SAYONA QUÉBEC

AUTHIER LITHIUM PROJECT

UPDATED DEFINITIVE FEASIBILITY STUDY

October 2019

In collaboration with:

BBA

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1. SUMMARY

1.1 Property Description and Location

The Authier property is located in the Abitibi-Témiscamingue Region of the Province of Québec and is centered in a well-developed mining region with associated resource industry support facilities and services. The towns of Rouyn-Noranda, Val-d'Or and Amos have populations of between 10,000 and 55,000 and are well known for their mining history. An experienced mining workforce and other mining related support services will come from these nearby cities. Val-d'Or, Amos, and Rouyn-Noranda have well established hospitals, regional airports, schools, accommodation and telecommunications, which are readily accessed from the project site.

The site can be accessed by a 5 km rural road, which connects to a sealed highway that links to Amos, Val-d'Or and Rouyn-Noranda as well as Montréal 590 km to the south.

Québec is a major producer of electricity as well as one the largest hydropower generators in the world. Green and renewable, it is well distributed through a reliable power network. Power will be accessed about 5 km to the south-east of the project site via an electricity grid supplied by low-cost, hydro-electric power.

CN Rail has an extensive rail network throughout Canada. One of the closest rail sidings connecting to an export shipping ports at Cadillac and Amos, both located roughly 20 km from the Authier site. The rail network connects to Montréal and Québec City, and to the west through the Ontario Northland Railway and the whole of the North American rail system.

1.2 Geology and Mineralization

The Authier project hosts two separate mineralized spodumene-bearing pegmatite systems including, Authier and Authier North. Authier is 1,100 m long, striking east-west, with an average thickness of 25 m, ranging from 4 m to 55 m, dipping at 40° to 50° to the north. The deposit outcrops in the central - eastern sector and then extends under up to 10 m of cover in the western and eastern sectors.

Authier North, located 400 m north of main Authier pegmatite, is approximately 500 m in strike length, 7 m average width, dipping 15° to 20° to the north. The Authier and Authier North pegmatite dykes remain open in all directions. A magnetic geophysical survey has demonstrated that Authier mineralization is hosted within a strong east-west trending magnetic low anomaly. Future exploration will focus on identifying extensions of know mineralization within this structural feature.

The lithium mineralization at Authier is related to multiple pulses of spodumene-bearing quartz-feldspar pegmatite. Higher lithium grades are related with high concentrations of fine and mid-to-coarse spodumene crystals (up to 4 cm long axis) in a mid-to-coarse grained pegmatite facies.

1.3 Drilling

The Authier project has been subject to more than 31,000 m of drilling. Between 2010 and 2012, Glen Eagle completed 8,990 m of diamond drilling in 69 diamond drill holes (NQ diameter) of which 7,959 m were drilled on the Authier deposit; 609 m (5 DDH) were drilled on the Northwest and 422 m on the south-southwest of the Property.

Sayona Québec has completed three phases of drilling since acquisition of Authier to Glen Eagle including 81 holes for 11,367.5 m. From this database, 199 drill holes were used for the solid modeling and updated resource estimate. All holes completed by Sayona in the three programs have been Diamond Core Drill holes (DDH) using HQ or NQ core diameter size with a standard tube and bit. Core diameter for metallurgical drilling was done using PQ core for 680 m and HQ core for 89.5 m of HQ core. Condemnation drilling was done using NQ core diameter.

Core was oriented using a Reflex ACT III tool for Phase 1 and Phase 2 whereas Phase 3 diamond core was not oriented. The drilling programs have been subject to very robust QA/QC procedures.

1.4 Mineral Processing and Metallurgical Testing

Samples from the Authier deposit have been subjected to several metallurgical test work programs (1999, 2012, 2016, 2017, and 2018). In 1999, testing on a 40-t bulk sample produced concentrate grading between 5.78% and 5.89% Li_2O with lithium recoveries between 68% and 70% from a sample with average head assay of 1.14% Li_2O.

In 2012, Glen Eagle tested a 270 kg sample from drill core. The batch tests incorporated magnetic separation and spodumene flotation without mica pre-flotation. Tests produced concentrate grading 6.4% Li_2O with 85% recovery.

In 2016, Sayona completed metallurgical testing on a representative 430 kg sample (including 5% mine ore dilution). Concentrate grades varied from 5.4% to 6.1% Li_2O at recoveries between 71% and 79%. Ore dilution had a negative impact on flotation performance.

In 2017, two representative samples were prepared, and flotation tests were undertaken to assess the impact of dilution and processing with site water. The program demonstrated the ability to produce concentrate grade of 6.0% Li_2O at recoveries greater than 80%.

A pilot plant testing program was undertaken in 2018 on a roughly 5 t sample. Two composite pilot plant feed samples were prepared from drill core to represent Years 0 to 5 and Years 5+ of the operation. Batch and locked-cycle-testing was undertaken on each composite prior to pilot plant operation. Optimized batch flotation tests produced 6.0% Li_2O concentrate grade at 82% recovery. Locked-cycle test results showed Composite 1 achieved 5.9% Li_2O concentrate grade at 84% recovery; and Composite 2 achieved 5.9% Li_2O concentrate grade at 83% recovery.

The pilot plant flowsheet included grinding, de-sliming, magnetic separation, mica and spodumene flotation. The optimized flowsheet produced a 6% Li_2O concentrate at a 79% lithium recovery. There was some variability in the results over the total program. For the optimized pilot plant flowsheets, Composite 1 produced concentrate ranging from 5.9% to 6.0% Li_2O with recoveries ranging from 67% to 71%. For Composite 2, concentrate grade ranged from 5.8% to 6.2% Li_2O with lithium recovery from 73% to 79%.

An optimization test program was undertaken in 2018 to investigate the impact of spodumene conditioning on flotation performance. The results reiterated the necessity to condition at high solids density (>55% solids) to achieve target lithium grade and recovery.

1.5 Mineral Resource Estimate

An independent JORC Mineral Resource (2012) estimate for the Authier deposit is reported as per Table 1-1.

Table 1-1: Authier JORC Mineral Resources Estimate (0.55% Li_2O Cut-off Grade) Inclusive of Reserves

Category	Tonne (Mt)	Grade (% Li_2O)	Contained Li_2O (t)
Measured	6.58	1.02	67,116
Indicated	10.60	1.01	107,060
Measured and Indicated	17.18	1.01	174,176
Inferred	3.76	0.98	36,848

The mineral resource estimates for the Authier deposit includes Authier Main and Authier North pegmatites and is based on 1.5 m composite analytical data, no top-cut, and a 0.55% Li_2O cut-off grade. The estimation was based on an Inverse Distance Cubed (ID^3) interpolation.

A block size of 3 m (N-S) by 3 m (E-W) by 3 m (vertical) was selected for the resource block model of the project based on drill hole spacing, width and general geometry of mineralization but primarily by the selected SMU from the advanced feasibility study. Three dimensional mineralized wireframes were used to domain the Li_2O data using a 0.4% Li_2O cut-off over a minimum drill hole interval length of 2 m as guideline to define the width of mineralized interpretations on sections, i.e., polygons. Sample data was composited to 1.5 m down hole lengths. Variable search ellipse orientations were used to interpolate the blocks.

For the Measured resource category, the search ellipsoid was 50 m (strike) by 50 m (dip) by 25 m with a minimum of seven composites in at least three different drill holes, with a maximum of two composites per hole. An ellipse fill factor of 60% was applied to the measured category, i.e. only 50% of the blocks were tagged as measured within the search ellipse. For the Indicated category, the search ellipsoid was twice the size of the measured category ellipsoid using the same composites selection criteria. An ellipse fill factor of 85% was applied to the Indicated category. All remaining blocks were in the Inferred category.

1.6 Mining

The project's life-of-mine (LOM) plan and subsequent ore reserves are based on an average lithium concentrate selling price of 693 $USD/t at 6.0% Li_2O purity and based on an exchange rate of 0.76 USD to 1 CAD.

Development of the LOM plan included pit optimization, pit design, mine scheduling and the application of modifying factors to the Measured and Indicated portion of the in-situ mineral resource. Tonnages and grades are reported as run-of-mine (ROM) feed at the crusher and are inclusive of mining dilution, geological losses and operational mining loss factors. Multiple mining phases were developed for the UDFS. The LOM was developed based on a range of constraints and results in a mine life of 14 years. A summary of the LOM is shown in Table 1-2.

Table 1-2: Life-of-mine Plan

Physicals	Units	Pre-Prod	Production														LOM Total
		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	
Total Moved (Expit + Rehandle)	**(kt)**	**794**	**3,599**	**3,958**	**6,625**	**9,617**	**12,582**	**14,657**	**14,662**	**12,440**	**9,623**	**6,636**	**3,183**	**2,396**	**2,116**	**1,718**	**104,608**
Total Expit	**(kt)**	**794**	**3,020**	**3,296**	**5,964**	**8,956**	**11,920**	**13,995**	**14,000**	**11,778**	**8,961**	**5,974**	**2,522**	**1,734**	**1,454**	**1,168**	**95,536**
Expit Waste Rock	(kt)	538	2,068	1,391	4,142	7,322	10,429	11,672	13,098	10,895	8,078	5,092	1,639	852	572	435	78,223
Expit Overburden	(kt)	257	180	1,022	939	751	609	1,440	20	0	0	0	0	0	0	0	5,217
Expit Ore to Mill	(kt)	0	193	221	221	221	221	221	221	221	221	221	221	221	221	183	3,024
Expit Ore to ROMpad	(kt)	0	579	662	662	662	662	662	662	662	662	662	662	662	662	550	9,072
Expit Ore	(kt)	0	772	883	883	883	883	883	883	883	883	883	883	883	883	733	12,096
Expit Ore Grade	(% Li_2O)	0.00	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	1.00
Stripping Ratio	**(t_{waste}:t_{RoM})**	**0.0**	**2.9**	**2.7**	**5.8**	**9.1**	**12.5**	**14.9**	**14.9**	**12.3**	**9.2**	**5.8**	**1.9**	**1.0**	**0.6**	**0.6**	**6.9**
Total Stockpile Rehandling	**(kt)**	**0**	**579**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**550**	**9,072**
Stockpile to Mill	(kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stockpile to Mill Grade	(% Li_2O)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
ROMpad to Mill	(kt)	0	579	662	662	662	662	662	662	662	662	662	662	662	662	550	9,072
ROMpad to Mill Grade	(% Li_2O)	0.00	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	1.00
Crusher Feed	**(kt)**	**0**	**772**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**733**	**12,096**
Crusher Feed Grade	(% Li_2O)	0.00	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	1.00

Table 1-3 presents the ore reserves for Authier Lithium project which are based on the results of the 2018 DFS (no updates were made to the reserves for the feasibility study update). Ore reserves are based on measured and indicated mineral resources contained within the final pit design following the application of modifying factors. Reserves are reported as ROM feed tonnes at the crusher above a cut-off grade of 0.55% Li_2O.

Table 1-3: Authier DFS Ore Reserves

Category	Tonne (Mt)	Grade (% Li_2O)	Contained Li_2O (t)
Proved	6.10	0.99	60,390
Probable	6.00	1.02	61,200
Total Proved & Probable	**12.10**	**1.00**	**121,590**

1.7 Recovery Methods

BBA designed a concentrator to process roughly 883,000 tpa of ore using conventional flotation technology suitable for a pegmatite orebody that will be located near the open-pit.

Run-of-mine ore (ROM) will be transported from the mine to the crushing plant. The ore will be crushed to a P_{80} of 9 mm in three stages of crushing. The crushed ore will be stored under a protected dome and conveyed to the ball mill. Crushed ore will be ground using a single-stage ball mill to a P_{80} of 180 µm. The ground ore will be passed through a magnetic separation circuit to remove iron-bearing silicate minerals and then de-slimed prior to mica flotation. Following mica flotation, the slurry will flow to an attrition scrubber and hydrocyclones for de-sliming prior to spodumene flotation. The plant will produce 6.0% Li_2O concentrate with 78% lithium recovery.

Magnetic and mica concentrates, slimes, and spodumene flotation tailings will be thickened and filtered prior to co-disposal with mine waste (dry stacking). Truck and loading units will be used to dispatch tailings to the waste rock facility. The spodumene concentrate will be filtered to roughly 6% moisture. The dried spodumene concentrate will be stored in a covered storage area prior to bulk shipment to a port and/or other Canadian off-taker.

The plant will produce a LOM average of roughly 112,700 t of 6.0% Li_2O concentrate suitable for sale to lithium conversion plants that supply feed-stock to lithium battery manufacturers.

1.8 Waste Dumps and Tailings

During the lifespan of the open pit mine, a total of 35.9 Mm^3 of waste rocks and 4.38 Mm^3 of tailings will be generated for a total of 40.28 Mm^3. Sayona Québec has opted for a co-disposal method to store tailings produced at the concentrator and waste rocks from the mine. The co-disposal strategy consists of using waste rocks to construct peripheral berms and peripheral roads and confining filtered tailings into waste rock cells. This method has the advantage of increasing the stockpile's global stability and the water drainage efficiency while ensuring long-term physical and geochemical stability. Furthermore, the co-disposal method reduces the site's footprint.

1.9 Project Infrastructure

The project infrastructure includes a process plant, overburden stockpile, ROM pad, waste and dry tailings co-disposal pile, service and haul roads, offices complex, mechanical shop, water treatment plant, warehouse, electrical distribution facilities, water storage and management ponds, fuel and explosive storage and communication systems (see Figure 1-1).

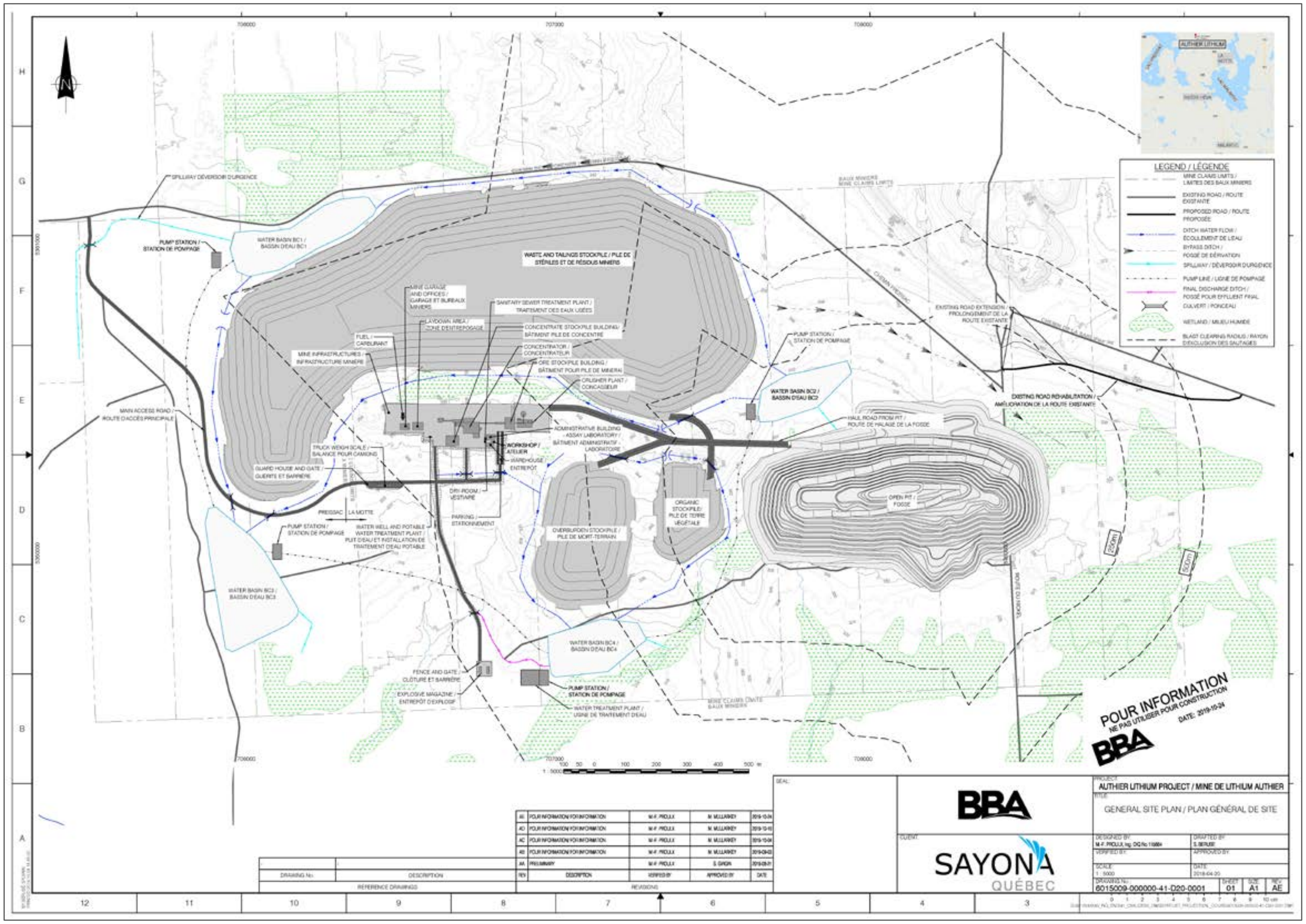


Figure 1-1: Site Layout

1.10 Market Analysis and Offtake Agreements

The market fundamentals for lithium products remain strong, with recent pricing forecasts being upgraded and indicating continued long-term strength in pricing for hydroxide, carbonate and spodumene concentrate. The study reviewed lithium industry pricing forecasts from leading investment groups and were compiled by Hatch Consultants (Market Study Division – London and New York). The pricing forecast is presented in Table 1-4 and the base case pricing was retained for the UDFS.

Table 1-4: Spodumene Concentrate Price Forecast 2019-2026 (USD)

	2019	2020	2021	2022	2023	2024	2025	2026
Base Case	609	545	564	558	644	713	712	730
High Case	670	599	620	641	740	820	818	839
Low Case	579	518	535	530	579	642	640	657

1.11 Environmental Studies

Environmental baseline studies were conducted in 2012 by Dessau. Surface water was sampled (2017, 2018 and 2019) as part of the fish habitat study in the streams that are likely to be impacted by the project. The results were compared to the provincial criteria for the protection of aquatic life for chronic toxicity established by the Ministère l’Environnement et de la Lutte contre les changements climatiques (MELCC) and the Canadian aquatic life protection guidelines for long term exposure of the Canadian Council of Ministers of the Environment (CCME). Laboratory analysis revealed that the water sampled is acidic, with low to medium alkalinity, relatively limpid, and slightly mineralized. During the sampling campaigns between December 2016 and September 2019, 14 to 21 piezometers were sampled for underground water quality analyses. Samples collected were analyzed for a variety of parameters including metals, nutrients, major anions and cations, volatile compounds, polycyclic aromatic hydrocarbons and C_{10}-C_{50} petroleum hydrocarbons.

Characterization of wetlands began in 2017 and completed in June 2018 and July 2019. A field inventory of snakes, salamanders and anuran was conducted by SNC-Lavalin during the summer 2017 and spring 2018. In 2012, visual characterizations were made for five ponds and one stream in the Preissac Lake watershed. An ichthyological fauna inventory was completed during the characterization of bodies of water and streams during the summer 2017 and 2018.

The southern part of the St-Mathieu-Berry esker is located within the area of influence of the mine. However, it is located fifty metres lower than the esker and isolated from it by a bedrock. As such, the Authier project will not threaten, <u>in any way and under any circumstances,</u> the water quality of this esker. Also, the effects of mine dewatering on residential wells are negligible since those wells are located at more than twice the radius of influence of the open pit. Groundwater quality should not be affected as the open pit will act like a drain, intercepting all possibly contaminated groundwater.

The project will create temporary and permanent modifications to the site. During the environmental assessment process, the impacts and mitigation measures were established for the physical, biological, and human environments. During operations, a monitoring program will be implemented. This environmental monitoring program aims to ensure compliance with the environmental laws and regulations and the conditions of the certificates of authorization that will be issued by the MELCC or conditions of the mining lease issued by the MERN.

1.12 Permitting

The global certificate of authorization frames the environmental component of the project, in respect to the regulation respecting the environmental assessment and review of certain projects (CQLR, cQ2, r23.1). The projects listed in Schedule 1 are subject to the environmental impact assessment and review procedure under the Environment Quality Act (article 31.1), and also to go through a Public hearing process (BAPE).

In addition, in accordance with Québec's Mining Act and Environmental Quality Act, permits are also required in order to build and operate the mine. A mining lease is required from the Ministry of Energy and Natural Ressources (MERN). From a federal point of view, no Environmental Impact Assessment is required. Other permits or leases will have to be obtained depending on planned development activities at the site. Other permits may also be required at the regional/municipal level.

1.13 Reclamation and Closure

In accordance with the Mining Act of Québec, closure and restoration requirements have been developed to return the Authier Project site to an acceptable condition, ensuring that the site is safe, and the surrounding environment is protected.

The cost of restoring the Authier site is estimated to be $10.9M. As required by the Ministère des Ressources Naturelles ("MERN"), this cost estimate includes the cost of site restoration, the post-closure monitoring as well as engineering costs (30%) and a contingency of 15%. In accordance with the regulations, Sayona intends to post a bond as a guarantee against the site restoration cost.

1.14 Transport and Logistic

Sayona will hire a third-party contactor to provide a turnkey transport service to transport the concentrate from the mine to the port of Montreal using a fleet of 40 t b-train tractor-trailers. A well-managed dedicated truck fleet would allow Sayona to have a complete control on the tonnage to be shipped according to production. Approximately nine 40-t loads will leave the mine daily.

1.15 Human Resources

The project will benefit from the local human resources and services in the Abitibi region with a population of over 120,000 people. The following table presents the project workforce requirements over the mine life.

Table 1-5: Mine Life Workforce

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mining	24	42	47	57	77	92	107	101	97	86	68	37	32	30	30
Mine Maintenance	10	11	13	16	20	25	29	27	26	23	18	11	8	7	6
Processing	9	22	22	22	22	22	22	22	22	22	22	22	22	22	22
Plant Maintenance	1	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Technical Services	5	7	7	7	7	7	7	7	7	7	7	5	5	5	5
Admin & Others	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total	**53**	**93**	**100**	**113**	**137**	**157**	**176**	**168**	**163**	**149**	**126**	**86**	**78**	**75**	**74**

1.16 Capital and Operating Cost Estimates

The following tables summarize the capital, sustaining capital and operating costs resulting from the UDFS work. The present costs estimate meets AACE Class 3 – estimate type criteria, which is usually prepared to establish preliminary costs forecast and assess the profitability potential of a project. The accuracy range for the costs estimate developed in this study has an expected accuracy range of -10% on the low side and +15% on the high side. Please note numbers in the following tables might not add up due to rounding.

Table 1-6: Initial Capital Cost Estimate Summary

Year		-1	1*	Total
Mine Preproduction	M$	3.2	0.0	**3.2**
Mine Equipment- Financed	M$	0.9	0.0	**0.9**
Mine Equipment- Purchased	M$	1.3	0.0	**1.3**
Process Plant and Infrastructure	M$	69.0	46.0	**115.0**
Total Initial Capital Costs	**M$**	**74.5**	**46.0**	**120.5**

**Capital costs in year 1 for all items except for Process Plant and Infrastructure are captured under sustaining capital costs. The Process Plant and Infrastructure costs are assumed to be spent over years -1 and 1 in a split of 60% and 40% respectively.*

Table 1-7: Process Plant and Infrastructure Capital Cost Summary

Area	Value	Cost
Direct Costs		
Mine Infrastructure (Road)	M$	0.5
Process Plant	M$	65.8
Tailings Storage Facility	M$	9.2
Common Services	M$	3.3
On Site Infrastructure	M$	4.1
Off Site Infrastructure	M$	2.8
Total Directs	**M$**	**85.6**
Indirect Costs		
Owner's Costs	M$	3.7
Indirects (EPCM, Overhead, General)	M$	13.6
Contingency	M$	12.1
Total Indirects	**M$**	**29.4**
Total Process Plant and Infrastructure Costs	**M$**	**115.0**

Indirect costs for EPCM, Owner’s team consultants and contingency were estimated using cost allowances. EPCM costs assume 11.15% of total direct costs within acceptable industry range at the feasibility level. Owner’s team consultants’ costs were calculated using 2% of total direct costs as benchmarked on other projects. A contingency value of 12% of total project costs has been assumed, which is in line with industry standard values.

Table 1-8: Sustaining Capital Cost Estimate Summary

Year		1	2	3	4	5	6	7	8	9	10	11	12	13	14	LOM
Mine Equipment- Financed	M$	1.8	1.8	3.2	4.1	4.8	6.2	5.4	4.1	3.9	2.9	2.0	1.3	0.1	0.0	**41.6**
Mine Equipment- Purchased	M$	0.3	0.1	0.2	0.1	0.0	0.2	1.2	0.0	0.0	0.0	0.0	0.4	0.0	0.0	**2.5**
Process Plant Mobile Equipment	M$	1.2	0.0	0.0	0.0	0.0	0.0	1.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**2.3**
Remaining Site Preparation Activities	M$	9.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**9.8**
Building Capital Rental	M$	1.0	0.8	0.8	0.8	0.8	0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**4.7**
Tailings and Water Management Infrastructure	M$	0.0	0.0	0.0	10.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**10.4**
Water Treatment Plant - Capital Rental	M$	0.2	0.2	0.2	0.2	0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**1.5**
Wetland Compensation	M$	0.3	0.3	0.3	0.3	0.3	0.3	0.2	0.2	0.2	0.3	0.3	0.3	0.4	0.4	**4.1**
Royalties buyback	M$	3.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**3.0**
Reclamation and Closure	M$	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	10.9	**10.9**
Total Sustaining Capital Costs	**M$**	**17.5**	**3.2**	**4.7**	**15.9**	**6.4**	**7.2**	**7.9**	**4.2**	**4.1**	**3.3**	**2.4**	**2.0**	**0.5**	**11.3**	**90.7**

Table 1-9: Operating Cost Estimate Summary

Cost Area	LOM (M$)	$/t Milled	CAD/t Conc Prod (Dry)	USD/t Conc Prod (Dry)
Open Pit Mining	302.3	25.0	191.5	145.5
Mineral Processing (Includes mobile eqpt)	226.7	18.7	143.7	109.2
Analytical Laboratory	12.7	1.1	8.1	6.1
Water Treatment	16.1	1.3	10.2	7.7
Tailings Management	6.0	0.5	3.8	2.9
General and Administration	67.1	5.5	42.5	32.3
Total Onsite Costs	**630.9**	**52.2**	**399.7**	**303.8**
Royalties	20.4	1.7	12.9	9.8
Total Onsite Costs + Royalties	**651.3**	**53.8**	**412.7**	**313.6**
Concentrate Transport and Logistics Costs	108.5	9.0	68.8	52.3
Total Operating and Shipping Costs	**759.8**	**62.8**	**481.4**	**365.9**

The low capital and operating costs associated with the project are attributed to:

- Close proximity to established infrastructure – power lines (5 km), sealed national highways (5 km), rail (20 km), local water supplies, and skilled local workforce;
- No requirement for on-site infrastructure such as accommodation camps and power plants;
- Low electricity costs in Québec; and
- Simple deposit geology, mining and production processes.

1.17 Economic Analysis

An economic, life-of-mine cash flow model of the project was constructed using the production mining and processing production schedules developed for the UDFS. The key outcomes of the economic evaluation for 100% of the project, before financing costs, are presented in Table 1-10.

Table 1-10: Economic Analysis Summary

Authier Lithium Project Highlights		
Description	**Unit**	**Value**
Average Annual Ore Feed to the Plant	t	874 594
Average Annual Grade to the Plant	% Li_2O	1.0
Annual Average Spodumene Production - Dry (6% Li_2O)	t	114 116
Li_2O Recovery	%	78

Authier Lithium Project Highlights		
Description	**Unit**	**Value**
Life of Mine (LOM)	year	13.8
LOM Strip Ratio	waste to ore	6.90
Average Spodumene Price	USD/t	693
LOM Operating Costs (mine gate) – Excluding Royalties	CAD million	631
LOM Transport and Logistics Costs (mine to port)	CAD million	109
Royalties purchase	CAD million	3.0
Initial Capital Costs	CAD million	120
LOM Capital Costs	CAD million	211
Royalties	CAD million	20.4
Total Net Revenue	CAD million	1 412
Total Project EBITDA	CAD million	461
Average Life of Mine Cash Costs (mine gate) – Excluding Royalties	CAD/t	400
Average Life of Mine Cash Costs (FOB Port of Montreal) – Excluding Royalties	CAD/t	469
Average Life of Mine Cash Costs (FOB Port of Montreal) – Excluding Royalties	USD/t	356
Net Present Value (real terms @ 8% discount rate)	**CAD million**	**216**
Pre-tax Internal Rate of Return	**%**	**33.9**
Project Payback Period (After Start of Production)	year	2.7
Exchange Rate	CAD:USD	0.76

A pre-tax sensitivity analysis was conducted on the economic model to test changes in key economic assumptions, namely commodity price, Li_2O recovery, operating cost, capital cost, and exchange rate. The project's pre-tax Net Present Value (NPV) was most sensitive to the commodity price and least sensitive to the Li_2O recovery and CAPEX.

1.18 Project Implementation and Execution

The project execution schedule is developed to a feasibility level and therefore preliminary in nature. Detailed execution planning will be undertaken at the beginning of the project execution phase.

The preliminary project execution schedule, developed during the DFS and described herein, covers the period from the end of the DFS to the achievement of commercial operation in Q3 2022. Table 1-11 summarizes key milestones of the execution strategy.

Table 1-11: Preliminary Schedule

Activity	Start Date	Completion Date
Definitive Feasibility Study, incl. revised mine plan	-	Completed
Decision from Government on permitting	-	Q4 2020
Construction permit issuance (approximate date)	-	Q1 2021
Operation permits issuance (approximate date)	-	Q3 2021
Early work phase (engineering and procurement only)	Q4 2020	Q1 2021
Execution phase to be started	Q1 2021	-
Detailed engineering and procurement	Q4 2020	Q1 2022
Construction (when permits are issued)	Q2-Q3 2021	Q3 2022
Commissioning	Q3 2022	Q3 2022
Hand-over to Operation	-	Q3 2022
Start of mining operations	Q3-Q4 2022	-

1.19 Risk and Opportunity

Significant risks are associated with the development, commissioning and operation of a mine. The 2018 (DFS) risk assessment was updated during the UDFS to identify the critical project risks and develop mitigation strategies. A significant amount of drilling has been undertaken to establish the resource and metallurgical testing has led to the development of a proven processing route to produce a saleable spodumene concentrate. The Authier project is in an advanced stage of development and a major focus is risk mitigation which will allow the project to progress successfully. Sayona is now expanding its in-house technical and project delivery capabilities to manage the outcomes.

1.20 Conclusion and Recommendations

The UDFS incorporates the 2018 JORC resource, results from many technical optimisation programs, and realignment of pricing to reflect more recent industry forecasts. The UDFS confirms the technical and financial viability of constructing a simple, open-cut mining operation and processing facility producing spodumene concentrate. The positive UDFS demonstrates the opportunity to create substantial long-term sustainable shareholder value at a low capital cost. Given the technical feasibility and positive economic results of the UDFS, it is recommended to continue the work necessary to develop the project.

2. INTRODUCTION

2.1 General and Purpose of the Technical Report

The following document was prepared to provide an update to the technical Definitive Feasibility Study (DFS, 2018) of the Authier Lithium mineralization on the Authier Property (Property) in compliance with JORC Code 2012. The main objectives of the Updated Definitive Feasibility Study (UDFS) were to:

- Revise the mine and plant design to a capacity of 2600 tpd (from ca. 1900 tpd in the DFS);
- Update the site layout to modify the design of the co-deposition waste piles;
- Further the design related to site water management;

and ultimately demonstrate that the Authier Lithium project has sufficient merit from a technical, environmental and economic standpoint to justify moving forward with the EPCM phase. The purpose of the UDFS was to evaluate the potential for mining and milling processes and all associated infrastructure required for development of the project.

This UDFS is based upon developing the project over a 14-year production period, using a conventional open-pit truck and shovel operation and a conventional milling process to produce a spodumene concentrate (6% Li_2O).

BBA provided engineering services for the UDFS on the Authier Lithium project in collaboration with SNC-Lavalin (SNCL) and Hatch. BBA performed the ore reserve estimates and co-ordinated mining and milling activities, associated infrastructure, capital and operating cost estimates and economic analysis. All aspects related to waste, tailings and water management and the associated infrastructure and capital and operating cost were managed by BBA based on the concept developed by SNCL at the DFS phase of the project.

2.2 Study Contributors

Table 2-1: List of Qualified Persons and Their Respective Sections of Responsibility

	DFS SECTION	AUTHOR
1	SUMMARY	Sayona/BBA
2	INTRODUCTION	Sayona/BBA
3	RELIANCE ON OTHER EXPERTS	Sayona/BBA
4	PROPERTY DESCRIPTION AND LOCATION	Dr. Gustavo Delendatti
5	ACCESSIBILITY, PHYSIOGRAPHY, CLIMATE, INFRASTRUCTURE, SURFACE RIGHTS	Dr. Gustavo Delendatti
6	PROJECT HISTORY	Dr. Gustavo Delendatti

	DFS SECTION	AUTHOR
7	GEOLOGICAL SETTING AND MINERALIZATION	Dr. Gustavo Delendatti
8	DEPOSIT TYPES	Dr. Gustavo Delendatti
9	DRILLING	Dr. Gustavo Delendatti
10	SAMPLE PREPARATION, ANALYSIS AND SECURITY	Dr. Gustavo Delendatti
11	DATA VERIFICATION	Dr. Gustavo Delendatti
12	METALLURGICAL TESTING	Sayona
13	MINERAL RESOURCE ESTIMATES	Dr. Gustavo Delendatti
14	GEOTECHNICAL	
	Waste Dumps	SNCL
	Pit Slopes	Journeaux Assoc.
15	ORE RESERVE ESTIMATES	
	Dilution and Ore Loss	BBA
	Pit Optimization	BBA
	Mine Design	BBA
16	MINING METHODS	
	Cutback Design	BBA
	Production Schedule	BBA
	Mine Operation and Equipment Selection	BBA
	Mine Workforce Requirements	BBA
17	RECOVERY METHODS	
	Process Flow Sheet	BBA
	Unit Operation	BBA
	Reagents and Utilities	BBA
	Layout	BBA
	Process Workforce Requirements	BBA
18	TAILINGS AND WASTE ROCKS MANAGEMENT	
	Waste Dumps Design	BBA/SNCL
	Co-disposition Strategy	BBA/SNCL
19	PROJECT INFRASTRUCTURE	
	Off-Site Infrastructure	Sayona
	On-Site Infrastructure	BBA
	Water Management	BBA
	Waste Management	SNCL
20	MARKET STUDIES AND CONTRACTS	Hatch
21	ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT	Sayona
22	TRANSPORT, LOGISTICS AND PORT INFRASTRUCTURE	Sayona

	DFS SECTION	AUTHOR
23	HUMAN RESOURCES	BBA/Sayona
24	CAPITAL COST ESTIMATE	
	Mining	BBA
	Process Plant	BBA/ASDR
	Common Services	BBA
	Infrastructures	BBA/ASDR
	Waste Dumps, Tailings and Water Management	BBA
	Pre-production	BBA
	Indirect	BBA
	Salvage Value	BBA
	Mine Closure	SNCL
25	OPERATING COST ESTIMATE	
	Mining	BBA
	Process	BBA
	G&A	Sayona
	Transport	Sayona
26	ECONOMIC ANALYSIS	
	CFs Forecast, NPV and Sensitivity Analysis	BBA
	After-Tax Analysis	Sayona
27	PROJECT IMPLEMENTATION AND EXECUTION	
	Project and Execution Strategy	Sayona
	Roles, Responsibilities and Owners Team Organization Chart	Sayona
	EPCM	Sayona
	Commissioning	Sayona
28	RISK AND OPPORTUNITY	Sayona
29	ADJACENT PROPERTIES	Dr. Gustavo Delendatti
30	OTHER RELEVANT DATA AND INFORMATION	Dr. Gustavo Delendatti
31	CONCLUSIONS AND RECOMMENDATIONS	Sayona
32	REFERENCES	BBA

2.3 Units and Currency

The following units and currency are used throughout this report:

- All units are metric, unless noted otherwise;
- All currency is in Canadian dollars (CAD), unless noted otherwise.

2.4 Accuracy of Cost Estimate

The present Capital Cost Estimate pertaining to this study meets AACE Class 3 – estimate type criteria, which is usually prepared to establish a preliminary Capital Cost Forecast and assess the profitability potential of the project. This will allow management, or the project sponsor, to obtain authorization for funds for further project definition. As such, this estimate forms the initial "Control Estimate" against which subsequent cost estimates developed in the next study and engineering phases will be compared and monitored. The accuracy range for the Capital Cost Estimate and the Operating Cost Estimate developed in this study has an expected accuracy range of -10% on the low side and +15% on the high side.

2.5 Site Visits

Two BBA employees within the Earth and Infrastructure group visited the Authier Lithium property on August 30, 2019: Yves Thomassin, Senior ESIA specialist and Vahid Marefat, Engineer.

3. RELIANCE ON EXPERTS

The authors of the feasibility study relied upon information provided by experts, who were not authors of the report. The authors of the various sections of the report believe that it is reasonable to rely upon these experts, based on the assertion that the experts have the necessary education, professional designation and related experience on matters relevant to the technical report.

- **Richelieu Hydrogéologie Inc.**: Richelieu Hydrogéologie was founded in 2005 to provide hydrogeological consulting services. The company specializes in numerical modeling of underground water flows around mines, quarries and sand pits, e.g. evaluation of dewatering rates for open pits, optimization of dewatering well spacing, evaluation of the impact of groundwater pumping, as well as risk assessment associated with the transport of dissolved contaminants.
- **SGS Geostat**: SGS Geostat is known globally as the expert in ore body modeling and reserve evaluation with over 35 years of experience, providing the mining industry with computer-assisted mineral resource estimation services using cutting edge geostatistical techniques).
- **Bolloré Logistics**: Bolloré Logistics provided services, which extend across five core categories: Multimodal Transport, Customs and Regulatory Compliance, Logistics, Global Supply Chain and Industrial Projects. Its performance stems from a worldwide network of experts and from value-added integrated information systems, which afford complete visibility on all operations throughout the entire supply chain.
- **Services Forestiers et Exploration GFE inc.**: GFE did provide technical personnel, between spring and fall 2019, to collect samples of water in the various creeks at the mine site.
- **Services d'ingénierie Norinfra inc.:** Norinfra did work on the environmental evaluation EES1 and soil characterization. Norinfra are well known in Abitibi and do provide engineering services to numerous mining companies.
- **Groupe-conseil Nutshimit-Nippour:** This First Nation consulting company, a member of Groupe Desfor, did contribute to environment expertise and to the landscaping architecture and related matters. Their expertise of the local Algonquin community and other first nations particularities is bringing a unique and complementarity expertise to this study.
- **Hatch** Consultants, through their office specialized in market studies for the mining sector located in New York, USA and London, UK who provided the research, analysis and the writing of Chapter 20 related to the lithium price and comodity.
- **Consultant GCM, Del Degan, Massé et Associés inc., SNC Lavalin, Patricia Desgagné, anthropologist, Martin Pérusse, biologist and Lamont Expert Conseil** also participated in the drafting of the Environmental Impact Assessment.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Authier property is located in the Abitibi-Témiscamingue Region of the Province of Québec, approximately 45 km northwest of the city of Val-d'Or and 15 km north of the town of Rivière-Héva. The centre of the Property is situated on NTS sheet 32D08 at about UTM 5,361,055 m N, 706,270 m E, NAD 1983.

The Property is accessible by a high-quality, rural road network connecting to the main highway, Route 109, situated a few kilometres east, which links Rivière-Héva to Amos.

Route 109 connects at Rivière-Héva to Highway 117, a provincial highway that links Val-d'Or and Rouyn-Noranda, the two regional centers of the Abitibi-Témiscamingue region, to Montréal, the closest major city, almost 500 km to the southeast (Figure 4-1 and Figure 4-2).

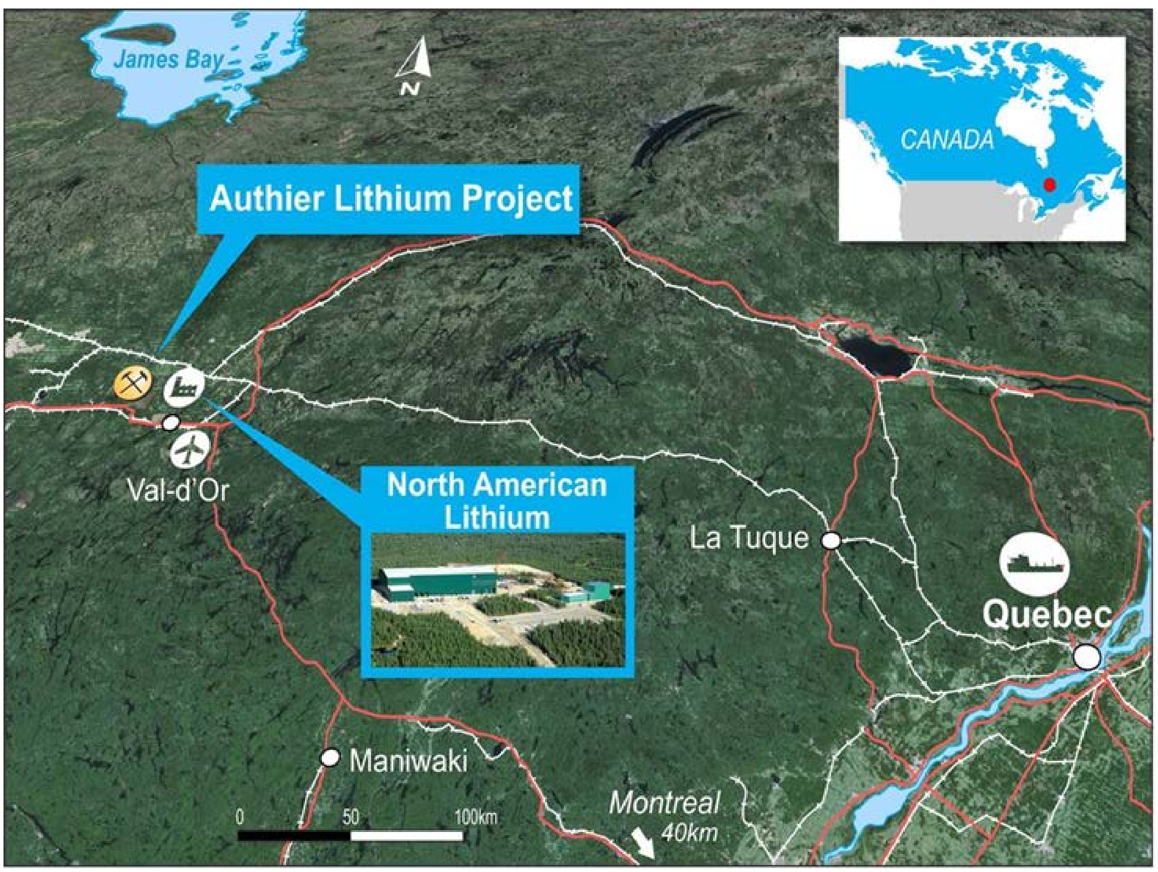


Figure 4-1: Location of the Property Relative to a Number of Nearby Regional Townships

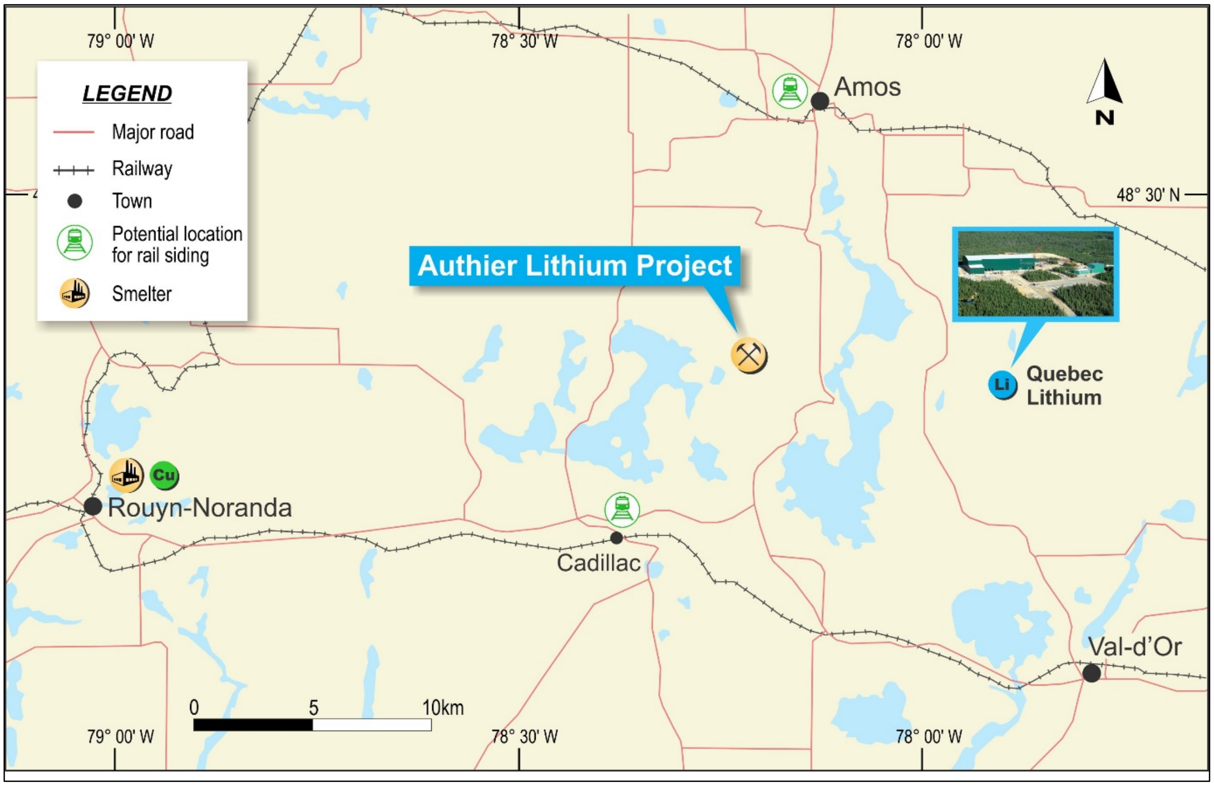


Figure 4-2: Authier Proximity to Nearby Mining Services Centres

4.2 Property Ownership and Agreements

As of September 9th, 2019, the property consists of one block totaling 24 mineral claims covering 884 ha. The claims are located in the La Motte Township except the westernmost claims, which are located in the Preissac Township. The claims are located on Crown Lands. The Property area extends 4.1 km in the east-west direction and 3.3 km in the north-south direction. All of the claims comprising the Property are map designated cells (CDC). Figure 4-3 shows the claim map of the Property and a detailed listing of the Authier property claims is included in Table 4-1.

