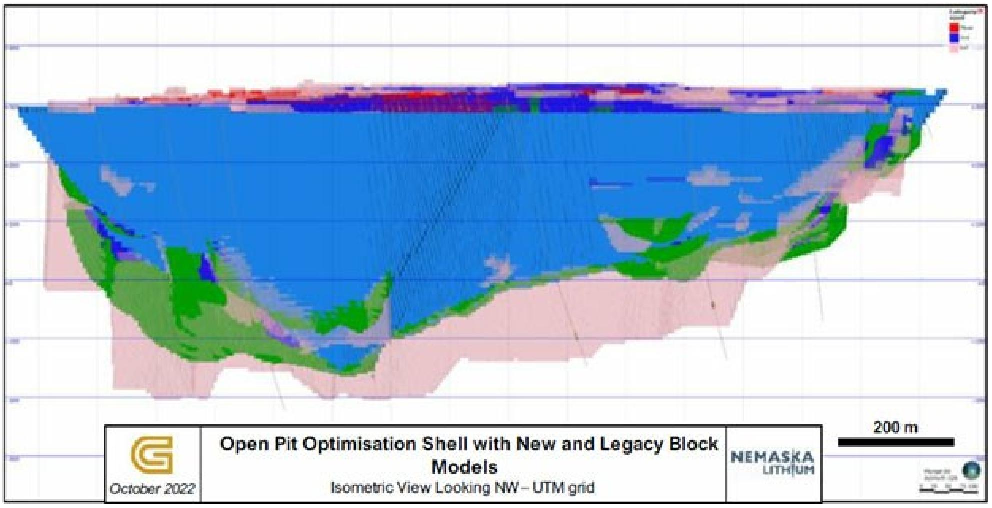
* New Model: includes new geological model, new classification (such as transfer of inferred to indicated resources and new limits on inferred resources), new block model parameters, but excluding increase in grades and density.

Figure 11-20 Waterfall Chart of metal content of the Updated 2022 MRE – Inferred only (open pit and underground combined)



* New Model: includes new geological model, new classification (such as transfer of inferred to indicated resources and new limits on inferred resources), new block model parameters, but excluding increase in grades and density.

Figure 11-21 Longitudinal View (Looking NW) of the 2022 and 2019 Resource Categories over the C$1,264 Pit Optimization – 2022 Inferred in Green and 2019 Inferred in Pale Pink



11.14 Kriging Neighbourhood Analysis

To support the selection of various interpolation parameters (block size, maximum sample in search ellipses and ellipsoid search ranges), a Kriging Neighbourhood Analysis (KNA) was conducted on the Main1 domain. This domain has robust variography and is the main dyke hosting the majority of tonnes in the deposit.

The following parameters remained fixed throughout the analysis:

- Variography parameters;
- Minimum of 7 samples selected;
- Minimum of 3 drill holes in the search;
- Dynamic anisotropy, or adaptative search ellipsoid orientation.

Figure 11-22 shows results of varying block size, while Figure 11-23 and Figure 11-24 show results of varying maximum sample selection and varying ellipsoid ranges respectively. Selected parameters are highlighted and generally represent the best alternative between slope of regression, kriging efficiency and the sum of negative weights.

Figure 11-22 Kriging Estimation Results for Varying Block Sizes. Orange: Kriging Efficiency, Blue: Slope of Regression and Grey: Kriging Variance

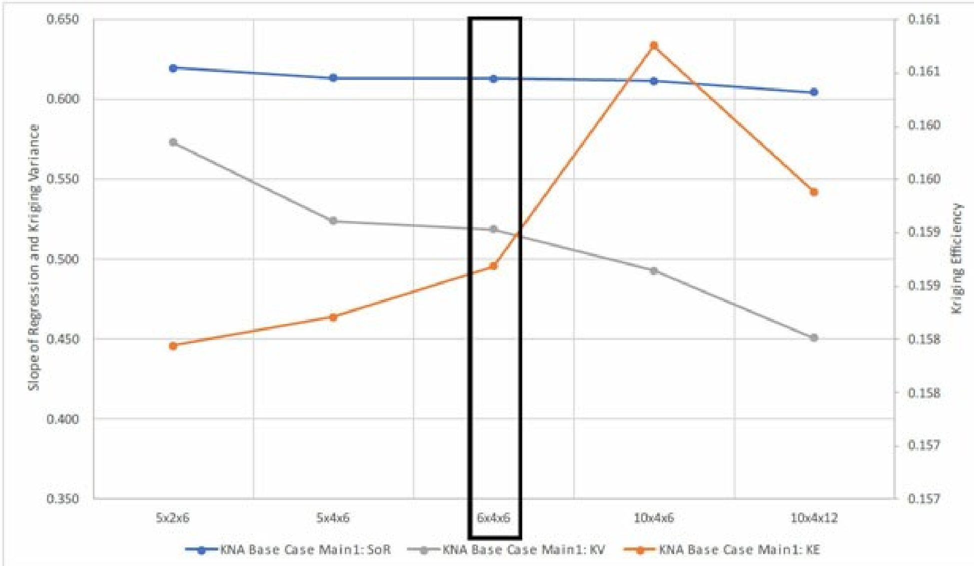


Figure 11-23 Kriging Estimation Results for Varying Maximum Sample Selection. Orange: Slope of Regression, Blue: Sum of negative weights and Grey; Kriging Efficiency

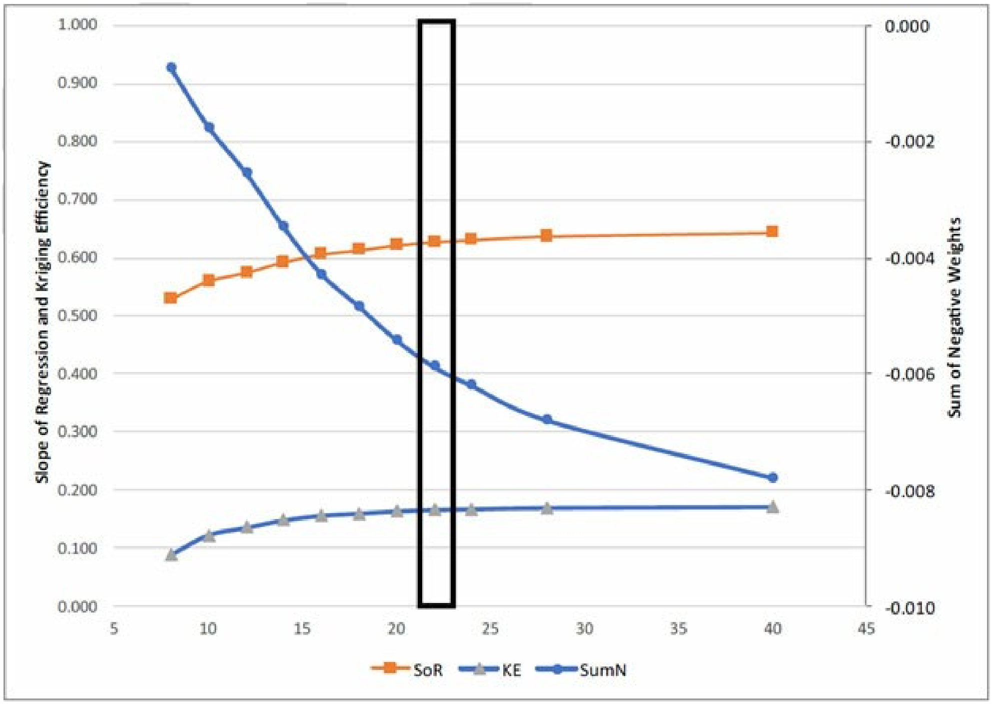
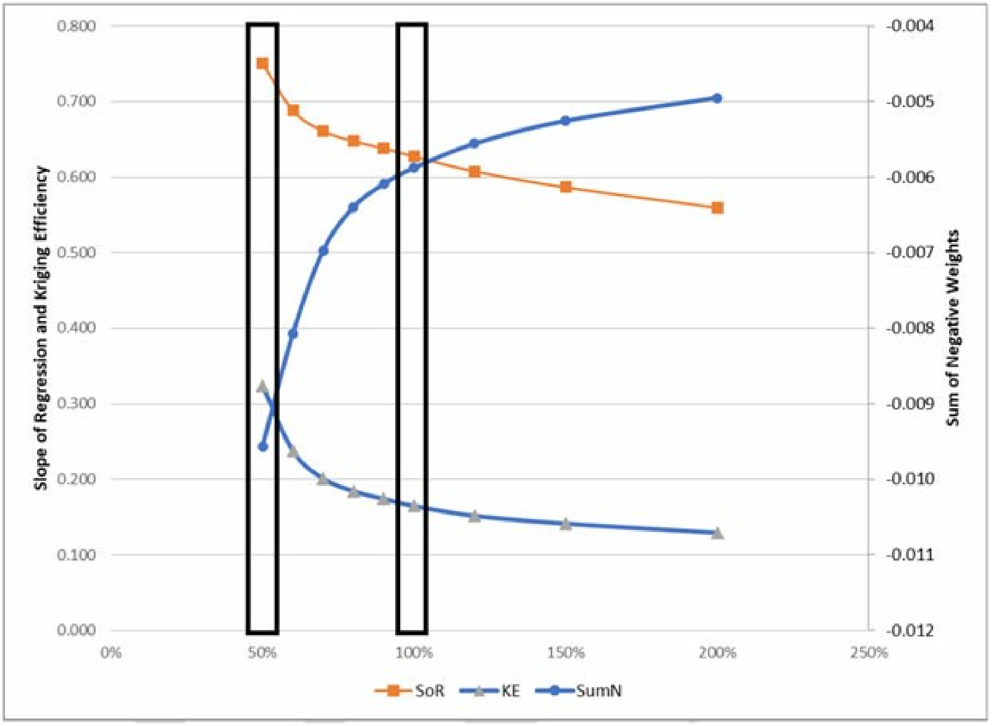


Figure 11-24 Kriging Estimation Results for Varying Search Ellipsoid Sizes. Orange: Slope of Regression, Blue: Sum of negative weights and Grey: Kriging Efficiency



11.15 QP Conclusions

The QP is of the opinion that with consideration of the recommendations summarized in Section 23 of this Technical Report Summary, any issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

12 MINERAL RESERVE ESTIMATES

The Whabouchi deposit will be mined using conventional open pit mining for the first 24 years of operation, followed by ten (10) years of underground mining. The Project life of mine (LOM) plan and subsequent Mineral Reserves are based on a lithium Spodumene selling price of C$1,264/t. It should be noted that the forward-looking selling price for Spodumene concentrate market studies (Section 16) used in the economic analysis (Section 19) vary from year to year and are higher than the static selling price used for the reserve estimate. Thus, as of the effective date of the reserve estimate, the mineral reserve estimates are more conservative (lower) than might be expected based on price forecasts presented in Section 16. The effective date of the Mineral Reserve estimate is December 31, 2022.

Development of the LOM plan included pit optimization, pit design, mine scheduling and the application of modifying factors to the Measured and Indicated Mineral Resources. The reference point for the Mineral Reserves is the feed to the primary crusher. The tonnages and grades reported are inclusive of mining dilution, geological losses, and operational mining losses.

The Mineral Reserves for the open pit component of the Whabouchi Mine were prepared by BBA's mining QP.

The Mineral Reserves for the underground component of the Whabouchi Mine were prepared by DRA's mining QP.

The Contributing Authors are of the opinion that no other known risks including legal, political, or environmental, would materially affect potential development of the Mineral Reserves, except for those risks already discussed in this Report.

Mineral Reserves are reported on a total basis for the property and on an attributable basis consistent with Livent's ownership interest in the property.

Table 12-1 presents the Mineral Reserves that have been estimated for the open pit component of the Whabouchi deposit which include 10.5 Mt of Proven Mineral Reserves at an average grade of 1.40% Li_2O and 16.0 Mt of Probable Mineral Reserves at an average grade of 1.27% Li_2O for a total of 26.5 Mt of Proven and Probable Mineral Reserves at an average grade of 1.32% Li_2O. To access these Mineral Reserves, 1.6 Mt of overburden and 70.2 Mt of waste rock must be mined, resulting in a stripping ratio of 2.8:1.

In 2017, NLI mined a portion of the deposit for a bulk sample used to feed a demonstration plant in Shawinigan. Approximately 19,000 tonnes of ore at an average grade of 1.49% Li_2O are currently stockpiled about 600 m from the primary crusher and are included in the Mineral Reserves.

Table 12-2 presents the Mineral Reserves that have been estimated for the underground part of the Whabouchi deposit. Table 12-3 presents the combined open pit and underground Mineral Reserves that have been estimated for the Whabouchi deposit.

Table 12-1 Whabouchi Open Pit Mineral Reserves

Category	Total Tonnes (Mt)	Attributable Tonnes (Mt)	Li_2O Grade (%)
Proven	10.5	5.2	1.40
Probable	16.0	8	1.27
Proven & Probable	**26.5**	**13.2**	**1.32**

1. The Qualified Person for the Open Pit Mineral Reserve Estimate is BBA's mining QP.
2. The effective date of the estimate is December 31, 2022.
3. The Mineral Reserves are based on a Spodumene concentrate selling price of $1,264/t CAD at an average concentrate grade of 5.50% Li_2O.A metallurgical recovery of 85.0% was used.
4. A cut-off grade of 0.40% Li_2O was used.
5. The stripping ratio for the open pit is 2.8 to 1.
6. The Mineral Reserves are inclusive of mining dilution and ore loss.
7. The reference point for the Mineral Reserves is the primary crusher.
8. Totals may not add due to rounding.

Table 12-2 Whabouchi Underground Mineral Reserves

Category	Total Tonnes (Mt)	Attributable Tonnes (Mt)	Li_2O Grade (%)
Proven	0.0	0	1.29
Probable	11.7	5.8	1.29
Proven & Probable	**11.7**	**5.8**	**1.29**

1. The Qualified Person for the Underground Mineral Reserve Estimate is DRA's mining QP.
2. A variable cut-off grade was used (0.5-0.72%), depending on mining method.
3. Mineral Reserves are based on a Spodumene concentrate selling price of $1,264/t CAD at 5.50% Li_2O.
4. A metallurgical recovery of 85.0% was used.
5. The Mineral Reserves are inclusive of mining dilution and ore loss.
6. The reference point for the Mineral Reserves is the primary crusher.
7. The economic viability of the Mineral Reserve has been demonstrated.
8. The reserves estimate has an effective date of December 31, 2022.
9. The Mineral Reserve is estimated with a variable COG which was calculated by mining method.
10. Li_2O content (tonnes) are estimated as reported (include dilution and mining recovery) values.
11. The Mineral Reserve is estimated with a stope mining recovery of 90%.
12. The Mineral Reserve includes both internal and external dilution.
13. External dilution included a mining dilution of 0.5 m on the hanging and footwalls for the long-hole mining method.
14. A minimum true mining width of 4 m was used.
15. Totals may not add due to rounding.

Table 12-3 Whabouchi Combined Open Pit and Underground Mineral Reserves

Category	Total Tonnes (Mt)	Attributable Tonnes (Mt)	Li_2O Grade (%)
Proven	10.5	5.2	1.40
Probable	27.7	13.8	1.28
Proven & Probable	**38.2**	**19.1**	**1.31**

1. Totals may not add due to rounding.
2. Mineral Reserves are based on a Spodumene concentrate selling price of $1,264/t CAD at 5.50% Li_2O.
3. A metallurgical recovery of 85% was used.
4. The Mineral Reserves are inclusive of mining dilution and ore loss.
5. The reference point for the Mineral Reserves is the primary crusher.
6. The economic viability of the Mineral Reserve has been demonstrated.
7. The reserves estimate has an effective date of December 31, 2022.

For the Open Pit Mineral Reserves:

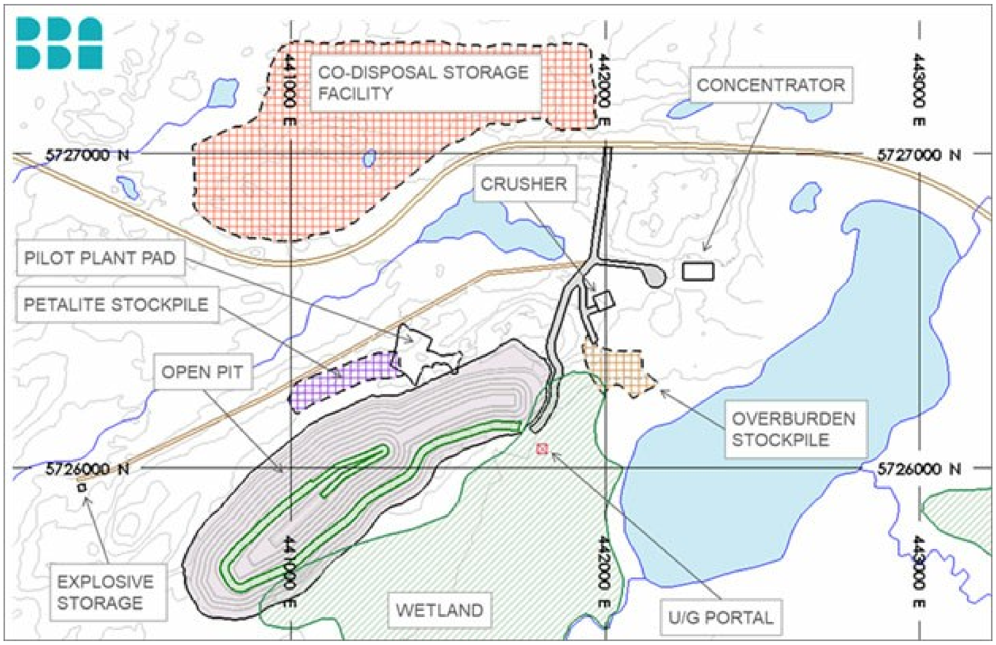
8. The Open Pit Mineral Reserves were prepared by BBA's mining QP.
9. A cut-off grade of 0.40% Li_2O was used.
10. Estimated variable mining costs (CAD) of $2.25/MT for overburden and $3.46/MT for rock, variable processing and tailings management costs of $11/MT milled, transportation costs of $159 MT of concentrate, and $46.7M/yr of fixed costs.
11. The stripping ratio for the open pit is 2.8 to 1.

For the Underground Mineral Reserves:

12. The underground Mineral Reserves were prepared by DRA's mining QP.
13. A variable cut-off grade was used (0.5-0.72%), depending on mining method.
14. Li_2O content (tonnes) are estimated as reported (include dilution and mining recovery) values.
15. The Mineral Reserve is estimated with a stope mining recovery of 90%.
16. The Mineral Reserve includes both internal and external dilution.
17. External dilution included a mining dilution of 0.5 m on the hanging and footwalls for the long-hole mining method.
18. A minimum true mining width of 4 m was used.

Figure 12-1 presents a general layout of the mine site.

Figure 12-1 Mine General Layout



12.1 Comparison to 2019 Mineral Reserves

The combined open pit and underground Mineral Reserves for the Whabouchi deposit have increased by 1.6 Mt at an average grade of 1.52% Li_2O. The contained quantity of Li_2O units increased from 477 kt to 502 kt. The open pit Mineral Reserves have been reduced by 1.4 Mt and 21 kt of Li_2O units. The reason the open pit reserves have decreased is because a smaller pit was considered in this DFS to ensure that the waste rock will fit within the Phase 1 CSF, as discussed in Section 12.3.2.2. The underground Mineral Reserves have increased by 3.0 Mt and 45 kt of Li_2O units. Table 12-4 presents the change between the current and 2019 Mineral Reserves.

Table 12-4 Comparison to 2019 Mineral Reserves

Description	Category	Tonnes (Mt)	Li_2O Grade (%)	Li_2O Units (kt)
Open Pit (OP)	Proven	-7.8	1.42	-111
	Probable	6.4	1.42	90
	P&P	**-1.4**	**1.47**	**-21**
Underground (U/G)	Proven	-0.7	1.43	-9
	Probable	3.7	1.48	54
	P&P	**3.0**	**1.50**	**45**
Total OP & U/G	Proven	-8.4	1.42	-120
	Probable	10.0	1.44	145
	P&P	**1.6**	**1.52**	**24**

12.2 General Parameters Common to the Open Pit and Underground Mineral Reserves

The following section discusses the geological information that was used for the open pit and underground mine designs and Mineral Reserve Estimates. This information includes the topographic surface, the geological block model and the material properties for ore, waste rock, and overburden.

The mine planning work carried out for the Feasibility Study Update was done using Hexagon's Mine Plan 3D Version 16.04 (formerly known as MineSight) and Deswik Version 2022.1.

The mine design work was completed using the UTM Zone 18 NAD83 coordinate system, in metric units.

12.2.1 Topographical Data

The mine design was carried out using a topographic surface based on 1 m contour intervals. The contours were supplied by NLI and derived from a LiDAR survey that took place in 2017. The topography surface was modified to consider both the area mined out during the 2019 construction period and the bulk sample pit.

12.2.2 Mineral Resource Block Model

The open pit mine design is based on the 3-dimensional geological block model that was prepared by SGS and presented in Section 11. The model is a sub-blocked model with a parent block size of 6 m wide, 4 m long and 6 m high. The model is rotated 330°. Table 12-5 presents the block model parameters.

Table 12-5 Block Model Dimensions

Model Direction	Parent Block Size (m)	Minimum Sub-block Size (m)	Sub-block Count (#)
Easting (x)	6	3	2
Northing (y)	4	1	4
Elevation (z)	6	3	2

Each block in the model contains the lithology, Li_2O grade, density, resource classification, and the percentage of waste rock. The blocks contain a percentage of waste rock since a dilution skin was added for narrow veins, as discussed in Section 11, and the Li_2O grades provided for the block consider the grades of waste rock component.

An overburden/bedrock contact surface was also provided as well as wireframes representing the Petalite zones. The petalite zones contain mineralization for which a lower metallurgical recovery has been applied for the Reserves.

12.2.3 Material Properties

The material properties for the different rock types are outlined below. These properties are important in estimating the Mineral Reserves, the equipment fleet requirements as well as the CSF and stockpile design capacities.

12.2.3.1 Bulk Density

Bulk density is an important measurement that converts volumes modeled by the geologists into tonnages and contained tonnes of lithium. It is also used to estimate mine equipment requirements. The methodology used to estimate the bulk densities, which are presented in Table 12-6, was presented in Section 11.

Table 12-6 Bulk Densities (t/m^3)

Material	Value
Mineralized Pegmatite	2.76
Waste Rock	3.04
Unmineralized Pegmatite	2.67
Overburden	2.10

12.2.4 Swell Factor

The swell factor reflects the increase in volume of the material from its in-situ state to its state after it has been blasted and loaded into the haul trucks. The swell factor is an important parameter that is used to determine the loading and hauling equipment requirements, as well as the CSF and stockpile designs. A swell factor of 40% has been considered as well as a compaction factor of 5% for when waste rock is placed in the CSF. A swell factor of 20% was used for overburden.

12.2.5 Moisture Content

Mineral Resources and Mineral Reserves are reported as in-situ dry tonnes. The moisture content reflects the amount of water present within the rock formation. It affects the estimation of haul truck requirements and must be considered during the payload calculations. The moisture content is also a contributing factor for the process water balance. A moisture content of 2% has been used for blasted rock, and 10% for overburden.

12.3 Modifying Factors that Affect the Open Pit Mineral Reserves

The following section presents the modifying factors that were applied to convert Mineral Resources into Mineral Reserves for the open pit component of the Whabouchi Project, as well as the Reserve pit optimization analysis and open pit design.

12.3.1 Mining Dilution and Mining Recovery

In every mining operation, it is impossible to perfectly separate the ore and waste due to the large scale of the mining equipment and the use of drilling and blasting. BBA used the Stope Optimizer tool in the Deswik Software (DSO) to model mining dilution for the Whabouchi open pit.

Table 12-7 presents the parameters that were used in the DSO tool. The parameters are based on mining using 120 tonne sized excavators equipped with 6.5 m3 buckets which have a total bucket width of 2.7 m. The DSO tool runs an algorithm on the sub-blocked model and provides “mineable shapes” (solids) that meet the minimum mining dimension criteria, include a dilution skin on both the highwall and footwall, and that still provides a mill feed grade above the cut-off grade. Mineralization that does not fall within a mineable shape as a result of the diluted grade falling below the cut-off is considered as a geological loss.

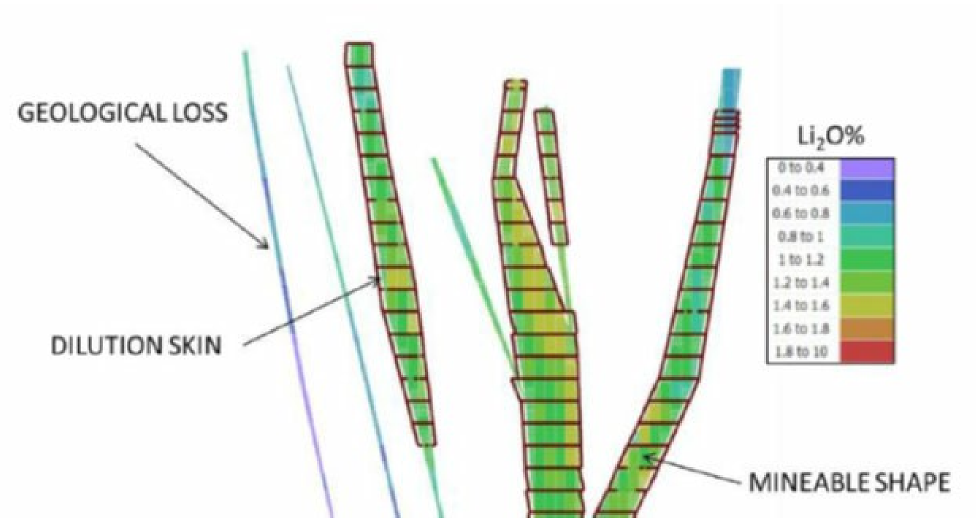
Prior to running the DSO algorithm, BBA removed the waste dilution skin that was included in the Mineral Resource model and worked with the in-situ mineralization thicknesses and grades.

Figure 12-2 presents a typical section through the deposit illustrating mineralization that did not pass as a mineable shape, the dilution skin, and mineable shapes that pass the minimum cut-off grade. Note that the selection of the cut-off grade is discussed in Section 12.3.2.

Table 12-7 DSO Parameters

Material	Unit	Value
Minimum Mining Width	m	2.50
Shape Height	m	6.00
Shape Length	m	10.00
Footwall Dilution Thickness	m	0.75
Highwall Dilution Thickness	m	0.75
Minimum Waste Pillar	m	3.00
Cut-off Grade (Li_2O)	%	0.40

Figure 12-2 Typical Section of DSO Shapes



In addition to mining dilution, BBA calculated the mining recovery that can be expected for the open pit mining operations. Like mining dilution, as a result of the large scale of the mining equipment and the use of drilling and blasting, it is not feasibly possible to recover 100% of the material that is identified as ore. For the mining recovery calculation, BBA considered that on average, a width of 0.25 m on each side (footwall and highwall) of the mineable shape would not be recoverable and would therefore be considered as an operational loss.

The following conventions have been used to calculate mining dilution, mining recovery:

$$\text{Mining Dilution (\%)} = \frac{\text{Dilution Waste Tonnage}}{(\text{In– Situ Ore Tonnage} + \text{Diltution Waste Tonnage})}$$

$$\text{Mining Recovery (\%)} = \frac{(\text{In– Situ Ore Tonnage} - \text{Operational Loss Tonnage})}{(\text{In– Situ Ore Tonnage})}$$

Within the final open pit, the modeling work resulted in 0.5 Mt of Measured and Indicated Mineral Resources being considered as geological losses, as well as the addition of 14.7% mining dilution (composed of 0.5% non-mineralized pegmatite and 14.2% amphiboles). The average mining recovery for the final open pit averages 97.2%.

Mining dilution is a critical modifying factor when estimating Mineral Reserves since there is an added cost associated with the processing of the waste dilution material. In addition to the increased cost, the waste rock may contain deleterious elements that can be harmful in the ore concentration process. There may also be deleterious elements in the waste rock that may affect the quality of the concentrate produced.

To counter this, the NLI process flowsheet includes X-Ray Transmission (XRT) ore sorters whose purpose is to remove the amphibole dilution prior to the concentration process. Section 14 of this Report presents the process flowsheet and discusses how the ore sorters have been sized based on the mining dilution that has been modeled.

Upon completion of the mining dilution and mining recovery modeling, the sub blocked model was converted into a regularized model that was used for the pit optimization analysis and mine planning. Each block in the regularized model includes the quantities of in-situ mineralized tonnes and grades as well as the tonnages of mining dilution and ore loss.

12.3.2 Reserve Pit Optimization

A pit optimization analysis has been completed to determine the extent of the deposit that can be mined and processed economically. Because this analysis differs from the pit optimization process used to define resources, in that discounted cash flows are considered, we refer to it in this Section as the Reserve Pit. The pit optimization was done using the pseudo-flow algorithm in the Economic Planner module of Hexagon MinePlan 3D. The algorithm determines the economic limits of the open pit at a range of selling prices based on input of mining and processing costs, revenue per block, and operational parameters such as the metallurgical recovery, pit slopes and other imposed physical constraints. The pseudo-flow algorithm provides similar results as the Lerch-Grossman algorithm with the benefit of shorter computing times. Inferred Mineral Resources were not considered in the pit optimization and mine plan and have therefore been considered as waste rock.

The pit optimization used the activity-based costing methodology that distinguishes fixed costs from variable costs. Fixed costs are time related with no direct production drivers while variable costs are directly related to a production driver in the system. The total fixed costs per year were then allocated to the system bottleneck, which for the Whabouchi deposit is the concentrator throughput of 996,000 t/y. This number was back-calculated to 1.132 Mt/y of feed to the crusher which considers an ore sorter circuit recovery of 88%.

Table 12-8 presents the input parameters that were used for the pit optimization analysis, with all figures being presented in Canadian Dollars. The input parameters were developed from the results of the 2019 Feasibility Study with adjustments for inflation and updated project knowledge. The costs used for the pit optimization are inputs and should therefore not be confused with the final operating costs for this Report.

The pit optimization considered pit slopes that were developed by WSP and presented in Section 12.2.4. The pit slopes were adjusted for the pit optimization to account for the ramp system that is added in the Reserve Pit design stage.

Table 12-8 Reserve Pit Optimization Parameters (in $CAD)

Item	Unit	Value
Operating Costs - Variable		
Mining Cost (Overburden)	$/t (mined)	2.25
Mining Cost (Ore & Waste)	$/t (mined)	3.46
Incremental Bench Mining Cost (per 6 m)	$/t (mined)	0.015
Reference Bench for Incremental Cost	m	284
Processing Cost	$/t (processed)	7.98
Tailing Management Cost	$/t (processed)	3.00
Operating Costs – Fixed		
Mining Cost	$ M/y	10.0
Processing Cost	$ M/y	11.4
Maintenance Services & Consumable	$ M/y	7.5
G&A	$ M/y	10.9
Sustaining Capital	$ M/y	6.9
Total Fixed Costs	$ M/y	46.7
Production Bottleneck		
Bottleneck	Mt/y	1.132
Bottleneck Cost	$ /t	41.1
Other Parameters		
Spodumene Concentrate Sales Price	$/t	1,264
Concentrate Transportation Costs	$/t	159
Concentrate Grade	%	5.50
Metallurgical Recovery	%	85.0
Metallurgical Recovery (Petalite Zone)	%	67.0
Pit Slope	degrees	50

12.3.2.1 Cut-Off Grade

A marginal cut-off grade is calculated to determine if material within the open pit should be sent to the mill for processing or sent to the waste rock pile. The marginal cut-off grade, which is referred to as the “Open Pit Discard Cut-off” in the CIM Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines, differs from the breakeven cut-off grade since mining costs are excluded from the calculation. The reason for excluding the mining costs is that material already defined to be within the limits of the open pit must be mined regardless of if it is classified as ore or waste to access the bench below.

The only exception where a mining cost would be included in the marginal cut-off grade calculation is if there is an incremental cost for mining ore relative to mining waste. The following calculation was used to calculate the marginal open pit mining cut-off grade for Whabouchi deposit.

$$\text{Marginal COG} = \frac{(\text{Processing Cost} + \text{Tailings Cost} + \text{Bottleneck Cost})\text{ x Concentrate Grade}}{(\text{Sales Price} - \text{Transportation Cost})\text{ x Mill Recovery}}$$

Using the economic parameters presented in Table 12-8, the marginal open pit cut-off grade was calculated to be 0.31% Li_2O. To ensure an average feed grade to the processing plant that can provide a high-quality concentrate, the cut off grade for the open pit was artificially elevated to 0.40% Li_2O.

There are approximately 70,000 tonnes of Measured and Indicated Mineral Resources that have a grade between 0.31 and 0.40% Li_2O. The average grade of these resources is 0.36% Li_2O.

12.3.2.2 Reserve Pit Optimization Results

Using the cost and operating parameters, a series of 25 pit shells was generated by varying the selling price (revenue factor) from $379 to $1,517/t CAD. The tonnages and grades associated with each of the pit shells are presented in Table 12-9. The Net Present Value (NPV) of each shell was calculated assuming a selling price of $1,264/t CAD of spodumene concentrate, a discount rate of 8% and an annual production rate of 1.132 Mt/y of ore. It is important to note that the NPV's presented do not include initial and sustaining capital costs and are therefore not indicative of the Project's NPV, they are merely used as a relative comparison of the pit shells.

Figure 12-3 presents the results in a graphical format and Figure 12-4 presents a typical section through the deposit highlighting several of the important pit shells.

The pit shell with the maximum NPV is the Revenue Factor (RF) 0.50, which has an NPV of $1,925 M CAD. This pit shell contains 97 Mt of waste rock, while the area allocated for the CSF can store a maximum of 75 Mt of waste rock. To minimize the amount of waste rock, an evaluation was done to determine at which depth it would become more economic to transition the open pit into an underground mining operation. The evaluation considered an underground mining cost of $80/t CAD, and the results are presented in Figure 15.5. The chart shows that underground mining would be more economic beyond the RF 0.80 pit, at which point the mining cost in the open pit on a dollar per tonne of ore equate to the same $80/t CAD. As the pit deepens (beyond the RF 0.80 pit), and the strip ratio continues to rise, open pit mining becomes less favorable relative to underground mining. For the RF 0.50 pit shell with the maximum NPV, open pit mining is more economic than underground mining, with an incremental mining cost of $33/t CAD of ore, however, the quantity of waste rock exceeds the 75 Mt capacity. It was, therefore, decided to limit the open pit to the RF 0.47 pit shell and consider an underground mine below. An in-pit waste rock storage facility was evaluated, but not retained for the DFS since it would be operationally challenging.

The RF 0.47 pit shell that was selected to guide the design of the ultimate pit contains 28.8 Mt of Measured and Indicated Mineral Resources with an average diluted Li_2O grade of 1.32% and a stripping ratio of 2.6:1.

Table 12-9 Reserve Pit Optimization Results

Revenue Factor	Ore (Mt)	Li_2O (%)	OB (Mt)	Waste Rock (Mt)	Total Waste (Mt)	Stripping Ratio	Mine Life (y)	NPV ($M)
0.300	0.6	1.81	0.1	0.1	0.1	0.2	1	137
0.320	1.9	1.66	0.2	0.5	0.7	0.4	2	382
0.340	3.5	1.57	0.3	1.4	1.7	0.5	3	623
0.360	8.5	1.46	0.5	7.0	7.5	0.9	8	1,165
0.380	12.4	1.42	0.7	14.2	14.9	1.2	11	1,446
0.400	18.1	1.39	1.0	31.1	32.0	1.8	16	1,726
0.410	20.3	1.38	1.1	38.8	39.9	2.0	18	1,798
0.420	21.8	1.37	1.2	44.6	45.8	2.1	19	1,835
0.430	23.5	1.36	1.3	49.9	51.2	2.2	21	1,862
0.440	24.7	1.35	1.4	55.0	56.4	2.3	22	1,880
0.450	26.5	1.34	1.5	62.3	63.8	2.4	23	1,898
0.453	26.7	1.33	1.5	63.5	65.0	2.4	24	1,900
0.460	27.5	1.33	1.6	67.4	69.0	2.5	24	1,906
0.470	28.8	1.32	1.7	74.3	76.0	2.6	25	1,913
0.490	31.4	1.31	1.9	91.1	93.0	3.0	28	1,925
0.500	32.2	1.30	2.0	96.5	98.5	3.1	28	1,925
0.520	33.9	1.30	2.2	109.5	111.7	3.3	30	1,923
0.530	34.8	1.29	2.3	116.7	118.9	3.4	31	1,919
0.540	35.4	1.29	2.3	122.0	124.4	3.5	31	1,916
0.550	36.2	1.28	2.4	128.4	130.8	3.6	32	1,910
0.600	41.4	1.26	3.1	179.5	182.6	4.4	37	1,869
0.700	43.4	1.25	3.4	202.9	206.2	4.8	38	1,842
0.900	45.3	1.23	3.8	233.3	237.0	5.2	40	1,798
1.000	45.6	1.23	3.9	239.2	243.1	5.3	40	1,791
1.200	46.0	1.23	4.0	249.8	253.8	5.5	41	1,776

Figure 12-3 Reserve Pit Optimization Results

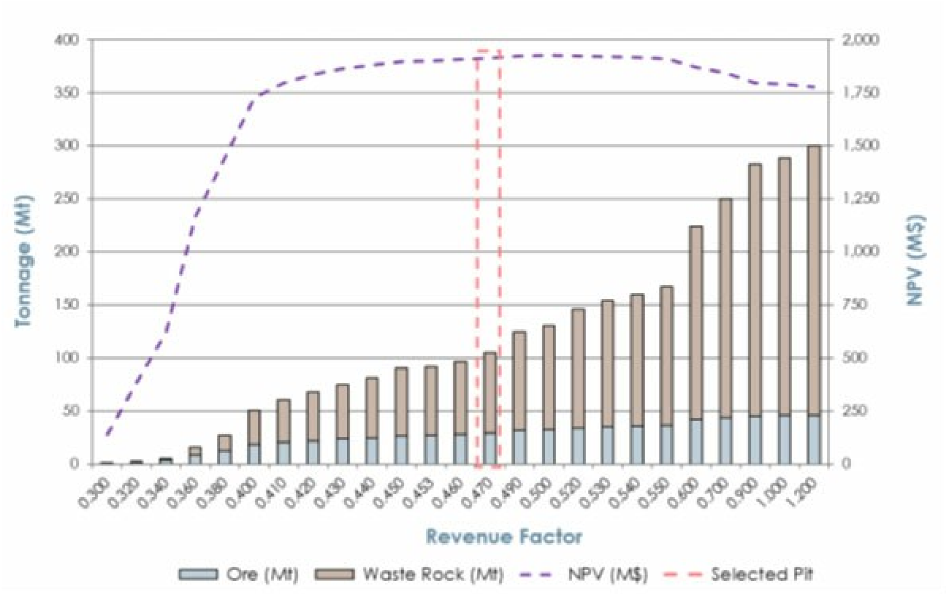


Figure 12-4 Reserve Pit Optimization Shells

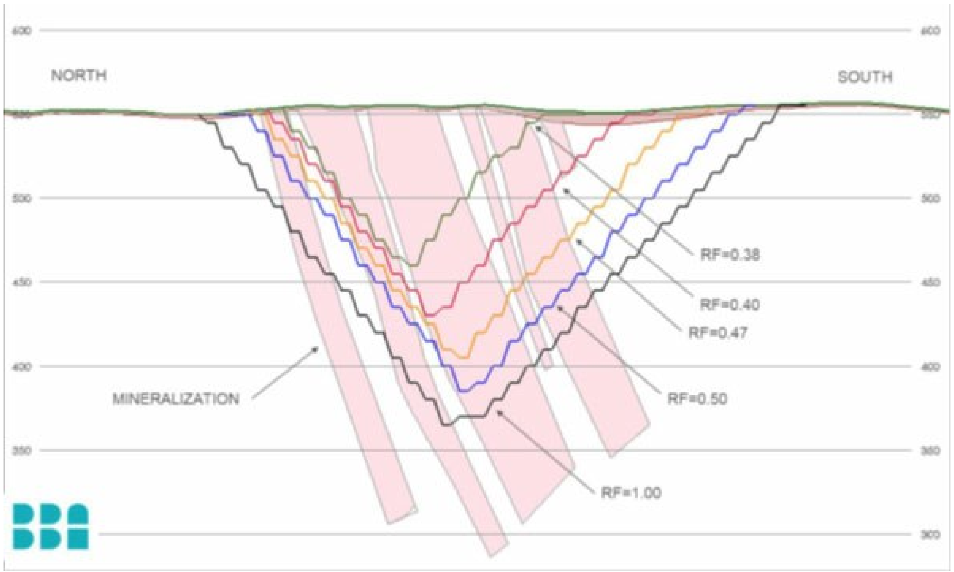
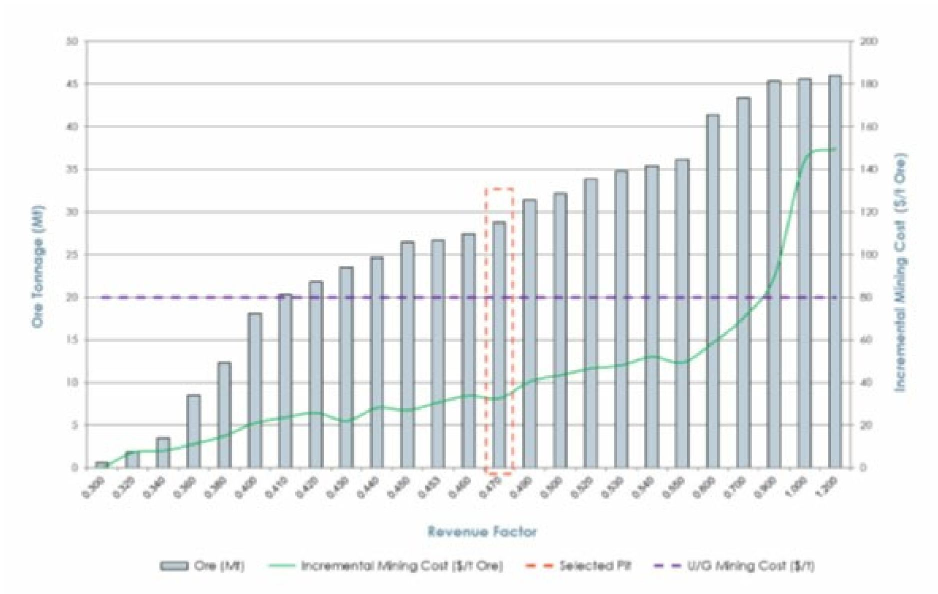


Figure 12-5 Open Pit versus Underground Mining



12.4 **Open Pit Design**

Using the results of the pit optimization analysis, an operational pit was designed, which is the basis of the LOM plan. This pit design uses the selected pit shell as a guide and includes smoothing the pit wall, adding ramps to access the pit bottom and ensures that the pit can be mined safely and efficiently. The following section provides the parameters that were used for the open pit design and presents the results.

12.4.1 Bench Height

A 12 m high bench height was selected for the Whabouchi Project. Ore and waste rock will be mined in two (2), 6 m flitches. Open pit ultimate walls in competent rock will be excavated is double benches, vertical height 24m.

12.4.2 Geotechnical Pit Slope Parameters

Golder Associates Ltd., (now WSP) was retained by Nemaska Lithium Inc. (NLI) to carry out a feasibility level geotechnical investigation. The scope of work and contractual basis for the work are presented in "Phase 2 – Recommended Field Investigation, Geotechnical Characterization and Hydrogeological Modelling", dated October 19, 2021.

The open pit rock slope design recommendations were developed based on the results of the 2021 – 2022 geotechnical investigation, involving three (3) geotechnical drillholes totalling 653 m, eight (8) hydraulic conductivity tests, nine (9) vibrating wire piezometer installations, and a pumping test, completed by WSP. Open pit overburden slopes are not characterized. Golder provided assumptions to the mine planners for overburden slope designs (Golder 2022).

This slope design study considers the Whabouchi open pit, ranging in depth from 120 to 220 m. The open pit area consists of approximately 1 m to 10 m of glacial till overburden, overlying dark, fine-grained gabbro to basaltic host rock, intruded by a spodumene bearing pegmatite dykes. The basalts and dykes have a steeply dipping structural fabric forming footwall conditions on the north wall and hanging wall conditions on the south wall. The rockmass is very strong and competent. The stability of individual benches controls slope design on the north wall and end walls. The south wall design considers bench scale and potential deep seated toppling.

The slope design terminology and the rock slope design recommendations, inter-ramp and overall, are presented after the text on Figure 12-6 and Figure 12-7, respectively and discussed below.

Figure 12-6 Typical Slope Design

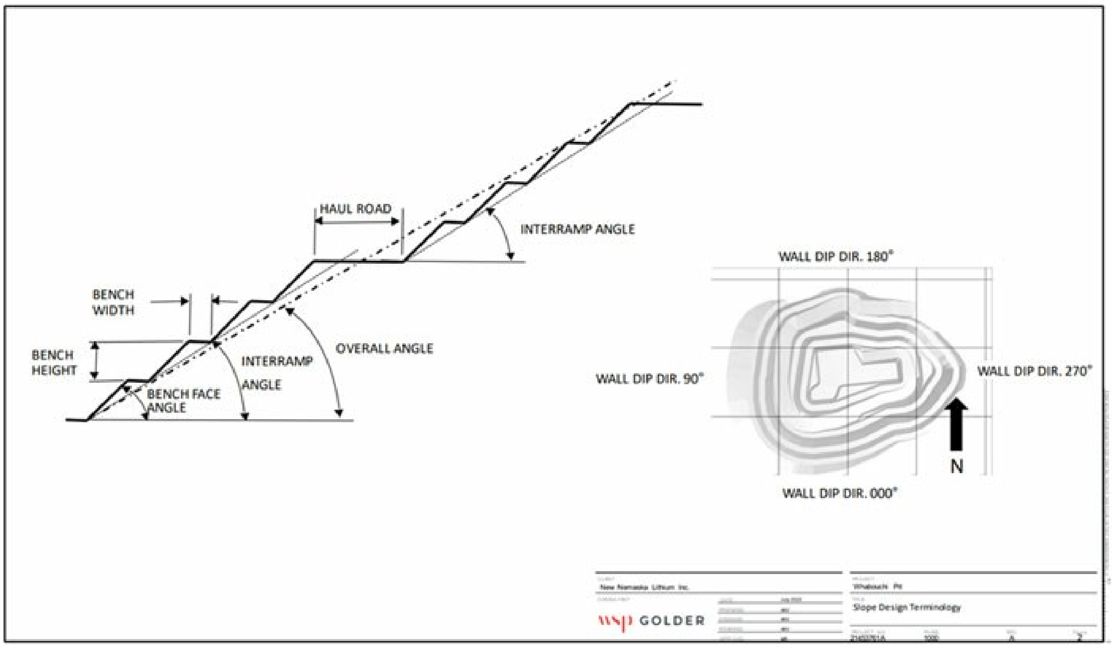
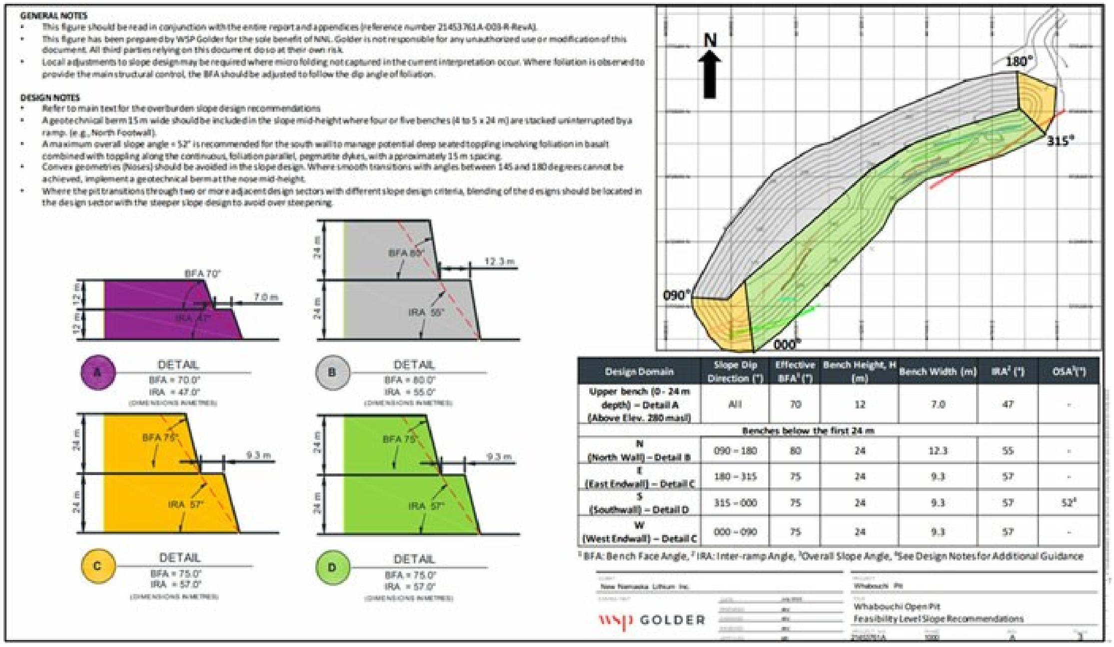


Figure 12-7 Slope Recommendation



Design Domain	Slope Dip Direction (°)	Effective BFA[1] (°)	Bench Height, H (m)	Bench Width (m)	IRA[2] (°)	OSA[3] (°)
Upper bench (0 - 24 m depth) – Detail A (Above Elev. 280 masl)	All	70	12	7.0	47	-
Benches below the first 24 m						
N (North Wall) – Detail B	090 – 180	80	24	12.3	55	-
E (East Endwall) – Detail C	180 – 315	75	24	9.3	57	-
S (Southwall) – Detail D	315 – 000	75	24	9.3	57	52[4]
W (West Endwall) – Detail C	000 – 090	75	24	9.3	57	-

[1] BFA: Bench Face Angle, [2] IRA: Inter-ramp Angle, [3] Overall Slope Angle, [4] See Design Notes for Additional Guidance

12.4.2.1 Overburden Slope Design

Overburden is assumed to be less than 10 m thick and consist of glaciolacustrine deposits, till, and local soft sediments. In some areas it has been observed to reach 25 m thick. No stability analyses have been completed for the overburden, nor has the overburden been differentiated or domained.

The following assumptions could be used by the project mine planners for the Whabouchi overburden slope design:

The slopes are comprised mainly of till, with an angle of repose of approximately 30 degrees.

- As per Chapter S-2.1, r. 14 of Quebec's occupational health and safety regulations "Regulation respecting occupational health and safety in mines", the overburden benches must have a slope less than its natural embankment and have a cleared bench width of a minimum 2 m.
- A bench width of 8 m at the toe (after accounting for crest loss) of the overburden slope is recommended to provide sufficient catchment in the long-term and sufficient width for access by workers in small equipment.

There will be adequate surface and groundwater management/control to reduce localized sloughing and slope erosion.

A diversion ditch should be constructed at the overburden/rock interface. The ditch could adversely reduce the travel width of the bench and therefore the bench might need to be proportionally wider.

- It is assumed that the ditch would be 0.5 m deep and excavated at 2:1 angle. The geometry of the toe bench and its drainage ditch width should be confirmed based on the hydrology of the site and the equipment that NLI will assign to work this bench.

During operations, either as pro-active maintenance or as part of progressive closure reclamation, the overburden will be re-sloped to facilitate revegetation or to facilitate covering them with short haul clean rock that will act as riprap, providing erosion protection.

While the overburden may be excavated in benches depending on the height, the benches will be removed as part of re-sloping for closure and erosion control. If design for closure is a consideration for granular materials, a closure overburden slope geometry of 2.5:1 or 3:1 is suitable. For weaker materials such as silts and clays, 4:1 slopes or flatter may be required.

Overburden is assumed to be consistent from ground surface, extending to the bedrock contact.

12.4.2.2 Rock Slope Design

Open pit slope design requires integration of stability at the bench scale, the inter-ramp scale and overall slope scale. The slope design terminology is illustrated in Figure 12-6. The bench configuration is defined by the bench height, vertical bench separation, the catch-bench width, and the bench face angle (BFA). The inter-ramp slope is a stack of benches uninterrupted by a ramp or wider bench. The overall slope is formed by a series of inter-ramp slopes, separated by haul roads from the pit crest to pit toe. For details regarding the slope design methods refer to the Golder 2022 slope design geotechnical report.

The rock mass at Whabouchi varies from Strong (compressive strength of the order of 50 to 100 MPa) to Very Strong (compressive strength of the order of 100 to 200 MPa) and of Fair quality in Gabbro (RMR76 of the order of 60 to 75) and of Good quality in the Basalt and Pegmatite (RMR76 of the order of 80 to 90). Given the modest depths of the proposed pits relative to the rock mass strength of the lithological units, rock mass failure is not a concern at Whabouchi and is not expected to control slope design. Rather, the pit slope design will mainly be controlled by optimizing the achievable bench geometries due to structural fabric and specifically bench scale and possible deep-seated toppling on the south wall.

The prominent 75° to 80° dipping foliation and continuous foliation parallel dykes are expected to control the performance of footwall and hanging wall bench face angles. This will be exacerbated by cross-cutting structures that could increase crest-losses.

Domaining of the open pit was primarily based on the orientation of foliation. The pegmatite dykes are considered major structures with potential to deliver groundwater recharge to the slopes and potential to contribute to deep seated toppling. Wall orientations were assessed for kinematic planar, wedge, and toppling failures. Stability for the end walls and north wall were assessed using the 2D limit equilibrium program Slide® to determine factors of safety for roTckmass and wall scale planar and bi-linear failures, respectively. Stability of the south wall was assessed for bench scale and wall scale toppling using the 2D block toppling stability software, RocTopple®.

The RocTopple assessment was completed to determine the factors of safety of bench-scale and inter-ramp toppling failure considered the foliation and foliation parallel dykes for a range of column spacing and slope water conditions based on fracture frequency data, direct shear laboratory strength testing, the NLI geological model and engineering judgment. Bench scale toppling is expected and can be managed with inclined pre-shear bench face angles. The potential for deep seated toppling is more speculative and has been assessed for completeness and due diligence.

12.4.2.3 Rock Slope Failure Mechanisms

Three (3) types of slopes (domains) with specific failure modes and design criteria can be used to characterize pit walls at Whabouchi:

a) **FOOTWALL (FW):** Benches and Walls controlled by foliation planes, with high potential for planar failure to develop. The mean foliation orientation is consistent with the primary pegmatite intrusion. Areas where smaller dykes branch off from the main intrusion can result in localized variations in the foliation orientation, and will have to be addressed by continuous wall mapping during the operation.

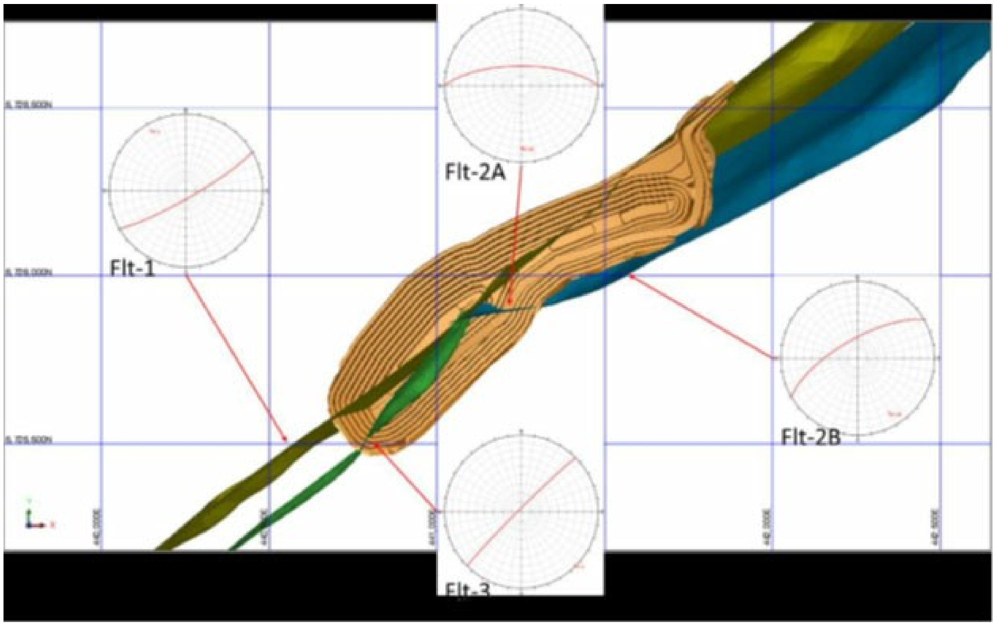
b) **HANGING WALL (HW TOPPLING):** Walls controlled by foliation planes where toppling may occur, with some potential for toppling failure to develop along the steeply dipping foliation structures combined with continuous, foliation parallel, pegmatite dykes.

c) **OBLIQUE WALL (HW) OR END WALL (EW):** Walls not directly controlled by foliation (Hanging wall, End Walls), with potential for wedges to occur, but offering the opportunity to excavate steeper inter-ramp (IRA) angles, provided that proper controlled blasting trim techniques are implemented.

Major Structures

The NLI structural model includes three faults that are parallel or subparallel to the dyke trends. Two of the faults (Fault 1 and Fault 3) are mined out by the proposed pit and will only be exposed on the end walls. Fault 2 crosses the south wall at an oblique angle, while the eastern extent of Fault 1 intersects the northeast wall at an oblique angle. Bench scale raveling may occur locally involving Fault 2 and the eastern portion of Fault 1. Table 12-10 presents the fault traces on the planned ultimate pit as well as the interpreted orientations at the Whabouchi Mine.