Approximately 75% of the mineral resources are situated in CDC 2183455, 2194819 and 2116146, with the remainder in claims 2183454 and 2187652 (Figure 4-4).

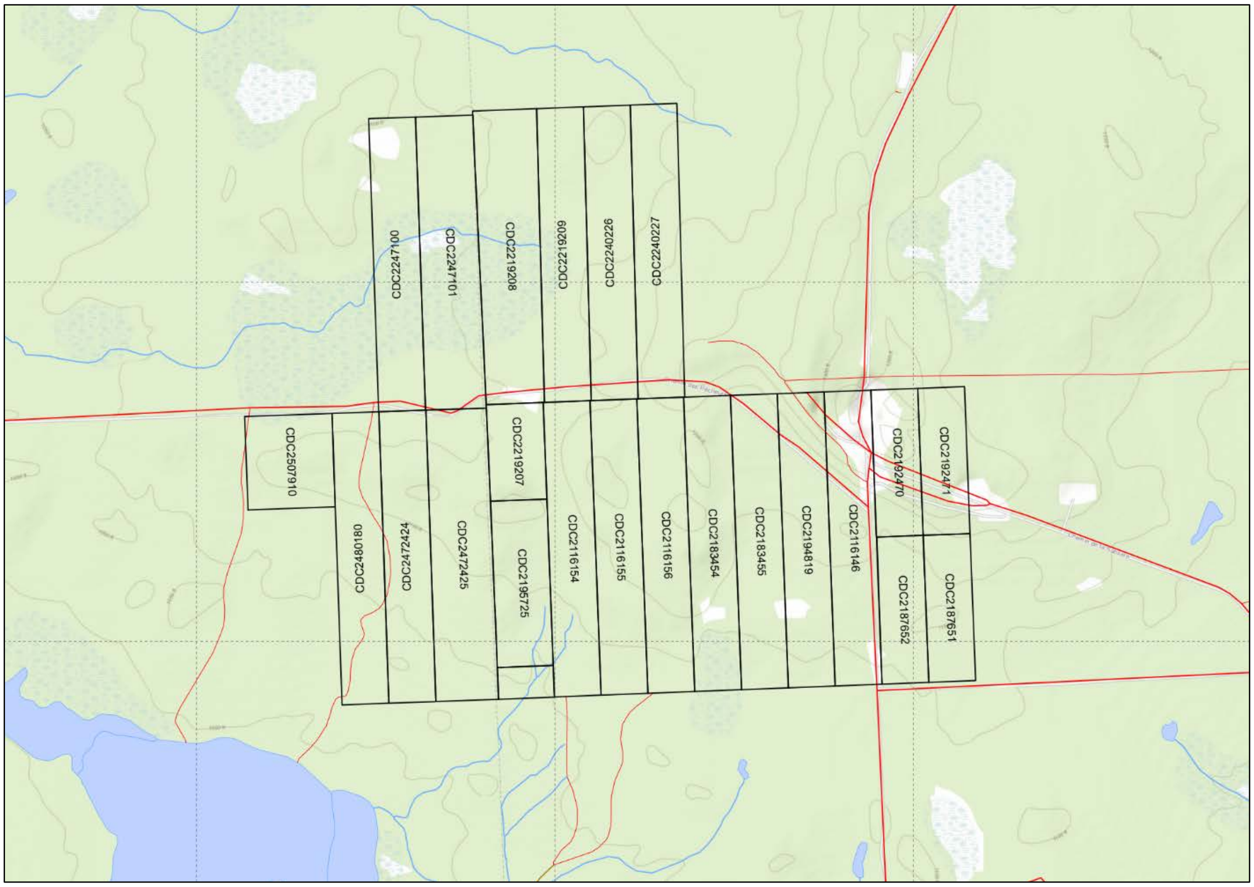


Figure 4-3: Property Mining Titles Location Map (as of September 9, 2019)

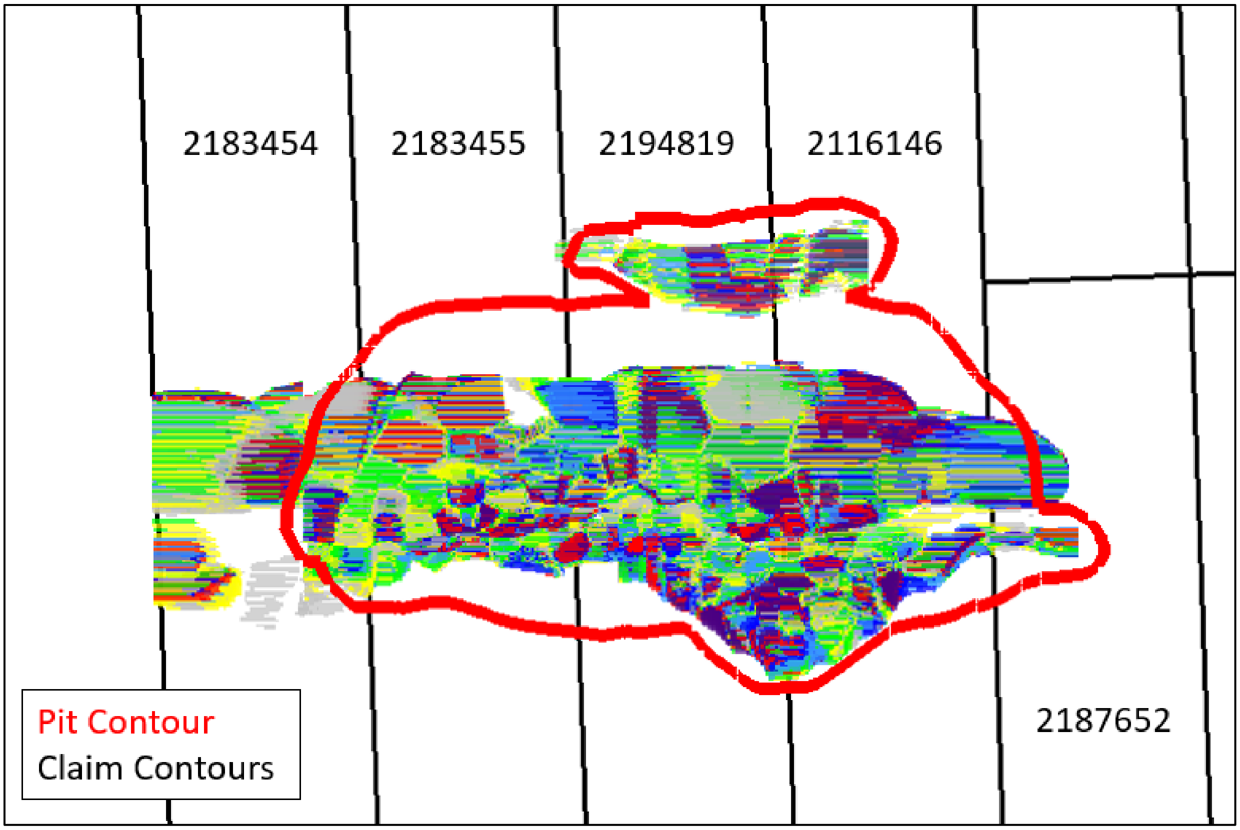


Figure 4-4: Pit Contours Showing Resource Relative to Claim Boundaries

Table 4-1: List of Authier Property Claims

Claim Number		Registered holder	Status	Registration date	Expiry date	Area (ha)	Required work ($)	Required Fees ($)
CDC	2195725	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2009/11/27	2021/11/26	29.03	$ 1,800.00	$ 65.25
CDC	2219206	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/04/22	2022/04/21	5.51	$ 750.00	$ 33.25
CDC	2219207	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/04/22	2022/04/21	17.06	$ 750.00	$ 33.25
CDC	2507910	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2017/12/15	2021/12/14	25.35	$ 1,200.00	$ 65.25
CDC	2116154	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2007/08/08	2021/08/07	42.88	$ 2,500.00	$ 65.25
CDC	2116155	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2007/08/08	2021/08/07	42.87	$ 2,500.00	$ 65.25
CDC	2192471	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2009/10/22	2021/10/21	21.39	$ 750.00	$ 33.25
CDC	2187651	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2009/09/02	2021/09/01	21.39	$ 750.00	$ 33.25
CDC	2219208	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/04/22	2022/04/21	55.96	$ 1,800.00	$ 65.25
CDC	2219209	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/04/22	2022/04/21	42.71	$ 1,800.00	$ 65.25
CDC	2240226	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/07/09	2022/07/08	42.71	$ 1,800.00	$ 65.25
CDC	2240227	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/07/09	2022/07/08	42.71	$ 1,800.00	$ 65.25
CDC	2247100	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/08/23	2022/08/22	42.75	$ 1,800.00	$ 65.25
CDC	2247101	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2010/08/23	2022/08/22	53.77	$ 1,800.00	$ 65.25
CDC	2480180	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2017/02/22	2021/02/21	42.51	$ 1,200.00	$ 65.25
CDC	2472424	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2017/01/11	2021/01/10	42.50	$ 1,200.00	$ 65.25
CDC	2472425	Sayona Québec Inc. (95797) 100 % (responsible)	Active	2017/01/11	2021/01/10	55.96	$ 1,200.00	$ 65.25
CDC	2116146	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2007/08/08	2021/08/07	43.24	$ 2,500.00	$ 65.25
CDC	2116156	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2007/08/08	2021/08/07	42.86	$ 2,500.00	$ 65.25
CDC	2183454	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2009/06/02	2021/06/01	42.85	$ 1,800.00	$ 65.25
CDC	2183455	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2009/06/02	2021/06/01	42.84	$ 1,800.00	$ 65.25
CDC	2187652	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2009/09/02	2021/09/01	21.29	$ 750.00	$ 33.25
CDC	2192470	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2009/10/22	2021/10/21	21.08	$ 750.00	$ 33.25
CDC	2194819	Sayona Québec Inc. (95797) 100 % (responsible)	Suspended*	2009-11-19	2021-11-18	42.82	$ 1,800.00	$ 65.25
Total						**884.04**	**$37,300.00**	**$1,374.00**

* Claim with status "suspended" are pending to be converted to mining lease.

4.3 Royalty Obligations

Table 4-2 summarizes the royalties payable from the Authier project. As at November 2017, only four tenements contain ore reserves that would create royalty obligations. These are CDC 2183454, 2183455, 2194819 and 2116146.

Table 4-2: Authier Project Summary Royalties

Tenement	Royalty	Royalty Details
2116146	2% NSR royalty payable to Jefmar Inc.	▪ The Royalty payable will be based upon the Gross Value less the deductions (costs for treatment and refining, sales, brokerage, certain taxes and transportation). ▪ Gross Value is attributable to a London Metal Exchange (LME) benchmark price (not necessarily the price actually received). ▪ The Royalty enables the owner to transact (for sales or smelting) with an affiliate. However actual prices and treatment charge deductions would be substituted with an arm's length value for the purposes of calculating the Royalty. ▪ 1% of the royalty can be purchased for CAD 1.0 M.
	1.5% NSR royalty payable to RNC	▪ The Royalty payable will be based upon the Gross Value less the deductions (costs for treatment and refining, sales, brokerage, certain taxes and transportation). ▪ No buy-back provision.
2183454 **2483455**	2% NSR royalty payable to 9187-1400 Québec Inc.	▪ Net Smelter Returns (NSR) means actual proceeds received by GER from any mint, smelter or purchaser for sale of ores, metals or concentrated products from the Property and sold after deducting: o Smelting, refining charges; o Penalties, marketing costs; o Transportation of ores, metals or concentrates from the Property to any mint, smelter or other purchaser; o Insurance on all ores, metals or concentrates; and o Any export or import taxes on ores, metals or concentrates in Canada or the receiving country. ▪ A 1% NSR can be repurchased on the claims CDC 2183454, 2183455 and 2194819 for CAD 1,000,000 leading respectively to a 1%, 1% and 0% on the CDC 2183454, 2183455 and 2194819. *Note: Prior to these clams being able to be mined, the final option consideration, due on the day on which a positive feasibility study is completed, will need to be paid to Québec Inc. (QI). This amount is equal to CAD 500,000 plus an amount equivalent [in cash] to 1,000,000 GER share at that date. This is in addition to the royalty. This remains outstanding and the substitution of GER shares for Sayona shares has not yet been raised with QI.*
2194819	1% NSR royalty payable to 9187-1400 Québec Inc.	

Tenement	Royalty	Royalty Details
2194819	1% GMR royalty payable to Globex Enterprises Inc.	▪ 1% Gross Metal Royalty (GMR) to Globex. ▪ GMR is a percentage of all metals or mineral compounds including but not limited to lithium, lithium compounds, gold, silver, tungsten etc. produced from the Property. ▪ No costs to be included in the Globex Royalty calculation. ▪ To be paid in cash or in kind at Globex's option.
2116154 2116155 **2116156 2187651** **2192470 2192471** **2219206 2219207** **2219208 2219209** **2247100 2247101**	2% GMR royalty payable to Globex Enterprises Inc.	▪ 2% Gross Metal Royalty (GMR) to Globex. ▪ GMR is a percentage of all metals including but not limited to lithium, gold, silver etc. produced from the Property. ▪ No costs to be included in the Globex Royalty calculation. ▪ To be paid in cash or in kind at Globex's option. ▪ Globex's Royalty and metals or minerals shall exclusively be the property of Globex immediately upon production.
2187652	1.5% NSR royalty payable to Canuck Exploration Inc.	▪ 1.5% of Net Smelter Return (NSR) payable to Canuck on any resource extracted for commercial purpose derived from the Claim with the exception of surface minerals substances. ▪ NSR is a percentage of the actual proceeds derived from any smelter or mill for the sale of all Payable Metals less Deductions. ▪ Quarterly payments, Canuck has right to audit calculations.

CDC 2187652 is subject to an option-to-purchase with Canuck Exploration Inc. as per an agreement dated March 16, 2017. The agreement allows for the exercise of the option at any time in the next five years by the paying of $25,000 on signing and $5,000, between Years 2 and 5, and $75,000 on exercise.

4.4 Work Permits

Sayona is conducting exploration activities under a valid forest intervention permit delivered by the provincial *Ministère des Forêts, de la Faune et des Parcs*. Permits are required for any exploration program that involves tree-cutting to create road access for drilling and/or stripping activities. As of the date of this report, the Company has all the valid work permits.

4.5 Environmental Liabilities

There are no environmental liabilities pertaining to the Property.

5. ACCESSIBILITY, PHYSIOGRAPHY, CLIMATE, INFRASTRUCTURE, SURFACE RIGHTS

5.1 Accessibility

The Property is accessible by well-maintained secondary gravel roads that connect to Route 109, situated a few kilometres to the east; Route 109 links Rivière-Héva to Amos and continues on to Matagami. Route 109 meets Route 117 at Rivière-Héva, which is the provincial highway linking Val-d’Or and Rouyn-Noranda.

5.2 Physiography

The Property is characterized by a relatively flat topography, with the exception of the north-eastern area, where gently rolling hills occur. Outcrops represent approximately 5% of the project area.

The overburden is relatively thin and is characterized by glacial tills and clays. The land is drained westward by small creeks and local grassy swamps occur in topographic lows.

The area is generally covered by forest populated by mixed balsam, spruce and aspen trees. The Property’s elevation above sea level ranges from 320 m at the lowest point to 380 m in the northeastern sector, with an average elevation of 350 m.

5.3 Climate

The region has a continental climate marked by cold dry winters and hot humid summers. The hottest month is July and the coldest month is January. Table 5-1 shows average temperatures per month.

Table 5-1: Average Temperatures by Month

Month	Temperature (ºC)
January	-17.0
February	-14.9
March	-8.1
April	1.2
May	9.3
June	14.6
July	17.1
August	15.5
September	10.7

Month	Temperature (ºC)
October	4.3
November	-3.6
December	-12.8
Annual	**1.4**

The extreme temperatures measured between 1981 and 2010 were 37.2ºC and -52.8ºC. Temperatures are above freezing approximately 210 days per year.

Average annual precipitation is 903 mm. This comprises average annual rainfall of 651 mm (maximal monthly rainfall is 106 mm in August) and average annual snowfall of 253 mm (maximal monthly snowfall is 53 mm in both December and January). The snow stays on the ground from mid-November until the ice leaves the lakes in early to mid-May. In 1988, the maximum snow cover recorded at Amos was 107 cm. Winters can be bitterly cold with temperatures averaging -15°C in January and February. The ground is frost-free from May to October. Summers are warm and relatively dry with a mean temperature of 22°C. Table 5-2 shows the total precipitation with the proportions of rain and snow.

Table 5-2: Average Monthly Precipitation with the Proportions of Rain and Snow

Month	Precipitation (mm)	Rain (mm)	Snow (mm)
January	55	2	53
February	44	2	42
March	49	12	37
April	55	38	17
May	75	74	1
June	95	95	0
July	105	105	0
August	106	106	0
September	105	105	0
October	84	75	9
November	71	31	41
December	59	6	53
Annual	**903**	**651**	**253**

On average, about 72% of the precipitation fell as rain and 28% as snow. August is the month with the most precipitation (106 mm) and February is the month with the least precipitation (44 mm).

Under normal circumstances, exploration and mining operations are conducted year-round without interruption due to weather conditions.

5.4 Local Infrastructure

The project is located in a well-developed mining region with readily-available support facilities and services. The towns of Val d'Or and Rouyn-Noranda, with populations of roughly 26,000 and 42,000, respectively, are well known for their mining history. The agricultural town of Amos, 20 km to the north, has a population of roughly 13,000.

An experienced mining workforce and other mining-related support services will come from these nearby cities.

Val-d'Or and Rouyn-Noranda have well-established hospitals, regional airports, schools, accommodation and telecommunications, which are also readily accessed from the project site.

Québec is a major producer of electricity as well as one the largest hydropower generators in the world. Green and renewable, it is well distributed through a reliable power network. Power will be accessed 5 km to the east of the project site via an electrical grid supplied by low-cost, hydroelectric power.

CN Rail has an extensive railway network throughout Canada. The closest rail connections to export shipping ports are located at Cadillac and Amos, 20 km to the southwest of the Property. The rail network connects to Montréal and Québec City, and to the west through the Ontario Northland Railway and North American rail system.

High and low pressure natural gas pipelines are located in close proximity to the Authier site, although no immediate reliance upon natural gas is anticipated.

5.5 Surface Rights

All of the claims composing the Property are situated on Crown Lands. There is no reason to believe that Sayona will not be able to secure the surface rights needed to construct the infrastructure related to a potential mining operation, including tailings storage and waste disposal areas as well as a processing plant and other infrastructures in the mine industrial area (MIA).

6. PROJECT HISTORY

A series of geological surveys and geoscientific studies were conducted by the Québec Government in the project area between 1955 and 1959, and again in 1972 (Leuner, 1959 and Brett et al., 1976).

In 1956, an electrical resistivity (potential) survey was completed by Kopp Scientific Inc. in the central portion of the Property. In 1958, East-Sullivan Mines Ltd. conducted magnetic and polarization surveys, followed by six drill holes located in the southwestern area of the Property. In 1963, Space Age Metals Corp., exploring for magmatic sulfides, completed magnetic and electromagnetic surveys in the area of the main pegmatite dyke. In 1965, Delta Mining Corp. Ltd. conducted additional magnetic surveys in the area.

From 1966 until 1969, exploration work was conducted under the direction and supervision of Mr. George H. Dumont, consulting engineer. The exploration programs, originally designed for magmatic sulfides, successfully outlined the main spodumene-bearing pegmatite on the Property. The work included magnetic and electromagnetic surveys as well as 23 diamond drill holes totalling 2,611.37 m.

In 1969, the Québec Department of Natural Resources carried out a series of flotation tests on two drill core composite samples. The bulk sample was composed of split core from drill hole AL-14 (50 m) and hole AL-19 (38.1 m). The results confirmed that the material was amenable to concentration by flotation, producing commercial grade spodumene concentrate, assaying between 5.13% and 5.81% Li_2O with recovery ranging from 66.86% and 82.25%.

In 1978, Société Minière Louvem Inc. completed two diamond drill holes, AL-24 and Al-25, on the western extension of the pegmatite dyke for a total of 190.5 m.

In 1980, *Société Québécoise d'Exploration Minière* (SOQUEM) completed six diamond drill holes (80-26 to 80-31), totalling 619.96 m in the central portion of the spodumene-bearing pegmatite. At the same time, 224 core samples from previous drilling, done between 1967 and 1980 on the pegmatite dyke, were re-assayed for Li_2O.

In 1989, the MRNF released the results of a regional metallogenic study on lithium prospects and other high technology commodities in the Abitibi-Témiscamingue region (Boily et al. 1989).

In 1991, Raymor Resources Ltd. (Raymor) conducted small-scale metallurgical testing of pegmatite rocks mineralized in spodumene sampled on the Property. An 18.3 kg sample grading 1.66% Li_2O was tested in 1991 by the Centre de Recherche Minérale (CRM). Results of the metallurgical testing returned a concentrate grade of 6.3% Li_2O with recovery rate of 73%.

In 1993, Raymor conducted additional drilling of 33 holes for a total of 3,699.66 m with the objective of verifying the presence and detailing the geometry of the spodumene-bearing pegmatite. Raymor also conducted geological mapping, trenching and started a 30 t bulk sampling of the pegmatite dyke, which was completed in 1996.

In 1997, Raymor contracted the CRM to conduct additional metallurgical testing. The tests were conducted on two different samples weighing roughly 18 t (with an average grade of 1.32% Li_2O), and 12 t, (with an average grade of 1.10% Li_2O). Testwork results for the first sample returned a concentrate grade of 5.61% Li_2O with a recovery rate of 61% following magnetic separation. The second sample returned a final concentrate grade of 5.16% with a recovery rate of 58%.

Historical mineral resource estimates from 1994 were then revised in 1999 by Karpoff for SOQUEM and Raymor. The final historical mineral resources totalled 2,424,400 t at an average grade of 1.05% Li_2O, using a cut-off grade of 0.5% Li_2O. To these mineral resources, Karpoff defined an additional 1,580,000 t of historical resources in the *possible* category, without specifying the Li_2O grade.

Raymor concluded an agreement with SOQUEM in 1999. The group completed a pre-feasibility study on the project, including additional metallurgical testing. The metallurgical test results underlined the difficulty of generating a high quality spodumene concentrate. The economic analysis returned a negative investment return rate (IRR), making the project uneconomic at that time.

Glen Eagle Resources acquired the project in 2010, and completed a number of mapping, sampling, drilling, metallurgical, and resource definition programs as well as a Preliminary Economic Study in 2012.

In November 2010, a ground magnetic survey was performed on the Authier property. The survey was executed by Services Forestiers & d'Exploration GFE and the data was processed by MB Geosolutions at the request of Glen Eagle. The survey totalled 53.5 line-km and was done through the forest without a cut line grid. The lines were read with a GSM-19 Overhauser magnetometer, built by the company GEM of Toronto, which was used in *walking* mode with the locations of the readings determined by an integrated GPS.

The magnetic measurements were taken continuously along 23 traverse lines for a total of 66,027 readings at every 1.25 m. Magnetic diurnal was monitored with a base station and the magnetic readings were corrected accordingly. Figure 6-1 presents the results of this survey.

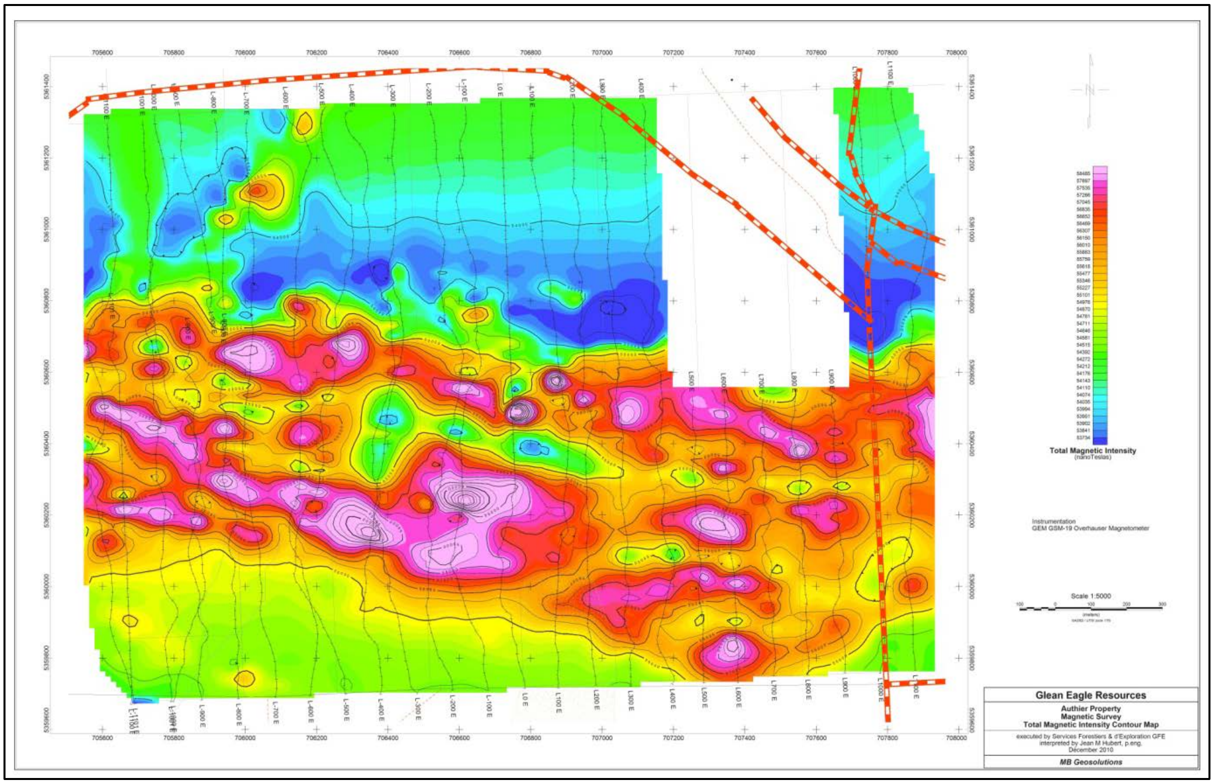


Figure 6-1: 2010 Authier Property Magnetic Survey

In August 2011, a geochemical survey program was completed in an effort to discover new spodumene-bearing pegmatites. Eighty-six (86) samples were collected, mainly in the northwest sector of the Property. Four samples were collected on the main pegmatite and were analyzed for the major elements. The geochemical signature of the collected samples was compared to the signature of the main pegmatite and only a few samples were determined to have a similar signature. Three drill holes were drilled in the area of these samples; muscovite-bearing pegmatites were discovered with little or no spodumene.

From 2010 to 2012, Glen Eagle completed 8,990 m in 69 diamond, NQ diameter drill holes on the Authier property; 7,959 m were drilled on the Authier deposit; 609 m (5 DDH) were drilled on the northwest and 422 m on the south-southwest sectors of the Property.

From these drill holes, 1,474 samples for analysis were collected representing approximately 18% of the drill core material. The drill holes are generally spaced 25 m to 50 m apart, with azimuth generally south dipping (180°) and dip ranging from 45° to 70°. The mineralized drill intersection ranged from near true thickness to 85% true thickness.

The spodumene-bearing pegmatite is principally defined by one single continuous intrusion or dyke, which contains local rafts or xenoliths of the amphibolitic host rock that can be a few metres thick and up to 200 m in length.

In 2012, Glen Eagle conducted further testing on a 270 kg composite sample and achieved very attractive results, including an 88% metallurgical recovery to a 6.09% Li_2O concentrate. The results were achieved using batch flotation, after passing the concentrate through WHIMS and two-stage cleaning, without prior mica removal by flotation. Bumigème Inc. used the results of this program to design a conventional process flowsheet incorporating crushing, grinding and flotation for the Authier NI 43-101 Preliminary Economic Assessment (2013). The flowsheet contemplated the processing of 2,200 tpd of ore at 85% metallurgical recovery, producing a 6% Li_2O spodumene concentrate. This assessment suggested the technical and commercial viability of developing the deposit and reported a Measured and Indicated mineral resource of 7.67 Mt at 0.96% Li_2O (Table 6-1).

Table 6-1: Glen Eagle 2013 Mineral Resource Estimate (NI 43-101 Compliant at 0.5% Li_2O Cut-off)

Category	Tonnes	Grade (% Li_2O)	Contained Li_2O (t)
Measured	2,244,000	0.95	21,318
Indicated	5,431,000	0.97	52,681
Total	7,675,000	0.96	73,999
Inferred	1,552,000	0.96	14,899

6.1 Pre-acquisition Due Diligence Evaluation by Sayona Québec

Sayona Québec signed a term sheet with Glen Eagle in May 2016, which was subject to completing a detailed technical due diligence and finalizing acquisition financing. The due diligence program covered all the legal and technical aspects of the proposed acquisition.

Sayona has commissioned independent assessments, including:

1. Resources

An Independent Competent Person visited the Authier property, completed a geological and historical drilling data review, and released an in-house, JORC 2012 compliant mineral resource estimate, tabulated below at a 0.5% Li_2O cut-off grade (Table 6-2). The effective date of this mineral resource estimate is July 7, 2016.

Table 6-2: Due Diligence JORC (2012) Mineral Resources Estimate (0.5% Li_2O Cut-off Grade)

Category	Million Tonnes (Mt)	Grade (% Li_2O)	Contained Li_2O (t)
Measured	2.08	0.95	19,730
Indicated	5.16	0.97	50,092
Inferred	1.88	0.93	17,480
Total	**9.12**	**0.96**	**87,302**

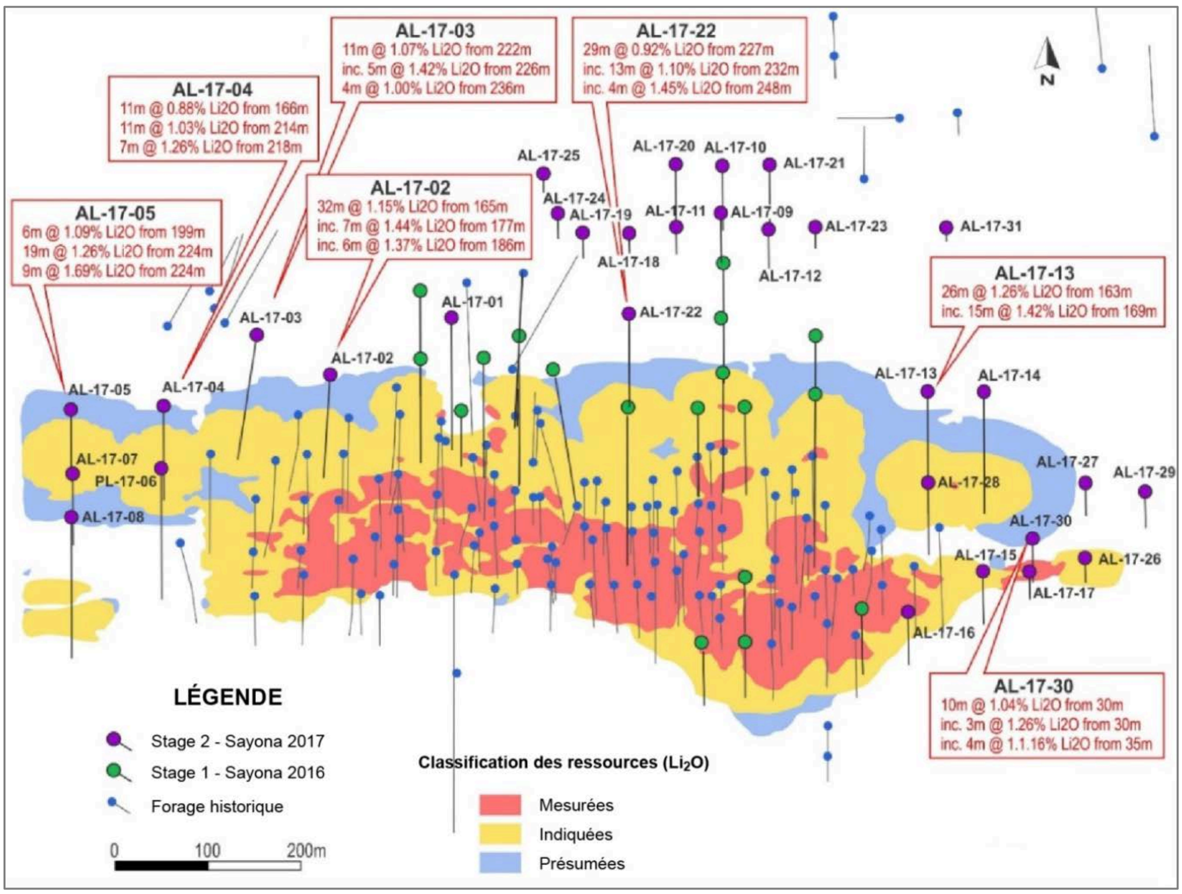


Figure 6-2: Drill Hole Location Plan and Significant Interception of the Sayona Québec 2017 Drilling Program

Sayona's in-house resource estimate correlated strongly with the Glen Eagle resource estimate from the 2012 technical report. Sayona's in-house resource was estimated and reported in accordance with the guidelines of the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code 2012).

Sayona notes that the Canadian NI 43-101 Standards of Disclosure system is broadly comparable to the JORC Code of reporting, and whilst the reporting methodologies are different, the mineral resource estimates are broadly similar.

2. Technical and economic

SGS Canada Inc. and Bumigème Inc. completed an economic assessment, including updating the entire mine, processing, administration, operating and capital cost estimates previously disclosed in the Glen Eagle Resources 2013 Technical Report. As well, SGS completed Whittle pit optimizations using the new operating and capital cost estimates, and revised spodumene concentrations pricing and exchange rate assumptions. The review concluded that Authier had strong technical and economic merit and recommended that Sayona complete the acquisition.

3. Legal

A legal review was completed to confirm that the tenure was in good standing and that there were no major environmental or cultural issues that would affect a potential project development.

In August 2016, Sayona completed the acquisition of the Authier property for CAD 4.0 M.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Authier property is located in the southeast part of the Superior Province of the Canadian Shield craton, more specifically in the Southern Volcanic Zone of the Abitibi Greenstone Belt. The spodumene-bearing pegmatites observed on the Property are genetically related to the Preissac-La Corne batholith (Figure 7-1) located 40 km northeast of the city of Val-d'Or (Corfu, 1993; Boily, 1995; Mulja et al., 1995a).

The Preissac-La Corne batholith is an Archean-age syn- to post-tectonic intrusive complex that intruded along the La Pause anticline into the volcano-sedimentary units of the Malartic Composite Group. The rocks of the Malartic Group are metamorphosed to the greenschist to lower amphibolite metamorphic grade and are bounded to the north by the Manneville fault and by the Cadillac-Larder Lake fault to the south. The units comprising the Malartic Group are mafic to ultramafic metavolcanic rocks (serpentinized peridotites, amphibolitic mafic flows) and metasedimentary units (biotite schists derived from greywackes). The Preissac-La Corne batholith comprises early-stage metaluminous intrusive suites, dioritic to granodioritic in composition, and four late-stage peraluminous monzogranitic plutons: Preissac, La Corne, and La Motte and Moly Hill plutons. Late Proterozoic-age diabase dykes crosscutting all the lithologies can also be observed in the region (Boily, 1995; Mulja et al., 1995; Desrocher et Hubert, 1996).

The pegmatite dykes and other aplitic dykes and veins observed in the region are genetically derived from the late peraluminous plutons. More than one thousand intrusions of mineralized, but mostly barren pegmatite dykes have been mapped in the vicinity of the Preissac-La Corne batholith. These intrusions cross-cut all of the units of the Malartic Group and intrusive lithologies of the batholith, with the exception of the late Proterozoic diabase dykes. The pegmatites and the aplitic intrusions occur in two distinct morphologies: tabular, generally strongly dipping dykes with sharp contacts, and irregularly shaped dykes, often comprised of mixed pegmatitic and aplitic lithologies in contact with the country rocks. The dykes can be up to hundreds of metres in length with a thickness varying from a few centimetres to tens of metres, with the majority having less than 1 m in thickness.

The pegmatites can be classified by their spatial distribution within and around the lithologies of the Preissac-La Corne batholith. The pegmatites occurring within or in the vicinity of the La Motte and La Corne plutons are generally mineralized in beryl and columbite-tantalite as opposed to the pegmatites observed in association with the Preissac pluton, which are mostly un-mineralized. The spodumene-bearing pegmatites almost exclusively cross-cut lithologies located outside the late stage plutons of the Preissac-La Corne Batholith and can be uniform or present internal zoning enriched in spodumene. The hydrothermal veins mineralized in molybdenite occur inside, near the edges, of the intrusives related to the Preissac and Moly Hill plutons.

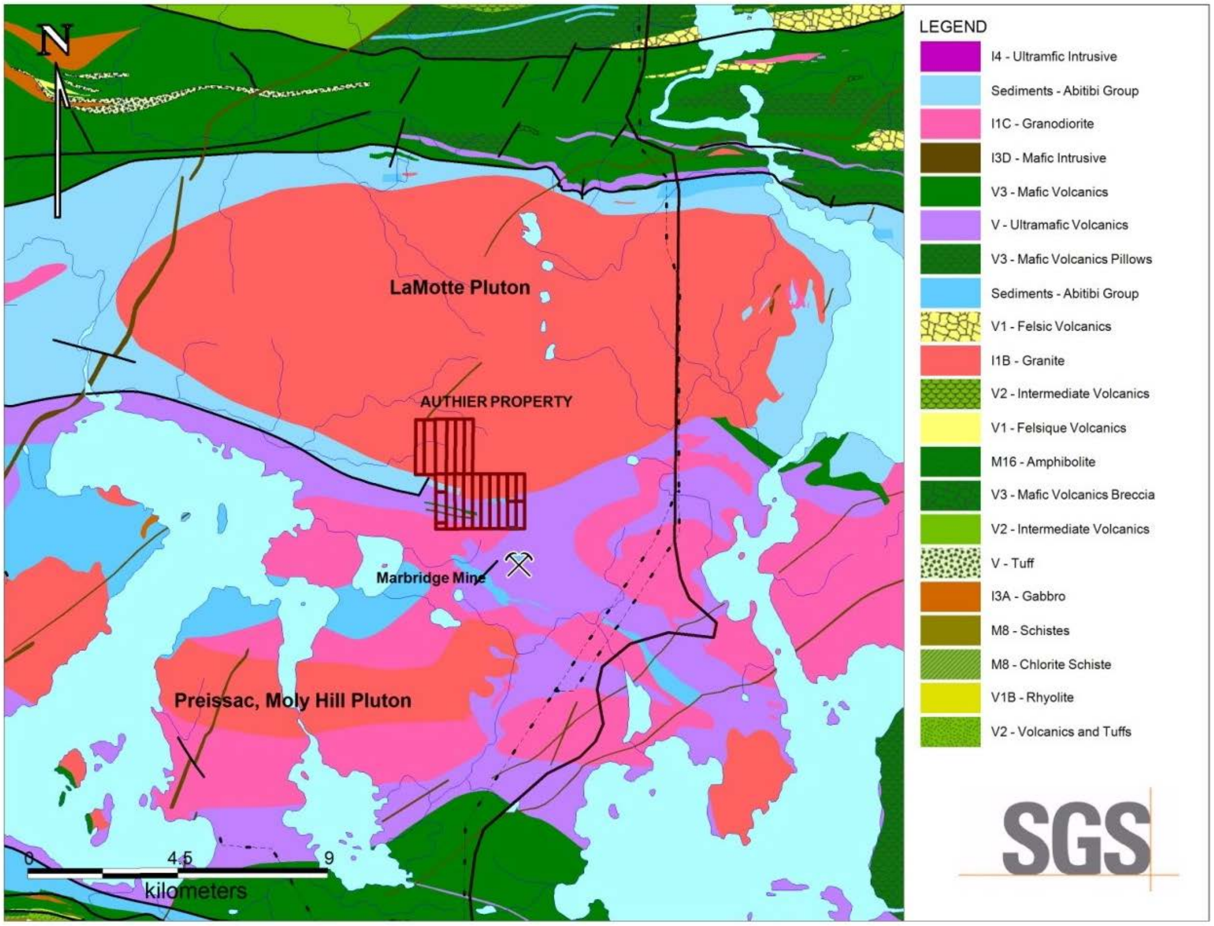


Figure 7-1: Regional Geology Map (note: 2018 claims)

7.2 Property Geology

The Property geology comprises intrusive units of the La Motte pluton to the north and Preissac pluton to the south, with volcano-sedimentary lithologies of the Malartic Group in the centre (Figure 7-2). The volcano-sedimentary stratigraphy is generally oriented east-west and ranges between 500 m and 850 m in thickness (north-south). The volcanic units comprise principally ultramafic (peridotitic) metavolcanic flows with less abundant basaltic metavolcanics. Several highly metamorphosed metasedimentary units described as hornblende-chlorite-biotite schists occur on the south-central portion of the Property, generally in contact with the La Motte pluton to the north (Karpoff, 1994).

The northern border of the Preissac pluton, composed of granodiorite and monzodiorite, runs east-west along the southern edge on the Property. To the north, muscovite monzogranitic units of the La Motte pluton cover the Property. Numerous small pegmatites, generally composed of quartz monzonite, are intruding the volcanic stratigraphy, including the larger Authier spodumene-bearing pegmatite, which is the focus of study.

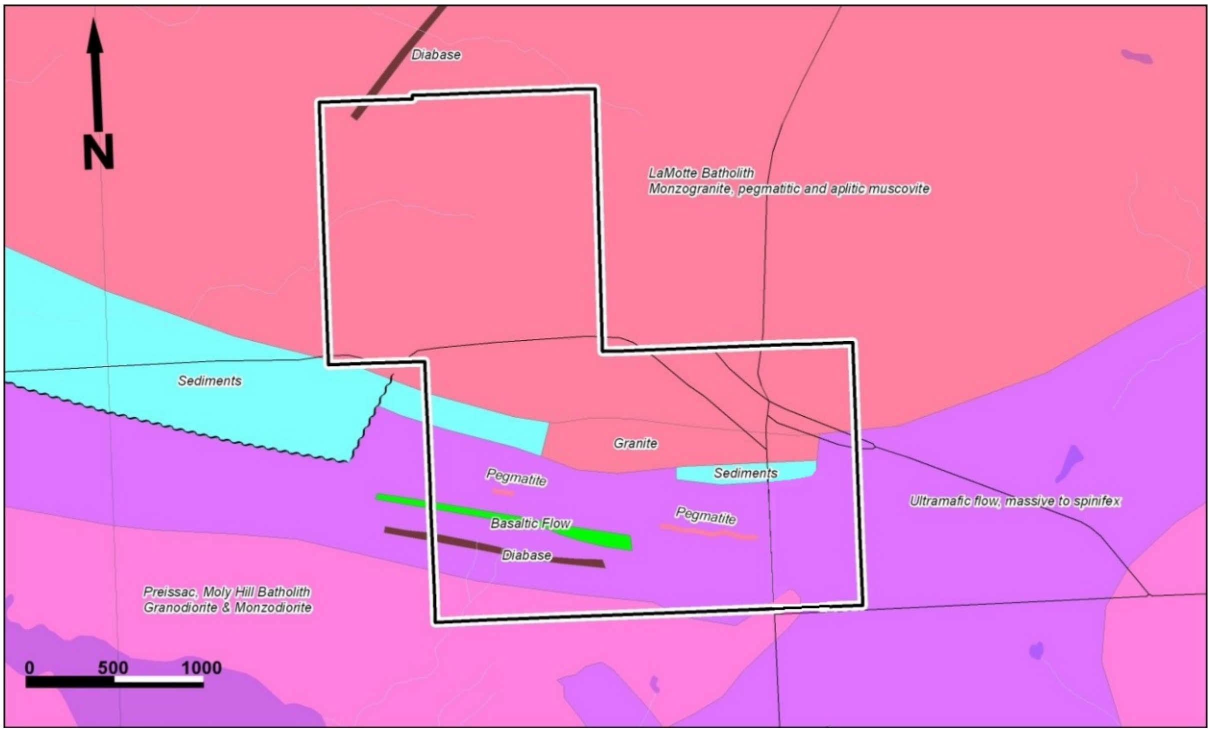


Figure 7-2: Local Geological Map (note: 2018 claim boundaries)

7.3 Mineralization

The mineralization observed at the Authier project in the spodumene-bearing pegmatites is principally lithium with trace amounts of beryllium, molybdenum, tantalum, niobium, cesium and rubidium. Figure 7-2 details the typical geochemical composition of the Authier pegmatite averaged from the project database.

Detailed logging of drill core suggests that the main pegmatite at Authier is composed of several internal phases related to intrusive placement and progressive cooling. The outside border of the pegmatite in contact with the host rocks has been identified as a transition zone or border zone. This transition zone is often significantly less mineralized in spodumene and is characterized by a centimetre-scale fine to medium-grained chill margin, followed by a medium to coarse-grained decimetre to metre-scale zone. The transition zone often includes fragments of the host rock and can also be intermixed with the material from the core zone.

The main intrusive phase observed in the pegmatite is described as a core pegmatitic zone characterized by large centimetre-scale spodumene and white feldspar minerals. The core pegmatitic zone shows internally different pegmatitic phases characterized by different spodumene crystal lengths, ranging from coarse grained (earlier) to fine grained (later). The contacts between different spodumene-bearing pegmatite phases are transitional and well defined at core logging scale. Higher lithium grades are correlated with higher concentrations of larger spodumene crystals. Late mineral to post-mineral aplite phases cut earlier spodumene-bearing mineralization, causing local diminishing of lithium grade. The core zone hosts the majority of the spodumene mineralization at Authier. Figure 7-3 is a photograph that illustrates the transition and core zones from drill hole AL-10-03.

The spodumene-bearing pegmatite is principally defined by one single continuous intrusion or dyke that contains local rafts or xenoliths of the amphibolitic host rock, which are a few metres thick and up to 200 m in length at shallow levels within the western zone. The main pegmatite outcrops in a small, 50 m by 20 m area at the central-eastern sector that orients east-west and is mostly covered by up to 10 m of overburden. Based on the information gathered from the drilling, the pegmatite intrusion is more than 1,100 m in length and can be up to 60 m thick. The intrusion is generally oriented east-west, dips to the north at angles ranging between 35° and 50° and reaches depths of up to 270 m below surface in drilling to date.

A second spodumene-bearing pegmatite, not visible from the surface, was intersected by diamond hole AL-16-10 at shallow levels, between 15 m to 22 m downhole depth, approximately 400 m north of the main pegmatite. Follow-up drilling in early 2017 and 2018 outlined this new body, the Authier North pegmatite, which has a strike extension of 500 m east-west, 7 m average width, gently dipping 15 degrees to the north. The Authier North pegmatite appears at shallow levels, 15 m to 25 m vertical depth, and is open in all directions. Figure 7-4 is a photograph showing spodumene mineralization from the new shallow pegmatite intersected by drill hole AL-16-10.

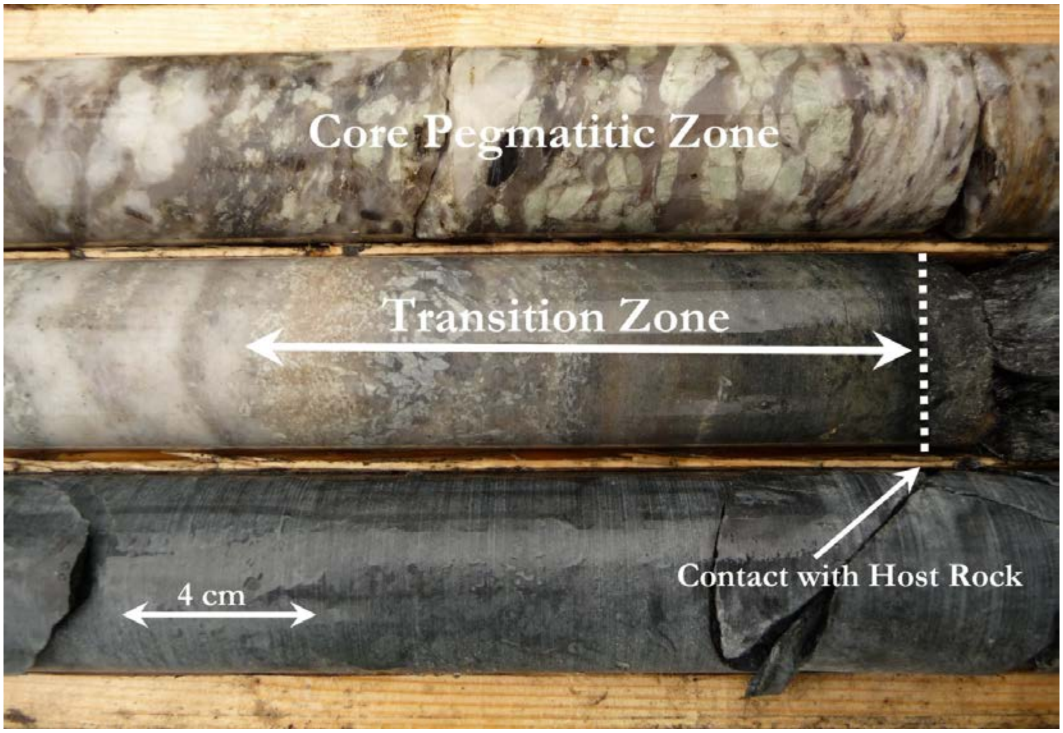


Figure 7-3: Drill Core from Hole AL-10-03 Showing Core and Transition Zones

Figure 7-4: Drill Core from Hole AL-16-10, Showing Spodumene Mineralization in the New Authier North Pegmatite

8. DEPOSIT TYPES

8.1 General

The deposit type for the lithium mineralization occurring on the Authier property is a granitic pegmatites type, more specifically the rare-element pegmatites sub-type, due to the presence of spodumene.

Rare-element pegmatites typically occur in metamorphic terrains and are commonly peripheral to larger granitic plutons, which in many cases represent the parental granite from which the pegmatite was derived.

The late Archean pegmatites of the Superior Province are typically located along deep fault systems that, in many areas, coincide with major metamorphic and tectonic boundaries. Most pegmatites range in size from a few metres to hundreds of metres long and from centimetric-scale to several hundred metres wide, and even more for a few known cases.

Rare-element pegmatites can have complex internal structures where the internal units in complex pegmatites consist of a sequence of zones, mainly concentric, which conform roughly to the shape of the pegmatite, but differ in mineral assemblages and textures. From the margin inward, these zones consist of a border zone, a wall zone, intermediate zones and a core zone.

The border zone is generally thin and typically aplitic or fine grained in texture. The wall zone, composed mainly of quartz-feldspar-muscovite, is wider and coarser grained than the border zone and marks the beginning of coarse crystallization characteristic of pegmatites. Intermediate zones, where present, are more complex mineralogically and contain a variety of economically important minerals such as sheet mica, beryl and spodumene.

In the intermediate zones of some pegmatites, individual crystals size can reach metres to tens of metres. The core zone consists mainly of quartz, either as solid masses or as euhedral crystals.

Rare-element pegmatites, typically associated with granitic intrusions, are distributed in zonal patterns around such intrusions. In general, the pegmatites most enriched in rare metals and volatile components are located farthest from intrusions (Figure 8-1).

Rare-element pegmatites are generally considered to form by primary crystallization from volatile-rich siliceous melt related to highly differentiated granitic magmas.

The lithology of the source rocks for these melts is a major control on the ultimate composition of subsequently formed rare-element pegmatites (Cerny, 1993; Sinclair, 1996).

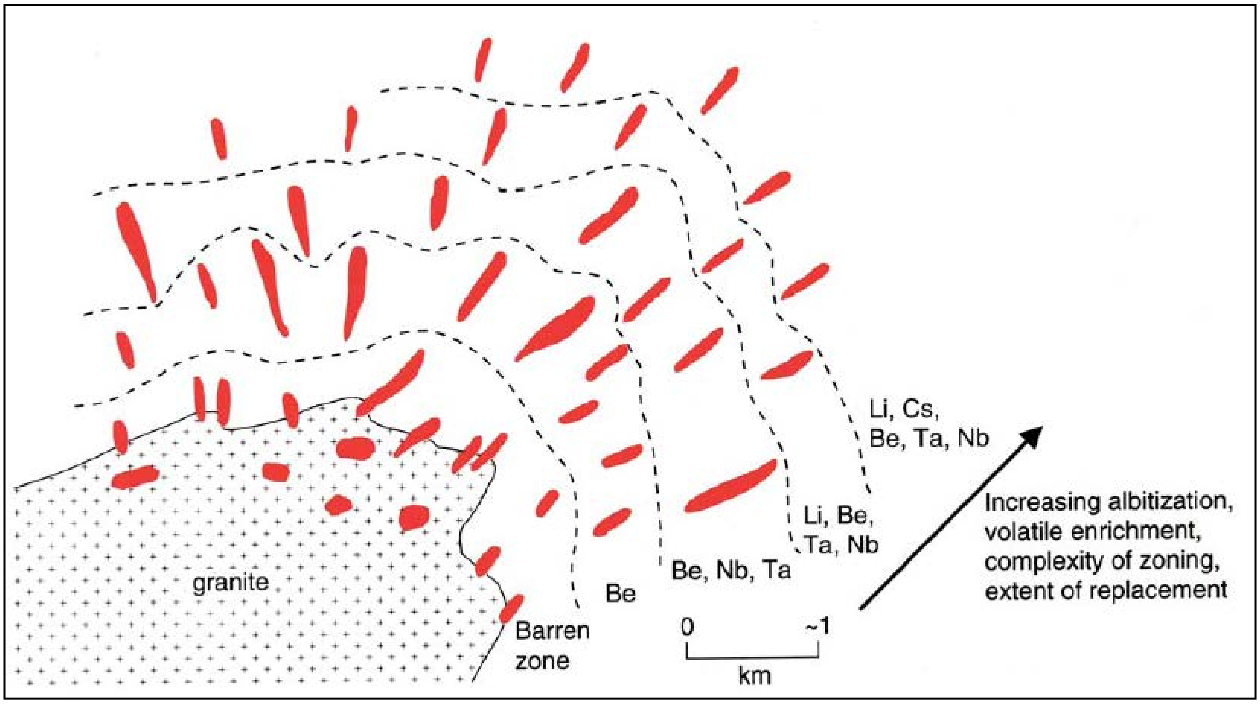


Figure 8-1: Schematic Representation of Regional Zonation of Pegmatites Source (Image from Sinclair 1996 (modified from Trueman and Cerny 1982))

9. DRILLING

9.1 Historical Drilling

Several exploration companies have previously conducted drilling programs on the Authier property and mainly the pegmatite as described in Chapter 6.

A total of 19,736 m of historical drilling were done on the Property. Figure 9-1 shows a plan view of the historical drill holes. All of the historical drilling that predates Sayona was diamond core NQ diameter.

Table 9-1: Summary of Drilling Done on the Property Prior to the Sayona Acquisition in 2016

Period	Drill Holes Series	Number of DDH	Metres Drilled
Historical	GM-XX	5	1,176
	LG-XX	12	2,437
	AL-XX	31	3,433
	R-93-XX	33	3,700
Glen Eagle Resources	AL-10-XX	18	1,905
	AL-11-XX	27	4,051
	AL-12-XX	24	3,034
Total		**150**	**19,736**

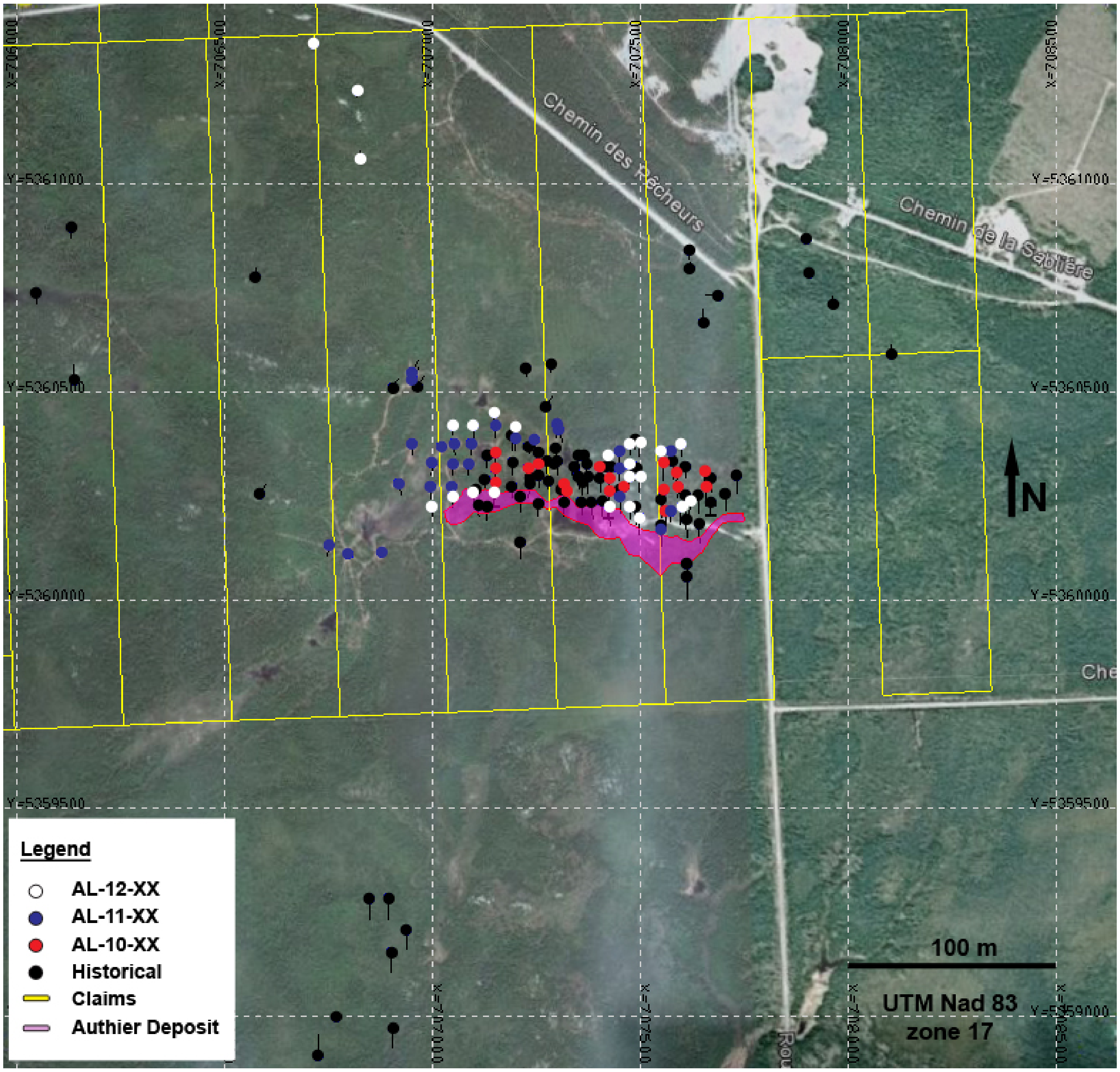


Figure 9-1: Authier Lithium Property Diamond Drill Hole Location Map Prior to 2016

9.2 Sayona Drilling Summary

Sayona Québec has completed three drilling programs at the Authier property, including:

- Phase 1 program in October/November 2016 of 18 holes, totalling 3,967 m. Following the drilling program, Sayona completed an upgrade of the resource and completed a Prefeasibility Study, dated February 2017;
- Phase 2 diamond drilling program in May 2017 of 31 holes totalling 4,117 m; and
- Phase 3 diamond drilling program in November/December 2017, which comprised 7 diamond holes (PQ and HQ) for 769.5 m and the collection of five tonnes of core for pilot metallurgical testing; January / March 2018, which comprised 19 holes NQ diameter totalling 2,170.45 m; April 2018, involving condemnation drilling, 6 holes NQ diameter for 342.65 m.

The total drilling performed by Sayona since acquiring the Authier property from Glen Eagle is 81 holes for 11,367.5 m. From this database, 199 drill holes were used for the solid modelling and updated resource estimate (MRE). All holes completed by Sayona in both programs have been diamond core drill holes (DDH) using HQ or NQ core diameter size with a standard tube and bit. Core diameter for metallurgical drilling was done using PQ core for 680 m and HQ core for 89.5 m of HQ core. Condemnation drilling was done using NQ core diameter.

Core was oriented using a Reflex ACT III tool for Phase 1 and Phase 2, whereas Phase 3 diamond core was not oriented.

The drilling programs were planned and managed by Sayona's Competent Person, assisted by one of Sayona's project geologists. In addition, Sayona contracted Services Forestiers et d'Exploration GFE (Services GFE) for the permitting and logistic support of the drilling program. Services GFE provided the office, core logging and storage facilities to Sayona, which are located less than 4 km southeast from the main pegmatite dyke, near the town of La Motte.

All drill core handling was done onsite with logging and sampling processes conducted by employees and contractors of Sayona.

Drill core of HQ size was placed in wooden core boxes and collected twice a day at the drill site and then transported to the core logging facilities. The drill core was first aligned and measured by a technician or the geologist for core recovery. After a summary review of the core, it was oriented and geologically and geotechnically logged, including rock type, spodumene abundance, mica abundance, RQD, orientation data (alpha and beta angles) for structures (faults, fractures, etc.). Point load tests (1 each, 10 m average) have also been undertaken. The logging of the geological features was predominately qualitative. Parameters such as spodumene abundance are visual estimates by the logging geologist.

The observations of lithology, structure, mineralization, sample number and location were noted by the geologists and geotechnicians in hard copy and an excel spreadsheet, and then recorded in a Microsoft Access digital database. Copies of the database are stored on external hard drive for security. Sampling intervals were defined by a geologist. Before sampling, core was photographed after metre marks using a digital camera and sample intervals have been clearly marked on the core. The core was photographed dry and wet. The core boxes were identified with the box number, hole ID, from and to using aluminum tags. The entire target mineralization type core (i.e., spodumene pegmatite) and surrounding barren host rock has been logged, sampled, and assayed.

The footwall and hanging wall barren host rock has been summary logged. Main rock units, i.e. pegmatite and host rock, are competent with average core recovery around 99%. High competence of the core tends to preclude any potential issue of sampling bias and sampling is considered representative.

Sampling intervals were determined by the geologist, marked and tagged based on observations of the lithology and mineralization. The typical sample length is 1.0 m starting 2 m to 3 m above and below of the contact of the pegmatite with the barren host rock. In general, at least two host rock samples were collected each side from the contact with the pegmatite. High to low-grade lithium-bearing mineralization, i.e. spodumene, is visible during geological logging and sampling.

The drill core samples were split into two halves with one half-placed in a new plastic bag along with the sample tag; the other half was placed in the core box with the second sample tag for reference. The third sample tag was archived onsite. The samples were then catalogued and placed in rice bags or sealed pails for shipping. The sample shipment forms were prepared onsite with one copy inserted into one of the shipment bags and one copy kept for reference.

Full core was sent to the laboratory for PQ and NQ diameter samples taken for the metallurgical drilling program.

As with the 2017 and 2016 samples, the 2018 samples were transported on a regular basis by a courier truck contracted by Sayona, directly to the SGS facilities in Lakefield, Ontario. Sample preparation and assaying techniques are within industry standard and appropriate for this type of mineralization.

All core drilling before 2016 was NQ core diameter size only, standard tubes and bit, and not oriented.

9.3 Sayona Québec Drilling 2016

Sayona Québec completed a Phase 1 diamond drilling program at the Authier property, including 18 holes for 3,967 m (Figure 9-2), which had the following objectives:

- Converting the inferred mineral resources to be measured and indicated through further drilling;
- Exploring for extensions to the existing mineral resources and other potential mineralization within the tenement package;
- Collecting geotechnical data for incorporation in the Authier prefeasibility study; and
- Collecting additional drill core for any additional metallurgical testing that may be required to complete a definitive feasibility study.

Holes were typically drilled perpendicular to the strike of the mineralized pegmatite to provide high confidence in the grade, strike and vertical extensions of the mineralization.

The final diamond drill holes (Table 9-2) have all intersected high-grade spodumene mineralization, including:

Table 9-2: Phase 1 Sayona Drill Hole Collar Location and Intercept Information (downhole intersections in metres)

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-16-001	707525	5360175	330	180	-45	87	12	74	62	1.35
including							27	43	16	1.65
AL-16-002	707525	5360245	330	180	-45	111	50	99	49	1.18
including							81	98	17	1.49
AL-16-003	707600	5360500	331	180	-55	267	170	197	27	1.46
including							181	192	11	1.66
							213	223	10	1.24
including							218	221	3	1.63
AL-16-004	707525	5360430	331	180	-55	246	156	206	50	1.13
including							157	168	11	1.4
							200	205	5	1.89
AL-16-005	707500	5360520	332	180	-55	294	197	202	5	1.44
							218	243	25	1.08
including							218	232	14	1.18
AL-16-006	707650	5360210	330	180	-45	105	16	60	44	1.02
including							16	35	19	1.45

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-16-007	707479	5360174	330	180	-45	90	3.81	44	40.19	1.27
including							13	33	20	1.47
AL-16-008	707475	5360425	330	180	-60	234	162	198	36	0.93
including							163	173	10	1.32
AL-16-009	707245	5360478	330	180	-60	249	192	230	38	1.1
including							192	215	23	1.35
AL-16-010	707500	5360580	330	180	-55	330	15	22	7	1.36
including							17	19	2	2.24
							236	241	5	1.36
							258	266	8	0.85
including							264	266	2	1.42
AL-16-011	707220	5360420	330	180	-65	204	135	181	46	1.26
including							137	161	24	1.62
AL-16-012	707500	5360460	331	180	-55	240	161	208	47	1.05
including							167	194	27	1.31
AL-16-013	707175	5360478	331	180	-60	234	184	208	24	1.25
							216	224	8	0.91
AL-16-014	707600	5360440	331	180	-55	241	148	193	45	1.08
including							149	157	8	1.36
							171	189	18	1.34
							203	207	4	1.65
AL-16-015	707175	5360550	330	180	-60	279	242	262	20	1.32
including							248	259	11	1.61
AL-16-016	707400	5360425	331.47	180	-60	252	158	186	28	1.2
including							162	180	18	1.39
AL-16-017	707280	5360500	330	180	-60	240	190	235	45	1.28
including							190	213	23	1.77
AL-16-018	707318	5360465	330	170	-55	264	197	201	4	0.99
							206	213	7	0.95
							218	228	10	1.2
including							219	225	6	1.48
Note: Downhole widths are not true widths										

The highlights of the 2016 drilling program include:

- Fourteen (14) new drill holes successfully tested the deep extensions of mineralization on the main Authier pegmatite;
- Four (4) holes successfully tested the geometry of the Authier pegmatite at shallow levels in the eastern and central sectors to upgrade the resource categories from indicated to measured;
- Holes AL-16-01, 02, 06 and 07 successfully tested the geometry of the Authier pegmatite at shallow levels in the eastern and central sectors to upgrade the resource categories from indicated to measured;
- Hole AL-16-16 intersected a thick zone of spodumene mineralization in the gap zone, between eastern and western zones of the main pegmatite;
- Holes AL-16-03, 04, 05, 08, 10, 12 and 14 extended the lithium mineralization in the eastern sector of the main Authier pegmatite, beyond 200 m of vertical depth;
- In addition, hole AL-16-10 intercepted a new pegmatite at shallow levels between 15 m to 22 m downhole depth, which is not visible from the surface and located 400 m north of the main Authier pegmatite; and
- Holes AL-16-09, 11, 13, 15, 17 and 18 extended the lithium mineralization in the western sector of the main Authier pegmatite, beyond 200 m of vertical depth.

The mineralization remains open in all directions.

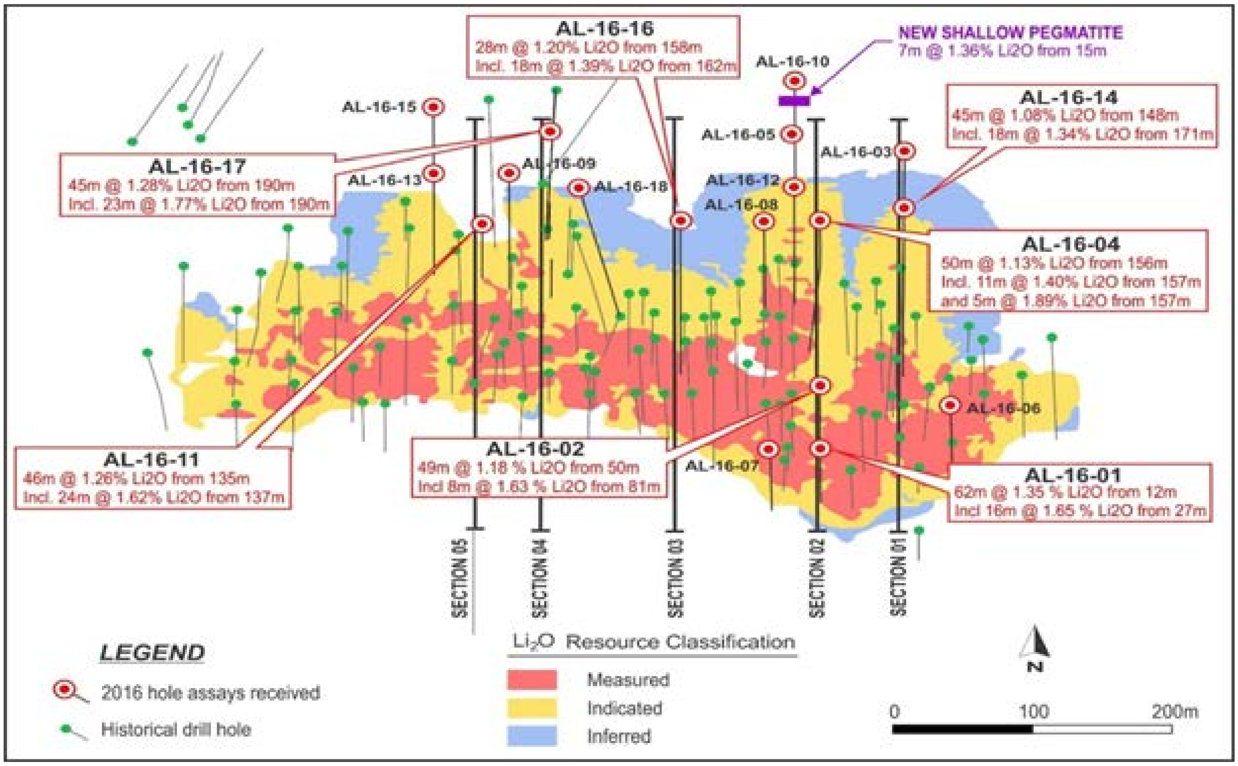


Figure 9-2: Drill Hole Collar Location Plan and Significant Intercepts from the Sayona 2016 Drilling Program

9.4 Sayona Québec Drilling 2017

Sayona Québec completed a Phase 2 diamond drilling program at the Authier property, including 31 holes for 4,117 m (Figure 9-3), having the following objectives:

- Defining the mineralized boundaries and lifting the resource categories in zones in the western sector that were drilled during the 2016 drill program. The 2016 drilling program in the west zone highlighted a number of new high-grade intersections between 120 m to 220 m vertical depth, such as hole AL-16-11, which returned 46 m of 1.26% Li_2O from 135 m, including 24 m of 1.62% Li_2O from 137 m;
- Testing for mineralization in the eastern strike extension at both shallow and deeper levels at similar vertical level to hole AL-16-14, which intercepted 45 m of 1.08% Li_2O from 148 m, including 8 m of 1.36% Li_2O from 149 m and 18 m of 1.34% Li_2O from 171 m;
- Testing for a vertical extension of the mineralization in the gap zone to follow up hole AL-16-16, which intersected 28 m of 1.20% Li_2O from 158 m, including 18 m of 1.32% Li_2O from 149 m; and

- Assessing the resource potential of the new northern pegmatite, which intersected 7 m of 1.36% Li_2O from 15 m in Sayona's 2016 drilling.

The Phase 2 diamond drill holes are detailed as follows (Table 9-3):

Table 9-3: Phase 2 Sayona Drill Hole Collar Location and Intercept Information (downhole intersections in metres)

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-17-01	707210	5360520	331.5	180	-60	283.0	241.8	251.5	9.7	NSR
AL-17-02	707080	5360460	331.0	180	-65	253.0	165.0	197.0	32.0	1.15
including							177.0	184.0	7.0	1.44
and							186.0	192.0	6.0	1.37
AL-17-03	707000	5360500	330.0	180	-60	268.0	222.0	233.0	11.0	1.07
including							226.0	231.0	5.0	1.42
							236.0	240.0	4.0	1.0
AL-17-04	706900	5360425	335.4	180	-70	264.0	166.0	177.0	11.0	0.88
including							166.0	169.0	3.0	1.26
							214.0	225.0	11.0	1.03
including							218.0	222.0	7.0	1.26
AL-17-05	706800	5360425	344.9	180	-75	303.0	199.0	205.0	6.0	1.09
							224.0	243.0	19.0	1.26
including							224.0	233.0	9.0	1.69
AL-17-06	706900	5360360	331.9	180	-55	240.0				NSR
AL-17-07	706803	5360356	339.0	180	-55	246.0	210.0	211.0	1.0	0.64
							214.0	219.0	6.0	0.89
including							215.0	216.0	1.0	1.48
AL-17-08	706802	5360310	335.0	180	-45	219.0	165.0	173.0	8.0	1.07
including							167.0	170.0	3.0	1.31
AL-17-09	707500	5360630	339.2	180	-55	90.0	26.0	31.0	5.0	0.84
including							28.0	29.0	1.0	2.34
AL-17-10	707500	5360680	340.3	180	-55	78.0	20.0	21.0	1.0	0.62
AL-17-11	707450	5360615	336.9	180	-55	48.0	23.0	29.0	6.0	1.32
including							24.0	27.0	3.0	1.76
AL-17-12	707550	5360615	338.7	180	-55	72.0	27.0	32.0	5.0	0.90
including							30.0	31.0	1.0	1.71
AL-17-13	707720	5360440	332.5	180	-55	228.0	153.0	156.0	3.0	1.17
including							154.0	156.0	2.0	1.32
							163.0	189.0	26.0	1.26

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
including							169.0	184.0	15.0	1.42
AL-17-14	707780	5360440	332.3	180	-55	213.0	169.0	189.0	20.0	0.95
including							170.0	180.0	10.0	1.19
AL-17-15	707780	5360250	329.8	180	-55	81.0	11.0	14.0	3.0	1.02
including							12.0	13.0	1.0	1.40
AL-17-16	707700	5360210	328.6	180	-50	87.0	8.0	15.0	7.0	0.76
including							10.00	11.0	1.0	1.10
AL-17-17	707830	5360250	327.0	180	-60	57.0	22.0	23.0	1.0	1.13
AL-17-18	707400	5360610	335.8	180	-55	39.0	22.0	26.0	4.0	0.82
AL-17-19	707350	5360610	335.9	180	-55	45.0	10.73	19.0	8.27	0.88
including							10.73	15.0	4.27	1.27
AL-17-20	707450	5360680	338.4	180	-55	51.0				NSR
AL-17-21	707550	5360680	341.6	180	-90	69.0				NSR
AL-17-22	707400	5360525	334.06	180	-60	271.0	227.0	256.0	29.0	0.92
including							232.0	245.0	13.0	1.10
including							248.0	249.0	4.0	1.46
AL-17-23	707600	5360615	338.7	180	-55	36.0	16.0	24.0	9.0	0.82
including							21.0	24.0	3.0	1.53
AL-17-24	707323	5360628	335.9	180	-55	39.0	12.0	15.0	3.0	0.56
including							12.0	13.0	1.0	1.13
AL-17-25	707308	5360671	336.27	180	-65	42.0				NSR
AL-17-26	707890	5360265	332.5	180	-65	60.0	27.0	39.0	13.0	0.73
including							27.0	31.0	4.0	0.95
including							37.0	39.0	2.0	1.33
AL-17-27	707890	5360345	332.5	180	-65	87.0				NSR
AL-17-28	707720	5360345	331.1	180	-65	181.0				NSR
AL-17-29	707935	5360341	332.5	180	-45	71.0				NSR
AL-17-30	707833	5360286	332.5	180	-45	66.0	16.0	19.0	3.0	0.84
							30.0	40.0	10.0	1.04
including							30.0	33.0	3.0	1.26
including							35.0	39.0	4.0	1.16
AL-17-31	707740	5360615	332.5	180	-65	30.0				NSR

Note: Downhole widths are not true widths
NSR: Not Significant Results

The highlights of the 2017 drilling program include:

- Extension of the mineralization within the main pegmatite orebody by 150 m to the east, up to 300 m to the west within the deeper levels, and 200 m to the west at shallower levels and at depth in the gap zone;
- The east-west strike length of the main deposit has now been extended from 850 m to 1,100 m, with an average thickness of 25 m, ranging from 4 m to 55 m, dipping at 40 to 50 degrees to the north. The orebody remains open to the east, west and at depth; and
- Delineation of the Authier North pegmatite, which has 670 m of drilling completed in 13 holes. The northern pegmatite has a narrow and gently-dipping geometry between 10 m and 25 m vertical depth, not visible from the surface, and downhole intersections typically averaging 5 m to 8 m in width. The pegmatite remains open in all directions. Sayona Québec aims to delineate a resource at shallow levels that would be amenable to open-cut mining at a low stripping ratio.

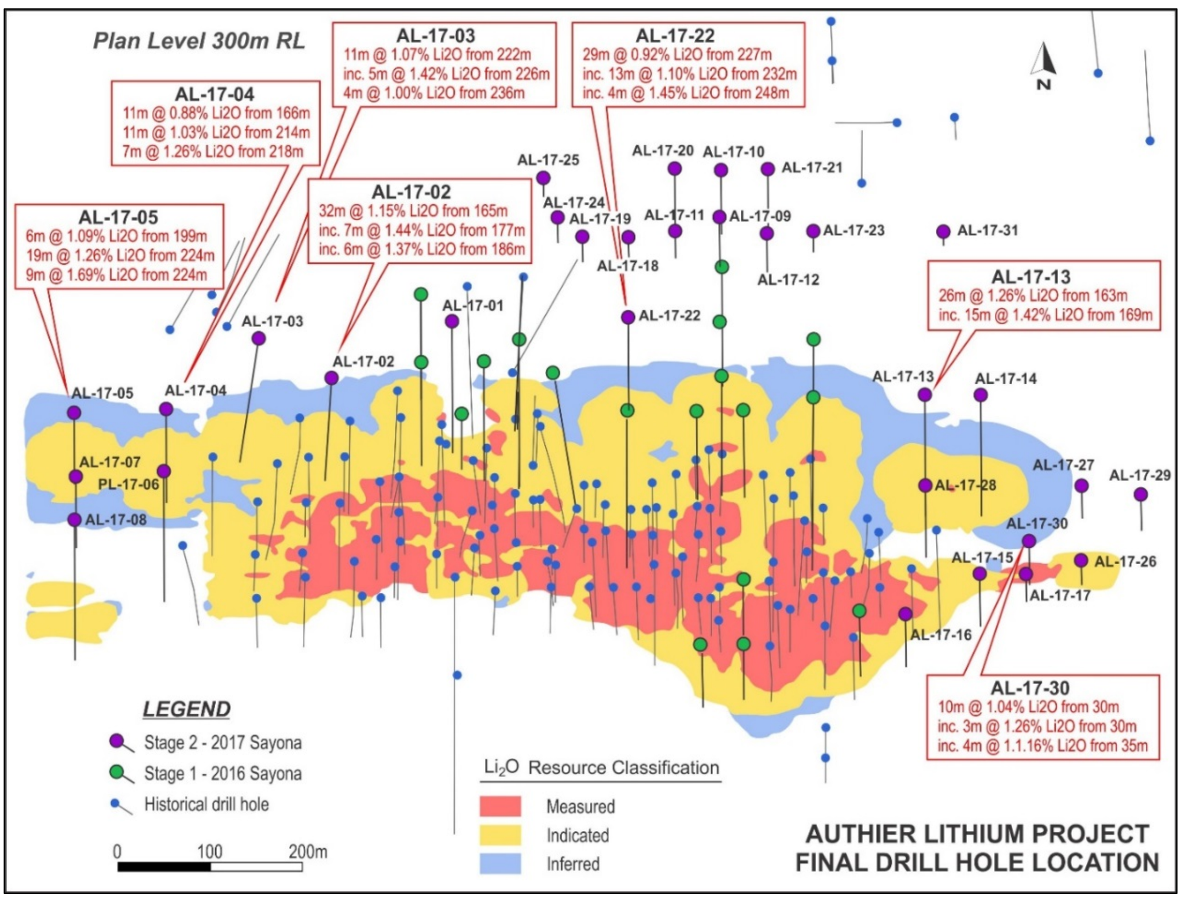


Figure 9-3: Drill Hole Collar Location Plan and Significant Intercepts from the Sayona 2016 Drilling Program

9.5 Drill Hole Results by Sector

Western Zone

Drilling has successfully defined a 300 m western extension of the main Authier pegmatite at between 110 m and 220 m vertical depth, including:

- AL-17-02: 32 m of 1.15% Li_2O, including 7 m of 1.44% Li_2O;
- AL-17-05: 19 m of 1.26% Li_2O, including 9 m of 1.69% Li_2O; and
- AL-17-08: 8 m of 1.07 % Li_2O from 165 m, including 3 m of 1.31% Li_2O from 167 m.

AL-17-02 and AL-17-05 demonstrated similar widths and grades to those in the deeper, Phase 1 holes, which included:

- AL-16-13: 24 m of 1.25% Li_2O from 184 m and 8 m of 0.91% Li_2O from 216 m; and
- AL-16-15: 20 m of 1.32% Li_2O from 242 m, including 11 m of 1.61% Li_2O from 248 m.

The results indicate a potential western plunge of the high-grade mineralization at deeper levels within the western sector. The higher-grade mineralization below the economic open-cut pit depths could be amenable to future underground mining.

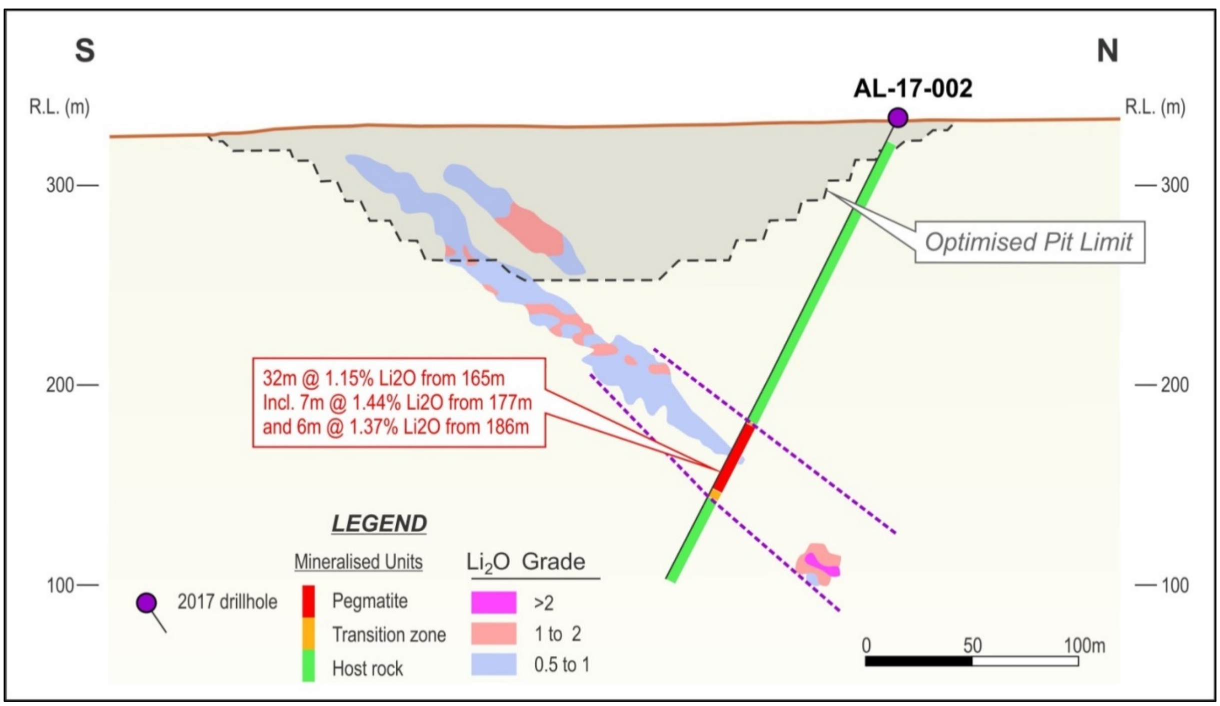


Figure 9-4: Section 707080 mE Looking West, Demonstrating the Extension of Mineralization Below the Open-cut Pit Limit Outlined in the February 2017 Prefeasibility Study

AL-17-01, AL-17-06 and AL-17-07 have intercepted narrow zones of low-grade to barren pegmatite, which has been affected by a large north-south fault cross-cutting the mineralization in the Beaver Dam area (Section 707560 m East). The pegmatite pinches within the fault zone, but shows no significant evidence of post-mineral displacement.