Table 12-10 Major Structure Interpretation at the Whabouchi Mine

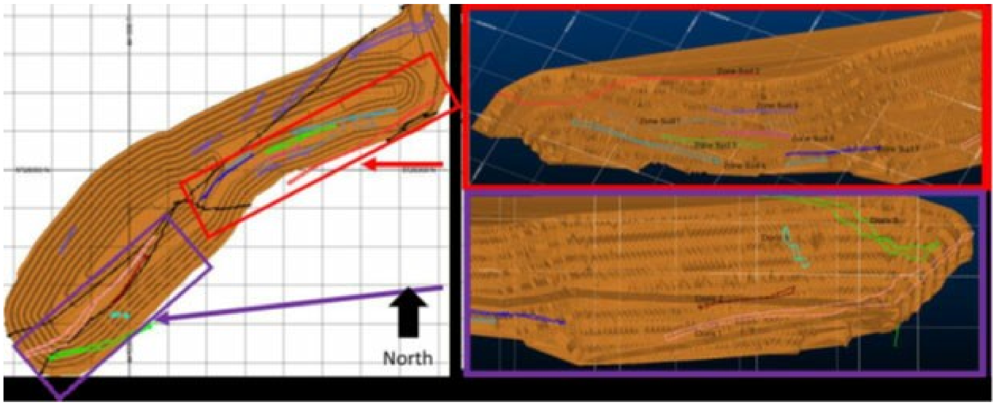


Structure ID	Dip (°)	Dip Direction (°)
Flt-1	82	150
Flt-2A	69	000
Flt-2B	70	330
Flt-3	87	315

Interactions of the faults with foliation or major joint sets could create the potential for localized failures. In particular, Fault 2A daylighting on the south wall has the potential to increase toppling potential by acting as a basal plane that interacts with foliation and the foliation sub-parallel dykes.

The pegmatite dykes are considered major structures with potential to deliver groundwater recharge to the slopes and potential to contribute to deep seated toppling on the south wall, shown Figure 12-8. The orientation of the dykes is generally sub-parallel to the mean foliation orientation. Areas where smaller dykes branch off from the main intrusion can result in localized variations in the foliation orientation, contributing to localized failures. The south wall (hanging wall) has an increased dyke density that has the potential to contribute to deep seated toppling, given the dyke's continuous, moderately dipping characteristics.

Figure 12-8 Increased Density of Dykes on the Southeast and Southwest Final Pit Walls



The dykes presented in Figure 12-8 have an approximate spacing of 10 – 15 m, with the highest dyke density occurring in the southeast pit wall area.

12.4.2.4 Standard Of Care in Pit Slope Development

Rock slope design for open pit mines and quarries includes consideration of both mining economics (the steepness and overall stability of the slopes) and operating safety (particularly mitigation of rockfall hazards). Design factors affecting pit economics can be modified to optimize financial returns. Design factors related to safety cannot be compromised, whether for permanent or temporary slopes, and slope designs must be implemented to meet the current standard of care in the mining industry for operating safely below rock slopes. This standard includes incorporating effective catch berms into pit slopes.

The minimum standard of care for safety in development of mine slopes is defined in Canada by provincial mining codes. In addition, operating practices, and slope designs to enhance operator safety are often developed at the corporate level, and these may be supplemented at the Operating level based on site conditions at individual pits.

Based on Golder (now WSP's) experience in similar hard rock mines, catch-bench widths less than 6.5 m wide do not provide adequate catchment for control of rockfall. In Quebec the duty of the Engineer is to demonstrate that the catch-bench width is adequate. A catch-bench width of 9.3 m has been assigned for every 24m vertical separation (double bench) based on the Modified Ritchie equation (Ryan and Pryor, 2000).

Catch berms are designed to provide safe working conditions below the slope, and therefore it is equally important to develop effective catch berms for both Ultimate and interim or Phase slopes, since the safety risks are identical for both cases. Where the Ultimate slope is designed assuming expensive perimeter blasting techniques, it may be warranted to avoid perimeter blasting costs for Phase slopes by accepting flatter interim slopes with alternative design bench configurations.

12.4.2.5 Acceptance Criteria

Factor of Safety (FoS), commonly applied for pit slope design, depends on the reliability of the information used for the analyses, the sensitivity of calculated FoS to the design assumptions, and the consequences of slope failure. Typical values for acceptable FoS incorporates consequences of failure, as defined in Table 12-11 (after Read & Stacey, 2009).

Table 12-11 Pit Slope Design Acceptance Criteria (adapted from Read & Stacey, 2009)

Slope Scale	Consequences of Failure[2]	Acceptance Criteria[1]		
		Min. Static FoS	Min. Dynamic FoS	PoF (max) P[FoS ≤1]
Bench	Low – High	1.1 – 1.2	Not Applicable	25 – 50% (50%)
Inter-Ramp	Low	1.15 – 1.2	1.0	30%
	Medium	1.2	1.0	20%
	High	1.2 – 1.3	1.1	10%
Overall	Low	1.2 – 1.3	1.0	15 – 20%
	Medium	1.3	1.05	5 – 10%
	High	1.3 – 1.5	1.1	≤5 %

1. Needs to meet all acceptance criteria.

2. Semi-quantitatively evaluated.

Slope designs are commonly based on FoS values of 1.1 to 1.3 when: the consequences of potential failures are of limited economic impact; slope failures can be remediated; and failures do not involve any critical facilities, do not represent a significant risk to safety of personnel, and do not threaten the continuous supply of ore. The following criteria are used for this study:

- FoS = 1.1 as the Acceptance Criteria for kinematic analyses of bench angles and bench geometry.

- FoS = 1.2 as the Acceptance Criteria for analyses for the inter-ramp slopes and overall slope stability including the final slope configuration with ramps and catch-berms.

12.4.2.6 Slope Design Assumptions and Recommendations

The following slope design assumptions were used for the Whabouchi rock slope design:

- Operating single bench height of 12 vertical meters and double bench height of 24 vertical meters.

- The maximum achieved Bench Face Angle (BFA) at Whabouchi where the bench is not undercut by structures and with excellent blasting practices, is expected to be 75°. For context, where the structural controls are limited:

  - Production blasting can typically consistently deliver a 55° to 60° BFA.

  - Trim or cushion blasting can typically consistently deliver a 65° to 70° BFA.

  - Vertical pre-shear blasting can typically deliver a 70° to 75° BFA, and sometimes better.

  - Poor blasting and scaling practices will reduce the achieved slope angles by 5° to 10°.

- A geotechnical berm 15 m wide should be included in the slope mid-height where four or five benches (4 to 5 x 24 m) are stacked uninterrupted by a ramp.

- Rock fall hazard requires control using adequately wide (effective) and debris free catch benches above the workers according to Quebec regulations. An equation based on bench height (Ryan & Pryor, 2000) is widely used in the mining industry for estimating an effective catch-bench width after any assumed back-break and is represented by:

$$\text{Effective CBW (m)} = 4.5 + 0.2 \blacksquare \text{Bench Height (m)}$$

  - This equation would yield an effective Catch Bench Width (CBW) of 6.9 m, rounded to 7.0 m, for the 12 m high single benches and 9.3 m width for a 24 m double bench.

- Convex geometries (Noses) should be avoided in the slope design. Where smooth transitions with angles between 145 and 180 degrees cannot be achieved, implement a geotechnical berm at the nose mid-height.

- Where the pit transitions through two or more adjacent design sectors with different slope design criteria, blending of the designs should be located in the design sector with the steeper slope design to avoid over-steepening.

- Initial benching will be single benching, to manage the expected poorer ground with higher fracture frequency expected in the upper 24 m of the open pit wall rock slopes. Single benching will also help manage early mining less than excellent blasting practices, which is also expected.

- In more competent rock, below the near surface, double benching (2 x 12 m) will occur on all final walls.

  - Bench faces shall be excavated with pre-split-controlled blasting over the full 24 m, not two single benches. Double height pre-splits eliminate the need for two set-ups and eliminate the need for a step-out between pre-split rows that often results in a ledge that can launch rockfall and increase its horizontal trajectory. The angle of the pre-split should be optimized by the operation.

    - East and West End Walls: There are limited structural controls on the east and west end walls. It is assumed that vertical pre-split blasting will be used. This practice may need to be reviewed after the first several benches.

    - North Walls: It is expected that the benches typically backbreak to the mean foliation dip. Cushion blasting or inclined pre-shear blasting should be used. This practice should be reviewed and optimized during mining of the phases. An allowance for 3 m of crest loss has been accounted for in the design.

    - Local adjustments to slope design may be required where micro folding not captured in the current interpretation occur. Where foliation is observed to provide the main structural control, the BFA should be adjusted to follow the dip angle of foliation.

    - The open pit benches with potential for bench scale toppling must be excavated with an inclined pre-shear. Golder recommends the inclination be 75 degrees as a starting point.

  - No sub-grade drilling in the vicinity of final benches shall occur, to reduce crest loss.

  - As a starter best-practice, it is recommended that operations restrict production blasts to within 50 m of an unblasted presplit line. Once presplit is shot, production blasts will be taken to within 10 m of the presplit and then a trim shot used to clean the face. Given that larger production shots may be more likely to damage the final walls, all blasts shall be monitored, and blast designs shall be adjusted to avoid this.

12.4.2.7 Single Benching for the First One or Two Benches

The initial benches at Whabouchi are more weathered with more open fracturing and consistently more prone to crest lost. Review of the core and core logging data at Whabouchi indicates increased fracturing and a lower rock quality designation (RQD) in the upper portions of geotechnical and exploration drillholes. It is also assumed that the first bench may also be susceptible to some less-than-excellent controlled blasting results, as the blasting crew learns from experience, the best methods (stab holes, trim shots, cushion blasting) and mine operators scale. To allow flexibility to the operation, the surface (0 – 24 m) should be excavated in a single-bench configuration with a BFA of 70°.

12.4.2.8 Slope Design Below the First One or Two Benches

Rock quality and structural conditions generally appear favorable for the development of double benches below the top of the more weathered and fractured near-surface bedrock interval. Given the prevalent foliation and dyke orientation the open pit can be described as having hanging wall conditions on the north wall, end wall conditions to the east and west, and foot wall conditions on the south wall. The engineering geology, design issues, and slope design recommendations by wall domain follow.

- **South Domain (Hanging wall):** Experience from the previous operations as well as kinematic stability analyses results indicate potential for significant structurally controlled toppling-planar failures along foliation planes and continuous, foliation parallel pegmatite dykes on the west wall.
  - An effective BFA of 75° developed with an inclined pre-shear is required to manage bench scale toppling. The actual inclination of the inclined pre-shear can be modified based on experience with initial benches, however, a vertical pre-shear will exacerbate toppling.
  - It is crucial that the pre-shear shots have no stemming. Stemming will cause crest and wall damage.
  - Bench and slope stability in this sector will be sensitive to the presence of groundwater pressures along structures where slope stability is marginal.
  - It is critical that depressurization is verified through the monitoring of vibrating wire piezometers, particularly for this hanging wall slope where multi-bench stability may be sensitive to groundwater pressure.
  - Prisms are required in tandem with the vibrating wire piezometer installations to monitor for potential deep seated toppling and to allow time for remedial measures such as a horizontal drain program.
  - WSP Golder recommends that the mine planners keep the ramp on the south wall to keep the overall slope angle at 52° or lower to manage the potential for deep seated toppling. Alternatively, the south wall should be interrupted with geotechnical benches to achieve the same overall slope angle. There is a potential for steepening if deep seated toppling can be shown not to be a concern based on slope performance and additional information.
- **North Domain (Footwall):** It is expected for the benches to backbreak to the mean foliation dip. For the design, the mean foliation dip measured in oriented core (75° - 80°) was used to define the bench face angle. Careful final wall blasting and scaling will also be required to successfully apply this design and reduce back-break.
  - The single and double bench face angles in this domain are expected to break back to foliation. This may mean that cushion blasting and scaling to foliation will suffice, rather than an inclined pre-shear along the assumed foliation dip for the next bench.
  - An inclined pre-shear parallel to the assumed foliation dip may be attempted. The risk to operations could be when foliation dips flatter than the pre-shear and the resulting slab fails after mucking and scaling.
- **Other Domains (End walls)**, where toppling and planar failure involving foliation is not a concern because the walls are too oblique to the discontinuity set, steep bench face angles should generally be achievable with careful blasting, excavation, and scaling. While wedge or toppling failures may occur locally, available structural data does not indicate these failure mechanisms to be a widespread control on bench design.

The design sectors were developed and adjusted based on rock mass domains, the wall orientation and the major kinematic controls influencing the pit wall stability. The main kinematic controls for each Design Sector are discussed in the slope design geotechnical report and summarized in Table 12-12.

Table 12-12 Pit Slope Recommendation

Domain	Bench Face Angle (º)	Bench Height (m)	Bench Width (m)	Inter-ramp Angle (º)	Overall Slope Angle (º)
North Wall	80	24	12.3	55	-
South Wall	75	24	9.3	57	52
Upper Benches	70	12	7.0	47	-

Note:

1. A geotechnical berm 15 m wide should be included in the slope mid-height where four or five benches (4 to 5 x 24 m) are stacked uninterrupted by a ramp. (e.g., North Footwall).
2. A maximum overall slope angle = 52° is recommended for the south wall to manage potential deep seated toppling involving foliation in basalt combined with toppling along the continuous, foliation parallel, pegmatite dykes, with approximately 15 m spacing.
3. Convex geometries (Noses) should be avoided in the slope design. Where smooth transitions with angles between 145 and 180 degrees cannot be achieved, implement a geotechnical berm at the nose mid-height.
4. Where the pit transitions through two or more adjacent design domains with different slope design criteria, blending of the designs should be located in the design domain with the steeper slope design to avoid over steepening.
5. Bench configuration: Double Bench – 2 x 12 m.

12.4.2.9 Updated Pit Designs Using the Geotechnical Recommendations

Any updated pit designs developed using these recommendations should be reviewed by a geotechnical engineer to validate that the slope designs have been applied correctly.

12.4.3 Haul Ramp Design

The haul ramps were designed for haulage with 64-tonne sized rigid frame mining trucks, with an overall width of 25 m. For double lane traffic, industry practice indicates the running surface width to be a minimum of 3½ times the width of the largest truck. The overall width of a 64-tonne rigid frame mining truck is 5.5 m which results in a running surface of 19.3 m. The allowance for berms and ditches increases the overall width to 25 m. Single-lane traffic has been considered for the final 4 benches (48 m in elevation), reducing the overall ramp width to 18 m. Figure 12-9 presents the haul road configuration for 2-way traffic. A maximum ramp grade of 10% was used.

Figure 12-9 Ramp Design

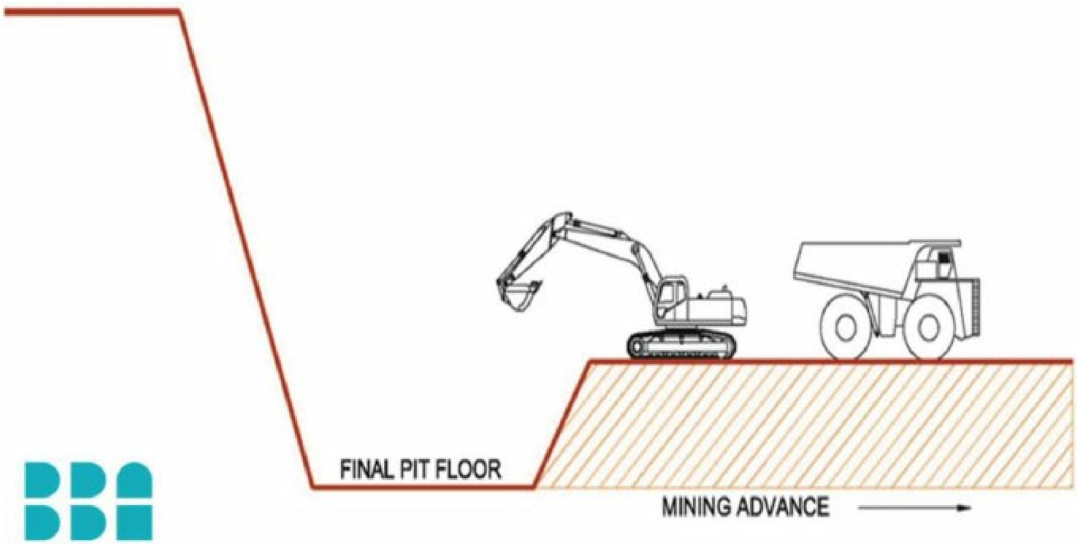
12.4.4 Minimum Mining Width

A minimum mining width of 30 m was considered for the pit design. This width must be respected to ensure that a 64-tonne haul truck, which has a turning radius of 12 m, can safely enter the mining area and make a 180° turn to be positioned for loading.

12.4.5 Final Bench Access

To reduce the strip ratio as much as is feasibly safe and efficient, the access ramp has not been designed to the bottom of the pit. When mining the final bench, the haul trucks will be positioned on the bench crest rather than on the bench toe. Figure 12-10 illustrates this operating scenario, commonly referred to in the industry as a good-bye cut. The final bench has been designed at a height 6 m.

Figure 12-10 Final Bench Access



12.4.6 Open Pit Design Results and Mineral Reserves

The open pit that has been designed for the Whabouchi Project is approximately 1,400 m long and 400 m wide at surface. The total surface area of the pit is roughly 42 ha. The pit ramp enters at the 280 m elevation on the east side of the pit and runs down the southern wall, wrapping around to the North wall at the 175 m elevation. The ramp becomes single-lane access at the 135 m elevation. A switchback is incorporated at the 111 m elevation and the ramp heads down to the bottom of the pit which is at the 82 m elevation. The deepest part of the pit is 230 m below surface.

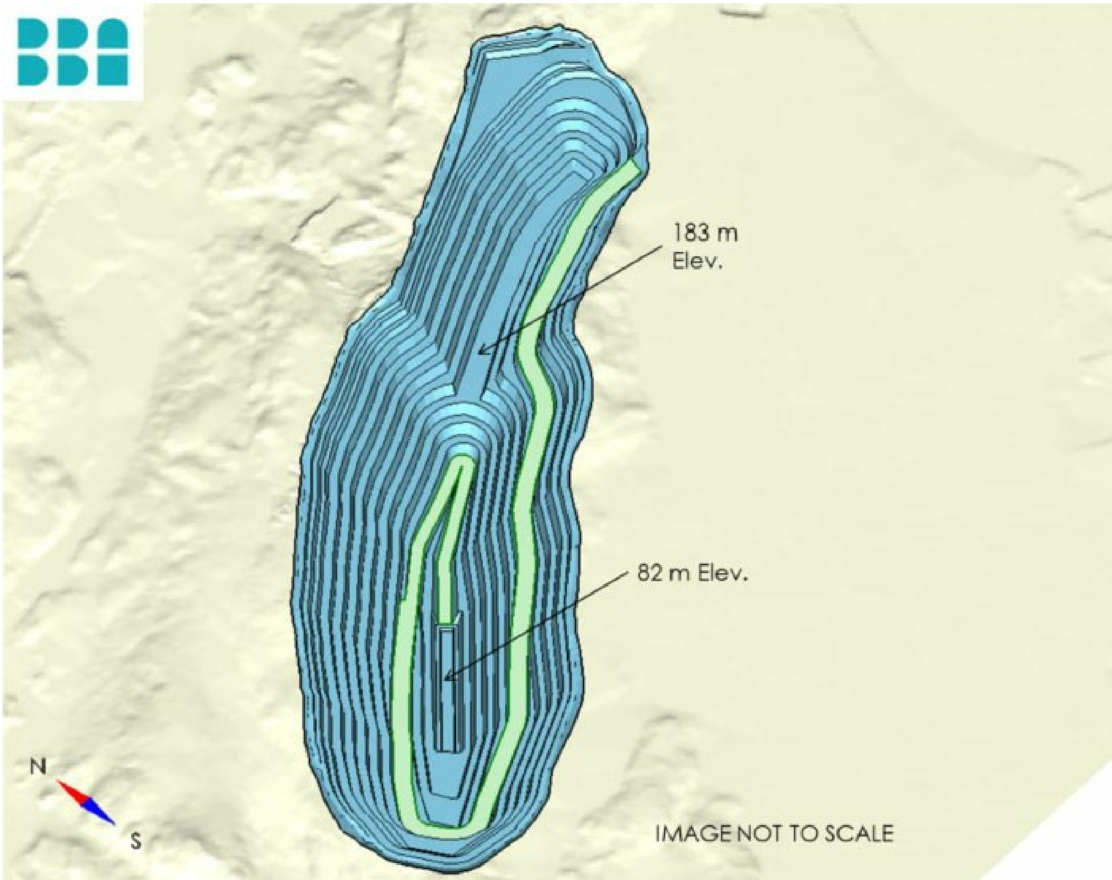
The open pit design attempts to avoid the wetlands that are located on the south side of the pit as much as possible while not losing a significant amount of resources that fall within the optimized pit shell.

The overburden thickness within the open pit ranges from 0 m to a maximum of 10 m and averages 2.7 m thick. The overburden is mostly present on the south side of the pit.

Accounting for mining dilution and loss, the open pit design for the Whabouchi Project includes 10.5 Mt of Proven Mineral Reserves at an average grade of 1.40% Li_2O and 16.0 Mt of Probable Mineral Reserves at an average grade of 1.27% Li_2O for a total Proven and Probable Mineral Reserves of 26.5 Mt at an average grade of 1.32% Li_2O. To access these Mineral Reserves, 1.6 Mt of overburden and 70.2 Mt of waste rock must be mined, resulting in a stripping ratio of 2.8:1.

There are 575,000 t of Inferred Mineral Resources in the open pit which are included in the waste rock quantities. Figure 12-11 presents the open pit design.

Figure 12-11 Open Pit Design



12.5 Underground Mineral Reserves

12.5.1 Underground Parameters

DRA designed an underground mine to produce an average of 3,361 t/d of lithium-bearing ore from the Whabouchi underground mine deposit. The underground mine will be accessed through a portal from where the main decline will be driven to the underground production levels. The deposit will be mainly mined using conventional mechanized transverse long-hole mining method. Approximately 15% of the deposit will be mined using longitudinal long-hole and AVOCA methods. Table 12-13 presents the parameters used for the estimation of the underground Mineral Reserves.

Table 12-13 Underground Mineral Reserves Parameters

Description	Unit	Value
U/G Mining Cost	CAD/t (mined)	46.02
Mill operation cost	CAD/t (milled)	37.00
Milled Ore Transportation Cost	CAD/t (milled)	32.00
Administration & Infrastructure	CAD/t (milled)	11.00
Sales Price	CAD/t (conc.)	1,264
Royalties	CAD/t (conc.)	1.665
OP Mining Dilution	%	5.0
UG Mining Dilution	%	10.0
Mining Recovery - UG	%	90.0
Mill Recovery	%	85.0
Concentrate Grade	%	5.50

12.5.2 Cut-Off Grade

From the underground Mineral Reserves parameters, the resulting cut-off grade (COG) is 0.72 % of lithium oxide (Li_2O). In order to increase the average feed grade to the process plant, the final COG used for the stope generation process was established at 0.87 % Li_2O.

A marginal COG was calculated for the exercise at 0.5 % Li_2O. The marginal material comes from development tunnels. The marginal COG does not take into consideration the U/G mining cost since the material will already be excavated and transported to the surface.

The cut-off grade was calculated according to the following equation:

$$Li^2O\ COG\ [\%]\ = \frac{\{UG\ Mining\ Cost\ [CAD/t] + Mill\ Operation\ Cost[CAD/t] + Transportion\ Cost[CAD/t\ milled] + G\&A[CAD/t\ milled]\ \} \times \{1 + UG\ Mining\ Dilution\ [\%]\}}{\{Sales\ price[CAD/t\ conc.] - Royalties\ [CAD/t\ conc.]\ \} \times Mill\ Recovery\ [\%]}$$

12.5.3 Stope Optimizer Parameters

The stopes were designed using Deswik's Stope Optimizer (Deswik.SO) software. The stope sizes and the mining methods has been mainly determined by the geometry of the deposit. Detailed stope design parameters can be consulted in Section 13.2.1.

12.5.4 Mine Dilution

Dilution is the material (ore, waste, or backfill) that breaks off from the host rock walls, backs and end-walls which is inherent to underground mining. The dilution parameters included in the mineral reserve calculation are identified below:

- Stope footwall: 0.5 m;
- Stope hanging wall: 0.5 m;
- Stope internal dilution from sidewall (backfilled stopes): 0.3 m;
- Lateral / vertical development dilution: 6.5% / 5% of overbreak.

12.5.5 Underground Mineral Reserves

The economic viability of the underground Mineral Reserve has been demonstrated. The underground Mineral Reserves are estimated at 11.7 million tonnes of recoverable and diluted ore with a grade of 1.29 % Li_2O. The mineral underground reserves are 100% in probable category. Reserves are inclusive of dilution and ore loss. The Underground Mineral Reserves are presented in **Error! Reference source not found.**.

The Mineral Reserve is estimated with a stope mining recovery of 90%. The Mineral Reserve includes both internal and external dilution. External dilution included a mining dilution of 0.5 m on the hanging and footwall for the long-hole mining method. A minimum true mining width of 4 m was used. For the Underground Reserves Estimate, concentrate selling price $1,264/t CAD with a process recovery of 85%, a process cost of $48/t CAD (including G &A), a milled-ore transportation cost of $32/t CAD and a mining cost of $46/t CAD (including haulage and backfill) were used.

The underground Mineral Reserves are based on the production schedule presented in Section 13.2.11. The underground Mineral Reserves are the combination of stope and drift development tonnage (including material above the cut-off and internal low-grade dilution) to which dilution and mining recovery factors are added. Cutoff grades varied from 0.5% to 0.72% based on mining method.

12.6 **Note on Mineral Reserve Estimates**

It should be understood that the Mineral Reserve presented in this Report are estimates of the size and grade of the deposits based on a number of drillings and samplings and on assumptions and parameters currently available. The level of confidence in the estimates depends upon a number of uncertainties. These uncertainties include, but are not limited to, future changes in product prices and/or production costs, differences in size and grade and recovery rates from those expected, and changes in Project parameters.

13 MINING METHODS

13.1 Open Pit Mining Method

The Whabouchi pit will be mined using conventional open pit mining methods consisting of drilling, blasting, loading, and hauling. Vegetation, topsoil, and overburden will be stripped and stockpiled for future reclamation use. The ore and waste rock will be drilled and blasted with 12 m high benches and loaded into haul trucks using mining backhoes which will mine 6 m high flitches. Overburden will be hauled to an overburden stockpile and waste rock will be hauled to the CSF. Ore will be dumped on the ROM pad in several stockpiles which will be rehandled and trammed to the primary crusher by a front-end wheel loader. The purpose of this rehandling is to provide a properly blended ore feed to the mill.

The mine will operate on two (2) 12-hour shifts, seven (7) days per week, 50 weeks per year.

13.1.1 Geotechnical Pit Slope Parameters

The geotechnical pit slope parameters are presented in Section 12.4.2.

13.1.2 Hydrogeology and Hydrology Parameters

The hydrogeology and hydrology parameters for the Whabouchi Project are described in Section 17 of the Report.

13.1.3 Phase Designs

Phases, also referred to as pushbacks, have been designed to access ore quicker and to defer waste stripping. The phase design process was guided by the smaller revenue factor pit shells from the open pit optimization analysis. A minimum working width of 40 m between phases was considered acceptable based on the size of the mining equipment and the proposed scale of mining operations. The phase designs use the same pit wall configurations that were presented in Section 12.4.2. The phase designs also attempt to avoid mining the petalite zones in the initial years of the mining operation. A total of four (4) phases were designed for the LOM.

Phase 1 runs the length of the open pit and has a width of about 125 m at surface. The lowest elevation of Phase 1 is 255 m where it reaches a depth of 60 m from surface. Phase 1 includes 2.2 Mt of ore at a stripping ratio of 1.6 to 1. The only final pit wall developed during Phase 1 is at the far west end of the pit.

Phase 2 establishes the north wall of the final pit limits and reaches an elevation of 183 m. The phase is accessed via a temporary ramp that follows the south wall. Phase 2 includes 7.7 Mt of ore at a stripping ratio of 2.6 to 1.

Phase 3 continues to establish the north wall of the final pit limits and reaches an elevation of 159 m. The phase is accessed via a new temporary ramp that follows the south wall. Phase 3 includes 5.9 Mt of ore at a stripping ratio of 2.7 to 1.

Phase 4 establishes the final pit ramp and walls and includes 10.7 Mt of ore at a stripping ratio of 3.1 to 1.

Table 13-1 presents the Mineral Reserves for each phase and Figure 13-1 presents a typical section showing the phases. The figure includes the Measured and Indicated Resources above the cut-off grade of 0.4% Li_2O. Figure 13-2, Figure 13-3, Figure 13-4, and Figure 13-5 show the designs for Phase 1, Phase 2, Phase 3, and Phase 4, respectively.

Table 13-1 Mineral Reserves by Phase

Description	Ore Tonnes (Mt)	Li_2O Grade (%)	Waste Rock & Overburden (Mt)	Stripping Ratio
Phase 1	2.2	1.41	3.6	1.6
Phase 2	7.7	1.32	20.3	2.6
Phase 3	5.9	1.34	15.8	2.7
Phase 4	10.7	1.29	33.4	3.1

Figure 13-1 Phase Design Typical Section

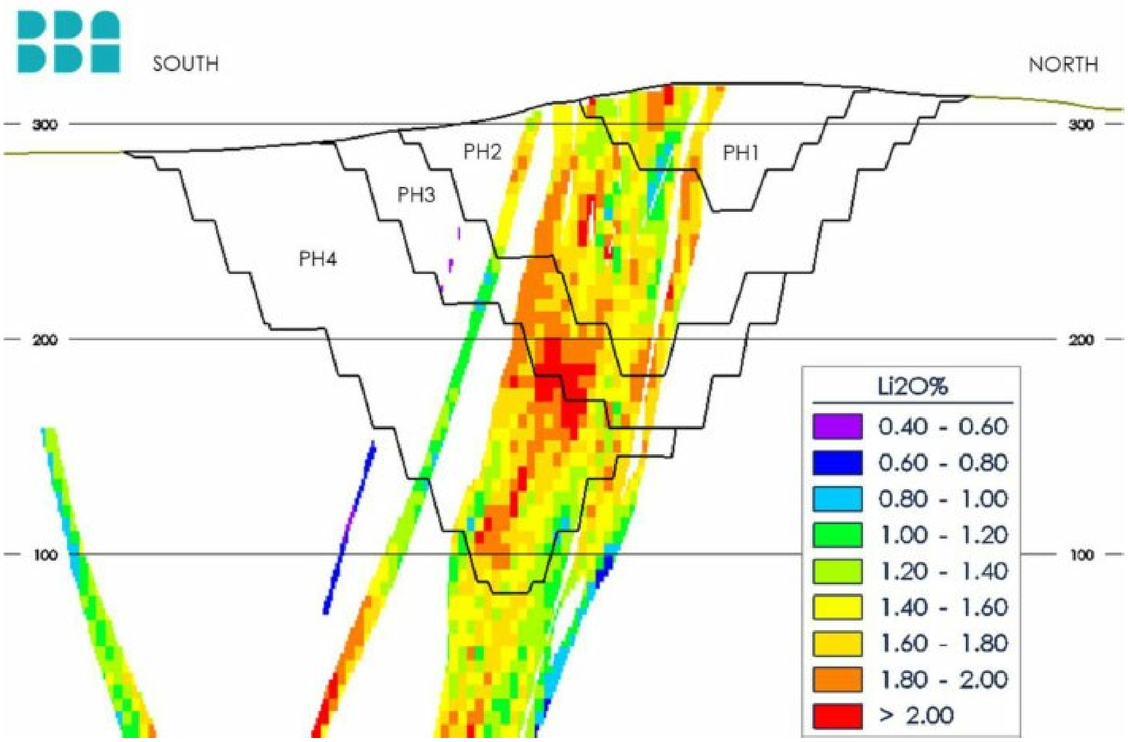


Figure 13-2 Phase 1

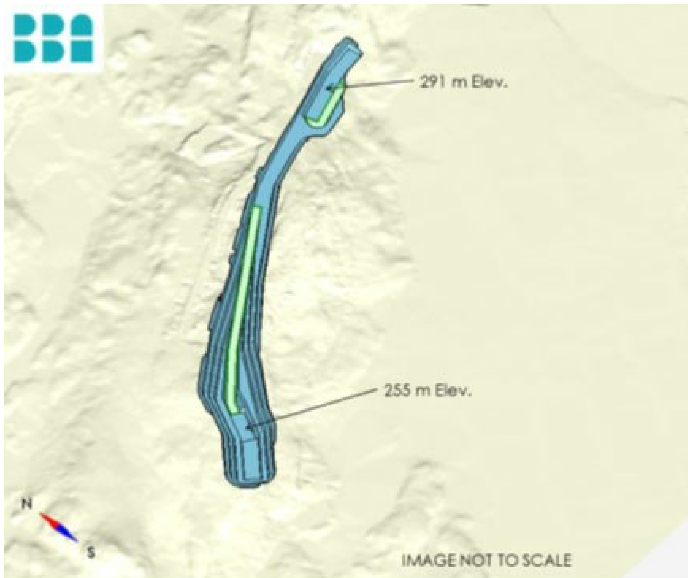


Figure 13-3 Phase 2

Figure 13-4 Phase 3

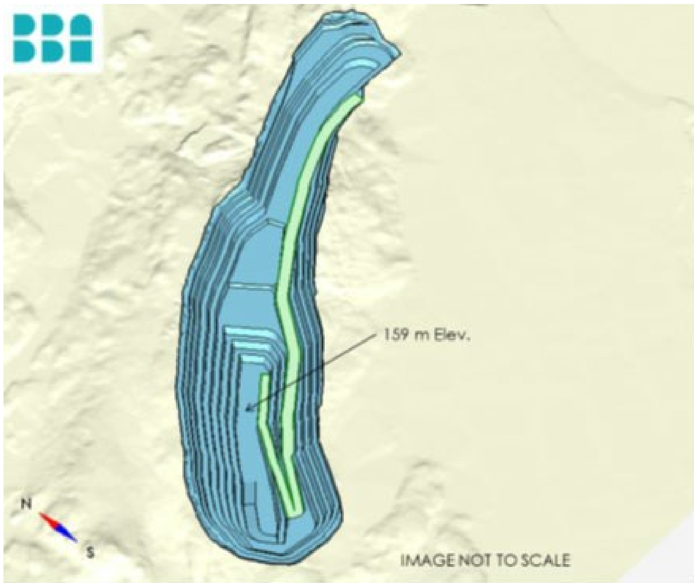
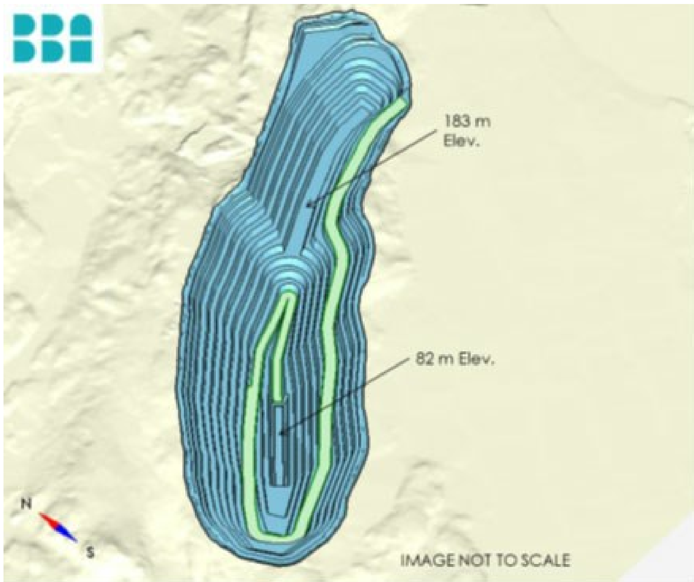


Figure 13-5 Phase 4



13.1.4 Waste Rock Storage Facility and Stockpiles

Material mined from the open pit that is not directly hauled to Run of Mine (ROM) Pad will be placed in several storage facilities across the site. These facilities, discussed in further detail below, include topsoil stockpiles, an overburden stockpile, a petalite ore stockpile, and the CSF. Note that trees will be cleared prior to placing material in these piles.

13.1.4.1 Topsoil Stockpiles

An average topsoil thickness in the open pit of 30 cm was considered for the DFS. The topsoil (organic material) will be stripped and placed separately in stockpiles and will be used for closure and reclamation activities. The topsoil stockpiles will be strategically located around the site to minimize haul distances. Topsoil will also be hauled directly to certain areas if they are available for reclamation, thus reducing costs by limiting re-handling activities.

13.1.4.2 Overburden Stockpile

The overburden stripped from the open pit will be placed in the overburden stockpile and used for future closure and reclamation activities. The overburden stockpile is located to the east of the open pit and south of the concentrator facilities.

13.1.4.3 Petalite Stockpile

The petalite zones mined during the first 5 years of the operation will be stockpiled to defer the loss of lithium units associated to the petalite and maximize the Project profitability in the early years of the operation. This will also allow time for process development and economical evaluation of concentrator modifications required to increase petalite recovery.

13.1.4.4 Co-Disposal Storage Facility

The waste rock excavated from the open pit will be hauled to the CSF. Information about the CSF is provided in Section 15.1.9 of the Report.

13.1.5 Open Pit Mine Production Schedule

13.1.5.1 Mine Planning Parameters

The mine production plan has been prepared using the MinePlan Schedule Optimizer (MPSO) tool in the Hexagon MinePlan 3D software. Provided with economic input parameters and operational constraints such as phase sequencing, maximum bench sink rates, and mining and milling capacities, the software determines the optimal mining sequence which maximizes the NPV of the mine production plan.

The mine plan has been prepared quarterly for the first 3 years of production, annually for the following seven (7) years, and in three (3) year increments thereafter. The mine plan also includes a three (3) month period of pre-production. The purpose of the pre-production period is for the mine to provide waste rock for construction material and to prepare the pit for mining operations.

The mine plan has been prepared using cuts that are 50 m x 50 m x 12 m high, for the first two (2) phases, and 100 m x 100 m x 12 m high for the final two (2) phases.

No specific maximum bench sink rate was used a constraining parameter for the mine plan, but upon completion, it was verified that the mining advances in each period were not too aggressive.

The following assumptions were used to determine the ore targets for each period of the mine plan as well as the concentrate tonnages that are expected to be produced.

- Ore is fed to the primary crusher, and once crushed it is conveyed to the ore sorters. The following formula, which is based on the percentage of amphibole dilution, is used to determine the expected ore sorter recovery;

  % Mass Ejection = (%Amphibole X 95.2% + (1-%Amphibole) x (1- White Rock Recovery)) x 83.35%

  If dilution is < 10% than the White Rock Recovery percent;

  % White Rock Recovery = 99%

  If dilution is > 10% than the White Rock Recovery percent:

  % White Rock Recovery = -7.2356 x %Amphibole x %Amphibole+(%Amphibole+0.9676)

- The ore sorter recovery is expected to increase as of Year 5 with the addition of a scavenger ore sorter;

  % Mass Ejection = (%Amphibole X 95.2% x 97.4% + (1-%Amphibole) x (1- White Rock Recovery)) x 83.35%

  If dilution is < 10% than the White Rock Recovery percent:

  % White Rock Recovery = 99.96%

  If dilution is > 10% than the White Rock Recovery percent

  % White Rock Recovery = (-7.2356 x %Amphibole x %Amphibole+(%Amphibole+0.9676) x 4.2%) + 95.8%

- The material that is not rejected by the ore sorters is conveyed to the mill which has a nominal capacity of 943,525 t/y. This is planned to increase to 990,700 as of Year 5.

- The following is used to calculate the overall metallurgical recovery (ore sorters and milling);

  Overall Metallurgical Recovery= 10.29 x (%Li_2O feed grade) - 0.36 x (%Amphibole) + 0.76

- As of Year 5 the formula becomes;

  Overall Metallurgical Recovery= 17.22 x (%Li_2O feed grade) + 0.18 x (%Amphibole) + 0.61

- To align with the capacity of a potential future conversion plant, an additional constraint was put on the mine plan such that the concentrate production cannot exceed 250,000 t/y.

The mine plan accounts for the following process plant utilization ramp-up prior to achieving nominal capacity during the 13th month of operations. This ramp-up is based on the Series 1 McNulty curve.

- Q1 – 28%;
- Q2 – 65%;
- Q3 – 90%;
- Q4 – 96%;

13.1.5.2 Open Pit Mine Production Schedule

The open pit has a 24-year mine life plus a three (3) month period of pre-production development referred to as Year 0. During pre-production, a total of 529 kt of material is planned to be mined, including 97 kt of overburden, 300 kt of waste rock, and 130 kt of ore.

During the mining operation, the total material mined from the open pit peaks at 5.4 Mt in Year 5 and averages 4.8 Mt/y from Years 2 to 19. The average Li_2O grade ranges from 1.26% to 1.50% over the life of the open pit mine. The mine plan is expected to produce an overage of 226,400 tonnes of concentrate per year, with a high of 238,500 tonnes in Years 20 to 23.

Table 13-2 presents the mine production schedule for the open pit. The tonnages presented are all on a dry basis and the totals may not add up due to rounding. Figure 13-6 to Figure 13-13 present various charts which display the mine production schedule and Figure 13-14 to Figure 13-18 present the pit advances at the end of Year 01, Year 05, Year 10, Year 13, and Year 19.

Detailed anticipated ore production figures for the production schedule depicted in Table 13-2 and Table 13-12 below, together with the anticipated lithium spodumene (5.5% concentrate) production, are included in Table 19-3 in Section 19.6 (Detailed Economic Analysis) of this technical report summary.

Table 13-2 Mine Production Schedule

Description	Unit	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11-13	Y14-16	Y17-19	Y20-23	Y24	TOTAL
Ore to Crusher	kt	0	749	1,087	1,147	1,116	1,130	1,109	1,123	1,099	1,098	1,117	3,450	3,427	3,443	4,437	966	26,497
Li_2O Grade	%	0.00	1.35	1.37	1.30	1.31	1.26	1.26	1.32	1.33	1.33	1.29	1.28	1.34	1.26	1.39	1.50	1.32
Amphibole Dilution	%	0.0	12.6	13.8	16.6	15.5	17.1	15.0	14.4	14.1	12.8	13.3	15.4	14.3	17.5	11.6	3.6	14.2
Ore Sorter Rejects	kt	0	87	145	204	177	152	130	127	121	110	125	416	384	474	401	27	3,079
Concentrator Feed	kt	0	662	942	943	939	979	979	996	978	988	992	3,034	3,044	2,968	4,036	939	23,418
Concentrate Produced	kt	0	155	227	219	218	218	213	227	222	224	219	671	705	661	954	226	5,358
Overburden	kt	97	161	247	57	134	140	106	71	329	32	1	195	0	0	0	0	1,571
Waste Rock	kt	300	1,476	3,044	2,513	3,748	4,047	3,443	3,685	2,572	2,870	3,194	11,355	11,573	12,012	5,488	109	71,427
Total Material Mined	**kt**	**529**	**2,355**	**4,389**	**3,791**	**5,075**	**5,396**	**4,647**	**4,879**	**3,995**	**3,994**	**4,312**	**14,926**	**14,888**	**15,321**	**9,925**	**1,075**	**99,496**
Stripping Ratio		3.0	2.3	3.0	2.1	3.3	3.5	3.2	3.3	2.7	2.7	2.9	3.4	3.5	3.6	1.2	0.1	2.8

Figure 13-6 Mine Production Schedule

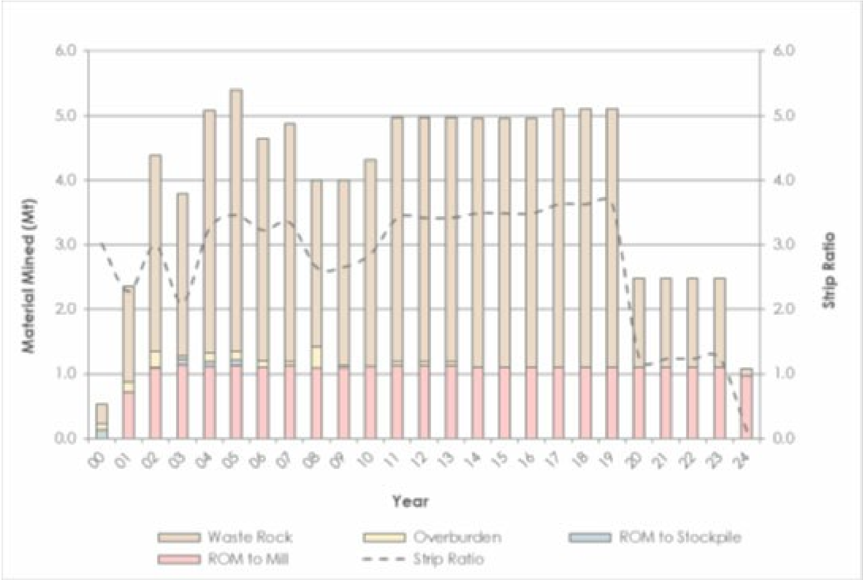


Figure 13-7 Crusher Feed

Figure 13-8 Concentrator Feed

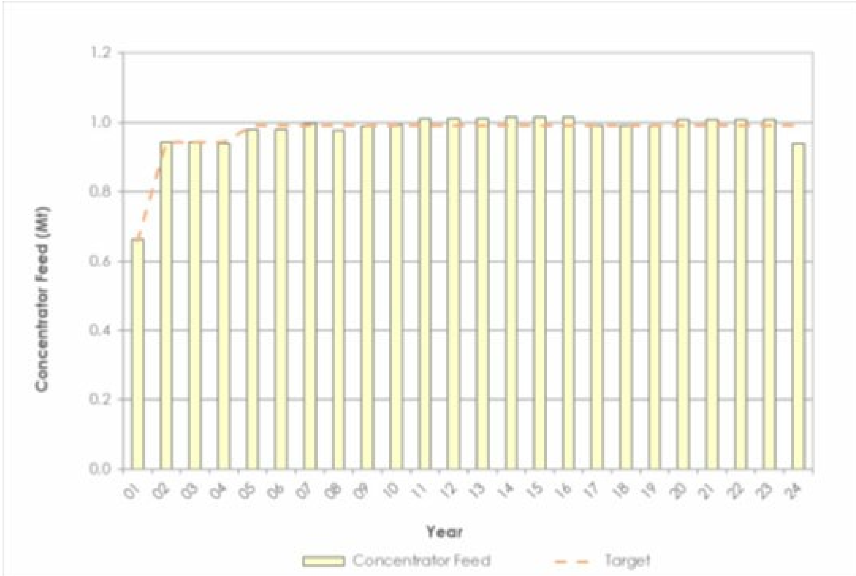


Figure 13-9 Concentrate Produced

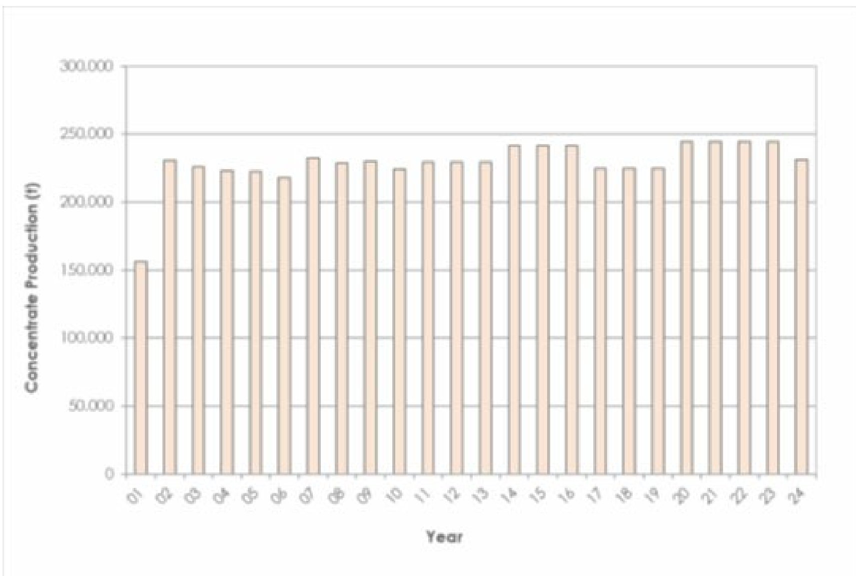


Figure 13-10 Material Mined by Phase

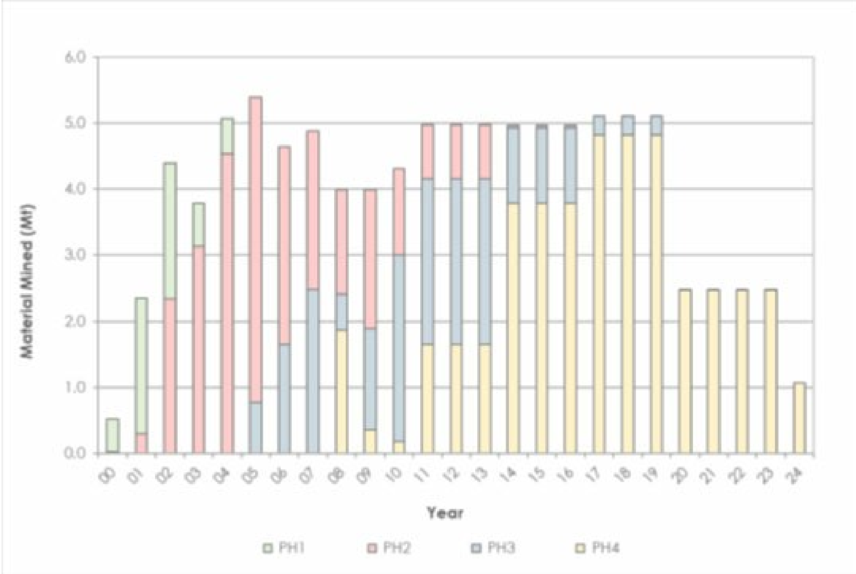


Figure 13-11 Material Mined by Reserve Category

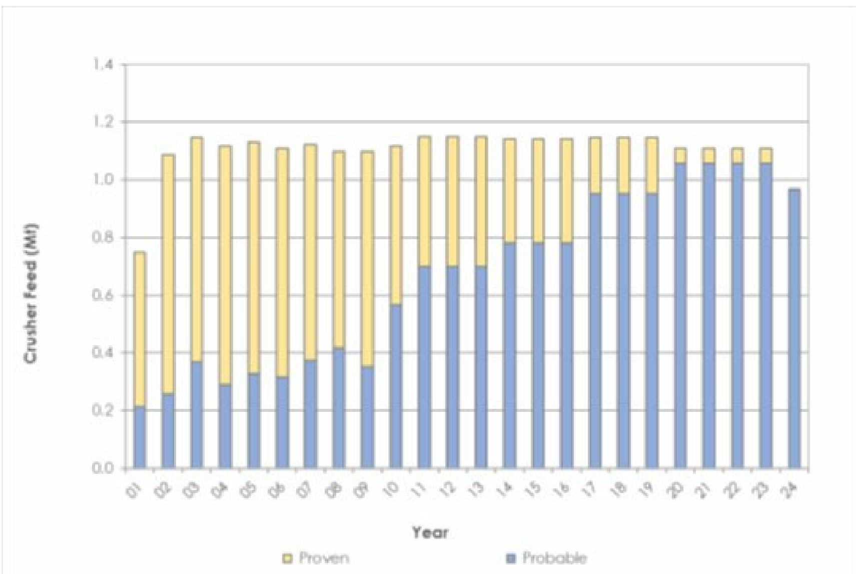


Figure 13-12 Aphibole Dilution

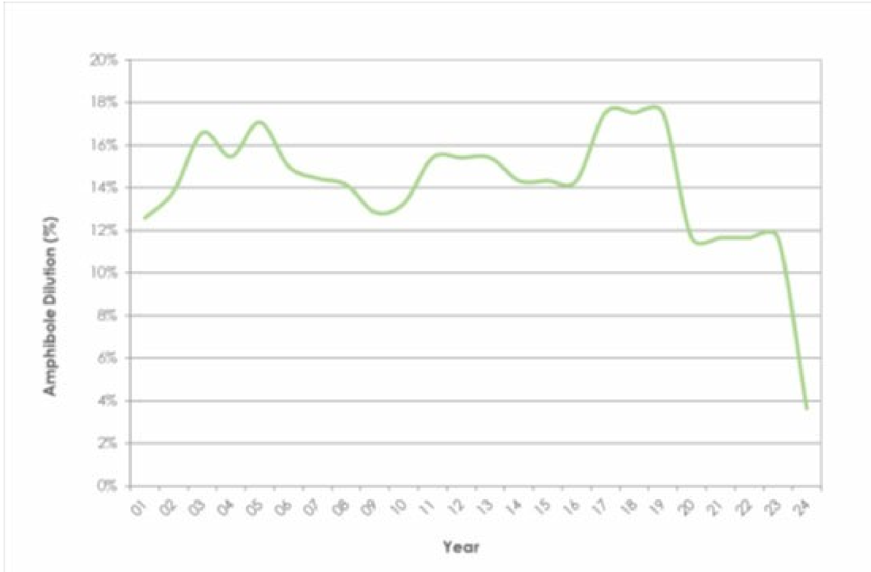


Figure 13-13 Petalite Tonnages to Crusher

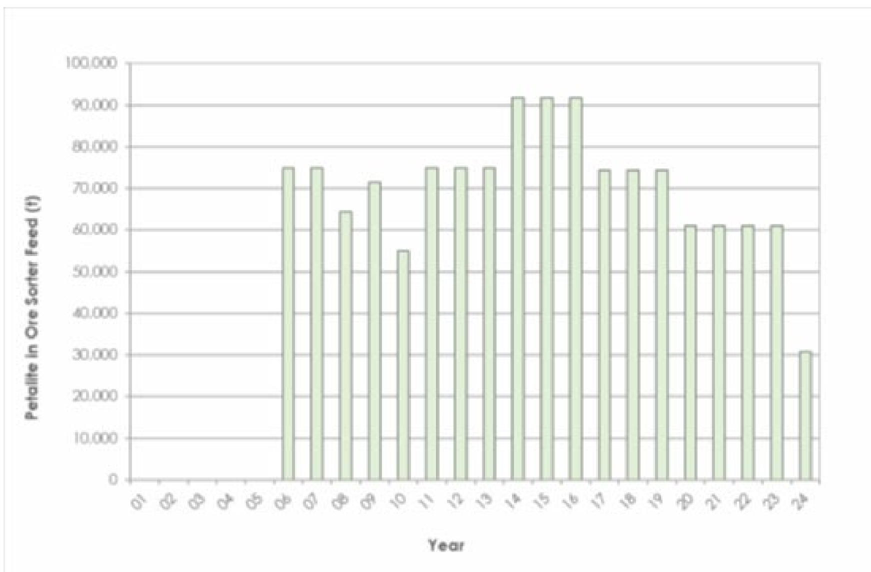


Figure 13-14 End of Year 01

Figure 13-15 End of Year 05

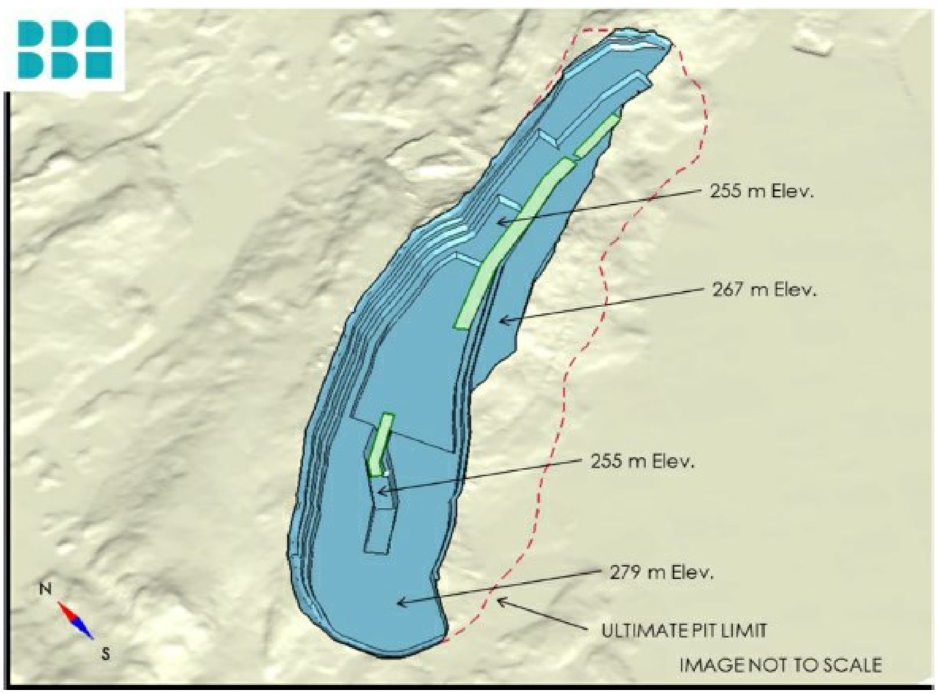


Figure 13-16 End of Year 10

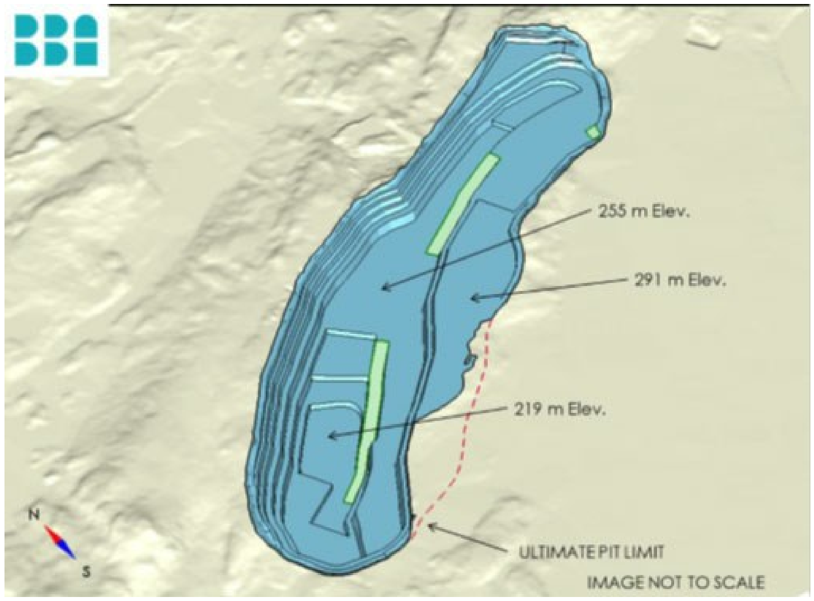


Figure 13-17 End of Year 13

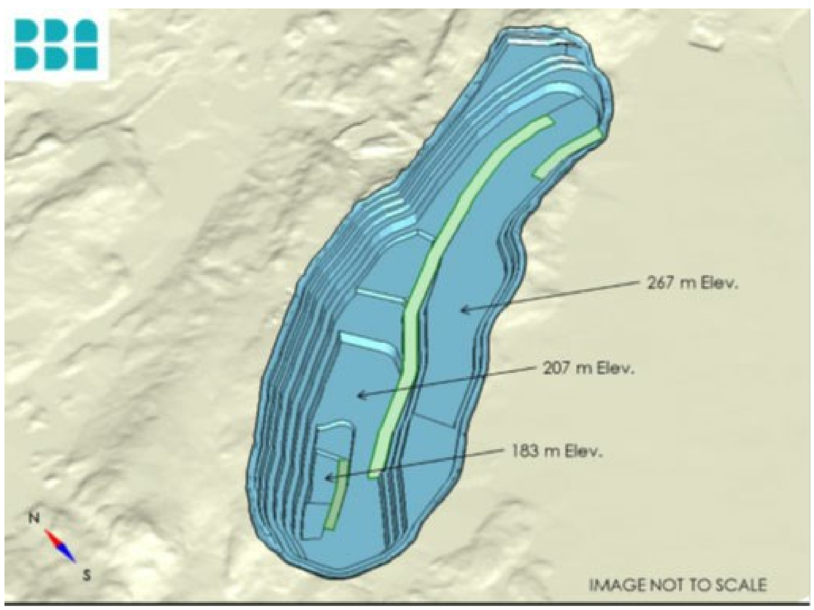
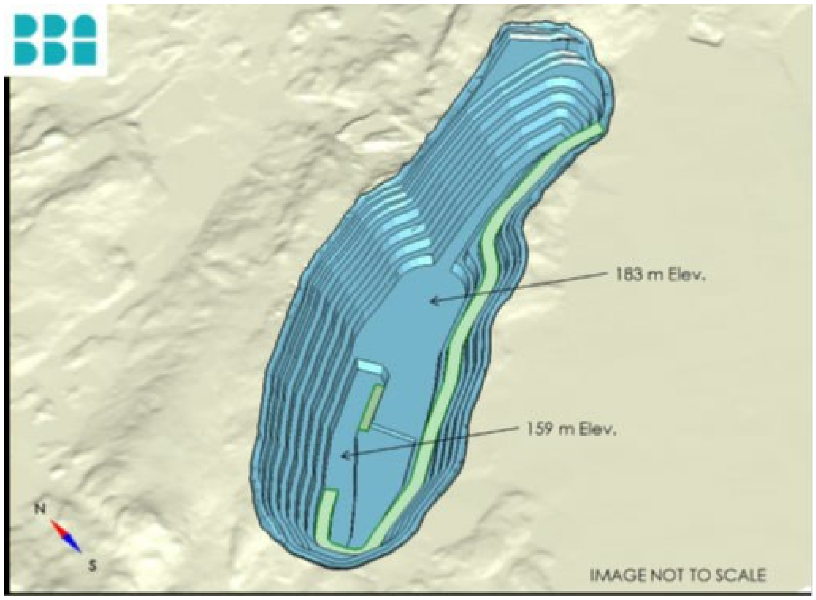


Figure 13-18 End of Year 19



13.1.6 Open Pit Mine Equipment

The following section discusses equipment selection and fleet requirements to carry out the mine plan for the open pit. The mine will be operated by an owner's fleet, except for the production drilling which will be carried out on contract. Table 13-3 presents the list of major and support equipment required during peak production. The table identifies the Komatsu or Caterpillar equivalent to give the reader an appreciation for the size of each machine although the specific equipment selection will be done during the procurement phase of the Project. Equipment required for the loading, transporting, and placement of the tailings and ore sorter rejects at the CSF is not included in this table and is discussed in Section 15.1.9.

To reduce the carbon footprint, a diesel fleet management system will be used, as well as a practice to stop all engines when not in use. In addition, trolley assist can be implemented when mining Phases 3 and 4 since the pit will be accessed via a long ramp segment.

Table 13-3 Mine Equipment Fleet

Equipment	Model	Description	Units
Haul Truck	Komatsu HD605	Payload – 64 tonnes	6
Hydraulic Excavator	Komatsu PC1250	Operating Weight – 120,000 kg	2
Wheel Loader	Komatsu WA600	Net Power – 395 kW	2
Track Dozer	Komatsu D155AX	Net Power – 264 kW	3
Road Grader	CAT 16M	Operating Weight – 232,000 kg	2
Water / Sand Truck	n/a	Capacity – 40,000 litres	1
Utility Excavator	Komatsu PC650	Net Power – 325 kW	1
Stemming Loader	n/a		1
Transport Bus	GMC	20 passengers	1
Powder Truck	n/a	n/a	1
Fuel Truck	n/a	n/a	1
Mechanic Service Truck	n/a	n/a	1
Boom Truck	n/a	n/a	1
Lowboy	n/a	n/a	1
Pickup Truck	Ford F250	Crew Cab	20
Light Plant	n/a	6 kW	8
Dewatering Pump	n/a	n/a	5

13.1.6.1 Operating Schedule

The schedule for the open-pit operations is based on two (2) 12-hour shifts per day, seven (7) days per week, for 50 weeks per year. The fleet calculations consider seven (7) days of lost mine production due to inclement weather. Figure 13-19 presents the equipment utilization model that is used to define the key performance indictors (KPI) that govern the fleet requirements. The definitions for each time component are presented below using haul trucks as an example.

Figure 13-19 Equipment Utilization Model

Scheduled Time (S)				
Available Time (AT)			Down Time (DT)	
Operation Time (GOH)		Standby Time (ST)	Planned Loss	Breakdown Loss
Utilized Time (NOH)	Delay Time (DL)			
Travelling empty	Waiting for Shovel	Shift Change	Scheduled Maintenance	Breakdown
Spotting at Shovel	Shovel Repositioning	Lunch & Coffee Breaks	Preventative Maintenance	Waiting for Parts
Loading	Crusher Down	Refueling	Inspections	Repair Time
Travelling Full	Queuing at Shovel	Pre-start Checks	Overhauls	
Spotting at Dump	Weather	No Operator Available		
Dumping				

- Scheduled Time – full calendar year less unplanned shutdowns;
- Down Time –unit is inoperable due to scheduled maintenance or unplanned breakdown;
- Available Time – scheduled time less down time;
- Standby Time – the unit is available mechanically but not being used (the engine will typically be shut off while the unit is on standby);
- Utilized Time – available time less standby time. This time is also referred to as the Gross Operating Hours (GOH);
- Operating Delays – the unit is available and not on standby but not effectively producing (the engine will be running during the operating delays);
- Operating Time – utilized time minus operating delays. This time is also referred to as the Net Operating Hours (NOH).
- The following KPI's can be calculated from using the formulas below;
- Availability – (NOH + Op. Delays + Standby) / (NOH + Op. Delays + Standby + Down);
- Use of Availability – (NOH + Op. Delays) / (NOH + Op. Delays + Standby);
- Machine Utilization – (NOH + Op. Delays) / (Scheduled Time);
- Operating Efficiency – (NOH) / (NOH + Op. Delays);
- Effective Utilization – (NOH) / (Scheduled Time).

Table 13-4 presents the KPI's and time assumptions used for the fleet of trucks and excavators.

Table 13-4 Mine Equipment KPI's

Description	Units	Trucks	Excavators
Availability	%	85.0	85.0
Use of Availability	%	80.9	82.2
Machine Utilization	%	87.9	75.1
Operating Efficiency	%	68.8	69.9
Effective Utilization	%	60.4	52.5
Scheduled Time	h/y	8,424	8,424
Down Time	h/y	1,314	1,314
Standby Time	h/y	1,420	1,324
Operating Delays	h/y	731	1,525
Utilized Time (GOH)	h/y	6,026	6,122
Operating Time (NOH)	h/y	5,295	4,597

13.1.6.2 Drilling and Blasting

Production drilling will be done by a contractor with diesel-powered down-the-hole (DTH) drills that will drill 4.5-inch (114 mm) diameter holes in ore, and 5.5-inch (140 mm) diameter holes in waste rock, on 12 m high benches. Drilling productivities have been calculated using an instantaneous drill penetration rate of 35 m/h and the fixed time drilling components presented in Table 13-5.

Table 13-5 Fixed Drilling Time per Hole

Description	Unit	Value
Steel retract	min	0.50
Jack up	min	0.30
Tramming	min	2.50
Jack down	min	0.50
Collar time	min	3.00
Bit change	min	0.30
Total	min	7.10

The drill productivities have been applied to the number of holes drilled per year to determine the annual hours of drilling and number of units required. In addition to the number of holes, which is based on the blast pattern presented in Table 13-6, an additional 2% has been considered for holes that will require re-drilling.

Table 13-6 Drilling and Blasting Parameters

Parameter	Unit	Ore	Waste
Bench Height	m	12	12
Blast hole Diameter	mm	114	140
Burden	m	2.80	3.40
Spacing	m	3.30	4.10
Sub drilling	m	1.0	1.0
Stemming	m	3.0	3.0
Powder Factor	kg/t	0.38	0.36

Blasting will be carried out using emulsion with an explosive density of 1.15 g/cm³. Blasting will be done using electric detonation and drill holes will be double primed (two detonators and two boosters per hole). Pre-split drilling and blasting will be done on the final pit walls.

A total of two (2) production drills are required.

Explosives products and accessories will be delivered to site by an explosive's supplier. The emulsion will be delivered to site in 20,000 kg tankers, and two (2) tankers will always be left on-site as emulsion storage. A total of two (2) explosives magazines will be required as well to store the explosive accessories. The storage tanks and magazines will be located to the West of the open pit and will respect the minimum distance requirements specified by Natural Resources Canada Explosives Regulatory Division. A mobile manufacturing unit (MMU), operated and maintained by the explosive's supplier, with a 12,500 kg capacity will deliver the emulsion from the transfer site to the blast patterns.

The Nemaska explosives team will consist of two (2) blasters and two (2) blaster helpers, working on a 2-crew system, day shift only.

There will be roughly one (1) blast per week which will generate between approximately 100,000 tonnes of rock. The quantity of explosives averages approximately 1,500,000 kg per year.

13.1.6.3 Loading

Loading will be done on 6 m benches using diesel-powered hydraulic backhoe excavators equipped with 6.7 m3 buckets. Productivities have been calculated considering bucket swing times of 40 seconds and an 85% fill factor.

During peak production, the fleet will include two (2) excavators.

The mine production fleet includes two (2) front-end wheel loaders equipped with 6.4 m3 buckets. One wheel loader will be dedicated to rehandling the ROM pad ore stockpile into the primary crusher and the second wheel loader will be used to rehandle ore from the Petalite Stockpile and to assist the excavators in the pit.

13.1.6.4 Hauling

Hauling will be done with 64-tonne rigid frame haul trucks. Haul productivities have been calculated considering effective payloads of 62.7 tonnes, which have been reduced from the nominal payloads to account for a carryback of 2%.

A haulage network was established in MPSO that considers the hauls for each mining cut to each potential dumping destination. Using rimpull curves provided by the truck manufacturers, MPSO calculated the travel times for each haul. The travel times were then added to the fixed haulage cycle times to arrive at the total cycle times. The fixed cycle times consider 42 seconds for truck spotting, 40 seconds for each bucket, and 72 seconds for spotting and dumping at the destination. It is assumed that the excavator will be waiting for a truck with a loaded bucket 50% of the time, resulting in a 5-second first bucket pass in those instances. A total of six (6) buckets is required to load each truck, resulting in an average total fixed cycle time of 320 seconds. In addition to these haulage parameters, the truck productivity calculations consider a 3% rolling resistance for in-pit and on the stockpile hauling, a 2% rolling resistance for surface haul roads, a maximum speed of 40 km/h and a downhill maximum speed of 25 km/h.

A total of three (3) trucks are required in preproduction, ramping up to six (6) by Year 14.

The average one-way haul distances for the open pit over the LOM are 1.7 km for ore to the crusher and 2.7 km for waste rock to the CSF.

13.1.6.5 Auxiliary Equipment

A fleet of support equipment has been included for haul road maintenance, drill pad preparation, and cleaning around the loading face. The fleet of support equipment includes dozers, graders, a water/sand truck, and a utility excavator.

A fleet of service equipment such as fuel and lube trucks, maintenance vehicles, and pick-up trucks is also included.

13.1.7 Co-Disposal Storage Facility Construction and Operation

The following section provides a brief description of the construction of the CSF with further details provided in Section 15.1.9. The CSF will be built with waste rock from the mine, ore sorter rejects, and tailings. In a full production year where the mill will process 949 kt of ore at an average grade of 1.32% Li_2O and 14% amphibole dilution, a total of 119 kt of ore sorter rejects, and 702 kt of tailings will be generated.

Table 13-7 presents the densities that are used to calculate the volumes that will be placed in the CSF. These densities consider the swell factor, compaction factor, and the moisture content.

Table 13-7 Densities

Material	Unit	Value
Tailings	t/m3	1.50
Ore Sorter Rejects	t/m3	1.80
Waste Rock	t/m3	2.33

The tailings and ore sorter rejects generated at the mill will be loaded by a Komatsu WA600 front end wheel loader into a fleet of Komatsu HD605 haul trucks. A total of two (2) trucks are required to haul the tailings and ore sorter rejects.

Material placed on the CSF will be spread by a fleet of Komatsu D65X dozers and compacted by a soil compactor. The lift thickness and number of dozer and soil compactor passes required will be determined by test-pad scale-tests to provide sufficient compaction. A quality assurance program including how the thickness of the layers and the compaction required would be achieved must be included into the operation manual and the deposition plan of the CSF.

Areas of the CSF that have reached the final construction design will be capped with a layer of overburden as part of the closure plan. The overburden will either come directly from the open pit or will be rehandled from the overburden stockpile. Placement and spreading of the overburden will be done by backhoe excavators.

The mine road graders and water/sand trucks will be used to maintain the roads related to the CSF.

The CSF construction activities will follow the same seven (7) day per week schedule as the mine operations.

Table 13-8 presents the equipment fleet for the CSF.

Table 13-8 Mine Equipment Fleet (CSF)

Equipment	Model	Description	Units
Haul Truck	Komatsu HD605	Payload – 64 tonnes	2
Wheel Loader	Komatsu WA600	Net Power – 395 kW	1
Track Dozer	Komatsu D65AX	Net Power – 162 kW	1
Hydraulic Excavator	Komatsu PC490	Net Power – 268 kW	1
Soil Compactor	BOMAG MW219	n/a	1

13.1.8 Open Pit Mine Workforce

The mine workforce for the open pit will total 84 employees at the start of pre-production and will reach a peak of 148 employees between Years 17 and 19 (Table 13-9). The workforce for the mine has been categorized into Mine Operations, Mine Maintenance, CSF Operations, and Mine Technical Services. The mine operations will be composed of four (4) crews to provide a 24 h/d continuous operation.

Table 13-9 Mine Workforce Requirements

Description	Number
Mine Operations	
Mine Manager	1
Pit Foreman	4
Shovel Operator	8
Truck Operator	21
Loader Operator	8
Grader / Dozer Operator	20
Water Truck / Excavator Operator	4
Labourer	8
Blaster	2
Blaster Helper	2
Mine Maintenance	
Maintenance Superintendent	1
Maintenance Foreman	4
Maintenance Planner	2
Mechanic	25
Mine Technical	
Mining Engineer	2
Geologist	2
Sampler	4
Surveyor	2
Tailings Operations	
Tailings Foremen	2
Tailings Planner	2
Truck Operator	8
Tailings Loader Operator	4
Tailings Dozer Operator	4
Tailings Excavator Operator	4
Tailings Mechanic	4
Total Mine Workforce	148

13.2 Underground Mining Methods

The geometry of the deposit gives the opportunity to select an underground productive mining method, namely the transverse long-hole mining method. Longitudinal long-hole and AVOCA mining methods were also selected for some remote areas where the lodes are thinner and smaller. The underground ore production rate will reach 3,361 t/d at the concentrator feed.

The underground mine will start production when the open pit mining is completed. A large amount of waste rock can be re-handled and used for stope backfilling purposes. The backfill method selected is cemented rock fill (CRF) for transverse stopes and rockfill (RF) for longitudinal and AVOCA stopes.

The underground operations will generate 11.7 Mt of ore at an average diluted grade of 1.29 % Li_2O. The current underground Mineral Reserves extend to 400 m below surface. The main underground mine plan consists of 14 levels set at 30 m intervals. The pit bottom will also be used to recover the crown pillar from the underground. The crown pillar will be mined at the end of the underground LOM. Access to the pit bottom should be maintained for drilling and backfill activities during the crown pillar excavation.

To access the deposit, a single decline will be driven from a single portal, located at surface near the exit of the pit at elevation 275 m. From the surface, the ramp will be driven toward the west of the deposit to reach the center of the deposit at elevation 185 m (090L). The same decline will then remain centralized and be driven to the lowest point of the mine, at elevation -115 m (390L) connecting all levels of the underground mine. From 090L, another ramp will be driven up towards the West where a second egress raise will then be excavated at the highest point of this ramp to reach surface.

For the transverse mining method, the main level and infrastructure will be excavated south of the deposit. The commercial production of the deposit will begin 3 years and 2 months after the start of the ramp.

No work has been performed for the geotechnical and hydrogeology aspects for the underground mine.

13.2.1 Underground Geotechnical Parameters

No geotechnical work has been performed to date for the underground mine. DRA used typical and industry-standard geotechnical parameters. These are listed below:

- Minimum wall angles for hanging wall and footwall at 70;
- Hanging wall and footwall dilution at 0.5 m each.
- Sub-level spacing at 30 m.
- Stope width at 20 m.
- Maximum stope length at 20 m.
- Minimum inter-lode pillar at 5 m.

Typical stope shapes have the following dimensions:

- Height: 30 m;
- Width: 20 m;
- Length: 5 to 20 m (avg. 12 m);
- Dip: 76° to 90° (avg. 84°).

Long support (i.e., cables) were considered for this exercise. The following assumptions were used:

- Cables in the back of the overcuts;
- Cables in the brow of the undercuts for the bottom of the pyramids;
- Cables in the face of the overcuts (transverse only); and
- Cables in two intersections per level.
- No empirical analyses (e.g., stability graph method), limit equilibrium analyses and 2D-3D numerical modelling were performed for the underground mine. For future exercises, a geotechnical study should be performed to evaluate and cover the following aspects:
- Rock mass characterization with a specific attention to the deeper portion of the deposit (i.e., for the underground mine areas);
- Stope dimensions and dilution;
- Stope inter-lode pillar dimensions
- Ground support design (development and stopes) and recommendations
- Crown pillar dimension and interaction with the open pit;
- Geotechnical guidelines for the mining sequence and development with a specific attention to the interaction with the open pit.

13.2.2 Hydrogeology Parameters

No work has been performed on the hydrogeology aspects for the underground mine.

13.2.3 Stope Design

Stopes were designed using the Mineable Stope Optimizer (MSO) module in Deswik (Deswik.SO) based on defined parameters (minimum and maximum stope dimensions, dilution, cut-off grade, etc.). The MSO parameters used are presented in Table 13-10.

Table 13-10 MSO Parameters

Description	Value
Maximum Length (along the transverses)	20 m
Height	30 m
Stope width	20 m
Minimum Mining Width (True Convention)	4 m
Minimum Pillar Thickness (True Convention)	5 m
Minimum Wall Dip	70°
Hanging Wall Dilution	0.5 m
Foot Wall Dilution	0.5 m
Li_2O Cut-off Grade	0.867 %

A mining recovery of 90% was used on the diluted stopes regardless of the mining method.

Ore and waste drives were designed to account for the selected mining fleet, the geometry of the deposit and the mining method.

13.2.4 Production Drilling and Blasting

All stopes will be drilled using a top hammer drill with a diameter of 114 mm / 4.5 inch. An initial slot of 0.76 m / 30 inch in diameter (i.e., V30 method) will be made in each stope using appropriate boring equipment to account for rock swelling after blasting.

Bulk emulsion product, manufactured on site, will be used for the underground development and production blasting. Explosives will be manufactured at the same explosives plant used for the open pit operation and then transported to the underground mine in quantities required to sustain production and development.

13.2.5 Ore Handling

Ore mucking will be done using 17-tonne Load-Haul-Dump (LHD) machines. LHDs will be equipped with a remote-control system for mucking in the stopes. Broken ore will be brought to a nearby drawpoint / footwall drive intersection, a temporary remuck, or to the level access / footwall drive intersection where loading of the haul trucks will occur. The 50-tonne underground mining trucks will haul the ore material to the surface through the ramp(s).

13.2.6 Backfilling

All underground stopes will be backfilled to maximize recovery of the deposit. Cemented rockfill (CRF) will be the main backfill material used followed by rockfill (RF). For transverse stopes, primary stopes will be backfilled using CRF and secondary stopes using RF. For longitudinal stopes, CRF backfill will be used except for the last stope in the sequence of a level will be backfilled with rockfill (RF). AVOCA stopes will be backfilled using RF only.

Only one (1) sill pillar is included in the design. The CRF backfill recipe in these stopes will need to be at a higher cement content for stability purposes and to develop through it. The cement content in that sill pillar will need to be assessed in detailed in future studies or revisions.

Most of the waste rock used for the backfill will come from open pit mining activities. When available, waste rock will be transported from the waste development rounds directly to the stopes to be filled.

Any waste rock surplus will be hauled to surface until it can be hauled back underground later in the LOM, as active lateral development alone will not provide sufficient waste rock for backfill requirements.

A dry cement truck will transport the cement to the underground cement mixing truck. The cement milk will be produced underground in the cement mixing truck parked in a drawpoint near the backfill activity. The cement milk will be poured directly into the LHD bucket filled with waste rock. The mixing of the waste rock with the cement milk will be done naturally while transporting the CRF to the stope and while unloading the cemented rockfill in the stope.

CRF backfill will cure for 21 and 28 days for longitudinal and transverse stopes respectively following the end of the backfilling. It is expected that the cement content of the CRF be at 4%. The curing of the CRF backfill will allow to mine the adjacent stope on the same level. The cement content and subsequently the backfill strength and strategy will need to be reviewed in future studies / revisions.

The CRF backfill production rate was set at 900 t/d considering backfill operations occur during day- and nightshift. For RF backfill, a rate of 1,000 t/d was used.

There is an opportunity to use paste backfill (PF) for the Whabouchi deposit. This should be assessed in detail in future studies.

13.2.7 Mine Development

A development schedule was prepared to ensure that stopes will be available on time to sustain a nominal ore concentrator feed of 3,361 t/d. Early in the LOM, the development plan consists of a single face decline (i.e., ramp) that will transition to multiple faces once the production level is reached.

The ramp will be driven from the surface to the 090L using a single development crew. After reaching this level, a second mining crew will be added to complete the main level and the infrastructures required to start the production. This includes the excavation of the main ventilation raise air-intake and the second egress through a raise and safescape system. A third crew will be added six (6) months after the start of the ramp to develop the footwall drive and drawpoints for production.

All lateral and ramp development has a single face productivity rate of 120 m/month. For multiple headings, the rate is at 250 m/month. These are typical rates used in similar underground operations.

Table 13-11 details the development schedule for the pre-production and production years. A total of 37.5 km of equivalent lateral development and 1.8 km of vertical development will be required, as sustaining capital expenses, to access and develop the different stopes on time. Completion of the escape way raises and the main ventilation raises are required before starting stope production, as they are required to comply with legal requirements regarding two means of egress and ventilation requirements.

The development and production plan will be used to define the operating and capital requirements. The underground development schedule is presented in Table 13-11.

13.2.8 Portal

The underground mine portal will be located near the surface plant. The portal will be in a boxcut excavation. Ground support specific to this kind of infrastructure will be installed accordingly to support walls of the boxcut and the brow between the tunnel and the boxcut. A short section of corrugated steel structure will be erected at the drift entrance to eliminate personnel's exposure to potential falls of rocks or ice blocks from the vertical rock face.

Table 13-11 Underground Development Meters by Year

Description	Total	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35	Year 36	Year 37
CAPEX Lateral Meters (m)	**16,819.6**	4,163	5,210	4,515	1,267	596	1,026	41								
CAPEX Vertical Meters (m)	**1,783.7**	329	747	619	88											
OPEX Waste Meters (m)	**10,055.7**		630	1,295	1,143	2,490	2,402	1,633	200		115		119	23		7
OPEX Marginal Meters (m)	**1,985.8**			30	870	370	314	327	33		3			39		
OPEX Ore Meters (m)	**8,601.8**				2,576	2,201	1,823	1,748	71	11	65		10	97		
Subtotal OPEX Meters (m)	**20,643.4**		630	1,325	4,589	5,061	4,539	3,708	304	11	182		128	159		7
Total Lateral Development Meters (m)	**37,463.0**	**4,163**	**5,840**	**5,840**	**5,856**	**5,657**	**5,565**	**3,750**	**304**	**11**	**182**	**0**	**128**	**159**	**0**	**7**

13.2.9 Mine Ventilation

The ventilation design follows the underground mine design version NEM-WHA22UGMY-002. The air requirements comply with the Quebec ventilation regulations rate plus a 20% contingency factor (leakage included); based on Canmet's approved Diesel Engine guidelines. In estimating the aggregate rate of fresh airflow for the entire mine, a utilization rate has been applied to account when machines may be mechanically unavailable, or simply not in use. This initiative enables to improve the air quality while maintaining airflow requirements as per regulations. The utilization rates are as follow: 100% for production equipment, 50% for most service equipment, and 25% for machinery that operates primarily with electricity. The total fresh air rate required for the entire mine is calculated at 372 m3/s (790 kCFM).

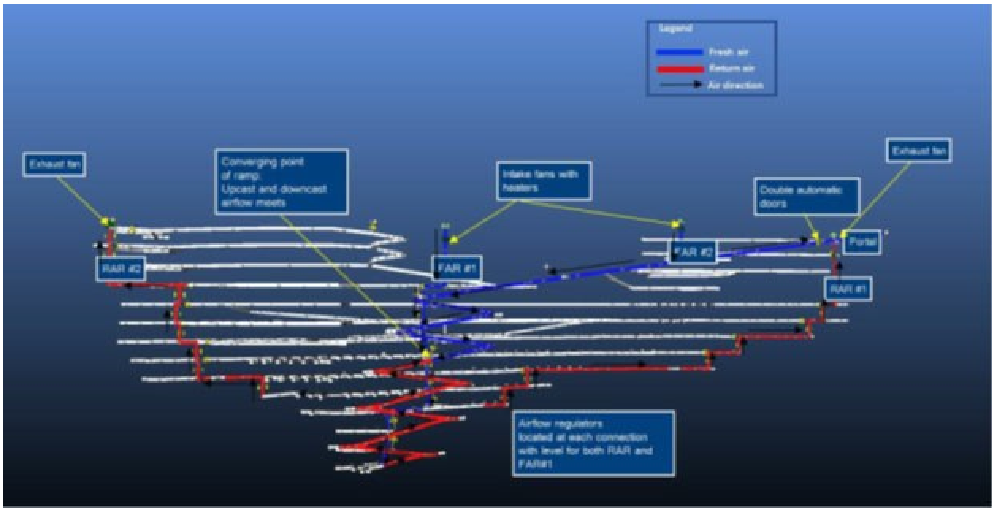
The ventilation system consists of a push-pull system with two systems of air intake and two exhaust systems. One intake raise system pushes air to the bottom of the ramp using a series of Alimak/drop raises while the other one is located close to the portal and uses a single raise and the ramp itself as the intake. For the intake raise connected close to the portal, double automatic doors will be installed between the raise and surface to add resistance so that the air is pushed down the ramp. This system was chosen to divide the fresh air flow in two (2) systems in order to optimize Capex with respect to the ventilation raise and fan sizes. The two (2) intake raises are each equipped with main fans and heaters located on surface. The two (2) exhaust systems are each located at the extremity of the production levels. They are each equipped with main fans located on surface. Figure 13-20 depicts a typical schematic of the ventilation network.

All four (4) fan systems would have a set of parallel fans in horizontal arrangement. For the main intake and two (2) exhausts systems, the selected fans are the same and each would be equipped with a 447 kW (600 HP) motor. The ramp intake system fans would be equipped with two (2) 112 kW (150 HP) motors. The total ventilation installed motor power for main fans is of 2,910 kW (3,900 HP). Each motor will be equipped with a Variable Frequency Drive (VFD) for energy savings. The intake systems will be equipped with heaters of 9,600 kW (32.7 MBtu/h) capacity each for a total of 19,200 kW (65.4 MBtu/h).

The ventilation raises have been sized as follows:

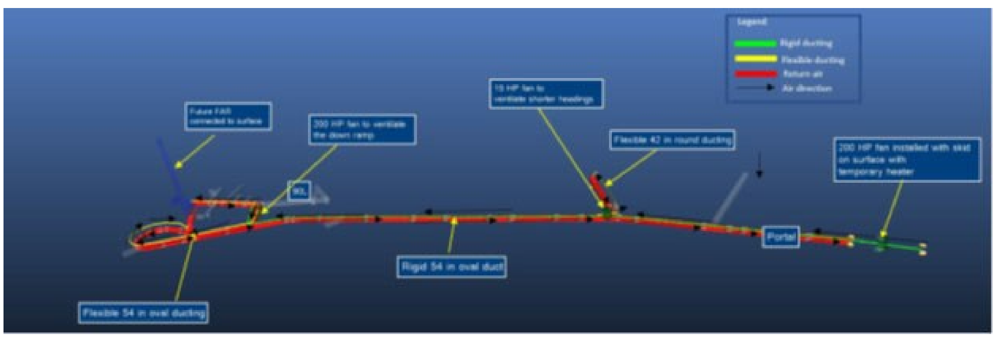
- Two (2) intake air raises connected to surface are Raisebore of 4 m (13.1 ft) diameter;
- Two (2) return air raises connected to surface are Raisebore of 3 m (9.8 ft) diameter;
- All other raises developed from the underground levels are Alimak/drop raise 3 x 3 m (9.8 x 9.8 ft).

Figure 13-20 Ventilation Network Schematic



The ramp development is performed assuming fresh air requirements for one 8 yd LHD, one 30 tons haul truck and a mine pick-up truck for a total of 32 m3/s (67 kCFM) including 5% leakages. Prior to constructing the ventilation raise, a temporary 149 kW (200 HP) fan will be installed on surface with a skid with a temporary heater with a power of 1,900 kW (5.2 MBtu/h). The fan will push the air in a rigid ducting of 1.37 m (54 in) diameter oval. Additional headings will require a 200 HP fan that can be connected to either rigid or flexible ducting of 1.37 m (54 in) diameter oval. Shorter headings that will only have an LHD accessing will be ventilated with flexible round ducting of 1.07 m (42 in) diameter and an 11 kW (15 HP) fan. A schematic of the ramp development to 090L ventilation system prior to the construction of the ventilation raise is shown in Figure 13-21.

Figure 13-21 Ramp Development Ventilation Schematic



Once the new ventilation raise is constructed, the 200 HP fan located on surface will be moved underground as a main ventilation circuit is established.

For the production levels, airflow regulators will be installed in bulkheads at the connection with the exhaust raise, those regulators will be manually adjusted to maintain the required amount of airflow depending on the equipment expected to be present. The headings will be ventilated with flexible round ducting of 1.07 m (42 in) diameter and an 11 kW (15 HP) fan to supply enough air for an LHD.

13.2.10 Typical Mine Layout

Access is provided by a decline, located South of the deposit. The decline starts at surface to 390L. Levels are spaced at 30 m with a level access located in the centre of the economic lodes. The stopes are mainly mined using transverses long-hole mining. A small portion of the mineralization will be mined using AVOCA or longitudinal mining method.

As shown in Figure 13-22 and Figure 13-23, the levels are all designed to contain a set of standard infrastructure required for operation. Each level is connected to the main ventilation intake raise, and an electrical bay or substation is installed to provide power. A sump is also located at the level access to control water inflows and avoid sending process water into the production ramp.

The intersection of the level access with the footwall drive will be used as a loading point for haul trucks. The intersections of the drawpoints with the footwall drive may be used as well as a loading point. Backfill operations will occur in the drawpoints. This includes storing waste rock and parking the cement mixing truck.

The main level will contain additional infrastructure, when required. For example, refuges will be excavated at strategic locations in the mine, and secondary egress accesses will be connected to the main level below 090L. The fresh air raise will be excavated near center of the main level and the egress. At the end of each level, a raise will be excavated between levels to create two (2) exhaust circuits all the way to the surface level where the main exhaust raises to the surface are located.

Figure 13-22 Typical 90L Layout

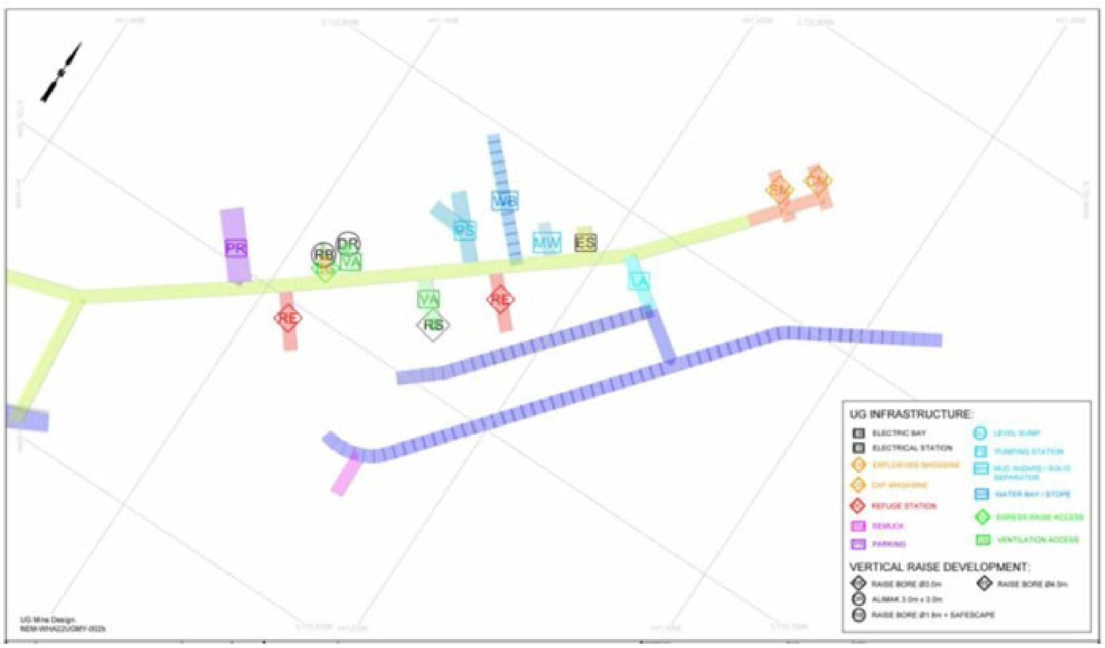


Figure 13-23 Typical 300L Layout

13.2.11 Production Schedule

A production schedule was generated from the Mineral Reserves presented in Section 12 of this Report. The schedule targets an average concentrator feed of 3,361 t/d of ore (development and stope combined). Commercial production starts at year 25 (Y25) to sustain the concentrator feed at full capacity. The underground production will sustain the concentrator feed of 1.227 Mt of ore per year.

Development and production milestones are listed below:

First production stope will be mined in the first quarter of Y25;

Commercial production (60% of 3,361t/d of ore) will be reached in the first quarter of Y25 (3 years and 2 months after the start of the ramp); and

Full production (100% of 3,361t/d of ore) will be reached mid-year Y25.

Table 13-12 presents the underground production schedule.

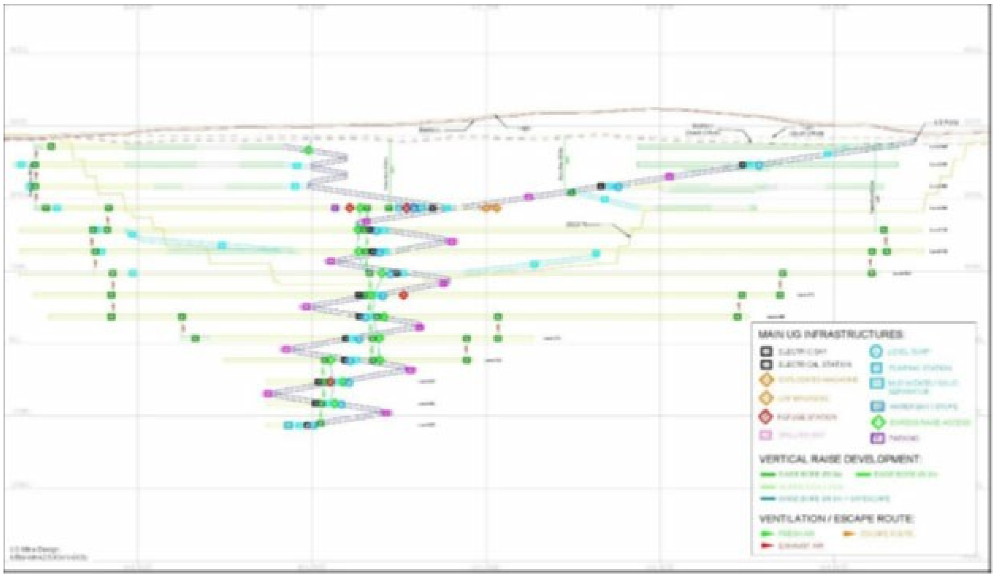
Table 13-12 Underground Production Schedule

Material	Source	Description	Total	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34
Ore	Stope	Tonnage (t)	10,943,502	0	882,190	1,152,278	1,164,157	1,165,413	1,163,110	1,197,995	1,247,917	1,238,223	1,240,685	491,533
		Li_2O Grade (%)	1	0.00	1.23	1.29	1.28	1.30	1.22	1.32	1.28	1.37	1.36	1.28
		Li_2O Tonnes (t)	141,872	0	10,893	14,918	14,850	15,199	14,190	15,767	15,985	16,926	16,860	6,283
	Dev.	Tonnage (t)	609,465	17,961	246,063	86,590	68,967	68,471	77,925	32,188	2,558	6,760	204	1,777
		Li_2O Grade (%)	1.30	1.19	1.26	1.32	1.39	1.41	1.38	1.39	1.54	1.27	1.38	1.10
		Li_2O Tonnes (t)	8,043	213	3,092	1,145	961	965	1,072	446	39	86	3	20
	Stope & Dev.	Tonnage (t)	11,552,967	17,961	1,128,253	1,238,868	1,233,125	1,233,884	1,241,035	1,230,184	1,250,475	1,244,983	1,240,890	493,311
		Amphibole Tonnes (t)	213,438	219	19,089	32,321	19,052	22,039	32,629	10,891	30,107	23,421	16,205	7,464
		Li_2O Grade (%)	1.30	1.19	1.24	1.30	1.28	1.31	1.23	1.32	1.28	1.37	1.36	1.28
		Amphibole Percent (%)	1.80	1.22	1.69	2.61	1.55	1.79	2.63	0.89	2.41	1.88	1.31	1.51
		Li_2O Tonnes (t)	149,914	213	13,985	16,064	15,812	16,164	15,262	16,213	16,024	17,011	16,863	6,303
Marginal	Stope & Dev.	Tonnage (t)	156,234	9,604	89,785	15,297	8,774	11,165	12,758	5,843	531	1,363	0	1,113
		Amphibole Tonnes (t)	7,673	493	3,396	823	835	513	1,016	310	10	195	0	82
		Li_2O Grade (%)	0.70	0.67	0.68	0.69	0.68	0.67	0.67	0.70	0.72	0.57	0.00	0.79
		Amphibole Percent (%)	4.90	5.14	3.78	5.38	9.52	4.60	7.96	5.30	1.88	14.30	0.00	7.36
		Li_2O Tonnes (t)	1,063	65	611	105	60	75	86	41	4	8	0	9
Ore & Marginal	Stope & Dev.	Tonnage (t)	11,709,201.10	27,564	1,218,038	1,254,165	1,241,899	1,245,049	1,253,793	1,236,026	1,251,006	1,246,346	1,240,890	494,424
		Amphibole Tonnes (t)	221,111	712	22,485	33,144	19,887	22,553	33,645	11,201	30,117	23,616	16,205	7,546
		Li_2O Grade (%)	1.29	1.01	1.20	1.29	1.28	1.30	1.22	1.32	1.28	1.37	1.36	1.28
		Amphibole Percent (%)	1.89	2.58	1.85	2.64	1.60	1.81	2.68	0.91	2.41	1.89	1.31	1.53
		Petalite Percent (%)	3.61	0.00	0.78	2.49	2.78	2.60	2.31	3.77	3.37	6.09	7.08	7.66
		Li_2O Tonnes (t)	**150,977**	**278**	**14,596**	**16,169**	**15,871**	**16,239**	**15,348**	**16,254**	**16,028**	**17,019**	**16,863**	**6,312**

13.3 Underground Infrastructure

Figure 13-24 illustrates the general underground infrastructure location.

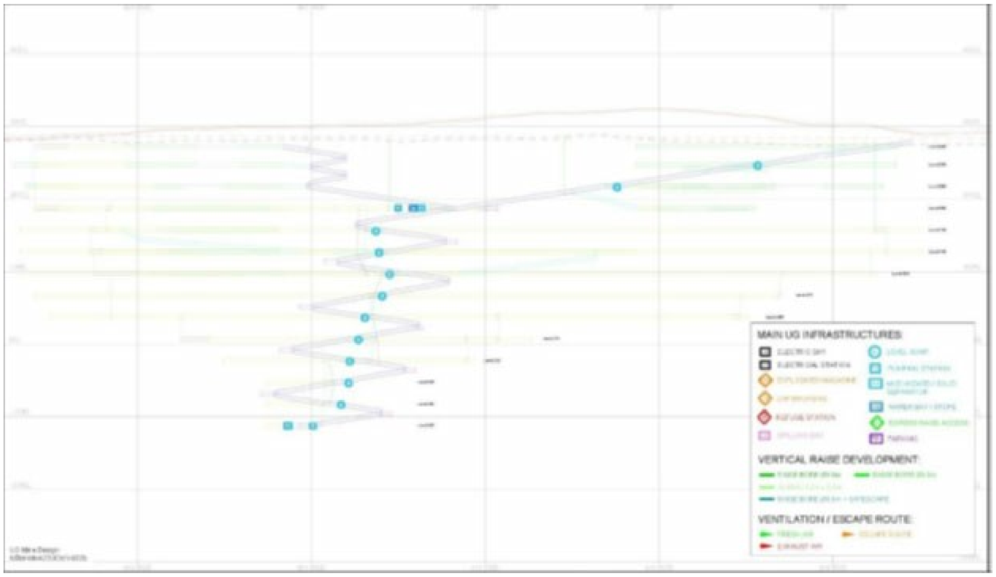
Figure 13-24 Underground Infrastructure Location



13.3.1 Water - Mine Drainage and Water Pumping Arrangement

The process water system will consist of managing dirty water to direct it to a central area and centralizing the mud extraction process in one single location prior to returning clean water to water stopes prior to reinsertion in the process. Meanwhile, mud will be disposed of either in mined out stopes, or in trucks of ore, should it reveal economical values generated by production holes cuttings in the mud. Figure 13-25 depicts a typical mine drainage and water pumping arrangement.

Figure 13-25 Underground Water / Pumping Network Infrastructure Location



13.3.1.1 Dewatering Strategy & Pumping System

All the upper levels, on the East and West side will have drain holes to direct the dirty water to the lower levels. Sumps will consist of two rounds deep enough to store dirty water and to drill a drain hole out of the main circulation way which will allow water to cascade down from sump to sump until it reaches a pumping station or a mud extractor. There is also the strategy to pump in cascade from levels to levels until it reaches a temporary pumping station or mud extractor location during the development phase. For these duties, a series of 5 HP submersible pumps have been selected. Also located near the mud extractor, 090L, a water bay / stope will be excavated to store and supply water for the drilling and other mining operation. The mud extractor will be located in its own excavation as well as the pumping station.

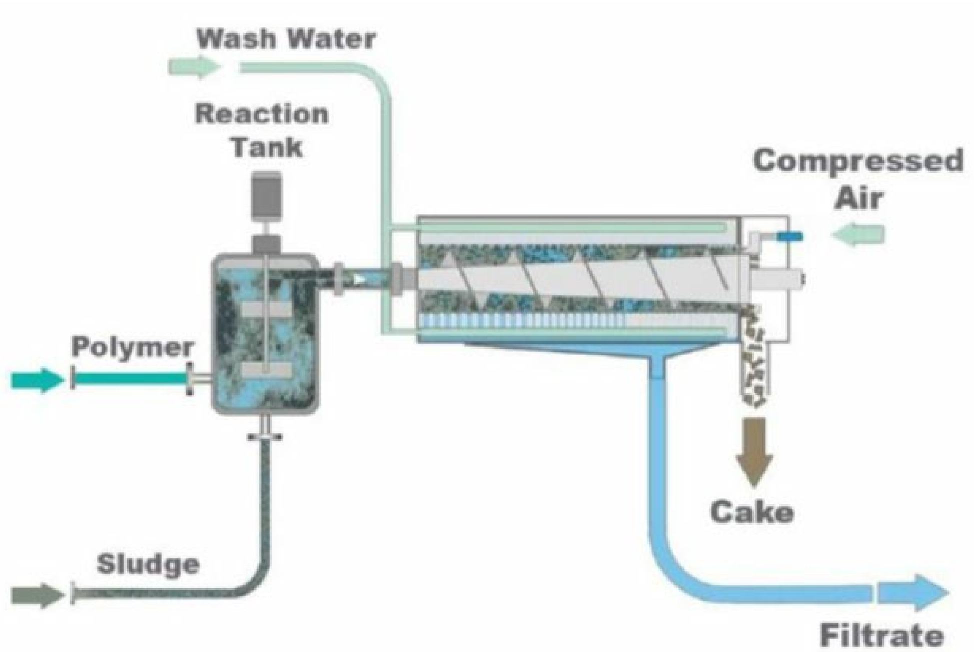
Same strategy will also apply from Levels 120 to 390, the dirty water will get directed to the lower levels through drill holes from sump to sump. During development, the water will get pumped from the lower level in cascade to 090L, where a pumping station will be established during development to pump the water all the way to the mud extractor next to it on 090L. A cavity pump with a capacity of pumping 50 m3/h for a minimum head pressure of 200 m shall be use.

After 390L is reached, the lower point of the mine, the main mud extractor excavation and the main pumping station will be completed. The main pumping system consists of one (1) centrifugal pump with a capacity of 50m^3/h that will supply water to the water stope / bay located on 090L.

13.3.1.2 Mud Extractor System

It is recommended to install a screw press to extract the mud from the dirty water. A cake of 60 to 70% solid produced by the screw press will be much easier to handle with scoops and trucks than a 30% slurry if using other methods. Figure 13-26 is a typical schematic representation of the screw press system.

Figure 13-26 Screw Press Schematic



13.3.1.3 Clear Water Installation

For the early development phase of the mine, the clean water supply will come from the surface. During the progression of the mine development, a water stope / bay will be excavated at 090L. The pumping and mud extraction system is described in Section 13.3.1.2 and will supply that reservoir with the required water to support all the drilling operation of the mine. Clean water will be distributed to the mine operation through a 2-inch pipe installed in the ramps and the level access of the mine.

13.3.2 Air

Air compressors have been located on surface. The compressor room dimensions are 6.6 m by 5.6 m (or 37 m2) and will enclose two air compressors developing 1,000 cfm/each. Compressed air will be distributed using a 6-inch insulated steel pipe that will be extended into the mine as development progresses.

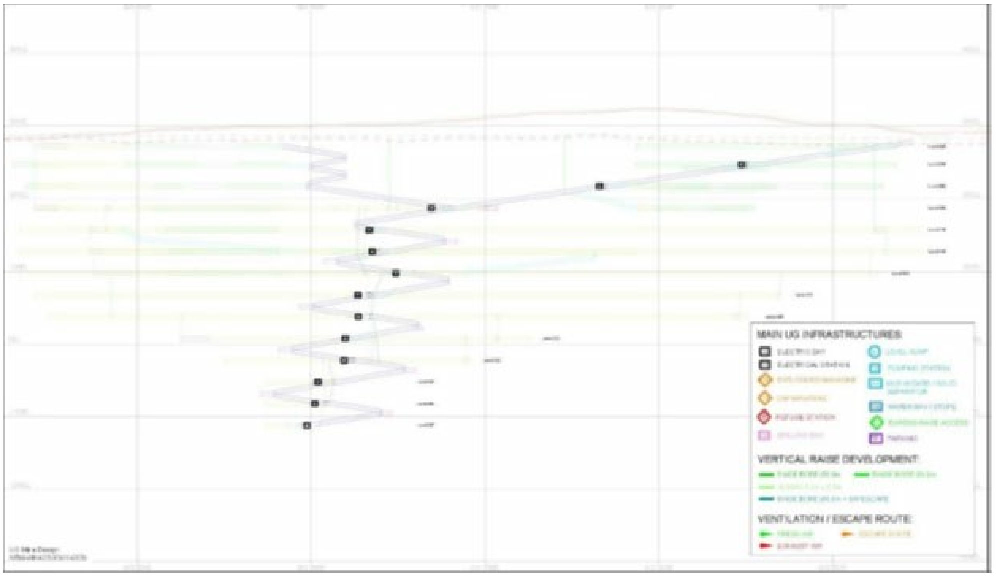
13.3.3 Fuel, Oil, and Grease

It is assumed that all underground equipment, with the exceptions of man carrier; Landcruiser; UG haul trucks; boom truck; and zoom boom, will be filling up their fuel reservoirs underground, from the fuel truck, unless they must go to surface for other specific reasons. Scooptrams might eventually become an exception as their travelling speed is reasonable and their fuel consumption is important. They might also be required to refill directly from the fuel truck when it comes to refill the SatStat☐ to avoid emptying the fueling station too fast. Remuck excavations in the ramps will be repurposed as fuel stations, materials bays, temporary refuges, or any other purposes as deemed necessary.

13.3.4 Electrical Distribution

Power will be distributed to the various consumers from each electric substation using a 600-V cable installed as the development progresses. Each substation will connect through service holes to an electric bay on one level above and below. This will allow to reduce the distance to be covered and minimize the number of substations required. Junction boxes will be installed in the electric bay on each level without a substation. The electric distribution network is illustrated in Figure 13-27.

Figure 13-27 Underground Electrical Network Infrastructure Location



13.3.5 Communication System

Communications and controls hardware will be distributed — and the network expanded — as development progresses. Leaky feeder and Wi-Fi access points will be distributed throughout the mine over time. Fixed radios in the mobile equipment and handheld radios for the personnel will be completed by strategically located base stations distributed in key locations (refuges, shop, offices, etc.).

13.3.6 Lunchroom / Mine Refuge

Refuges have been planned in order to meet legal requirements in terms of distance between each refuge. A 1 km distance is respected between an active workplace and the refuge in the underground mine. These are also typically used as lunchrooms and are designed as such. With the current mine configuration, it has been established that three (3) separate refuges will be required at Levels 90, 210, and 330.

13.3.7 Material Storage

No specific excavation has been planned specifically for the purpose of material storage. It has been assumed that these would be built on an as-needed basis in remucks or other temporary excavations that are repurposed over time as development and as activities progress.

13.3.8 Maintenance Bay

There will be no underground maintenance shop. A small tire changing bay has been planned for the purpose of changing flat tires and completing minor repairs. It has been assumed that these would be built on an as-needed basis in remucks or other temporary excavations that are repurposed over time as development and as activities progress. Any full-scale maintenance work will be performed in the surface maintenance shop.

13.3.9 Cap Magazine

Magazine dimensions are 5 m wide by 12 m long. All components (front wall, paint, door, shelves, etc.) conform with Quebec's regulation on occupational health and safety in mines.

The Cap Magazine is located on Level 90 (090L).

13.3.10 Explosives Magazine

The magazine can hold up to 24 emulsion bins or pallets of explosives for a total capacity of 30,000 kg. All components (front wall, paint, door, shelves, lighting, and heating units, etc.) conform with Quebec's regulation on occupational health and safety in mines.

The Cap Magazine is located on 090L.

13.3.11 Cement Distribution Station

After being transferred on surface storage area from tote bags to a small silo, dry cement will be transferred by a screw conveyor to an underground dry cement delivery truck. The truck will then travel underground to deliver cement to the area where it will be mixed with water in an underground cement mixing truck. Once mixed, the cement milk will then be poured directly in the LHD bucket and then dumped in the stope.

13.3.12 Mobile Equipment

Table 13-13 summarizes the underground mine equipment fleet that will be required during the peak period considering the development and production plan defined in the DFS. This fleet was selected in consideration of the quantities of material to be mined and moved, the geometry of the orebody. The mine will be operated by a Contractor who will supply the equipment fleet.

Table 13-13 Underground Equipment Fleet

Equipment	Capacity	Quantity required
Main Equipment		
Truck	50 t	6
Truck	30 t	2
Production scoop	12 yards	6

Development scoop	8 yards	2
Production drill with V30 attachment	3½ to 8½ inch	1 - Contractor
Production drill Top Hammer	3½ to 5 inch	3
Jumbo	2 Booms	2 to 3
Cable drill		1
Development loading unit		1
Bolters		3
Scissor lift		3
Auxiliary Equipment		
Auxiliary scoop	3.5 yards	1
Block holer		1
Boom truck		1
Zoom boom		2
Grader		2
Casette Truck		1
Boom truck Casette		1
Lube truck Casette		1
Production loading unit Casette		1
Landcruiser		6
Service tractor		4
Concrete truck		3
Dry cement transport truck		1
Mixing truck		1
Small excavator		1
Small loader		1

In addition to the underground fleet, a small surface fleet will be required to support the operation as presented in Table 13-14.

Table 13-14 Surface Support Equipment Fleet

Equipment	Capacity	Quantity required
Bus		1
Pickup	Mined Operations	2
Pickup	Maintenance	1
Surface loader	980 or eq.	1
Total		5

14 PROCESSING AND RECOVERY METHODS

Section 10 of this Report described the metallurgical test work and how the results were used to derive the Process Flow Diagrams (PFD) and mass balance. The process design is centered on the concentrator located 675 m north east of the Whabouchi mine open pit.

Descriptions of the various areas encompassing the processing plants are provided herein. This information serves as the basis for the development of the capital and operating cost estimates presented in Section 18.

14.1 Whabouchi Concentrator

The Whabouchi concentrator was originally designed in 2014 to produce 216,485 dry tonnes per year of 6.0 % Li_2O spodumene concentrate. The concentrator building was erected in 2016 based on the 2014 flowsheet. The flowsheet was modified in 2018 in order to increase spodumene concentrate grade to 6.25 % Li_2O at 215,000 dry tonnes per year of production, which included the switch from mechanical flotation cells to a combination of coarse particle flotation and column flotation. The increase in concentrate grade was implemented to mitigate the process risks associated with flash calcining of the spodumene concentrate at the conversion plant. In 2019, this design was refined based on the expected mine plan and, due to a lower feed grade, the production target was reduced to 205,000 dry tonnes per year of 6.25 % Li_2O spodumene concentrate, and this design was maintained for equipment procurement and the start construction of the plant prior to halting of the project at the end of 2019. In 2021, under new ownership, a detailed evaluation was performed to mitigate the process risks associated with flash calcining leading to the decision to revert to a rotary kiln for spodumene calcination. This decision allowed for a reduced concentrate specification with increased lithium recovery. Based on additional metallurgical testing and a revised metallurgical balance, the plant design was updated for a target spodumene concentrate of 5.5 % Li_2O.

Due to the complexity of the flowsheet and the constraints associated with the existing procured equipment, constructed building and half constructed plant, and current overall design, it is the opinion of the QP that the selected availability of 91.5% is optimistic and a significant effort will be required to achieve it. A review of the process data and modifications is in progress and will continue as the Project moves forward to flag and mitigate identified risks and establish contingency plans for dealing with process upsets. A list of high-level recommendations and modifications have been prepared and are listed in Section 23. An additional detailed list of risks, proposed modifications, and verifications to complete has been prepared by DRA and will be actioned in this phase of the project with the goal of achieving targeted plant performance. However, there is still a significant risk that ramping up to the stated production values will take longer than indicated in this report and will require additional sustaining capital projects. In the QP's opinion, the upper limit of plant availability may be lower than 91.5% and may be as low as 75 to 85%, or recovery may decrease to maintain concentrate production. It should be recognized that reductions in plant capacity will lead to reduction in tonnes of concentrate produced by the process plant.

The process plant effective utilization, or effective operating hours per year at nominal tonnage as a percent, was simulated to determine an estimated availability based on the process complexity, equipment sizing, available buffers, planned maintenance, and unplanned downtime. Following the initial simulation which suggested an effective availability of 85.9%, efforts were focused on investigating potential modifications to ensure that the concentrator output matches the conversion plant nominal feed throughput during the entire project life. Following the dynamic simulations, it was identified that the absence of buffer at the flotation feed and the forced stoppage of the concentrator during the Goose Break were two factors limiting overall plant availability. Operating the concentrator during the Goose Break increases yearly uptime by 11 days (or 3.0% on an annual basis) and adding a six-hour residence time flotation feed buffer tank increases availability by an additional 2.6%, which brings the effective concentrator availability at 91.5%. This corresponds to an average annual concentrate production of 229,797 tonnes throughout the entire Life-of-Mine. This remains slightly under the initial design capacity of 235,000 tonnes at 5.50% Li_2O, but the remaining shortfall will be compensated by stockpiling the concentrate produced before the conversion plant start-up and then progressively reclaiming this stockpile in subsequent years.