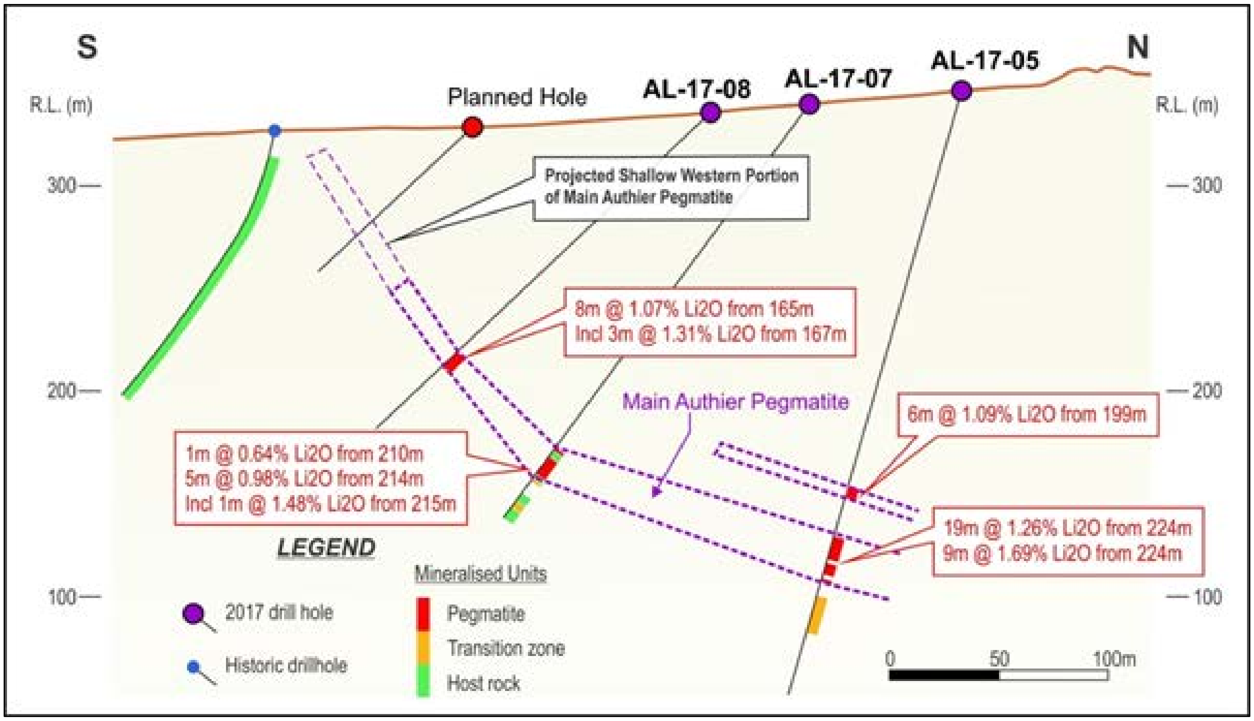


Figure 9-5: Section 706800 mE Looking West Demonstrating Western-most Extensions Mineralization Between 110 m to 220 m Vertical Depth as well as the Potential Shallower Extension that has to be Tested in Step Forward Hole of AL-17-08

Sayona Québec believes that the western sector remains highly prospective for further mineralization west of AL-17-05, AL-17-07 and AL-17-08. Figure 9-6 shows the main Authier pegmatite in relation to the local magnetic geophysical image. The main orebody is strongly correlated to a deep magnetic low, which extends to the western tenement boundary. Additional drilling will be required to extend the mineralization further west.

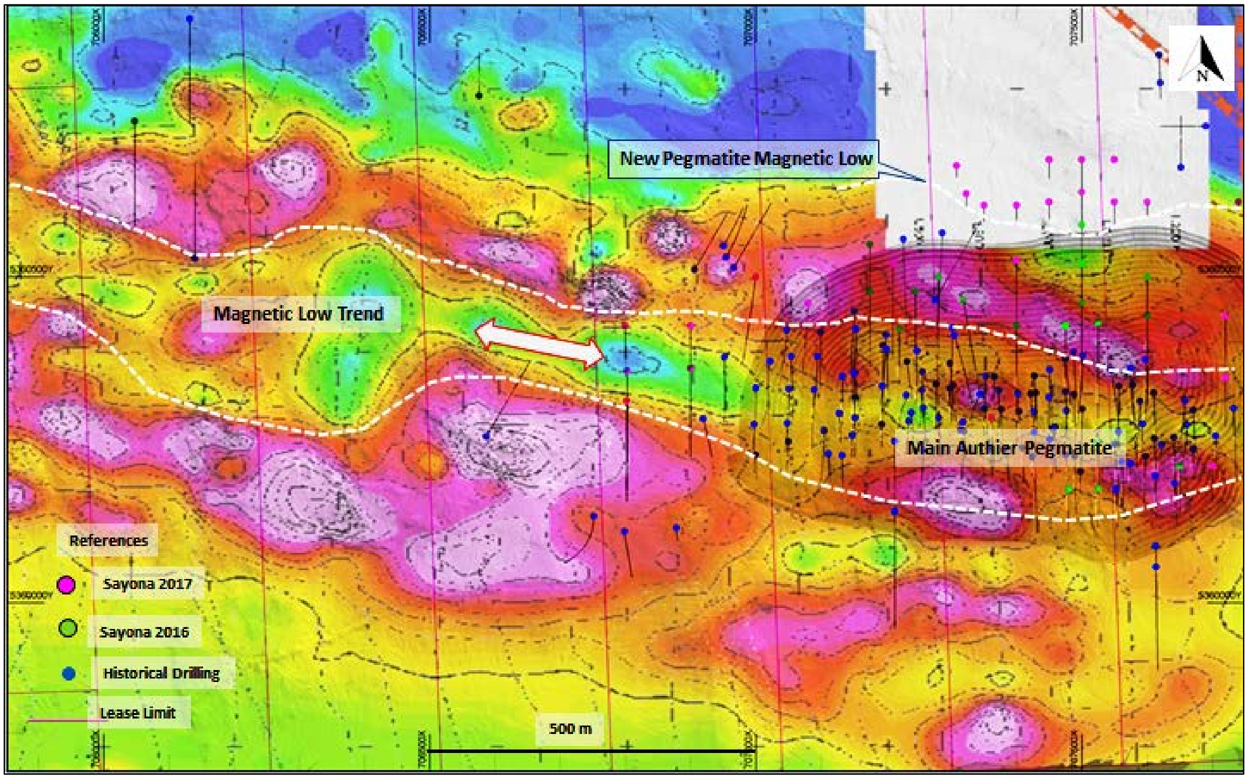


Figure 9-6: Magnetic Geophysical Image and the Main Authier Pegmatite Orebody, which has been Extensively Drilled (right side of image)

Gap Zone

AL-17-22 intersected a thick zone of spodumene mineralization in the gap zone, 29 m of 0.92% Li_2O, confirming an 85 m down-dip extension of the exploratory Phase 1 drill hole AL-16-16, which intersected 28 m of 1.20% Li_2O from 158 m, including 18 m of 1.39% Li_2O from 162 m. AL-17-22 has confirmed an extension of the resource down to approximately 200 m in the gap zone.

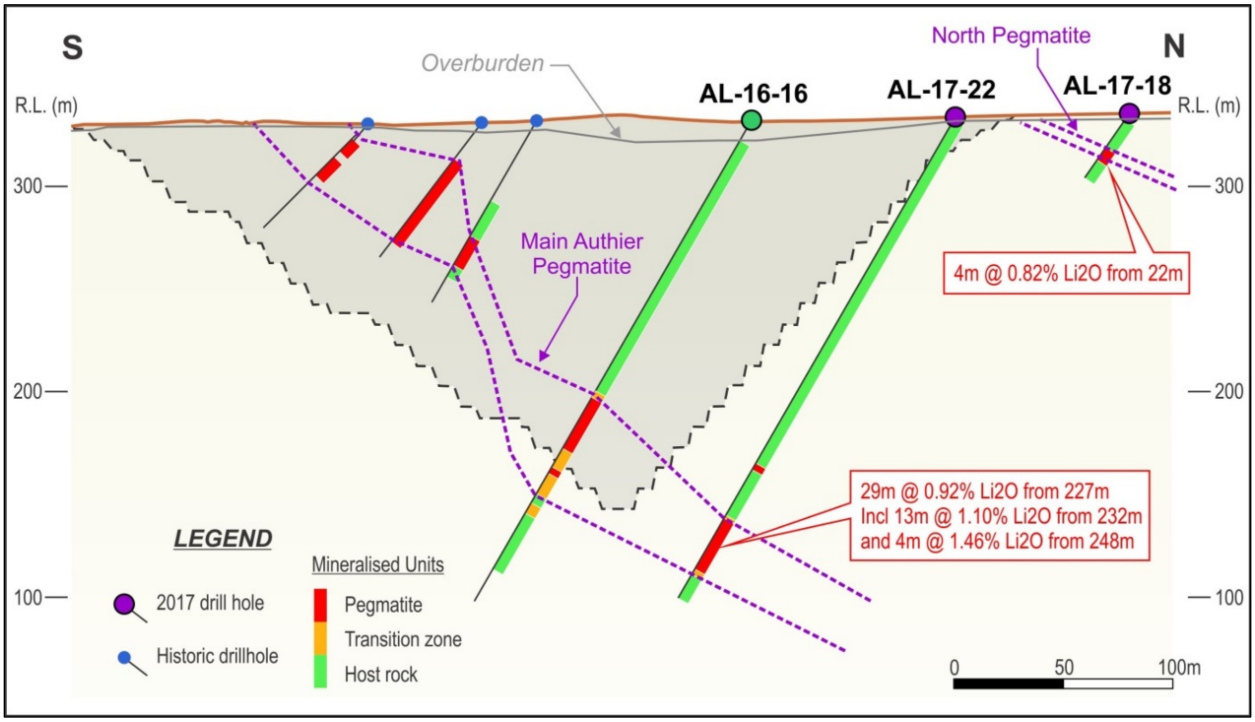


Figure 9-7: Section 707400 mE Looking West (Gap Zone)
Showing the Dip Extension of Mineralization Below the Open-cut Pit Outlined in the February 2017 Prefeasibility Study. Hole AL-17-18 is part of the new Authier North pegmatite and was collared 50 m west of hole AL-17-11 (6 m of 1.32% Li_2O from 23 m, including 3 m of 1.76% Li_2O from 24 m).

Eastern Zone Deep

Holes AL-17-13 and AL-17-14 in the eastern deep zone have extended mineralization 150 m to the east. Hole AL-17-13 yielded 26 m of 1.26% Li_2O from 163 m, including 15 m of 1.42% Li_2O from 169 m, and is located 120 m east of AL-16-14, which intercepted mineralized pegmatite from a vertical depth of 120 m and is expected to result in an 80 m deepening of the current pit outline.

Hole AL-17-28, a 100 m step forward from AL-17-13, intercepted low-grade pegmatite that was affected by a fault zone, which caused a local pinching of the main Authier pegmatite.

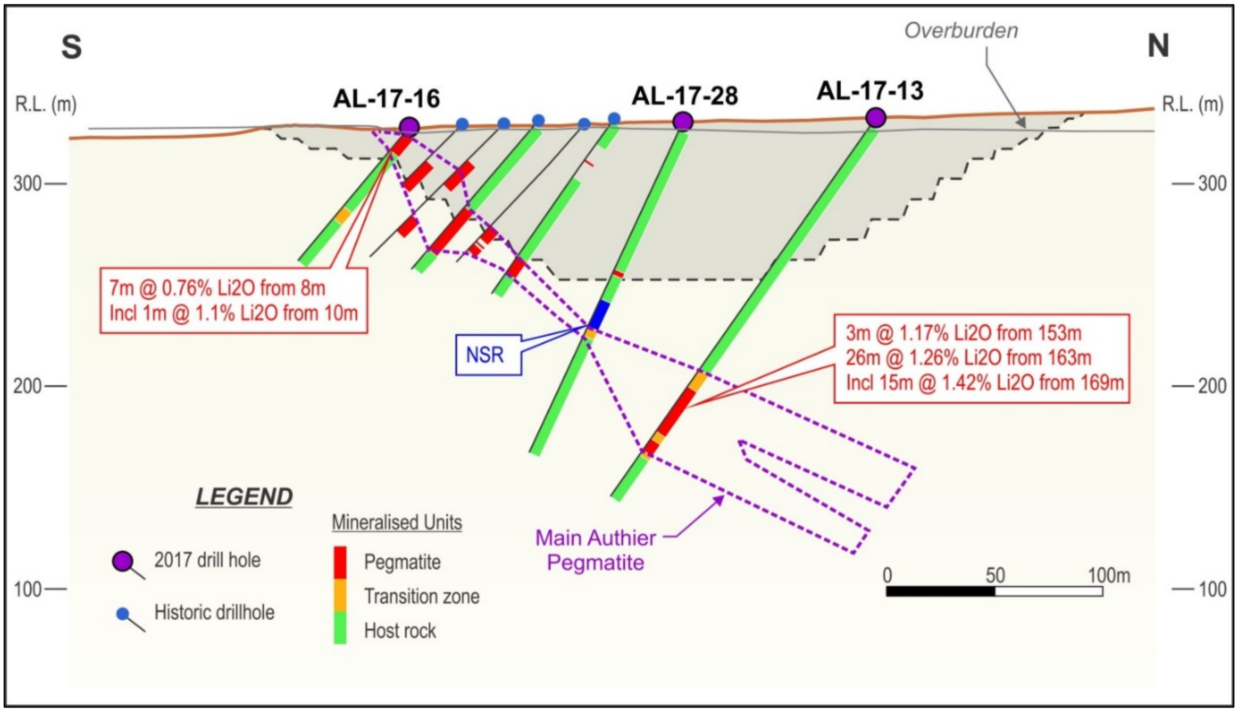


Figure 9-8: Section 707725 mE Looking West
Showing the Down-dip Extension of Mineralization Below the Open-cut pit Outlined in the February 2017 Prefeasibility Study. Hole AL-17-13 is located 120 m east of AL-6-14, and the bottom of the pit should deepen approximately 80 m to incorporate the mineralized interval from AL-17-13

Eastern Zone Shallow

Hole AL-17-16 intercepted a narrow zone of mineralized pegmatite, 7 m of 0.76% Li_2O, within a wider zone of low-grade to barren pegmatite at shallow levels. It is interpreted that the mineralization has been pinched with respect to the wider pegmatite intercepted by the following holes:

- AL-17-30: 10 m of 1.04% Li_2O from 30 m, including 3 m of 1.26% Li_2O from 30 m; and
- AL-17-26: 13 m of 0.73% Li_2O from 27 m, including 2 m of 1.33% Li_2O from 37 m.

Hole AL-17-17 intercepted the narrow, lower portion of the eroded pegmatite, 1 m of 1.03% Li_2O, immediately below 12 m of overburden being collared 35 m south (same section) of AL-17-30.

Holes AL-17-30 and AL-17-26, separated 65 m east-west, intercepted the main pegmatite slightly deeper than AL-17-15 and AL-17-17. The narrow mineralization intercepted by AL-17-15 was extended 165 m down-dip by AL-17-14, which yielded 20 m of 0.95% Li_2O from 169 m, including 10 m of 1.19% Li_2O from 170 m, from a vertical depth of 135 m and collared 185 m north in the same section.

Holes AL-17-27 and AL-17-29, the easternmost holes, intercepted narrow barren pegmatite in fault zones. The geometry of the pegmatite at narrow levels pinch and swells, but it is considered open and further drilling is required to test the easternmost strike extent.

Northern Pegmatite

During Phase 2, drilling began to define the geometry of the new northern pegmatite, located 400 m north of the main Authier pegmatite. During the Phase 1 drilling, AL-16-10 intersected 7 m of 1.36% Li_2O from 7 m in a step-back hole targeting deeper mineralization in the main pegmatite. Drilling from the Phase 2 program has now defined additional mineralization over 300 m in strike length and the system remains open in all directions.

Such a mineralized zone was built using a reference east–west line, 35 m north of AL-16-11, in a 50 m by 50 m drilling grid. The most significant holes are:

- AL-17-11: 6 m of 1.32% Li_2O from 23 m, including 3 m of 1.76% Li_2O from 24 m;
- AL-17-12: 5 m of 0.90% Li_2O from 27 m, including 1 m of 1.71% Li_2O from 30 m;
- AL-17-19: 8.27 m of 0.88% Li_2O from 10.7 m, including 4.27 m of 1.27% Li_2O from 10.7 m; AL-17-23: 8 m of 0.86% Li_2O from 16 m, including 3 m of 1.53% Li_2O from 21 m.

Fifty metres (50 m) step-back holes AL-17-10, AL-17-20, AL-17-21, AL-17-24 and AL-17-25, as well as scout hole AL-17-31, intercepted narrow and low-grade to barren pegmatite. While the grades were lower than anticipated, Sayona Québec believes the system has good potential to host further mineralization. Zones within the pegmatite occur as coarse-grained, narrow, high-grade mineralization, suggesting potential for a large feeder system at depth. Further drilling will be required to test the down-dip extensions of the pegmatite, which has only been drilled to shallow levels.

Figure 9-9: Hole AL-17-10 in the Northern Pegmatite, Which Intersected 7 m of 1.36% Li_2O from a Downhole Depth of 15 m (Vertical Depth of 12 m), Including 2 m of 2.24% Li_2O from 17 m

9.6 Sayona Québec Drilling 2018

Sayona Québec completed a Phase 3 diamond drilling program at the Authier property, including 33 holes for 3,282.6 m (Figure 9-3) and having the following objectives:

- Converting the inferred mineral resources to measured and indicated, and upgrading ore reserves for the DFS;
- Exploring for extensions to the existing mineral resources and other potential mineralization within the tenement package;
- Collecting geotechnical data for incorporation into the DFS and 5,000 kg of core for pilot metallurgical testing; and
- Condemnation drilling in areas planned for infrastructure.

Resource Expansion and Exploration Drilling

A total of 19 diamond core holes (NQ diameter) for 2,170 m were completed as part of the Phase 3 drilling program.

A number of diamond drill holes have intercepted high-grade spodumene mineralization with the best intercepts including:

- AL-18-09: 25 m of 1.48% Li_2O from 79 m including 6 m of 1.77% Li_2O from 80 m and 6 m of 1.78% Li_2O from 94 m;
- AL-18-10: 6 m of 1.26% Li_2O from 97.4 m, including 4 m of 1.52% Li_2O from 98.4 m;
- AL-18-16: 37 m of 1.03% Li_2O from 255 m, including 11 m of 1.24% Li_2O from 266 m and 3 m of 1.67% Li_2O from 281 m; and
- AL-18-17: 33 m of 1.18% Li_2O from 160 m, including 10 m of 1.25% Li_2O from 166 m and 3 m of 1.75% Li_2O from 190 m.

Drilling has successfully demonstrated depth extensions of the mineralization at the main Authier pegmatite. Infill drilling successfully targeted areas of low drilling density with the objective of upgrading the resource categories. A number of holes testing the eastern extensions of the main Authier pegmatite at shallow levels were stopped due to the presence of a fault zone, but warrant further testing in a future drilling program.

A potential third deep pegmatite dyke was intercepted at a depth of 300 m and returned low grade mineralization due to the replacement of spodumene by phengite. Further drilling will be required to test the potential of this system, especially at shallower levels.

Drilling has successfully extended the mineralization at the Authier North pegmatite from 300 m to 500 m in strike length and at depth. The system remains open in all directions.

Sayona Québec believes that the new drilling has the potential to expand the size of the existing mineral resource and ore reserve, and that the mineralization remains open in all directions.

9.7 Drill Hole Results by Sector

Main Authier Pegmatite

The following summarizes the key outcomes of the resource expansion and exploration drilling program within Phase 3 drilling:

- AL-18-01 and AL-18-02 were stopped before hitting the target due to a fault zone;
- AL-18-09, 18-04, 18-05, 18-06 and 18-07 tested the eastern extension of the main Authier pegmatite at shallow levels, intercepting narrow zones of weak lithium mineralization;
- AL-18-08 and AL-18-09 filled the gaps within the East zone of the main Authier pegmatite resource from 40 m to 70 m vertical depth. AL-18-09 yielded 25 m of 1.48% Li_2O from 79 m, including 6 m of 1.77% Li_2O from 80 m and 6 m of 1.78% Li_2O from 94 m;
- AL-18-10 intercepted a narrow lithium mineralized zone that filled the gap of the main Authier pegmatite resource in the central part, including 6 m of 1.26% Li_2O from 97.4 m, including 4 m of 1.52% Li_2O from 98.4 m;
- AL-18-12 drilled within a NNE fault zone intercepted narrow and weak lithium anomalies in the west zone;
- AL-18-16 at the deep west zone of the main Authier pegmatite intercepted a wide deep extension of the pegmatite at a vertical depth of 235 m to 270 m, 75 m step back of hole AL-16-15 (20 m of 1.32% Li_2O from 242 m, see ASX release of Nov 16, 2016). A potential third pegmatite dyke was intercepted at a vertical depth of 300 m with 25 m downhole width, which returned no significant spodumene mineralization due the replacement of spodumene by phengite. Additionally, AL-18-16 intercepted the Authier North pegmatite with lithium mineralization at shallow levels; and
- AL-18-17, an infill hole at the East zone of the main Authier pegmatite, intercepted a wide mineralized pegmatite zone of 33 m of 1.18% Li_2O from 160 m, including 10 m of 1.25% Li_2O from 166 m and 3 m of 1.75% Li_2O from 190 m (Figure 9-10).

Sayona Québec believes that the main Authier pegmatite is still open in all directions. The geometry of the mineralized pegmatite at shallow levels in both east and west extensions seems affected by post-mineral faulting, and further drilling should be conducted at mid-to-deep levels to test along strike extension of the main pegmatite. The deep extensions of the main pegmatite are demonstrating excellent grades and widths.

Northern Pegmatite

Holes AL-18-13, AL-18-14 and AL-18-16 successfully extended the mineralization from 250 m to 500 m in strike extension; AL-18-13, AL-18-18 and AL-18-19 were infill holes. The Authier North pegmatite is narrow, gently dipping to the north, and is still open along strike.

A new JORC resource incorporating all the new assay results was prepared and incorporated into the DFS (2018).

The resource expansion and exploration drill holes results as part of Phase 3 diamond drilling (Table 9-4) are detailed as follows:

Table 9-4: Sayona Phase 3 Resource Expansion and Exploration Drill Hole Collar Location and Intercept Information (downhole intersections in metres)

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-18-01	707939	5360341	333	180	-45	93				NSR
AL-18-02	707934	5360304	333	180	-55	39				NSR
AL-18-03	708127	5360298	333	180	-55	75				NSR
AL-18-04	708034	5360307	333	180	-55	90				NSR
AL-18-05	707984	5360279	333	180	-45	69				NSR
AL-18-06	707885	5360342	333	180	-50	153				NSR
AL-18-07	707829	5360348	331.1	180	-55	129.45				NSR
AL-18-08	707786	5360332	331.1	180	-45	132	57.3	64	6.7	0.94
AL-18-09	707720	5360345	331.1	180	-45	129	79	104	25	1.48
including							80	86	6	1.77
including							94	100	6	1.78
AL-18-10	707472	5360320	333.28	180	-55	156	97.4	103.4	6	1.26
including							98.4	102.4	4	1.52
AL-18-11	707400	5360360	335	180	-55	175				NSR
AL-18-12	706760	5360224	330	180	-45	138				NSR
AL-18-13	707250	5360600	333.1	180	-55	57	16.85	22.05	5.2	0.82
including							18	20	2	1.02
AL-18-14	707690	5360590	338.1	180	-55	36	8	14	6	0.85
including							10	11	1	2.01
AL-18-15	707325	5360606	330	180	-55	60				NSR
AL-18-16	707175	5360600	333.1	180	-70	342	18	22	4	1.08
							255	292	37	1.03
including							266	277	11	1.24
including							281	284	3	1.67

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-18-17	707665	5360440	332.46	180	-55	231	160	193	33	1.18
including							166	176	10	1.25
including							190	193	3	1.75
AL-18-18	707600	5360580	344	170	-55	39	11	18	7	0.94
including							11	12	1	1.46
including							14	18	4	1.12
AL-18-19	707550	5360580	344	170	-55	27	15.4	18	2.6	1.14
							16	17	1	1.84

Note: Downhole widths are not true widths
NSR: Not Significant Results

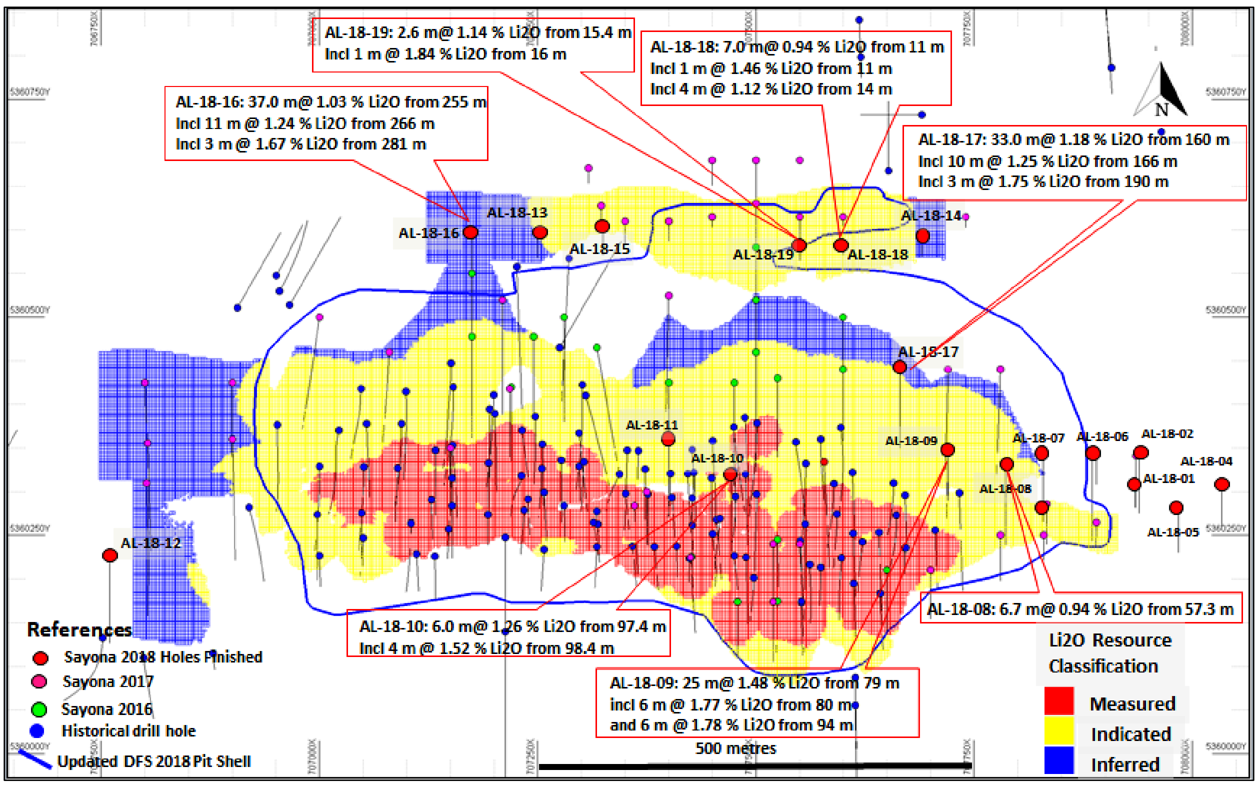


Figure 9-10: Drill Hole Collar Location Plan and Significant Intercepts from the Sayona 2018 Resource Expansion and Exploration Drilling Program

Metallurgical Pilot Plan Drilling Program

Phase 3 diamond drilling program includes a total of 7 diamond holes for 769.5 m, 680 m PQ core diameter and 89.5 m HQ diameter, which were drilled in November/December 2017 with the objective of collecting 5,000 kg of core for pilot metallurgical testing. The 7 holes were collared in the same pads as previous Sayona and historical pre-Sayona drill holes, using the same azimuth and dip parameters to drill perpendicular to the main Authier pegmatite body (Figure 9-11).

The metallurgical pilot plan drill holes results of Phase 3 diamond drilling (Table 9-5) are detailed as follows:

Table 9-5: Sayona Phase 3 Metallurgical Pilot Plan Drill Hole Collar Location and Intercept Information (downhole intersections in metres)

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-17-32	707520	5360175	328.8	180	-45	97.5	13	78	65	1.29
including							27	48	21	1.54
AL-17-33	707520	5360240	331.0	180	-45	119.5	53	99	46	1.28
including							54	66	12	1.50
AL-17-34	707550	5360240	331.0	177	-45	96.0	56	91	35	1.09
AL-17-35	707425	5360225	330.43	177	-45	73.5	4.7	42	37.3	0.98
including							27	42	15	1.10
AL-17-36	707150	5360350	329.9	180	-52	112	67	81	14	1.47
							83	94.9	11.9	1.57
							104	112	8	1.49
AL-17-37	707218	5360418	330.0	180	-65	186	139	146	7	1.15
							151	167	16	0.54
AL-17-38	707375	5360300	330.0	180	-45	85	34	52	18	0.96
							54	60	6	1.32
							63	65	2	1.30
Note: Downhole widths are not true widths										

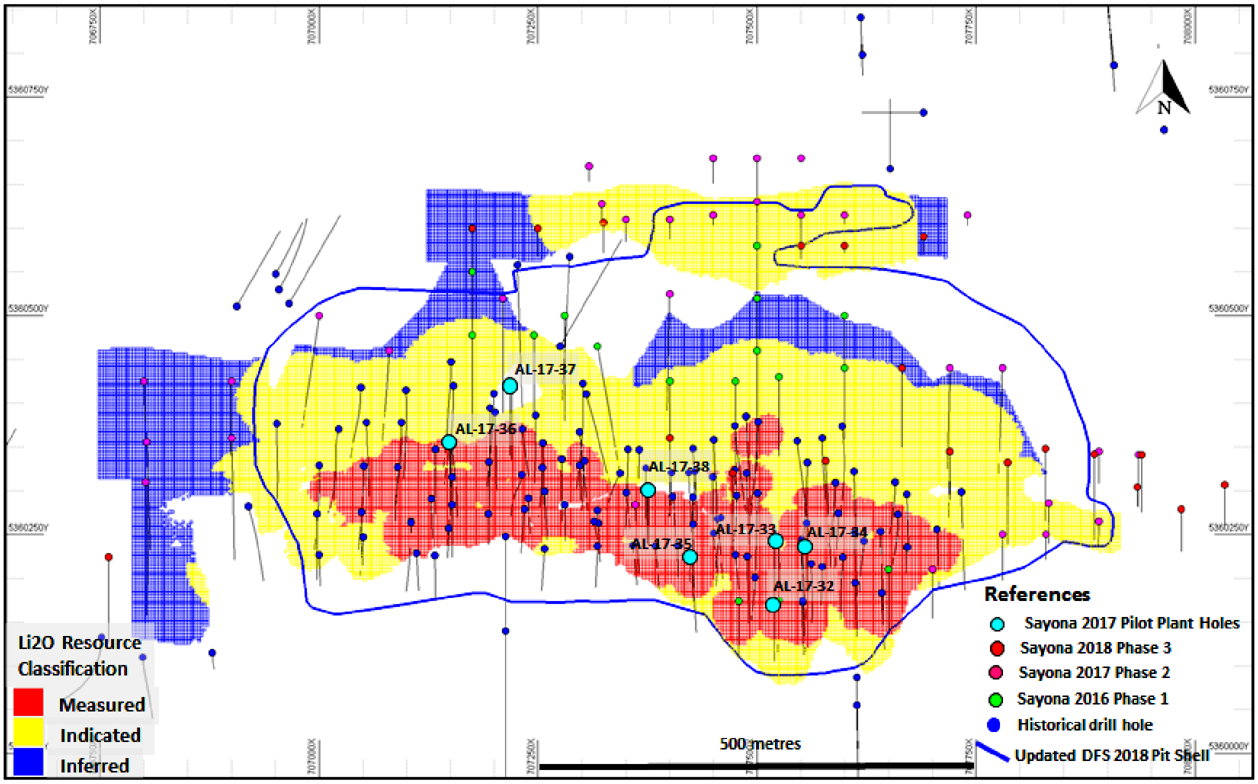


Figure 9-11: Drill Hole Collar Location Plan View Highlighting (in light blue) the Metallurgical Pilot Plant Drill Holes Completed During Phase 3 Drilling at Authier Project

Condemnation Holes

Seven diamond core holes NQ diameter for 342.65 m were completed in the zone north of the Authier deposit to test and discard potential mineralized pegmatite within the planned infrastructure zone. The areas tested were selected based on geological mapping and sampling, close to outcropping pegmatite, which returned low grade lithium anomalies after surface rock chip sampling or nearby historical drilling (Figure 9-12). All of the holes intercepted narrow zones of low grade to barren pegmatite dykes at different depths. Sampling has been performed to confirm the low grade to barren character of the pegmatites dykes and results will be made available.

The condemnation drill holes results of Phase 3 diamond drilling (Table 9-6) are detailed as follows:

Table 9-6: Sayona Phase 3 Metallurgical Pilot Plan Drill Hole Collar Location and Intercept Information (downhole intersections in metres)

Drill Hole	East	North	RL	Azimuth	Dip	Depth	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)
AL-18-20	707348	5360950	340	180	-50	48				NSR
AL-18-21	707036.7	5360304	341	180	-50	42				NSR
AL-18-22	706039	5360905	341	180	-50	51				NSR
AL-18-23	706115	5360890	340	180	-50	51				NSR
AL-18-24	706107	5361328	342	180	-50	48.65				NSR
AL-18-25	706446	5361165	341	180	-50	51				NSR
AL-18-26	706450	5360970	340	180	-50	51				NSR
Note: Downhole widths are not true widths NSR: Not significant results										

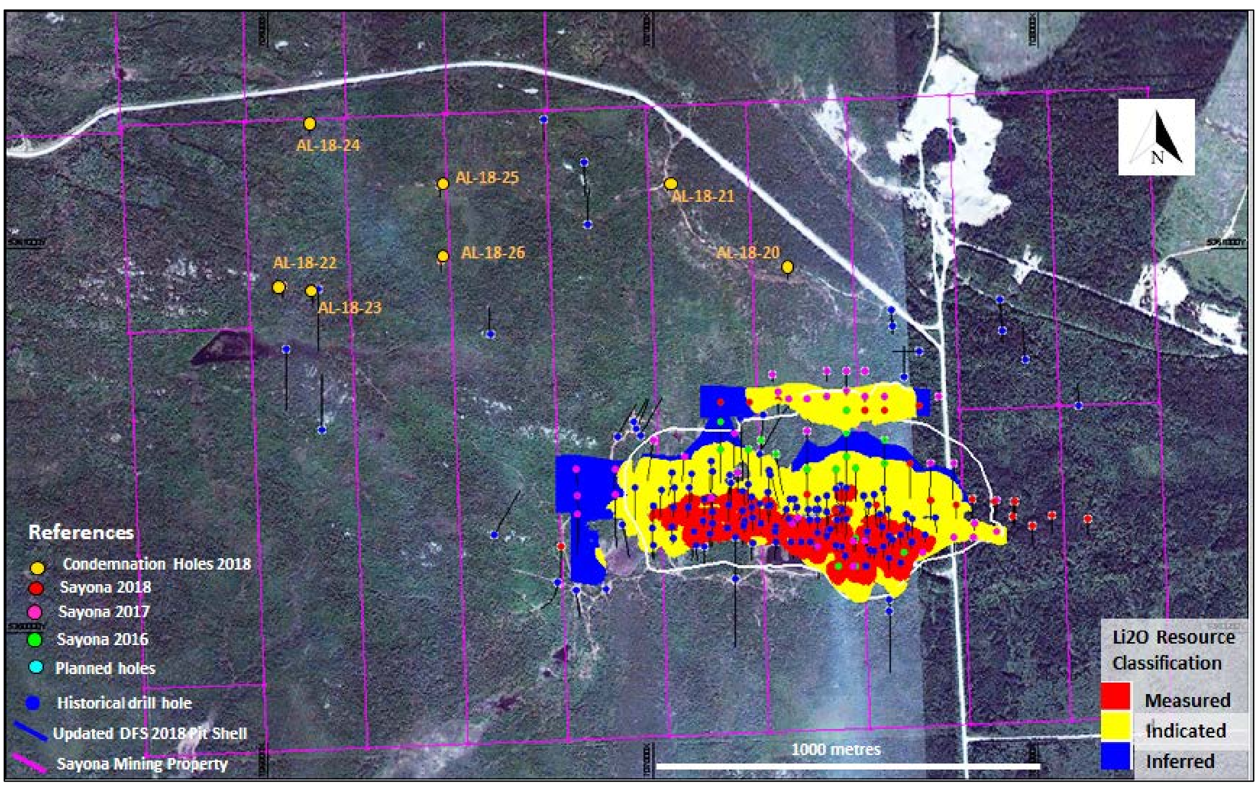


Figure 9-12: Drill Hole Collar Location Plan View Highlighting (orange) the Condemnation Drill Holes Completed during Phase 3 drilling at the Authier Property

10. SAMPLE PREPARATION, ANALYSIS AND SECURITY

10.1 General

The following section presents the sample preparation, analysis and security procedures followed during the various drilling campaigns. These procedures were reviewed by SGS in 2012, during the course of the preliminary economic assessment the Authier Lithium project and subsequently reviewed by Sayona Québec in 2016 during the pre-feasibility study.

10.2 ALS Minerals 2010 Procedures

All samples received at ALS in 2010 from the project were digitally inventoried using bar codes, then weighed. Samples with excess humidity were dried. Samples were crushed in a jaw and/or roll crusher to 70% passing 9 mesh. Crushed material was split in a rifle splitter to obtain a 250 g sub-sample, which was then pulverized to 85% passing 200 mesh using either a single component, i.e., flying disk, or a two component, i.e., ring and puck, mill.

The analyses were conducted at the ALS laboratory, an accredited laboratory under ISO/IEC 17025 standards, located in North Vancouver, British Columbia. Two analytical methods were used for samples from the Authier Lithium deposit. The first analytical method used by ALS was the 38 elements analysis, not including lithium, using lithium metaborate fusion, followed by inductively coupled plasma mass spectrometry (ICP-MS) (ALS code ME-MS81). The method used 0.2 g of the pulverized material and returned different detection limits for each element. The second analytical protocol used at ALS was the ore grade lithium four-acid digestion with inductively coupled plasma – atomic emission spectrometry (ICP-AES) (ALS code Li-OG63). The Li-OG63 analytical method uses approximately 0.4 g of pulp material and returned a lower detection limit of 0.01% Li.

SGS Geostat conducted independent check sampling of selected drill core from the project. The analyses of the check samples were conducted at SGS Canada Inc. Minerals Services laboratory located in Toronto, Ontario (SGS Minerals), which is an accredited ISO/IEC 17025 laboratory. The analytical method used by SGS Minerals is the ore grade analysis using sodium peroxide fusion with induced coupled plasma optical emission spectrometry (ICP-OES) finish methodology with a lower detection limit of 0.01% lithium (SGS code ICP90Q). This method uses 20 g of pulp material.

10.3 AGAT Laboratories 2011-2012 Procedures

Samples received at AGAT Laboratories in 2011-2012 were processed according to the following procedures at the AGAT preparation facilities in Sudbury, Ontario. All samples were inspected and compared to the chain of custody (COC) and logged into the AGAT laboratory management system (AGAT LIMS) then weighed. Drying was undertaken at 60°C on all samples. Sample material was crushed in a Rocklabs Boyd or a TM Terminator Jaw Crusher to 75% passing 10 mesh (2 mm). The crushed material was split with either a rifle splitter or a rotary splitter to obtain a 250 g sub-sample, which was then pulverized to 85% passing 200 mesh (75 µm) using TM, TM-2 pulverizers.

The analyses were conducted at the AGAT laboratory, an accredited laboratory under ISO/IEC 17025 standards, located in Mississauga, Ontario. The analytical protocol used at AGAT is the ore grade lithium four-acid digestion with inductively coupled plasma – optical emission spectrometry (ICP-OES) (AGAT code 201079) -Li. The analytical method uses approximately 0.5 g of pulp material and uses a lower detection limit of 0.0001% lithium.

10.4 SGS 2016 Sampling Procedures

Drill core samples collected during the 2016 diamond drilling program were transported directly by a courier truck contracted by Sayona Québec to the SGS laboratory preparation facilities in Sudbury, Ontario for sample preparation. Procedures followed were based upon industry best practice. All samples were inspected and compared to the chain of custody and logged into the SGS laboratory management system. Samples were then weighed and dried. Samples were crushed to 75% passing 10 mesh (2 mm), split to obtain a 250 g sub-sample, which was pulverized to 85% passing 200 mesh (75 µm). Samples were then shipped to SGS Mineral Services laboratories in Lakefield, Ontario, for analysis.

Analyses of all 2016 drilling samples were conducted at the SGS laboratory located in Lakefield, Ontario, which is an accredited laboratory under ISO/IEC 17025 standards accredited by the Standards Council of Canada. The analytical protocol used at SGS Lakefield was method GE ICP90A 29 element analysis - sodium peroxide fusion that involved the complete dissolution of the sample in molten flux for ICP-AES analysis. The detection limits for lithium are 10 ppm (lower) and 10,000 ppm (upper). No geophysical or handheld tools were used.

10.5 SGS 2017 Sampling Procedures

Drill core samples collected during the 2017 diamond drilling program were transported directly by a courier truck contracted by Sayona Québec to the SGS laboratory preparation facilities in Sudbury, Ontario for sample preparation. Procedures followed were based on industry best practice. All samples were inspected and compared to the chain of custody and logged into the SGS laboratory management system. Samples were then weighed and dried. Samples were crushed to 75% passing 10 mesh (2 mm), split to obtain a 250 g sub-sample which was pulverized to 85% passing 200 mesh (75 µm). Samples were then shipped to SGS Mineral Services laboratories in Lakefield, Ontario, for analysis.

Analyses of all 2017 drilling samples were conducted at the SGS laboratory located in Lakefield, Ontario, which is an accredited laboratory under ISO/IEC 17025 standards accredited by the Standards Council of Canada. The analytical protocol used at SGS Lakefield was method GE ICP90A 29 element analysis - sodium peroxide fusion that involved the complete dissolution of the sample in molten flux for ICP-AES analysis. The detection limits for lithium are 10 ppm (lower) and 10,000 ppm (upper). No geophysical or handheld tools were used.

10.6 Quality Assurance and Quality Control Procedure by Glen Eagle

Over and above the laboratory quality assurance quality control protocol (QA/QC) routinely conducted by ALS using pulp duplicate analysis, Glen Eagle implemented an internal QA/QC protocol consisting of the insertion of reference material, i.e. analytical standards and blanks, on a systematic basis with the samples shipped to ALS. The company also sent pulps from selected mineralized intersections to SGS Minerals for re-analysis. SGS Geostat did not visit the ALS or SGS Minerals facilities, or conduct an audit of the laboratories.

10.6.1 Analytical Standards

Two different standards were used by Glen Eagle for the internal QA/QC program: one low grade lithium (Low-Li) and one high grade lithium (High-Li) standard. Both standards were custom made reference materials from mineralized material coming from the main pegmatite intrusion at the Authier property. To evaluate their expected values, both Low-Li and High-Li standards were analyzed 15 times each at the SGS Minerals laboratory in Toronto and 15 times each at the ALS laboratory in North Vancouver, British-Colombia. The analytical protocol used at SGS Minerals was the mineral grade sodium peroxide fusion with ICP-OES finish described in section 11.1. The analytical protocol used at ALS was the ore grade lithium four-acid digestion with ICP-AES finish described in section 10.2.

For the Low-Li standard, the analytical results returned from SGS Minerals for the 15 samples averaged 0.63% Li_2O versus an average of 0.61% Li_2O for the 15 samples submitted to ALS. For the High-Li standard, the average of the 15 samples analyzed at SGS Minerals returned 2.91% Li_2O versus an average of 2.88% Li_2O for the 15 samples processed at ALS. Each laboratory showed relatively consistent analytical results from one sample to another for each standard analyzed. The averages for each standard also show a good correlation between SGS Minerals and ALS. The results from the analysis of these 30 samples for each Low-Li and High-Li were used to determine the expected values, based upon a mean value from the 30 samples, and the QA/QC warning/failure thresholds, i.e., ±2 standard deviations and ±3 standard deviations respectively. Table 10-1 shows the results for each standard using both analytical protocols.

Table 10-1: Results from Custom Low-Li and High-Li Standards

Glen Eagle Resources Inc. – Authier Project – Standards Certifications			
	Low Grade Standard (% Li_2O)		
	ALS Data	SGS Data	All Data
Count	15	15	30
Mean	0.614	0.629	0.622
Std. Dev	0.042	0.012	0.031
Min	0.588	0.603	0.588
Median	0.605	0.624	0.619
Max	0.764	0.646	0.764
QAQC Thresholds	Warning Range (2 x Std Dev)	Lower Limit	0.559
		Higher Limit	0.684
	Failure Limit (3 x Std Dev)	Lower Limit	0.528
		Higher Limit	0.715
	High Grade Standard (% Li_2O)		
	ALS Data	SGS Data	All Data
Count	15	15	30
Mean	2.884	2.911	2.898
Std. Dev	0.067	0.031	0.053
Min	2.756	2.820	2.756
Median	2.874	2.907	2.907
Max	3.090	2.950	3.090
QAQC Thresholds	Warning Range (2 x Std Dev)	Lower Limit	2.792
		Higher Limit	3.003
	Failure Limit (3 x Std Dev)	Lower Limit	2.739
		Higher Limit	3.056

10.7 2010-2012 Reference Materials (RM) Results

In 2010, Glen Eagle sent samples to ALS Minerals in Vancouver, British Columbia, and starting in 2011, to AGAT in Mississauga, Ontario. During this period, 31 High-Li and 32 Low-Li were inserted into the sampling procedure. A graphic representation of RM quality control failures and the labelling results are included in Table 10-1. The red lines represent three times the standard deviation (±3σ). Of a total of 63 RM samples tested since 2010, seven RM samples (11%) produced results exceeding ±3σ. Similarly, only two RM samples (3%) produced results exceeding 10% of the expected value. Almost all RM analyses fell under the 10% difference from the expected RM value.

10.7.1 Z-Scores

The Z scores were also calculated and plotted (Figure 10-1). The z-score is the difference between the observed RM result and the expected result divided by the expected standard deviation:

z-score = (x - μ) / s,

Where:

x is the observed assay;

μ is the expected assay for the RM;

s is the expected standard deviation for the RM.

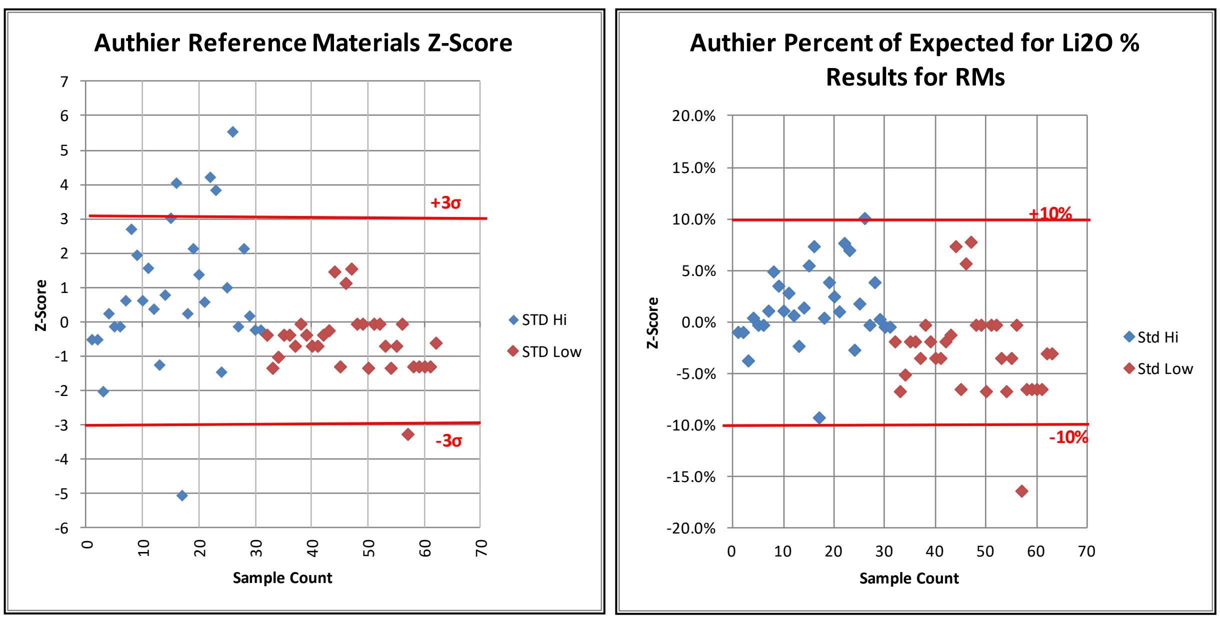


Figure 10-1: RM (STD High, STD Low) Results

10.7.2 ALS Minerals 2010 Reference Materials Results

In 2010, Glen Eagle sent samples to ALS Minerals of Vancouver. In Figure 10-2, the red lines represent the absolute limits of three times the standard deviations (±3σ) and the absolute percentage differences from the RM expected values. Of a total of 31 RM analyses, two RM (6%) produced results exceeding ±3σ the expected value. Additionally, no RM produced results exceeding 10% the RM expected value. Possible mislabels are included in this analysis.

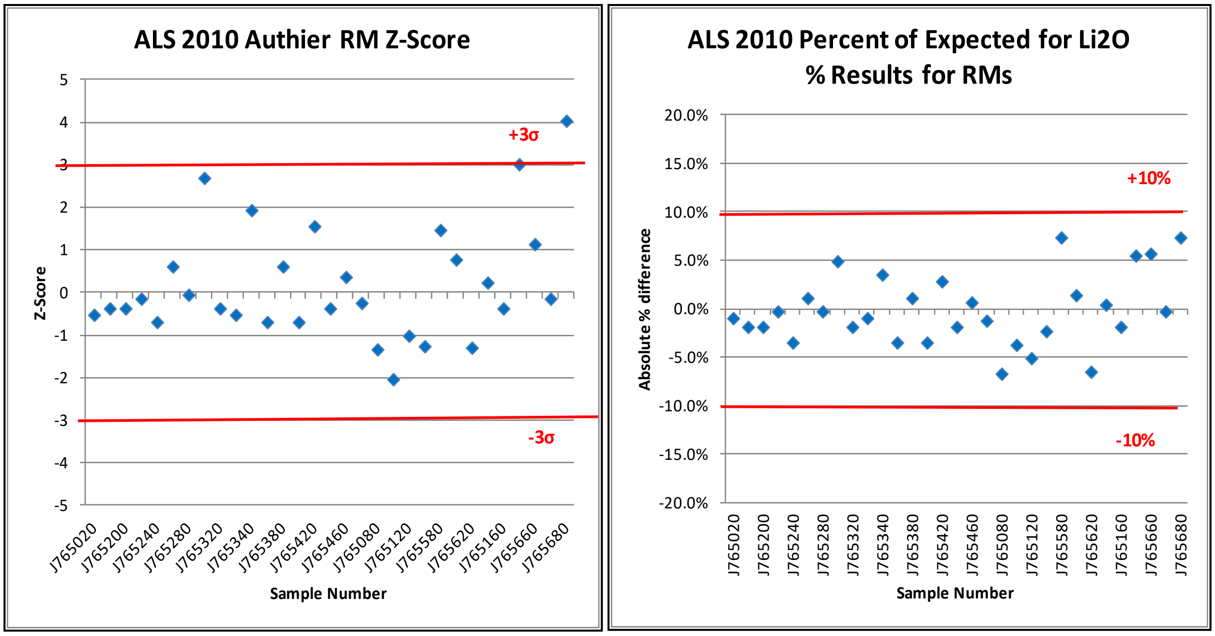


Figure 10-2: ALS 2010 RM Z-Score & Percentage from Expected RM Value

10.7.3 AGAT 2011-2012 Reference Materials Results

Beginning in 2011, Glen Eagle sent samples to AGAT Laboratories in Mississauga for analysis. In Figure 10-3, the red lines represent the absolute limits of three times the standard deviations (±3σ) and the absolute percentage differences from the RM expected values. Out of a total of 32 RM, five RM (15%) produced results exceeding ±3σ the expected value. Additionally, two RM produced results exceeding 10% of the expected value. Possible mislabels are included in this analysis. SGS Geostat is of the opinion that certain RMs were mislabelled at that time.

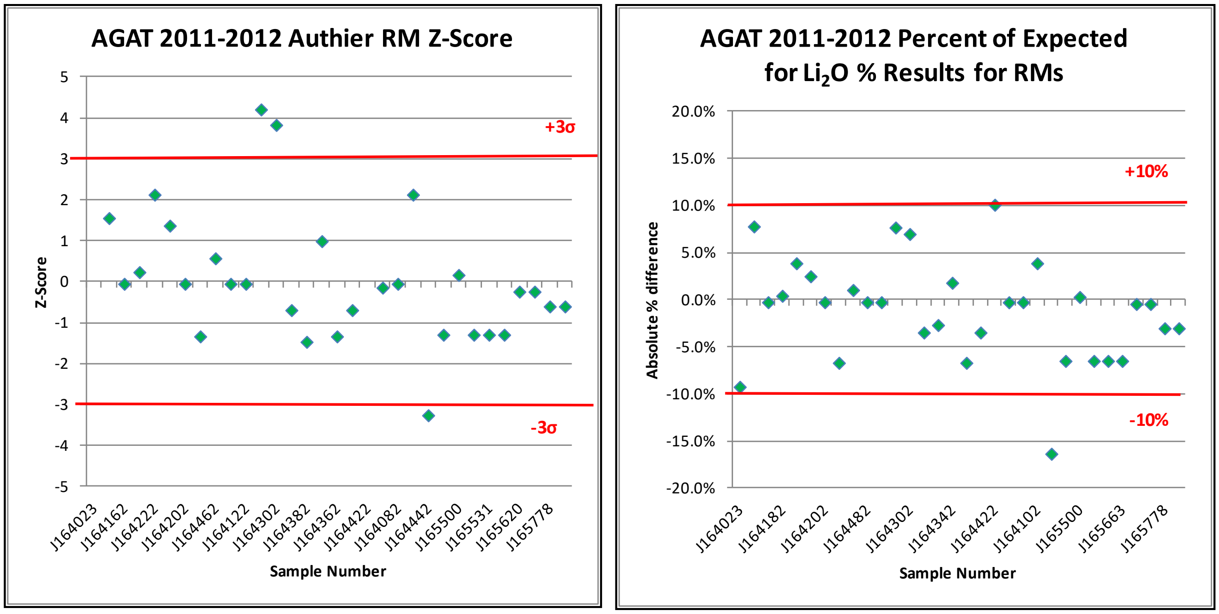


Figure 10-3: AGAT 2011-2012 RM Z-Score & Percentage from Expected RM Value

10.8 Quality Assurance and Quality Control Procedures by Sayona Québec

In addition to the laboratory QA/QC protocol routinely conducted by SGS using standards and pulp duplicate analysis, Sayona Québec applied a QA/QC protocol involving a review of laboratory supplied internal QA/QC and in-house controls consisting of the insertion of in-house reference standards, i.e., high and low grade, prepared with material from the project and certified by lab round-robin, and samples of *barren* material (blanks), on a systematic basis with the samples shipped to SGS. Sample sizes are considered appropriate with regard to the grain size of the sampled material.

10.8.1 Analytical Standards

Two different standards were used by Sayona Québec for the internal QA/QC program: one low grade lithium (Low-Li) and one high grade lithium (High-Li) standards. The samples were the same standards used by Glen Eagle for the 2010-2012 drilling programs. Both standards were custom made references produced from mineralized material from the main pegmatite intrusion at the Authier property. Both Low-Li and High-Li standards were analyzed 15 times each at the SGS Minerals laboratory in Toronto, Ontario, and 15 times each at the ALS laboratory in North Vancouver, British-Colombia.

The analytical protocol used at SGS Minerals was the mineral grade sodium peroxide fusion with ICP-OES finish described in section 10.2. The analytical protocol used at ALS was the ore grade lithium four-acid digestion with ICP-AES finish also described in section 10.2.

For the Low-Li standard, the analytical results returned from SGS Minerals for the 15 samples averaged 0.63% Li_2O versus an average of 0.61% Li_2O for the 15 samples submitted to ALS. For the High-Li standard, the average of the 15 samples analyzed at SGS Minerals returned 2.91% Li_2O versus an average of 2.88% Li_2O for the 15 samples processed at ALS. Each laboratory shows relatively consistent analytical results from one sample to another for each standard analyzed. The averages for each standard also show a good correlation between SGS Minerals and ALS. The results from the analysis of the 30 samples for each Low-Li and High-Li are used to determine the expected values, based upon a mean value from the 30 samples, and the QA/QC warning/failure thresholds, i.e., ±2 standard deviations and ±3 standard deviations, respectively. Table 10-2 shows the results for each standard using both analytical protocols.

10.9 2016 Reference Materials Results

The 2016 QA/QC follow-up was conducted by Rock Solid Data Consultancy Pty., mandated by Sayona Québec, which prepared a report that included performance of reference material (Sayona Québec and SGS).

In 2016, Sayona Québec included the two standards at random intervals during sampling at a rate of approximately 1:20 samples. All results for both the High-Li and Low-Li reported above the expected values and fell within ±10% from expected value. The results show a consistent bias with a mean of +4.91% for High-Li and +4.56% for Low-Li. The bias might be attributed to the difference between the SGS method by which the standard samples were analyzed (SGS GE_ICP90A) and the methods used for deriving the expected value for the standards (SGS ICP90Q and ALS Li-OG63).

In Table 10-4 and Table 10-5, orange lines represent the ±3σ from the expected value and the red lines represent ±10% of the expected value. The results for the 29 High-Li and 25 Low-Li samples are summarized in Table 10-2.

Table 10-2: Results from Custom Low-Li and High-Li Standards – Sayona Québec 2016

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp Value	Exp SD		Mean Li	SD	CV	Mean Bias
High_Li	FS_ICPES	FS_ICPES	1.346	0.0250	29	1.412	0.032	0.0224	4.91%
Low_Li	FS_ICPES	4A_ICPES	0.289	0.0140	25	0.301	0.005	0.0181	4.56%

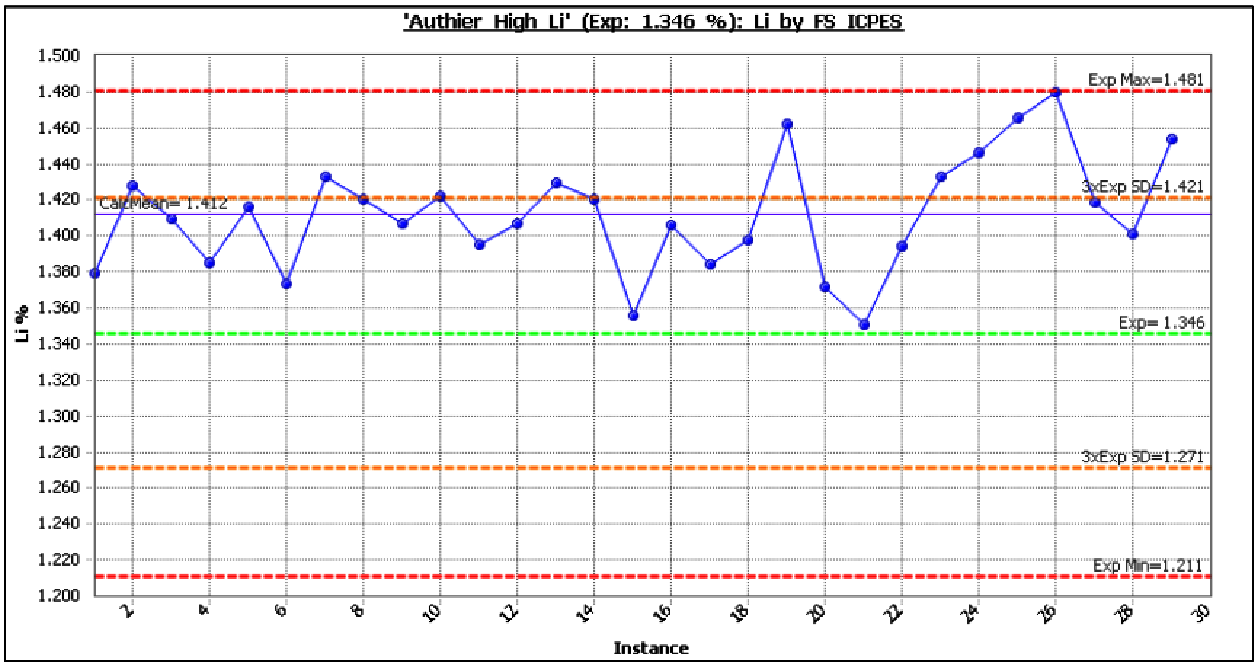


Figure 10-4: RM (STD High) Results

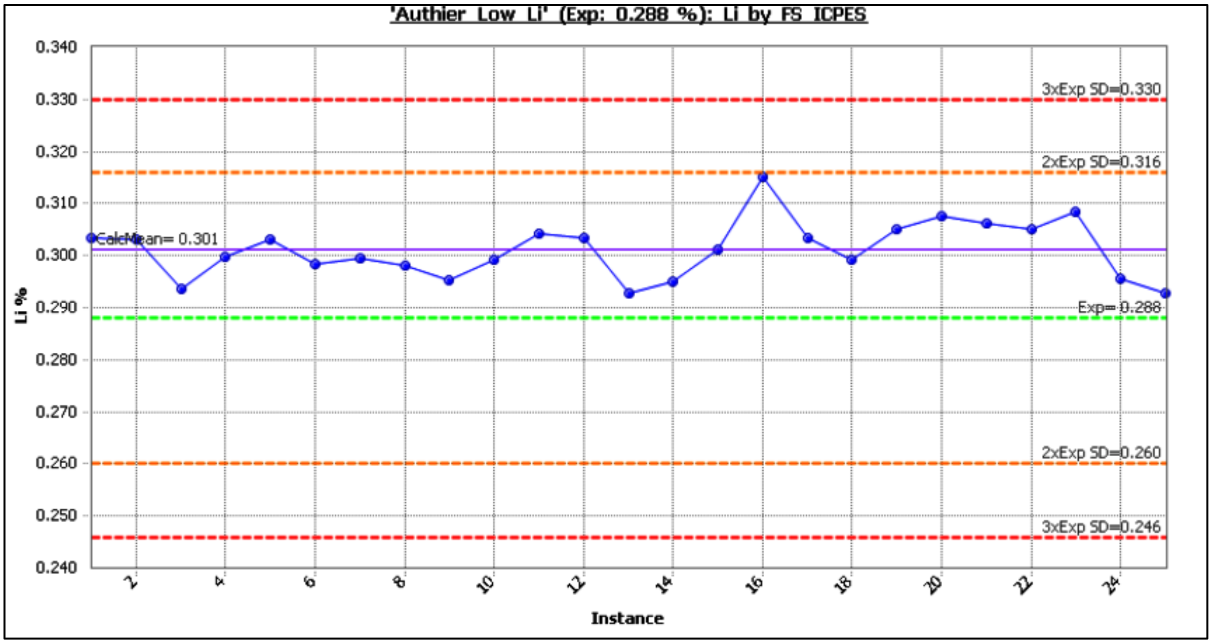


Figure 10-5: RM (STD Low) Results

10.9.1 Company Blank Material

Sayona Québec used one non-certified silica blank during the 2016 and 2017 drilling campaigns to test for potential sample contamination during sampling, preparation and analysis processes. The material was "Special Kitty Litter" purchased from Walmart and was stored in airtight plastic tubs to prevent contamination. Each sample consisted of approximately 200 g of material scooped with a dedicated mug into the plastic sample bags. The blanks were included at routine intervals during sampling at a rate of approximately 1:20 samples. The blanks were analyzed for lithium by SGS using the sodium peroxide fusion ICPOES (GE_ICP90A) method.

The expected value and standard deviation for the blank were set to 0.001% lithium, which is the detection limit for the analysis method. The control limits were set as ±3σ from the expected value.

The blank material performed well with all samples <0.003% and no outliers reported. The results for the 57 blank samples are summarized in the Table 10-3 and Figure 10-6.

Table 10-3: Blank Summary – Sayona Québec 2016

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp Value	Exp SD		Mean Li	SD	CV	Mean Bias
Blk_SpKi Litter	FS_ICPES	FS_ICPES	0.001	0.0010	57	0.000	0.001	0.0000	na

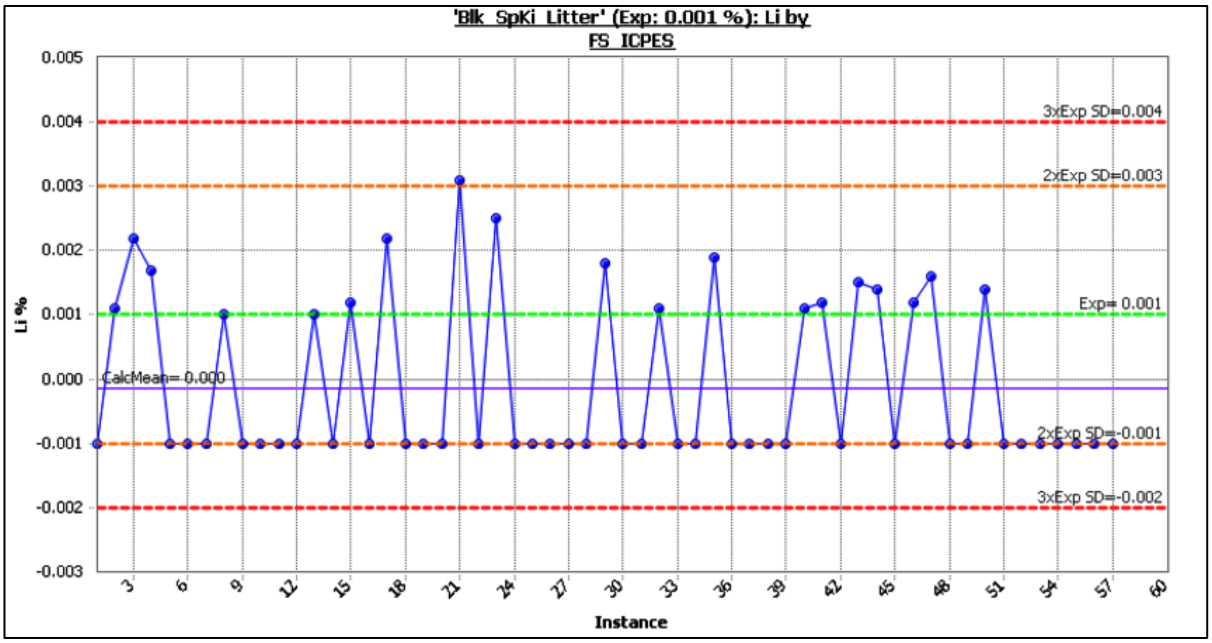


Figure 10-6: Blank Performance – Sayona Québec 2016

10.9.2 SGS Lakefield 2016 Reference Materials Results

In 2016, Sayona Québec sent samples to SGS in Lakefield, Ontario. SGS reported one high grade standard, one medium grade standard, two low grades standards and a blank during the drilling campaign. A total of 54 laboratory standards and 15 laboratory blanks were reported in the 18 SGS batches, with the results summarized in Table 10-4 and Figure 10-7. Orange lines in the figure represent ±2σ from the expected value and the red lines represent ±3σ from the expected value, except in the case of NBS183, which does not have an expected standard deviation. For this case, the red lines represent ±10% from expected value. No significant issues with the SGS Lakefield laboratory standards and blanks were detected.

Table 10-4: SGS Lakefield Laboratory Standards and Blank Summary – 2016

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp* Value	Exp* SD		Mean Li	SD	CV	Mean Bias
Blank	FS_ICPES	FS_ICPES	0.001	0.0010	15	-0.001	0.001	0.0000	na
NBS183	FS_ICPES	UN_UN	1.904	-	15	1.904	0.037	0.0195	0.01%
NIST97B	FS_ICPES	UN_UN	0.055	0.0010	12	0.054	0.001	0.0218	-1.62%
RTS-3A	FS_ICPES	UN_UN	0.001	0.0010	13	0.002	0.000	0.2076	na
SY-4	FS_ICPES	UN_UN	0.004	0.0002	14	0.004	0.000	0.0588	na

* Expected values for the Standard NIST97B were sourced from the “Certificate of Analysis” documents issued by National Bureau of Standards (now known as the National Institute of Standards and Technology). Expected values for the Standard NBS183 were sourced from the “Certificate of Analysis” documents issued by National Bureau of Standards. Expected values for the Standard SY-4 were sourced from the “Certificate of Analysis” documents issued by Natural Resources of Canada (NRCan). Expected values for the Standard RTS-3a were sourced from the “Certificate of Analysis” documents issued by Natural Resources of Canada.

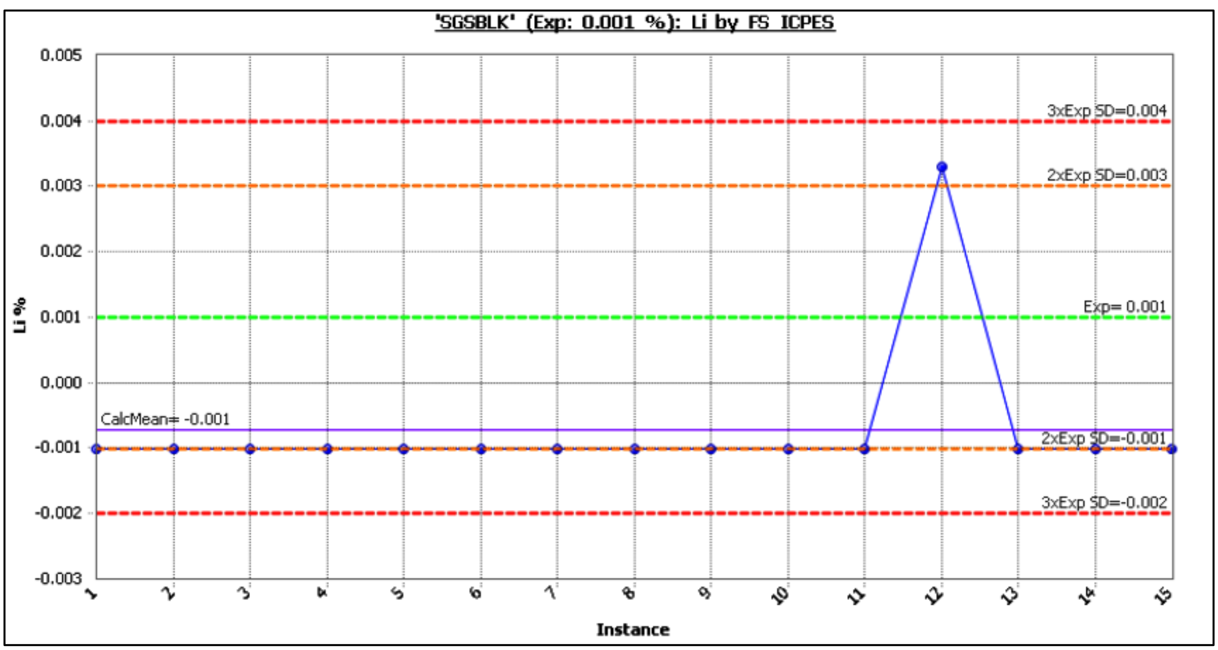


Figure 10-7: SGS Blank Performance – 2016

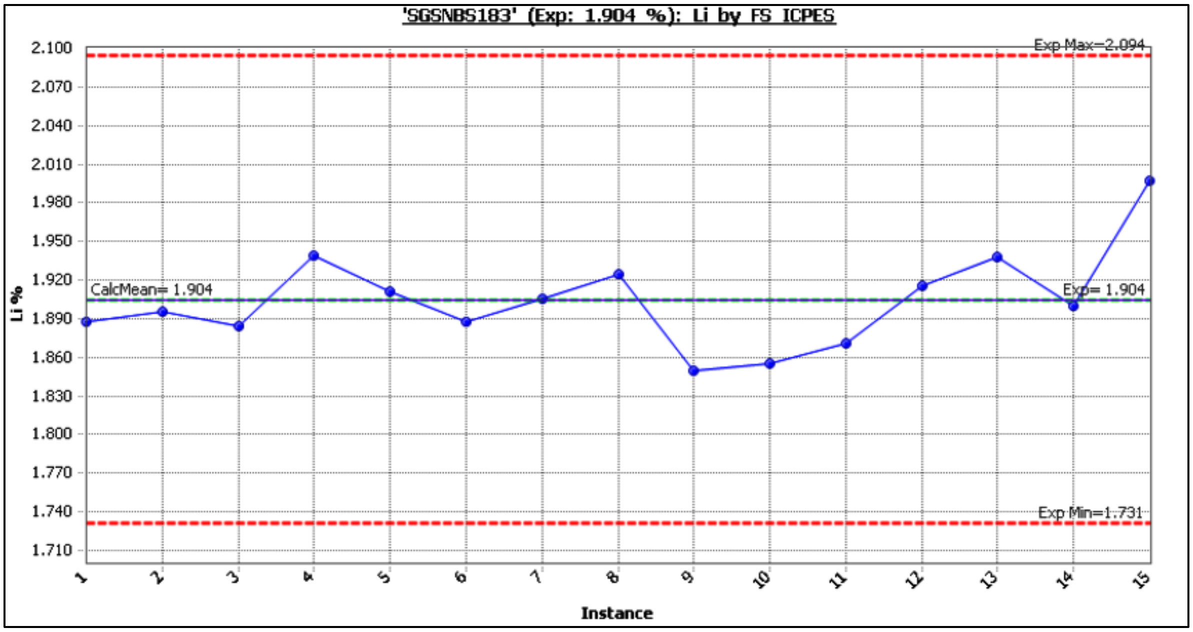


Figure 10-8: SGS Standard NBS183 Performance – 2016

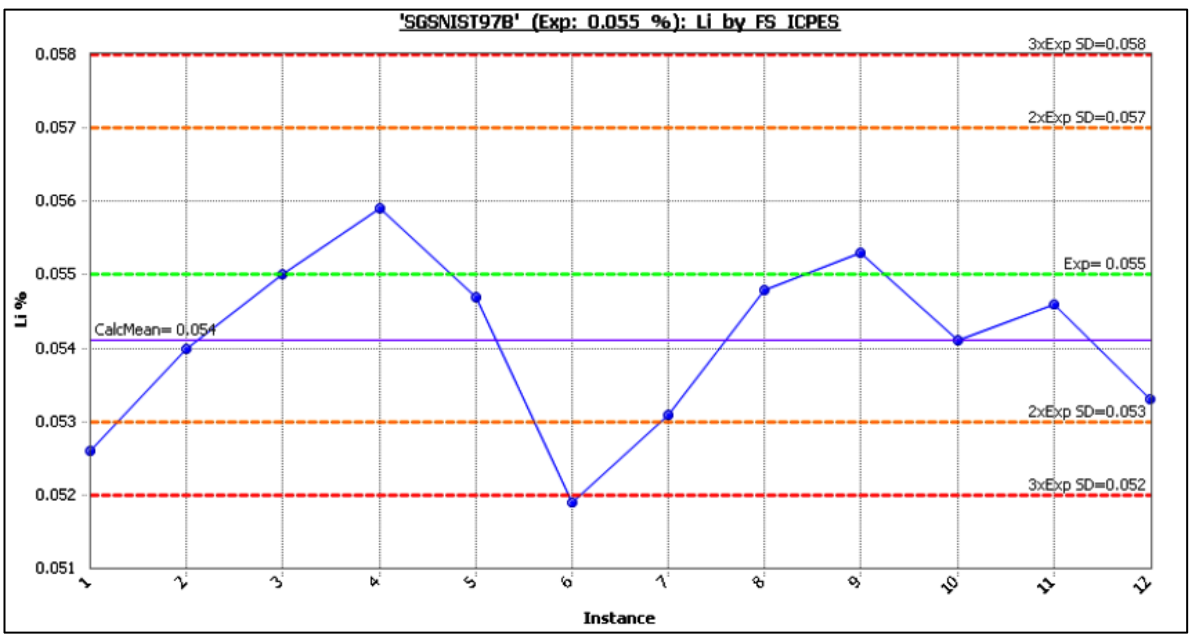


Figure 10-9: SGS Standard NIST97B Performance – 2016

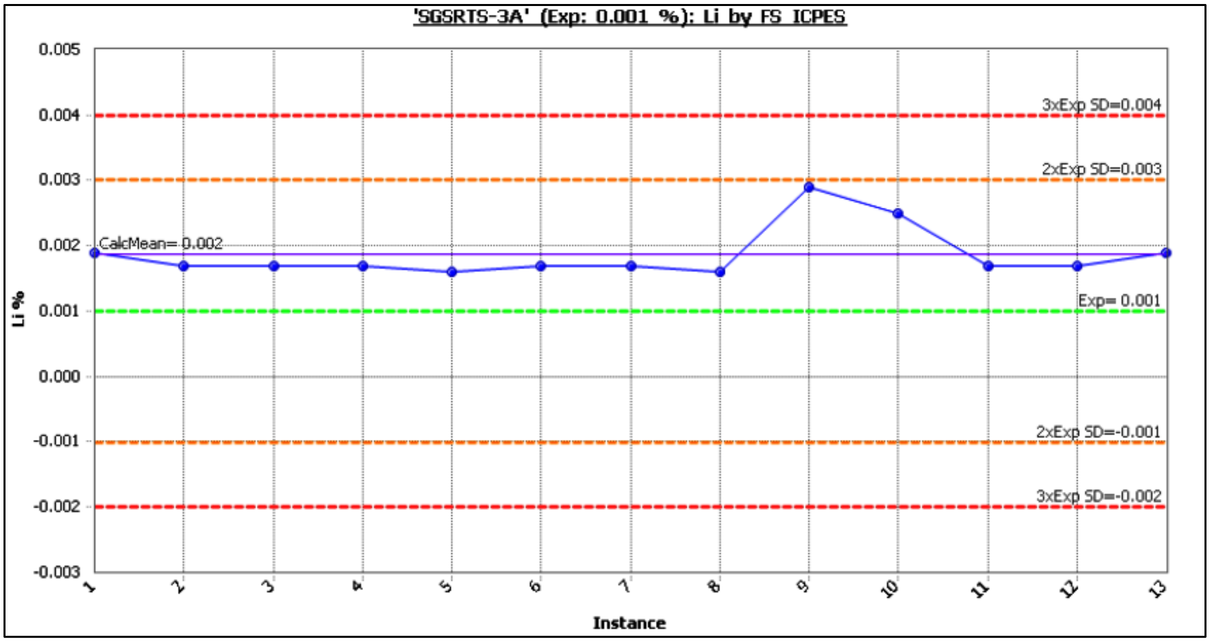


Figure 10-10: SGS Standard RTS-3A Performance – 2016

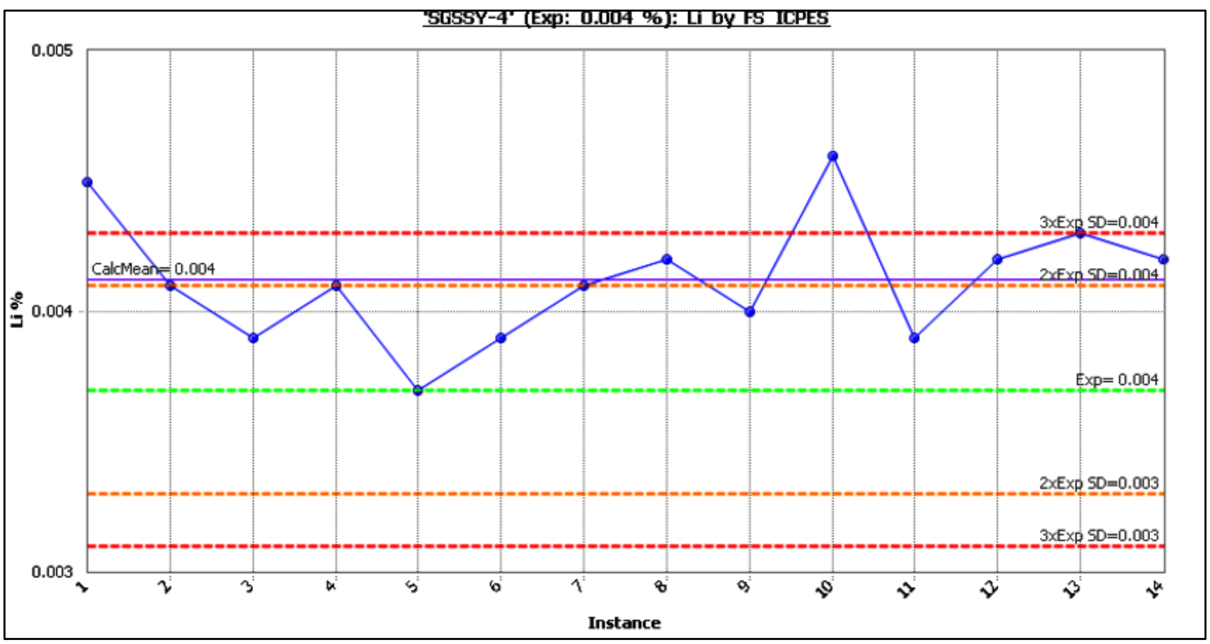


Figure 10-11: SGS Standard SY-4 Performance – 2016

10.9.3 Sayona Québec Duplicates 2016

Sayona Québec did not collect duplicate samples during the 2016 drill campaign. The SGS Lakefield laboratory reported two types of laboratory duplicates in their batches, a coarse duplicate and a pulp repeat, the results of which are included in Section 10.9.4 and 10.9.5.

10.9.4 SGS Lakefield 2016 Coarse Duplicates

Results for the thirty-three laboratory pulp repeat samples are summarized in Table 10-6 and Figure 10-13. Statistical analysis demonstrates good repeatability of the SGS GE_ICP90A procedure.

Table 10-5: SGS Lakefield Laboratory Duplicate Summary Statistics – 2016

No. of Samples	mean Li1	mean Li2	SD Li1	SD Li2	CV Li1	CV Li2	sRPHD (mean)
33	0.39	0.38	0.32	0.32	0.83	0.83	0.99

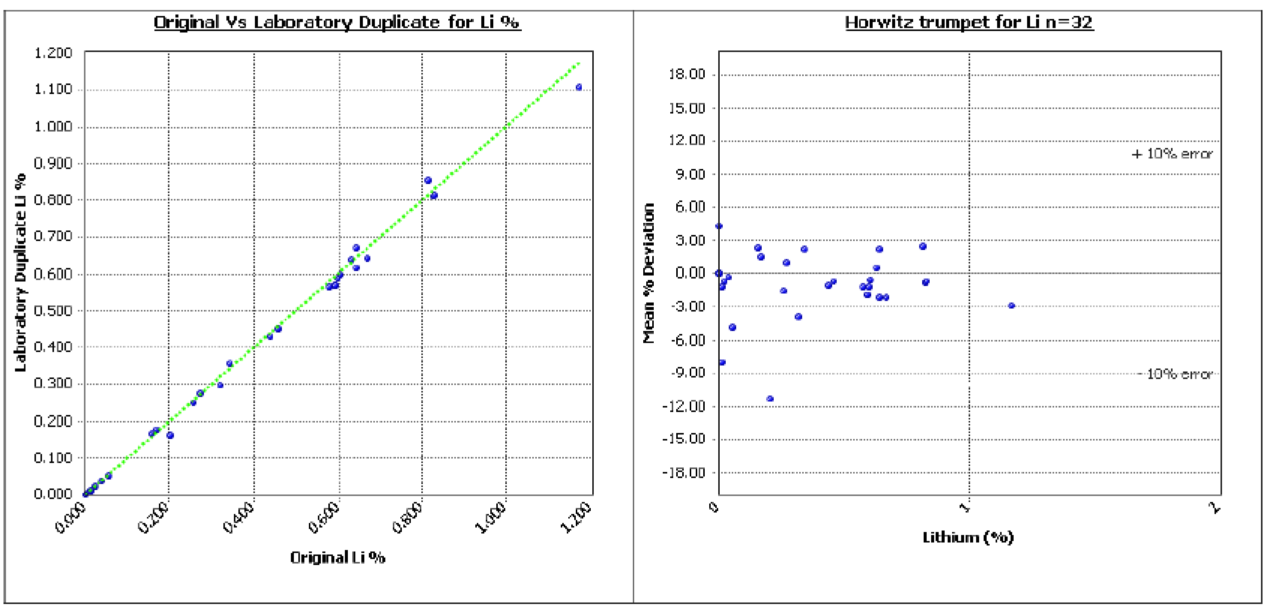


Figure 10-12: SGS Laboratory Duplicates Correlation Plot and Horwitz Trumpet Chart – 2016

10.9.5 SGS Lakefield 2016 Pulp Repeat

Results for the 33 laboratory pulp repeat samples are summarized in Table 10-6 and Table 10-13. Statistical analysis demonstrates good repeatability of the SGS GE_ICP90A procedure.