The dynamic simulation also predicted an effective utilization of 66.6% for the crushing circuit, as per design. It should be noted that a design availability of 80% was used for the DMS circuit to allow this circuit to catch-up and compensate for higher downtime compared to the rest of the plant. It is important to note that the current estimated effective utilization estimate assumes stable operation with skilled operators and an appropriate preventative maintenance program in place following ramp-up of production.

The process plant performance has been estimated based on the current mine plan data. Average annual throughputs, lithium grades, and other data has been estimated for Years 1-4 and 5-24 and 25-34. The Spodumene concentrate will be produced by two (2) distinct processes: DMS circuit and flotation circuits. Based on the process plant feed, the DMS will recover 8 to 11% by weight, while the flotation circuit will recover 12 to 14% by weight for a total of 21 to 24% weight recovery, depending on the year.

The concentrator is designed to produce a spodumene concentrate containing 5.5% Li_2O or 68.5% spodumene ($LiAlSi_2O_6$), from an ore containing an average 1.32% Li_2O. To achieve this concentration, the beneficiation processes include crushing, ore sorting, hydraulic separation, dense media separation (DMS), magnetic separation, grinding, attrition scrubbing, desliming, and flotation. Before leaving the concentrator, the concentrate will undergo further steps of thickening, filtration, drying (DMS concentrate only), and material handling, including storage and loading of spodumene concentrate in containers on road trucks. The concentrator production average for first four years is 220,846 dry tonnes of spodumene concentrate per year, it then averages 227,021 tonnes per year in years 5 to 24 and finally 238,841 tonnes per year during the underground mining in years 25 to 34 Tailings will be transported to the CSF by trucks.

NLI's process team was responsible for the general flowsheet design and process definition in this phase of the project, which includes predictions of plant performance, recovery, overall plant availability and utilization, and definition of the process design criteria.

14.1.1 Process Design Criteria

The 2019 Whabouchi concentrator design was based on the production of 215,000 dry tonnes per year of 6.25% Li_2O spodumene concentrate from a feed grade of 1.53% Li_2O, resulting in an annual throughput rate of 1,030,831 tonnes of ore.

Based on the updated metallurgical balance and flowsheet changed in 2021-2022, the expected average throughput for Years 1-4 is 1,103,988 tonnes per year. This is based on the current mine plan which targets an average production of 220,846 dry tonnes of 5.5% Li_2O spodumene concentrate from a feed grade of 1.32% Li_2O. The concentrate production and recovery estimates are based on individual stage recoveries from bench and pilot scale work and have been compiled into the metallurgical model. This overall estimate has not yet been confirmed through full pilot plant testing (not required at a PFS level).

The Whabouchi concentrator will operate 24 hours per day, seven (7) days per week, 52 weeks per year. The general concentrator operating availability will be 91.5% while the DMS section will operate at 80%. The crusher will be operated based on 66.6% availability. The concentrator capacity has been established at an average rate of 3,067 dry tonnes per day to the crusher.

The crusher and concentrator have been sized to meet the parameters Table 14-1.

Table 14-1 Process Design Basis

Concentrator Capacity					
Parameter	**Units**	**Design value from 2019 Report**	**Average Years 1-4**	**Average Years 5-24**	**Average Years 25-34**
Annual Ore Processing Rate	dry t/y	1,030,831	1,103,988	1,119,938	1,243,024
Average Ore Processing Rate	dry t/d	2,824	3,025	3,068	3,406
Spodumene Ore Grade	% (Li_2O)	1.53	1.33	1.32	1.29
Ore Sorter Stage Recovery (based on sorter feed, excl. bypass)	%	98.8	95.8	99.8	99.8
Concentrator Stage Recovery	%	86.2	85.8	86.9	82.1
Overall Lithium Recovery (Sorting + Concentrator)	%	85.2	82.8	84.3	82.0
Spodumene Concentrate Grade	%	6.25	5.50	5.50	5.50
Annual Spodumene Concentrate Production	dry t/y	205,000	220,846	227,021	238,841
Daily Spodumene Concentrate Production	dry t/d	562	605	622	654
ROM Ore Moisture	%	2.0	2.0	2.0	2.0
Crusher Effective Availability	%	66.6	66.6	66.6	66.6
DMS Circuit Operating Time	%	80.0	80.0	80.0	80.0
Concentrator Effective Availability	%	92.0	91.5	91.5	91.5
DMS Circuit Feed (F80)	mm	6.2	6.2	6.2	6.2
HydroFloat Separation Feed (F80)	mm	0.63	0.63	0.63	0.63
Column Flotation Circuit Feed (F80)	mm	0.17	0.17	0.17	0.17
DMS Circuit Operating Time	%	80.0	80.0	80.0	80.0

14.1.2 Mass and Water Balances

Table 14-2 shows a summary of the average in- and out-flows from the concentrator for the first four years of operating. The daily ore throughput is 3,025 tonnes requiring 254 m3/d of makeup water. Most of this water is lost in tailings which are about 12% moisture for dry stacking in a CSF with waste rock from the open pit.

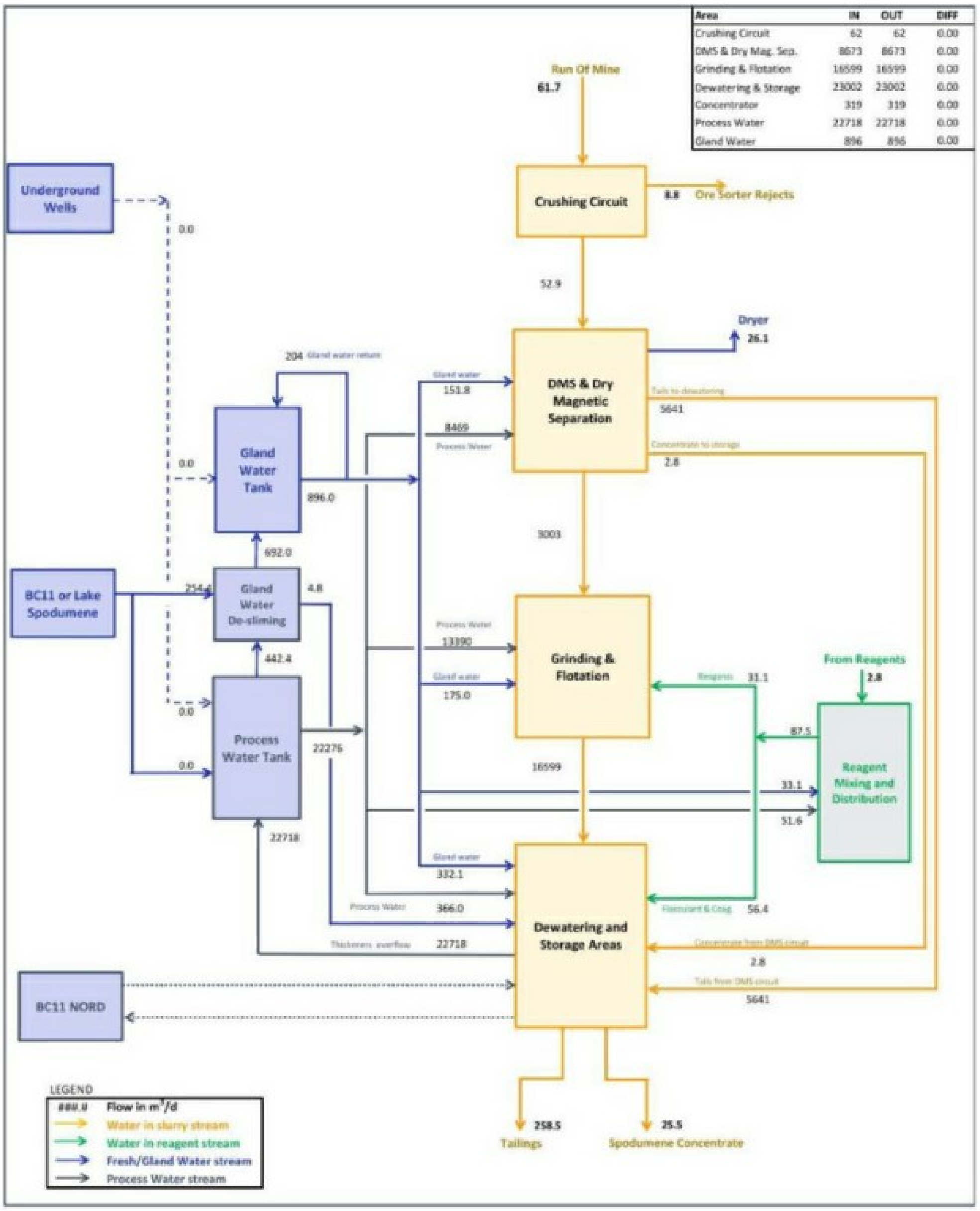
Table 14-2 Whabouchi Concentrator Summarized Process Mass Balance, Years 1-4

Mass Entering System				Mass Exiting System			
Streams	**Dry Solids (t/d)**	**Water (m3/d)**	**Total Mass (t/d)**	**Streams**	**Dry Solids (t/d)**	**Water (m3/d)**	**Total Mass (t/d)**
Fresh Water from several sources	0.1	254.4	254.5	Water Evaporation from Dryer	0.0	26.1	26.1
Spodumene Ore to Concentrator	3,024.6	61.7	3,086.4	Ore Sorter Rejects	447.9	8.8	456.7
Dense Media	0.8	0.0	0.8	Final Concentrate	605.1	25.5	630.5
Reagents	0.0	2.8	2.8	Final Tailings	1,972.6	258.5	2,231.1
Total Entering	3,025.6	318.9	3,344.4	Total Exiting	3,025.6	318.9	3,344.4

Differences may be due to rounding.

Figure 14-1 illustrates a summary of the water flows around the different circuits of the concentrator.

Figure 14-1 Typical Water Balance, Years 1-4



14.1.3 Flowsheets and Process Description

Simplified flowsheets are presented in Figure 14-2 and Figure 14-3. These flowsheets are indicative of the process. The crushing facility can operate independently from the concentrator to the extent that the fine ore can be stockpiled. The concentrator has three (3) distinct process areas: dense media separation (DMS), flotation, and dewatering. Figure 14-2 shows the crushing area with ore sorting. Figure 14-3 shows the concentrator simplified flow sheet.

14.1.3.1 Crushing and Ore Sorting

A run-of-mine (ROM) stockpile will be located near the primary crusher. The crushing area is split in three (3) areas, primary crushing, ore sorting, and fine crushing, as shown in Figure 14-2.

The ROM material from the mine will be dumped into the ROM stockpile. The material will be rehandled and blended via front end loader and transferred to the ROM feed hopper. The hopper is equipped with a static grizzly and rock breaker to provide top-size protection to the crushing circuit. The ROM material is withdrawn from the ROM feed hopper by an apron feeder located underneath the hopper and is transferred into the jaw crusher. The primary crusher discharge will have a particle size of 80% passing (P80) 95 mm. The primary crushed rock will be transported via conveyors to the coarse ore triple deck screen. The top deck removes the +80mm material and transfers it via conveyor to the secondary jaw crusher.

The second jaw crusher product has a P80 of 72 mm and discharged onto the primary crusher discharge conveyor prior to being returned to the triple deck screen. The triple deck screen middle deck produces a -80+35mm product (less than 80 mm, greater than 35 mm) which is sent to the coarse ore sorter, while the bottom deck produces a -35+10mm (less than 35 mm, greater than 10 mm) product which is sent to the fine ore sorter. The screen undersize, or -10mm material, is transported to fine crushing area.

The ore sorters operate by scanning and identifying individual particles and preferentially rejecting waste from ore using compressed air fired through a series of fine nozzles. The sensor technology used for the particle ore sorters is X-Ray Transmission (XRT). For each the coarse and fine ore sorters, material is transferred via an air-supported conveyor belt and fed to the ore sorter vibrating feeders. The vibrating feeders are used for feed presentation, which helps create a mono-layer of material on the ore sorter belt. The material passes through the ore sorter where each particle is identified as ore or waste.

The waste particles are targeted for removal and ejected with a controlled compressed air puff. For both sorters, the accepted material is transferred to the accepts conveyor via transfer conveyors, while the rejected material is discharged directly onto a common rejects conveyor. The waste material is conveyed to a 500 t rejects stockpile where it is rehandled by front-end loader and removed by mine trucks to the CSF. The accepted material is transported to the fine crusher ore facility via belt conveyors.

The fine crushing facility is composed of screening and 2-stage crushing. The coarse ore triple deck screen undersize and the ore sorter accepted material are combined and conveyed to the cone crusher screening area. The cone crusher screen feed conveyor discharge chute is equipped with a bypass to transfer material to a 500 t stockpile in case of upset. The combined material is screened on a double deck vibrating screen. The screen top deck creates a size separation at 20 mm, while the bottom deck separates at 9 mm. Both screen oversize discharges are transported to a coarse ore bin. This bin has two (2) compartments: one feeding the secondary crushers, -80+20mm, and one feeding the tertiary crusher, -20+9mm. There are two (2) secondary crushers and one tertiary crusher. All three (3) crushers operate in closed circuit with the double deck screen to produce a crushed product with P80 of 5.8 mm which will be conveyed to the fine ore overflow hopper near the concentrator building.

The secondary crushers will be standard cone crushers that crush the top deck oversize to a P80 of 21 mm. The tertiary crusher will be a short head cone crusher that crushes the bottom deck oversize to a P80 of 12.7 mm. All crusher discharges will be re-directed to the double deck vibrating screen via two (2) conveyors.

Figure 14-2 Simplified Flowsheet of Crushing and Ore Sorting

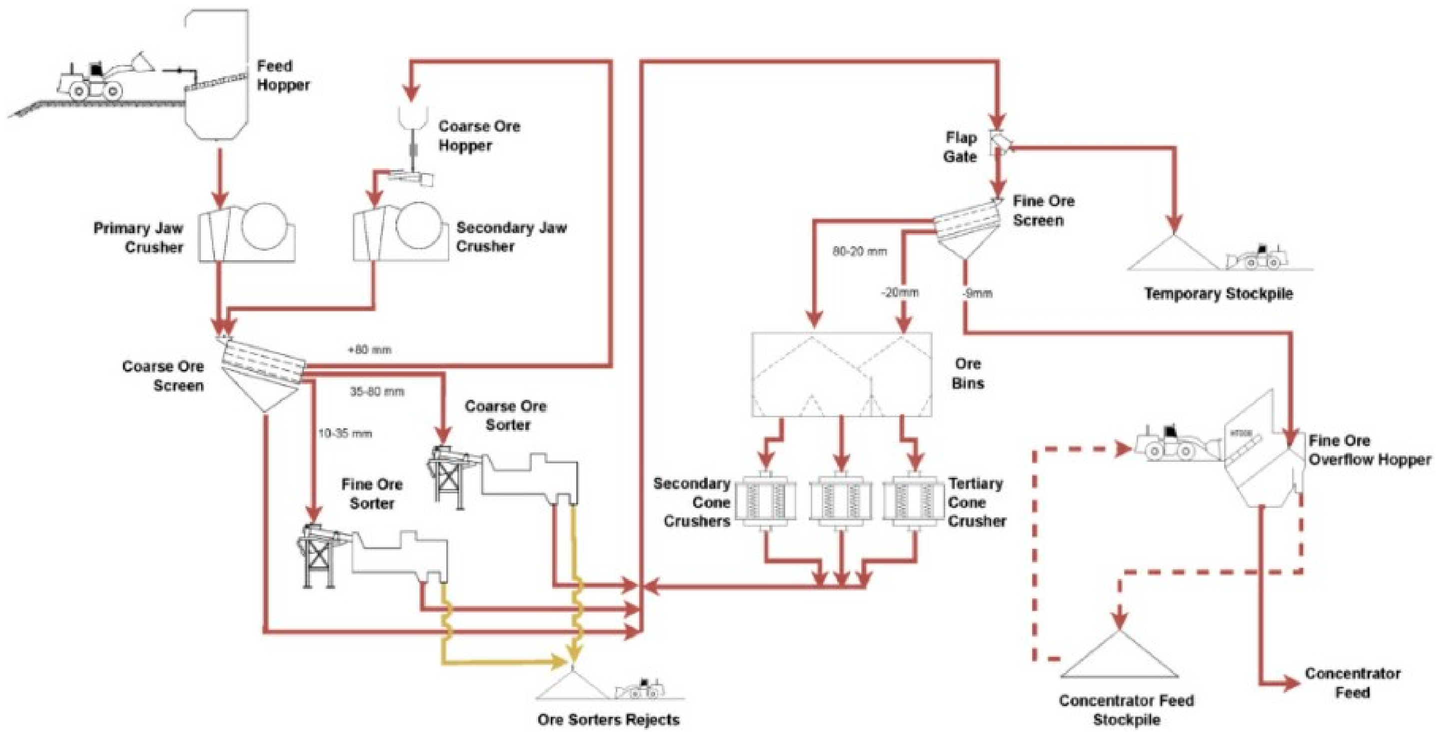
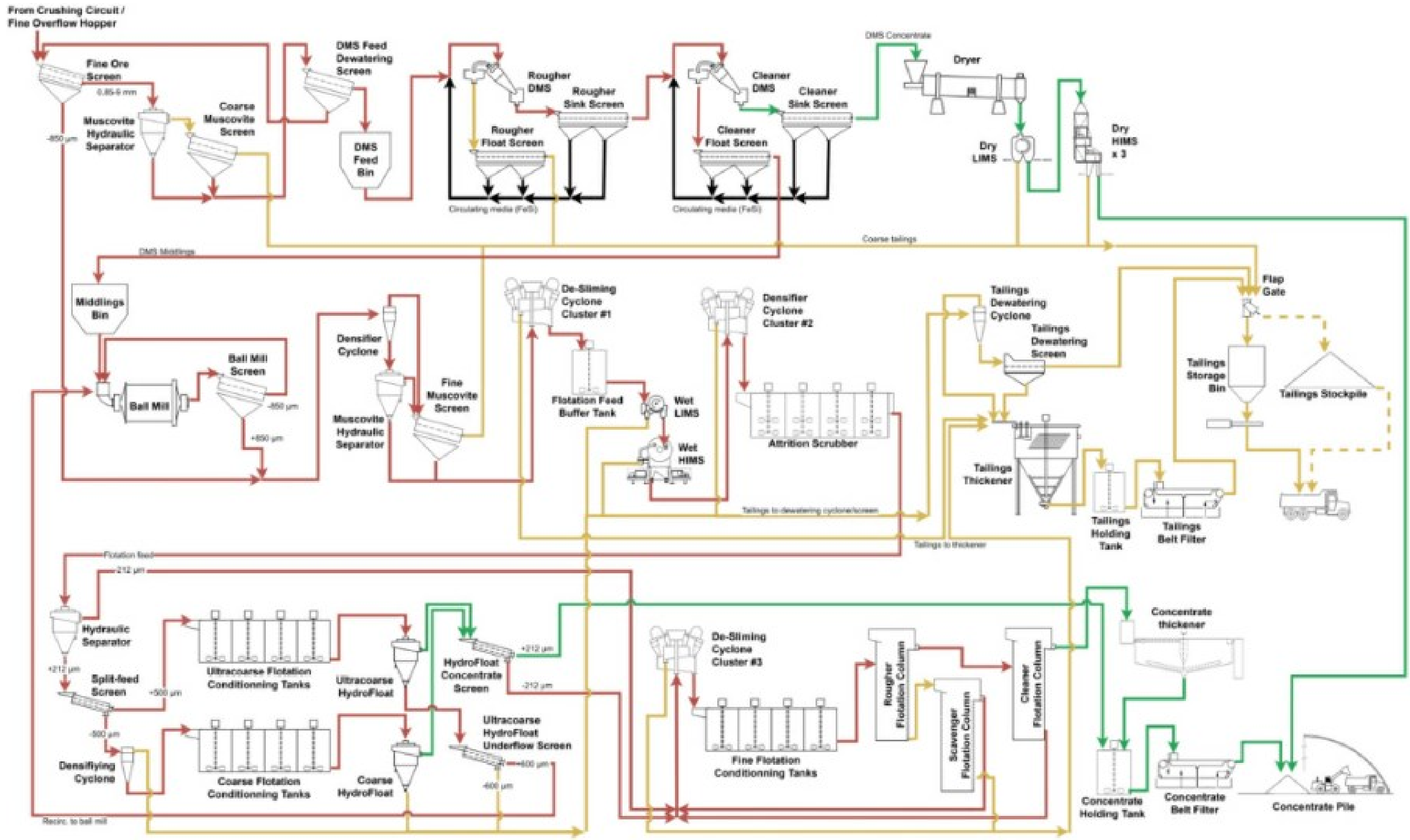


Figure 14-3 Simplified Flowsheet of Concentrator



14.1.3.2 Concentrator

The fine ore from the crushing area is transferred to the fine ore overflow hopper. The design of this hopper will allow material to direct feed the concentrator, while any surplus overflows from the hopper is transferred to the 10,000-t capacity fine ore stockpile. The material is rehandled via front-end loader and is transferred to the re-feed end of the overflow hopper. The hopper and re-feed areas are located within a covered dome.

The concentrator includes multiple stage of separation and concentration as well as products dewatering circuits. A simplified flow sheet is presented in Figure 14-3.

a) Preparation to Dense Media Separation

Ore is withdrawn from the fine ore overflow hopper by a belt feeder. The fine ore is transferred to the fine ore screen feed conveyor, which acts as the main plant feed conveyor and discharged onto the fine ore screen. The fine ore screen is a double deck vibrating screen that has a top deck with 2.7 mm slot openings and a bottom deck with 0.85 mm openings. The top deck is a protection deck to improve screening performance and prevent bottom deck damage from coarse gravel. The screen undersize is pumped to the fine muscovite removal circuit. Both top deck and bottom oversize report to the coarse muscovite removal circuit.

b) Coarse Muscovite Removal

The coarse muscovite removal hydraulic separator uses water as the separating medium. Water is injected into the lower part of the separator and flows upwards in order to elutriate the feed slurry. The hydraulic separator removes coarse muscovite flakes while entrains finer ore particles in the overflow. The hydraulic separator overflow is screened over a single deck screen with 2 mm openings. The screen oversize goes to final tailings, while the screen undersize is mixed with the hydraulic separator underflow and pumped to a dewatering screen prior to the DMS circuit. The screen oversize discharges into the DMS feed bin while the screen undersize returns to the fine ore screen.

c) Dense Media Separation

Dense Media Separation (DMS) is a gravity separation technique that uses fine ferrosilicon as dense medium to enhance the separation of material by specific gravity (SG) when proper feed preparation is performed. The fine ferrosilicon, when combined with water, behaves like a heavy liquid. The apparent specific gravity of this “heavy liquid” is controlled by water addition to the ferrosilicon slurry. The material with the higher specific gravity is referred to as sinks, while the less dense material is floats. In the case of spodumene concentration, the sinks contain the spodumene while the floats are generally gangue minerals.

The Whabouchi concentrator design is based on a 2-stage dense media separation unit; a rougher stage and a cleaner stage. The rougher stage consists of one (1) DMS cyclone and produces a rougher sinks and a rougher floats stream. The rougher floats, consisting of low SG gangue minerals, become the DMS tailings. The rougher sinks are sent to the cleaner stage for a second SG separation. The cleaner stage consists of a single DMS cyclone and produces a cleaner sinks and a cleaner floats product. The cleaner floats are referred to as middlings and contain unliberated or locked spodumene particles and other minerals. The middlings are transported by conveyor to the ball mill. The cleaner sinks are the DMS concentrate and are sent to the dryer.

The sinks and floats of each stage are separated from the dense medium using drain and wash screens. Each DMS circuit includes a circulating media circuit and dilute media circuit. The circulating media is at the correct density and is used to perform the SG separation. The dilute media has an excess of water. The ferrosilicon in the dilute media is densified using low intensity magnetic separators and densifying cyclones and reused in the DMS process to maintain density in the circulating media.

d) Concentrate Drying and Magnetic Separation

The DMS concentrate is dried in an electric rotary dryer to a moisture content of less than 1% as dry magnetic separation requires free flowing material to perform efficiently.

The first stage of magnetic separation is affected by low intensity magnetic separation; the second stage uses rare earth high intensity magnetic separation. The final DMS concentrate grade will be approximately 6.0% Li_2O. The magnetic material is transported to tailings via conveyors.

The final DMS concentrate is transferred to the combined concentrate conveyor prior to being discharged into the concentrate stockpile.

e) Grinding

The DMS middlings need to be ground to liberate the finer spodumene particles. They enter the ball mill via the ball mill feeder conveyor from the middlings storage bin. The ball mill discharge is pumped to the ball mill fine screen with apertures of 850 µm. The screen oversize reports back to the ball mill. The screen undersize, at a P80 of 640 µm, is pumped to the fine ore screen undersize to join the natural fines for fine muscovite removal stage.

f) Fine Muscovite Removal

The fine ore screen undersize and the ball mill product are pumped to a cyclone to densify the slurry. The cyclone underflow is introduced into the fine muscovite removal hydraulic separator. The muscovite and fine particles of less than 200 µm are pushed to the overflow. The overflow is screened at 212 microns to recover fine ore particles that would be lost with the muscovite flakes. The screen oversize is mostly muscovite and is sent to tailings.

The hydraulic separator underflow is mixed with the screen undersize and pumped to a cyclone cluster for de-sliming. The slimes (overflow) are rejected to tailings, while the de-slimed material (underflow) is transferred to the wet magnetic separation circuit.

g) Wet Magnetic Separation

Magnetic separation is performed in two (2) stages. The first step is the removal of residual dense media and steel from the ball mill with a Low Intensity Magnetic Separator (LIMS). The second stage is a Wet High Intensity Magnetic Separation (WHIMS) to remove paramagnetic material from the flotation feed. The magnetic products are sent to tailings, while the non-magnetic products are the feed to the densifying cyclone cluster. These cyclones have a dual duty to remove more slimes and to densify the attrition scrubber feed.

h) Attrition Scrubbing

This circuit is mainly for the removal of more deleterious particles prior to spodumene flotation. The attrition step is used to loosen tenacious fine particles from the surface of the coarser particles. It must be performed at a high pulp density with high sheer mixing to be effective. A dispersant is added to enhance loosening of the fines and caustic soda is used to increase pH to 12 and make the dispersant more effective.

The attrition scrubber discharges into a hydraulic separator. This hydraulic separator performs a size separation to ensure that the quantity of fines in the underflow are kept to a minimum before going to hydroflotation. The separator has a cut size of 200 µm. The underflow is sent to the coarse particle flotation circuit (hydroflotation), while the overflow goes to a third desliming stage to remove the liberated slimes/fines and provide the high slurry density required for spodumene conditioning before column flotation.

i) Spodumene Hydroflotation

The spodumene hydroflotation circuit consists of size separation, conditioning, flotation, and screening of the recovered concentrate. The hydraulic classifier underflow is sent to the hydroflotation split-feed vibrating screen, which separates the material into an ultracoarse +500 µm and a coarse -500 µm stream. The screen oversize is sent directly to the ultracoarse conditioning tanks, while the undersize is first sent to a densifying cyclone prior to discharging into the coarse conditioning tanks.

The ultracoarse and coarse circuits each start with four (4) stages of high-density conditioning using a spodumene collector. Sulfuric acid is introduced to lower the pH of the flotation feed and achieve a conditioning pH of 8. Proper conditioning can only occur at high solids density and high energy intensity in the conditioning tanks. The final stage of the high-density conditioning tanks is pumped directly to the ultracoarse and coarse hydroflotation cells.

Hydroflotation is a combination of a hydraulic separator and column flotation; the brand name provided by Eriez is known as the HydroFloat separator. This system is used to perform coarse particle flotation and combines an upwards flow of water with traditional flotation to decrease particle/bubble detachment events within the flotation device. The water up-flow entrains some finer particles non-selectively.

The hydroflotation concentrates are combined and are screened at 210 µm. The fine screen undersize is recycled to the column flotation circuit. The screen oversize, with a grade of approximately 4.1% Li_2O, is stored into the concentrate holding tank in the concentrate dewatering circuit.

The coarse hydroflotation underflow is sent to tailings, while the ultracoarse hydroflotation underflow is screened at 600 µm. The oversize is sent to the ball mill to liberate any finer spodumene particles while the undersize is sent to tailings.

j) Spodumene Column Flotation

The conditioning step of the column flotation circuit is adjusted for the fine particles, which requires a different collector dosage than the hydroflotation conditioning stage.

The spodumene column flotation circuit consists of three (3) flotation columns: a rougher column, a scavenger column, and a cleaner column. The conditioned feed is sent to the rougher column. The rougher concentrate is pumped to the cleaner column while its tailings are sent to the scavenger column. The scavenger column concentrate is recirculated to the pump box of the 3rd desliming cyclone cluster while its tailings are sent to tailings. The cleaner column concentrate is a final concentrate with a grade of approximately 5.1% Li_2O. The cleaner tail is recycled back to the pump box of the 3rd desliming cyclone cluster.

k) Spodumene Concentrate Dewatering and Storage

The hydroflotation concentrate screen oversize is sent directly to the concentrate holding tank. The column flotation concentrate is thickened to 62% solids by weight in a high-capacity thickener before being transferred to the concentrate holding tank. The combined flotation concentrate is then filtered to ~8% moisture using a vacuum belt filter. The filtered concentrate is discharged onto the combined concentrate conveyor where it is combined with the dried DMS concentrate. The blend of filtered and DMS concentrate is conveyed to the concentrate stockpile with a capacity of approximately 41 hours.

The combined DMS and flotation concentrate is expected to have 4 % moisture. The concentrate stockpile is within a covered dome. The combined concentrate is rehandled via front-end loader and transferred into containers on trucks for shipment offsite.

l) Tailings Dewatering and Storage

The coarse tailings sources are all screened and discharged onto the final tailings conveyor. Fine tailings sources are sent to an inclined plate settler where they are thickened to 61% solids by weight. The thickened tailings will be filtered to 15% moisture using a vacuum belt filter. The filtered tailings are discharged onto the final tailings conveyor. The combined tailings, containing 12% moisture, are transported by conveyor to a tailings loadout system which consists of a bin and emergency stockpile.

The tailings are either discharged directly into mine trucks or rehandled via front-end loaders. Mine haul trucks will transport the tailings to an on-site tailings CSF.

14.1.4 Whabouchi Processing Concentrator - Equipment Sizing and Selection

The equipment selection was based on the fulfillment of the design criteria during the 2018-2019 design, procurement, and construction phases prior to the change in ownership. Based on that design, an equipment list was prepared and the equipment was sized according to the developed design criteria, the flow sheet drawings and the mass balance. The design factor for crushing equipment was set at 30%, while the concentrator mostly uses a 15% design factor depending on equipment type and slurry pumps use 5%. The bulk of the equipment was procured and partially installed in 2018-2019 into the concentrator building which had been erected in 2016.

In 2021, under new ownership, a detailed evaluation was performed to mitigate the process risks associated with flash calcining leading to the decision to revert to a rotary kiln for spodumene calcination. This decision allowed for a reduced concentrate specification with increased lithium recovery. Based on additional metallurgical testing and a revised metallurgical balance, the plant design was modified and updated for a target spodumene concentrate of 5.5 % Li_2O.

Along with the change in product specification and flowsheet modifications, the plant throughput was increased compared to the 2019 design. Following the modifications, an exercise was carried out to verify the equipment sizing based on the new design parameters. Recommendations were made to address any major deficiencies; however, as a result of the changes, the design factors across the process plant was no longer systematic as described above. Some equipment have a large capacity to absorb process swings, while others have a decreased design factor. As with any plant, the risk of unforeseen bottlenecks exists during the ramp-up phase of the project; however, due to the modifications made, there is an increased risk of bottlenecking on equipment with low design factors which may limit overall production.

The sections below describe the equipment selection for the concentrator based on the updated concentrator flowsheet.

14.1.4.1 Primary Crushing

Primary crushing takes place in two (2) separate crushers. This circuit main equipment consists of two (2) jaw crushers and a triple deck vibrating screen. A primary jaw crusher is the most appropriate crusher for this facility based on throughput rate and cost. The secondary jaw crusher was selected by the vendor.

The static grizzly, with bars spaced at 800 mm, is on top of a surge hopper with a capacity of 26 m^3. An apron feeder with a length of 4,300 mm and a width of 1,524 mm withdraws material from the hopper. The feeder controls the ore fed in one 1,000 mm × 1,300 mm – 160 kW jaw crusher.

The crushed ore is transported on two (2) conveyors to a triple deck vibrating screen. The first conveyor is 1066 mm wide while the second is 915 mm wide. The screen has a top deck with 80 mm openings, a middle deck with 35 mm and a bottom deck with 10 mm openings. The top deck oversize is transported via a 915 mm conveyor onto a conveyor going to the 8 m^3 secondary jaw crusher feed bin. The feed bin discharges, via a vibrating feeder 0.6 m wide × 3.6-m long into an 800 mm × 1150 mm – 132 kW jaw crusher.

The secondary jaw crusher discharges back onto the conveyor feeding the surge hopper of the triple deck screen. The middle and bottom deck oversize are transported to the ore sorting circuit. The screen bottom deck undersize goes to the fine crushing circuit via a 762 mm belt conveyor.

It is estimated that approximately 83% of the crusher feed material will be sent to the coarse or fine ore sorter, while 17% reports to the screen undersize.

The crusher sizing was based on crushing work index test work results and verified by in-house DRA experts. The crushing circuit arrangement, screen sizing, and crusher selection were based on producing a minus 80 mm product for ore sorting limitations and were proposed by a supplier and verified by DRA.

The crushing area is now partially erected. The crushing area electrical room is installed and powered. The jaw crusher structural steel is erected, with mechanical equipment installation in progress. The secondary and tertiary crushing building is erected, with mechanical equipment installation in progress.

14.1.4.2 Ore Sorting

The middle and bottom deck screen oversize consist of 83% of the original feed and is sent to ore sorting. The coarse ore screen middle deck screen oversize is -80+35 mm in size and is transported to the coarse ore sorter via an air-supported conveyor with 762 mm belt width. The bottom deck screen oversize is -35+10 mm and is transported to the fine ore sorter via an air-supported conveyor with 762 mm belt width.

The ore sorting rejection rate is estimated in the range of 10-15% but is designed to handle larger instantaneous rejection rates (~30%). The accepted material is transported to the fine crushing circuit via two (2) 915 mm accepts conveyors and a 762 mm transfer conveyor. The ore sorter sizing is based on the bench scale test work at suppliers' facilities.

In order to increase ore sorting efficiency and recovery, one (1) future scavenger ore sorter is expected to be installed. It is planned to have the scavenger ore sorter operational in year 5.

The ore sorting building has been erected, and the ore sorters are in place on secondary steel, with additional mechanical equipment installation in progress. Conveyor assembly has begun, erection will begin once conveyor segments are complete and lifting operations in the area have been done.

14.1.4.3 Secondary and Tertiary Crushing

A double deck screen combined with three (3) cone crushers have been selected for this section based on tonnage and the final ore crushed size required.

The crushed ore from the primary crushers and the ore sorting circuit is transported via conveyors to the secondary crushing circuit. The cone crushers screen feed conveyor is 915 mm wide, while all other downstream conveyors are 762 mm wide. The crushed ore will be classified on the crusher vibrating screen consisting of one (1) 2.4 m wide × 8.3 long double deck vibrating screen with top deck screen apertures of 20 mm and the bottom deck screen apertures of 9.0 mm.

The top deck oversize will be crushed in two (2) standard secondary cone crushers which have 132 kW drives producing crushed ore at a P_{80} of 21 mm. The discharge from these crushers is recycled back to the cone crushers vibrating screen.

The bottom deck oversize will be crushed in the tertiary short head cone crusher which has a 132 kW drive and produces crushed ore at a P_{80} of 12.7 mm. The discharge from this crusher is also recycled back to the cone crushers vibrating screen.

The cone crushers vibrating screen undersize will have a top size of 9 mm and a P_{80} of 5.8 mm. It will be transported by conveyor to the fine ore overflow hopper located in a dome.

The crusher sizing was based on crusher work index test work results and performed by in-house DRA's experts and the supplier. Screen sizing and crusher selection were based on producing a final crushed material at 100% passing 9 mm.

14.1.4.4 Fine Ore Screening

The design of fine ore overflow hopper will allow material to direct feed the concentrator, while any surplus overflows from the hopper and is transferred to the fine ore stockpile with 10,000 t capacity. The material is rehandled via front-end loader and is transferred to the re-feed end of the overflow hopper. The hopper and re-feed areas are located within a covered dome. The hopper design has not been finalized and final volume and design is to be confirmed in detailed engineering.

Material is withdrawn from the fine ore overflow hopper via a belt feeder equipped with a VFD. The belt feeder controls the feed rate to the plant. The material is transferred to the fine ore screen feed conveyor and feeds the double deck fine ore screen. The screen is 2400 mm wide × 6.1 m long and has a 2.7 mm slot top deck and 0.85 mm bottom deck. The feed chute to this screen will flood the material with water to melt any ice agglomerates. The fine ore screen undersize (-0.85 mm) bypasses the DMS system and is pumped to the fine muscovite removal circuit. The fine ore screen oversize (-9.0+0.85 mm) is pumped to the coarse muscovite removal circuit. Adequate top-size protection will be implemented in detailed engineering to protect the oversize pumps.

14.1.4.5 Coarse Muscovite Removal

The screen oversize is fed into a 1.83 m × 1.83 m hydraulic separator. The hydraulic separator overflow flows over a sieve bend with 0.8 mm openings to remove the excess water and is then screened by a single deck screen (1.22 m × 3.05 m) with 2.0 mm square openings in the panels.

The screen oversize is directed to tailings, while the undersize is mixed with the separator underflow and pumped to the DMS feed dewatering screen. The dewatering screen (1.83 m × 3.66 m) has 0.85 mm openings in the panels. The dewatering screen undersize is mainly water and flows back by gravity to the fine ore screen, while the screen oversize is stored into the DMS feed bin with a capacity of 220 tonnes. The material is withdrawn from the DMS feed bin via a belt feeder with a belt scale to accurately control the feed to the DMS circuit.

The hydraulic separator sizing was based on the bench scale test work at the supplier.

14.1.4.6 Dense Media Separation

The DMS system is based on a modular design with modifications to suit the current plant. It includes two (2) separation stages: the rougher stage and the cleaner stage. Each stage operation is described below. The design is based on DRA's expertise for similar application.

a) Rougher DMS

   The material has been screened prior to the DMS bin to remove any fines. The material is transferred to the mixing box where it is mixed with the circulating media and is pumped to the DMS cyclones. The rougher stage is composed of one (1) 660 mm diameter cyclones. The cyclone discharge to two (2) dense medium single deck screens which act both as drain and wash screens. The rougher circuit also includes a circulating media circuit, dilute media circuit, densifying cyclone, and a low intensity magnetic separator to recover ferrosilicon.

   This rougher stage will be operated at low-density with a cut point at a specific gravity of 2.7. The cyclone overflows are the rougher floats and are considered the DMS tailings. The tailings are drained, rinsed, and transferred to the final tailings conveyor. The cyclone underflows are the rougher concentrate and are drained, rinsed, then pumped to the DMS cleaner feed pump box.

b) Cleaner DMS

   The material is pumped to the mixing box where it is mixed with the circulating media and is pumped to the DMS cyclone. The cleaner stage is composed of one (1) 510 mm diameter cyclone. The cyclone discharges to two (2) dense medium single deck screens which act both as drain and wash screens.

The cleaner circuit also includes a circulating media circuit, dilute media circuit, densifying cyclone, and a low intensity magnetic separator to recover ferrosilicon.

This cleaner stage will be operated at a higher density with a cut point at a specific gravity of 2.96. The cyclone overflow is the cleaner floats and is considered the DMS middlings. The DMS middlings are conveyed to the DMS middlings storage bin after washing and screening. The cyclone underflow is the cleaner concentrate and, after washing and screening, is conveyed to the DMS concentrate dryer.

14.1.4.7 Grinding

The DMS middlings are withdrawn from the 120-tonne capacity DMS middlings storage bin and fed into a 2.4 m diameter by 3.2 m effective grinding length (EGL) 285 kW grate discharge ball mill. The ball mill discharge is pumped to the ball mill screen.

The ball mill screen is a single deck high frequency vibrating fine screen with panels at 850 µm apertures. These panels will be used to produce a product size of P80 of 640 µm. The oversize is returned the ball mill, while the screen undersize is pumped to fine ore screen undersize pump box.

The ball mill sizing is based on the bond ball mill work index tests and simulations. A variable speed drive was specified to avoid overgrinding and to be flexible with grinding capacity.

14.1.4.8 Fine Muscovite Removal

The fine ore screen undersize and ball mill ground product are pumped into a densifying cyclone. This cyclone will thicken the fine muscovite hydraulic separator feed.

The cyclone underflow, at 60% solids by weight, is transferred to a 1.83 m × 1.83 m hydraulic separator. The hydraulic separator overflow combines with the cyclone overflow and flows over a sieve bend with 0.25 mm openings to remove the excess water and is than screened by a single deck screen with 200 µm openings. The screen oversize is sent to tailings, while the undersize is mixed with the hydraulic separator underflow and the combined mixture is pumped to a de-sliming cyclone cluster.

The desliming cyclone cluster consists of thirty (30) 4-in. cyclones with nineteen (19) in operation and eleven (11) as standby. The desliming cyclone overflow is sent to tailings, while the underflow is pumped to wet magnetic separation.

The hydraulic separator sizing was based on the bench scale test work at the supplier. Cyclone clusters sizing was based on DRA experience and manufacturer recommendations using simulations, with the goal of producing the proper density for the hydraulic separator and optimal slimes removal.

14.1.4.9 Wet Magnetic Separation

The desliming cyclone cluster underflow will be pumped to an agitated buffer tank with an effective residence time of 6 hours. The discharge of this tank will then feed the wet magnetic separation circuit to remove iron oxides bearing minerals and any residual dense media or grinding media chips. This will be carried out in two (2) steps. The first step is performed with a low intensity magnetic separator (LIMS). The LIMS is a 1.2 m diameter × 1.5 m wide counter rotation single drum magnetic separator operating at 950 gauss. The second step will use a vertical Wet High Intensity Magnetic Separator (WHIMS) at 12,000 gauss to remove paramagnetic material.

The magnetics rejects from both LIMS and WHIMS are combined and sent to tailings. The WHIMS non-magnetic product is sent to the densifying cyclone cluster prior to attrition scrubbing. The densifying cyclone cluster consists of twelve (12) 4-in. cyclones with eight (8) in operation and four (4) as standby. This cluster will remove residual fines and increase percent solids prior to attrition scrubbing. The cyclone overflow will contain 2% w/w solids and the cyclone underflow will have 60% w/w solids.

The magnetic separators were sized using bench scale and pilot plant test work results. The cyclones sizing was based on DRA experience and manufacturer recommendations using simulations.

14.1.4.10 Attrition Scrubbing

Attrition scrubbing is used to clean the mineral surfaces from slimes or other fine particles that adhere to the larger particle surfaces. The densifying cyclone cluster underflow discharges into the attrition scrubber circuit with four (4) attrition cells of 3.5 m³ each, where dispersant and caustic soda is added to assist with the loosening of persistent sticky fines. The scrubber discharge feeds the flotation split-feed hydraulic classifier which is used to create a size separation ahead of flotation. The hydraulic classifier underflow (at 50% solids) feeds the hydroflotation split-feed vibrating screen in the hydroflotation area, while the overflow is pumped to the third desliming cyclone cluster ahead of the column flotation circuit.

This third desliming cyclone cluster removes loosened fines from the attrition scrubber stage and ensures thickening before high density conditioning for column flotation. The cyclone cluster consists of nineteen (19), 4-in. diameter, cyclones, twelve (12) operating and seven (7) as standby. The cyclone overflow will contain <1% w/w solids which will be sent to tailings. The cyclone underflow will have 57% w/w solids.

The attrition scrubber cells residence time was derived from pilot plant test work and the hydraulic separator sizing was recommended by suppliers based on throughput and cut point. The cyclones sizing was based on DRA experience and manufacturer recommendations using simulations.

14.1.4.11 Spodumene Hydroflotation

The operation uses coarse particle spodumene, performed using hydroflotation, and fine spodumene flotation, performed using columns.

The hydraulic classifier underflow is sent to the hydroflotation split-feed vibrating screen, which separates the material into an ultracoarse +500 μm and a coarse -500 μm stream. The screen oversize is sent directly to the ultracoarse conditioning tanks, while the undersize is first sent to a densifying cyclone prior to discharging into the coarse conditioning tanks.

High-density conditioning is a requirement for spodumene flotation to have good performance. The ultracoarse and coarse circuits each are conditioned in a set of four (4) high-density condition tanks. Sulfuric acid and spodumene collector are added to the first conditioning tanks. The sulfuric acid is required to bring the pH down to 8 for proper conditioning to be achieved. The ultracoarse and coarse conditioning tanks are each a single tank, 6 m³ and 13 m³, respectively, each with four (4) agitators and four (4) sections. This provides high density and high energy intensity conditioning with minimal bypass. Each conditioning tank section is equipped with a 22 kW agitator.

The conditioned slurry is pumped from the last conditioning tanks are pumped to the ultracoarse and coarse spodumene hydroflotation cells, respectively. The ultracoarse hydroflotation cell is 0.9 m diameter × 1.8 m high, while the coarse hydroflotation cell is 1.8 m diameter × 3.7 m high. Air, water and frother are injected near the bottom of the cells and cause the particle-bubble interaction events where spodumene adheres to air bubbles and float to the surface of the cell.

The coarse hydroflotation underflow is sent to tailings. The ultracoarse hydroflotation underflow is screened at 600 μm with the oversize being sent to the ball mill and the undersize sent to tailings.

The ultracoarse and coarse concentrates are sent to the hydroflotation screen which is a multiple stack high frequency vibrating screen with panels having 210 μm apertures. The oversize goes to the spodumene concentrate filter holding tank, while the screen undersize is sent to the third desliming cyclone cluster feed pump box. This way the material is introduced to column flotation.

The conditioning cells residence time was derived from pilot plant test work. The original hydroflotation cell sizing was provided by the supplier based on pilot scale test work. As part of the 2021-2022 modifications, the hydroflotation circuit was split into the ultracoarse and coarse circuits. The equipment sizing is based on the 2022 supplier test work and recommendations.

14.1.4.12 Spodumene Column Flotation

The third desliming cyclone cluster underflow slurry feeds the column flotation conditioning tank by gravity while the overflow is sent to tailings. Again, sulfuric acid and spodumene collector are added as spodumene flotation reagent. The sulfuric acid brings the pH down to a pH=8 for proper conditioning to be achieved. Frother is added to the last section. This is a four (4) section conditioning tank of 13 m^3 to provide a residence time for conditioning of the flotation feed. Each conditioning section is equipped with a 15 kW agitator.

The slurry is pumped from the last conditioning tank into the rougher flotation column. This flotation column is 2.5 m diameter × 8.0 m high. Air is injected near the bottom of the column, and this causes the spodumene to float. The column underflow is rougher tailings, while the column overflow is rougher concentrate.

The rougher concentrate feeds the cleaner flotation column. This cleaner flotation column is 2.5 m diameter × 8.0 m high. Air is injected near the bottom of the column. The column underflow is cleaner tailings, while the column overflow is final concentrate. The cleaner tail is recycled back to the third desliming cyclones cluster to be reintroduced to the rougher flotation column. The concentrate from the cleaner column flows by gravity to the concentrate thickener.

The rougher tailings feed the scavenger flotation column. This scavenger flotation column is 2.5 m diameter × 8.0 m high. Air is injected near the bottom of the column. The column underflow is scavenger tailings, while the column overflow is scavenger concentrate. The scavenger concentrate is sent to the 3rd desliming cyclone cluster to be reintroduced to the rougher column while the scavenger tailings is sent to tailings.

The conditioning cells residence time was derived from pilot plant test work. The flotation column sizing was based on the pilot scale test work at the supplier's laboratory.

14.1.4.13 DMS Concentrate Drying

The DMS concentrate will be conveyed to a 1.65 m diameter × 15.5 m long rotary dryer indirect electric dryer with a 2.2 MW heating capacity.

The dried concentrate is then transported to the dry magnetic separation circuit.

The dryer sizing was based on the bench scale test work at the supplier and calculations for the concentrate moisture and feed tonnage.

14.1.4.14 Dry Magnetic Separation

Based on test work results, dry magnetic separation was found to be efficient on coarse material separation.

The magnetic separation of dried DMS concentrate will be carried out in two (2) stages. The equipment has a 120ºC temperature maximum rating. The first stage is designed with a lower field intensity and is used as Low Intensity Magnetic Separator (LIMS) to remove residual dense media or any fine ferrous material. The second stage uses three (3) rare earth magnetic separator drums as High Intensity Magnetic Separator (HIMS) in parallel. The peak magnetic field intensity on the HIMS drums go up to 8,000 gauss and is effective in removing coarse paramagnetic constituents. The DMS concentrate is transferred to the combined concentrate conveyor, while the magnetic fractions are sent to tailings.

The dry magnetic separator sizing was based on the full-scale test work at the suppliers' facilities.

14.1.4.15 Concentrate Dewatering and Storage

The final column concentrate goes to the concentrate thickener, which is a 6.0 m diameter high-rate thickener. The thickener overflow is pumped to the process water tank for recirculation of process water, while the concentrate thickener underflow at 62% solids will be pumped to spodumene the concentrate holding tank (5.5 m diameter × 8.4 m high). The solids will be kept in suspension with a 90 kW agitator.

From the holding tank, the concentrate is pumped to the concentrate filter. The vacuum belt filter was sized based on filtration tests. The filter has a filter area of 6 m². The filtrate will be re-circulated to the spodumene concentrate thickener via a filtrate pump.

The filter cake at about 8% moisture is combined with the dried DMS concentrate and conveyed to the concentrate stockpile. The combined DMS and flotation concentrate is expected to have 4.3 % moisture. The concentrate stockpile is within a covered dome. The combined concentrate is rehandled via front-end loader and transferred into containers on trucks for shipment offsite.

The high rate concentrate thickener was sized based on sedimentation test work conducted at SGS. The filter was sized by filtration test work performed on concentrate made from the pilot plant testing.

14.1.4.16 Tailings Dewatering and Storage

Various streams from the concentrator, including overflow from desliming and densifying cyclone clusters, magnetics from LIMS and WHIMS and tailings from column flotation circuit and hydroflotation circuit are pumped through a single (1) 420 mm dewatering cyclone. The cyclone underflow is screened on the tailings dewatering screen. The screen undersize is pumped with the cyclone overflow to the 10.5 m diameter × 12.6 m high inclined plate settler or thickener.

The tailings screen is a standard dewatering where screen oversize is discharged on the final tailings conveyor.

The thickener overflows into the process water tank, while the tailings thickener underflow at 61% solids is pumped to the tailings holding tank.

The tailings holding tank has dimensions of 5.6 m diameter × 7.2 m high.

From the holding tank, the tailings are pumped to the tailings vacuum belt filter. The belt filter, with 12 m² of filtering surface, will produce a filter cake with a moisture content of 15%. The filtrate will be re-circulated to the tailings thickener by a filtrate pump. The filter cake is combined with the other tailings on the final tailings conveyor and conveyed to a tailings loadout system which consists of a bin and emergency stockpile.

The tailings are either discharged directly into mine trucks or rehandled via front-end loaders.

The inclined plate settler was sized based on bench scale sedimentation test work and hydraulic loading capacity with recommendation from the manufacturer.

Trucks will be used to transport the tailings to the on-site CSF.

14.1.5 Whabouchi Processing Concentrator – Utilities

14.1.5.1 Concentrator Water Services

The water consumption is based on the concentrator nominal water consumption per hour.

a) Fire Water

Water collection pond BC-11 and freshwater wells will be the main water sources for fresh/fire water make-up. The water will be pumped to an 8 m diameter × 11 m high combined fresh water / fire water tank.

b) Process Water

Process Water will be recycled back, at a nominal rate of about 1093 m^3/h, from the tailings and concentrate thickeners. The process water tank is 10 m diameter × 12.8 m high. Any make-up water will be provided from water collection pond BC-11.

c) Gland Water

The gland water system has a separate 30 m^3 gland seal water tank. Process water is deslimed with a hydrocyclone cluster and passed through a water filtration system to produce 43.8 m^3/h of gland water. This cyclone and filtration system cleans the process water to be sufficient for gland seal service.

d) Potable Water

Potable water will be used at a rate of 2.0 m^3/h. This water is treated in the site potable water treatment system located near the camp prior to entering the concentrator. The potable water system will supply water to safety shower, concentrator offices and dry, mine dry and laboratory.

The fresh / fire water tank and process water tank have both been erected. The fresh / fire water tank is in service, currently providing the site with fresh water.

14.1.5.2 Concentrator Compressed Air (High Pressure)

One (1) air compressor complete with air dryer is dedicated to the ore sorters and is slightly oversized to provide compressed air in other areas of the crushing circuit. This will be used for dust collection pulsation and other small requirements.

The concentrator is equipped with three (3) air compressors. One (1) compressor is dedicated for the flotation air and will have a separate compressed air network to maintain a constant pressure.

The remaining two (2) are dedicated to plant/instrument air (one operational, one standby). An air dryer and accumulator tank will be used for instrument air only.

15 INFRASTRUCTURE

The mine and the concentrator infrastructure are located at Whabouchi and are described in Section 15.1.

15.1 Whabouchi Mine Infrastructure

The Whabouchi Mine Site has been under construction since 2017 and has a number of partially completed facilities prior to Project closure in late 2019. The mine garage building, concentrator building, ore sorter building, concentrate storage dome, fine ore dome, laboratories, the main electrical and crusher E-Rooms, the administration building, and the gate house were enclosed with some equipment and services installed.

Other supporting facilities and infrastructure such as the potable water treatment plant, sewage treatment plant, roads, propane infrastructure, various roads were established to support legacy execution activities.

The main substation and emergency power generator, the fire water tank, and the raw water supply, were completed and were supplying power and water for the construction activities.

Some of the above-mentioned infrastructure will require to be completed, modified, or replaced. This is described in the following sub-sections.

15.1.1.1 General Site Plan

This section describes the main Project elements related to process, followed by supporting infrastructure. Two (2) different scale plans were produced for the Project. Drawing. W000-EN-DWL-DRA1-0001 (Figure 15-1) illustrates the overall general site plan and Drawing No. W000-EN-DWL-DRA1-0002 (Figure 15-2) shows the Whabouchi Village area and related infrastructure.

The drawings contain the topographic information from LiDAR laser mapping technology provided for the Project by NLI.

The site is accessible via the Route du Nord through a fully manned gate house which leads to the Project areas, the administration facilities and the temporary camp which will continue to be used during the construction of the Project.

All mining activities will be concentrated in a 75 ha. area; delimited by the UTM 5,728,000 mN, NAD83 Zone 18 to the north, the UTM 5,724,500 mN, NAD83 Zone 18 to the south, the Mountain Lake to the west and the Spodumene Lake to the east.

Figure 15-1 General Site Plan

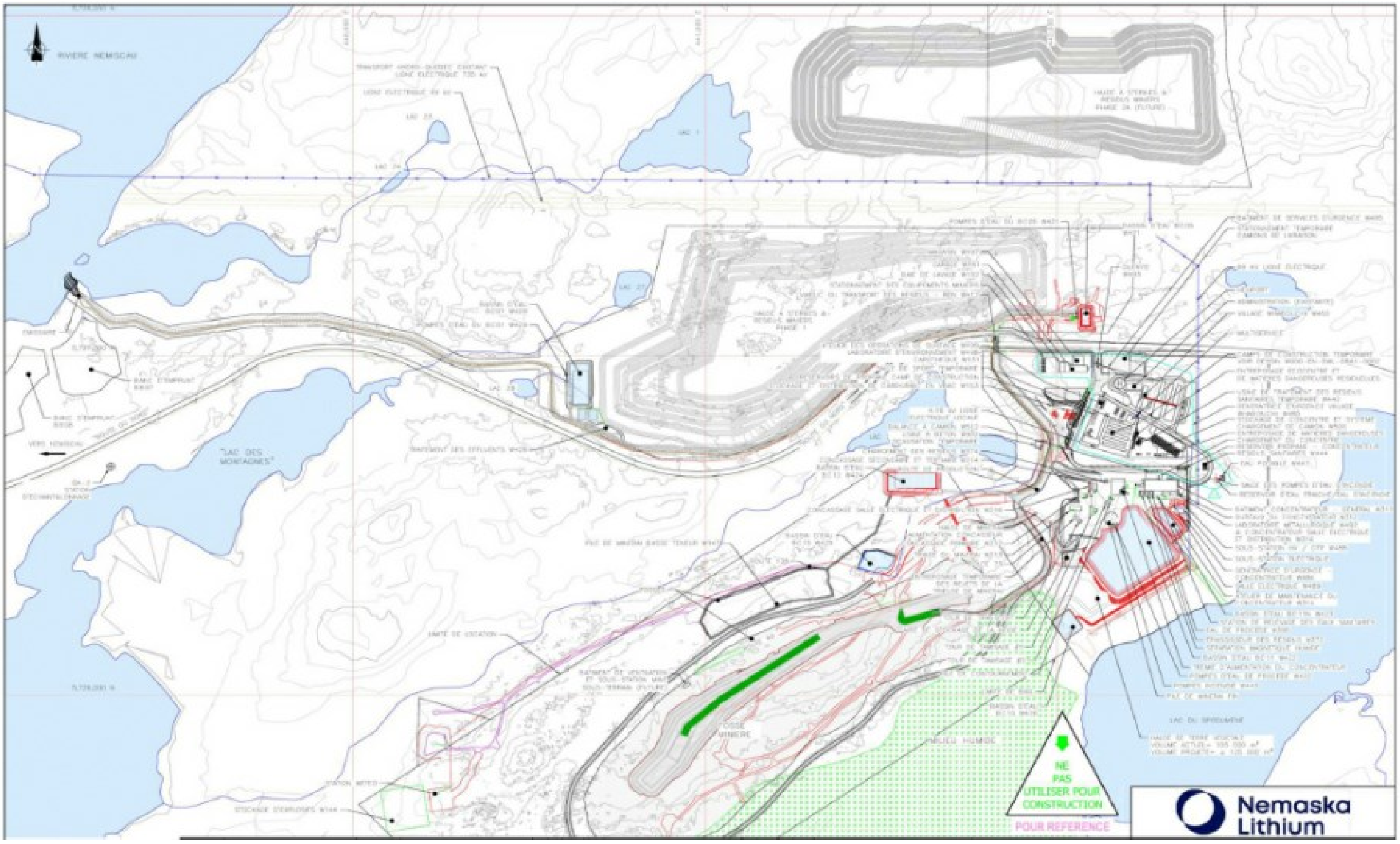
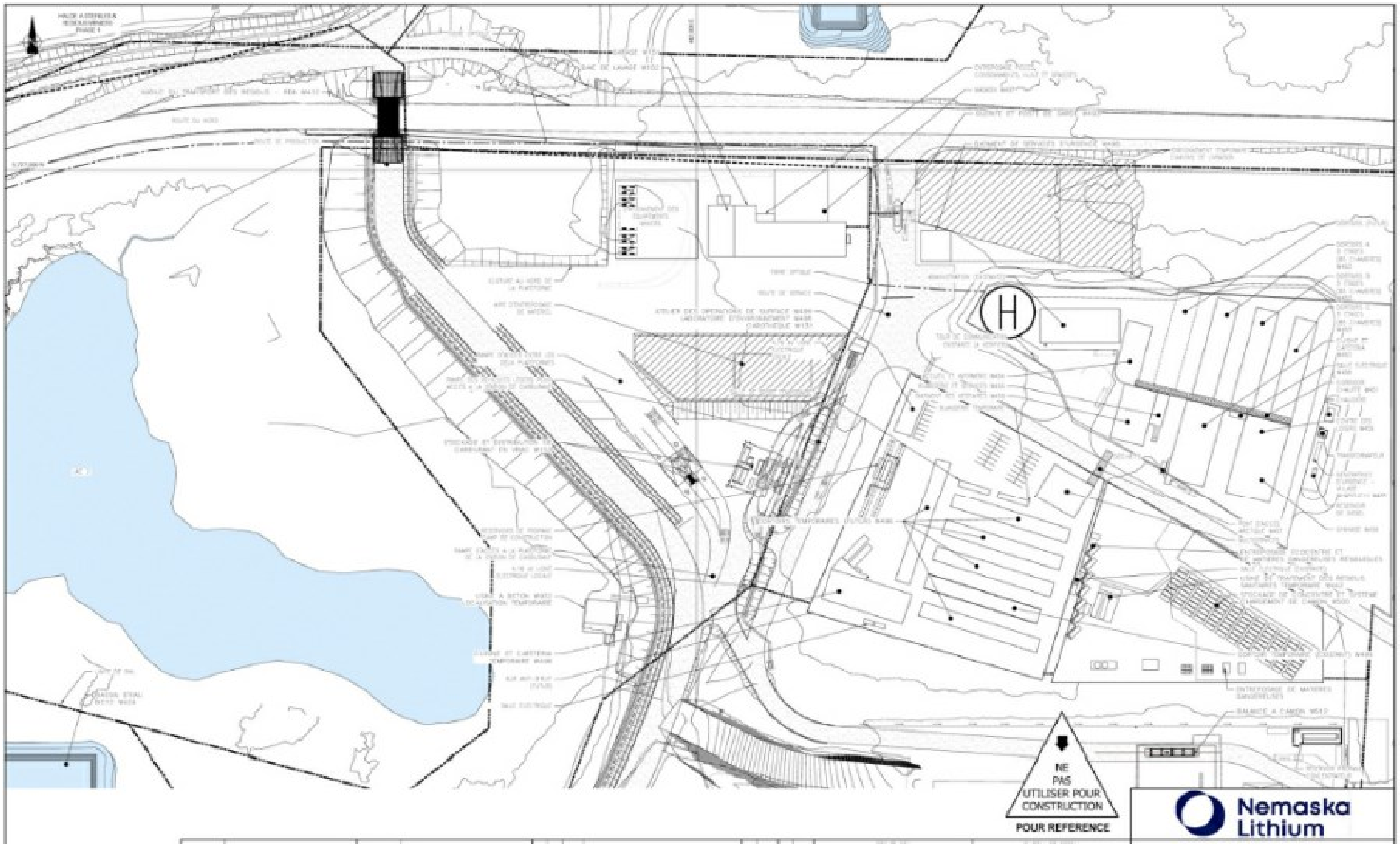


Figure 15-2 General Site Plan – Whabouchi Village Area View



15.1.2 General Plan

15.1.2.1 Geotechnical

Geotechnical investigations have been performed by:

- Journeaux Assoc. (Geotechnical Investigation Whabouchi Mine Nemaska Lithium, Quebec, Report # L-11-1452, issued December, 2011);
- Journeaux Assoc. (Geotechnical Investigation Whabouchi Mine Nemaska Lithium, Quebec, Report # L-16-1920, issued September 26, 2016);
- GHD (Campagne d'investigation géotechnique complémentaire. Bassins de collecte des eaux, Report # 11132417-A2-RPT-3-FINAL, Rev. 3, issue janvier 31, 2019);
- Journeaux Assoc. (Geotechnical Investigation Whabouchi Mine Nemaska Lithium, Quebec, Report # L-22-2413, Rev. 0, issued November 03, 2022).

The purpose of those reports is to characterise the ground condition for foundation design or for any borrow materials for the Whabouchi Infrastructure location presented in this Section. Based on the available Geotechnical information as well as Journeaux Assoc 's Recommendations, DRA was able to develop preliminary designs and Material Take-Offs (MTOs).

15.1.2.2 Route du Nord

The Route du Nord is located north of the open pit and north of most of the Project infrastructure, except for the tailings and waste rock CSF which are located on the north side of the Route du Nord. The Project is located at km 276 Route du Nord. The portion of the road on which the Project is located is administered by Société de développement de la Baie-James (SDBJ) as part of its strategic assets in the area. The Route du Nord provides site access from the west, via the Route Billy-Diamond (formerly Route de la Baie-James) and the town of Matagami, and from the south-east via the town of Chibougamau.

A part of the Route du Nord will be modified involving around 300 m of the road which will be paved to avoid visibility problems due to dust created by traffic on this area. Most of this length will be widened from the existing width to have four (4) lanes to have fluid traffic in case there is truck transportation at the same time as normal traffic in the Route du Nord.

15.1.2.3 Main Access Road to Concentrator

The main access road to the concentrator was designed using the general site topography and attention was given to grades and turning radius for long B-Train concentrate hauling trucks to be able to circulate around the concentrator building and the concentrate loadout system for loading the concentrate transport trucks.

The road is constructed with cut and fill material completed with a base of 300 mm natural granular fill MG112 (pit run 0-4 in) topped with 150 mm of MG20b crushed stone (0-3/4 in).

The road width is 8.5 m and a 2-m additional rolling width is allowed on the inside of each road bend for the concentrate trucks. The road is one-way, clockwise, to allow concentrate trucks to access the Overpass and to assure safety of users especially in winter.

In addition, the main access road and mining hauling roads are kept separate most of the time to avoid having large mining vehicles on the same road as the concentrate, delivery, and services vehicles. Only when the concentrate truck takes the ramp from the concentrator road through the Fuel Station area towards the Overpass is where mine trucks and concentrate traffic would need traffic control for safety reasons.

An overpass over the Route du Nord will be added to allow the mining trucks unimpeded access to the CSF and to avoid traffic with the Route du Nord.

Mine Service and Haul Roads

The mine haul roads were designed for 64-tonne class haul trucks with a 121,000-kg target operating weight. The quantities required for building the roads, were established based on draft profiles for each section of road. The profiles were determined based on the roads with a maximum slope of eight percent (8%). To suit the truck dimensions, the mining roads were designed to be 17 meters wide.

The quantities of fill required to construct the service roads were established using a standard AASHO-H20 highway live load. All service roads were designed to be ten (10) meters wide.

15.1.2.4 Gatehouse

The Whabouchi Village's guests will enter the Whabouchi site at the Gatehouse. The existing gatehouse and associated septic system will be demolished and replaced by a new building which will be connected to the main sewage water treatment plant via underground piping. The Gatehouse will be located near the Route du Nord and will be used to control all access to the site. A gate will be continuously operated by a security guard.

There will be a parking area outside the Gatehouse for trucks awaiting entry to the site.

15.1.2.5 Truck Scale

A truck scale will be used to track loads leaving the site with concentrate shipments. The truck scale will be located within the concentrate storage dome next to the concentrator.

15.1.3 Construction Camp

A temporary camp is located north of the ROM pad. It currently houses the visitors and the personnel in charge of care and maintenance of the Whabouchi site. It is operated and maintained by a local contractor, including all the required services. The camp is comprised of dormitories and a kitchen and dining area. The fuel for the kitchen appliances and the camp's heating requirements is provided by nearby propane tanks. These tanks are currently leased from a local propane distributor.

The existing temporary camp will be upgraded and used for the early works until the new Whabouchi Village dormitories are commissioned. Five (5) additional temporary dormitories, a small gymnasium and laundry facility will be added for the construction phase, to bring the temporary camp capacity up to approximately 225 people. These new facilities will consist of used refurbished camp to be purchased and installed by NLI. The temporary camp will be located at the same location as the camp that was used for construction during the previous phase of the project and has been decommissioned since.

15.1.4 Whabouchi Village

The new Whabouchi Village will be located to the north of the Whabouchi processing area, just south of the Route du Nord. The Whabouchi Village will comprise a new welcome center, administration center, mine dry building, recreation facilities, kitchen and dining area and accommodations for 255 persons. All facilities will be linked by heated corridors.

15.1.4.1 Welcome and Medical Centre

New guests will arrive at the Welcome Centre where they will be assigned a room and where their luggage will be inspected for unauthorised items. The Medical Centre will be conveniently located within the Welcome Centre to assist the newcomers for medical screening, such as COVID-19 screening, and similar activities. Ambulance parking and shelter will be accessible via the Medical Centre.

A loading dock will be included for receiving luggage, near the luggage inspection area. A lockable sliding door will be provided between the Welcome Centre and the Heated Corridor.

15.1.4.2 Administration Building

The existing Administration Building will be demolished. Administration will take place in the new multi-function building which will house offices and conference rooms.

15.1.4.3 Mine Dry

The presence of crystalline silica at the Whabouchi site poses serious health risks and proper measures will be taken to ensure the safety of the Whabouchi Village guests. As such, once the guests have been registered, their only way into the Whabouchi Village will be via the Mine Dry. This is meant to limit the ingress of crystalline silica dust into the facilities.

The mine and processing plant workers will transit via the mine dry whenever entering and exiting the Whabouchi Village. From the outside, the workers will enter the mine dry in their work clothes and work boots. The workers will then proceed to the dirty change room where they will take off their work clothes and work boots before proceeding to the clean changing room where they will change into their casual clothes.

The mine dry will be divided into two (2) completely separate sections: one for women, the other for men.

15.1.4.4 Recreational Centre

The Recreation Centre will be south of the Heated Corridor. The facility will include an indoor court, weights and cardio area, an aerobic and yoga area, a recreation room, a cinema room, and a television and gaming lounge.

15.1.4.5 Kitchen and Dining

The Kitchen and Dining will be located to the north of the Heated Corridor and will be designed to accommodate 180 people at a time.

The supplies for the kitchen will be delivered by 53’ semi-trailers. As such, in addition to the kitchen area and cafeteria, the Kitchen and Dining facility will be equipped with a loading dock to accommodate the deliveries. Sufficient space will be allocated to unload and store up to 26 pallets (48” x 40”) of supplies to make truck deliveries as efficient as possible.

Kitchen waste will be stored outside in a covered container.

15.1.4.6 Dormitories

The dormitories will be located on the east side of the Whabouchi Village. Three (3) separate three (3) story dormitories will house a total of 250 rooms. There will be provision for potential future dormitories which could provide an additional 87 rooms.

Each room will be equipped with a private bathroom complete with toilet and shower. Laundry rooms will be included at each level of the dormitory to allow guests to do their laundry.

Dormitory rooms will be air conditioned with Packaged Terminal Air Conditioners (PTAC) heat pump units in addition to glycol baseboard heaters hooked up to a dual energy diesel-electric boiler.

Each dormitory wing will include a mechanical room on each floor. The mechanical rooms located on the top floor will be equipped with a ladder for roof access.

All rooms will be equipped with an electronic key card access system.

15.1.4.7 Laundry and Services

The Laundry and Services facility will be located to the west of the Village, adjacent to the Mine Dry. The layout will allow for the possibility of delivering new washing machines with a pallet jack.

15.1.4.8 Covered Walkways

There will be two (2) unheated covered pedestrian walkways that will connect the Whabouchi Village to the Mine Garage and to the Concentrator. They will provide shelter from the wind, rain, and snow to the workers walking between the various facilities, and physical protection from vehicles and construction equipment. Exits and entrances located at road crossings will be equipped with safety gates complete with automatic flashing light to signal to the vehicles that pedestrians are crossing.

15.1.5 Mine Support Facilities

15.1.5.1 Mine Garage / Excavator Service Bay Annex

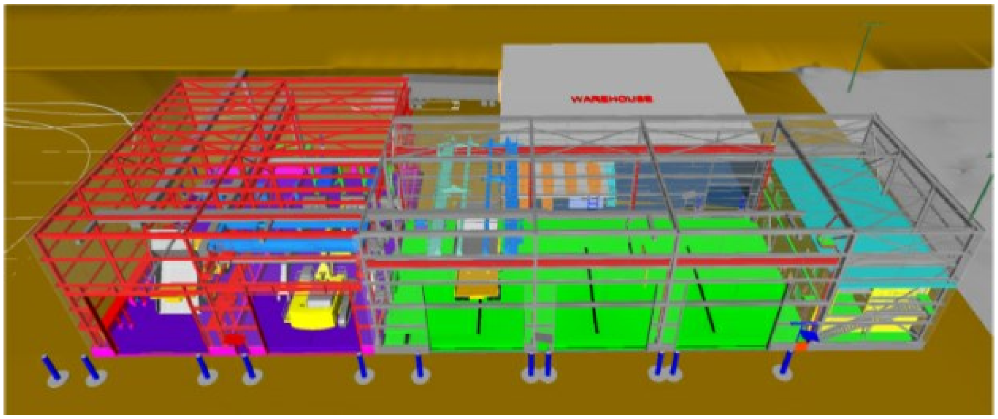
The mine garage building is located west of the Whabouchi Village. The mine garage will be used for servicing the mine equipment in addition to other mobile equipment. It will house one (1) excavator service bay, three (3) maintenance bays for the mine trucks and other large mobile equipment and two (2) light-vehicle maintenance bays. The mine garage will be equipped with three (3) overhead cranes (one (1) 20,000 kg for the excavator maintenance and two (2) 10,000 kg independent overhead cranes which can be operated together with a spreader beam for Haul Truck Dump Body changes (up to 20,000 kg lift)), tools and air compressor.

The maintenance garage has been erected with three (3) bays only, and interior finishing remains to be completed and is shown in Figure 15-3.

Figure 15-3 Maintenance Garage

The existing building will be augmented by the addition of a bay for longer mobile equipment and the addition of a wash bay, both of which will be added to the east as illustrated in Figure 15-4.

Figure 15-4 3D Model of the Mine Garage/Wash Bay



The wash bay will be constructed adjacent to the mine garage for the cleaning and de-icing of the equipment prior to servicing. The wash bay will be equipped with a high-pressure wash system with water recycling and an oil and water separator.

15.1.5.2 Bulk Fuel Storage And Distribution Facility

The fuel storage facility will be built on a lined pad. The system will include the following:

- Three (3) 50,000 litre capacity double wall diesel fuel tanks;
- One (1) 20,000 litre capacity double wall gasoline fuel tank;
- One (1) 50,000 litre capacity double wall Diesel Exhaust Fluid (DEF) tank;
- One (1) station for fueling heavy vehicles;
- One (1) station for fueling light vehicles;
- One (1) station for fuel delivery combined with the refueling of a fuel truck that will deliver fuel to slow speed equipment in the mine pit.

A complete Command Centre for the transfer pump system and distribution station, will be installed in a modified storage container.

The facility will include a fuel accounting system to keep track of fuel distribution by equipment and operator to facilitate cost accounting and reporting, and to prevent fuel theft. The fueling station will be completely unmanned and automated such that equipment operators will fuel their own mobile equipment.

Exterior lighting will be provided for safety and security. A containment trench will be constructed to collect any spillage that might occur.

15.1.6 Non-Process Buildings Terrace

There are a number of support facilities required to service the Whabouchi operations and located on the non-process building terrace, south of the Village and north of the processing area. These facilities range from warehouses, core shack and others as enumerated below.

15.1.6.1 Warehouse

The warehouse will store spare parts, wear parts, spare equipment needed to perform day to day tasks all around the mine and concentrator. The building will have racking arranged to optimize storage space, secure storage room and office. There will be exterior laydown space for certain reagents, parts and equipment not affected by weather conditions.

15.1.6.2 Emergency Services Building

The emergency services building will house fire truck and fire fighting equipment.

15.1.6.3 Environmental Laboratory

The environmental laboratory will be located near the Emergency Services Building in a separate building.

15.1.6.4 Surface Operations Workshop

The Surface Operations Workshop will be used by site maintenance personnel such as plumbers and carpenters.

15.1.6.5 Core Shack

A new Core Shack will be provided to replace the existing core storage racks. The new Core Shack will contain the core cutting equipment. New core storage shelves will be located adjacent to the Core Shack. The existing Core Shack will be dismantled and discarded.

15.1.6.6 Industrial Waste Management

A waste storage facility will be built to ensure the proper waste storage until their collection by outside contractors.

Seven (7) open top 20’ containers will be provided to store the following:

- Industrial waste;
- Domestic waste;
- Hazardous waste;
- Metal;
- Wood;
- Paper;
- Contaminated soil

A small building will also be provided nearby for administration. The containers will be picked up on a regular basis and trucked to regulated off-site disposal areas. New containers will be put in place during the pick-up.

Hazardous waste will be managed by outside authorized contractors. Hazardous material will be collected in specific and clearly identified locations. The selected contractor will be responsible for gathering hazardous materials on-site, proper identification, safe transport, and safe shipment of material directly to specialized disposal areas, when required.

15.1.7 Service Facilities

The Whabouchi site is serviced by potable water, sewage, fire protection and detection, and maintenance facilities.

15.1.7.1 Effluent Treatment Plant

The Whabouchi site water will collect in various collection ponds and the excess water will be pumped to the Effluent Treatment Plant to be treated prior to being pumped into the Nemiscau river. The Effluent Treatment Plant building will house water treatment process equipment and reagent plants. The reagents will be delivered to the Effluent Treatment Plant via the access road south of the CSF.

15.1.7.2 Potable Water Treatment Plant

The existing potable water treatment plant will be decommissioned and relocated in the vicinity of the existing Fire Water Tank.

15.1.7.3 Temporary Sewage Water Treatment Plant

The existing Sewage Water Treatment Plant will be used for treatment of the sewage from the Construction Camp. It will run concurrently with the new permanent Sewage Treatment Plant for the first few years of operations before being decommissioned.

15.1.7.4 Sewage Water Treatment Plant

A new Sewage Water Treatment Plant will be installed north-east of the Concentrator and will be sized for up to 350 people. It will treat the sewage water for the whole Whabouchi site. Sanitary and shower wastewater will be collected from each building via underground piping and discharged into the Sewage Water Treatment Plant.

15.1.7.5 Fire Protection

An existing fire water pumping station is located at the site with some existing fire water piping and hydrants to some of the existing facilities.

The existing diesel-powered fire water pump and fire water tank are undersized for the new requirements of the Whabouchi site. The Fire Water Tank will remain in operation to feed the Jockey Pump but the diesel fire water pump will be decommissioned. A picture of a fire pumping module is provided in Figure 15-5.

Two (2) new vertical turbine fire water pumps will be installed in pond BC-11 to service the fire water ring main. One of the pumps will be electric powered and the other will be diesel powered.

The new fire water design provides for a site-wide fire protection system, including all electrical rooms and other high-risk areas. The fire water will be distributed through an independent underground and heat traced piping system.

Figure 15-5 Fire Water Pumping Module

15.1.7.6 Fire Detection

The fire alarm system will consist of multiple panels located within key facilities (such as gatehouse, emergency response centre, administration building) with notification appliances, detectors and manual pull stations installed to cover all process and non-process facilities at the Whabouchi site. There will be designated emergency muster stations in the event of a fire alarm, along with evacuation routes and procedures. Fire extinguishers will be provided sitewide as required in areas such as offices, laboratory, warehouse, lunchrooms, and fuel stations.

15.1.7.7 Metallurgical Laboratory

The existing Metallurgical Laboratory is located to the south-east of the concentrator building and is made of modular buildings.

15.1.8 Heating, Ventilation and Air Conditioning

Heating, Ventilation and Air Conditioning (HVAC) will be provided in the various facilities of the Whabouchi Site. Most of the HVAC equipment remains to be purchased and installed with the exception of the direct fired makeup air units at the concentrator and some propane forced air heaters.

The Concentrator Building's HVAC system will consist of air extractors, a combination of electric and propane fired make-up air units, and of electric and propane forced air heaters. The offices will be air conditioned and will be heated with baseboard heaters.

The Ore Sorter Building will be equipped with an electric makeup air unit and electric forced air heaters.

No HVAC equipment will be installed at the Cone Crushers Building with the exception of louvers. The hydraulic power units room will be equipped with a pressurization HVAC unit and electric forced air heaters.

The Mine Garage and Wash bay will be equipped with an electric makeup air unit complete with recovery cube and peripheric forced air heaters. A piping loop will be included in the slab on grade to allow for a potential future radiant heating system. An exhaust fumes system will be provided.

The buildings of the Whabouchi Village will be heated by a combination of electric HVAC equipment such as heat pumps, makeup air units, baseboard heaters and Packaged Terminal Air Conditioners (PTAC) heat pump units.

An insulated heat traced piping loop will be provided between the Concentrator and Whabouchi Village to allow for a potential future District Heating system.

15.1.9 Co-Disposal Storage Facility

The planned Co-disposal Facility (CSF) is located to the north of the Route du Nord. The CSF is positioned at a minimum distance of 30 m from the road centre line and 60 m from the surrounding lakes and streams. A right-of-way of 80 m was considered for the Hydro-Québec power lines (735 kV existing high-voltage power lines). Figure 15-6 illustrates the Phase 1 and proposed extension CSF locations shows the location on the mine site plan.

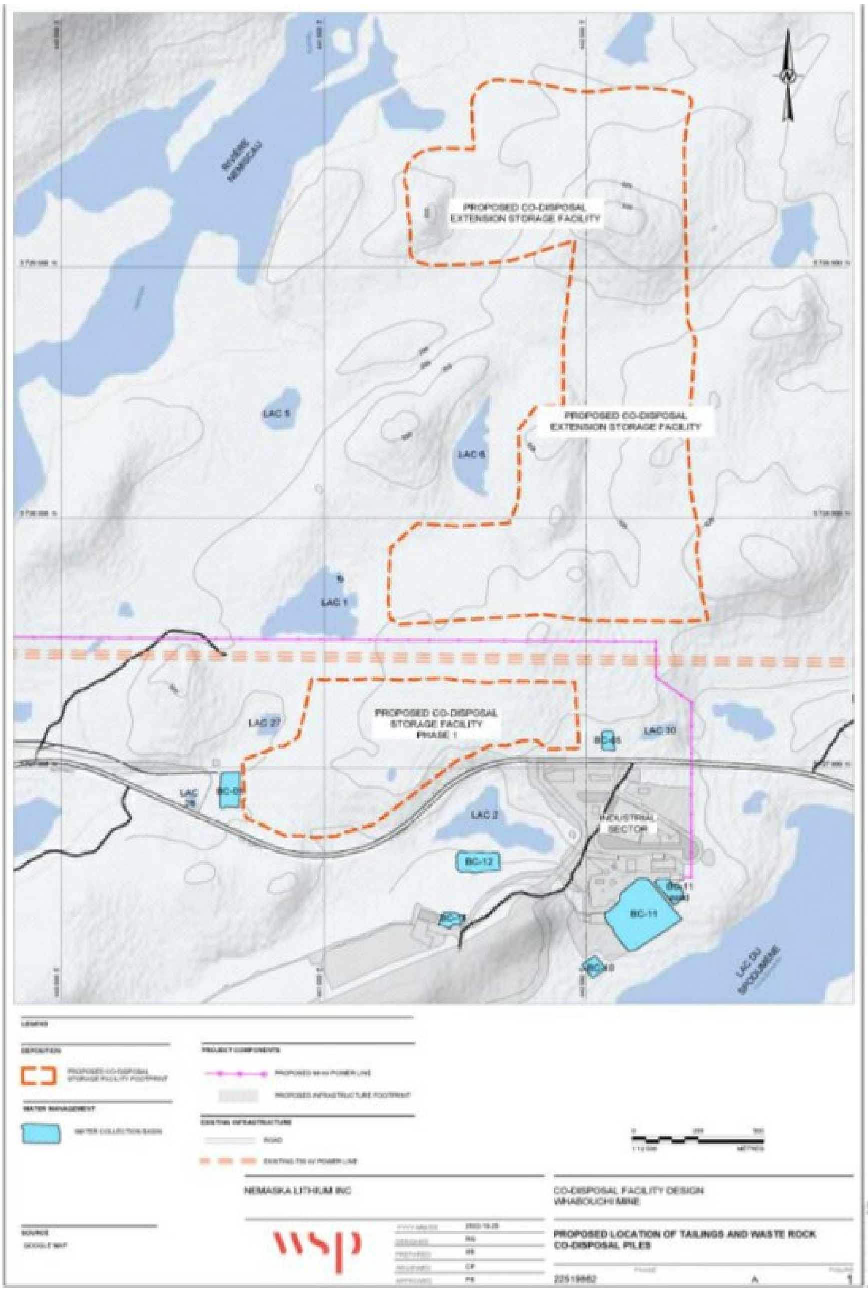
With open pit and underground mining, the Project life is 34 years, generating 52.8 million cubic metres (Mm^3) of material managed within the CSF. Table 15-1 shows the details of waste rock and tailings production for both mining methods.

Table 15-1 Summary of Tailings and Waste Rock Production

	Duration (year)[(a)]	Waste Rock (Mm^3)	Filtered Tailings (Mm^3)	Ore Sorter Rejects (Mm^3)	Total (Mm^3)
Open pit	24	33.9	11.3	1.7	46.9
Underground	10	0.0	5.8	0.1	5.9

(a) For two years (Year 23 to 24), underground and open pit mines are exploited simultaneously.
Mm^3 = million cubic metres.

Figure 15-6 Co-Disposal Storage Facility Location



All phases of co-deposition of waste rock and filtered tailings are planned to meet the anticipated duration of the project. All waste rock filtered tailings and ore sorter rejects would be contained in the co-disposition piles. Table 15-2 summarizes the total mining waste to manage, and the associated capacity of the CSF.

Table 15-2 CSF Capacity

Parameters	Value		
	Phase 1	Other Phases	Total
Co-disposal pile volume	13.1 Mm^3	39.7 Mm^3	52.8 Mm^3
Co-disposal pile tonnage	25.9 Mt	76.1 Mt	102.0 Mt
Waste rock volume	9.7 Mm^3	24.2 Mm^3	33.9 Mm^3
Tailings volume	2.9 Mm^3	14.3 Mm^3	17.2 Mm^3
Ore sorter rejects volume	0.5 Mm^3	1.3 Mm^3	1.8 Mm^3

Mm^3 = million cubic metres
Mt = million tonnes

Soil parameters from various geotechnical investigation data (GHD 2018; Journeaux Assoc. 2011; SNC 2016; Richelieu 2019; WESA Envir-Eau 2012) and surface deposit maps were used in the design. The geotechnical parameters used for the mining waste rock design are:

- waste rock dry density: 2.11 t/m³ (SNC 2019);
- filtered tailings dry density: 1.60 t/m³ (WSP Golder 2022);
- ore sorter rejects density: 1.80t/m³.

Future phases (Figure 15-6) are planned but would need to be redefined to confirm total capacity and water management infrastructure to secure the necessary environmental permits prior to use (Section 17). Approximately 6 Mm³ of waste rock can be disposed in the pit; thereby, reducing operating costs, decreasing transportation distances for waste rock, and reducing the environmental footprint of the CSF.

15.1.10 Water Management Infrastructure

All seepage and runoff water generated on areas impacted by mining activities are considered as “contact water.” Contact water and water from pit dewatering activities will be collected and retained for the settlement of sediment and treatment prior to being released to the environment.

The current water management strategy assumes that a water treatment plant (WTP) will treat excess contact water generated on the mine site, if required, prior to its release at final effluent point. The primary components of the contact water management system and process plant water supply for the Phase 1 of the Whabouchi mine are described below.

15.1.10.1 Collection Ditches

Contact water collection ditches will be constructed to collect all contact water generated at the mine site, including runoff from roads built with waste rock material. The design of collection ditches will be completed or reviewed at the detailed engineering design phase, and ditches will be designed to manage peak flows generated by 24-hour rainfall events with a return period of 100 years.

Two collection ditches around CSF draining towards the BC-01 basin were preliminary designed in 2019 (SNC, 2019a), with a combined length of about 2.5 km. The design considered ditches with a trapezoidal section with side slopes of 1.5H:1V, minimum freeboard of 0.3 m above the maximum water level, and riprap armour as erosion protection, except for segments excavated on roc on which riprap protection is not required. The total minimum required depth varies between 1.0 m for the ditch located north of CSF and 1.5 m for the one located south.

15.1.10.2 Water Basins

Whabouchi mine water management strategy accounts for six contact water basins (BC-01, BC-05, BC-10, BC-11, BC-12, and BC-15). Together, these basins were designed to manage contact water volume generated by a 30-day snowmelt from a snow accumulation with a return period of 100 years, combined with the contact water volume generated by a 24-hour rainfall event with a return period of 1,000 years (SNC, 2019b). Basins will be mostly excavated to store contact water, and basins BC-11, BC-15, and BC-10 have already been constructed. Table 15-3 presents the design volume of each basin.

Table 15-3 Design Volumes of Contact Water Basins

Basin	Required water storage volume (m³)	Total excavation volume (m³)
BC-01	46,100	56,500
BC-05	20,000	26,000
BC-10	2,600	9,200
BC-11	227,700	345,400
BC-12	63,800	73,000

BC-11 receives all the water from the concentrator sector and the industrial area. The BC-11 basin can also receive water from the BC-01, BC-05, BC-10, BC-12, and BC-15 basins to provide a water reserve for the concentrator in winter and summer. Its water can be sent to the concentrator to supply fresh water, and/or to the pit during significant floods when the WTP cannot discharge all the water to the effluent.

BC-12 ultimately receives all the water from the portion of the mine site located south of the North Road and west of the main mining road (industrial zone). It receives pit water (pit dewatering) and runoff from the low-grade stockpile as well as from overburden stockpile 2. Water collected at BC-12 is pumped to BC-11.

Contact water from the waste rock and tailings CSF will be collected in perimeter ditches that drain to either the BC-01 basin, located western of the CSF, or to the BC-05 basin, located east of CSF. BC-05 will also collect snow melt from the snow dump located eastern of CSF. During normal operations, water collected at BC-05 will be pumped towards BC-01, from where it will be pumped to the mine site main water management basin, BC-11. During flood conditions, excess water from both basins (BC-01 and BC-05) will be pumped towards BC-11.

BC-15 is a temporary basin constructed to manage water from pit dewatering before construction of BC-12. Water collected at BC-15 is pumped to BC-11.

15.1.10.3 Process Plant Water Supply

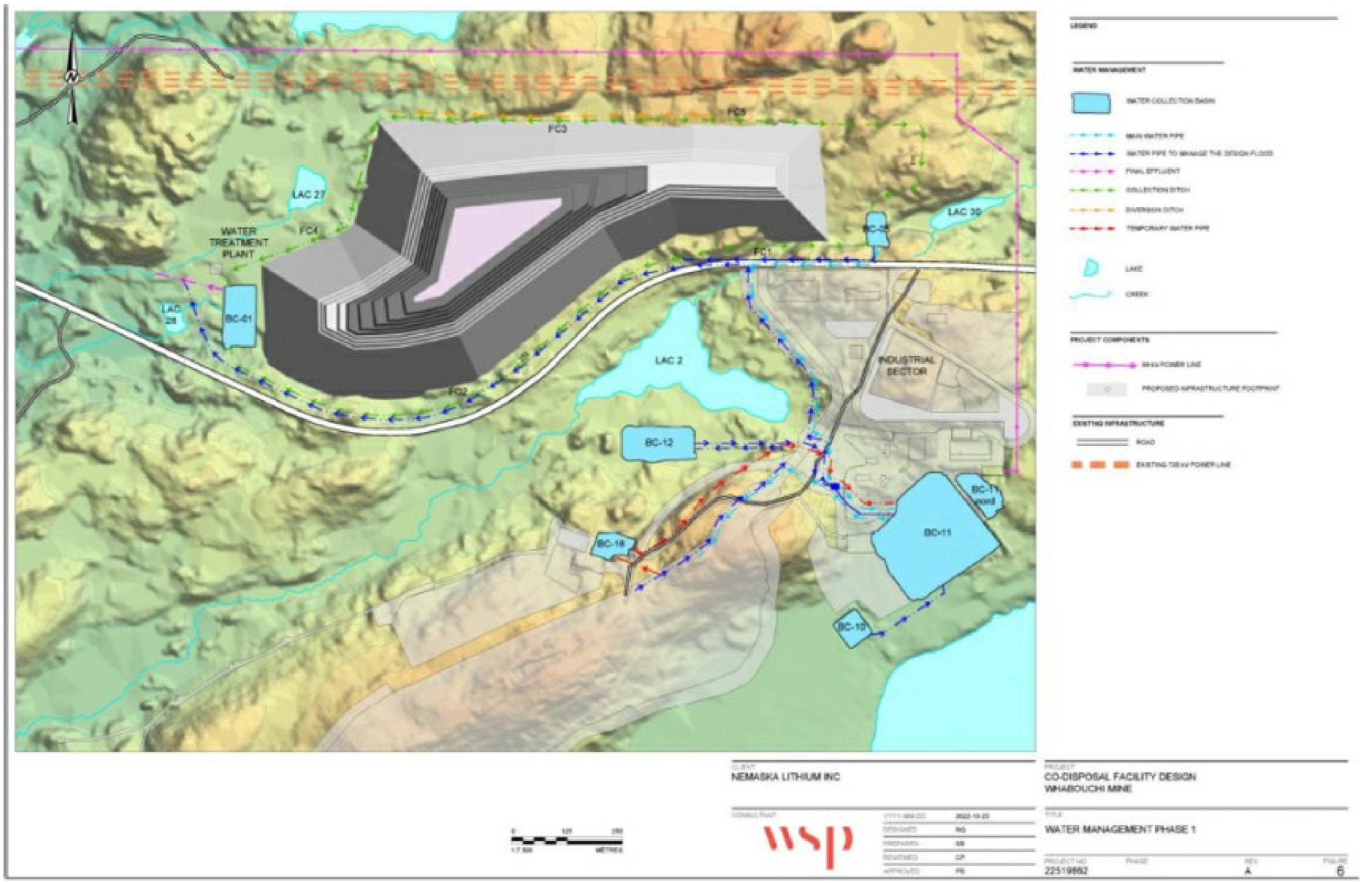
To meet the process plant make-up water requirements, fresh water will come from BC-11 basin, that has a permanent pump station. The annual water balance in the region of Whabouchi mine is positive even during dry historical years (precipitation exceeds evapotranspiration), and the process plant demand could be supplied by the site runoff and pit dewatering flows. The excess of water is expected to be discharged to the environment at the effluent point even during years with low precipitation rates (dry years).

The precipitation patterns in the region of Whabouchi mine, with little rainfall during the winter season (November-April) requires well-defined operational procedures and controls of water storage at BC-11 basin to reduce risks associated with low water reserves during the winter season. Additional quantities of water should be reserved in the BC-11 prior to the onset of winter to account for process plant water needs and water unavailability due to surficial ice formation for a prolonged period (typically from November to May). A water balance study is undergoing to assess the potential risks associated with a prolonged dry season or a prolonged winter period and define a water management procedure to ensure a constant supply of water.

An existing emergency basin, BC-11-Nord, can receive all the water and pulp from the concentrator large tanks and thickeners in case of an emergency stop of the mill or during scheduled maintenance of the concentrator infrastructure. This basin is not part of the flood management system, and water from this basin can only be pumped to the concentrator once emergency conditions or maintenance works are completed.

Figure 15-7 depicts the Whabouchi Mine water strategy for the exploitation Phase 1.

Figure 15-7 Whabouchi Mine Water Management for the Exploitation Phase 1



15.1.11 Power Supply and Distribution

15.1.11.1 Power Line, Main Sub-Station and Electrical Distribution

The total running power demand of the Whabouchi site was determined to be approximately 17.3 MW in winter and 11.4 MW in summer based on the estimated connected load, running load, and running power. Table 15-4 presents the power demand breakdown by sector. Using a power factor (pf) of 0.95, the peak load of 17.3 MW equates to 18.2 MVA.

Table 15-4 Estimated Total Project Power Demand

WBS	WBS Description	Area Operating %	Connected kW	Winter kW	Summer kW	kWh/y
W100	Mine General Area	67	400	300	300	1300
W200	Crushing Area	67	2800	1700	1700	9900
W300	Concentrator	85	10000	5700	5400	41300
W400	Infrastructure	67	1500	700	800	4200
W458 & W491	Camp and Admin Buildings	67	2000	1500	1300	8400
Various	Future Loads	50	2000	1400	1000	5300
Various	Electrical HVAC	67	7000	6100	1000	20900
		Total		25700	17300	11400
W800	Future UG Mine	67	4500	2500	2600	19300

The Whabouchi Project electrical needs are supplied from the Hydro-Québec Nemiscau sub-station at 69 kV. From that point, a new 69 kV overhead power line of approximately 12 km was constructed by NLI for Hydro-Québec terminating at the Project 69 kV to 4.16 kV main outdoor substation. One (1) 15/20 MVA transformer is installed in the Main Substation. The transformer then feeds the 4.16-kV switchgear installed in a dedicated prefabricated electrical room W489-REL-900. The electrical room is installed in the vicinity of the main substation switch yard.

Currently, 10.2 MW (10.7 MVA @ pf of 0.95) is contracted from Hydro-Québec. An increase to 16 MW (17 MVA @ pf of 0.95) has been proposed by Hydro-Québec. These modifications to the Nemiscau substation are required to supply 16 MW, the estimate provided by Hydro-Québec for this work is included in the Capex. Figure 15-8 shows the actual and planned upgrades to the Hydro-Québec substation.

Figure 15-8 Hydro-Québec Nemiscau Sub-Station Upgrades Required

In order to provide redundancy for the existing 69/4.16kV, 15/20MVA Transformer that supplies the power to the mine, a second Transformer feeder bay has been included in the Capex. Details of the layout and configuration will be refined in the next phase of the Project. Figure 15-9 shows the actual and planned upgrades to the transformer feeder bay. An illustration of the substation expansion and power factor corrector unit is shown in Figure 15-10.

Figure 15-9 Second 69 / 4.16kV Transformer Feeder Bay (existing in Blue, new in Grey)

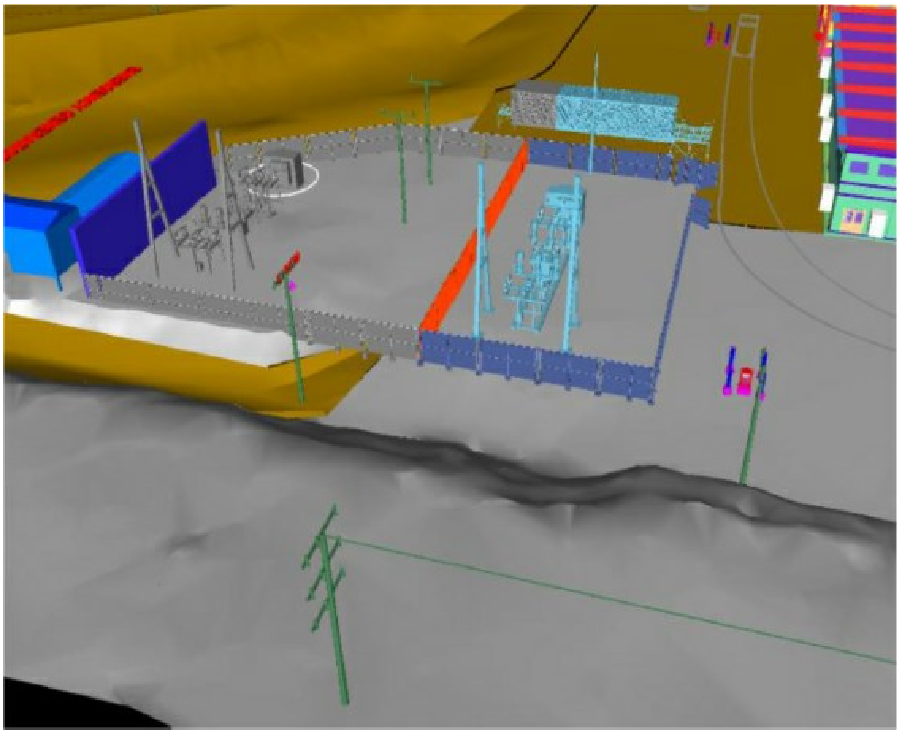
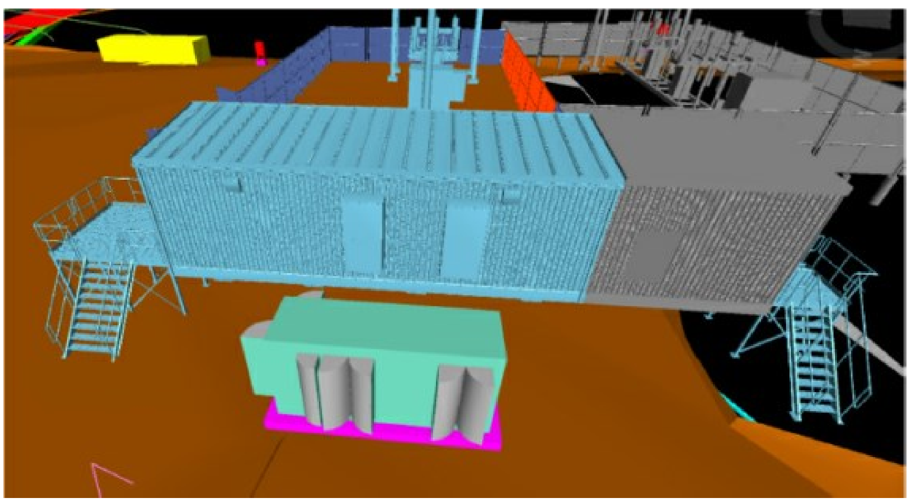


Figure 15-10 4.16kV Substation Expansion (Grey) and Power Factor Correction Unit for Whabouchi Mine (with Pink Base)



The 4.16-kV main switchgear is located in a prefabricated electrical room (W489-REL-900) and provides power to:

A capacitor bank through a feeder. The Power Factor Compensation Unit will improve the power factor to 95%. It will be located in front of the main switchgear electrical room.

The concentrator area through four (4) feeders from the 4.16-kV main switchgear to four (4) dedicated electrical rooms;

The primary, secondary, and tertiary crushing area through a feeder from the 4.16-kV main switchgear to one dedicated electrical room;

Two (2) pole lines at 4.16-kV supply the camp/administration/garage area, pond pumps and explosives magazine (Figure 15-11). The two (2) pole lines will be modified to balance their loads. For this, one pole line will predominantly feed the camp, and the second pole line will feed the mine garage, EFT, and water basins. A link between the existing batch plant and the mine garage is required to make this change.

A picture of the existing transformer bay at Whabouchi is shown in Figure 15-12.

The primary, secondary and tertiary crusher electrical equipment is supplied from the electrical room W489-REL-900. The main equipment installed inside the electrical room comprises a dry step-down 3 MVA, 4.16 – 0.6 kV transformer and the 600 V equipment for distribution and control (MCC, VFD).

The concentrator electrical equipment is supplied from a modular electrical room, comprised of three (3) existing modules: W316-REL-201, W316-REL-202, W316-REL-203 and one (1) new Electrical Room: W316-REL-204. An illustration of the new electrical room and transformer bay is provided in Figure 15-13.

The main equipment installed inside modules W316-REL-201 and W316-REL-202 each comprise a dry step-down 4 MVA, 4.16 – 0.6 kV transformer and the 600 V equipment for distribution and control (LV Switchgears, MCC, and VFD). A picture of the Switchgear (4.16 kV) building is provided in Figure 15-14.

The main equipment installed inside module W316-REL-203 comprise a dry step-down 3 MVA, 4.16 – 0.6 kV transformer and the special control equipment (thyristor control units, total power 2,250 kW, 600 V) dedicated to the control of the W351-HTR-031 DMS concentrate dryer heater. In this electrical room, the VFD is installed which supplies the W331-MLB-030 Ball Mill motor (285 kW).

The New Electrical Room (W316-REL-204) will be fed by the existing 1.5 MVA, 4.16-0.6 kV construction transformer. This electrical room will consist of MCC starter panels and VFDs that feed equipment located in the North West Corner of the Concentrator.

Figure 15-11 Pole line Power Distribution (4.16kV)

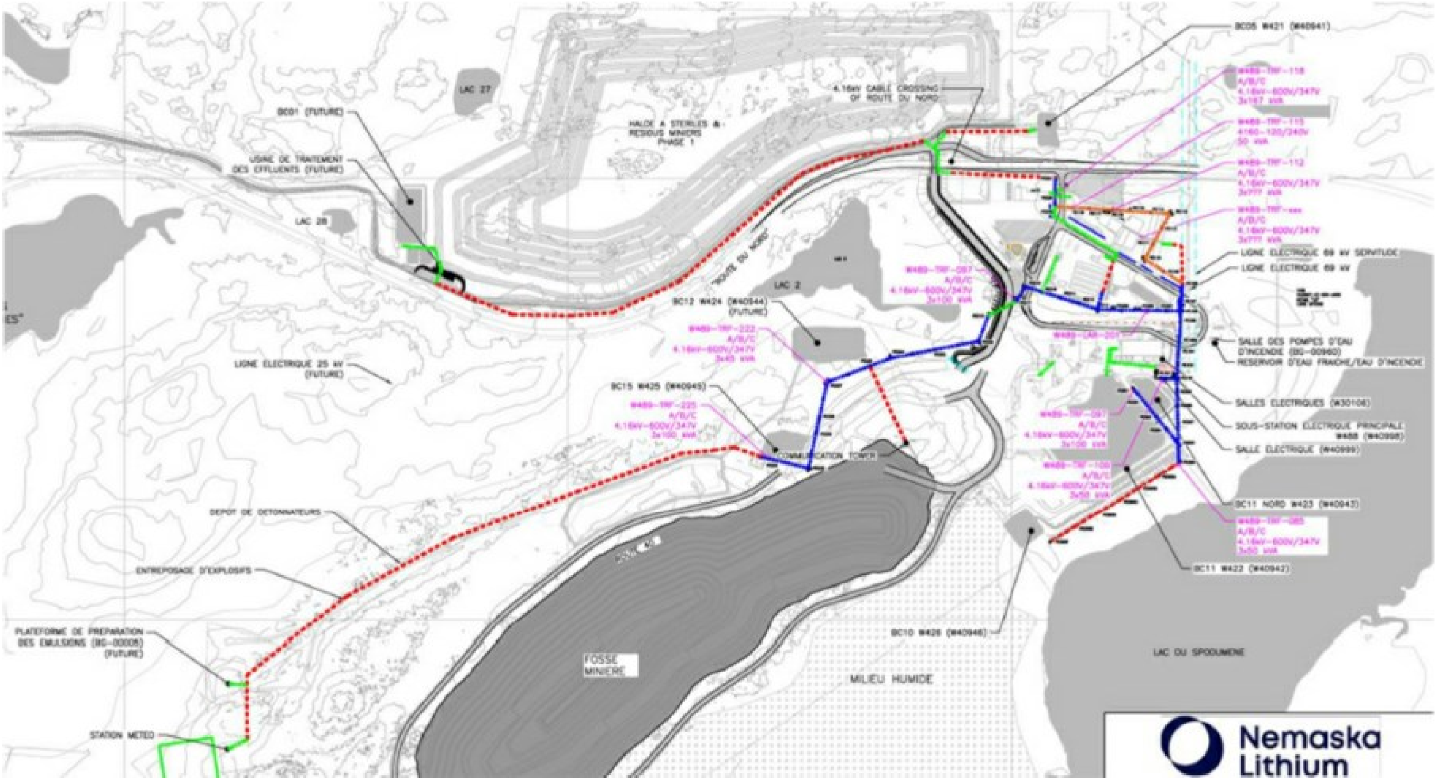


Figure 15-12 Existing Transformer Bay at Whabouchi Mine (69/4.16kV)

Figure 15-13 New Electrical Room and Transformer Bay

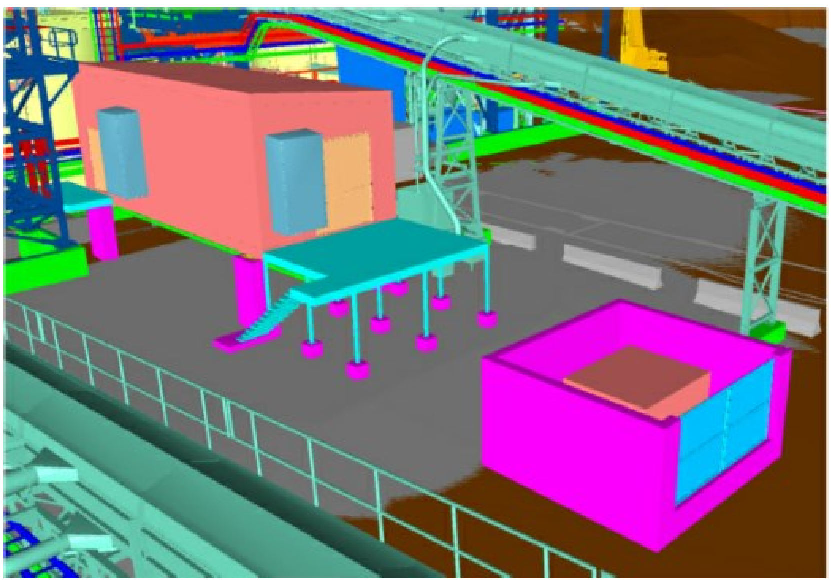


Figure 15-14 4.16 kV Switchgear Building

There are additional electrical rooms dedicated to the following:

- W458-REL-205 dedicated to supply the Camp and Administration building;
- An electrical room located inside the Mine Garage;
- The various basin Pump Stations will be modular structures that contain an electrical section for the pump starter panel or VFD.

A 4.16 kV pole line which will supply power to the water basins (BC-11N, BC-11, BC-05 and BC-01), Temporary Camp, Administration Building, Mine Garage and Wash Bay, Bulk Fueling Station, Communication Tower; and Explosives plant.

There will be no electrical distribution to and within the open pit mine, as all mining equipment, including pumps, will be diesel powered. At Year 25, a circuit will be added to feed the underground ventilation system and services. This will require a 25 kV power line upgrade.

15.1.11.2 Electrical Room

The Whabouchi Village Electrical Room will house the main step-down transformer that will feed the various Whabouchi Village facilities, the medium voltage transfer switch, and the 600 V MCC. The existing diesel-powered Concentrator Emergency Power Generator will be relocated near the Whabouchi Village. The medium voltage transfer switch will switch over electrical feed to the Whabouchi Village Electrical Room to the Emergency Power Generator in case of power outage. Contactors will be provided in the Electrical Room to allow for shutting off low priority consumers whenever the total requirements exceed the capacity of the Emergency Power Generator.

15.1.11.3 Emergency Generators

The existing concentrator Emergency Power Generator (1250 kVA, 4.16kV, diesel) will be relocated near the Whabouchi Village and will be dedicated to the Whabouchi Village's facilities. This will be required to run while Hydro-Québec is upgrading the power available to the mine, during the first year of operation, when the load exceeds the power available. The existing generator will be relocated closer to the Whabouchi Village to reduce the risk of a power outage due to the failure of a power line from the substation to the Whabouchi Village. The Whabouchi Village generator will be connected to the camp electrical room via a changeover switch.

A new Emergency Power Generator (2500 kVA, 4.16kV, diesel) will be purchased for the Concentrator, and loads connected to the pole line, except for the camp. The load on this generator will be managed by the DCS, which will grant or deny permission for loads to start based on their criticality and the power available. This generator will be connected to the 4.16 kV main switchgear W489-SWG-901.

Some of the process equipment that will be connected to the emergency power include:

- Tank agitators;
- Thickener rakes;
- Sump pumps;
- Partial heating/lighting;
- Communication and control equipment;
- Effluent treatment plant; and,
- Fresh and fire water pumps.

15.1.11.4 Future Underground Mine

The current design does not consider the addition of the equipment required for the operation of the underground mine. This will need to be addressed in the next phase of the Project.

Hydro-Québec will need to be consulted, to increase the power consumption limit of the mine from 17 MVA. The load of the underground mine will require an upgrade of the power line from 4.16 kV to 25 kV. The existing power line that is on-site has been designed and constructed for 25 kV, but all the transformers and surge arrestors will need to be changed. A step-up transformer will also be required from 4.16 kV to 25 kV.

15.1.12 Control System

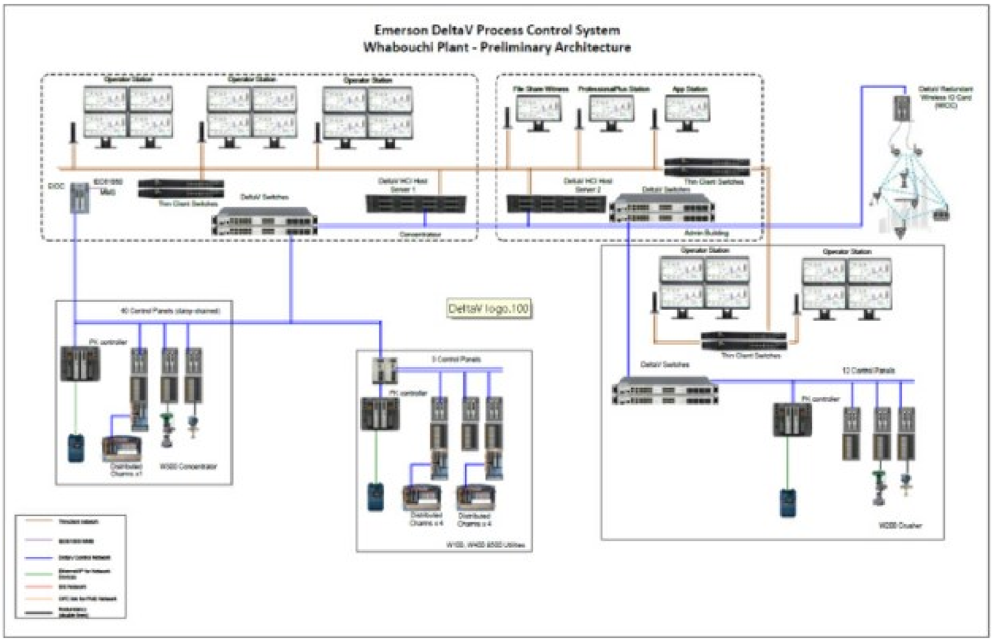
15.1.12.1 Automation Process Network

The topology of the Whabouchi Process Control System (PCS) has a mix of redundant and non-redundant networks (Figure 15-15). A redundant network is formed between the various main controller components four (4) and the various remote chassis. There is also redundancy at the level of servers, switches, and operating PCs. At the level of the electrical rooms, a ring topology is adopted to minimise the risks of network failure. At the I/O level in the process plant, the Remote I/O panel network is connected in a redundant star topology, but a non-redundant topology is used to connect the field components. The network connects all major automation equipment, such as the distributed control system (DCS), historian, human-machine interface (HMI) and system processor. There is an integration platform that links the DCS to the applications and databases.

The proposed network includes fibre optic linking of the following main areas of the Whabouchi concentrator:

- Central Control Room;
- Electrical Room in the Crushing area;
- Electrical Rooms in the Concentrator area (4);
- Electrical Room for the Main Sub-Station;
- Small remote Process;
- Administration Building.

Figure 15-15 Whabouchi Plant Preliminary Architecture



Network automation communication services are:

- Operator stations are located in the central control room and the field (in the concentrator control booths);
- Process control system processors inter-communication;
- PCS/Remote Input/Output (I/O) communication;
- PCS direct interface to the Motor Control Centres (MCCs) and VFD lineups;
- Redundant IEC61850 interface to the power distribution equipment;
- Field device communication including communication with 3rd-party Programmable Logic Controller (PLC) supplied with mechanical equipment;
- A safety camera system will be installed to cover all processing plant areas. This system will include cameras for process control intermittent viewing purposes.

15.1.12.2 Process Control System

The process control system will be of Distributed Control System (DCS) type. Emerson DeltaV has been selected as the DCS. A DCS controller will be assigned to each strategic area with remote I/O panels. These controllers will be located inside the electrical rooms of the respective areas.

Four (4) main controllers will be included to control the Crushing, Concentrator and Utilities areas.

Some vendor PLCs may be supplied with mechanical equipment (crushers, water treatment, filters, etc.). There is an integration platform that links the DCS to the applications and database. The DCS system can control and supervise all the vendor PLCs.

15.1.12.3 Wiring and Junction Boxes

All the field instruments and switches will be wired to the PCS through remote I/O cabinets.