Table 10-6: SGS Lakefield Laboratory Repeats Summary Statistics – 2016

No. of Samples	mean Li1	mean Li2	SD Li1	SD Li2	CV Li1	CV Li2	sRPHD (mean)
33	0.41	0.41	0.34	0.34	0.82	0.82	-0.36

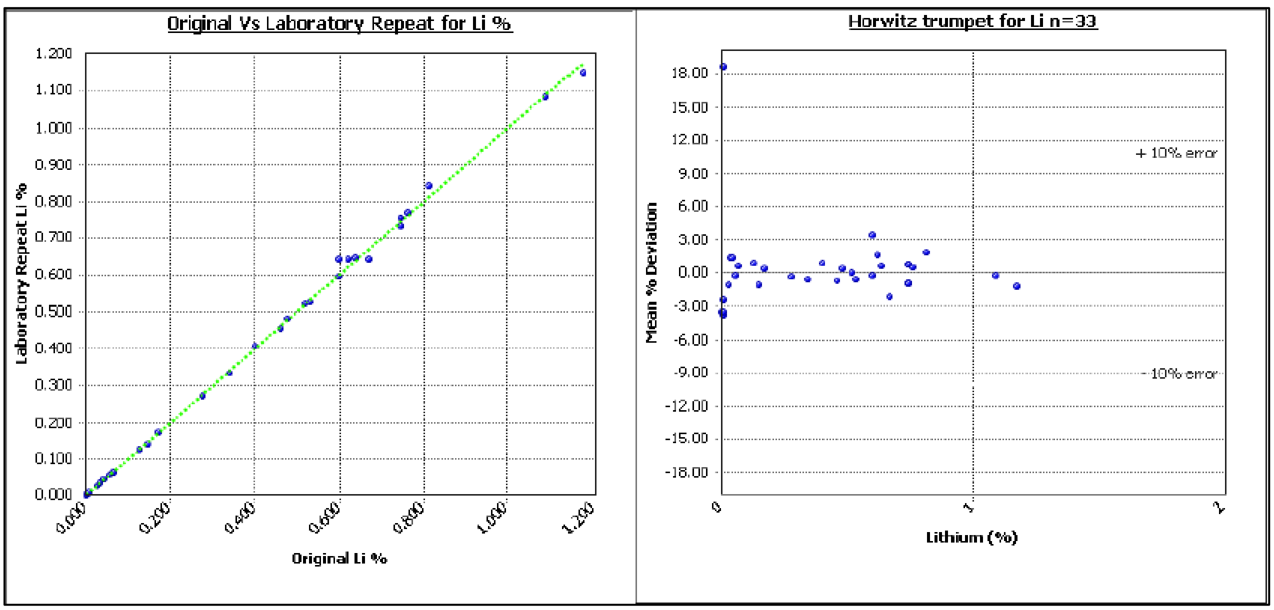


Figure 10-13: SGS Laboratory Repeats Correlation Plot and Horwitz Trumpet Chart – 2016

10.10 2017 Reference Materials Results

The 2017 QA/QC follow-up was conducted by Rock Solid Data Consultancy Pty., mandated by Sayona Québec, which included performance of reference material of both Sayona Québec and SGS.

In 2017, Sayona Québec included the two standards at random intervals during sampling at a rate of approximately 1:20 samples, the same procedure as 2016. The Sayona Québec blank material, the SGS blank and low grade SGS laboratory standard (RTS-3A) performed well with all samples within control limits.

The two Sayona Québec standards, High-Li and Low-Li, and SGS laboratory standards, NBS183, NIST97B and SY-4, exhibited a bias shift in the results reported during April 2017 compared to the results reported in March 2017. All results for laboratory standard NBS183, reported during April 2017, fell below 3σ from the expected value, which is in contrast to the results for March 2017 and for the 2016 drilling campaign, where all results reported within ±3σ from the expected value. The apparent bias could be due to laboratory calibration error and will be controlled by Sayona Québec through pulp check assaying in both the same lab and another lab during the Phase 3 drilling program.

In the charts that follow, the orange lines represent the ±3σ from the expected value and the red lines represent ±10% from expected value. The results for the 17 High-Li and 19 Low-Li samples are summarized in Table 10-7.

Table 10-7: Results from custom Low-Li and High-Li Standards – Sayona Québec 2017

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp Value	Exp SD		Mean Li	SD	CV	Mean Bias
High_Li	FS_ICPES	UN_UN	1.346	0.0250	17	1.360	0.051	0.0377	1.05%
Low_Li	FS_ICPES	UN_UN	0.288	0.0140	19	0.289	0.010	0.0350	0.29%

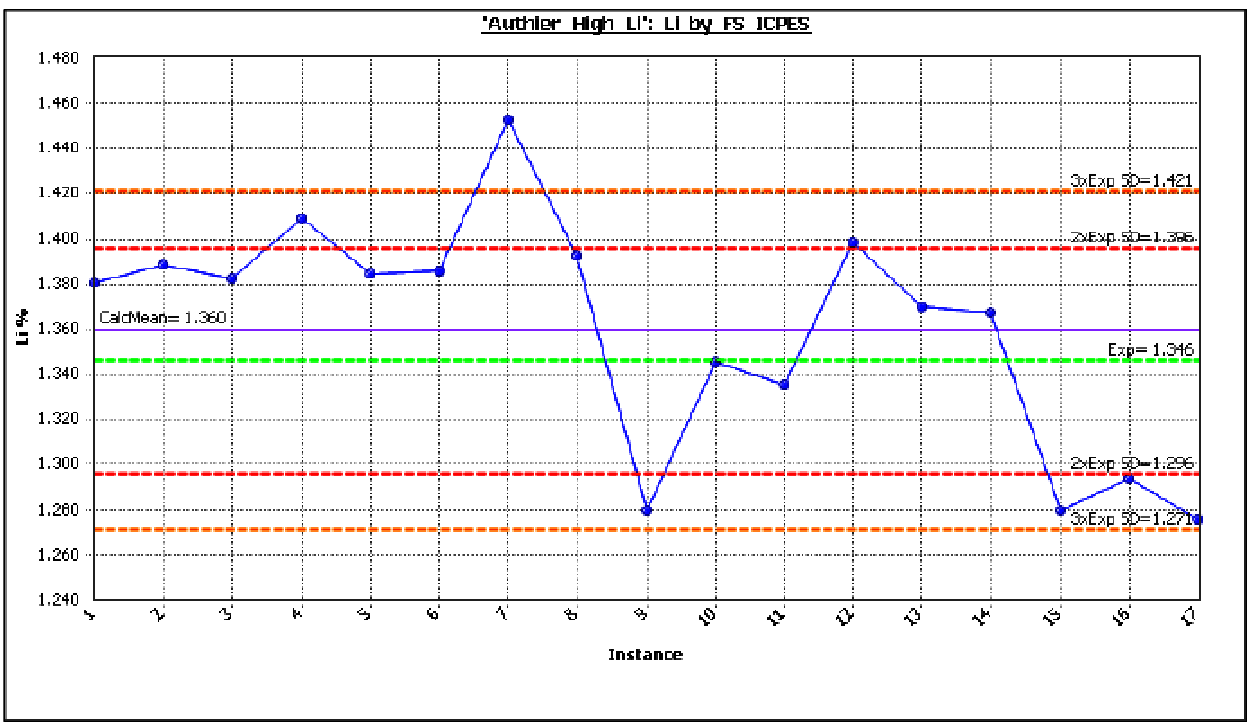


Figure 10-14: RM (STD High) Results

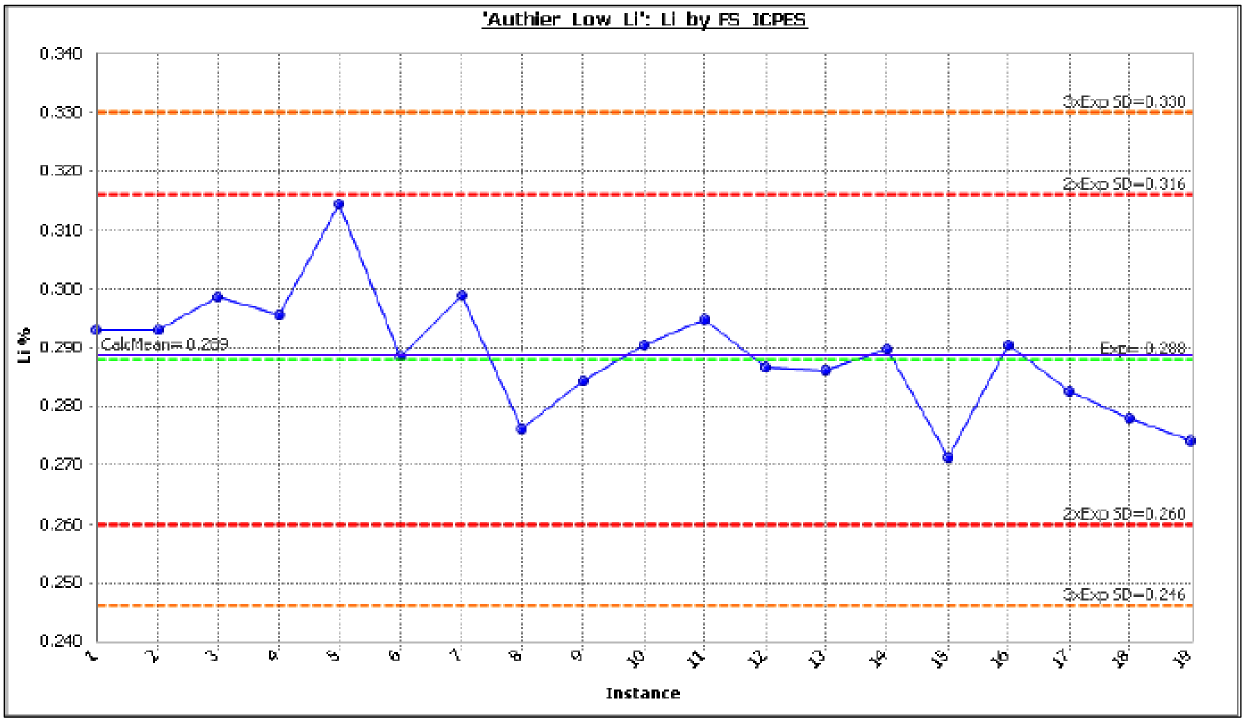


Figure 10-15: RM (STD Low) Results

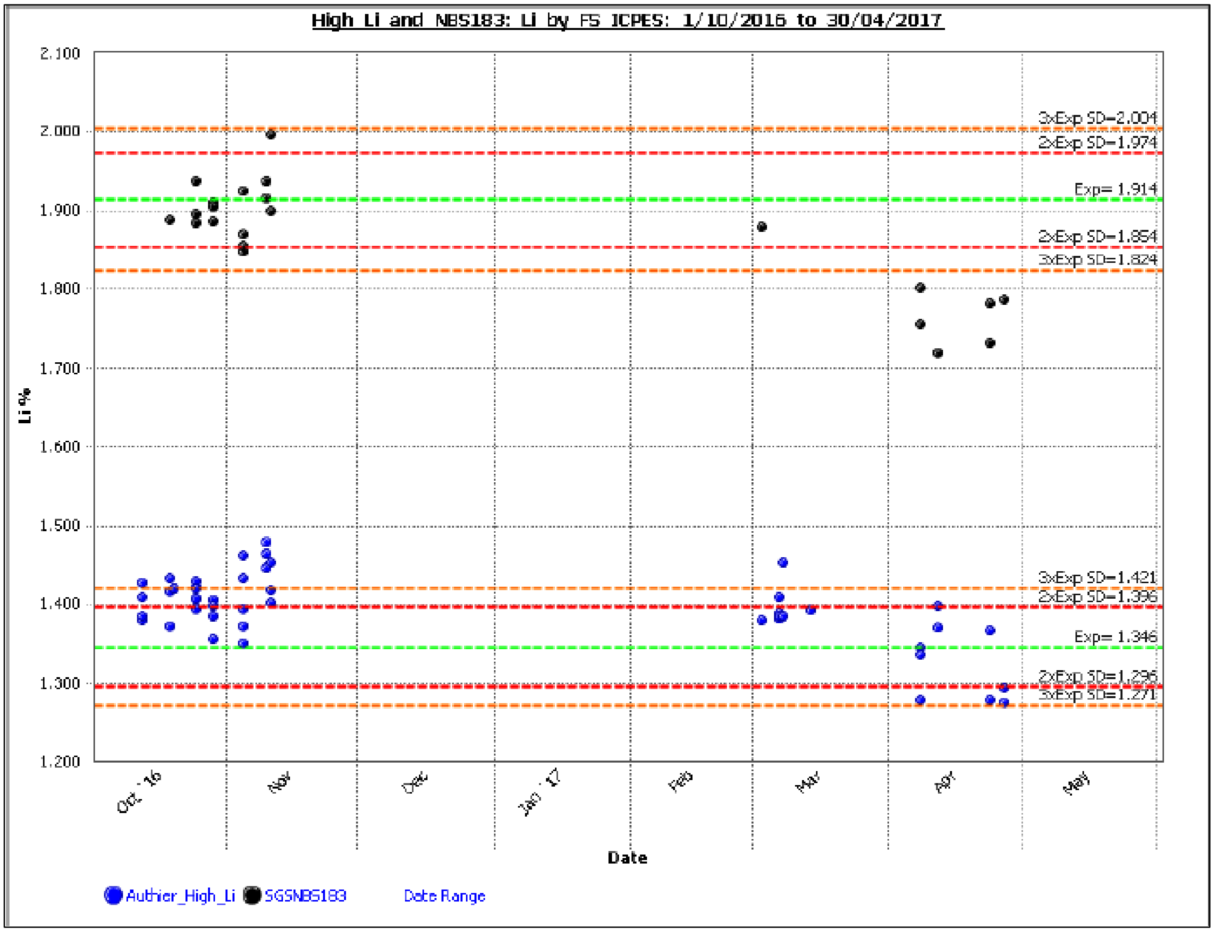


Figure 10-16: Authier High_Li and SGS NBS183 Performance 2016-2017

10.10.1 Company Blank Material

Sayona Québec utilized one non-certified silica blank during the 2016 and 2017 drilling campaign to test for potential sample contamination during sampling, preparation and analysis processes. The material was "Special Kitty Litter" purchased from Walmart and was stored in airtight plastic tubs to prevent contamination. Each sample consisted of approximately 200 g of the material scooped with a dedicated mug into the plastic sample bags.

The blanks were included at routine intervals during sampling at a rate of approximately 1:20 samples. The blanks were analyzed for Li by SGS sodium peroxide fusion ICPOES (GE_ICP90A).

The expected value and standard deviation for the blank were set to 0.001% lithium, which is the detection limit for the analysis method. The control limits are set as ±3σ from the expected value.

The blank material performed well with all samples <0.003% and no outliers reported. The results for the 44 blank samples are summarized in Table 10-8 and Figure 10-17. Orange lines in the figure represent the ±3σ from the expected value and the red lines represent ±2σ from the expected value.

Table 10-8: Blank Summary – Sayona Québec 2017

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp Value	Exp SD		Mean Li	SD	CV	Mean Bias
Blank	FS_ICPES	FS_ICPES	0.000	0.001	44	-0.001	0.001	0.0000	0.00%

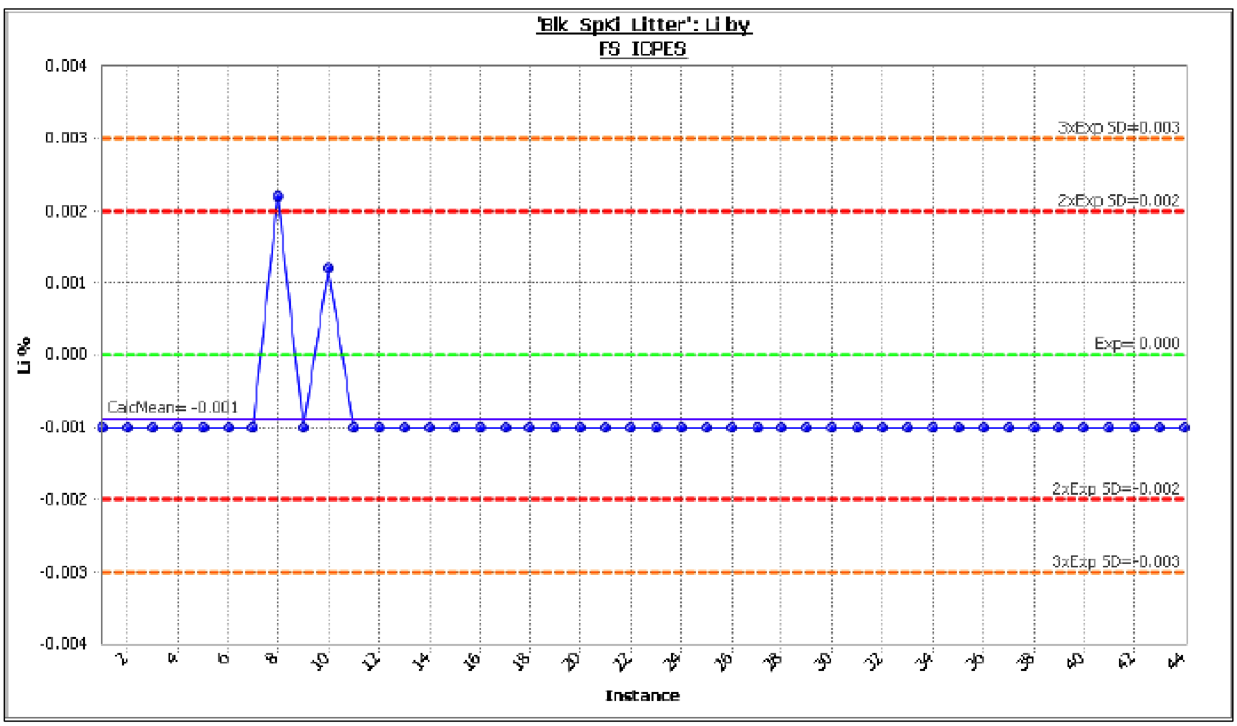


Figure 10-17: Blank Performance – Sayona Québec 2017

10.10.2 SGS Lakefield 2017 Reference Materials Results

In 2017, Sayona Québec sent samples to SGS Lakefield. SGS reported one high grade standard, one medium grade standard, two low-grade standards and 13 laboratory blanks during the drilling campaign, with the results summarized in Table 10-9 and Figure 10-18 to Figure 10-22.

The results for NBS183, NIST97B and SY4 exhibit a low bias in the results reported during April 2017. All results for NBS183 reported during April 2017 fall below three standard deviations of the expected value, which is in contrast to the results for March 2017 and for the 2016 Phase 1 drilling campaign, where all results reported within ±3σ from the expected value. In the figures that follow, the orange lines represent the ±2σ from the expected value and the red lines represent ±3σ from the expected value. No significant issues with the SGS Lakefield laboratory standards and blanks were detected.

Table 10-9: SGS Lakefield Laboratory Standards and Blank Summary – 2016

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp* Value	Exp* SD		Mean Li	SD	CV	Mean Bias
Blank	FS_ICPES	FS_ICPES	0.000	0.0010	13	-0.001	0.000	0.0000	0.00%
NBS183	FS_ICPES	UN_UN	1.914	0.0300	7	1.780	0.053	0.0296	-6.98%
NIST97B	FS_ICPES	UN_UN	0.055	0.0010	11	0.054	0.003	0.0595	-2.30%
RTS-3A	FS_ICPES	UN_UN	0.001	0.0010	12	0.002	0.000	0.0732	-
SY-4	FS_ICPES	UN_UN	0.004	0.0002	12	0.004	0.000	0.0463	-

*Expected values for the Standard NIST97B were sourced from the "Certificate of Analysis" documents issued by National Bureau of Standards (now known as the National Institute of Standards and Technology). Expected values for the Standard NBS183 were sourced from the "Certificate of Analysis" documents issued by National Bureau of Standards. Expected values for the Standard SY-4 were sourced from the "Certificate of Analysis" documents issued by Natural Resources of Canada (NRCan). Expected values for the Standard RTS-3a were sourced from the "Certificate of Analysis" documents issued by Natural Resources of Canada.

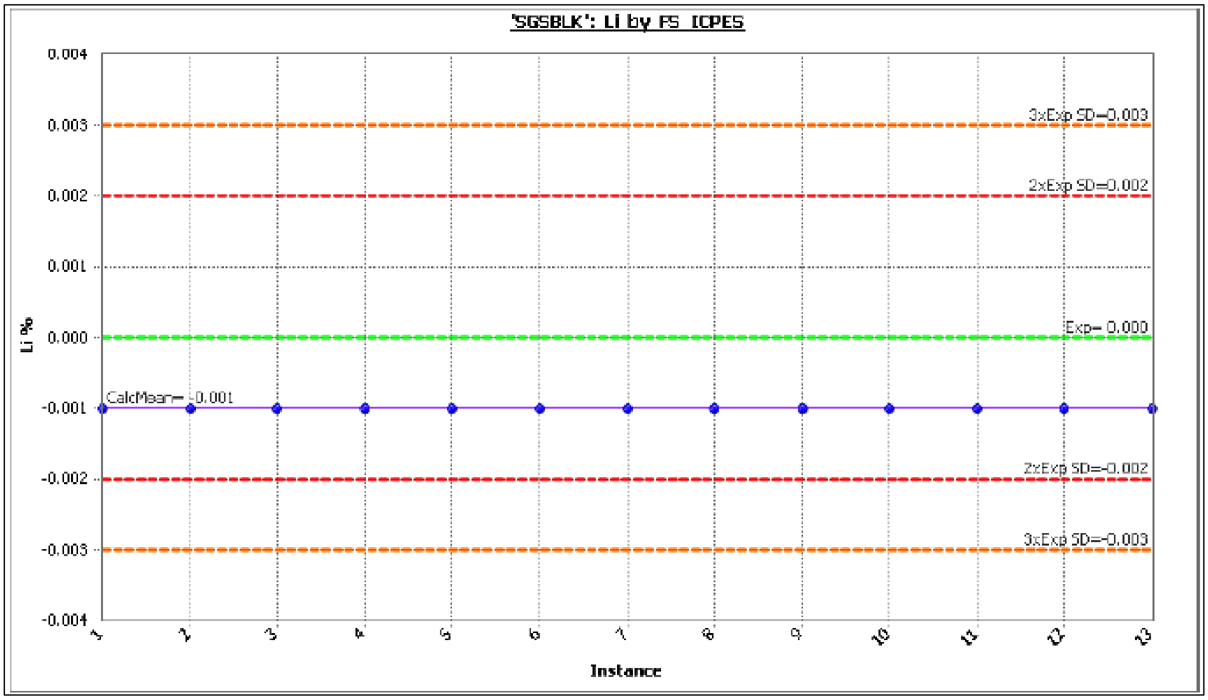


Figure 10-18: SGS Blank Performance – 2017

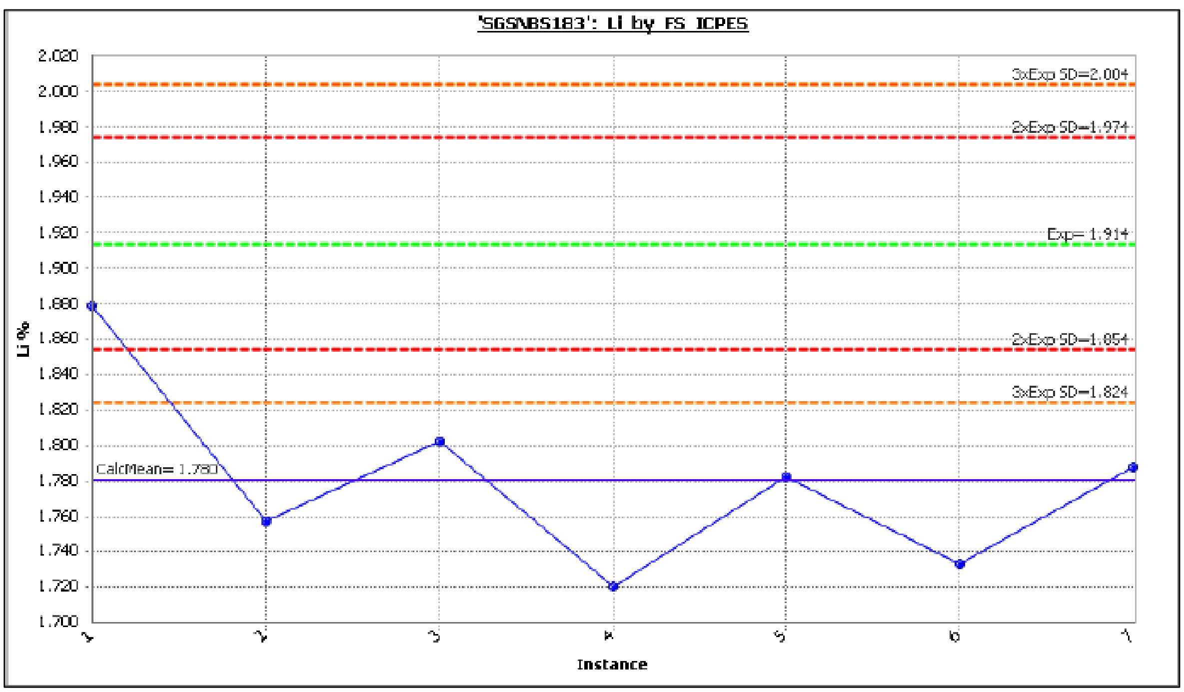


Figure 10-19: SGS Standard NBS183 Performance – 2017

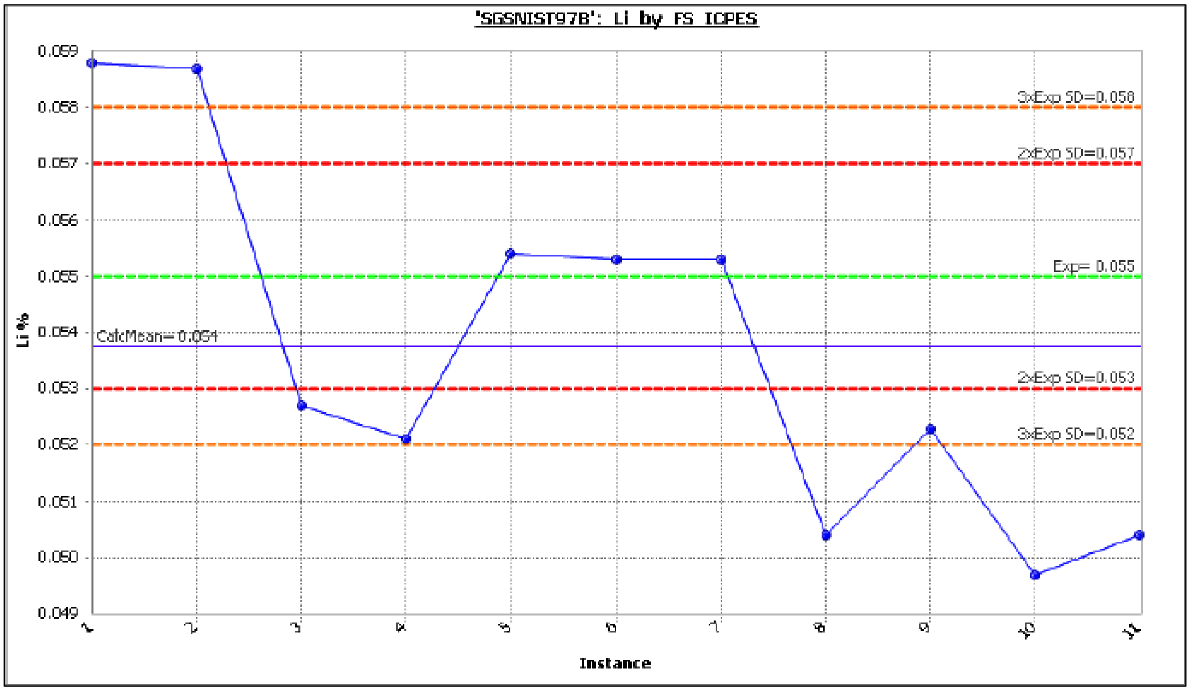


Figure 10-20: SGS Standard NIST97B Performance – 2017
(Points 1-3 were reported during March 2017 and Points 4-11 were reported during April 2017)

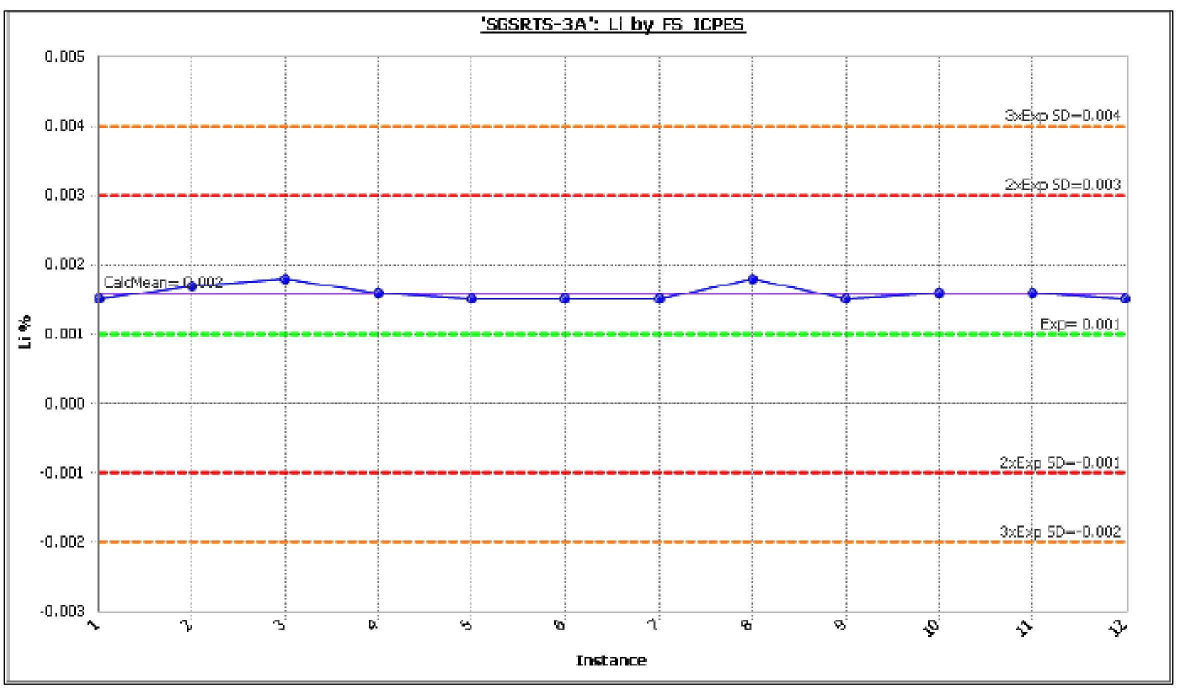


Figure 10-21: SGS Standard RTS-3A Performance – 2017
(Points 1-5 were reported during March 2017 and Points 6-12 were reported during April 2017)

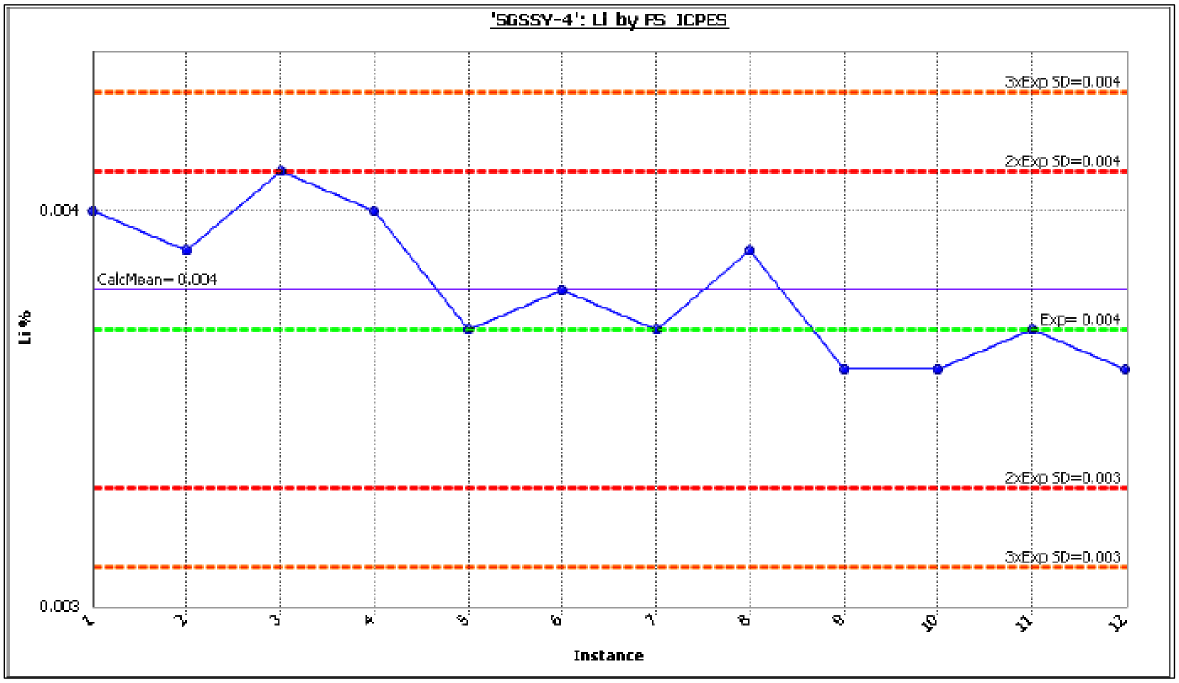


Figure 10-22: SGS Standard SY4 Performance – 2017
(Points 1-5 were reported during March 2017 and Points 6-12 were reported during April 2017)

10.10.3 Sayona Québec Duplicates 2017

Sayona Québec did not collect duplicate samples during the 2017 drill campaign. The SGS Lakefield laboratory reported two types of laboratory duplicates in their batches, a coarse duplicate and a pulp repeat, the results of which are included in sections below.

10.10.4 SGS Lakefield 2017 Coarse Duplicates

Results for the 18 laboratory coarse duplicate samples are summarized in Table 10-10 and Figure 10-23. Statistical analysis demonstrates good repeatability of the SGS laboratory preparation and GE_ICP90A analysis procedure.

Table 10-10: Laboratory Duplicate Summary Statistics – 2017

No. of Samples	mean Li1	mean Li2	SD Li1	SD Li2	CV Li1	CV Li2	sRPHD (mean)
18	1604.28	1583.00	2243.50	2215.04	1.40	1.40	0.08

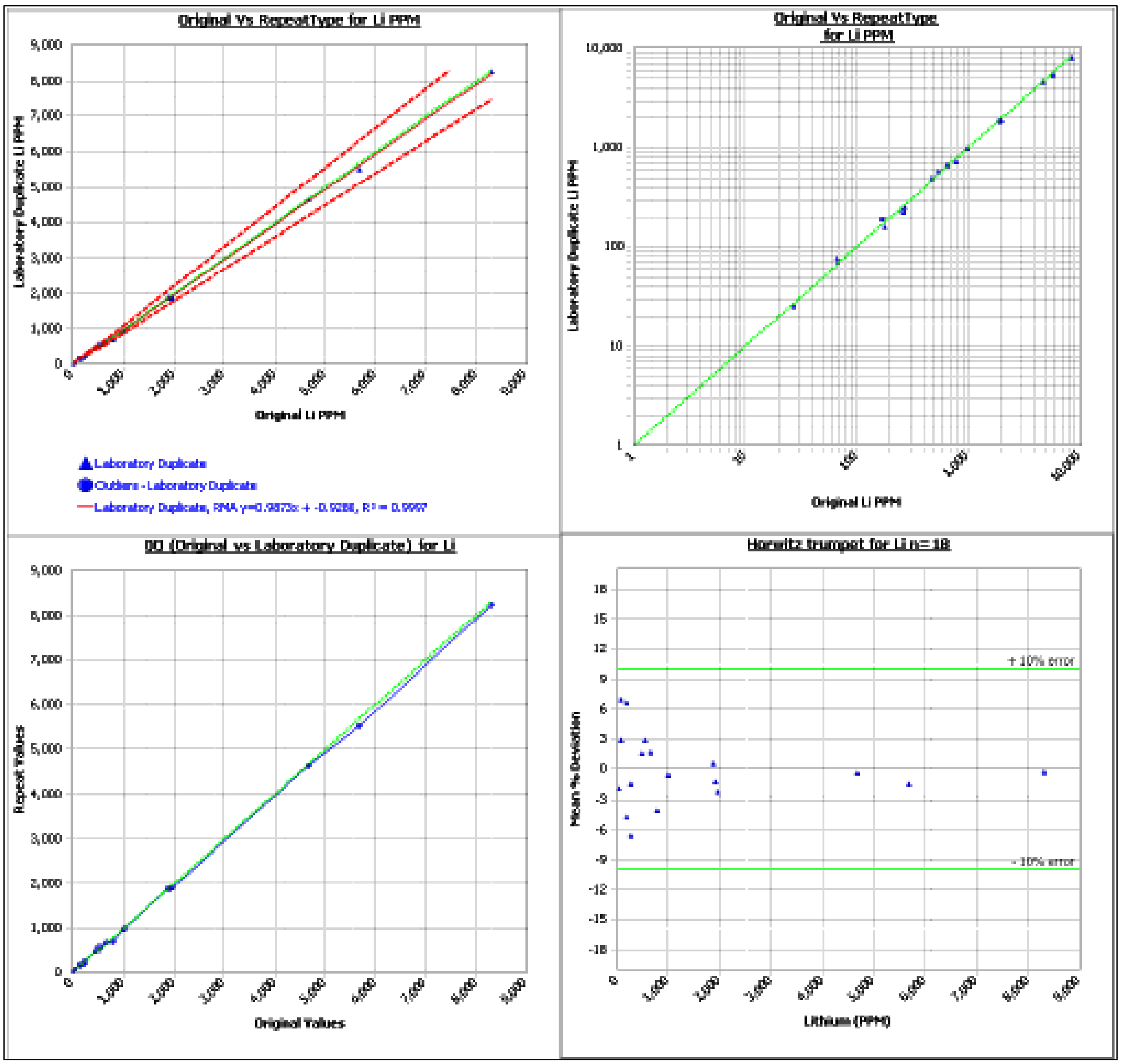


Figure 10-23: Laboratory Duplicates Correlation Plot, QQ Plot and Horwitz Trumpet Chart – 2017

10.10.5 SGS Lakefield 2017 Pulp Repeat

Results for the 22 laboratory pulp repeat samples are summarized in Table 10-11 and Figure 10-24. Statistical analysis demonstrates good repeatability of the SGS GE_ICP90A analysis procedure.

Table 10-11: Laboratory Repeat Summary Statistics – 2017

No. of Samples	mean Li1	mean Li2	SD Li1	SD Li2	CV Li1	CV Li2	sRPHD (mean)
22	1483.55	1494.86	1961.45	1979.31	1.32	1.32	0.87

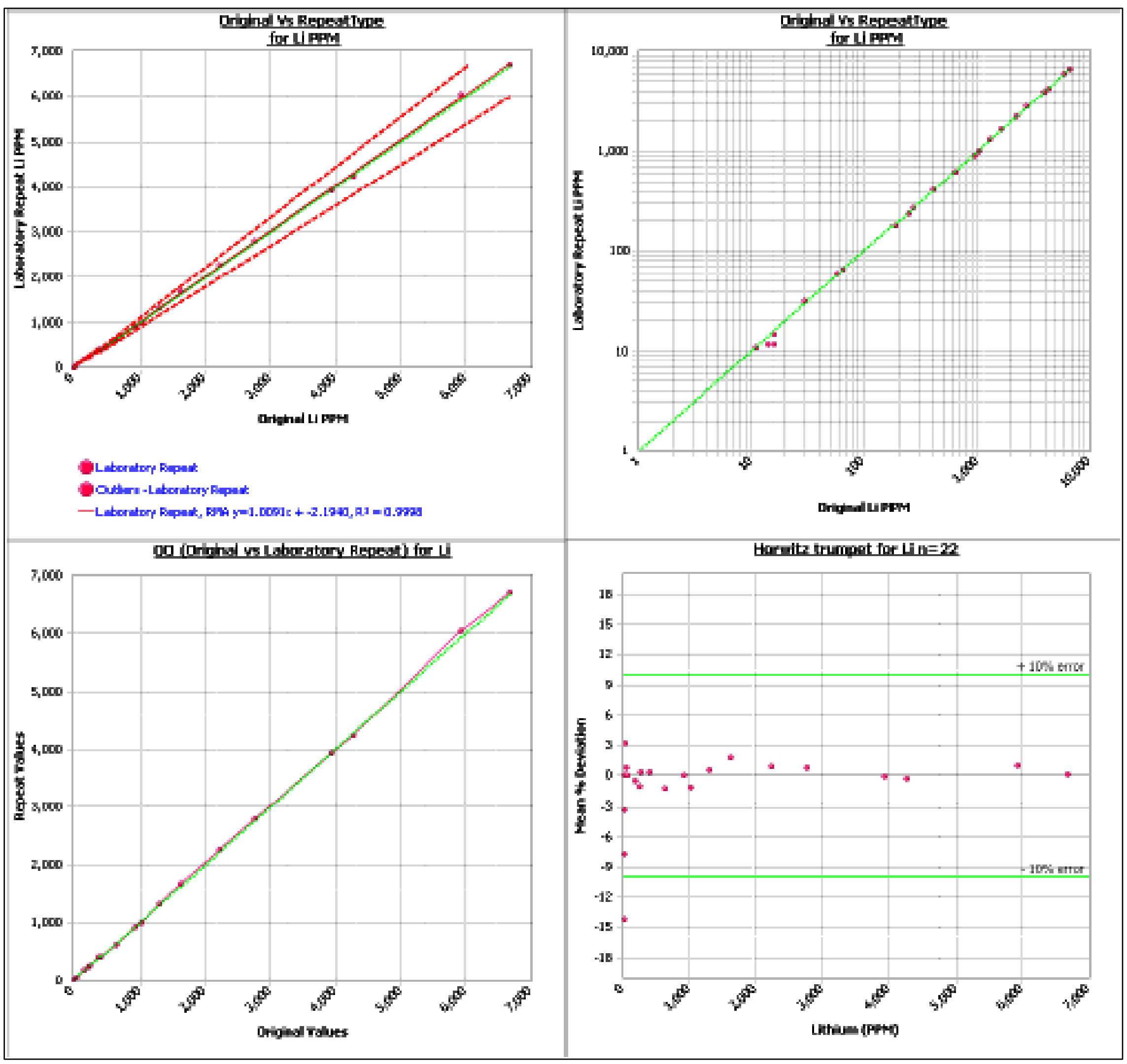


Figure 10-24: Laboratory Repeats Correlation plot, QQ Plot and Horwitz Trumpet Chart – 2017

10.11 Sayona Québec 2018 Reference Materials Results

The 2018 QA/QC follow-up was conducted by Rock Solid Data Consultancy Pty., mandated by Sayona Québec. The report included performance of reference materials for both Sayona Québec and SGS. The sampling data was managed by Rock Solid Data and stored in a custom-relational SQL database.

The report is based on quality control data associated with 2,154 m of NQ diamond drilling (DD) from 19 drill holes (AL-18-001 to AL-18-019). A total of 364 half-core samples were collected from mineralized intersections between January and March 2018. Available quality control data include two company standards and one company blank as well as laboratory duplicates, blanks and standards. Sayona Québec did not collect duplicate samples during the report period.

The drill and quality control samples were submitted to SGS Lakefield, where they were analyzed for lithium and 27 additional elements by sodium peroxide fusion ICP-OES with HCl finish (GE_ICP91A). The lower detection limit for lithium is 0.001%. SGS reported a total of seven batches, listed in Appendix II, between 22 February and 27 March 2018. The lithium analyses are the subject of this report and values are reported in percent.

The amount of drill samples, duplicates and standards reported during the sampling program are summarized in Table 10-12. Approximately 8% of all samples submitted to SGS are Sayona Québec standards and blanks. Laboratory standards and laboratory duplicates represent approximately 11% of the reported samples.

Table 10-12: Authier 2018 SGS Lakefield Batch Summary Statistics

Number of Batches	Drill Samples	Drill Duplicates	Company Standards	Company Blanks	Laboratory Duplicates	Laboratory Standards and Blanks
7	364	0	13	20	20	28

During the Authier 2018 drilling campaign, Sayona Québec used two company standards and one blank to monitor the accuracy of the laboratory assay results. The company standards were a high grade High-Li (approx. 1.4%) and a low-grade Low-Li (approx. 0.3%) lithium standard.