The wiring system will include field junction boxes for instrument power supply to digital signals and analog signals. The motor thermistor signals and RTD signals for motor protection will be wired directly to the related motor protection relays while equipment RTD signals for monitoring will be connected directly to the PCS remote I/Os.

The I/O cabinets will be located and installed in all processing areas of the crusher, the concentrator and in certain places outside near the basins.

15.1.12.4 Distributed Control System

The DCS system will be hosted on two (2) servers. There will be four (4) operator stations and one (1) engineering station. The system will be located in the central control room and the Administration Building.

There will be ten (10) local operation client stations for the various vendor packages throughout the crusher and concentrator. The ten (10) field stations will be located in the primary and secondary crusher area, in the ore sorter area, WHIMS area, the grinding area, and the pumping area.

The PCS will include two (2) historians. One will be local and the other will be corporate. The local Historian is included with Delta-V. The second Historian is the historian that will be shared with the Whabouchi Mine.

15.1.12.5 Uninterruptible Power Supply

In case of a power outage, a diesel generator will supply emergency power to different electric loads throughout the concentrator. The PCS, switches, main servers, phone system, and security systems will be fed by Uninterruptible Power Supply (UPS). The UPS will be powered by the emergency power diesel generator. UPS status will be monitored.

The UPS power supply to critical devices will be fed from two (2) UPS power sources to the redundant power supplies of the devices.

15.1.12.6 Redundancy

Process Control System (DeltaV) and IEC 61850 networks are redundant for increased safety and reliability.

15.1.12.7 Process Analog Instruments

Process analog instruments will support primarily the Hart protocol and they will be wired to remote I/O cabinets to the process controller by traditional 4-20mA cables. Hart Instruments allow better monitoring (than 4-20mA instruments) of the instrument themselves and facilitate configuration and commissioning.

15.1.13 Whabouchi Communication System (Local and External)

15.1.13.1 Telecommunication Guidelines

The telecommunication system will be based on Ethernet links throughout the concentrator buildings and administrative buildings.

A single-mode fibre optic backbone will be deployed through the site buildings to accommodate both automation and corporate services. This will be done using a multicore fibre backbone throughout the site.

For some remote buildings, an additional link will be supplied to communicate with corporate services. For short cable runs, a CAT6 cable will be used. The CAT6 cable will be armored when installed in an instrumentation cable tray. For longer runs, microwave antennas will be used. Secure WI-FI and wired connections will be deployed in every building where required.

15.1.13.2 Telecommunication Services

The site is now connected to the Internet Service Provider (ISP) Telus via an optical fibre (a partnership between Telus and Eeyou Communications Network) ending at the Relais routier Nemiscau (15 km west of the main facility) and a microwave link for the last 15 km. The microwave link is maintained by the local provider, Communications Télésignal Inc. The bandwidth in place offers 30 Mb/s on a 5-year contract basis with the possibility of an increase, as needed. Another distinct 100 Mb/s link is also in place via the same infrastructure to provide internet in the construction camp rooms via WIFI.

A backup system will use a modem or satellite technology. The current cellular coverage (Telus 4G LTE) and the usage of the 4G technology.

The 15 km of the fibre optic link from the Relais routier Nemiscau to the site has been installed.

15.1.13.3 Communication and Mobile Radio Systems

The communication and radio systems require a communication tower at the Whabouchi site hosting the microwave antennas and radio communication equipment.

The communication systems include:

- IP PBX and IP Phones;
- Mobile Radio System.

15.1.13.4 Corporate Network

The Ethernet backbone network comprising 48 fibre optics, in a ring type topology described in the previous section will be used for the automation, the process and security camera video, the IP phone system and the corporate network applications.

All the major network equipment will be located in a dedicated server room located in the administrative office, the telecom shelter, the control room, and electrical rooms.

Corporate services are:

- Wired/Wireless internet/network connection;
- Phones and System Server;
- Process and Security Camera System;
- Access Control System (gate, door);
- Spodumene concentrate containers RFID reader and tracking system;
- Fire Detection.

A camera system, with recorder and a viewer, will be installed in the main gate office. Aside from the gate cameras, various cameras will be installed in the concentrator for process control purposes. One (1) viewing station will be installed in the control room for process control purposes.

16 MARKET STUDIES

The Section was written by Livent based on its extensive knowledge of the lithium industry. Livent relies on Benchmark Mineral Intelligence (BMI) Lithium Forecast Q1 2023 and spodumene concentrate (SC6) contract pricing forecasts by Wood Mackenzie in Q1 2023 for its market analysis and economic analysis, respectively.

16.1 Highlights

The key deliverables of NLI's commercial strategy are as follows:

- To prioritise the needs of its customers to provide a bespoke service during the supply of Nemaska's lithium hydroxide monohydrate and by-products.
- It is envisaged to start production of the Whabouchi mine in late 2024/early 2025 and sell spodumene concentrate prior to the development and coming online of a downstream converstion plant (Bécancour).
- In the future, NLI plans to develop, and deliver spodumene concentrate to, a downstream conversion plant (Bécancour) to supply LiOH.
- To remain profitable throughout the pricing cycle by leveraging its position on the cost curve.

The Whabouchi Mine will produce spodumene concentrate, which may be converted to lithium hydroxide. In turn, NLI's primary objective is to be the supplier of choice for its customers by supplying one of the world's most environmentally sustainable, low-carbon premium lithium chemicals, leveraging Quebec's hydroelectric power network and the regions superior connectivity to international markets. This will further be achieved by meeting customers' needs/preferences and maintaining a strict license to operate but without eroding NLI's bottom line.

NLI's lithium stands to benefit downstream battery and electric vehicle customers in the USA, due to the Inflation Reduction Act which provides consumer tax credits for critical minerals (including lithium) that are processed and refined in countries with free-trade agreements with the USA. NLI's lithium is also expected to benefit under the European Union Critical Raw Materials Act as western countries look to create regional, resilient supply chains with allied countries (including Canada) and decrease their dependence on China where a significant portion of global lithium refining capacity is located today.

16.2 Lithium Market Overview

Lithium demand spans rechargeable batteries for a wide range of end uses, non-rechargeable batteries, aerospace alloys, agrochemicals, air treatment, ceramics, glass, lubricating greases, metallurgy, pharmaceuticals, polymers, and other specialty chemicals. Lithium carbonate is the most widely used form of lithium and has applications in rechargeable batteries, ceramics, glass, metallurgy, and other specialty chemicals.

Lithium hydroxide is widely used in rechargeable batteries, lubricating greases and other specialty chemicals applications. Lithium hydroxide production requires application know-how and extensive manufacturing process technology. Lithium hydroxide must meet specific performance requirements in each customer's manufacturing application. As a result, lithium hydroxide is often developed in collaboration with customers and undergo rigorous qualification processes to ensure they can meet these requirements. Customer qualification processes take a few quarters and may be longer depending on the specification, customer, and application. Customers typically require considerable technical support during and after the qualification process.

16.2.1 Lithium Demand Overview

16.2.1.1 Historical Demand Growth

Since the 2000s, demand growth for lithium was led by the use of lithium in rechargeable battery applications. Lithium-ion cells typically consist of a lithium-metal oxide/phosphate compound as the cathode active material (cathode), a graphite anode and an electrolyte—a lithium salt in an organic solution. Most lithium in a lithium-ion cell application is used in the production of cathode active materials. Cathode materials are produced using a variety of processing techniques that combine lithium carbonate or lithium hydroxide with one or more metal hydroxides or phosphates. For example, lithium hydroxide is used to produce high-nickel content nickel-manganese-cobalt (NMC) oxide and nickel-cobalt-aluminium (NCA) oxide cathodes which are typically used for long-range electric vehicle applications and non-electric vehicle applications requiring high energy density such as gardening tools, power tools and some energy storage systems (ESS) applications. Lithium carbonate is typically used in lithium-iron-phosphate (LFP), low-to-medium-nickel content NMC, lithium cobalt oxide (LCO), lithium manganese oxide (LMO) cathodes which are best suited for short- to medium-range EV applications, portable electronics, or for residential, commercial and industrial-scale, ESS applications due to their long cycle life.

According to Benchmark Mineral Intelligence's Lithium Forecast Q1 2023, rechargeable batteries comprised about 80-83% of global lithium demand in 2022, followed by ceramics and glass (about 6% of global lithium demand in 2022). Lubricant/grease, metallurgy, air treatment, medical and other applications comprised the remaining 11-14% of global lithium demand in 2022.

According to Benchmark Mineral Intelligence, lithium demand from rechargeable batteries grew about 9~10x from 2015 to 2022. Lithium demand from rechargeable batteries represented about 50% of overall lithium demand since 2017.

Historically, the portable electronics market was the largest consumer of lithium in rechargeable batteries, though market growth has been significantly outpaced by electric transportation and ESS more recently. The rechargeable battery industry has been dominated by lithium-ion battery technologies in electric transportation applications since the mid-2010s. Supported by government subsidies, increasingly strict vehicle tailpipe emissions regulations and improving customer sentiment for electric drivetrains, passenger vehicles, buses and commercial vehicles, two- and three-wheeled vehicles, agricultural and industrial vehicles, off-road vehicles have been increasingly lithium-ion battery-powered.

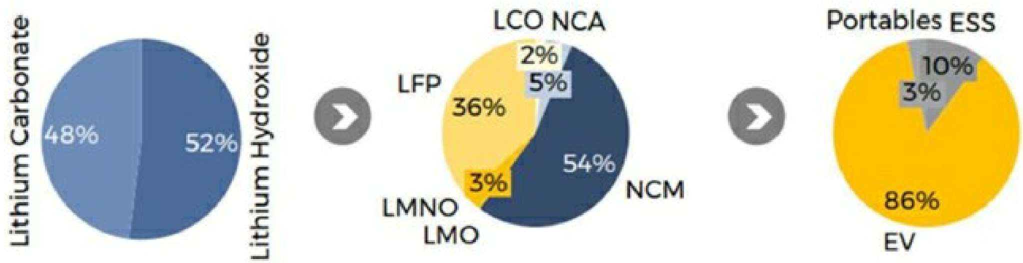
In 2022, China was the largest consumer of lithium. China's lithium demand has increased rapidly since 2015, largely through the rapid expansion of its domestic lithium-ion battery sector to serve the growing EV and energy storage industries. China has been the world's largest market for electric passenger cars, buses, and commercial vehicles. Japan and South Korea are also large consumers of lithium, because of their cathode capacity expansions. Both Europe and North America are mature markets for traditional lithium applications. Since the last few years, several cathode producers are building new cathode capacity in Europe and North America.

16.2.1.2 Forecast Demand Growth

Forecast demand growth will be driven principally by expansion of the EV andESS markets. EV Volumes, in their January 2023 global battery electric and plug-in hybrid electric passenger cars and light commercial vehicles forecast, anticipates sales to be approximately 46 million units in 2030, rising to approximately 74 million units in 2035, representing a penetration rate of 46% and 69%, respectively, of all passenger cars and light commercial vehicles sold globally. Battery-electric vehicles are expected to comprise a clear majority of the EV sales mix. Besides electrification of transportation, electricity generation is expected to continue its decarbonization trend with solar and wind installations crossing new milestones; many of these commercial-, retail- and utility-scale installations are expected to be coupled with lithium-ion battery-based ESS.

As a result of the strong growth in rechargeable battery applications, Benchmark Mineral Intelligence's Lithium Forecast Q1 2023 forecasts lithium carbonate and lithium hydroxide demand to grow approximately 3x and 6x, respectively, from 2022 to 2030. By the end of this decade, Benchmark Mineral Intelligence expect lithium hydroxide demand to exceed lithium carbonate demand. Overall refined lithium demand is expected to grow from approximately 0.8~0.9 million t LCE (Lithium Carbonate Equivalent) in 2022 to 2.8~2.9 million t LCE in 2030. Refined lithium demand growth will translate into mined lithium supply growth.(Figure 16-1).

Figure 16-1 Lithium Demand Breakdown by End-Use, 2030



Source: Benchmarck Mineral Intelligence, 2022

16.2.1.3 Outlook By Geography

Lithium demand will geographically diversify over the next decade, driven by significant expansion of cathode production capacity in Europe and North America. China is likely to continue to be the largest consumer of lithium followed by South Korea in the medium term, because of the ongoing cathode capacity expansions. Japan's cathode capacity will continue to grow at a slower pace compared to that in China and South Korea. Meanwhile, Europe and North America are expected to witness build-out of substantial cathode capacity. India, Indonesia, and Vietnam are also likely to build cathode capacity starting this decade.

Risk Of Substitution

Lithium is the lightest metal and has the highest electrochemical potential, enabling it to achieve very high energy and power densities. It is, therefore, unlikely to be substituted for alternative rechargeable battery technology in EVs. For example, lithium-ion batteries are lighter weight and offer higher energy density than traditional nickel cadmium or nickel metal hydride batteries. Currently all types of active cathode chemistries (nickel-rich like NCM and NCA, LFP/Lithium-iron-phosphate-manganese (LFMP) and manganese-rich) will continue to use either lithium in the form of carbonate or hydroxide.

While sodium-ion batteries are viewed as a potential next-generation rechargeable battery system, its lower energy density (compared to lithium-ion cells) will likely preclude it from mass use in standard to long range EVs and be reserved only for budget, short range EVs, if the technology reaches commercialisation.

Solid-state batteries (SSBs) are an alternative rechargeable battery technology but do not represent a significant substitution risk. SSBs function in a similar manner to lithium-ion technology, using the movement of lithium ions to charge and discharge a cell. However, current SSB technologies use a lithium metal anode and solid electrolyte, instead of a carbon-based anode and liquid electrolyte, in combination with typical cathode materials i.e., SSBs use lithium hydroxide or carbonate to formulate the cathode active materials. The emergence of SSBs would likely put greater demand pressure on the lithium industry requiring greater lithium metal production for the anode, in addition to lithium chemical requirements for the cathode. Despite years of research and development, SSB technology is still not mature enough to achieve widespread commercial use, with further development required to make SSBs viable on a large scale.

16.2.2 Lithium Supply Overview

Lithium is mainly found in three (3) different types of deposits: hard-rock, brine, or clay. Hard-rock deposits like Whabouchi commonly contain lithium in the minerals, spodumene, lepidolite, pegmatite or petalite. Other examples of hard-rock deposits include Western Australia's Greenbushes, Mt Marion, Wodgina, Pilgangoora, Mt Cattlin deposits. Lithium is recovered in the form of spodumene concentrate, which can be further processed into lithium chemicals such as lithium hydroxide and lithium carbonate. Access to low- or zero-carbon energy for mining of lithium bearing ore, processing, and concentrating the ore followed by converting into lithium chemicals can afford hard-rock deposits a favourable sustainability profile.

Lithium-rich continental brines are commonly found in arid climates and result from the weathering of lithium-bearing rocks by groundwater. There are significant resources in South America, particularly in Chile's Atacama, Argentina, Bolivia, and China's Qinghai and Tibet provinces. Lithium is extracted from brine deposits by transferring underground brine to evaporation ponds where the water progressively evaporates to leave a lithium-rich solution. The solution is then treated with reagents followed by purification steps to produce lithium carbonate. Some lithium producers in Chile, China, Russia, and the US convert lithium carbonate into lithium hydroxide. Recently, geothermal brines in Europe and the US have also been under development.

Clay deposits are less common and can contain a variety of soft, lithium-rich clay minerals. These deposits result from the chemical alteration of volcanic rocks by circulating groundwater or incorporation of lithium into existing clay deposits during interaction with lithium-rich groundwater. Currently, no meaningful lithium production comes from clay deposits. Two sites of potential future production include Thacker Pass (USA) and Sonora deposit (Mexico).

16.2.2.1 Historical Mine Supply

According to Benchmark Mineral Intelligence's Lithium Forecast Q1 2023, in 2022, active hard-rock lithium mine supply totalled approximately 0.4 million t on an LCE basis and represented approximately 54% of global lithium supply. This was dominated by operations in Australia which accounted for approximately 80% of global hard-rock lithium mine supply. Other countries with hard-rock lithium mine supply are China, Brazil, Portugal, and Zimbabwe.

Mine capacity in Australia has increased rapidly in recent years to keep up with refined lithium demand from the battery industry. Supply has lagged mine capacity. Since 2015, hard-rock mine supply in Australia increased about 5x to approximately 0.3 million t LCE in 2022.

China comprised the second-largest hard-rock mine supply in 2022, predominantly from its lepidolite deposits, totalling approximately 50,000 t LCE. Brazil-based Mibra mine had the largest hard-rock mine supply outside of Australia and China, of approximately 13,000 t LCE in 2022.

There was no meaningful North American mined hard-rock supply in 2022.

According to Benchmark Mineral Intelligence's Lithium Forecast Q1 2023, in 2022, brine-based supply accounted for approximately 0.3 million t LCE and represented approximately 46% of global lithium supply. Chile accounted for approximately 68%of brine-based supply in 2022 followed by China and Argentina.

There was no meaningful clay-based supply in 2022.

16.2.2.2 Historical Refined Supply

According to Benchmark Mineral Intelligence's Lithium Forecast Q1 2023, refined lithium supply in 2022 totaled approximately 0.7 million t LCE, with lithium carbonate totalled approximately 0.4 million t LCE, and lithium hydroxide totalled approximately 0.2 million t LCE. Lithium chloride and lithium sulfate account for the remainder refined lithium supply.

China dominates the global refined lithium supply, based on domestic lepidolite, brine and spodumene as well as imported spodumene. In 2022, China also recovered LCEs from battery production scrap, and end-of-life lithium-ion batteries.

Outside China, Chile, Argentina, and the US comprised major refined lithium supply in 2022. Australia has seen a build-out of refining capacity but did not see a meaningful refined lithium supply in 2022.

16.2.2.3 Mine Supply Outlook

According to Benchmark Mineral Intelligence's Lithium Forecast Q1 2023, mine supply (hard-rock, brine, clay) will increase from approximately 0.7 million t LCE in 2022 to 2.3 million t LCE in 2030. Hard-rock mine supply is expected to account for approximately 61% of total mine supply in 2030. Hard-rock mine supply projects typically come online faster than brine-based projects. Although actual hard-rock mine supply from Australia increases, Australia's share of global hard-rock mine supply is expected to decline from approximately 80% in 2022 to approximately 47% in 2030 as the combined share of hard-rock mine supply from China, various countries in Africa, and Canada increases.

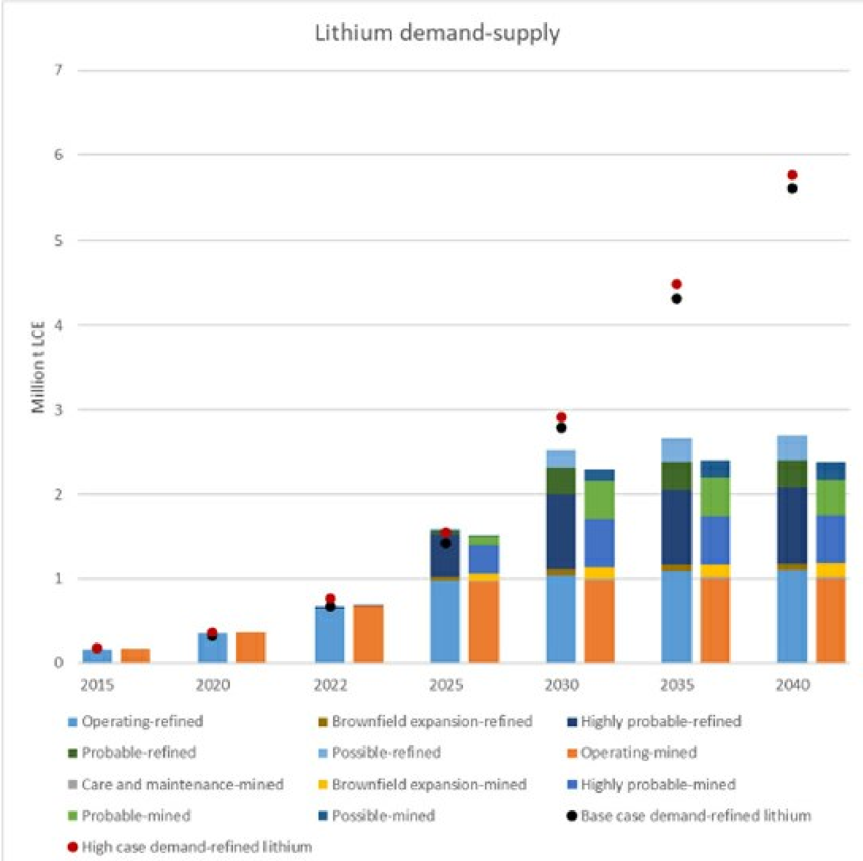
16.2.2.4 Refined Lithium Supply Outlook

According to Benchmark Mineral Intelligence's Lithium Forecast Q1 2023, refined lithium supply (lithium carbonate, hydroxide, chloride, and sulfate) will increase from approximately 0.7 million t LCE in 2022 to 2.5 million t LCE in 2030. Lithium supply forecasts, through 2040, is provided in Figure 16-2.

16.2.2.5 Market Balance and Prices

Spodumene concentrate can be converted to technical and battery-grade lithium carbonate and hydroxide products. Each have distinct supply-demand dynamics and market balances as not every lithium producer is capable of reaching battery-grade purities and qualifying its product with top tier cathode producers or cell manufacturers. While technical-grades may make their way into the lithium-ion battery supply chain through reprocessing, it is likely to have different end-use market that does not require as strict specifications. Demand for battery-grade lithium hydroxide, or that destined for lithium-ion cell production, is forecast to experience unprecedented growth as Western EV manufacturers favor battery chemistries to include high-nickel cathodes.

Figure 16-2 Lithium Demand-Supply, million t LCE



Source: Benchmark Mineral Intelligence Lithium Forecast Q1 2023

In a structural LCE supply (refined and mined) deficit scenario, it is generally difficult to split LCE supply between lithium carbonate and lithium hydroxide. Benchmark Mineral Intelligence estimate a mined supply deficit of just under 0.1 million t LCE in 2022. They expect the mined supply to fluctuate between deficit and surplus from 2023 until 2028 and then a return to widening mined supply deficits starting in 2029 into the next decade.

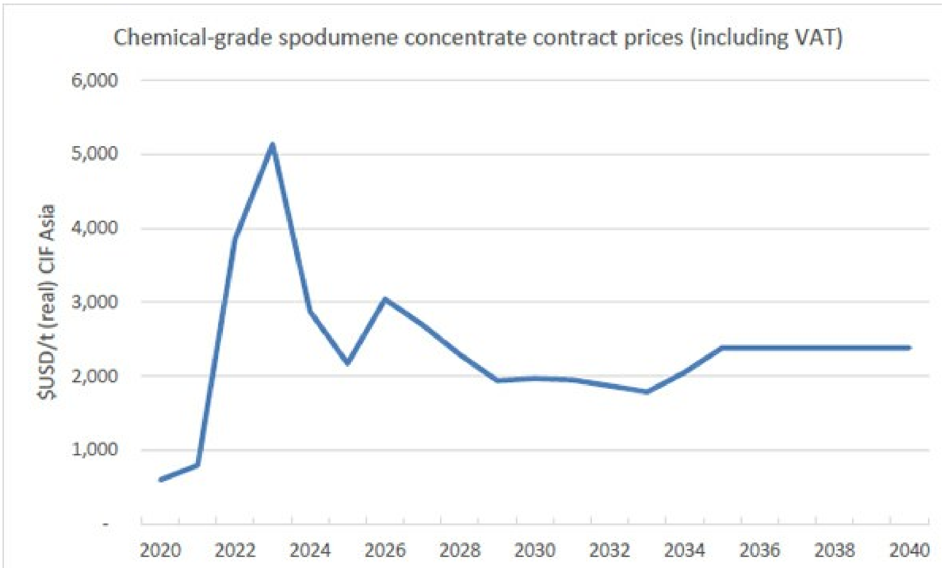
With the ongoing overall market tightness, battery-grade products are likely to experience a more pronounced supply deficit than technical-grade products. The lithium hydroxide market is expected to experience tighter conditions than carbonate as supplies are constrained by producers' struggle to expand production in line with demand while maintaining increasingly strict quality standards. In addition, lithium hydroxide shelf-life challenges (e.g., clumping over extended periods of storage) will hamper prolonged stockpile growth in times of subdued demand. As such, producers will need to proactively schedule production in line with contractual needs and market conditions as part of efforts to mitigate production volume versus product sale risk. This will naturally tighten market supply and provide additional upward pressure on pricing.

Lithium raw materials is unlike traditional commodities that can be physically traded on transparent metal exchanges, such as the London Metal Exchange (LME). Lithium companies increasingly refer to market pricing from price reporting agencies such as Fastmarkets and Wood Mackenzie as a single reference index for pricing does not yet exist. While each market research firm's pricing forecast shows a degree of variation (due to differing respective supply-demand models), following recent extreme price increases, due to rapidly improving demand for lithium chemicals, analysts expect prices to show only moderate improvement in the short-term before prices stabilise or show a slight decline as supply from new conversion plants increases over the medium-to-long-term.

- A review of the market studies prepared for NLI, anchored on BMI and Wood Mackenzie reports, market input, and leveraging Livent's knowledge and experience with potential customers.

Wood Mackenzie's historic and forward-looking chemical grade spodumene contract price forecast is provided in Figure 16-3.The sales pricing adopted for the economic analysis (reported in $USD on a per tonne basis, with VAT), uses a long-term (2035 and beyond) price for chemical-grade spodumene concentrate sales of $2,381/t.

Figure 16-3 Spodumene Price Forecast



Source: Wood Mackenzie, April 2023

16.3 NLI's Supplier Differentiation

NLI's operations are based entirely in Quebec, a Tier-1 investment and mining jurisdiction. Its location provides several strategic advantages for prospective clients which are highlighted below:

- Stable political and fiscal environment: Canada is an attractive investment destination with an established mining history. This is a major advantage over other lithium operations where the risk of corruption and expropriation are significantly higher.

- Best-in-class environmental credentials: On a per tonne basis, NLI will have one of the lowest carbon, waste and water intensities of global lithium chemical production, important value levers for its potential customers, which are increasingly setting net zero emission targets, or significant emission reductions due to Quebec's abundant hydroelectric power. Conversion of spodumene to lithium hydroxide in China is powered by an electricity grid heavily reliant on fossil fuels.

- Cycle resistant, fully integrated operation: NLI's fully integrated, cost competitive operations will source high-quality spodumene concentrate from its Whabouchi Mine. This will allow it to have greater quality control on lithium hydroxide production and optimise plant throughput compared to Chinese converter which need to source external feedstock. This will enable Nemaska to operate profitably through the lithium market cycles, enabling a consistent supply to its customers.

- Geographic proximity to North American customers: As lithium demand is forecasted to significantly increase and North America seeks to reduce its dependence on battery materials produced in Asia, NLI is ideally located to provide lithium hydroxide to the significant number of new battery manufacturing facilities due for commissioning in the coming years. The announcement of the Inflation Reduction Act in August 2022 includes several incentives to strengthen the US supply chain for critical minerals. Lithium produced in countries with free-trade agreements with the USA, such as Canada, will stand to benefit from these incentives.

- Logistical flexibility to serve all customers: NLI's lithium hydroxide processing facility in Bécancour is well served by CN Rail's transcontinental rail system in addition to being located close to major highways and the Ports of Montreal/Trois-Rivières/Quebec City. This will facilitate flexible access to North American and global markets.

- Quality assurance: NLI will have an information-based production line with an associated quality control system and a professional production management team to ensure consistent product quality that exceeds customer specifications. Nemaska will target certification under international standards including the EU's REACH and RoHS, the USA's Toxic Substances Control Act (TSCA), International Automotive Task Force (IATF) 16949 as well as ISO 9001 (quality management), ISO 14001 (environmental management) and ISO 45001 (health and safety).

- De-risked operations: NLI's lithium hydroxide process has been heavily tested and designed by shareholders with decades of experience in project development, resource extraction and battery-grade lithium chemical production.

- Permitting and social acceptance already acquired: NLI has secured all the necessary permits for mining and has built a strong licence to operate with the local community through regular engagement with the Cree Nation of Nemaska, which is solidified by a complementary Impact and Benefit Agreement. NLI has also secured significant backing from the Quebec government through Investissement Québec's 50 shareholding in NLI and its continued support through project development.

16.4 Contracts

As of the date of this Technical Report, NLI has entered into four agreements with Livent or its subsidiaries, as described below.

NLI and Livent USA Corp., a subsidiary of Livent, entered into, in May 2022, a technology and technical advisory support agreement under which Livent provides NLI with development, optimization and technical advisory support in connection with NLI's development of facilities and operations for the conversion of lithium sulfate to lithium hydroxide. The agreement is effective for five years unless earlier terminated.

NLI and Livent are also parties to a marketing services agreement under which Livent provides exclusive marketing and sales services to NLI, and a project management services agreement under which Livent provides NLI with project management technical expertise and support.

In June 2023, NLI, Investissement Québec and Québec Lithium Partners (UK) Limited, a subsidiary of Livent, entered into an amended and restated unanimous shareholder agreement, which establishes terms for, among other things, the governance, funding and operations of NLI.

Livent believes that the aforementioned contracts were negotiated on an arm's length basis and contain terms, rates or charges that are substantially the same under the circumstances as could be obtained had the contracts been negotiated with an unaffiliated third party.

In May 2023, Ford and NLI entered into a long-term supply agreement (over 11 years), beginning in 2025. The agreement initially calls for spodumene concentrate and later transitions to delivery of battery-grade lithium hydroxide. Spodumene concentrate produced exclusively from the Whabouchi mine will provide the feedstock for lithium hydroxide at volumes up to 13,000 metric tons per year, which is equivalent to approximately 100,000 dry metric tons per year of spodumene concentrate.

Additional supply agreements will be negotiated on a case-by-case basis and with prices linked to an internationally recognized and industry-accepted price index.

17 ENVIRONMENTAL STUDIES, PERMITTING, AND PLANS, NEGOTIATIONS, OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

17.1 Environmental Baseline Studies

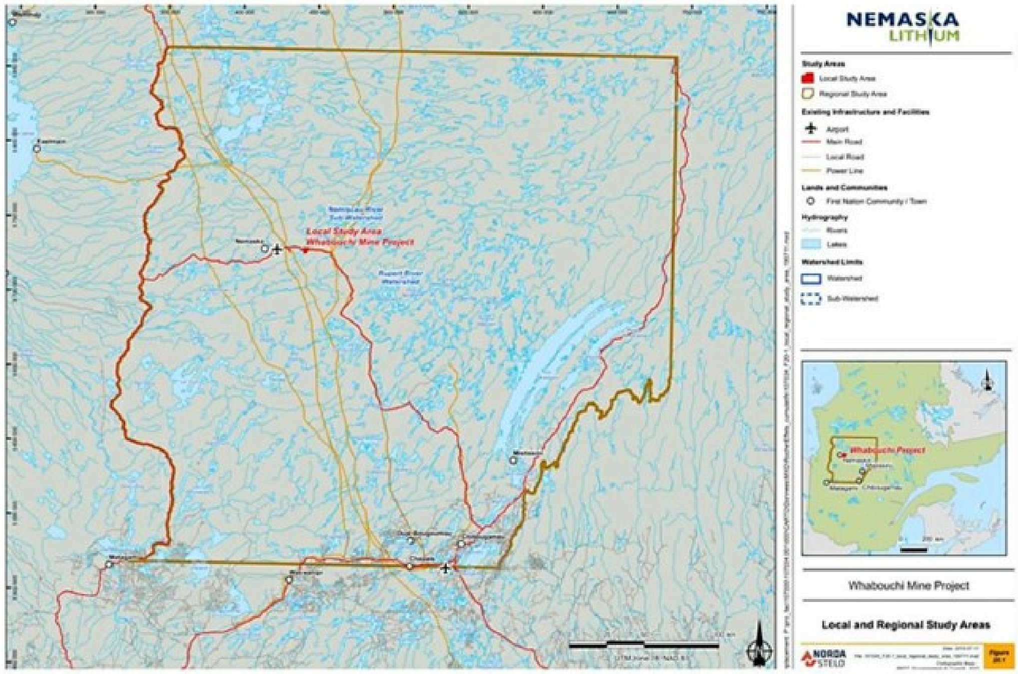
Baseline environmental studies at the Whabouchi Mine Project site began in August 2010 with field surveys for water quality, sediment quality, benthic invertebrates, and fish. During 2011 and through 2012, additional data were collected, focusing on fish, surface water quality, bathymetry, hydrology, ground water quality, soil quality, air quality, noise, large mammals, small mammals, bats, birds, amphibians, and reptiles.

Two (2) study areas have been identified for the ESIA and the associated environmental and baseline studies. The “local study area” includes all the areas likely to be directly physically impacted by the mine development (pit, buildings, and roads) or that is in its immediate vicinity. Such area encompasses zones that are likely to be disturbed by the activities on-site (site preparation, noise, dust emissions, ore extraction, waste rock and tailings disposal, discharge of mine effluent, etc.). The “regional study area” is a larger area extending out of the Property and to which are potentially associated cumulative effects with other projects or infrastructure; such effects are typically associated with water quality, wildlife, and socio-economic aspects. The extents of both study areas are shown in Figure 17-1.

Physical and biological environment of the Whabouchi Mine Project area were described based on information collected from various sources:

- Field surveys;
- Aerial photographs and/or satellite images, maps, and geomatics tools;
- Information provided by various governmental agencies as well as by other project proponents active in the territory (municipality, other mine or hydropower projects, etc.);
- Studies from the scientific and technical literature.

Figure 17-1 Whabouchi Mine Project – Environmental Baseline Studies



17.1.1 Surface and Ground Waters

The characterization and validation of water courses and watersheds were done using topographical and thematic maps, Digital Elevation Models (DEM), aerial photos and field surveys. Watershed limits and stream alignments within the limits of the Property were established using LiDAR data and drone surveys. Calculations of stream flow rates derived from standard CEHQ models were made and supplemented with field checks.

The study area is characterized by the presence of numerous water bodies, streams, and wetlands. The mine site is in the Rupert River watershed, which has a surface area of 43,400 km² and flows from east to west towards the Rupert Bay and ultimately into James and Hudson bays. The hydrographic network on the Property is constituted of five (5) creeks, named A, B, C, D, and E.

All of those small watersheds near the mine site drain into the Nemiscau River (2,000 km² watershed upstream from the site) itself, a tributary to the Rupert River that joins 70 km downstream from the site. The Nemiscau River is on the western perimeter of the site and the on-site drainage trends west southwest towards the Nemiscau River. The largest stream on-site is the one that drains Spodumene Lake (namely, creek D) with a watershed of over 100 km². The other on-site watershed surface areas are well under 5 km².

The largest lake in the vicinity of the site is Mountain Lake, with a surface area of 1,375 ha. It lies on the western side of the Whabouchi Property and is actually a widening of the Nemiscau River. Spodumene Lake, to the east of the site, is only 61 ha and is the second largest lake in the mine site vicinity. The Mine Project study area is characterized by the presence of unconsolidated deposits, essentially of glacial and fluvio-glacial origin. Globally, their thickness is less than four (4) m, except in an area northwest of Spodumene Lake, where their thickness can reach about 15 m. The landscape in the study area is dominated by rocky hills that are mostly oriented northeast/southwest. Most of these hills are low, with an elevation difference of less than 50 m.

A major part of the study area is characterized by the presence of undifferentiated till in thicknesses varying from one (1) m or two (2) m to more than 10 m. At the southwest of the study area, particularly on either side of the Nemiscau River, segments of the Sakami Moraine can be seen.

As part of the soil characterization program completed on the Whabouchi Property (during exploratory trenching), no results exceeded the applicable criteria C (for industrial areas such as mine sites). The results of the soils characterization completed for the Project indicate that there is no contamination in the unconsolidated deposits at the site.

Field hydrogeological data was used to perform groundwater flow modelling (Richelieu Hydrogéologie, 2012 and 2014). The results of that modelling enable portraying the hydrogeological conditions prevailing on the projected mine site and identifying the potential impacts of the Project on groundwater.

Groundwater was sampled and analysed at several well locations from 2011 to 2022 and results suggest that it is generally of good quality. Some increases in concentration of some parameters are observed since the beginning of more extensive surface disturbance related to pit overburden removal and infrastructure area earthworks. Groundwater quality is characterized by a slight acidic pH. Natural background exceedances of resurgence in surface water criteria of the Soil Protection and Rehabilitation of Contaminated Sites Intervention Guide (SPRCSIG) criteria, were measured for copper, zinc, mercury, aluminum, barium and nickel, all common metals in the Canadian Shield geological region.

The surface waters quality on and near the site are typical of the Quebec boreal zone of the Canadian Shield. They are clear, very soft and of very low conductivity. The pH values range from 4.53 to 6.88. Acidic waters are common for Canadian Shield water bodies. They are poor in nitrogenous and phosphorous nutrients, what is typical of oligotrophic environments. Alkalinity levels are very low, indicating high sensitivity to acidification (low to very low buffering capacity). Results indicate that metal and metalloid contents in surface water of the study area are low. Exceedances of applicable criteria were measured for aluminum, iron, mercury, arsenic, beryllium, copper, and lead.

Sediment quality data collected in 2014 suggests that they are generally of good quality with some elevated arsenic, copper, mercury, lead and zinc concentrations.

17.1.2 Biological Environment

The Whabouchi Mine Project is located at the northern limit of the spruce-moss forest domain, in the continuous boreal forest sub-zone. The site is not located in or nearby any protected areas, as designated by the Quebec or Canadian governments.

Black spruce is the dominant tree species, with other species being found, notably white birch, trembling aspen and balsam poplar. Forest fires have modeled the forest dynamic in the Nemiscau area. The presence of many recent burns (less than 20 years old) near the Project site is an indication of this phenomenon. In fact, recent burns represent the most important class of vegetation in the study area (80% of the total impacted area).

Twenty-six (26) special-status plant species could potentially be present in the Project study area. Due to the features of the Project site, three (3) of these special-status species are susceptible of being found in bogs, namely dragon’s mouth, linear-leaved sundew, and twin-scaped bladderwort. Even though specific attention was paid to those species as part of all field works, none was observed. Moreover, no exotic invasive plant was observed on the Property.

Ten (10) species of amphibians and reptiles could be present in the study area. These include the yellow-spotted salamander, American toad, wood frog, mink frog and common garter snake. Field surveys confirmed the presence of some of these species, but of no special-status species of amphibian or reptile in the study area.

With regards to mammals, several species of large and small fauna are present in the study area. Moose and black bear are the two (2) main species of large mammals found in the study area. During the aerial surveys, five (5) moose yards with individuals present were observed, while no black bear was seen. Gray wolf, North American beaver, marten, and red fox are among the other mammals observed in the Study area.

Six (6) bat species could be present in the study area. Field surveys carried out in 2012 have confirmed the presence of bat species of the Myotis and Lasiurus genders. The Myotis gender comprises two (2) species (little brown bat and northern long-eared bat) that are listed as endangered under the Species at Risk Act while the Lasiurus gender comprises two species (hoary bat and eastern red bat) that are considered as likely to be designated threatened or vulnerable by the Quebec Government. A nursery roost of approximately 300 little brown bat individuals is present near the Mine Project site, more precisely at the northern end of the Spodumene Lake. This maternity located in an old cabin is monitored by the Ministère des forêts, de la faune et des parcs (MFFP)). In addition, there is a prior record of one (1) sighting of a hoary bat near Spodumene Lake.

Otherwise, 13 species of micromammals (small mammals, mice, and voles) might be present in the study area. These include the rock vole, southern bog lemming, deer mouse and arctic shrew. The deer mouse was the species captured most often during the inventories completed for the Whabouchi Mine Project.

A total of ten (10) special-status mammals (Quebec's Act Respecting the Conservation and Development of Wildlife and Canadian Species at Risk Act (SARA)) could be present in the study area: the least weasel, rock vole, southern bog lemming, wolverine, woodland caribou (forest ecotype), silver-haired bat, northern long-eared bat, hoary bat, eastern red bat, and little brown bat. However, except for the hoary and little brown bats for which confirmed sightings exist, none was observed during the field surveys conducted on the Mine Study Area up to this date.

According to the documentation that was consulted, 131 bird species from the four (4) following groups could be present in the study area: waterfowl (geese, ducks, and loons), other aquatic birds (gulls, herons, etc.), raptors (falcons, eagles, owls, etc.) and terrestrial birds (grouse, nighthawks, woodpeckers, etc.). Among the waterfowl, the Canada goose is the most abundant species during the spring migration, while during the fall migration, American black duck is the most numerous. Seven (7) species susceptible of being present in the study area have a special status at the provincial and/or federal level: golden eagle, common nighthawk, peregrine falcon, short-eared owl, olive-sided flycatcher, bald eagle, and rusty blackbird. One (1) bird species, the common nighthawk, listed as “threatened” under the Federal Species at Risk Act, has been confirmed in the study area. In addition, some sightings of bald eagles have been reported around the Mountain Lake.

Fisheries and fish habitat assessment works were completed at various periods since 2010. Fish populations were inventoried using experimental fishing nets, bait traps, hoop nets and electrofishing. Thirteen (13) fish species have been identified in the lakes and streams near the site during fieldwork. The species caught most frequently is the lake whitefish. Other common species include white sucker, walleye, and brook trout.

Several of the fish species present are caught and eaten by local residents, notably brook trout, walleye, lake whitefish, and pike. In November 2011, over 100 fish tissue samples were collected and preserved for chemical analyses. Preliminary evaluation of the results from 40 of these fish suggests that levels of metals are generally low, even though mercury levels were above detection limits in all samples.

To date, no special-status aquatic species has been found on the site. Lake sturgeon is a species considered as likely to be designated threatened or vulnerable in Quebec and is present in some nearby lakes (i.e., Nemaska Lake) and in the Rupert River system. However, there are no indications of its presence on the Nemiscau River drainage area this far up. Local people have stated that they have not found it in this area.
Background benthic invertebrate studies were undertaken in 2010. Benthic invertebrates are important indicators of the quality of aquatic habitats. Results show high variability between sites. In addition, characterization of benthic invertebrates within the final effluent discharge area were carried out in September 2017.

17.1.3 Social Development

The following studies were used to feed both the voluntary impact study and the environmental permitting request for the process plant:

- Allochthonous environment baseline (population, demography, socio-economic conditions, quality of life and community health and well-being, current land-use);
- Indigenous environment baseline (concerns, socio-demographic conditions, cultural heritage, traditional know-how);
- Traffic baseline;
- Archaeological potential study.

Information pertaining to local demographics, economic development, land use, cultural heritage, health and social services, and infrastructure has been collected in order to provide a snapshot of the community’s needs and priorities and to determine how current conditions may be affected by the proposed Whabouchi Mine Project.

Results are based largely on data obtained from the local First Nations government (Cree Nation of Nemaska Band Council), local social service providers, educational institutions, law enforcement, regional Cree entities and Census Canada, as well as on data obtained from engagement with land users and community stakeholders (ex. in-depth semi-structured interviews with trapline R^{2}0 family and other Cree land users, focus groups, etc.).

Efforts were made to create and maintain a collaborative and cordial relation with the community, in particular with families affected by the Mine Project, in order to address issues as they emerge throughout the course of consultations and Project development. To that regard, it should be noted that NLI has engaged in a Community Dialogue and Planning initiative led by the Cree Nation of Nemaska in collaboration with NetPositive and which aimed at planning for and managing natural resource development activities affecting the community.

The process also convened a discussion among the Nemaska community and other key stakeholders (ex. NLI, Cree Nation Government) about the socio-economic development of Nemaska and the community’s vision for the future. A phase 2 is being developed and will implement several priority recommendations from the first assessment phase and continue the visioning and planning work that was begun.

Phase 2 will focus on formalizing the community of Nemaska’s vision by linking existing visioning activities to mining. Furthermore, the project will focus on formalizing a comprehensive community plan for mining aligned with that vision, including by building internal community capacity and by establishing the necessary communication and coordination structures.

Historical Background and Social Changes

The James Bay Crees occupies the immense territory called Eeyou Istchee, of which the limits are defined in the James Bay and Northern Québec Agreement (JBNQA), the first major agreement concluded between the Government of Quebec, the Crees and the Inuits of Northern Quebec in 1975. The JBNQA, the Paix des Braves (Agreement Respecting a New Relationship Between the Cree Nation and the Government of Quebec) and the Agreement Concerning a New Relationship Between the Government of Canada and the Cree of Eeyou Istchee constitute the legal, political, and administrative framework throughout which we can understand the accelerated growth and social development of the signatory communities, which in turn influenced the way of life and the occupation of the land.

The territory of the Crees, namely Eeyou Istchee, comprises nine communities. These communities are located on the shore of the James and Hudson bays (Whapmagoostui, Chisasibi, Wemindji, Eastmain, Waskaganish) as well as inland (Nemaska, Waswanipi, Oujé-Bougoumou, Mistissini). The total population of the Cree Nation of Eeyou Istchee is currently estimated at close to 17,000 persons (in 2015), the majority of which speaks English.

The territory of each Cree community is subdivided into a variable number of family hunting grounds or trapping territories (traplines), each under the authority of a tallyman whose role consists, among others, in ensuring the proper management of exploitable resources and of the areas to preserve.

The Cree communities are united under the Grand Council of the Crees of Eeyou Istchee (GCCEI) and its administrative branch, the Cree Nation Government (CNG). Furthermore, since January 1, 2014, the Cree and Jamesians (the non-Aboriginal regional population) are unified under the new Eeyou Istchee/James Bay Regional Government, established pursuant to the Agreement on Governance in the Eeyou Istchee/James Bay Territory. This regional government offers a new model, formally constituted with equal representation of Aboriginal and non-Aboriginal populations.

It exercises powers of local and regional municipal governance, regional development and land and resource use planning over the Category III lands of the Eeyou Istchee/James Bay region.

Socio-Economic Baseline of the Cree Nation of Nemaska

The Cree Nation of Nemaska, which means “plenty of fish”, is situated on the shores of Champion Lake in Eeyou Istchee. Nemaska is a new and modern village comprised of Cree families originally living at the Nemiscau trading post on Lake Nemiscau. The community of Nemaska is accessible all year long from Matagami via the James Bay Highway (over 390 km away), and from Chibougamau by the Route du Nord (approximately 330 km).

It is a fairly small, but fully serviced community of around 800 people. Nemaska followed the trend seen in other Eeyou Istchee communities, with a rapid population growth over the last 30 years. However, for the past ten (10) years, this growth rate has been decreasing gradually, as in the rest of the Cree population of Eeyou Istchee.

The main demographic feature of the Nemaska community is its youth. Thirty-one percent (31%) of the Nemaskau Eenouch (the “people of Nemaska”) is less than 15 years old and according to the 2006 Canada census data, the mean population age was 25 years old. Elders represent the smallest age group at 4% of the total population.

The Nemaska community is an important Cree administrative center in the Eeyou Istchee region. The offices of the GCCEI and of the CNG are located in the community.

Social service facilities in the community include the wellness center, Nemaska clinic, social services center, school, daycare, sports complex and multi-service centre and youth centre.

Land and Resource Use

Since the creation of beaver preserves that began in the 1930s, the territory of each Eeyou Istchee Cree community is subdivided into a number of family hunting grounds or trapping territories (traplines).

The Whabouchi Mine Project is located on part of the traditional territory of the Cree Nation of Nemaska, more specifically on Trapline $R^2$0. The Whabouchi Mine is located in the extreme south corner of Trapline $R^2$0, and thus occupies only a minor portion of it. The trapline is under the direction of a hunting leader or tallyman, known as uuchimaau in Cree, who is responsible for the continued monitoring and management of the resources.

Trapline $R^2$0 is regularly used by family members and their extended family, as well as by other Crees who have camps along the Route du Nord, who frequent the Bible Camp or fish in Mountain Lake and the Nemiscau River.

Traditional harvesting activities continue all year long. The spring goose hunt is practiced in several sectors of the trapline, and many lakes and rivers offer good fishing grounds. Big-game hunting, as well as waterfowl and ptarmigan hunting, are practiced mainly in the fall and winter. Furbearer trapping is practiced in the winter. Harvesting activities decrease during the summer season, except for fishing and berry harvesting.

Trapline R^{2}0 hosts a site that has long been used for community activities and that was recently transformed into a Bible Camp to receive Nemaska children and families for summer camps, religious gatherings, and traditional Cree ceremonies.

Non-Natives also visit the territory during the summer for fishing and camping, or during the hunting season. Among them are workers from Nemaska community and from nearby work camps such as Hydro-Québec's Nemiscau Camp.

Economy, Employment and Education

Even though Eeyou Istchee is considered a resource region (i.e., where the main work providers are primary sector companies such as hydroelectricity, forests, and mineral), Nemaska is distinct from other Cree communities in that it is an important administrative center for the Eeyou Istchee territory.

In fact, in 2008, the tertiary sector (health, social and education services, municipal services or other governmental services) was the predominant economic sector, representing 85.4% of the employment in the community. Moreover, nearly a third (31%) of the jobs is in public administration, while this category of employment provides only 20% of the jobs elsewhere in Eeyou Istchee.

In Nemaska, the primary sector represented only 8.3% of the jobs in the community in 2008 compared to close to one fourth (4th) of the jobs (23.8%) in the whole Cree Nation. As in the case of the primary sector, construction is a marginal economic activity in Nemaska, representing only 4.9% of the jobs in the community while, in Eeyou Istchee, it represents almost twice the proportion of jobs (8.7%).

Several regional organizations and businesses are active in the Eeyou Istchee territory, and more specifically in Nemaska. These organizations and businesses (ex. CreeCo, Cree Construction and Development Company, Air Creebec, PetroNor, Kepa Group, Nemaska Eenou Company, NDC-Fournier, VPC, Iywaashtin Enterprises, Meeyobin Company) belong to the Crees, the Band Councils or to a Cree-held entity. Although Nemaska constitutes an administrative centre, very few among these businesses have their headquarters within this community.

However, local economic opportunities in Nemaska have increased over the past several years. Contributing factors include Hydro-Québec contracts being awarded to local contractors for construction projects and environmental monitoring, as well as new community infrastructure projects undertaken by local government and associated economic spin-offs. Taking into consideration the large youth population in Nemaska, the Cree School Board, in partnership with Nemaska Lithium and other regional partners, is currently rolling out various training and skills development programs (ex. heavy machinery operation, ore processing, drilling and blasting, etc.) which will play a key role in building local capacity to meet future needs.

A recent development in the community is the emergence of employment at the entrepreneurial level. Individuals are offering such services as equipment rentals (dump trucks, snowmobiles), video rentals, painting, laundry, towing and equipment maintenance and repairs. As well, the Nemaska clinic, fire hall, band council headquarters and justice facilities were recently built. However, in spite of efforts to stimulate the local economy, people in the community still report a lack of jobs.

In general, the Cree communities experience a higher unemployment rate than the Quebec average. In 2008, although the unemployment rate in the community of Nemaska was lower than in the Eeyou Istchee territory, it was much higher (16.4%) than for the population of 15 years old and more in Quebec as a whole (6.9%). Furthermore, the unemployment rate among the youth (15 to 20 years old; 23%) is much higher than among adults (24 to 64 years old; 15%).

Another important element that must be accounted for is that the rate of graduation and participation in post-secondary education programs is generally lower in the Cree communities than in the rest of Quebec. The graduation rate in the community of Nemaska is similar to the Cree average with 11.2% of vocational diplomas and 11.2% for college diplomas, which is significantly lower than the Quebec average.

Community Health and Wellness

Health and social services in all Eeyou Istchee communities are provided by the Cree Board of Health and Social Services of James Bay (CBHSSJB). Each of the nine (9) Cree communities is served by a clinic that offers mainly primary care and a dental clinic.

The community of Nemaska has a Wellness Centre, a social services office, a multiservice day care centre (MSDC) and a Youth office. Looking at the high birth and fecundity rate trends in the population, it is foreseeable that the demand for this type of service will increase in the upcoming years.

Cultural and Archaeological Heritage

A study on the archaeological potential and an archaeological inventory were completed in 2011 and 2012. Even if a consultation of the Inventaire des sites archéologiques du Québec (ISAQ - Archaeological sites inventory database in Quebec) identified three (3) known archaeological sites in the vicinity of the Project, no remains of human establishments prior to the 1950s were identified. Therefore, no significant issues have emerged that would be considered jeopardizing to the Whabouchi Mine Project.

First Nations

Early in the preliminary phases of the development of its Project, NLI devoted time and resources to ensure a concrete and constructive involvement of the various stakeholders, notably the Cree Nation of Nemaska. Even before launching the ESIA process, the local authorities of the Nemaska Cree community took part in information and consultation activities.

Since the project inception in 2009, several activities were undertaken to present the Project to the stakeholders and collect their concerns about the Project and its potential environmental and social impacts. Various subjects were discussed during these information and consultation activities, for example the open-pit mining processes, the main project infrastructure planned and the lifecycle of the ore.

Such consultation led to significant improvement to the site general arrangement plan from the original project concept. The location of the Waste Rock and Tailings stockpile, as well as of sedimentation basins and final effluent, were modified so that they are now located further from Mountain Lake.

In August 2009, the financial arm of the Cree community, Nemaska Development Corporation, agreed to purchase shares in NLI, therefore ensuring their financial participation in the development of the Project.

In the fall 2009, discussions were held on negotiating and signing a Memorandum of Understanding (MOU) between the community and NLI that recognized the respective rights and expectations of the parties, and particularly the need for the company to respect Cree culture and traditions in its activities on the territory. The MOU was signed in August 2010.

The implementation of the Communication and Consultation Plan began in November 2011, after a series of discussions with the Nemaska Band Council administration and an initial presentation of the Project during a community general assembly.

The Communication and Consultation Plan included the completion of various engagement activities with the following stakeholders:

- Nemaska Band Council (meetings);
- Members of the Community of the Cree Nation of Nemaska and local organizations (interviews);
- Trapline R^{2}0 tallyman, neighboring tallymen (R16, R18, R19, and R21) and other Cree land users (in-depth semi-structured interviews, meetings);
- Youth; Elders; Land users, hunters and trappers; and Women (focus groups);
- Community Consultative Committee which was set up to provide a platform for exchanges between the proponent and the various stakeholders in the Nemaska community (meetings).

A 3D video animation describing the main elements of the Project through the construction, operation, and closure phases was made available to the community of Nemaska along with multiple documents and reports, including all those produced as part of the provincial and federal environmental and social impact assessment processes.

Several field visits on the Whabouchi Mine Project site were also organized to explain the Project to the representatives of the Nemaska community and to the Cree land users. Moreover, in March 2012, NLI opened a local office in Nemaska and hired a community liaison agent to facilitate the exchanges of information between Nemaska and the community.

The comments, concerns, demands, and suggestions expressed by stakeholders during the information and consultation activities were documented and compiled by topic. Following the filing of the ESIA in April 2013, NLI and its consultants were still actively involved in the public information and consultation sessions held in Nemaska by the Cree Nation Government's Environment and Remedial Works Department and the Canadian Environmental Assessment Agency (the Agency).

In November 2013, NLI participated to public consultations held by the Agency in Nemaska, and in March 2015, it participated to the public hearings organized by the Quebec's Review Committee (COMEX) also in Nemaska.

In November 2014, the GCCEI, the CNG, the Cree Nation of Nemaska and NLI announced that they have entered into the Chinuchi Agreement regarding the development and operation of the Whabouchi Mine Project. The Chinuchi Agreement is a binding agreement that will govern the long-term working relationship between NLI and the Cree parties during all phases of the Whabouchi Mine Project.

Community approval was expressed through the support of the Chief and Council of the Cree Nation of Nemaska on September 18, 2014. The approval of the Chinuchi Agreement by the GCCEI and the CNG on September 23, 2014, represents the support of the Cree Nation as a whole, and ensures a stable regional environment for the development and operation of NLI's Project.

After extensive and cooperative negotiations, the Cree announced that they were satisfied that concerns expressed regarding a range of issues, including business opportunities, training, and employment, as well as other matters, will be addressed by NLI. The Agreement, which will be in effect throughout the life of the mine, contains items pertaining to training, employment and business opportunities for the Crees during construction, operation and closure of the mine.

In addition, the parties have established a framework to ensure Cree involvement and participation in environmental matters, such as monitoring, mitigation and closure. The Chinuchi Agreement reflects the commitment of the Crees to collaborate with NLI during the development and operations of this new mine in the Eeyou Istchee territory. Moreover, the Agreement aligns the parties' respective interests in the economic success of the Project and ensures that the Crees will receive financial benefits from the Project in compliance with the best practices in the industry and with the Cree Nation Mining Policy.

17.1.4 Waste Rock and Tailings Characterization

Applicable Requirements

The design of the co-disposal facility must meet the requirements of *Directive 019* on the mining industry (MELCC, 2012). The containment measures required for the design of the installation are described in this document. Containment measures depend on the geochemical properties of the waste rock and tailings that will be deposited in the facility.

In June 2020, *the Ministère de l'environnement et de la lutte contre les changements climatiques – now Ministère de l'environnement, de la lutte contre les changements climatiques, de la faune et des parcs (MELCCFP) -* published the Guide to the Characterization of Mine Tailings and Ore (*Guide caractérisation des résidus miniers et du minerai - GCRMM*) (*MELCCFP*, 2020) which details the expectations of the *MELCCFP* for the classification of mine residues. The *GCRMM* also emphasizes the size of a representative sample of each geological unit with distinct geochemical properties for the characterization of mine tailings. If the mining residue is classified as low risk, there are no specific confinement requirements for the design of the disposal facility. If it is classified as acid generating, leachable or radioactive, a level-A confinement design should meet the maximum seepage criterion of 3.3 L/m2/day for the whole facility. If it is classified as «high risk», a level-B confinement design should be implemented which includes a HDPE liner with leak detection systems.

NLI also requires that the CSF design include best practices in the mining residue management as described in the «Towards Sustainable Mining» standards from the Mining Association of Canada (MAC), the Global Acid Rock Drainage Guide (GARD Guide) from the International Network for Acid Prevention (INAP) and the Global Industry Standard on Tailing Management (GISTM) from the International Council on Mining and Metals (ICMM).

Available Geochemical Information

Three (3) environmental geochemistry studies were conducted on the ore, waste rock and process tailings of the Whabouchi mine Project.

In 2011-2012, Lamont Inc. (2013) performed the initial characterization of mining materials as part of the environmental and social impact assessment process. A total of 83 samples were taken from the exploration drill core. Five (5) tailings samples were also provided from metallurgical testing. These samples were analyzed in accordance with *Directive 019* in order to classify mining residue according to their geochemical properties and to provide design criteria for the co-disposal facility.

In 2014, Roche Ltée (2014) conducted an additional assessment to address issues raised by the Environmental and Social Impact Review Committee (COMEX) and the Impact Assessment Agency of Canada (IAAC). The purpose of this study was to quantify rare metals and rare earths in ore and waste rock and to obtain lower analytical detection limits for certain parameters. A total of 18 samples were taken from the sampling intervals that were investigated in Lamont (2013) and reanalyzed.

A research project was undertaken with the Université du Québec en Abitibi-Témiscamingue (*UQAT*) in 2016 to assess the expected seepage water quality. In the first stage of the project, a total of 64 samples were collected on drill cores to produce five (5) composites of the main orebody lithologies which were characterized for mineralogy, elemental chemistry and metal leaching potential with humidity cell and leach column tests. In the second stage of the project three (3) experimental field cells were built on site in October 2017. The purpose of these cells was to simulate the evolution of the tailings and waste rock in a co-disposal set-up and in the climatic conditions of the project site. These cells were monitored until end of summer 2021.

Based on discussions held with *UQAT* personnel in charge of the field cells and with the NLI team, the waste rock material used for the test cell seems the to be run-of-mine waste rock including pegmatite material from the orebody. The waste rock used for the field cells construction is not totally representative of the waste rock material that will be sent to the co-disposal facility. During the construction of the cells fine DMS sinks were used by mistake instead of coarse DMS tailings. As a result, the tailings used are not totally representative of the tailings that will be disposed of at the co-disposal facility. Due to the lack of material, only one bench of the co-disposal cell could be built. Tailings were thus not covered with waste rock as planned in the CSF. As a consequence, the objective to simulate the co-disposal configuration was not fully achieved. The field cell could, however, simulate a worst-case scenario where lithium recovery would be lower than expected at the concentrator and where mine operation experience dilution problems.

Acid Generation Potential

The acid generation potential of ore, tailings and waste rock was first evaluated by static Acid-Base Accounting tests and humidity cell tests in the Lamont 2013 study.

The results of this study can be summarized as follows:

- On average, all the ore and waste rock lithologies were classified as non-acid generating (NAG) due to an average total sulfur content of 0,19% which was below the 0.3% limit. Sulfur was mainly in sulfide state as no sulfate were detected. Larger sulfur content was observed in a few individual samples of basalt (0.6%) and gabbro (1.08%);

- Ore and waste rock samples showed low neutralization potential;

- The majority of the 88 samples were considered as non-potentially acid generating (NPAG): All of the samples of spodumene pegmatite, barren pegmatite, and felsic volcanic rocks were classified as NPAG, while 78% and 77% of the basalt and gabbro samples, respectively, were classified NPAG;

- Five (5) waste rock composites were tested in humidity cells: Average and potential acid generating (PAG) gabbro, Average and PAG basalt and pegmatite to assess further the acid generation potential. After 46 weeks (28 weeks for the average gabbro and basalt) of wet air/dry air injection cycles, the pH of the leachates of all the humidity cells remained circum-neutral, which confirm the low acid generation potential of the tested material;

- Tailings were considered as NPAG due to a total sulfur content lower than 0.01% for all the samples.

Directive 019 criteria have been subsequently replaced by those described in the *GCRMM* (2020): A mine waste is now considered as potentially acid generating when its total sulfur content is higher than 0,04% and one of the two following conditions are met: Net Neutralization Potential (NNP) lower than 20 kg $CaCO_3$/t or Neutralization Potential Ratio (NPR) lower than 2. It is expected that if Lamont (2013) static ABA results were reinterpreted with these new criteria, more samples would be classified as PAG or with an uncertain acidification potential. However, humidity cell test results remain valid.

In the *UQAT* testwork in 2016-2017, weathering cell tests were implemented on six (6) ore and waste rock composites of the Whabouchi deposit. According to URSTM (2015), conditions of the weathering cell weathering tests are more aggressive than the conditions of the humidity cell tests. The results showed that pH remained circum-neutral after 23 weeks of leach cycles.

Content and Metal Leaching Potential

Analysis of available metal content in solid fraction and static leach tests are used as screening tools to evaluate potential chemical elements of concern that may be leached out of the material when stockpiled on site and could impact the surface and ground water quality. To be classified as “potentially leachable”, the mine material has to show exceedance both on solid content (compared with Criteria A of the Guide d’intervention de la Potitique de protection des sols et des terrains contaminés, therafter Guide d’intervention (Beaulieu, 2019) and in leachate concentration (compared with groundwater seepage criteria of the Guide d’intervention) for the same chemical element.

In Lamont (2013), the water leaching test was the Shake Flask Extraction test, but some of the samples were retested with the CTEU-9 protocol in Roche (2014). Thus, it is considered that all of the data from the static leaching tests required by the *GCRMM* are available to determine the metal leaching potential of mining materials and the results of the two studies (Lamont 2013 and Roche 2014) should be interpreted together to meet the requirements of the *GCRMM*.

- Lamont (2013) results showed that the average metal content in ore and waste rock is lower than the background concentrations in the geological Superior province (criteria A of the Politique de protection des sols et des terrains contaminés (PPSRTC) except for copper (Cu). Other exceedances of the background concentrations were observed for cadmium (Cd), cobalt (Co), silver (Ag), barium (Ba) and nickel (Ni) in individual samples but the frequency of exceedance was too low to be representative.

- In Roche (2014), with the reanalysis of samples taken in the same drill core intervals as in Lamont (2013) with lower detection limits, exceedance of Criteria A was confirmed for Cu for most of the samples of waste rock but not for ore. Some other potential significative exceedances were noted for chromium (Cr), Co and Ni in waste rock especially in basalt and gabbro.

- In Lamont (2013), ore, waste rock (gabbro and basalt) and tailings were classified as leachable for Cu only but not ‘at high risks’ under Directive 019 guidelines. This conclusion was confirmed in Roche (2014) for ore and waste rock.

Four (4) different types of kinetic leach test were implemented on the Whabouchi mine material in Lamont (2013) and *UQAT* testwork: weathering cells, humidity cells, leach column and experimental field cells. However, due to specificities in their protocols (e.g., sample size and Liquid/Solid ratio), weathering cells and humidity cell tests are not totally suitable for quantitative evaluation of the metal leaching potential. Weathering cell tests are used as preliminary screening tests to observe the potentially leached out contaminants. Humidity cell tests are primarily used to characterize the acid generation potential. The liquid/solid ratio is considered too high to produce leachates concentration that are quantitatively representative of field alteration conditions and there is a bias toward overestimation of the leaching potential.

UQAT also conducted leach column tests in laboratory on composites of the four (4) waste rock lithologies, on an ore composite, and on sieved ore (fine DMS sinks) from the pilot process and on the two (2) types of process tailings (flotation tailings and coarse DMS tailings) or mixes of them. Based on the results recorded and provided by *UQAT*, pH remained circum-neutral for the ore and the waste rock composites. Leachates were analyzed for conventional parameters (pH, eH, ORP and electric conductivity) and metals by ICP-AES and ICP-MS, the second equipment allowing lower detection limits. Reinterpretation of the test results is ongoing by SNC-Lavalin to determine the leachability of these materials.

The analytical results of the Summer sampling campaigns at the field test cells showed circum-neutral pH values except for the leachate of the waste rock berm at Cell CC where some values were more acidic (pH 5-6). SNC-Lavalin is also working on the reinterpretation of the test results to determine the leachability of these materials. Concentrations of lithium (Li) are observed in the leachate of the tailings (1 to 5 mg/L) whereas in the leachates of the waste rock, Li concentrations are lower than 0.3 mg/L. These high values are most probably linked to the fact that they are composed of a 50:50 mix of sieved ore (Li-enriched) and flotation tailings and by the ore content in waste rock. There is no value for lithium in the Directive 019 final effluent criteria, but this element is indirectly regulated by effluent toxicity monitoring requirement. Based on Quebec’s surface water quality criteria (*MELCCFP*, 2017), the Li criterion for Aquatic Life Protection at effluent (acute effect) is 1.8 mg/L.

<u>Radioactivity</u>

Natural radioactivity is mainly associated with the presence of radioactive isotopes of the uranium (U) and thorium (Th) decay series. These two (2) elements are a common constituent of pegmatites. They are usually present in low concentrations. The activity of K-40 is also taken into account to evaluate the radioactivity of the samples according to the *GCRMM* but generally the contribution of this isotope to the total activity is negligible.

Gamma spectrometric analyzes were undertaken in the Lamont study on 50 waste rock samples from each lithology and five (5) tailings samples. The gamma ray emission of seven (7) isotopes of the U-238 and Th-232 and K-40 decay series was measured. For waste rock samples other than pegmatite, all results were below the detection limit and the total U content was also below the detection limit, confirming that they have no potential for radioactivity. For pegmatite and spodumene pegmatite, the total U content is between 0.8 and 32 ppm. Some samples showed low values of gamma ray emission radioactivity. No samples were analyzed for total Th content.

<u>Conclusions</u>

According to the results from the two (2) geochemical characterizations, and also, partly based on the results gathered up to this date from the ongoing in-situ experimental cells testing, following *Directive 019*, Whabouchi mining residues are classified as:

- Not “high risk”;
- Not potentially “acid-generating”;
- Not potentially “leachable” according to TCLP and kinetic tests results.

- In this context, no groundwater protection measures are required for the waste rock and tailings pile.

On-Going Geochemical Studies

A series of geochemical review and analysis is being conducted by SNC-Lavalin as a complementary geochemical assessment of the Whabouchi Project.

First step is to review of the number of existing lithologies with distinct geochemical properties based on NLI's geological model, integrating all available the analytical and geochemical results. If the presence of a lithology with distinct geochemical characteristics is identified or if the number of samples is insufficient, recommendations will be issued to proceed with complementary sampling and analysis. Once all the required data is available, it will be used to classify the associated tailings and waste rock with regards to their metal leaching potential.

Testing results from the *UQAT* previous geochemical program is also being assessed in comparison with applicable regulations criteria in *Directive 019* and *GCRMM*.

Process water should be chemically characterized as soon as representative samples are available. This water will be present in the pores of the filtered process tailings deposited at the co-disposal facility. Its quality will directly impact the quality of exfiltration water of the facility on the short and medium terms.

Concerning the testing of the kinetic leaching behavior of waste rock and process tailings, the implementation of additional field tests is considered to reduce uncertainties and thus risks on their potential environmental impact. A protocol for in-situ column tests is being developed to be implemented at Whabouchi site. In-situ column tests are carried out in barrels where crushed drill core composites or process tailings are placed. The top extremity of the barrel is open exposing the sample to the local climatic conditions. Leachates are collected at the bottom of the barrel. This test is well-suited when low quantities of material are available and has a relatively low construction cost. It requires the presence of employees on site on a regular basis for the monitoring during the non-freezing period. They will be run on several years to obtain pertinent data.

17.2 Jurisdictions and Applicable Laws and Regulations

The legal framework for the construction and operation of the Whabouchi projected mine facilities is a combination of provincial, national, and municipal policies, regulations, and guidelines. The design and the environmental management of the Mine Project facilities and activities must be done in accordance with this legal framework. Outlined below are the major steps through which NLI went or will have to go through as Whabouchi Mine Project development will move forward.

17.3 2013 Modifications to The Quebec Mining Act

The Quebec government is responsible for mining activities in the province. This activity is subject to the Mining Act which defines Ownership of the right to mineral substances (claims, mining exploration licenses, mining leases, mining concessions, etc.) and the rights and obligations of the claim holder or other mining right granted by the State. The Act was substantially amended and modernized by Bill 70, which the Quebec National Assembly adopted on December 9, 2013. This fourth attempt to update Quebec's mining legislation follows on the heels of the defeat of Bill 43 and that of Bill 79 and Bill 14 in previous legislative sessions.

Within the specific context of the Whabouchi Project, the following amendments are relevant:

- Provisions specific to aboriginal communities and referring to an aboriginal community consultation policy specific to the mining sector (obligation to consult aboriginal communities and requirement that the Minister consult aboriginal communities separately if the circumstances so warrant);

- On each anniversary date of a mining lease or mining concession, the lessee or grantee will have to send the Minister a report showing the quantity of ore extracted during the previous year, its value, the duties paid under the Mining Tax Act during that period and the overall contributions paid;

- Mining leases to be issued will require the prior approval of a rehabilitation and restoration plan and the issuance of a General Certificate of Authorization (CA) under the Environment Quality Act (EQA), unless the time needed to obtain a certificate is unreasonable.

It should be noted that with regards to these specific amendments, Nemaska Lithium obtained its mining lease following the adoption of Bill 70 and thus is fully complying with the Act.

With regards to the environment, it should be noted that the Mining Act, Chapter IV, Division III, specifies that the holder of mining rights has the responsibility to rehabilitate and restore the lands on which exploration and/or development activities have been carried out. This work must be completed in accordance with the restoration plan pre-approved by the Ministère des ressources naturelle et des forêts (*MRNF*). Under the Mining Act, the Regulation respecting mineral substances other than petroleum, natural gas and brine details certain procedural requirements of the Act, particularly in terms of the information and documents to be provided to the *MRNF* on restoration measures and locations established to store mine tailings.

On July 23, 2013, the Government of Quebec passed amendments to the Regulation respecting mineral substances other than petroleum, natural gas, and brine in order to set new rules concerning the financial guarantees required for the restoration of mining sites. Among other things, those result in an increase of the financial guarantee from 70% to 100% of the projected costs for the work required under the rehabilitation and restoration plan. The guarantee must cover not only restoration costs associated with the accumulation areas, but all costs for the entire mine site and associated infrastructure. It must be paid in three annual installments.

The first installment corresponds to 50% of the total amount of the guarantee and must be paid within 90 days following the receipt of the approval of the plan. The second and third installments each represent 25% of the guarantee and must be paid in full by the first and second anniversary date. To that regard, in June 2019, NLI paid the last installment of the financial guarantee required under the approved Rehabilitation Plan for the Whabouchi Mine Project.

The five (5) year statutory review of the closure plan was submitted in February 2021 and approved by *MRNF* in February 2022. Three (3) new installments are required to cover the increase in the cost estimate of the closure plan. The first two installments are paid and the last one due in February 2024. NLI is thus fully compliant with the applicable Regulation.

Lastly, the Regulation respecting Environmental Impact Assessment and Review was amended to require an environmental impact assessment for all metal ore processing plant construction projects, and all metal mine openings and operation projects where the processing or production capacity of the plant or the mine is 2,000 metric tonnes or more per day. However, it should be noted that since the Whabouchi Mine Project is located in the territory governed by the James Bay and Northern Quebec Agreement (JBNQA), this last modification does not apply to it. Indeed, on that territory, all mining projects were already subject to the environmental impact assessment process, as outlined in the following section.

On December 31, 2015, amendments to the Regulation were also adopted in order to require mining project proponents to establish, within 30 days following the issuance of the mining lease, a monitoring committee to facilitate the participation of the local community. The further-described Chinuchi Agreement signed in 2014 by NLI and its Cree First Nation partners includes the implementation of committees which complies with this requirement.

17.3.1 Quebec Procedure Relating to the Environmental Assessment of the Project

The Quebec EQA comprises two (2) Chapters. Chapter I gives general provision and Chapter II gives provisions applicable to the Eeyou Istchee/James Bay and Northern Quebec Region in accordance with the JBNQA, signed by the Native peoples of the northern regions.

The environmental assessment procedures established for northern projects vary according to whether the project is located south or north of the 55th parallel.

Section 133 of the EQA defines the territory south of the 55th parallel as: "the territory bounded to the north by the 55th parallel, to the west by the boundaries of Ontario and of the Northwest Territories, to the east by the 69th meridian and to the south by a line that coincides with the southern limit of the middle zone and the Cree traplines located to the south of the middle zone, as determined under the Act respecting hunting and fishing rights in the Eeyou Istchee/James Bay and New Quebec territories (chapter D-13.1), as well as to the Category I and II lands for the Crees of Great Whale River."

The Whabouchi Mine Site is located within the limit of the territory described above, also referred to as Eeyou Istchee/James Bay Region, or Eeyou Istchee for the Cree First Nation.

Section 153 of the EQA gives the list of the projects automatically subject to the environmental and social impact assessment (ESIA) and review procedure (as listed in Schedule A) and the project which are automatically exempt from that procedure (as listed in Schedule B).

The list of projects automatically subject to the procedure includes:

- All mining developments, including the additions to, alterations or modifications of existing mining developments.

The Whabouchi Mine Project is therefore subject to the procedure.

Section 154 of the EQA specifies that: "No person may undertake or carry out any project which is not automatically exempt from the assessment and review procedure, unless:

- A Certificate of Authorization has been issued by the Minister, after the application of the assessment and review procedure; or.
- An attestation of exemption of the project from the assessment and review procedure has been issued by the Minister."

The first step in the ESIA process involves the proponent gathering preliminary information on the project. The proponent must submit a Project Notice to the government administrator along with this preliminary information. Such Project Notice was tabled by NLI on August 2, 2011.

The Administrator sends this preliminary information to an Evaluation Committee (COMEV) which is responsible for defining the nature and extent of the impact study (ESIA). The COMEV formulates guidelines outlining the extent of the ESIA document to be prepared by the proponent. The guidelines are submitted to the administrator who transmits them to the proponent, something that was done on February 2, 2012.

The proponent prepares the ESIA document in accordance with the administrator's guidelines. Nemaska Lithium submitted its ESIA document to the Administrator on April 2, 2013, who forwarded the studies to a Review Committee (COMEX). First Nations, i.e., the Cree Nation Government, and the public then made representations to the committee, which may decide to hold public hearings or any other type of consultation. The COMEX held public hearings in March-April 2015 as well as other forms of consultation, enabling the Committee to consider the concerns of the people in the territory and ensure they were accounted for in the Whabouchi Mine Project and reflected in the General CA.

On September 4, 2015, following a positive recommendation by the COMEX, the Administrator granted authorization for the Project and NLI announced that it has received the General CA for the Whabouchi Project from the *MELCCFP*. NLI has already begun and is continuing to fulfill the provisions included in the General CA.

17.3.2 Federal Procedure

The Canadian Environmental Assessment Act (CEAA 2012) was introduced on July 6, 2012. Under this Act, an Environmental Assessment (EA) focuses on potential adverse environmental effects that are within federal jurisdiction, including:

- Fish and fish habitat;
- Other aquatic species;
- Migratory birds;
- Federal lands;
- Impacts that will or could potentially cross provincial or international boundaries;
- Impacts on Aboriginal peoples, such as land use and traditional resources;
- Impacts that are directly linked or necessarily incidental to any federal decisions about a project.

An EA will consider a comprehensive set of factors that include any cumulative effect, mitigation measure and comments received from the public.

The Regulations designating Physical Activities (Regulations) identify the activities that are subject to the federal environmental assessment procedure under the Canadian Environmental Assessment Act, 2012 (CEAA 2012) by the Canadian Environmental Assessment Agency (hereafter the Agency) or by the Canadian Nuclear Safety Commission or the National Energy Board. The Regulations identify types of major projects that may require an environmental assessment under the CEAA 2012. These projects have the greatest potential for significant adverse environmental effects in areas of federal jurisdiction and are called “designated projects”.

According to the Regulations, the construction, operation decommissioning and abandonment of a “metal mine, other than a rare earth element mine or gold mine, with an ore production capacity of 3,000 t/day or more” is subject to the federal environmental assessment procedure.

According to the CEAA 2012, proponents of designated projects are required to submit a description of the designated project to the Agency to inform on whether or not an EA of the designated project is required. The project description for the Whabouchi Mine Project was tabled by NLI on December 14, 2012.

After having approved the Project Description and determined that an EA is required, the Agency posted on January 29, 2013, a Notice of Commencement of the EA on the registry. The Agency then prepared a preliminary version of the guidelines relative to the environmental impact assessment. These guidelines were posted on the registry on January 29, 2013, allowing the public to comment on the proposed studies and methods as well as on the information that will be required for the environmental impact assessment.

The Agency took into account the general public’s comments, including the observations made by Aboriginal groups and federal ministries before providing the final version of the environmental impact assessment guidelines to the Proponent, which it did in April 2013.

The Proponent then has to submit to the Agency an environmental impact assessment identifying the environmental effects of the project and propose measures to mitigate these effects, while accounting for the Agency’s guidelines. NLI tabled its Environmental Impact Statement (EIS) in March 2013.

Following the submission of the EIS to the Agency, the latter will ensure of its relevancy and accuracy. The Agency may require that the proponent provides further clarifications or additional information to better understand the potential environmental effects and the proposed mitigation and preventive measures. Such additional information was requested by the Agency in late November 2013 and NLI provided that information on May 5, 2014. The Agency may also decide to hold public hearings, something which was completed in November 2013.

Following the completion of its analysis, the Agency prepares a preliminary version of the EA report, which includes the Agency's conclusions on the potential environmental effects of the Project, the proposed mitigation measures, the significance of the residual adverse environmental effects of the Project and the requirements of the monitoring program. The Agency then invites the public to comment on this preliminary report before finalizing it and submitting it to the Minister of Environment. The preliminary EA report was issued by the Agency on May 6, 2015.

On July 29, 2015, following a comprehensive assessment of the Whabouchi Mine Project, the Canadian Minister of Environment decided that the Project is not likely to cause any significant adverse environmental effects, and set out in her positive decision statement the conditions relative to the mitigation measures and monitoring program to be respected by NLI. The Agency issued on that same date its final EA report. NLI has already begun and is continuing to fulfill the provisions included in the Decision Statement.

It should finally be noted that on June 2, 2019, Bill C-69 ("An Act to enact the Impact Assessment Act and the Canadian Energy Regulator Act, to amend the Navigation Protection Act and to make consequential amendments to other Acts") received Royal Assent. The new Impact Assessment Act will overhaul both the National Energy Board Act (NEBA) and CEAA 2012, changing how major infrastructure projects are reviewed and approved in Canada. Changes would include replacing the National Energy Board with a new "Canadian Energy Regulator" and an altered federal environmental assessments process to include a broad range of impacts to be reviewed by a new "Impact Assessment Agency." However, since the Whabouchi Mine Project was approved in 2015 under CEAA 2012, this new Bill will have no effect on the project development and permitting requirements at the federal level.

17.3.3 Metal and Diamond Mining Effluent Regulations (MDMER) and Environmental Effects Monitoring Program (EEMP)

The EEMP is a requirement for regulated mines in accordance with the MDMER under the authority of the Fisheries Act. The objective of the EEMP is to evaluate the effects of mine effluents on fish, fish habitat and the use of fisheries resources by humans. *Directive 019* sets at the provincial level the criteria that mine effluents must comply with at the end-of-pipe. The EEMP examines the effectiveness of the environmental protection measures directly in the aquatic ecosystems, i.e., downstream of the final discharge point. The EEMP consists of biological monitoring studies as well of effluent and water quality studies. In the case of Whabouchi Mine Project, the monitoring will have to include the following elements:

- Effluent characterization;
- Effluent sublethal toxicity testing;
- Water quality monitoring;
- A study respecting the benthic invertebrate community.

The requirement of an EEMP is to be reviewed as more information is collected and when a better assessment of the impact of effluents on the aquatic environment is available.

Finally, it should be noted that the Whabouchi Project was designed in a way that no fish habitats will be directly impacted by the implementation of any accumulation areas (no encroachment in fish habitat). Consequently, those infrastructures will not have to be listed in Appendix 2 of the MDMER.

17.3.4 Environmental Permitting

Even though the Project underwent an environmental impact assessment and was authorized by the Government pursuant to the EQA and CEAA, it is still subject to other sections of the EQA, the Mining Act and to other applicable provincial and federal laws and regulations. Indeed, in addition to the authorizations required under Section 22 of the EQA, the proponent must obtain the permits, authorizations, approvals, certificates, and leases required from the appropriate authorities. Those are described in Table 17-1.

As well, along with the mitigation measures set out as part of the environmental impact assessment, the final Project design must comply with all applicable standards relating to the proposed infrastructure and equipment.

It must also be noted that the issuance of the certificate of authorization required under Section 22 of the EQA, however, should only be a formality as the certificate issued pursuant to the approval of the ESIA binds the Minister as to where he exercises the powers provided in Section 22. The authorization application and permitting process is expected to last the full construction phase and has started in Q1-2016. Applications are being filed in a timely manner with the construction works and have therefore no impact on Project schedule.

17.3.5 Recent Modifications to The Quebec Environment Quality Act (Eqa)

On March 23, 2017, the Quebec National Assembly passed Bill No. 102, entitled An Act to amend the Environment Quality Act to modernize the environmental authorization scheme and to amend other legislative provisions, in particular to reform the governance of the Green Fund (EQA 102). The adoption of EQA 102 follows on from the publication of the Green Paper in June 2015 and the introduction of Bill 102 in June 2016.

The new Environment Quality Act (new EQA) entered into force on March 23, 2018. This date marks the beginning of the progressive implementation of a new environmental authorization scheme in Quebec. The majority of associated regulations are currently being amended and will come into force progressively in the coming months or years. In the meantime, the Regulation respecting certain measures to facilitate the carrying out of the Environment Quality Act and its regulations helps to link the new authorization scheme with current regulations.

One of the most important amendments to the new, EQA is the establishment of a simplified and modulated authorization process based on environmental risks. Henceforth, projects, and the authorizations they require, will be classified in four (4) categories, according to the degree of risk they entail, namely:

High-Risk Activities subjected to the environmental impact assessment and review procedure before any authorization is granted by the Government of Quebec. Those activities are designated under the new Regulation respecting the environmental impact assessment and review of certain projects that came into force on March 23, 2018.

Moderate-Risk Activities, designated by Regulation (not yet issued), subjected to prior ministerial authorization, such as those historically provided for by the former EQA (ex. hazardous materials permit; authorizations under Sections 32, 48, etc.). These are activities designated by the EQA and the applicable regulations, as well as those which, although not specifically so designated, are not considered to be of a low or negligible risk nature. They will be subject to prior ministerial authorization.

Low-Risk Activities, designated by Regulation (not yet issued), that must be disclosed in a declaration of compliance, at least 30 days before the activities commence (ex. extensions to waterworks or sewer systems and certain rehabilitation works on contaminated lands).

Negligible-Risk Activities, which necessitate no prior ministerial authorization or prior declaration of compliance. An Instruction Note was published on that point in April 2019 and is entitled “Activités à risque négligeable - Listes des exemptions administratives de l’application des articles 22 et 30 de la LQE”. However, it must be pointed out that it is an administrative document and that it has no official value.

With few exceptions (e.g., water treatment and water withdrawal activities), municipal certificates of compliance that were required in the past to support authorization applications under the EQA are no longer required. Copies of authorization applications submitted to the *MELCCFP* must, however, be submitted to the municipality concerned as required under Section 23 of the new EQA.

Information concerning authorization applications and issued authorizations will also be more readily accessible, since the new EQA expands the required content of the environmental registers to be kept by the *MELCCFP* and provides that certain information shall be accessible to the public, such as the authorizations and documents that are an integral part thereof, the studies and other analyses submitted by the applicant on which authorizations were based, except, however, information that constitutes industrial or trade secrets as provided for in the EQA.

17.3.6 Future Modifications to Initial Permits and Authorizations

As described earlier in this Report, there will be a need to increase the area dedicated to waste rock and tailings management as part of the normal operations over the 34-year mine life. In order to do so, additional permitting will be needed since the initial General CA and other regional authorizations were obtained to cover the Project as it was described prior to the current Feasibility Study. However, such modifications (as provided for under section 30 of the new EQA) can be obtained in a timely manner without delaying by any mean the normal mine operations, especially considering that the already-permitted capacity enables mining operations to take place over a significant period of time.

Most of the required permits were obtained starting in 2016 during the construction period before the project was halted by the CCAA procedure in 2019. This includes, among many others, the general certificate of authorization as well as the authorization for the mine and co-disposal operation. There was a lingering issue with the validity of permits for drinking water and waste water treatment systems that was resolved in 2021.

Several minor changes to the CA were requested by NLI and authorized by the COMEX and the *MELCCFP*. The IAAC continues to monitor the acceptability of changes and operations with reference to the Decision Statement. The last request for change was suspended in 2019 when NLI went under the CCAA.

The Whabouchi Mine project now have some minor differences from the previous one. Some of those changes require modification requests to the already granted permits. Meetings with MELCC and IAAC helped identify which changes are sufficient to warrant a permit modification request to the initial General CA and other regional authorizations that were obtained to cover the Project as it was described prior to the current Feasibility Study. However, such modification requests now provided for under Section 30 of the new EQA can be obtained in a timely manner.

The list of changes and permit requests is detailed in Table 17-1. In summary, the Project for the most part is already approved; however, certain CAs will need to be amended due to some modifications to the Project as outlined in Table 17-1.

Table 17-1 Environmental Permit Strategy

ID	Permit Name	Regulation	Law Article	Description
102b	Operation of the Whabouchi mine (obtained to modify)	100-MELCC	LQE Article 22	Request a deadline: activities carried out in wetlands and waterways must start within 2 years of the date of issue of this authorization. Otherwise, the authorization for these activities is automatically canceled. So max. 2022-09-05 (phase I of the co-disposal facility is affected by this limitation). Submit the other modifications, if required
614	Operation of a wastewater treatment system - Camp - Whabouchi site	600-GREIBJ		Certificate of compliance
100a	CAMOD 7 - Relocation of the temporary construction camp on the mining site	100-COMEX	LQE Article 22	In accordance with the commitment mentioned in your correspondence of May 12, 2020, NLI must submit a request for modification of a certificate of authorization for a new construction camp when the future of the project is better known.
208	Closure plan for the Whabouchi mining sit	200-MERN	Mining Act Article 232.6	Next five-year Review Due for February 10, 2027
100	Condition 5 - Land occupation	100-COMEX	LQE Article 22	Condition fulfilled but update will be necessary
124	Condition 17 - Residual materials	100-COMEX	LQE Article 22	File to be followed with the Cree. The requirements have been met, subject to the approval of the new place of trench landfill proposed by the Cree Nation of Nemaska. Informed of any significant change made to the management method of residual materials in the future.
122	Conditions 18 and 19 - Emergency measures plan	100-COMEX	LQE Article 22	COMEX considers that the requirements of condition 18 have been fulfilled, however certain elements will have to be completed in the next revised version.
109	Mining wastewater treatment plant (UTM)	100-MELCC	LQE Article 22	The mineral wastewater treatment plant will be set up and functional from the start of mining and for all operating periods.
113	Request for authorization for the operation of an industrial establishment (old pollution reduction certificate)	100-MELCC	LQE Article 22	113 100-MELCC LQE Article 22 Must be deposited 30 days max. Following the start of mining operations
104	Installation of a final effluent pipe in the Nemiscau river	100-MELCC	LQE Article 22	The work relating to the establishment of the aquatic portion of this pipe was completed at the end of August 2018. We may have to file another request to complete the land portion of the work.
	Drilling campaign	300-MFFP		Renew the intervention permit before April 30, 2022

ID	Permit Name	Regulation	Law Article	Description
	Relocation of the permanent camp	100-COMEX	LQE Article 22	Get the new location approved.
	Co-Disposal facility	100-COMEX		Transmit the modifications in connection with the deposition plan 100
	The construction and operation of the waste rock and mining residue facility, phase 2, as well as the water management associated with it	100-MELCC		As included in the Mining Request ID 102B Transmit the modifications in connection with the Deposition Plan
	Installation of a drinking water production and distribution system at the camp and the permanent living base	100-MELCC		included in the Mining Request ID 102B
	Development and operation of the domestic wastewater treatment system for the permanent camp	100-MELCC		As included in the Mining Request ID 102B
	Connection of buildings to raw water, drinking water and domestic water systems	100-MELCC		As included in the Mining Request ID 102B
	Construction of a new pad for explosives Construction of a washing bay to be built			Modification of mining and other operations to be validated or new request
	Extension to the concentrator and the installation of a ball mill for the concentrate			Modification of mining and other operations to be validated or new request
	Overpass (bridge and access road)			Modification of mining and other operations to be validated or new request
	New loading bay			Modification of mining and other operations to be validated or new request
300	Intervention permit in a forest environment for mining activities	300-MFFP	Mining Act Article 213	To be confirmed if possible to include the entire mining site. Extend Nemaska Lithium's intervention permit for the entire site by completing the form. Ref. MFFP: 3024549 (old permit) N / ref.: 107034.005-312
	Transshipment Centre - Matagami			Modify all the CAs that mention the Chibougamau transshipment center. City of Matagami will apply for authorization for this centre
	Conditions 10 and 1	100-COMEX		Submit a modification of the communication plan

NLI is continuing to fulfill the provisions included in the General CA as well as the DS for Whabouchi, since those apply to all project phases from construction to site restoration. Whabouchi regional construction permits have now expired and will need to be renewed once plans to restart construction are confirmed.

As described earlier in this Report, there will be a need to increase the area dedicated to waste rock and tailings management as part of the normal operations over the 34-year mine life. In order to do so, additional permitting will be needed since the initial General CA and other regional authorizations were obtained to cover the Project as it was described prior to the current Feasibility Study.

17.4 Rehabilitation and Mine Closure Plan

Section 232.1 of the Mining Act states that:

"*The following persons must submit a rehabilitation and restoration plan to the Minister for approval and carry out the work provided for in the plan:*

- *every holder of mining rights who engages in exploration work determined by regulation or agrees that such work be carried out on the land subject to his mining rights;*
- *every operator who engages in mining operations determined by regulation in respect of mineral substances listed in the regulations;*
- *every person who operates a concentration plant in respect of such substances;*
- *every person who engages in mining operations determined by regulation in respect of tailings.*

The obligation shall subsist until the work is completed or until a certificate is issued by the Minister under Section 232.10."

As stated in Section 101, "*the [mining] lease cannot be granted before the rehabilitation and restoration plan is approved in accordance with this Act, and the CA mentioned in Section 22, 31.5, 164 or 201 of the EQA (chapter Q-2) has been issued*."

Hence, an initial rehabilitation plan was prepared as part of the Project and approved by the *MRNF* in September 2017. Theses rehabilitation and restoration plans were elaborated in accordance with the provincial Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements (2016) which provides to the proponents the rehabilitation requirements. This study accounted for costs of all works needed for the rehabilitation of a mining site following the Regulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine. Mine rehabilitation and closure costs, as approved by the *MRNF*, are estimated at C$9.2 M.

In June 2019, NLI paid the last installment of the financial guarantee required under the initially approved Rehabilitation Plan for the Whabouchi Mine Project. The Mining Act include a five (5) years statutory review of the closure plan. In February 2021, NLI submitted to *MRNF* the updated closure plan. It was approved by *MRNF* in February 2022 and the new approved cost estimate is now C$15.0 M. As requested by *MRNF*, a first installment of C$3.0 M was paid in May 2022 and two (2) installments of C$1.4 M will be paid in February of 2023 and 2024. NLI is, thus, fully compliant with the applicable Regulation.

17.4.1 General Principles

The main objective of mine site rehabilitation is to restore the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring the public safety;
- Limiting the production and circulation of substances that could damage the receiving environment and trying to eliminate long-term maintenance and monitoring;

- Restoring the site to a condition which is visually acceptable to the community;
- Reclaiming the areas where the infrastructures are located (excluding the accumulation areas) for future use.

Specific objectives are to:

- Restore degraded environmental resources and land uses;
- Protect important ecosystems and habitats of rare and endangered flora and fauna, which favors the re-establishment of biodiversity;
- Prevent or minimize future environmental damage;
- Enhance the quality of specific environmental resources;
- Improve the capacity of eligible organizations to protect, restore and enhance the environment; and
- Undertake resource recovery and waste avoidance projects and prevent and/or reduce pollution.

The general guidelines of a rehabilitation plan include:

- Favouring a progressive restoration to allow for a rapid re-establishment of biodiversity;
- Implementing a monitoring and surveillance program;
- Maximizing recovery of previous land uses;
- Establishing new land uses;
- Promoting habitat rehabilitation using operational environmental criteria;
- Ensuring sustainability of restoration efforts.

The mine site rehabilitation plan focuses on land reclamation, reclamation of tailings area and water basins, and of surface drainage to prevent erosion. The successful completion of a rehabilitation plan will ensure that the Whabouchi Mine Project will result in a minimum of disturbance. Site inspections will be carried out before the Property is returned to the Government.

The rehabilitation concept for the current mine project is described below and complies with the requirements described in the Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements and the current legislation.

17.4.2 Mining Site Rehabilitation Plan Concept

The rehabilitation and restoration plan concept is summarized as follows:

- Waste Rock and Tailings Pile:
  - Exposed surfaces of the accumulation areas (waste rock and tailings piles, overburden piles) will be covered with a layer of topsoil/overburden and revegetated when feasible. The production of filter pressed tailings co-disposed with coarse waste rocks will enable the progressive revegetation of the pile during operation, thus limiting potential dust emissions and runoff.
- Haul Roads:
  - Surface will be scarified and revegetated.
- Industrial Complex and Buildings:

- No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During dismantling works, beneficiation/recycling of construction material will be maximized. Remaining waste will be disposed of in an appropriate site:
    - All equipment and machinery will be disposed of or recycled off-site;
    - The explosives magazine, if any, and related facilities will be dismantled;
    - The drinking water supply and domestic wastewater treatment facilities will be dismantled;
    - Infrastructure relating to electricity supply and distribution will be dismantled with the exception of Hydro-Québec requirements;
    - All underground services (power lines, pipelines, water, and sewer pipes, etc.) shall remain in place since they are unlikely to cause any environmental damage. Openings and access to such pipelines, however, shall be sealed.

17.4.3 Open Pit

The surface exploitation of a mineral substance is common in Quebec. Many open pits that are created to extract a mineral substance or ore are therefore found throughout the province. Unlike quarries that are essentially developed on rock outcrops, ore deposits can be located below the surface, which means pits could be filled with groundwater. In many open pit mines, water could rise to the overburden contact without the dewatering wells.

Once mining activities cease, the pit will gradually fill up to its equilibrium level with rainfall and groundwater. The overburden slope around the pit will have already been established for a safe operation of the mine. No special work in this regard will be required upon the cessation of mining activities.

To permanently close pit access roads, an embankment 2-m high will be built using waste rock, along with an equivalent crest line. A ditch 2-m wide and 1-m deep will be excavated in front of the embankment.

17.4.4 Environmental Aspects

- Drainage:
    - Whenever possible, the surface water drainage pattern will be re-established to a condition similar to the original hydrological system. The open pit discharge will flow towards the natural low point where an outlet will be implemented.
- Topsoil Management:
    - During the site construction period and overburden stripping over the orebody, overburden and topsoil will be stored separately and used for revegetation purposes.
    - Slopes of the overburden storage area and flat surfaces will ultimately be seeded and revegetated.
- Waste Management:
    - Waste material from demolition activities will be:
        - Decontaminated when required;
        - Recycled when cost-effective;
        - Buried in an appropriate site.
    - All non-contaminated waste will be sent to an appropriate site.

- Hazardous Materials:
  - Facilities containing petroleum products, chemicals, solid waste, hazardous waste, and/or contaminated soil or materials will be dismantled and managed according to regulatory requirements.
  - All hazardous waste will be managed according to existing laws and regulations and will be transported off site.
- Characterization Study:
  - The Land Protection and Rehabilitation Regulation, which came into force on March 27, 2003, contains several provisions concerning land protection in the new Section IV.2.1 of the EQA. The term "land" also includes groundwater and surface waters. The Regulation sets limit values for a range of contaminants and specifies the categories of targeted commercial or industrial activities. The mining industry is one of the categories subject to the Regulation.
  - For the mining industry, this generally entails an undertaking of a site characterization study within six (6) months following the termination of the mine operations. In cases where the contamination was to exceed the criteria set for in the Regulation, a rehabilitation plan which would specify the environmental protection measures to be undertaken must be submitted to the *MELCCFP* for its approval.
  - Waste rocks and mine tailings are not soils and are not covered by this Regulation. The characterization study will address the areas that are likely to have been contaminated by human activities, specifically the handling of petroleum products.

17.4.5 Monitoring Program and Post-Closure Monitoring

According to *Directive 019*, a Monitoring Program will have to be implemented during the mine operation to account for all the requirements specified in that Directive, especially with regards to noise levels, vibrations, surface, and ground waters.

- Physical Stability:
  - The physical stability of dams and of the Waste Rock and Tailings pile will need to be assessed, and signs of erosion will be noted;
  - This monitoring will be conducted on an annual basis for a minimum of five (5) years following mine closure.
- Environmental Monitoring:
  - Monitoring of the water quality (surface and groundwater) will continue for a minimum of five (5) years after the completion of the restoration work.
- Agronomic Monitoring:
  - The agronomic monitoring program is designed to assess the effectiveness of the revegetation which will be done as part of mining rehabilitation effort.

To document the success of the revegetation efforts over the accumulation areas, an agronomic monitoring will be undertaken following the establishment of a vegetative cover on the areas subject to the progressive restoration program. This monitoring will be conducted annually for three (3) years following the revegetation efforts. Reseeding will be carried out, as required, in areas where revegetation is found unsatisfactory.

17.5 **Water Management**

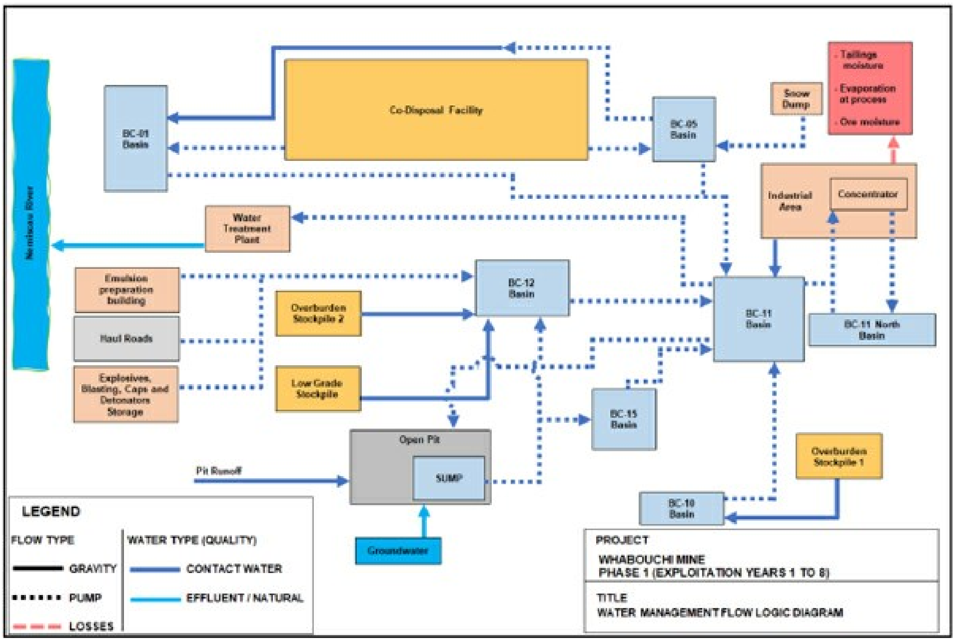
17.5.1 Water Management Strategy and Infrastructure Design

Water management strategy and infrastructure design for Whabouchi mine have been completed in accordance with recommendations from Quebec's *Directive 019* (*MELCCFP*, 2012) and Canada's Environmental Code of Practice for metal mines (ECCC, 2009), and in compliance with Quebec's Environmental Discharge Objectives (EDO) determined for the Whabouchi Mine by the *MELCCFP*. All seepage and runoff water generated on areas impacted by mining activities is considered as "contact water." Contact water and water from pit dewatering activities will be collected and retained for settlement of sediment and treatment prior to being released to the environment. The water management strategy specific to Whabouchi mine includes the following:

- Divert natural runoff from areas not affected by mining activities to limit mixing of natural water and contact water;
- Collect all contact water from areas impacted by mining activities;
- Prioritize the reclamation of contact water collected on sumps to fulfill the process plant water needs;
- Limit the risk of non-treated water discharge to the environment, by designing water management infrastructure to at minimum safely manage floods of 100-year return period;
- Have one single effluent point (Nemiscau River).

Figure 17-2 presents the Whabouchi Mine water management flow logic diagram.

Figure 17-2 Whabouchi Mine Water Management Flow Logic Diagram



17.5.2 Water Diversion

Most of the Whabouchi mine infrastructure such as waste rock and tailings co-disposal storage facilities (CSF), open pit, mill, garage, and camp, are located on high topographic areas, close to natural watershed divide, limiting the amount of required diversion ditches or channels.

Water diversion needs for current projected site layout is limited to the area North of CSF, where a diversion channel will be constructed. The design of access and haul roads include the installation of culverts or longitudinal drainage ditches allowing for diversion or free passage of natural runoff.

17.5.3 Collection Ditches and Sumps

Collection ditches will be constructed surrounding the CSF, the overburden stockpile and temporary ore stockpile, south of the mine pit and around the mill and garage area to collect and safely convey contact water to water sumps. Haulage roads will also have contact water collection ditches. Contact water will be managed at sumps strategically located at site low drainage points from where it will be continuously recirculated to the ore process plant or stored for future use.

17.5.4 Site Water Balance Model

A deterministic water balance model simulates average, wet, and dry historical climate conditions (also including climate change projections) to support the water management strategy defining monthly targets at water sumps and estimating water treatment and effluent discharge needs.

17.5.5 Water Treatment and Effluent Discharge

Excess contact water collected on sumps will be sampled and treated at a water treatment plant (WTP) if required prior to its discharge as an effluent to Nemiscau River via an already constructed outfall sewer, designed to discharge a maximum flow rate of 500 m³/h.

Monitoring at the final effluent will take place on a continuous basis as per applicable regulations to ensure full compliance with all applicable quality criteria.

17.5.6 Climate Change Considerations

Over time, changes to long-term trends or extreme climate events may exceed infrastructure design parameters, or changes to the mine operations may decrease the original design capacity. The effects of climate change will be greatest for water management structures with a long service life, such as structures that will remain at the closure phase (not yet designed) and will depend on the robustness of the initial design and the vulnerability/sensitivity of the downstream environment that could be impacted by the failure or the under design of a particular structure. The mine life of Whabouchi mine is currently estimated to be about 34 years. All water management infrastructure was currently designed to withstand extreme events based on statistical analyses using historical climate data. The climate change effects during operations will likely be absorbed by the freeboard considered on ditch and sump design (minimum freeboard of 0.3 m for ditches and 1.0 m for sumps).

Nevertheless, to address the growing awareness that a changing climate and its impacts on the mine water management strategy, climate change projections for the Whabouchi mine was accessed on the climate analysis, and climate change projections were considered on the site water balance model to evaluate the capacity of the current designed infrastructure to appropriately manage water on site.

The potential for climate change to affect water management infrastructure needs to be reassessed during the operation of the mine as new climate data become available and based on updated climate change projections.

17.6 **Waste and Filtered Tailings Management**

CSF Phase 1 is located between the Route du Nord (north) and the Hydro-Québec 735-kV high-voltage power lines (south). The co-deposal pile has an approximate capacity of 13.1 Mm³, which is sufficient to contain the waste rock and the filtered tailings for 6.5 years of production, with a footprint of 50.2 ha and a top elevation of 354 m. The volume was assessed based on the optimized pit shell. A typical cross-section of the CSF (Phase 1) concept is presented in Figure 17-3.

The CSF would be expanded following the sequence of the starting berm (Figure 17-3; sequence 1A, 2A, 3A, and so on). Distinct zones are identified for stockpiling waste rock and filtered tailings; gray for waste rock, and beige for filtered tailings mixed with ore sorter reject. A transition zone would be implemented, if needed, between waste rock and filtered tailings. Each bench of waste rock would be approximately 3 m (height of each lift could change during the deposition). The perimeter access road, having a 22 m width, would be constructed on the first bench of the CSF.

The waste rock section covers the filtered tailings piles, in accordance with the following design criteria:

- Overall slope: 3H:1V (18.4°);
- Construction temporary slope: 1.4H:1V (35.5°);
- Bench height: 3 m;
- Final crest elevation: 354 metres above sea level (masl);
- Co-disposal height: mostly 60 m.

The co-disposal strategy uses waste rock to construct the perimeter berm and roads around the CSF. Filtered tailings and waste rock would be deposited together. The central access roads would act as a filter berm providing drainage of the CSF.

- Filtered tailings deposition in the CSF would include the following criteria:
- Place wet tailings in pre-established and identified areas of the facility;
- Deposit filtered tailings in thin layers, compacted between each lift;
- Grade and seal filtered tailings deposition areas during operations to reduce water infiltration and accumulation;
- Transport tailings from the concentrator to the CSF on the same roads used by open-pit operations haul trucks;
- Remove snow from active construction areas and place in designated disposal areas.

An annual horizontal layer of approximately 1.2 m of waste rock will be included in the filtered tailings area as presented in Figure 17-3 Advantages of this configuration are:

- Allows for a potentially simpler operation in wet conditions, particularly in spring;
- Provides a potential small volume gain by penetration of filtered tailings in the waste rock;
- Acts as protection against erosion;
- Increases the tailings drainage.

During construction, regular visual inspections will be required by qualified technical personnel. The facility would be monitored by geotechnical instruments such as vibrating wire piezometers to track the water table in the filtered tailings.

The area of contact between the waste rock and the topography would be natural soil that is cleared, but not stripped. The subsoil (natural soil) underneath the CSF is assumed to be till composed of sand and gravel. If deleterious materials are encountered during earthworks, they would be removed to ensure stability of the CSF. Low areas in the co-disposal sector would be pumped and drained outside of the CSF to reach water management system.

Topography was considered for the construction sequence. The terrain where the pile will be built consists of a series of hills separated by flatter areas. The construction sequence would begin east of the CSF Phase 1 site and involve a hill to constrict the filtered tailings on the northeast side. Figure 17-4 presents the final stage of development of the CSF Phase 1 (after 6.5 years)

17.6.1 CSF Design Criteria

Two (2) provincial guidelines, namely *Directive 019* (*MELCCFP* 2012) and the Ministère de ressources naturelles et des forêts (*MRNF*) Closure Guidelines (2017), were used to support tailings facility design basis memorandum of the CSF (WSP 2023). In addition, the information presented by the Canadian Dam Association (CDA; 2013, 2014, 2019), the Mining Association of Canada (MAC 2019a, b), Guidelines for Mine Waste Dump and Stockpile Design (Hawley and Cunning 2017), and the Global Industry Standard on Tailings Management (GISTM 2021) were considered.

The CDA consequence classification system was used as the rating system for the population at risk and the potential incremental losses in the categories of Loss of Life, Environmental and Cultural Values, and Infrastructure and Economy to establish the consequence rating for a tailings or water storage facility.

Figure 17-3 Typical Cross-Section of CSF (Phase 1)

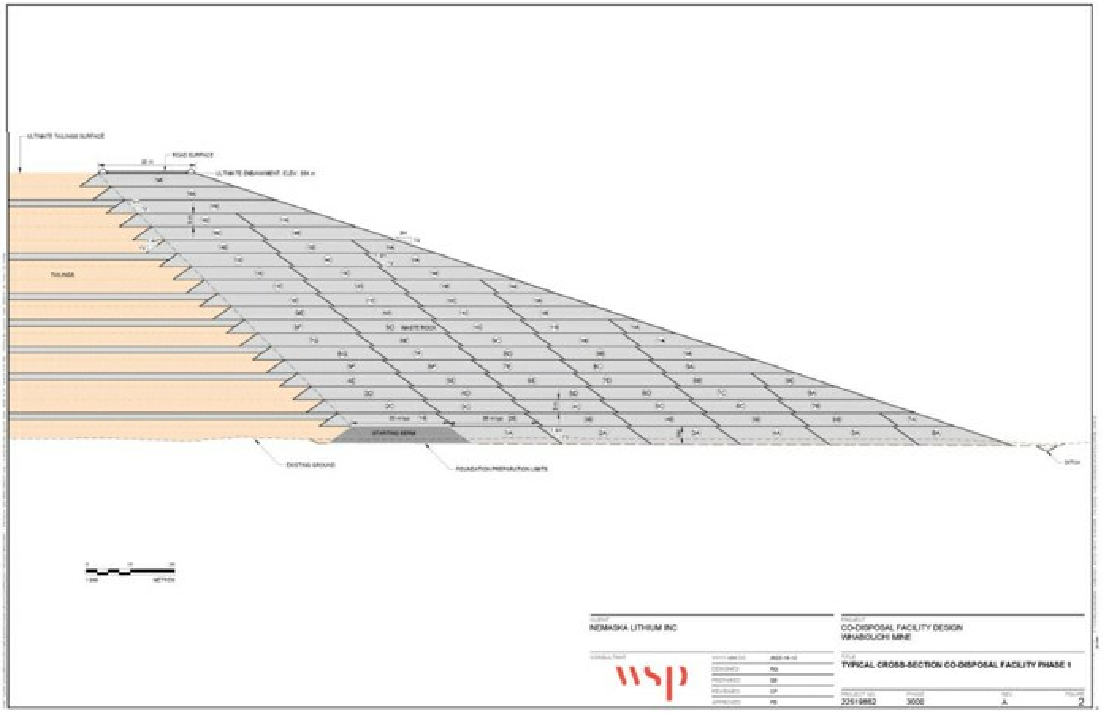
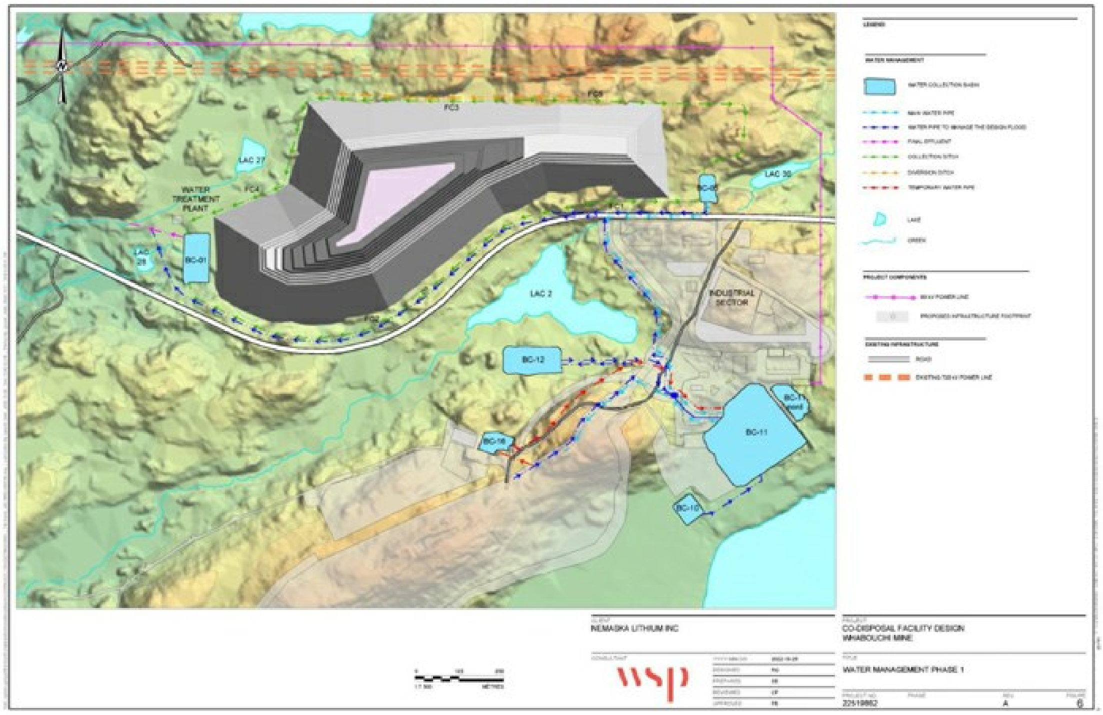


Figure 17-4 Final Stage of CSF Development after 6.5 Years



The CSF is rated as “high” consequence of failure with possible loss of ten (10) or less lives due to the Route du Nord nearby. Risks to environmental and cultural values were considered with possible significant loss or deterioration in fish and wildlife habitats, despite restoration, as being highly possible. High economic losses affecting infrastructure, public transportation, and commercial facilities are also possible risks due to the Hydro-Québec powerline and the Route du Nord nearby.

According to the classification of the Waste Dump and Stockpile Stability Rating and Hazard Classification system and the information currently available, the CSF stability rating is 65, which corresponds to a “low hazard” level of danger (Category II). The most important factors contributing to the instability hazard of the CSF are “foundation liquefaction” and “material liquefaction,” with rating of -2.5/20 assigned since liquefaction can not be fully discounted at this time.

The pile design was completed in accordance with *Directive 019*, related to the mining industry. Geochemical investigations to assess the acid rock drainage (ARD) and metal leaching (ML) potential of waste rock, ore, and tailings from the Whabouchi Lithium deposit generally indicate low ARD and low ML potential (Lamont 2013). Therefore, Level A groundwater protection measures would have to be applied. No basal waterproofing measures is included for groundwater protection. If the ARD and ML were to increase above the *Directive 019* criteria, design changes are expected.

17.6.2 Stability Factors of Safety

Preliminary stability analyses were conducted by WSP to reflect the co-disposal concept and possible filling sequence. Table 17-2 shows the minimum factors of safety (FSs) for the different loading conditions evaluated in the stability analyses.

The minimum FS for static and pseudo-static conditions and for the periods of construction and end of operation periods were established using an approach developed by Hawley and Cunning (2017) for hazard and stability assessment of waste rock piles, and according to the Guide de Préparation du Plan de Réaménagement et de Restauration des Sites Miniers au Québec (*MRNF* 2017). The FS depends on the level of consequence of failure, rated “high” for the CSF; but, also the level of confidence in the parameters or inputs of the stability analysis model and the failure mechanisms, a level estimated as “low” at this stage. As recommended by the Global Industry Standard on Tailings Management (GISTM), the chosen approach would be the “Extreme Consequence Classification” for the external load criteria of the CSF design, namely the 1/10000 year earthquake and maximal probable precipitation.