The High-Li and Low-Li standards were custom made from mineralized material from the main pegmatite intrusion at the Authier site and were used by Glen Eagle during their 2010-2012 drilling campaigns. The expected value and standard deviation for the standards were derived by Glen Eagle from 30 Lithium (Li_2O) analyses from SGS Toronto and ALS Vancouver. The 15 SGS Toronto analyses were by sodium peroxide fusion with ICP-OES finish (SGS code ICP90Q) and the 15 ALS Vancouver analyses were by ore grade lithium four acid digestion with ICPAES finish (ALS code Li-OG63). The control limits were set as ±3σ from the expected value. For further details regarding the two standards, refer to document "March-01-2013_PEA_Glen-Eagle_rev_March-11.pdf". The two standards were included at routine intervals during sampling at a rate of approximately 1:30 samples. The standards were analyzed for lithium by sodium peroxide fusion ICP-OES with HCl finish (GE_ICP91A).

Sayona Québec used one non-certified blank, logged as Blk_Spki_Litter to test for potential sample contamination during sampling, preparation and analysis processes. The Blank, Blk_SpKi_Litter was sourced from Walmart under the name "Special Kitty" Natural Clay Cat Litter. It was stored post purchase in airtight plastic tubs to prevent contamination.

The blanks were included at routine intervals during sampling at a rate of approximately 1:20 samples. Each sample consisted of approximately 200 g of the material scooped with a dedicated mug into the plastic sample bags. The blanks were analyzed for lithium by sodium peroxide fusion ICP-OES with HCl finish (GE_ICP91A).

10.11.1 Sayona Québec 2018 Standards Results

The lithium results for the company standards are summarized in Table 10-13, Figure 10-25 and Figure 10-26. A total of 13 standards were analyzed. All results for High-Li were within ±3σ from the expected value and all results for Low-Li were within ±2σ from the expected value.

Table 10-13: Sayona Québec Standard Reference Material Summary

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp Value	Exp SD		Mean Li	SD	CV	Mean Bias
High_Li	GE_ICP91A	-	1.346	0.0250	6	1.366	0.023	0.0017	1.50%
Low_Li	GE_ICP91A	-	0.288	0.0140	7	0.294	0.008	0.0027	2.25%

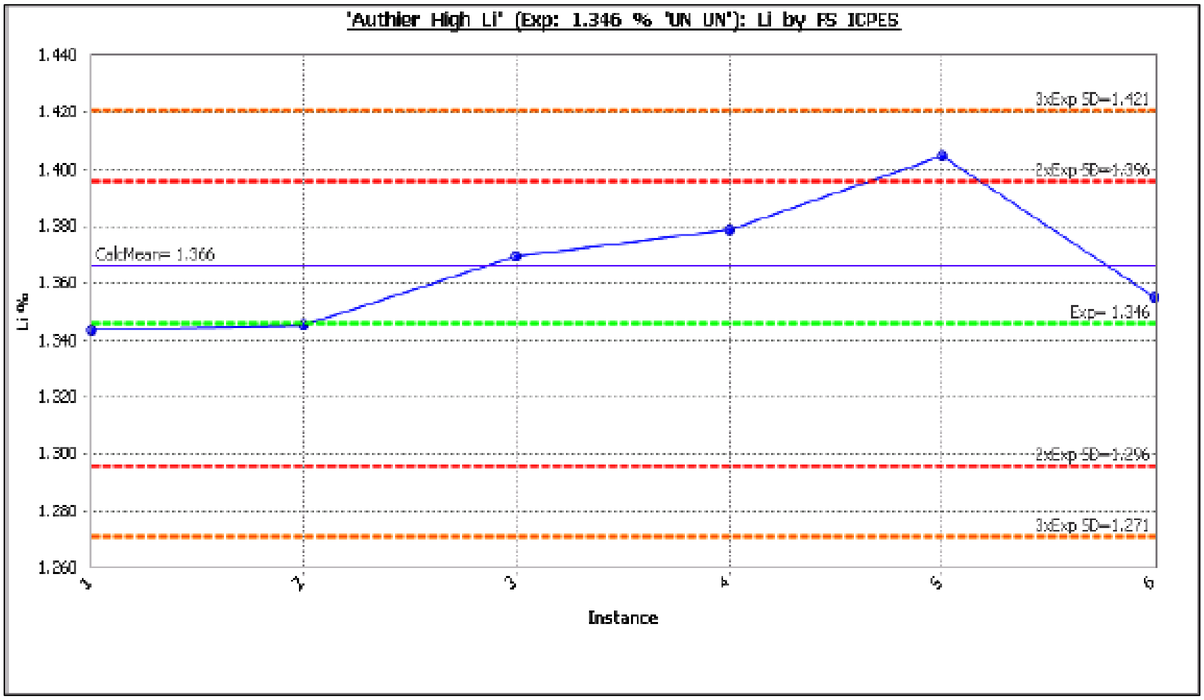


Figure 10-25: Authier High-Li Performance

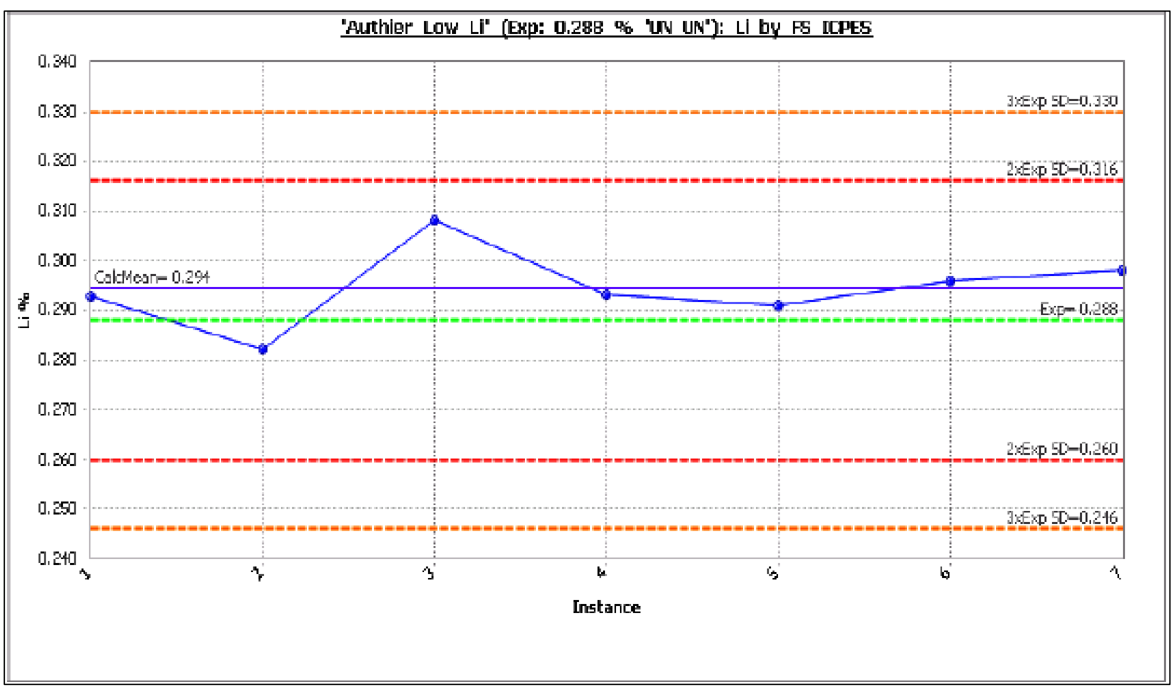


Figure 10-26: Authier Low-Li Performance

10.11.2 Sayona Québec 2018 Blank Results

During the report period a total of 20 blank samples were analyzed. Results for the blanks are summarized in Table 10-14 and Figure 10-27.

Table 10-14: Sayona Québec Blank Summary

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp Value	Exp SD		Mean Li	SD	CV	Mean Bias
Blank	GE_ICP91A	-	-	-	20	0.004	0.001	0.161	-

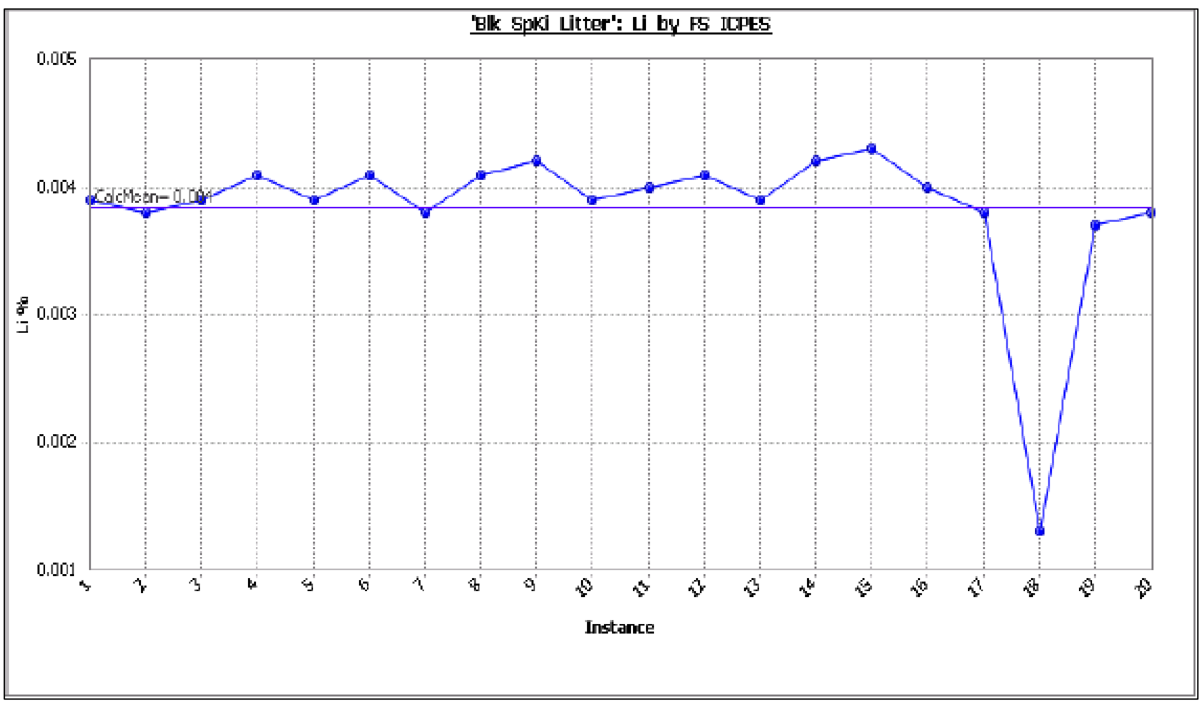


Figure 10-27: Sayona Québec Blank Performance

10.11.3 SGS Lakefield 2018 Reference Materials Results

During the report period, SGS reported lithium results for 21 laboratory standards and seven laboratory blanks. The standards and blanks were analyzed for Li by sodium peroxide fusion ICP-OES with HCl finish (GE_ICP91A).

The expected values for the standards were sourced from the certified reference material certificates. Expected values for OREAS standards were sourced from Ore Research and Exploration certificates. Expected values for the standard NIST97B were sourced from the Certificate of Analysis documents issued by National Bureau of Standards, now known as the National Institute of Standards and Technology. Expected values for the standard SY-4 were sourced from the Certificate of Analysis documents issued by Natural Resources of Canada.

The analytical performance of the standards and blanks is demonstrated in Table 10-15 and Figure 10-28 to Figure 10-33. Outliers are recorded in Table 10-16 and Table 10-17.

Table 10-15: Laboratory Standards and Blank Summary 2018

Li Standard(s)					No. of Samples	Calculated Values			
Standard	Method	Exp Method	Exp* Value	Exp* SD		Mean Li	SD	CV	Mean Bias
Blank	GE_ICP91A	GE_ICP91A	0.000	0.0010	7	-0.001	0.001	0.000	0.00%
NIST97B	GE_ICP91A	Unspecified	0.055	0.0010	7	0.053	0.001	0.021	-3.66%
OREAS-147	GE_ICP91A	Fusion ICP	0.227	0.0110	4	0.227	0.002	0.010	-0.03%
OREAS-147	GE_ICP91A	Fusion ICP	0.476	0.0110	2	0.483	0.016	0.034	1.47%
OREAS-147	GE_ICP91A	Fusion ICP	1.030	0.0300	4	1.019	0.019	0.019	-1.07%
SY-4	GE_ICP91A	Unspecified	0.004	0.0002	4	0.004	0.000	0.082	10.81%

* Expected values for the Standard NIST97B were sourced from the "Certificate of Analysis" documents issued by National Bureau of Standards (now known as the National Institute of Standards and Technology). Expected values for the Standard NBS183 were sourced from the "Certificate of Analysis" documents issued by National Bureau of Standards. Expected values for the Standard SY-4 were sourced from the "Certificate of Analysis" documents issued by Natural Resources of Canada (NRCan). Expected values for the Standard RTS-3a were sourced from the "Certificate of Analysis" documents issued by Natural Resources of Canada.

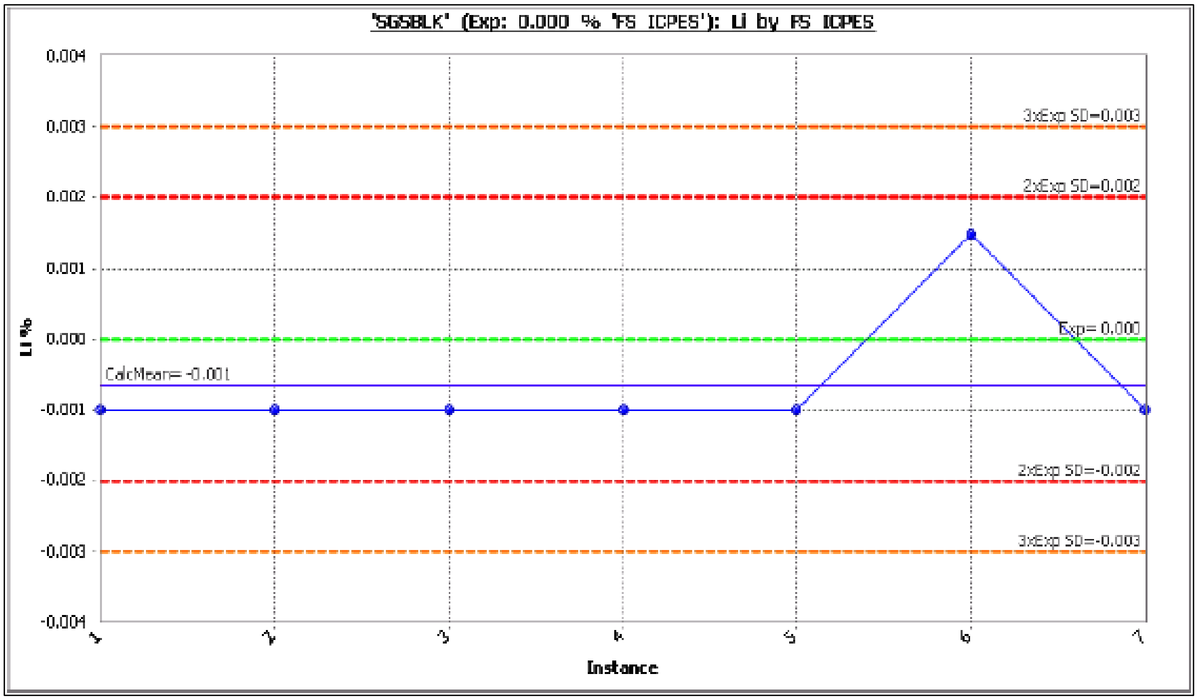


Figure 10-28: Blank Performance

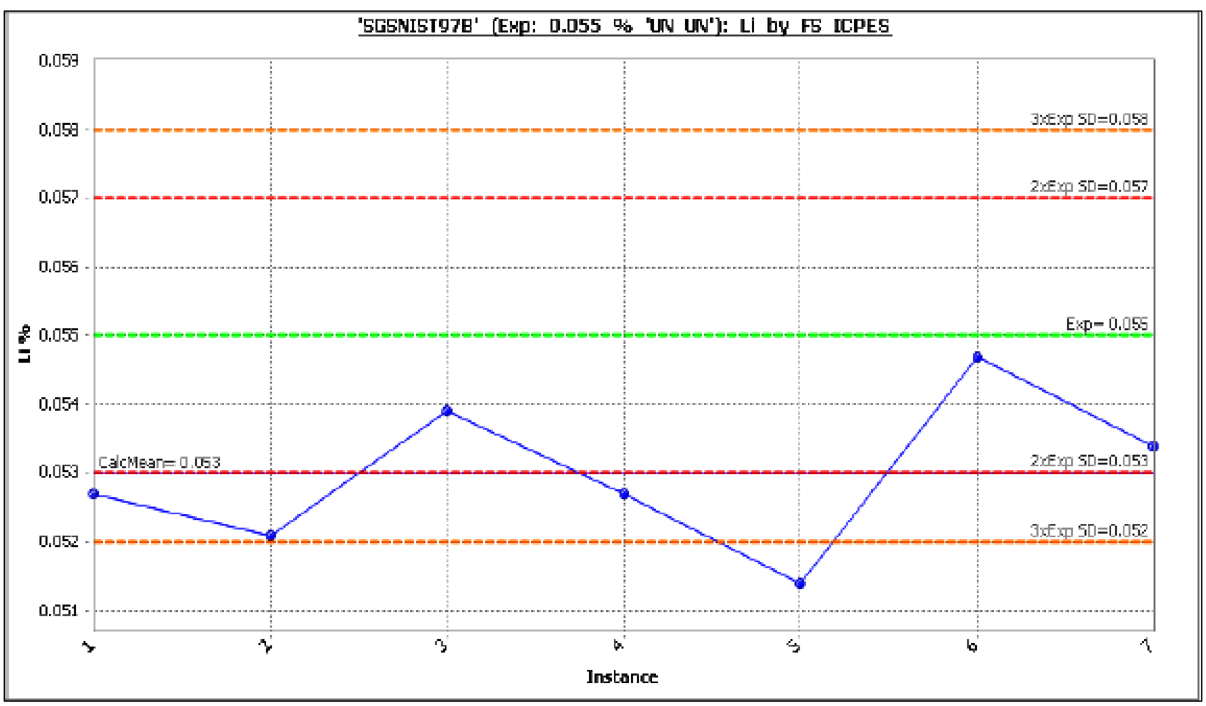


Figure 10-29: Standard NIST97B Performance

Table 10-16: SGS Laboratory Standard NIST97B Outlier

Standard	Batch	Sample Id	Method	Element	Value	Difference
SGSNIST97B	SU1800289	*STD-NIST97B_SU1800289_64	FS_ICPES	Li	0.051	-6.55

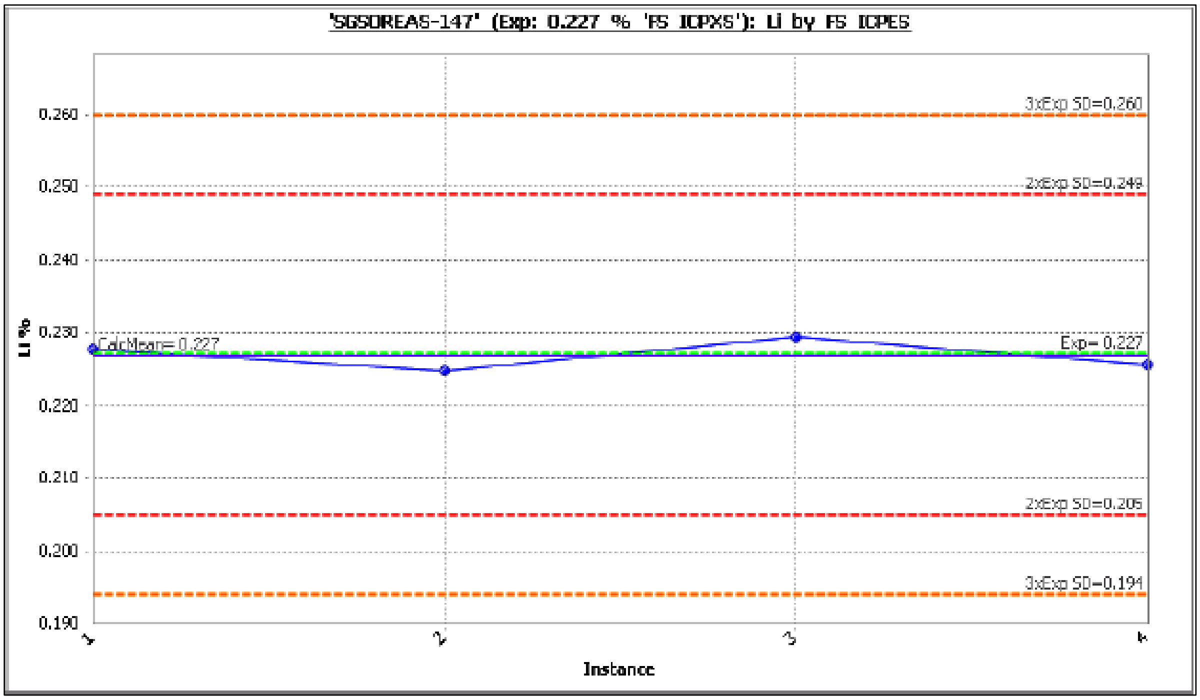


Figure 10-30: Standard OREAS-147 Performance

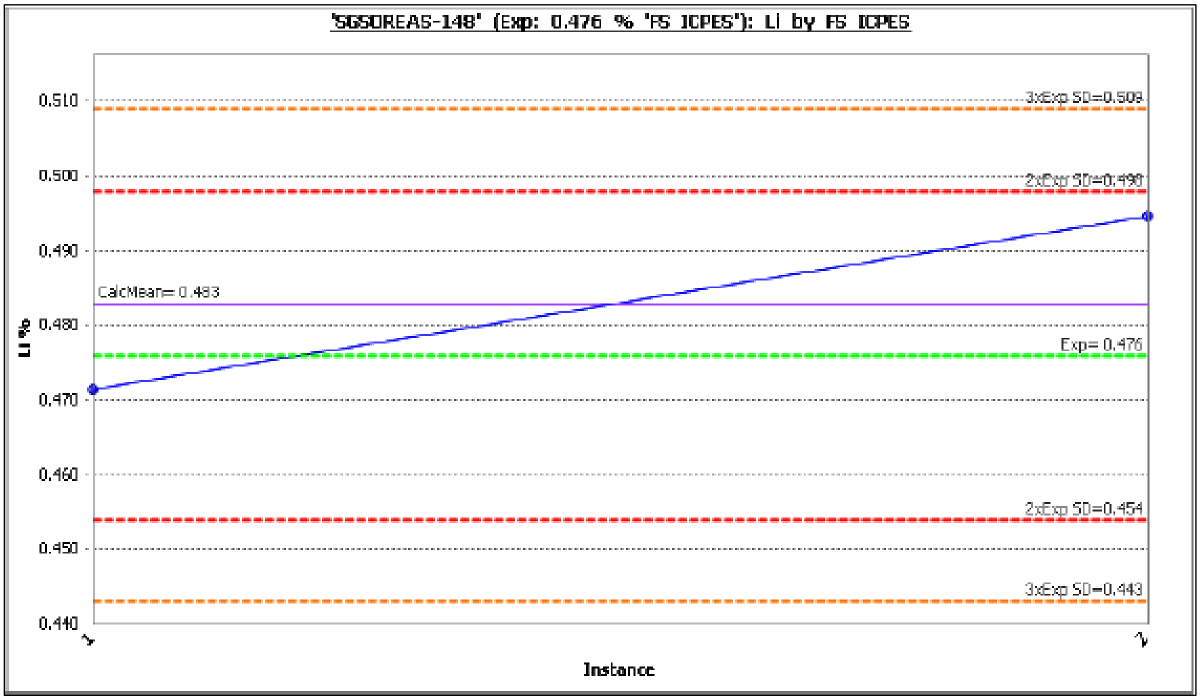


Figure 10-31: Standard OREAS-148 Performance

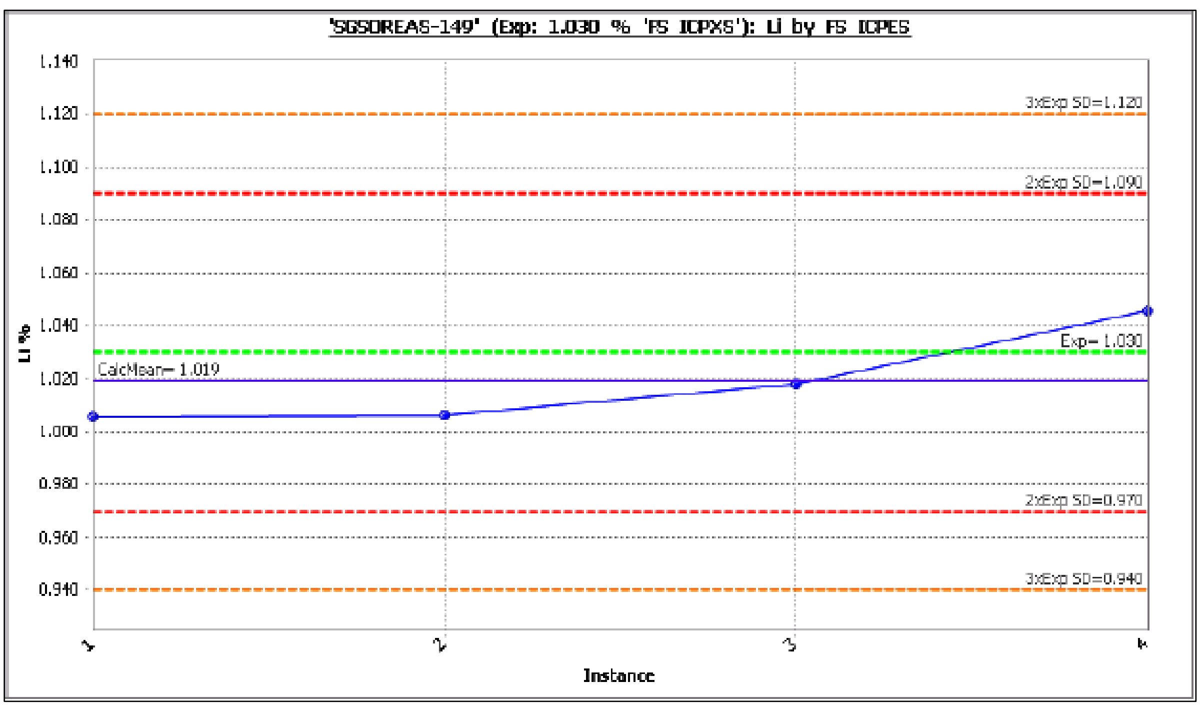


Figure 10-32: Standard OREAS-149 Performance

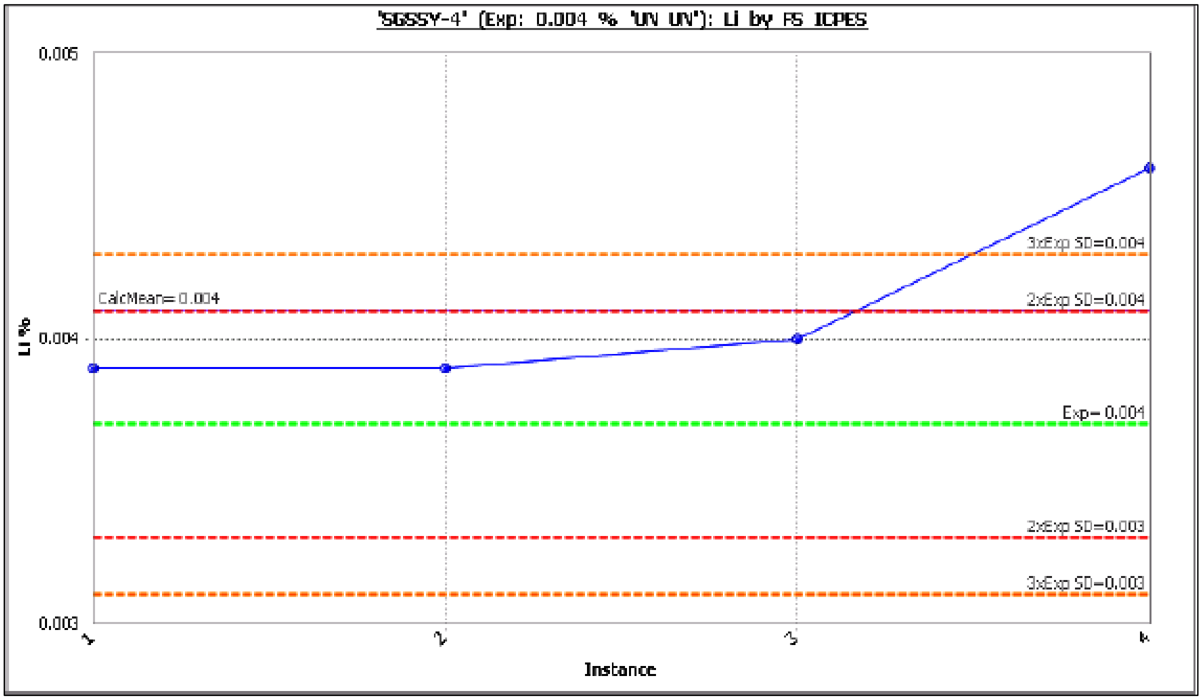


Figure 10-33: Standard SY-4 Performance

Table 10-17: SGS Laboratory Standard SY-4 Outlier

Standard	Batch	Sample Id	Method	Element	Value	Difference
SGSSY-4	SU1800178	*STD-SY-4_SU1800178_57	FS_ICPES	Li	0.005	24.32

10.11.4 SGS Lakefield 2018 Laboratory Duplicates and Repeats

During the 2018 diamond drilling campaign, SGS analyzed 20 routine laboratory duplicates in seven batches. The duplicates comprised seven DUP coarse duplicates and 13 REP pulp repeats. The DUP is a sample preparation duplicate, where a coarse sample is split into two and each sample is prepared and analyzed separately. The REP is a laboratory replicate analysis, where a pulp sample is analyzed twice. In 2018, the duplicates and repeats were analyzed by sodium peroxide fusion ICP-OES with HCl finish (GE_ICP91A).

10.11.5 Sayona Québec Duplicates 2018

Sayona Québec did not collect duplicate samples during the 2018 drill campaign. The SGS Lakefield laboratory reported two types of laboratory duplicates in their batches, a coarse duplicate and a pulp repeat, the results of which are included in the sections below.

10.11.6 SGS Lakefield 2018 Coarse Duplicates

SGS reported lithium values for seven routine laboratory duplicates. A comparison between the parent assays (Li1) and the laboratory duplicates (Li2) is summarized in Table 10-18 and Figure 10-34.

Table 10-18: Laboratory Duplicate Summary Statistics – 2018

No. of Samples	mean Li1	mean Li2	SD Li1	SD Li2	CV Li1	CV Li2	sRPHD (mean)
7	0.11	0.11	0.19	0.19	1.62	1.62	0.68

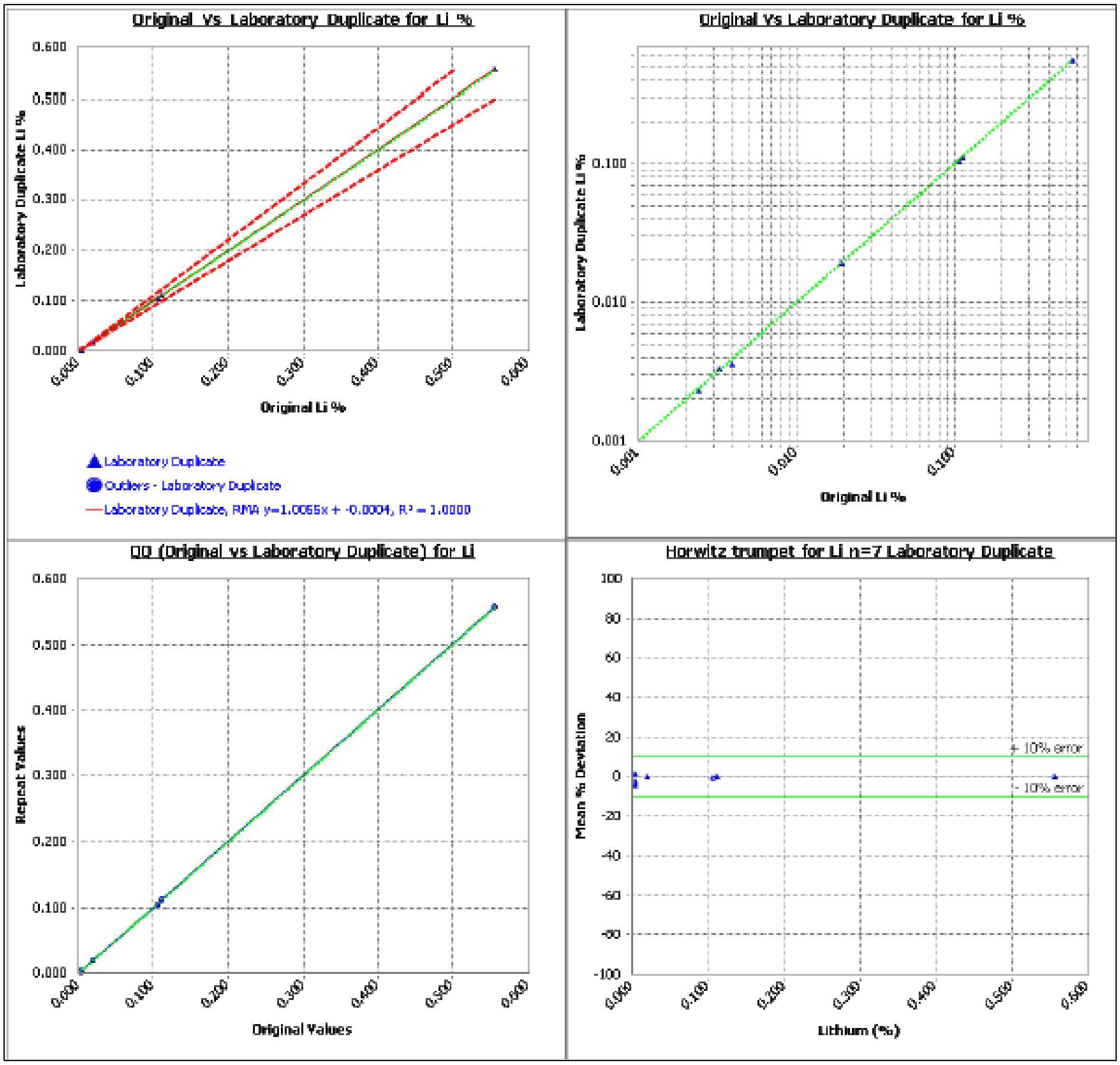


Figure 10-34: Laboratory Duplicates Correlation Plot, QQ Plot and Horwitz Trumpet Chart

10.11.7 SGS Lakefield 2018 Pulp Repeat

SGS reported lithium values for 13 routine laboratory repeats during the 2018 drilling campaign. A comparison between the parent assays (Li1) and the laboratory repeats (Li2) is summarized in Table 10-19 and Figure 10-35.

Table 10-19: Laboratory Repeat Summary Statistics – 2018

No. of Samples	mean Li1	mean Li2	SD Li1	SD Li2	CV Li1	CV Li2	sRPHD (mean)
13	0.17	0.18	0.26	0.27	1.52	1.53	-0.31

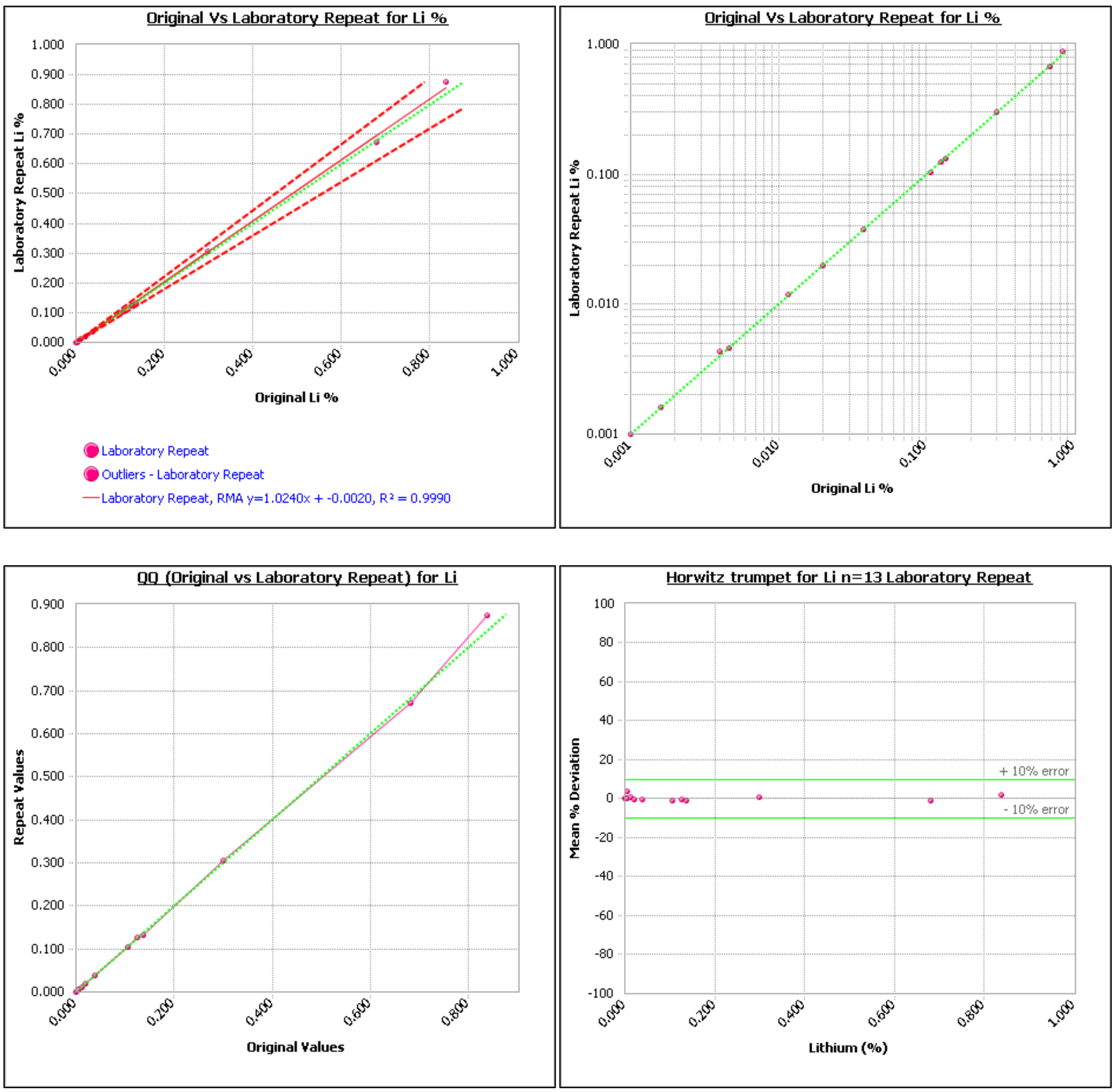


Figure 10-35: Laboratory Repeats Correlation Plot, QQ Plot and Horwitz Trumpet Chart – 2017

11. DATA VERIFICATION

11.1 General

Sayona Québec conducted the current mineral resources estimate (MRE) for the Authier deposit using an updated validated database, which incorporates diamond drilling programs completed by Sayona in 2016, 2017 and 2018. The database also includes validated historical drilling data from the Glen Eagle programs between 2010 and 2012. The Glen Eagle drilling database was also validated by SGS Geostat for the MRE released on November 19, 2013, by Glen Eagle.

The validation did not return any significant issues. The AL-10-XX, AL-11-XX and AL-12-XX collar coordinates present in the database were taken from signed originals and authorized copies of surveyed collar data from independent land surveying companies.

As part of the data verification, the analytical data from the database has been validated with values reported in the laboratories' analytical certificates. The total laboratory certificates verified amounts to a minimum of 20% of the overall laboratory certificates of the Property. There were no relevant errors or discrepancies noted during the validation.

The database used to produce the MRE is derived from a total of 225 holes from across the entire Authier property, including:

- 81 historical;
- 69 drilled by Glen Eagle between 2010 and 2012; and
- 75 drilled by Sayona Québec between 2016 and 2018.

The database contains the survey collar location, lithology and analytical results.

The database cut-off date is April 24, 2018. Sayona Québec is of the opinion that the final drill hole database is adequate to support the MRE.

From this database, 199 drill holes were used for the solid modelling and MRE.

There are a total of 5,049 assay intervals in the database used for the current MRE and 2,456 of them are contained inside the mineralized solids.

11.2 Check Sampling of 2010 Assay Results by SGS Geostat

As part of the 2010 data verification program, SGS Geostat completed independent analytical checks of drill core duplicate samples taken from Glen Eagle's 2010 diamond drilling program. SGS Geostat also conducted analysis of twin holes completed by Sayona Québec to validate the historical analytical data. Finally, verification of the laboratories' analytical certificates and validation of the project digital database supplied by Glen Eagle were verified for errors or discrepancies.

Thirty (30) mineralized drill core duplicates were collected from holes AL-10-01 and AL-10-11 by SGS Geostat. The comparison of the 2010 original and duplicate analytical values is suggesting a small analytical bias toward the original samples processed by ALS. The 2010 Glen Eagle pulp duplicate program also came to this conclusion. The 2010 analytical bias was not very significant, with the duplicate samples returning an average Li_2O value 7.9% higher compared to the original samples.

11.3 Check Sampling of 2011-2012 Assay Results by SGS Geostat

SGS Geostat completed analytical checks of drill core duplicate samples taken from selected Glen Eagle 2011-2012 diamond drill holes on the Authier deposit as part of the independent data verification program. SGS Geostat also conducted verification of the laboratories analytical certificates and validation of the database supplied by Glen Eagle for errors and discrepancies.

During the July 30, 2012, site visit by the author, Maxime Dupéré (P.Geo.), a total of 38 mineralized core duplicates from the Authier pegmatite were collected from holes AL-11-01, AL-11-16 and AL-12-20, and submitted for analysis at SGS Minerals' laboratory in Lakefield (SGS Lakefield), Ontario, Canada which is an accredited ISO/IEC 17025 laboratory. The analytical method used by SGS Lakefield is the ore grade analysis using sodium peroxide fusion with induced coupled plasma optical emission spectrometry (ICP-OES) finish methodology with a lower detection limit of 0.01% Li (SGS code ICP90Q). This method uses 20 g of pulp material. Blanks were inserted respectively at the beginning and the end of the sample series. Two homemade reference materials were also inserted in the samples series: High-Li and Low-Li.

Figure 11-1 shows the correlation plots for the check data versus the original data. A summary of the statistical analysis conducted on the data is shown in Table 11-1.

There is a good assay correlation for Li_2O. The correlation coefficient is above 0.9. The average Li_2O grade of the duplicate assays is 13% higher than the original samples. The sign test shows that the proportion of pairs with an old sample value greater than the new sample value is 8 out of 43. The sign test clearly showed a bias at a 95% confidence level. In comparison to the previous check samplings done by SGS Geostat, results show a clear variability of assay results between laboratories. It is SGS Geostat and the author's opinion that this difference in favour of the 2010 samples for ALS and the one in favour of the old assay results from the 2012 sampling (AGAT) is less than 15% and is considered acceptable. Recommendations will be made to mitigate this difference in the recommendation section.