Table 17-2 CSF Factors of Safety

Analysis Condition	Minimum Required Proposed FS		Guideline Minimum FS			
	Construction and Operation	Closure	ICMM (2020)	CDA (2019)	Hawley and Cunning (2017)(a)	MERN (2017)
Static						
peak parameters	1.5 (1.1)	1.5 (1.2)	N/A	1.5 (1.1)	1.3 to 1.4	1.3 to 1.5
post-peak parameters	1.2	1.2	N/A	1.2	N/A	N/A
Seismic						
design earthquake recurrence period (high consequence)	2,475 yr	10,000 yr	2,475/ 10,000 yr(d)	2,475 yr	N/A(b)	N/A
pseudo-static analysis - peak parameters	1.1	1.1	N/A	1.0	1.0 to 1.05	1.1 to 1.3
post-seismic - dynamic liquefaction in granular soil	1.2	1.2	N/A	1.2	N/A	N/A

(a) Based on a Moderate consequence classification due to nearby critical infrastructure and on a moderate confidence classification due to the limited information on the foundation conditions.
(b) 475 years is commonly used as exceedance probability but there are no requirements.
(c) The PGA of a 1:10,000 years earthquake will be estimated based on a linear extrapolation of available NRCC data due to the absence of a site-specific seismic hazard assessment.
(d) 2,475 years for construction and operation; 10,000 years for closure.
Note: Values in parentheses are the target FS for local failures;
Bold values indicate recommended FS for stability analysis.
Legend: N/A = not applicable; FS = factor of safety; ICMM = International Council on Mining and Metals;
CDA = Canadian Dam Association; MERN = Ministère de l'Énergie et des Ressources naturelles

The consequence classification and the level of confidence in the parameters could be revised after additional information is received. For post-earthquake analyses and closure analyses, the selection of minimum FS also considers the *MRNF* Guide (*MRNF*, 2017). The *Directive 019* (*MELCCFP*, 2012) and the CDA (2019) do not propose stability criteria for structures without water impoundment and structures that do not contain material susceptible to liquefaction.

17.6.3 Additional Design Criteria and Assumptions

The detailed design of the CSF would address the following objectives:

- Provide a configuration that reduces encroachment on areas not affected by mining activities.
- Provide a configuration that fits within the primary reclamation objectives of the CSF to reduce the amount of reclamation work required when the CSF is reclaimed.
- Limit site preparation requirements, such as the construction of additional access roads.
- Limit water ponding on the surface of the CSF and water infiltration.

- Ensure the water management system can handle a projected flood without uncontrolled discharge to the environment and without major damage to infrastructure.
- Allow for flexibility in the design should the amount of waste rock stored be smaller than estimated at the beginning of the project.
- Limit the amount of earthwork and preparation work to allow for reclamation.
- Ensure adequate risk management throughout the life cycle of the CSF.

The following additional general assumptions will be adopted for the Whabouchi Project mine waste storage facility preliminary design:

- During construction and operation, the water runoff would have to be pumped outside the CSF footprint to mitigate the need of a spillway.
- All seepage and runoff from the CSF would be collected in perimeter ditches and/or trenches and directed to the water treatment plant.

17.7 **Relations with Stakeholders**

During the period of corporate restructuring, the new NLI continued to respect the commitments of the Chinuchi Agreement. The Band Council of the Cree Nation of Nemaska was met to explain the restructuring and introduce the new partners.

With the resumption of the project, meetings with the community have multiplied. The statutory committees of the Chinuchi Agreement have been reformed and have resumed regular meetings. These are the Whabouchi Implementation Committee (WIC) and the Environment Committee (EC). Meetings are quarterly. Two (2) other thematic committees have been formed to deal with priority topics. These are the Training and Employment Committee (TEC) and the Economic Development Committee (EDC). Both thematic committees also meet quarterly. In addition, to facilitate the organization of the committees and the implementation of the proposed actions, a coordination committee made up of representatives of the administrative staff of each stakeholder meets every two weeks.

A new liaison officer who is a Nemaska community member has been hired. And a new a new community communication plan is being developed in coordination with the band communication officer. The specialized firm Transfert Environnement et Société is supporting NLI with stakeholders relation activities.

Privileged relations are also being maintained with the non-Cree communities of Jamésie. The two (2) communities of interest are Matagami and Chibougamau. The town of Matagami owns and operates the rail terminal where our containers of concentrate will be stored and transferred between road trucks and trains. The city of Matagami has been supported in its requests for subsidies to repair and expand the terminal and we are discussing a long-term service contract.

Chibougamau is the closest town to the Whabouchi mine. A good part of the goods and services delivered to the mine come from or pass through Chibougamau. City authorities and social and economic organizations are met to explain the new Project, which is generating a lot of interest.

In the QP’s opinion, the current plans for environmental compliance, permitting, and addressing issues with local individuals or groups are adequate.

18 CAPITAL AND OPERATING COSTS

The capital cost estimates (Capex) and operating cost estimates (Opex) were developed to pre-feasibility standards (+/- 25%). The Capex and Opex for the Whabouchi Project and shipment to market are included in Section 18.1 (Capex) and 18.2 (Opex).

The Capex and Opex values were provided by a number of qualified firms, each with their scope of work and then blended by NLI into one Capex and Opex. The Owner's costs were provided by NLI. A working capital equal to two (2) months trade payables and trade receivables has been included as well.

The Capex are reported in Canadian Dollars (CAD). The Capex includes pre-production capital cost, working capital and the initial required costs to complete the initial construction.

Table 18-1 presents the Capex for the entire Project.

Table 18-1 Capex – ($ 000's CAD)

Description	Capex
Direct Costs	283,565
Indirect Costs (incl. Owner's Cost)	147,589
Contingencies	42,000
Total Capex	**473,154**

18.1 **Capital Costs Summary – Whabouchi Mine Site**

The Capex is based on the scope of work as presented in earlier sections of this Report. The Capex consists of direct and indirect capital costs as well as contingency which includes an allowance for escalation. An estimate for the sustaining capital has been prepared for use in the financial model.

A provision for contingency of 9.7% was selected to cover the remaining work to be completed. The work generally covered labour-intensive installation and site work. A significant portion of the equipment has been purchased and delivered to site.

18.1.1.1 Detailed Capex

The Capex associated to the remaining scope of work at Whabouchi is $473.2 M CAD for the initial capital costs, and $198.4 M CAD for sustaining costs over the life of the mine. The closure costs are not included in the Capex nor sustaining capital.

18.1.1.2 Major Assumptions

The Capex is based on the Project obtaining all relevant permits in a timely manner to meet the Project schedule.

The estimate reflects a standard EP project delivery and a hybrid self-execution CM type construction mode and is based on the assumption that construction contracts will be attributed on the base of a competitive bidding process amongst qualified installation contractors. It is expected that a high level of site management supervision, contract administration, quality control and thorough safety management will be required during the site execution phase.

Other key site execution parameters are the following:

- A limited number of contractors on site;
- Soil conditions will not require special foundation designs such as piling;

- All excavated material will be disposed of within the site battery limits;
- Concrete will be mixed in the existing batching plant and will be free issued to contractors. The cost will be included in the direct cost for quantity per WBS.

18.1.1.3 Major Exclusions

The following items were not included in this Capex, but have been considered in the economic analysis:

- Currency fluctuations and interest expense incurred during construction;
- Project financing costs;
- All duties and taxes.

18.1.2 Currencies

All expenditures to date are in Canadian dollars. The exchange rates are shown in Table 18-2.

Table 18-2 Foreign Currency Exchange Rates

Currency Codes	Currency Name	Rate
CAD	Canadian Dollar	1.000
USD	US Dollar	1.31
EUR	Euros	1.42

18.1.3 Contingency

Contingency is an integral part of the estimate and can best be described as a provision for undefined items or cost elements that will be incurred, within the defined project scope, but that cannot be explicitly foreseen due to a lack of detailed or accurate information.

It should not be considered as a compensation for estimating inaccuracy, nor is it intended to cover any costs due to potential scope changes, “Act of God”, labour strikes, labour disruptions outside the control of the project manager, fluctuations in currency, or cost escalation beyond the predicted rates.

The overall contingency factor is established by assessment of major risks and uncertainties identified during the course of the Project.

Based on the level of engineering definition for the Project as well as assessment of major uncertainties, a factor of 9.7% on direct and indirect costs was established to estimate the provision for contingency. For the Whabouchi mine site, the provision amounts to $42.0 M CAD.

It is nevertheless expected that sufficiently developed engineering, adequate project management and tight construction cost control will be implemented at Project re-start in order to meet the budget.

18.1.4 Sustaining Capital

A provision of $198.4 M CAD has been estimated to cover the sustaining capital over the life of the mine. The sustaining capital refers to the purchase of equipment or development of facilities which would otherwise be capitalized. It is not a subset of the operating costs. The sustaining capital costs include mine equipment purchased in future years, replacement of equipment, development of the underground mining operation and required equipment, equipment replacement for the concentrator areas as well as the provision for an online analyzer, and a new scavenger ore sorting facility.

18.2 Whabouchi Operating Cost

Operating costs were estimated for the Whabouchi Mine operation and transport costs to bring spodumene concentrate to market. For costing purposes, we assume ground transportation to Matagami (405 km by truck) and 900 km by rail to Bécancour, a port city located on the St. Lawrence River, where a future lithium hydroxide conversion plant is planned. This transport route is within a single jurisdiction and can operate all-year round with little risk of disruption. Additional operating costs include costs related to ore extraction, spodumene concentration, management of tailings, waste, and water, General and Administration (G&A) costs including site services, transport and lodging of workers and operation expenses and concentrate shipping.

The unit operating costs were based on a typical steady state spodumene concentrate production of 221,400 t/y (dry).

The sources of information used to develop the operating costs include in-house databases and outside sources.

The average operating cost estimate is summarized in Table 18-3.

Table 18-3 Annual Opex for Whabouchi

Unit Cost	Estimate $CAD/t Concentrate
Mining	$91
Processing	$394
Tailings and water management	$11
Concentrate transport	$263
General and administration	$210
Unit Cost	**$970**

19 ECONOMIC ANALYSIS

The economic assessment of the Project is based on price projections in U.S. currency and cost estimates in Canadian currency. An exchange rate of 1.31 CAD per USD was assumed to convert USD market price projections and particular components of the cost estimates into CAD. No provision was made for the effects of inflation. The base-case evaluation was carried out on a 100%-equity basis.

Current Canadian tax regulations were applied to assess the corporate tax liabilities and the regulations adopted in 2013 were applied to assess the Quebec mining tax liabilities. Historical tax losses carried forward were considered. This assessment is based on the fact that the Project is emerging from a care and maintenance phase. Consequently, all funds invested up until April 2023 are considered sunk and are omitted from the capital expenses in the present economic analysis. Only that part of the capital expenditure that remains to be incurred to bring the Project to the production phase is considered.

19.1 Summary of Assumptions

The following criteria have been used to develop the economic projections:

- Annual cash flow forecasts using pricing for spodumene concentrate (5.50% Li_2O) based on Wood Mackenzie forecasts throughout the life of asset.
- Operating costs include all raw materials, packaging, labor, utilities, maintenance, and overhead.
- Expected asset operating life of 34 years is supported by sustaining capital as well.
- Anticipated mining lease renewal fees are included.
- DCF was carried out on a constant money basis, so there is no provision for escalation or inflation on costs or revenue.
- For project DCF evaluation purposes, it has been assumed that 100% of capital expenditures, including pre-production expenses, are financed with owners' equity.

19.2 Results

Table 19-1 presents the financial indicators under base case conditions.

Table 19-1 Base Case Scenario Results

Base Case Financial Results	Unit	Value
Pre-Tax (P-T) NPV @ 8%	M CAD	3,588.4
After-Tax (A-T) NPV @ 8%	M CAD	2,080.1
P-T IRR	%	50.2
A-T IRR	%	36.9
P-T Payback Period	Years	1.0
A-T Payback Period	Years	1.4

A sensitivity analysis reveals that the Project's viability will not be significantly vulnerable to variations in capital and operating costs, within the margins of error associated with feasibility study estimates. However, the Project's viability remains somewhat vulnerable to a strengthening CAD/USD exchange rate and more vulnerable to the larger uncertainty in future market prices.

19.3 Assumptions

19.3.1 Macro-Economic Assumptions

The main macro-economic assumptions used in the base case are given in Table 19-2 The price forecasts for spodumene were derived using projections from Wood Mackenzie Global Lithium outlook. Details concerning the derivation of these forecasts can be found in Section 16 of this Report. The sensitivity analysis examines a range of prices 25% above and below the base case price forecasts.

Table 19-2 Macro-Economic Assumptions

Item	Unit	Long-term Value (2035)
Spodumene Concentrate (5.50% Li_2O) Price Forecast	$USD/tonne	2,381
Exchange Rate	CAD/USD	1.31
Discount Rate	% per year	8
Discount Rate Variants	% per year	6 and 10

The sensitivity of base case financial results to variations in the exchange rate was also examined.

The current Canadian tax system applicable to Mineral Resource Income was used to assess the Project's annual tax liabilities. This consists of federal and provincial corporate taxes as well as provincial mining taxes. The federal and provincial corporate tax rates currently applicable over the Project's operating life are 15.0% and 11.5% of taxable income, respectively. The marginal tax rates applicable under the mining tax regulations in Quebec are 16%, 22% and 28% of annual profit and depend on the profit margin.

For taxation purposes, the Project is subject to corporate income taxes and mining tax. A processing allowance rate of 20% is assumed for mining tax purposes, as the Mining Operation produces a concentrate that is shipped for further processing.

The base-case assessment was carried out on a 100%-equity basis. Apart from the base case discount rate of 8.0%, two (2) variants of 6.0 and 10.0% were used to determine the Net Present Value (NPV) of the Project. These discount rates represent possible costs of equity capital.

19.3.2 Resource Development Partnership Agreement Payments

The present economic analysis incorporates provisions for a Resource Development Partnership Agreement (RDPA).

Payments under the RDPA are based on the net after-tax cumulative cash flows of the Project. The actual terms are confidential and cannot be disclosed.

19.3.3 Technical Assumptions

The main technical assumptions used in the base case are given in Table 19-3.

Table 19-3 Technical Assumptions

Item	Unit	Base Case Value
Open Pit Total Reserve Mined	k tonnes	26,497
Open Pit Average Mill Head Grade	% Li_2O	1.32
Open Pit Design Mining Rate (ore only)	k tonnes/year	1,121
Underground Total Reserve Mined	k tonnes	11,709
Underground Average Mill Head Grade	% Li_2O	1.29
Underground Design Mining Rate (ore only)	k tonnes/year	1,243
Mine Life	years	34
Average Process Recovery	%	83.4
Concentrate Grade	% Li_2O	5.50
Total Concentrate Production	k tonnes	7,605
Average Whabouchi Concentrate Costs	(CAD/tonne)	968

19.4 Cash Flow Model and Results

Investment decisions are forward-looking. The issue consists of assessing whether an initial capital expense incurred from today onwards for the purpose of obtaining future benefits is justified. Any related past expenses are irrelevant to the decision. If an ensuing economic analysis shows that the future benefits are greater than the initial capital expense, then this capital expense is justified.

In the case of this Project, the initial capital expense is the expenditure required today to bring the Nemaska Whabouchi Spodumene Mine to the production phase. The resulting benefits are the stream of expected net cash flows generated over the life-of-mine. These consist of all revenues and expenses associated with mine production. Any project-related expenses incurred in the past, generally referred to as "sunk costs", are irrelevant to the decision today. Consequently, this economic assessment ignores sunk costs in the determination of cash flows and economic indicators.

Figure 19-1 illustrates the after-tax cash flow and cumulative cash flow profiles of the Project for base case conditions. The intersection of the after-tax cumulative cash flow curve with the horizontal dashed line represents the payback period (measured from the start of Year 1, which is not the start of commercial production).

A summary of the base case results is given in Table 19-4.

The summary and cash flow statement indicate that the total pre-production (initial) capital costs were evaluated at $473.2 M CAD. The sustaining capital requirements at the mine site were evaluated at $198.4 M CAD, which includes transportation equipment, underground mine development and equipment. Mine closure costs were estimated at $15.0 M CAD (of which $13.6 M has already been paid).

The cash flow statement shows a capital cost breakdown by area and provides an estimated capital spending schedule over the remaining pre-production period of the Project. Working capital requirements were estimated at two (2) months of accounts payable and accounts receivable balances and the movement of inventory. Since operating costs vary annually over the mine life, additional amounts of working capital are injected or withdrawn as required.

The total revenue derived from the sale of concentrate from the mine was estimated at $23,154.9 M CAD or, on average, $606/t CAD milled. The total operating costs were estimated at $7,950.1 M CAD, or on average, $208/t CAD milled.

The financial results indicate a pre-tax NPV of $3,588.4 M CAD at a discount rate of 8.0%. The pre-tax Internal Rate of Return (IRR) is 50.2% and the payback period is 1.0 year.

The after-tax NPV is $2,080.1 M CAD at a discount rate of 8.0%. The after-tax IRR is 36.9% and the payback period is 1.4 years (see Table 19-5).

Figure 19-1 After-Tax Cash Flow and Cumulative Cash Flow Profiles

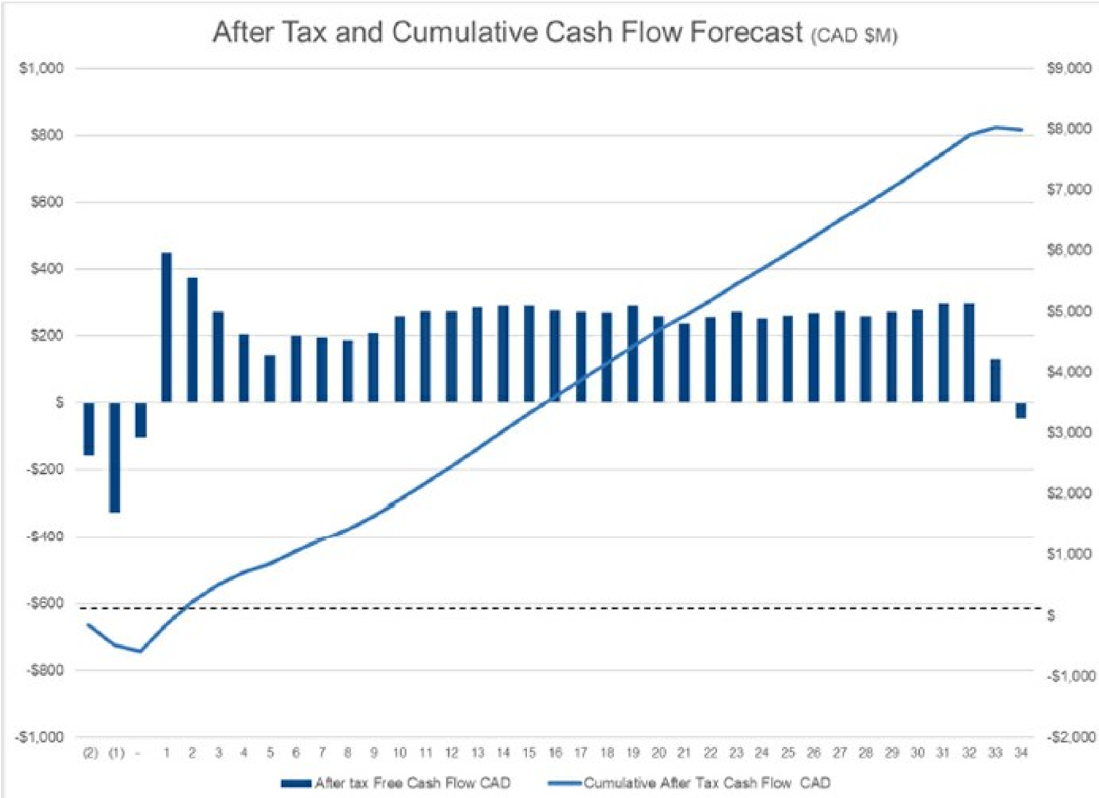


Table 19-4 Project Evaluation Summary – Base Case (CAD)

Item	Unit	Value
Concentrate Sales	$ M	23,154.9
Total Operating Costs	$ M	7,950.1
Initial Capital Costs (excludes Working Capital)	$ M	473.2
Sustaining and Other Capital Costs	$ M	198.4
Mine Rehabilitation Trust Fund Payments	$ M	11.4
General and Administration Corporate Costs	$ M	1,649.4
RDPA Payments	$ M	323.0
Total Pre-tax Cash Flow	$ M	13,301.6
Pre-tax NPV @ 6% [1]	$ M	4,752.8
Pre-tax NPV @ 8%	$ M	3,588.4
Pre-tax NPV @ 10%	$ M	2,776.3
Pre-tax IRR	%	50.2
Pre-tax Payback Period [2]	Years	1.0
Total After-tax Cash Flow	$ M	7,990.5
After-tax NPV @ 6%	$ M	2,789.8
After-tax NPV @ 8%	$ M	2,080.1
After-tax NPV @ 10%	$ M	1,584.3
After-tax IRR	%	36.9
After-tax Payback Period	Years	1.4

1 NPV calculation based on mid-period convention.
2 Measured from the start of commercial production.

Table 19-5 After-Tax Financial Summary

Post-Tax Net Present Value (CAD)	Post-Tax Internal Rate of Return	Payback Period (Years)	Life-of-Mine (Years)	Total Initial Capital (CAD)
$2,080.1 M	37%	1.4	34	$473.2 M

19.5 Sensitivity Analysis

A sensitivity analysis has been carried out, with the base case described above as a starting point, to assess the impact of changes in total pre-production Capex, Opex, product prices (PRICE) and the USD/CAD exchange rate (F/X) on the Project's NPV @ 8.0% and IRR. Each variable was examined one-at-a-time (all product prices are varied together). An interval of ±25% with an increment at 10% from the base case were used for the first three (3) variables. It is to be noted that the margin of error for cost estimates at the pre-feasibility study level is typically ±25% (without contingency). Revenue reported as spodumene price may be affected by project risks, including but not limited to plant availability and recovery, and market price volatility. While lower bound sensitivity is reported in accordance with pre-feasibility standards, lower revenue, and in turn NPV, may occur in certain circumstances. Table 19-6 presents the base case sensitivity analysis results.

Table 19-6 Base Case Sensitivity Analysis (CAD)

Cost Variable	Sensitivity Factor				
	-25%	-10%	0	10%	25%
Spodumene Price $ /Mt	$1,121 M	$1,699 M	$2,080 M	$2,458 M	$3,023 M
Initial Capital	$2,219 M	$2,136 M	$2,080 M	$2,024 M	$1,941 M
Operating Expense	$2,423 M	$2,217 M	$2,080 M	$1,940 M	$1,727 M

19.6 Detailed Economic Analysis

Table 19-7 Detailed Economic Analysis (LOM period)

Year	Total Ore Produced (kt)	Average Grade (% Li_2O)	Concentrate Produced at 5.5% Li_2O (kt)	Total Li_2O Units in Concentrate (kt)	Spodumene Concentrate Price ($/t)	Initial Capital ($M)	Sustaining Capital ($M)	Revenue ($M)	Operating Expense ($M)	DD&A ($M)	Royalties ($M)	Pre-Tax Cashflow ($M)	Taxes ($M)	Post-Tax Cashflow ($M)
2023	–	–	–	–	NA	116	6	–	46	–	3	-157	–	-157
2024	–	–	–	–	NA	250	3	–	70	–	2	-325	–	-325
2025	574	1.35	99	5	2,841	107	23	280	159	23	–	-88	16	-104
2026	1,078	1.37	224	12	3,981	–	20	894	247	23	–	578	129	449
2027	1,140	1.3	217	12	3,528	–	13	765	242	23	–	532	158	375
2028	1,120	1.31	221	12	2,999	–	7	664	252	23	–	430	158	272
2029	1,128	1.26	218	12	2,537	–	4	553	254	23	–	314	109	205
2030	1,113	1.26	214	12	2,578	–	47	552	252	23	–	252	109	142
2031	1,120	1.32	225	12	2,552	–	4	573	238	23	–	325	126	200
2032	1,103	1.33	223	12	2,447	–	4	545	235	23	–	310	115	196
2033	1,098	1.33	224	12	2,338	–	3	523	232	23	–	292	107	185
2034	1,113	1.29	220	12	2,685	–	4	592	233	23	–	344	136	208
2035	1,144	1.28	223	12	3,118	–	3	695	242	23	–	435	177	259
2036	1,150	1.28	224	12	3,118	–	3	697	241	23	–	452	178	275
2037	1,150	1.28	224	12	3,118	–	3	697	241	23	–	453	178	275
2038	1,144	1.34	233	13	3,118	–	4	727	248	23	–	473	187	285
2039	1,142	1.34	235	13	3,118	–	3	733	245	23	–	483	191	292
2040	1,142	1.34	235	13	3,118	–	3	733	250	23	–	480	189	291
2041	1,147	1.26	223	12	3,118	–	3	695	241	23	–	454	177	278
2042	1,148	1.26	220	12	3,118	–	3	687	240	23	–	445	174	271
2043	1,148	1.26	220	12	3,118	–	3	687	240	23	–	443	174	270
2044	1,116	1.39	235	13	3,118	–	2	733	240	23	–	485	194	290
2045	1,109	1.39	238	13	3,118	–	43	743	237	15	–	460	203	258
2046	1,109	1.39	238	13	3,118	–	64	743	240	15	–	439	201	237
2047	1,109	1.39	239	13	3,118	–	46	746	242	15	–	457	202	255
2048	1,013	1.49	232	13	3,118	–	15	722	257	15	–	457	185	272
2049	1,181	1.2	216	12	3,118	–	7	674	290	32	–	391	140	251
2050	1,248	1.29	236	13	3,118	–	5	737	312	32	–	416	157	259
2051	1,244	1.28	237	13	3,118	–	2	738	309	32	–	426	159	267
2052	1,245	1.3	242	13	3,118	–	3	755	311	32	–	440	166	274
2053	1,252	1.22	229	13	3,118	–	4	713	304	32	–	409	150	258
2054	1,239	1.32	240	13	3,118	–	2	749	302	32	–	439	167	272
2055	1,249	1.28	239	13	3,118	–	2	745	295	32	–	447	169	278
2056	1,247	1.37	253	14	3,118	–	2	790	299	32	–	484	187	298
2057	1,242	1.36	253	14	3,118	–	2	790	296	17	–	491	195	296
2058	619	1.28	120	7	3,118	–	3	374	226	17	–	180	50	129
2059	82	NA	36	2	3,118	–	2	111	223	0	–	-46	0	-46
Totals	***38.2E+6***		***7,604,910***	**418**		**$ 473**	**$ 373**	**$ 23,155**	**$ 9,031**	**$ 817**	**$ 5**	**$ 13,302**	**$5,311**	**$ 7,990**

Notes:
Currency in $CAD
kt = 1,000 tons
M = million
DD&A = depreciation, depletion, and amortization

20 ADJACENT PROPERTIES

Critical Elements Lithium Corporation (Critical Elements) and Li-Ft Power hold most of the adjacent properties to the Whabouchi deposit (Figure 20-1). According to SIGEOM, no lithium deposit, showing or mine is surrounding the immediate vicinity of the Whabouchi Property (date of search: February 14, 2023). However, Critical Elements has reported the discovery of many lithium-bearing showing on their Lemare and Duval properties from their summer 2022 exploration program (October 27, 2022 press-release, https://www.cecorp.ca). The Duval property is adjacent to NLI Whabouchi's project, while the Lemare property is located 20 km north-east.

Li-FT Power has reported drill-ready lithium targets generated from till geochemistry anomalies on the adjacent Rupert Project (Li-FT Power Corporate Presentation, 2023).

The only known deposit that is relevant to the report is the Rose deposit held by Critical Elements, located 43.3 km north-northwest of Whabouchi.

Critical Elements filed a NI 43-101 Technical Report for the Rose Lithium-Tantalum Feasibility Study on July 27, 2022 (WSP, 2022).

The estimate comprises a pit-constrained Indicated Mineral Resource of 30.4 Mt grading 0.91% Li_2O; and a pit-constrained Inferred Mineral Resource of 2.0 Mt grading 0.76% Li_2O.

The estimate also comprises an underground Indicated Mineral Resource of: 1.1 Mt grading 0.86% Li_2O; and an underground Inferred Mineral Resource of: 0.7 Mt grading 0.78% Li_2O.

The project also holds a Probable Mineral Reserve amenable by open pit mining of: 26.3 Mt grading 0.87% Li_2O.

Cut-off grades of the Rose deposit are based on Net Smelter Return (NSR) values. For the Mineral Resource, the cut-off grades used were:

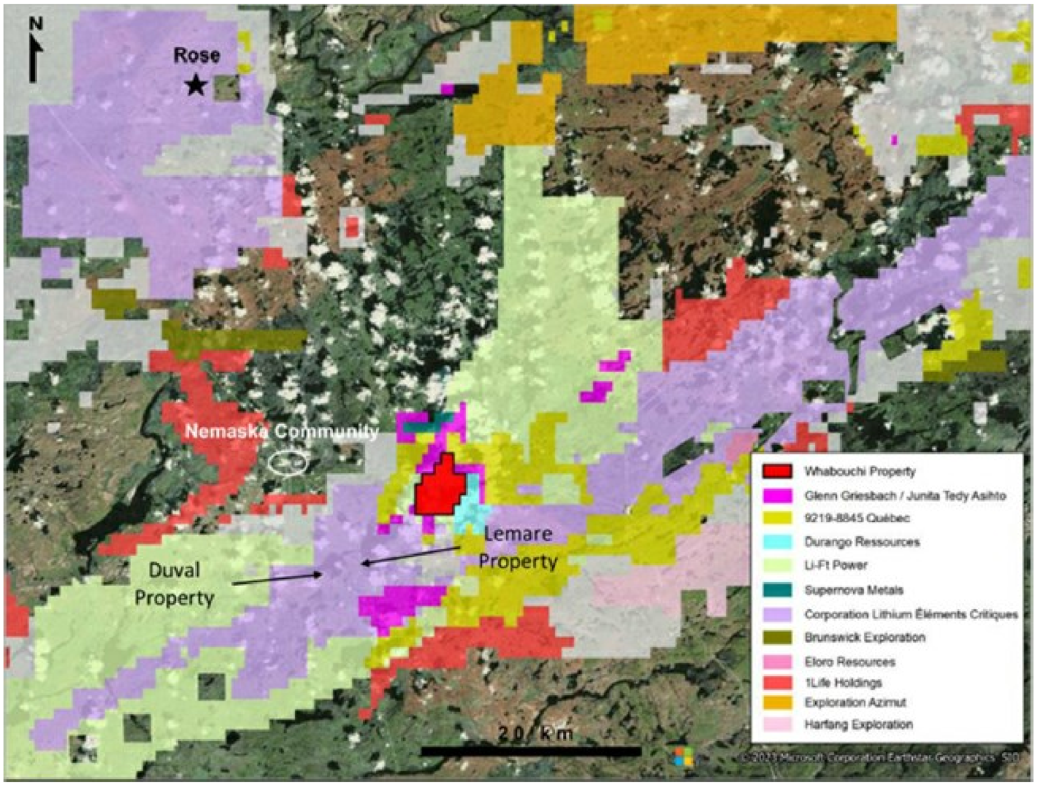
- C$31.40/t and C$121.12/t for open pit and underground respectively.

For the Mineral Reserve, a NSR cut-off grade of: C$29.70/t, was used. Metallurgical recoveries are set at a constant 85% for Li_2O. Metal prices used are:

- US$20,000/t Li_2O; and
- Using an exchange rate of: 1.25 (C$:US$).

The Contributing Author has been unable to verify the information presented in this section. The information is not necessarily indicative of the mineralization on the property that is the subject of the Technical Report.

Figure 20-1 Location Map Showing Adjacent Mineral Properties



21 OTHER RELEVANT DATA AND INFORMATION

21.1 **Project Status**

As of the effective date of writing of this Technical Report, project construction has started and is partially completed. Overall construction advancement stands at approximately 50%.

Site infrastructure work is started, and overall construction advancement stands at approximately 35%.

The required permitting for the construction of all site infrastructure will have to be permitted with the latest addition of the Whabouchi Village and supporting infrastructure. A current municipal construction permit will be sought prior to the start of any construction works.

The concentrator completion strategy takes into account that detailed engineering is well advanced; remaining work is centered on electrical, automation, secondary steel and process piping checks.

21.1.1 Mine

In 2018, purchase orders for mining equipment were placed. Deliveries and assembly were completed, and pre-production mining activities started, with the ROM Pad, the start of the tailings/co-disposal haul road, and certain other mine roads completed. The mining equipment leases were terminated in 2020, and the mining equipment will need to be procured going forward.

The support facilities for the mine include the mine garage and wash bay, the bulk fuel storage and distribution, the warehouse, waste rock and tailings co-disposal area and the low grade stockpile.

The mine garage facility building is constructed. Some work will be required to complete the construction, such as a concrete floor, addition of offices, an additional maintenance bay to accommodate the mining excavator, 20-tonne overhead crane, equipment, tooling, and hydraulic lift for the service equipment.

The wash bay, warehouse and tank farm have been designed and are a new construction.
WSP has completed preliminary design of the waste rock and tailings co-disposal areas. However, detailed design of the area and purchase of equipment and issuance of construction tender documents have not yet been prepared.

The layout and design of the low-grade stockpile has not been prepared to date.

21.1.2 Crushing System

The detailed design of the crushing system, comprising the primary crusher, ore sorting building, secondary and tertiary crusher building and the interconnection conveyors, transfer towers and screening towers, is complete. The purchase of the equipment is also essentially complete with only minor equipment remaining to be procured.
The earthworks and foundations for the crushing system facilities is about 85% completed. The primary crusher steel and hoppers are installed. The Primary and Secondary Jaw Crushers are not installed. A new rock breaker will be installed on the ROM pad. The ore sorting facility is completed, but no equipment has been fully installed to date. The secondary and tertiary cone crusher building have been erected, and equipment's are installed.

The crushing system electrical module is in place. The module is connected to the main substation via a buried line.

The remaining work focuses on the installation of the equipment in the ore sorting facility, the installation and reinforcements of all interconnecting conveyors, the finalization of the erection of the equipment in the secondary and tertiary cone crusher as well as the building structure, the completion of the transfer towers and screening tower.

Figure 21-1 shows the construction status of the primary crusher (located on the right of the picture), the electrical room (center) and the secondary and tertiary crusher building (located on the side of the picture).

Figure 21-1 View of Primary and Cone Crusher Buildings

Figure 21-2 shows the screening tower in the foreground and the fine ore storage and concentrator in the background. The construction camp facilities are shown in the background on the top of the hill. This picture was also taken in December 2019.

Figure 21-2 View of Screening Tower and Concentrator

21.1.3 Concentrator

The design of the concentrator is nearing completion. The balance of the design focuses on finalizing the new hydrofloat and piping layouts. There are only a few equipment or systems that have to be procured. The fire detection system was re-evaluated, a new vertical fire water pump will be installed at BC-11.

The fine ore and tailings dome will be modified to allow a direct feed system. The three (3) electrical modules are installed adjacent to the concentrator. The concentrate dome is also complete, a hoist system inside the dome to manipulate the container lids will be installed.

The balance of work to be done in the fine ore and tailings facility is the construction of the conveyor foundations, installation of the process equipment and conveyors and power connections.

The work in the concentrator has been put on hold and much of the interior foundation and structural steel work is completed. The final design of the secondary steel will need to be completed with the supplier for fabrication as well as the pipe supports. The building shell for the process water tank and thickener has not been erected, nor has the steel stairs linking the electrical rooms, laboratories, and the concentrator.

Figure 21-3 shows the interior of the concentrator with two (2) operating cranes, some of the erected equipment and installation of cable trays along the south wall.

Figure 21-3 View of Concentrator Interior

The erection of the equipment is on-going with some equipment installed. Two (2) overhead cranes are in place to assist in the erection of the equipment. Power for the work in the concentrator is provided by the main substation.

The office facilities and control room are enclosed, and a contract has been issued for the completion of the interior features. Pictures showing existing equipment are provided in Figure 21-4 through Figure 21-11.

- Figure 21-4 shows the status of the offices and control room.
- Figure 21-5 shows the wet, high intensity magnetic separator,
- Figure 21-6 shows the reagent area,
- Figure 21-7 shows the process water tank and thickener area,
- Figure 21-8 shows the dryer,
- Figure 21-9 shows the hydrocyclone,
- Figure 21-10 shows the hydraulic separating equipment, and
- Figure 21-11 shows the exterior of the main offices, laboratory and main electrical room for the Concentrator.

Figure 21-4 View of Offices and Control Room

Figure 21-5 View of the Wet High Intensity Magnetic Separator

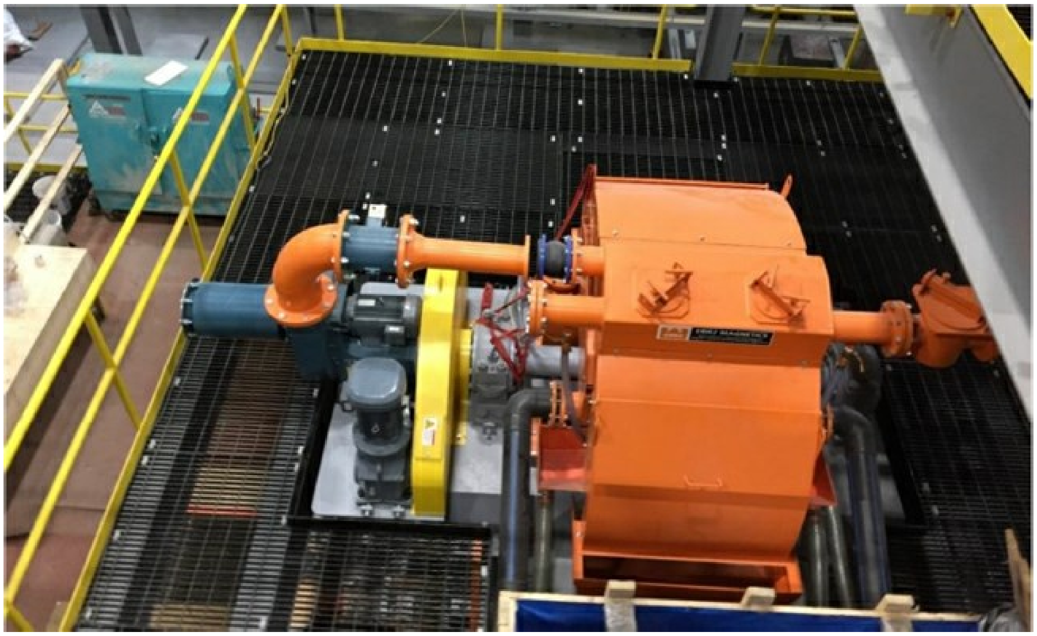


Figure 21-6 View of the Reagent Area

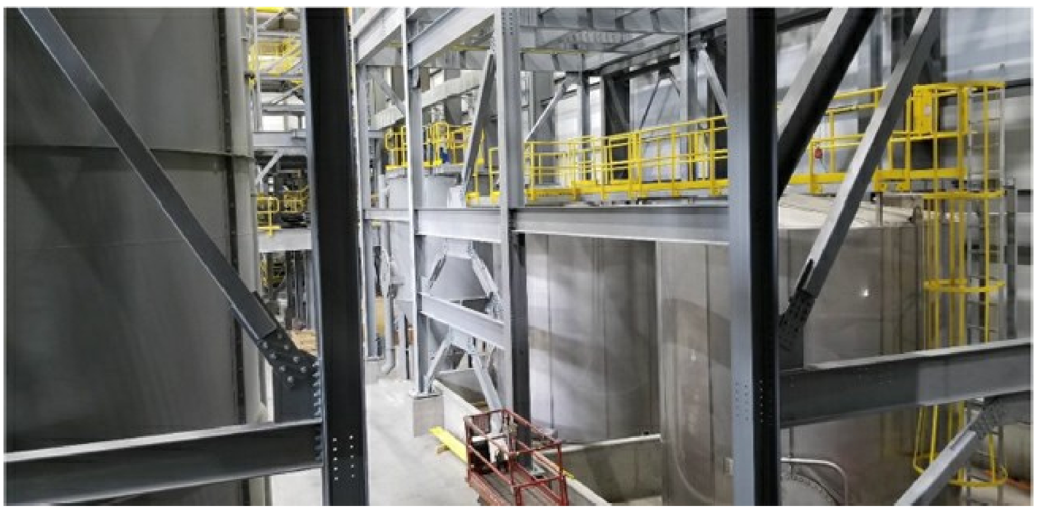


Figure 21-7 View of the Process Water Tank and Thickener Area

Figure 21-8 View of the Dryer

Figure 21-9 View of the Hydrocyclone

Figure 21-10 View of the Hydraulic Separators

21.1.4 Infrastructure

The administration building was completed and is currently occupied by NLI field personnel. This building will need to be demolished to make way for the new Welcome center. The state of the building precludes its economical and practical re-use on the site.

The laboratories have been procured and the five (5) modules are on site and assembled.

The gate house is fully functional. The truck scale will be relocated near the gate house at the entrance of the Whabouchi mine site.

Most of the site work, roads, and parking lots have been completed and provision is made for final leveling. A new mine haul road overpass has been included for the mine trucks to have easy access to the co-disposal area.

Work will be required to the Route du Nord to repair ditches.

Figure 21-11 Concentrator Main Electrical Room, Offices and Laboratory

The water management system includes basins and equipment to manage the excess water from the pit and from the general area. Some of the basins have been completed.

The existing camp is a fully functional camp housing the NLI care and maintenance team. The catering and camp services are currently undertaken by a Cree company. Following completion of the construction, NLI will replace the existing camp with the Whabouchi Village The temporary camp modules will be dismantled. The camp is serviced with potable water and sewage treatment. A concrete batch plant and aggregate crushing plant is mobilized to site.

A 169-room camp is already installed at the Relais Routier Nemiscau and operated by a Cree company that can accommodate supplemental current Project needs, as required.

21.2 Project Schedule and Key Milestones

The schedule for the Whabouchi Area is based on the start of detailed engineering as work on this study concludes, mobilization to site in the second quarter of 2023 and full start of construction by October 2023 with construction completion in February 2025. The planned production start-up is March 2025.

The Project key milestones are outlined in Table 21-1.

Table 21-1 Project Key Milestones

Milestone	Plan
Submit Detailed Feasibility Study	Mar 2023
Approval of Basic Permits & Start of Construction	Mar 2023
Temporary Camp Available for Use	May 2023
Start of Crushing Plant Commissioning	Sep 2024
Start of Concentrator Commissioning	Dec 2024
Declare Commercial Production	Mar 2025

21.2.1 Project Constraints and Dependencies

There are several constraints and dependencies that must be considered in the Project Schedule:

- The completion of water management basins BC-11 is required primarily to store water for the first winter of operation and manage major surface water flows particularly during the spring freshet. This basin will be completed prior to the end of the summer of 2023 in time to serve as a freshwater basin for the spring freshet of 2024 and commissioning activities expected to begin in the winter of 2024/2025.

- Winter weather

  These conditions may impact the productivity of certain construction activities; and have been considered given the expected construction remobilization in spring of 2023. Should the remobilization be delayed, this may impact actual productivities and cost, particularly for those exterior works which depend on warm weather for optimal productivity – such as the crushing area structural steel retrofits, all civil and concrete works, and turnkey building erection.

- Increase available power from Hydro-Québec from 10 MW to 16 MW will be required for continuous commercial production. If not available, some heating loads may need to be fueled by propane or diesel, and/or some temporary diesel power generation may be required.

22 INTERPRETATION AND CONCLUSIONS

22.1 Conclusions

The Whabouchi Mine is part of a broader corporate project (the Nemaska Integrated Lithium Project), which includes the development of the Whabouchi spodumene mine and concentrator located approximately 300 km North of Chibougamau and a lithium hydroxide conversion plant to be built in Bécancour, mid-way between Montreal and Quebec City.

During the 24-year life of the open pit mine, a total of 33.9 Mm³ of waste rock and 13.0 Mm³ of tailings will be generated for a total of 46.9 Mm³. The underground mine will generate an additional 0.7 Mm³ of waste rock and 5.9 Mm³ of tailings. In total, the Project will generate 53.4 Mm³ of waste materials. All the waste rock and filtered tailings will be contained in the designed storage facilities, except 4.0 Mm³ of waste rock that will be used as backfill material for the underground operation. Approximately 6.0 Mm³ of waste rock could also potentially be disposed in the open pit mine.

The underground project will take three (3) years and two (2) months to develop the required underground infrastructure to start commercial production.

The site-wide water management strategy and the water balance study will be updated during future engineering design phases and mine operations; accounting, whenever possible, for the construction sequence of the water management infrastructure based on the planned site development.

The results indicate that the site runoff and the groundwater inflows to the pit can supply the plant process water demand during the life of mine, and that no uncontrolled overflow of contact water to the environment is expected under the analyzed climate conditions.

The final effluent will release water to the Nemiscau River with regular monitoring of flow and water quality. If required, a water treatment plant will be implemented to ensure full compliance with all applicable quality criteria. Tests indicate that the ore and waste are non-acid generating, and no elements are leachable. The water management system has been designed to allow for sufficient settling time.

During the preparation of this Report, extensive additional laboratory testwork, vendor testwork and orebody characterization were complete to further reduce technological risks of the project.

The QPs have examined the technical aspects of the Project within the level of precision appropriate for the current status of the Project.

A computed cash flow analysis was developed by NLI from the technical aspects and based on SC6 price projections provided by Wood Mackenzie (April 2023). This Technical Report resulted in a Mineral Reserve Estimate that contains 38.2 Mt of Mineral Reserves averaging 1.31% Li_2O.

Consequently, the QPs conclude that the Project is technically feasible as well as economically viable and the authors of this Technical Report consider the Project to be sufficiently robust to warrant pursuing the implementation phase.

The main source of lithium at Whabouchi is spodumene in pegmatites. However, recent mineralogical studies have shown that other lithium-bearing minerals are present and are currently part of the mineral resource which is based on chemical content (Li_2O). Based on preliminary results, it is believed that two or three petalite-rich zones may exists. These zones currently occupy less than 5% of the mineral resource tonnage. Outside the petalite-rich domains (data mainly coming from Main1), fragmentary results show that approximately 2.3% of the lithium content is coming from petalite. With the current recovery methods, petalite is unrecoverable. Should petalite remain unrecoverable for the duration of the project, the LOM may be reduced by less than 2 years.

There are near-surface gaps in diamond drilling, resulting in indicated mineral resources. Indicated mineral resources, by definition, hold a certain degree of uncertainty and it is not recommended for first-year mining.

Lithium is considered as an industrial mineral and the sales prices for the different lithium compounds are not public. Sales agreements are negotiated on an individual and private basis with each different end-user. Therefore, it is possible that the sales prices used in the financial analysis be different than the actual market when NLI is in fact in a position to sell lithium compounds

22.2 **Opportunities**

The Whabouchi deposit has an opportunity to increase its mineral resources at depth by confirming the extension of known spodumene-bearing dykes. There are also some areas within the current mineral resource pit footprint that could see an increase in mineral resources by connecting interpreted dykes in different directions.

Drill core analytical data at Whabouchi varied during exploration campaigns. The sodium peroxide and the 4-acid fusion were the most used lithium digestion methods. Following a study by SGS (Camus and Dupéré, 2022), it was found that samples analyzed with 4-acid fusion underestimate grades by 4%. The impact of this bias on the mineral resource was evaluated at an underestimating of lithium grades globally by 1.6%.

23 RECOMMENDATIONS

23.1 General Recommendations

Based on the Project's modelled economic returns, it is recommended to proceed to the implementation phase of the Project. Based on the comments received to date, there are no issues that would materially affect the ability of NLI to develop and put the Project into production. However, NLI is still in consultation with regulators and stakeholders, and potential future conditions of approval could require refinements to Project components or additional mitigation measures to be implemented.

While the detailed design and procurement activities are on-going, it is recommended to monitor or to complete the specific activities listed below.

23.2 Mineral Resource Estimate

Diamond drilling and channel sampling is recommended for the following opportunities or to mitigate the risks:

- Channel sampling and near-surface diamond drilling targeting the indicated mineral resource area in the first five years of mining operations to convert to measured mineral resources,
- Diamond drilling at moderate depth (first 150 m) to convert indicated mineral resources to measured mineral resources. This conversion would require a minimal amount of drilling,
- Diamond drilling in areas of low drilling density to convert inferred mineral resources to indicated mineral resources,
- Targeted diamond drilling to confirm connexion between dykes of different orientations,
- Exploration diamond drilling at depth to confirm the continuity of spodumene-bearing dykes.

NLI should pursue all chemical analysis with the peroxide fusion method, while maintaining a robust QA/QC protocol.

It is recommended that NLI continue the analysis on lithium deportment. Once results are satisfactory and well distributed in the deposit, it is recommended to integrate these results into a mineralogical block model. It should also be investigated if other geometallurgical factors can be integrated, such as mineral liberation and grain size.

23.3 Open Pit Mining

For the open pit mine, BBA proposes that NLI:

- Evaluate the possibility of incorporating a fleet of battery powered equipment (Estimated cost $40,000 USD);
- Complete a trade-off study to assess the optimal bench height and excavator size (Estimated cost $25,000 USD).

23.4 Geotechnical – Open Pit

The open pit overburden has not been characterized geotechnically. This gap requires investigation through drilling, sampling and geophysics, particularly to identify localized areas of thicker overburden (>5m). Once completed, update the overburden waste dump design based on the geotechnical characterization of the overburden to be stripped from the open pit area.

It is recommended that operations restrict production blasts to within 50 m of an unblasted presplit line. Once presplit is shot, production blasts will be taken to within 10 m of the presplit and then a trim shot used to clean the face. Given that larger production shots may be more likely to damage the final walls, all blasts shall be monitored, and blast designs shall be adjusted to avoid this.

It is recommended that the mine planners keep the ramp on the south wall to keep the overall slope angle at 52° or lower to manage the potential for deep seated toppling. Alternatively, the south wall should be interrupted with geotechnical benches to achieve the same overall slope angle. There is a potential for steepening if deep seated toppling can be shown not to be a concern based on slope performance and additional joint spacing and hydrogeologic information.

Plan and assign budget for:

- Routine bench mapping to document:
    - Achieved bench face angles vs. slope design;
    - Map fault and dyke exposures for predictive analyses;
    - Map orientation variability and persistence in foliation and jointing;
    - Seepage, which may preferentially occur along the foliation, dykes, or at the top of bedrock.
- Vibrating wire piezometer installations with emphasis on the south wall because of the potential for deep-seated toppling.
- Drain holes along the south wall ramp.
- Prism installations and resources to monitor the prisms.

Any updated pit designs developed using these recommendations should be reviewed by a geotechnical engineer to validate that the slope designs have been applied correctly.

23.5 **Underground Mining**

For the next phase of the of the Project for the underground mine, DRA proposes the following recommendations to be performed:

- Trade-off Study to determine the ultimate open pit vs underground economic limit;
- Trade-off Study to combine the underground ore production with open pit production prior to Year 25;
- Geotechnical (hard rock) study for the underground mine design including the interaction (i.e., crown pillar extraction) between the open pit and underground mine;
- Hydrogeology study for the underground mine design;
- Backfill study including a trade-off study on paste backfill.

Prior to the Project detailed engineering phase for the underground mine, DRA proposes the following work as optional to be performed:

- A fully electric underground equipment fleet study; and
- Autonomous underground equipment fleet study.

23.6 **Process**

Based on the work performed and the test results, additional testwork and design modifications can be performed to both optimise and de-risk the process design and equipment selection. It is recommended to perform certain work for the next stage of the Project:

- To confirm closed circuit performance, recirculation performance, and overall plant recovery, it is recommended that an independent laboratory test the flowsheet at pilot scale as one continuous process.
- In order to control the grinding product, NLI should consider an automated media feeder. This would ensure near constant ball loading in the grinding circuit and reduce somewhat the slime production.
- An on-stream analyser is planned for installation in Year 2. In order to simplify the integration of the analyser, it is recommended to install it with the rest of plant and commission it once the plant has achieved stability.
- It is recommended to evaluate the material flow properties for the plant feed and tailings areas material to ensure proper bin design.
- It is recommended to carry out variability testing on samples representing the mine plan for Years 6-24. The geometallurgical methods for this testing will be prepared in 2023 for subsequent testing.
- It is recommended to perform a metallurgical development program investigating the recovery of petalite (currently planned and anticipated to start in 2024).
- A review of the plant feed bin (fine ore overflow hopper), fine ore screen feed chute, and tailings loadout designs is recommended to be performed in the next phase to ensure smooth operation.
- A complete review of all coarse pumping applications is required in the next phase. The current pumping arrangements will cause decreased availability. Larger pumps, ceramic lined pipes, fluidizers, and proper top-size protection is recommended.
- A complete review of all pumps, pipeline sizes, and equipment feed boxes is recommended following the water balance review.
- A review of the ore sorter feed arrangement is recommended, as the vibrating feeders are not designed to be used for feed control, only feed presentation.
- Due to the complexity of the flowsheet, it is recommended for NLI to be proactive with their operational readiness, development of operator training manuals, and on-boarding of skilled labor and operators. This will reduce commissioning and start-up risks.
- It is recommended to investigate the use of dozer traps for front-end loader applications or the use of de-lumpers to make it easier to feed/operate the plan.

23.7 **Mine Waste and Water Management**

It is recommended to account for the following points when reviewing the mine waste management and the water management strategy for the site and completing the engineering design of CSF infrastructure:

23.7.1 Mine Waste Management

The CSF structure is designed to store filtered waste rock and tailings. Inadequate filtration over a long period of time can lead to significant operational problems. The design of the CSF will require a prescribed low water content to achieve the required level of compaction. In this regard, compaction and water content control protocol should be implemented.

The placement of properly filtered tailings is not simple in a cold and wet climate. It will require some learning on the part of the operator. NLI must plan for the possibility of lost time, particularly early in the mine ramp-up.

Geotechnical investigations (boreholes and test pits) will be required as part of the detailed design of the CSF to specify the nature of the soils in the footprint and to specify the extent and costs of preparatory work.

It is assumed that the tailings do not generate acid and do not leach metals. If these conditions change, it could significantly alter the CSF foundation. Nemaska shall inform the designer if a change in tailings or waste rock geochemistry is observed.

The advancement of the mine closure plan by Nemaska is recommended.

23.7.2 Water Management

The side-wide water management strategy and the water balance study are to be updated during future engineering design phases and mine operations; accounting, whenever possible, for the construction sequence of the water management infrastructure based on the planned site development.

For the current water management strategy, WSP recommended evaluating to have a polishing pond within the ETP in order to increase the robustness of the water management system.

The current water management strategy considers that all collected contact water is pumped towards the BC-11 pond before the ETP. It is possible that contact water from different sources have different contaminant concentrations. It may be advantageous to have a ETP water intake at BC-01 and provide a connection between ponds BC-12 and BC-01 to treat the water without passing through BC-11. This should be considered when water quality analysis become available.

Considering that water will need to be managed during the winter period, Nemaska must ensure that the system does not freeze and remains operational all year long.

Ponds and pump systems should be equipped with flow and level monitoring instruments, necessary for water management during operations.

Climate and runoff monitoring should be undertaken during mine operations to reduce uncertainties in the water balance predictions.

Given the limitation of the discharge flow into the Nemiscau River. At the beginnings of the operation, Nemaska will have to evaluate the possibility of increasing the effluent capacity.

The design of the water management infrastructure must be based on improved topographic survey data.

23.7.3 Engineering Activities

It is recommended to continue with the engineering activities prior to implementation of the Project. The activities would include the following so as not to delay the construction activities and complete the Project on time:

- Complete the contracting strategy for the Project.
- Preparation of equipment specifications for long delivery items, proceed to tender, recommendation of bidders and prepare purchase orders.
- Preparation of contractual documents for civil and concrete works, proceed to tender and recommendation of contractors and prepare contracts.

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24.10 **Economic Analysis**

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25 RELIANCE ON INFORMATION PROVIDED BY THE REGISTRANT

Through its ownership stake in Nemaska, Livent considers information provided to the QPs by Nemaska to have been provided on Livent's behalf. The preparation of this report by the QPs is supported by certain information, including but not limited to published literature, technical reports prepared by other parties (including Nemaska's prior consultants and contractors), laboratory analyses performed by commercial laboratories, and operational data supplied by Nemaska.

The QP's opinions contained within this report are based in part upon information provided by Nemaska on behalf of the Registrant which was deemed appropriate for use. The QPs consider it reasonable to rely on this information because the registrant is the most qualified entity to provide current information on its stakeholder engagements for permitting; and macroeconomic factors that affect its costs and forward-looking economic analyses.

In accordance with the provision set forth in §229.1302(f), the QPs relied on information provided by Nemaska for subject matters outside their areas of expertise. Such information provided by the registrant was relied on in the preparation of the portions of the following sections of this report:

- Market studies (Section 16)
- Environmental Studies, permitting and plans negotiations or agreements with local invidividuals or groups (Section 17)
- Operating and capital costs (Section 18)
- Forward-looking economic analyses (Section 19).

26 SIGNATURE PAGE

Signed on behalf of:

BBA Inc.

By: /s/ Jeffrey Cassoff
Name: Jeffrey Cassoff
Title: Principal Mining Engineer

DRA Americas Inc.

By: /s/ Daniel M. Gagnon
Name: Daniel M. Gagnon
Title: VP Mining, Geology and GM, Montréal office

SGS Geological Services

By: /s/ Marc-Antoine Laporte
Name: Marc-Antoine Laporte
Title: Global Business Manager

Signed in an individual capacity:

By: /s/ Carl Pednault
Name: Carl Pednault
Title: Principal Mine Waste Engineer

By: /s/ Marc Rougier
Name: Marc Rougier
Title: Principal Geotechnical Engineer