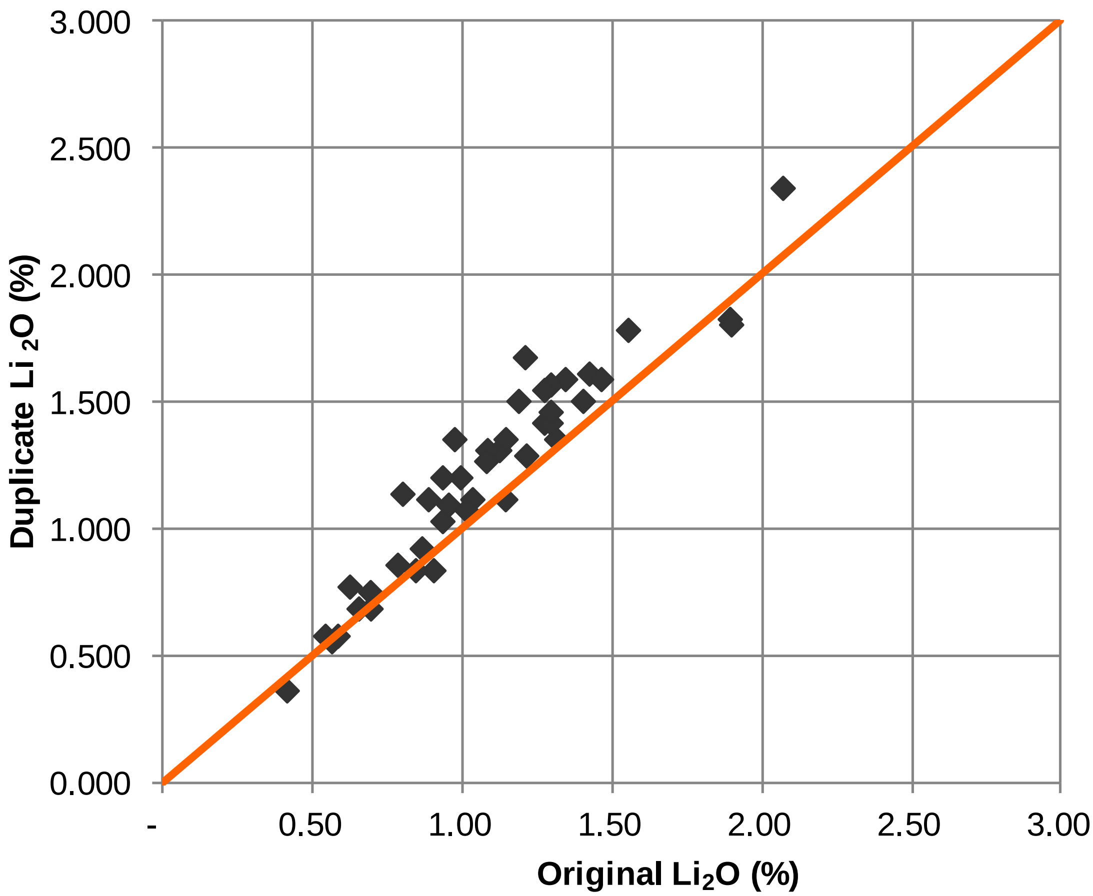


Figure 11-1: Correlation Plot for Independent Check Samples

Table 11-1: Summary Statistical Analysis of Original and Check Assay Results

Criteria	Count	Original < Duplicate	Original > Duplicate
All samples	43	35 81%	8 19%
> 0.75%	35	30 86%	5 14%
> 0.75% & <= 1.5%	31	28 90%	3 10%
> 1.5%	4	2 50%	2 50%

Criteria	Count	Relative Percent Difference Within Range		
		±10%	±25%	±50%
All samples	43	20 47%	19 44%	4 9%
> 0.75%	35	20 57%	11 31%	4 11%
> 0.75% & <= 1.5%	31	12 90%	15 48%	4 13%
> 1.5%	4	2 39%	2 50%	0 0%

11.4 Check Sampling of 2016 Assay Results by Sayona Québec

Sayona Québec has not conducted check sampling for 2016 samples or historical drill holes as part of the 2016 drilling program.

11.5 Check Sampling of 2017 Assay Results by Sayona Québec

Sayona Québec has not conducted check sampling for 2017 samples or historical drill holes as part of the 2017 drilling program.

11.6 Check Sampling of 2018 Assay Results by Sayona Québec

Sayona Québec has not conducted check sampling for 2018 samples or historical drill holes as part of the 2017 drilling program.

11.7 Twinning of Historical Drill Holes

As part of the Stage 3 drilling program in December 2017, Sayona Québec drilled seven diamond core holes for 769.5 m, PQ diameter, to collect 5.5 tonnes of pegmatite material for the pilot plant program.

All PQ drill holes were from the same drilling pad as both of Sayona Québec's historical holes; PQ holes were sampled metre by metre (full core). The diamond core was assayed and stage-crushed to the appropriate particle size to feed the pilot plant. The samples were processed and assayed at SGS Lakefield for lithium using sodium peroxide fusion, followed by ICP-OES analysis and whole rock analysis (major elements) using X-ray fluorescence (XRF76V) with majors by lithium metaborate fusion. No internal or laboratory QA/QC was applied for the metallurgical sampling as the aim of the analysis was to estimate composition of the two composite pilot plant feed samples, which represented Years 0 to 5 and Years 5+ of the operation.

Comparisons between holes were performed, when possible, based upon holes that were collared and positioned, i.e. azimuths and dip, close enough to the original. Table 11-2 shows the results obtained.

Table 11-2: Comparative Results for Metallurgical Pilot Plant Drill Holes vs Original Drill Holes - Authier Property

Drill Hole	From (m)	To (m)	Thickness (m)	Grade (% Li_2O)	Relative Difference (%)
AL-17-32	13	78	65	1.29	4.55
AL-16-01	12	74	62	1.35	
AL-17-33	53	99	46	1.28	8.14
AL-16-02	50	99	49	1.18	
AL-17-34	56	91	35	1.09	15.05
AL-14	49.38	99.36	49.98	1.27	
AL-17-35	4.7	42	37.3	0.98	NC (1)
AL-12-09	6	33	27	0.85	
AL-17-36	67	81	14	1.47	NC (2)
	83	94.9	11.9	1.57	
	104	112	8	1.49	
AL-10-01	72	112.5	40.5	1.38	
AL-17-37	139	146	7	1.15	NC (3)
	151	167	16	0.54	
AL-16-11	135	175	40	1.39	
AL-17-38	34	52	18	0.96	NC (4)
	54	60	6	1.32	
	63	65	2	1.3	
R-93-06	36.58	70.1	33.52	1.12	

Table 11-2 shows a good correlation between AL-17-32 vs AL-16-01 and AL-17-33 vs AL-16-02, which were collared less than 5 m from original and drilled at the same azimuth and dip. The correlation is fair for AL-17-34 vs AL-14.

Note that NC means *no comparison* done due to technical or operational differences:

NC (1): No comparison was made between AL-17-35 and AL-12-09 because both holes were drilled at different azimuths and dips;

NC (2): No comparison was made between AL-17-36 and AL-10-01 because 2 m portions of pegmatite cores from AL-17-36 were used during the pilot plant setup and assays were not reported for such intervals;

NC (3): No comparisons were made between AL-17-37 and AL-16-11 because 2 m portions of pegmatite cores from AL-17-37 were used during pilot plant setup and assays were not reported for such intervals;

NC (4): No comparison was made between AL-17-38 and R-93-06 because 2 m portions of pegmatite cores from AL-17-38 were used during pilot plant setup and assays were not reported for such intervals.

Considering the grade and geometry variability observed in the Authier pegmatite intrusive body, the results of the metallurgical drill hole program showed a fair to good correlation between the metallurgical versus recent and historical drill holes.

Sayona Québec has not conducted twinning of historical drill holes as part of the Phase 1 (2016) and Phase 2 (2017) drilling programs.

Before Sayona Québec's acquisition and to validate the historical drilling data, SGS Geostat recommended that the Glen Eagle complete twin holes of selected historical drill holes from the AL-XX and the R-93-XX series. In 2010, following SGS Geostat's recommendations, Glen Eagle completed three (3) twin drill holes to verify the historical R-93-XX drill holes series. Holes R-93-01, R-93-13, and R-93-25 were twinned with holes Al-10-11, AL-10-06 and Al-10-01, respectively.

Hole AL-10-11 intersected the mineralized interval at a distance varying between 1 m and 5 m from hole R-93-01. Hole AL-10-11 returned 0.87% Li_2O over 35.90 m, which is 3.68% lower compared to the original mineralized interval of 0.90% Li_2O over the 43.28 m intersected in hole R-93-01.

Hole AL-10-06 intersected two mineralized intervals at a distance varying between 4 m and 4.5 m from hole R-93-13. The first mineralized interval intersected by hole AL-10-13 returned 1.17% Li_2O over 8.55 m, which is 9.36% lower compared to the original mineralized interval of 1.29% Li_2O over the 8.08 m intersected in hole R-93-13. The second mineralized interval intersected by hole AL-10-06 returned 0.83% Li_2O over 8.30 m, which is 27.31% lower compared to the original mineralized interval of 1.14% Li_2O over 9.75 m that was intersected in hole R-93-13.

Hole AL-10-01 intersected the mineralized interval at a distance less than 7.5 m from hole R-93-25. Hole AL-10-01 returned 1.35% Li_2O over 51.25 m, which is 8.46% higher compared to the original mineralized interval of 1.25% Li_2O over the 49.38 m that was intersected in hole R-93-25.

Due to localization difficulties encountered in the field by Sayona Québec, the twin drill holes planned for the AL-XX drill hole series were collared too far, more than 15-20 m, from the historical holes, to be considered valid for data verification. After reviewing all the drill data, two holes, one by the recent Glen Eagle drilling (Al-10-15) and one from the R93-XX series (R93-12), intersected mineral intervals near enough holes from the AL-XX series to be considered valid for data verification.

Hole AL-10-15 intersected a mineralized interval at a distance less than 4.5 m from hole AL-18. Hole AL-10-15 returned 1.20% Li_2O over 15.4 m, which is 75.3% higher compared to the original mineralized interval of 0.69% Li_2O over 15.24 m that was intersected in hole AL-18.

Hole R-93-12 intersected a mineralized interval at a distance less than 5 m from hole AL-24. Hole R-93-12 returned 0.81% Li_2O over 12.19 m, which is 11.8% lower compared to the original mineralized interval of 0.92% Li_2O over 11.52 m that was intersected in hole AL-24.

Hole AL-12-09 intersected one mineralized interval at a distance varying between 1.5 m and 5 m from hole AL-16. The mineralized interval intersected by hole AL-12-19 returned 0.81% Li_2O over 27 m, which is 22.1% lower compared to the original mineralized interval of 1.01% Li_2O over the 27.4 m that was intersected in hole AL-16.

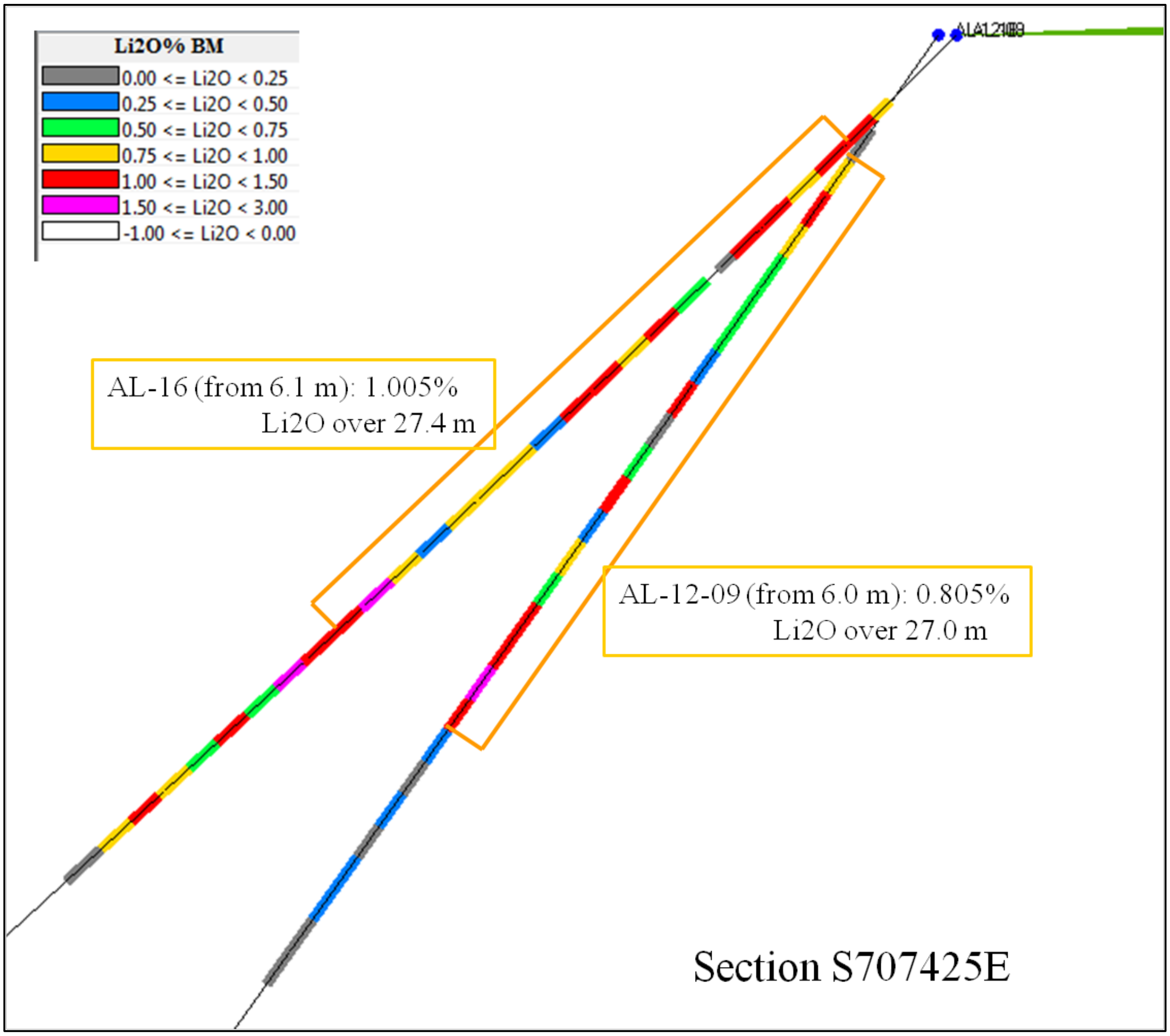


Figure 11-2: Oblique View Showing Results for Twin Holes Al-16 and AL-12-09

Hole AL-12-14 intersected one mineralized interval at an average distance of less than 8.5 m from hole AL-19. The mineralized interval intersected by hole AL-12-19 returned 0.74% Li_2O over 36 m, which is 43.4% lower compared to the original mineralized interval of 1.15% Li_2O over the 35.1 m that was intersected in hole AL-19.

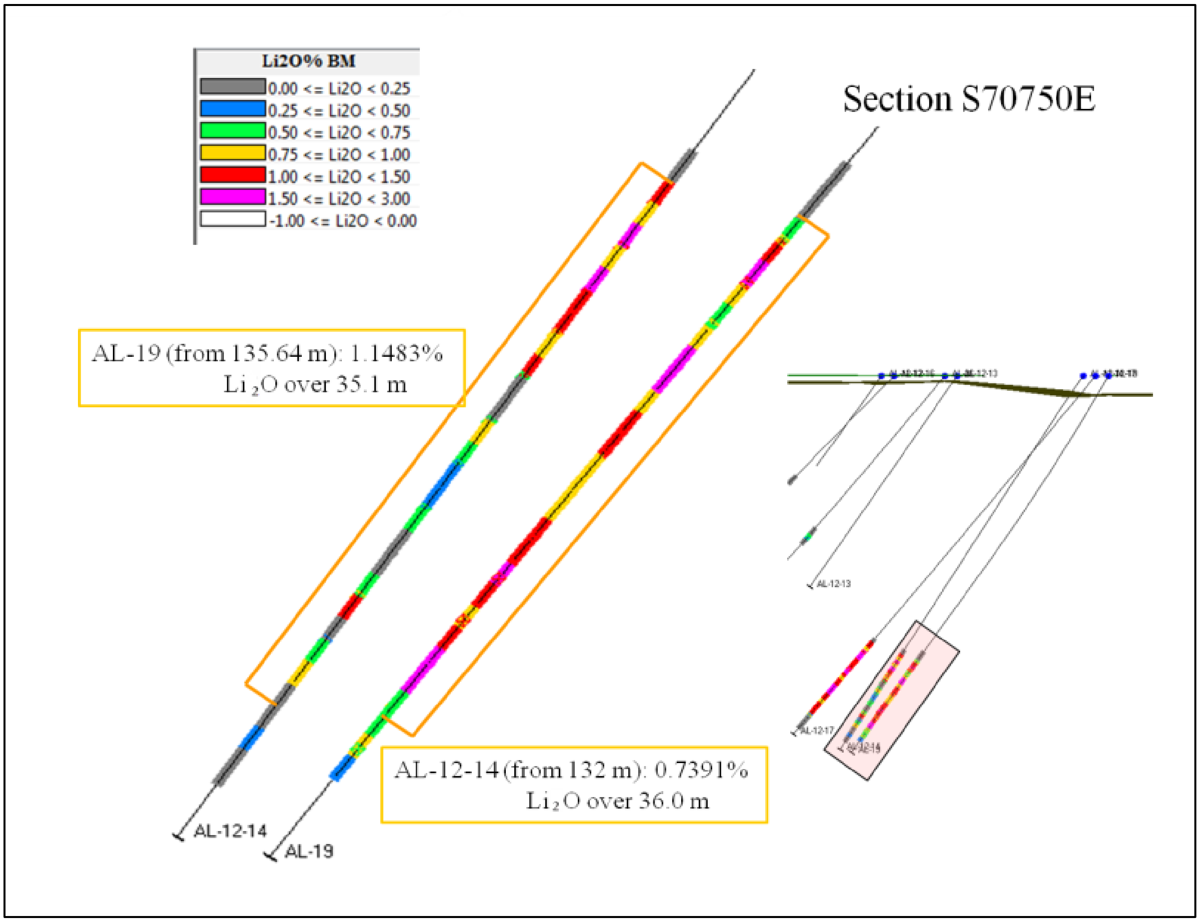


Figure 11-3: Oblique View Showing Results for Twin Holes Al-19 and AL-12-14

Considering the grade and geometry variability observed in the Authier pegmatite intrusive body, the results of the twin drill hole program showed a fair to good correlation between the recent and historical drill holes, except between historical R-93-13 and AL-10-06 as well as historical AL-19 and AL-12-14, lower mineralized intercepts of which returned Li_2O grade differences in excess of 30% and 40% differences, respectively. No systematic analytical bias was outlined. Based upon the results of the twin hole drill program, SGS Geostat considers the historical drill data to be of acceptable quality to be included in the final drill hole database of the project. Table 11-3 summarizes the overall results of the 2010-2012 twin hole drilling program.

Table 11-3: Comparative Results from the 2010-2012 Twin Hole Drill Program at Authier

Hole ID	From	To	Length	Weighted Average Li_2O (%)	Relative Percent Difference (%)
R-93-01	35.97	79.25	43.28	0.90	3.75
AI-10-11	38.55	74.45	35.90	0.87	
R-93-13	7.16	15.24	8.08	1.29	9.82
AI-10-06	6.55	15.10	8.55	1.17	
R-93-13	31.09	40.84	9.75	1.14	31.63
AI-10-06	32.70	41.00	8.30	0.83	
R-93-25	76.20	125.58	49.38	1.25	8.11
AI-10-01	72.00	123.25	51.25	1.35	
AL-18	96.62	111.86	15.24	0.69	54.72
AI-10-15	81.00	96.40	15.40	1.20	
AI-24	79.34	90.86	11.52	0.92	12.59
R-93-12	96.93	109.12	12.19	0.81	
AL-16	6.10	33.53	27.43	1.01	22.1
AL-12-09	135.64	170.69	27.00	0.81	
AL-19	135.64	170.69	35.05	1.15	43.4
AL-12-14	132.00	168.00	36.00	0.74	

The final database includes the historical and the 2010-2012 drilling data compiled from the Glen Eagle exploration programs. Table 11-3 lists the data contained in the final drill hole database. Although the sign test clearly showed a bias at a 95% confidence level with a 7.9% difference in favour of the duplicate (SGS) Li_2O results, SGS Geostat is of the opinion that the final drill hole database is adequate to support mineral resources estimation.

11.8 Specific Gravity

As part of the 2010 independent data verification program, SGS Geostat conducted specific gravity (SG) measurements on 38 mineralized core samples collected from drill holes AL-10-01 and AL-10-11. The measurements were performed using the water displacement method, i.e. weight in air divided by volume of water displaced, on representative half-core pieces weighing between 0.67 kg and 1.33 kg, with an average of 1.15 kg, yielding an average SG value of 2.71 t/m³.

Table 11-4: Specific Gravity Measurements Statistical Parameters (2010 Program)

	Unit	Mineralized Material
Count	#	38
Mean	t/m^3	2.71
Std Dev	t/m^3	0.01
Minimum	t/m^3	2.64
Median	t/m^3	2.71
Maximum	t/m^3	2.81

In 2017, Sayona Québec performed a density validation program on both mineralized and non-mineralized material. Core samples were sent to ALS in Val-d'Or, Québec, which did the measurements using the same water displacement method. The results of these tests are presented Table 11-5.

Table 11-5: Specific Gravity Measurements Statistical Parameters (2017 Program)

	Unit	Non-mineralized Material	Mineralized Material
Count	#	14	15
Mean	t/m^3	2.90	2.70
Std Dev	t/m^3	0.07	0.05
Minimum	t/m^3	2.77	2.62
Median	t/m^3	2.91	2.70
Maximum	t/m^3	2.99	2.86

12. METALLURGICAL TESTING

12.1 Introduction

Initial testwork on the deposit was undertaken by the Québec Department of Natural Resources in 1969. Flotation tests were carried out on a bulk composite sample prepared from split drill core. Results confirmed the ore was amenable to concentration by flotation and the tests produced spodumene concentrates assaying between 5.13% and 5.81% Li_2O with lithium recovery ranging from 67% to 82%.

In 1991, Raymor Resources Ltd. conducted bench-scale metallurgical testing on mineralized pegmatite samples from the Property. An 18.3 kg sample grading 1.66% Li_2O was tested at the Centre de Recherche Minérale (CRM, now COREM) in Québec City. The testwork produced a spodumene concentrate grading 6.30% Li_2O with lithium recovery of 73%.

In 1997, Raymor Resources Ltd. completed testing at CRM on two samples from a pegmatite dyke on the Property: 1) 18 t sample grading 1.32% Li_2O and 2) 12 t grading 1.10% Li_2O. Metallurgical testing on the first sample produced a concentrate grading 5.61% Li_2O with 61% lithium recovery. Magnetic separation was used in the testing to remove iron-bearing silicate minerals. The second sample returned a final concentrate grade of 5.16% Li_2O with 58% recovery.

In 1999, metallurgical testing was conducted at COREM on a 40 t mineralized pegmatite sample from the main intrusion at the Authier property. The testing program was conducted as part of a pre-feasibility study. Results showed spodumene concentrate grades ranging from 5.78% to 5.89% Li_2O with lithium recoveries ranging from 68% to 70% from a sample with head grade of 1.14% Li_2O. A sample with head grade of 1.35% Li_2O produced a 5.96% Li_2O concentrate at 75% recovery.

Table 12-1 gives an overview of recent metallurgical testing programs operated by SGS Canada Inc. at their facilities in Lakefield, Ontario.

Table 12-1: Recent Authier Metallurgical Testing Programs

<table>
<tr><th>Year</th><th>Owner</th><th>Sample Size</th><th>Testwork</th></tr>
<tr><td>2012</td><td>Glen Eagle</td><td>270 kg</td><td>Flotation testing</td></tr>
<tr><td>2016</td><td rowspan="6">Sayona Québec</td><td>430 kg</td><td>HLS and flotation testing</td></tr>
<tr><td rowspan="3">2017</td><td>52 kg</td><td>HLS and flotation testing</td></tr>
<tr><td>66 kg sample</td><td>HLS and flotation testing</td></tr>
<tr><td>120 kg sample</td><td>HLS</td></tr>
<tr><td>2018</td><td>5 t sample</td><td>Pilot plant program</td></tr>
<tr><td>2019</td><td>Pilot plant sample</td><td>Batch optimization testing</td></tr>
</table>

Figure 12-1 shows the locations in the deposit from which the historical metallurgical testing samples were taken.

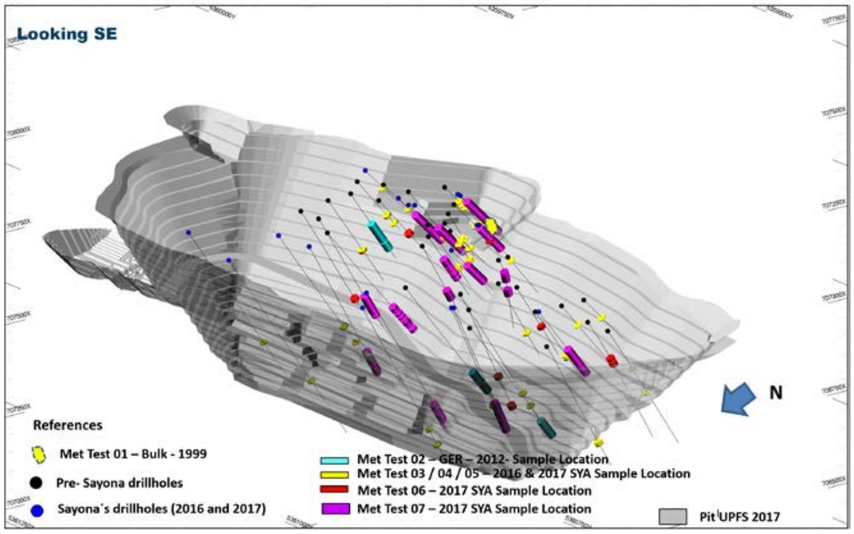


Figure 12-1: Drill Hole Locations for the Various Metallurgical Testing Samples

Glen Eagle Resources Inc. undertook a testing program in 2012 on a 270 kg sample as part of a Preliminary Economic Assessment (PEA) of the project. Batch testwork produced a concentrate grading 6.09% Li_2O with 88% lithium recovery after two stages of cleaning (without the use of mica pre-flotation). After four stages of cleaning and passing the concentrate through a Wet High-Intensity Magnetic Separator (WHIMS) at 15 G a concentrate grading 6.44% Li_2O was produced at 85% recovery.

In 2016, Sayona Québec completed a metallurgical testing program using drill core from 23 historical holes totaling 430 kg, representing the entire deposit geometry (including 5% mine ore dilution). Concentrate grades varied from 5.38% to 6.05% Li_2O with at lithium recovery ranging from 71% to 79%. Results indicated that ore dilution had a negative impact on flotation performance.

In 2017, two representative samples were prepared and flotation testing was undertaken to examine the impact of the presence of dilution material and the use of site water. Testwork demonstrated the ability to produce concentrate grading 6.0% Li_2O at with lithium recovery greater than 80%.

The majority of the testing for the project has focused on spodumene recovery by froth flotation. Recently (2016-17), Sayona performed several heavy-liquid separation (HLS) test programs to assess the viability of producing a coarse spodumene concentrate using dense media. Testwork and economic analysis showed that dense media separation was not a viable process option for the Authier deposit.

A pilot plant testwork program was undertaken in 2018 at SGS Canada Inc. as part of the feasibility study. The aim of the testwork was to confirm the spodumene concentration flowsheet, operational parameters, efficiencies, and consumptions. Roughly 5 t of drill core was used to prepare two composite samples representing: 1) years 0-5, and 2) years 5+ of operation. Testwork included batch, locked-cycle and continuous piloting.

In late 2018, an optimization batch testwork program was undertaken at SGS. Testing was performed using samples from the two pilot plant composite samples. Tests examined magnetic separation, the effect of spodumene conditioning, and spodumene collector optimization.

12.2 COREM Testwork (1991-1998)

In 1991, bench-scale metallurgical testing was conducted on ore from the Authier deposit at CRM in Québec City. Bench testing was followed by operation of a pilot plant in 1997-1998. A 40 t mineralized sample from a test pit (Figure 12-2) with head grade of 1.22% Li_2O was tested. Pilot testing was performed at a grind size of 75 µm (P_{75}). The process flowsheet consisted of mica, spodumene, and feldspar flotation (and to produce saleable concentrates for each). Magnetic separation was performed on the final spodumene concentrate to remove iron bearing silicate minerals. During pilot plant testing, a spodumene concentrate of 5.16% Li_2O with 70% lithium recovery was achieved.[1]

Figure 12-2 - Authier Bulk Test Pit

[1] The full report from the metallurgical program was not made available to Sayona Québec and the above references were derived from the Glen Eagle 2012 Preliminary Economic Assessment.

12.3 Glen Eagle Resources Inc. Testwork (2012)

In 2012, Glen Eagle Resources Inc. operated a testwork program at SGS Canada Inc. in Lakefield, Ontario on samples from the Authier deposit. A 270 kg representative sample was prepared from three drill holes along the strike length of the deposit. The average grade of the sample was 1.23% Li_2O (Table 12-2) which was higher than the 1.02% Li_2O resource grade outlined in the 2012 Glen Eagle 43-101 PEA.

Table 12-2: Feed Sample Chemical Analysis (2012 Testing)

Analysis	Grade (%)
Li	0.57
Li_2O	1.23
SiO_2	74.9
Al_2O_3	15.8
Na_2O	4.22
K_2O	3.08
Fe_2O_3	0.59
CaO	0.18
MgO	0.07
P_2O_5	0.02
MnO	0.10
Cr_2O_3	0.02
LOI	0.40

Mineralogical analysis (Table 12-3) showed major components of the sample to be albite (37%), quartz (27%), microcline (16%), spodumene (15%), and muscovite (5%).

Table 12-3: Mineralogical Analysis of the Feed Sample

Mineral	Weight (%)
Albite	37.2
Quartz	26.5
Microcline	16.2
Spodumene	14.9
Muscovite	4.8
Total	99.6

12.3.1 Grindability

Bond rod mill work index (RWI) and Bond ball mill work index (BWI) were determined to be 12.3 kWh/t and 15.6 kWh/t, respectively (Table 12-4). BWI was tested using a closing screen size of 150 µm.

Table 12-4: Grindability Results (2012)

Sample	RWI (kWh/t)	BWI (kWh/t)
2012 Composite	12.3	15.6

12.3.2 Bench-scale Flotation Tests

SGS completed ten batch flotation tests based on a typical spodumene flotation flowsheet. Variables investigated during the program included grind size, collector types, and the use of mica pre-flotation.

Stage-grinding was performed and scrubbing was performed in a Denver flotation cell at pH of 11 (using NaOH) and in the presence of lignin sulfonate (D618). De-sliming was by settling and decantation. Final concentrate was passed through a Wet High Intensity Magnetic Separator (WHIMS) for upgrading.

The testwork showed that a 6.0% Li_2O concentrate could be produced with lithium recovery greater than 80%. Test F8 showed the best flotation performance (conditions shown in Table 12-5) and resulted in the production of a spodumene concentrate grading 6.09% Li_2O (and 1.57% Fe_2O_3) with 88% lithium recovery after two cleaning stages without the use of mica pre-flotation. Flotation kinetics were fast with the majority of the concentrate floated within 1 min. Rougher flotation was typically completed in less than 3 min.

Table 12-5: Test F8 Test Conditions (2012)

Test	Objective	Grind (P_{100})	Reagent Dosage (g/t)				
			Fuel Oil	NaOH	Na_2CO_3	D618	FA-2
F8	Test without mica flotation	-210 µm	0	275	50	450	675

Table 12-6: Test F8 Bench-scale Flotation Results

Product	Weight (%)	Assay (%)		Dist. (%)	
		Li_2O	Fe_2O_3	Li	Fe
Non-mag final concentrate	16.1	6.44	1.06	85	24
4th Cleaner concentrate	16.7	6.29	1.58	86	37
3rd Cleaner concentrate	17.1	6.21	1.58	87	38
2nd Cleaner concentrate	17.6	6.09	1.57	88	39
1st Cleaner concentrate	18.4	5.89	1.57	89	40
Rougher concentrate	18.7	5.81	1.56	89	41
Rougher and scav. conc	21.8	5.07	1.52	90	46

Optimal flotation performance was achieved:

- Stage-grinding to a closed sizing of 150 µm;
- Without mica pre-flotation;
- Incorporating de-sliming using dispersant with pH adjustment with sodium hydoride (pH 11);
- Using soda ash as pH regulator in flotation (pH 8);
- Using a fatty acid spodumene collector.

12.4 Sayona Québec Metallurgical Testing (2016)

Nine composite samples weighing 358 kg were shipped to SGS Canada Inc. in Lakefield, Ontario for metallurgical testing. The composites were prepared using historical drill core samples. Drill core was selected to produce a sample with similar average grade and mineralogy to the deposit as a whole. In addition, 5% ore dilution from the hanging wall was added.

Sample AMET1 was the main composite tested and the remaining eight composites were considered variability samples. The head assays of the nine composite samples are shown in Table 12-7. Head grades varied between 0.88% Li_2O to 1.12% Li_2O. AMET1 had the highest head grade (average resource grade was 0.96% Li_2O for the July 2016 JORC Resource Estimate).

Table 12-7: Composite Sample Assays (2016)

AMET	Assay (%)												
	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	P_2O_5	MnO	LOI	Sum
1	0.52	1.12	73.9	15.7	0.81	0.62	0.34	4.42	2.83	0.03	0.11	0.67	99.4
2	0.44	0.95	72.2	15.0	0.83	0.35	0.32	4.73	2.85	0.03	0.09	0.47	96.9
3	0.41	0.88	74.2	15.6	0.59	0.19	0.21	4.88	3.13	0.02	0.09	0.57	99.5
4	0.51	1.10	72.6	15.3	1.20	1.25	0.49	4.06	2.89	0.03	0.11	0.80	98.8
5	0.51	1.10	73.6	15.3	0.95	0.98	0.47	4.17	3.06	0.02	0.10	0.70	99.4
6	0.49	1.05	73.3	15.5	0.91	0.89	0.42	4.34	3.04	0.04	0.10	0.68	99.2
7	0.52	1.12	71.2	15.0	0.67	0.21	0.17	4.50	2.80	0.03	0.09	0.50	95.2
9	0.48	1.03	73.5	15.1	1.03	1.09	0.42	3.96	3.08	0.02	0.09	0.83	99.1
10	0.41	0.88	74.0	15.3	0.66	0.19	0.2	4.48	3.15	0.03	0.09	0.60	98.7

12.4.1 Feed Characterization

Mineralogy (QEMSCAN analysis) of the nine composite samples is shown in Table 12-8. Major components of the samples were plagioclase, quartz, K-feldspar, spodumene and muscovite. Spodumene content in the AMET1 sample was 13.3% and ranged from 9.6% to 13.7% in the variability samples. Minor quantities of several iron-bearing silicate minerals were detected; notably biotite, tourmaline, amphibole/pyroxene, and chlorite.

Table 12-8: Mineralogical Analysis (2016)

	Mineral	AMET Sample No.								
		1	2	3	4	5	6	7	9	10
Mineral Composition (%)	**Spodumene**	13.3	11.0	9.6	12.4	11.1	11.8	13.7	13.4	10.8
	Quartz	24.8	27.4	24.8	27.3	25.4	25.5	26.7	28.5	27.4
	Plagioclase	38.5	38.8	40.7	34.2	35.9	37.5	37.8	33.2	38.3
	K-Feldspar	18.6	18.7	21.5	17.9	21.0	18.6	19.1	19.2	20.6
	Muscovite	1.88	1.77	1.58	2.49	1.83	1.96	1.68	2.19	1.84
	Biotite	1.02	0.55	0.44	0.69	0.55	2.08	0.33	0.12	0.30
	Tourmaline	0.29	0.22	0.34	0.28	0.24	0.22	0.26	0.20	0.13
	Amphibole/Pyroxene	1.29	1.11	0.48	3.21	2.77	2.12	0.06	2.39	0.43
	Chlorite	0.18	0.10	0.21	1.34	0.92	0.06	0.18	0.64	0.14
	Other	0.23	0.28	0.31	0.24	0.21	0.17	0.11	0.22	0.10
	Total	100	100	100	100	100	100	100	100	100

Liberation analysis was carried out on three size fractions; +425 µm, -425/+150 µm, and -150 µm. A relatively low portion (68.6%) of the spodumene in the -150 µm fraction was free and liberated (Table 12-9). Only roughly 35% of the spodumene was free and liberated in the coarser size fractions. The result demonstrates the relatively fine-grained nature of the sample.

Table 12-9: Spodumene Liberation for AMET1 Sample

Mineral / Association	Composition (%)			
	Combined	+425 µm	-425/+150 µm	-150 µm
Free Spodumene	**18.9**	**7.3**	**13.2**	**48.4**
Lib Spodumene	**23.5**	**28.0**	**21.6**	**20.2**
Spd: Quartz/Feldspar	50.3	58.2	57.0	24.7
Spd: Amphibole/Pyroxene	0.0	0.0	0.0	0.0
Spd: Phylosilicates	0.1	0.0	0.0	0.6
Spd: All Silicates	7.2	6.5	8.2	6.2
Total	**100**	**100**	**100**	**100**

Spodumene liberation analysis for all samples in the -150 µm fraction is shown in Table 12-10. Free and liberated spodumene ranged from 63.6% to 78.4%.

Table 12-10: Spodumene Liberation Analysis in the -150 µm Fraction

	Mineral / Association	AMET Sample No.								
		1	2	3	4	5	6	7	8	9
Composition (%)	Free Spodumene	48.4	54.8	39.7	45	43.6	51.3	58.7	54.5	46.9
	Lib Spodumene	20.2	23.6	26	18.5	21.6	13.6	15.9	20.2	21.7
	Midds Spodumene	19.7	15	20.8	25.8	26.1	21.6	15.6	16.1	20.8
	Sub Midds Spodumene	9.5	5.4	10.5	8.2	5.8	10.5	7	6.1	6.3
	Locked Spodumene	2.3	1.2	3	2.5	2.9	3	2.8	3	4.3
	Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**
	Free and Lib. Spod.	68.6	78.4	65.7	63.6	65.2	64.9	74.6	74.8	68.6

12.4.2 Grindability

Grindability test results are shown in Table 12-11. Bond abrasion tests were performed on five samples. Abrasion index (Ai) ranged from 0.968 g to 1.066 g and fell in at least the 98th percentile in the SGS database which classified the composite samples as very abrasive. BWI was performed on four samples and ranged from 14.2 kWh/t to 14.9 kWh/t classifying the composites as hard.

Table 12-11: Grindability Results (2016)

Sample	Ai (g)	BWI (kWh/t)
AMET1	1.032	14.2
AMET3	1.006	-
AMET4	-	14.8
AMET6	1.066	14.5
AMET9	0.968	14.9
AMET10	1.025	-

12.4.3 Heavy-liquid Separation

Bench-scale heavy liquid separation (HLS) tests were performed on five of the AMET samples. Samples were stage-crushed and the -6.4 mm / +0.86 mm size fraction was tested. Two process routes were available: 1) production of a high-grade concentrate (using heavy liquid with sg ca. 3.0), or 2) upgrading prior to flotation (using heavy liquid with sg ca. 2.7).

The results indicated that DMS was not a viable option for either of the process routes. Using a heavy liquid with sg of 2.95 g/cm^3, the testwork was unable to produce a 6.0% Li_2O concentrate (the highest grade achieved was 4.7% Li_2O). For upgrading tests (separation at 2.7 g/cm^3), results suggest that it was possible to reject more than 40% of the feed mass with lithium losses of between 6.3% and 15.0%. As an example, the AMET 1 test showed 42.2% mass rejection with associated lithium losses of 8.5%. Flotation feed for this test was increased from 1.18% to 1.87% Li_2O.

12.4.4 Bench-scale Flotation Tests

Testwork was undertaken on composite sample AMET1. Flotation charges were prepared with grind sizes ranging from 100% passing 300 µm to 75 µm. The results showed optimal grind size to be 100% passing 150 µm. Tests with finer grind size resulted in higher lithium losses to the slimes and generally poor flotation performance.

Magnetic separation was performed ahead of flotation in a rougher (5,000 G) - scavenger (10,000 G) arrangement to reject iron-bearing silicate minerals (e.g., biotite or amphiboles which would otherwise largely report to the final concentrate). Magnetic separation was performed ahead of flotation in an attempt to minimize lithium losses (previous testwork had performed magnetic separation on flotation concentrate). Results showed less than 1.5% lithium losses to the magnetic concentrate.

Table 12-13 summarizes the flotation test results on the AMET1 sample. Bench-scale results indicated that achieving a concentrate grade of 6.0% Li_2O was difficult and attributed to poor spodumene liberation (68.6% in case of the AMET1 composite). Results show that test F8 produced a concentrate grading 6.07% Li_2O with 71% lithium recovery. Test F15 achieved a concentrate grade of 5.88% Li_2O with 80% recovery. Both of the tests were operated with mica flotation. Based on the results, a locked-cycle test was operated (based on the F15 conditions) and resulted in the production of a 5.65% Li_2O concentrate with recovery of 82%.

Table 12-12: Summary of Batch Test Conditions for Tests F8 and F15 on AMET1 Sample

Test	Grind (µm)	Dosage (g/t)						
		H_2SO_4	NaOH	Na_2CO_3	Armac C	F100	FA-2	Na Silicate
F8	212	1200	250	56	75	400	575	50
F15	150	0	125	338	50	400	580	0

Table 12-13: Summary of Batch Flotation Tests F8 and F15 on AMET1 Sample

Test	Assay (% Li_2O)	Rec. (%)	Observations / Comments
F8	6.07	71.1	Mica flotation with Armac C at pH ~2.5, coarser grind produced >6.0% Li_2O.
F15	5.88	80.3	Mica flotation with Armac C at pH 10.5. Conc. grade 5.9% Li_2O, rec. >80%.

In summary, the testwork program demonstrated:

1. Non-liberated spodumene was mainly associated with quartz and feldspar.
2. Spodumene liberation did not exceed 74% in any size fraction.
3. 100% passing 150 µm was sufficient.
4. Achieving high grade spodumene concentrate (>6.0% Li_2O) at high recoveries (>80%) was challenging and attributed to poor spodumene liberation (in all size fractions).

12.5 Sayona Québec Metallurgical Test Programs (2017)

Several metallurgical testing programs were undertaken at SGS Canada Inc. in Lakefield, Ontario during 2017 to investigate:

- The effect of head grade and dilution on flotation performance (August 2017);
- Impact of grind size and the use of site water on flotation performance (October 2017);
- HLS testing (October and December 2017).

12.5.1 Bench-scale Flotation (August 2017)

Two composite samples were prepared from a total of 52 kg of drill core from four holes distributed about the ore body. The sample intervals were selected to provide representative: 1) grade (as compared to the resource), and 2) grain size domains (as identified in the core logging process: coarse, medium and fine). A high-grade (HG) composite and a low-grade (LG) composite were prepared based on the drill core lithia content. Sub-samples from the HG and LG composites were combined to produce an average composite (AG) grading roughly 1.0% Li_2O. Waste (dilution) material was included in certain test samples. Assays for each composite sample are given in Table 12-14. Note the high Fe_2O_3 content (9.39%) in the waste composite.

Table 12-14: Composite Assays for the August 2017 Test Program

Sample	Assay (%)												
	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	P_2O_5	MnO	LOI	Sum
HG	0.71	1.53	74.9	15.3	0.50	0.05	0.15	3.86	2.57	0.02	0.11	0.65	98
LG	0.20	0.43	74.4	15.4	0.60	0.17	0.21	5.98	2.68	0.03	0.10	0.56	100
AG	0.46	0.98	74.7	15.4	0.55	0.11	0.18	4.92	2.63	0.03	0.10	0.60	99
Waste	0.002	0.004	45.6	6.34	9.39	24.5	7.56	0.35	0.02	0.02	0.17	5.66	100

Tests were operated to confirm flotation response at a coarser grind than previously tested and to demonstrate the impact of head grade and the presence of dilution on spodumene concentrate grade and recovery. Test were undertaken on the HG, LG, and AG samples with grind sizes of 150 um and 180 um. Tests were also operated on AG samples which were diluted with waste rock.

Samples were stage-ground to the appropriate grind size and passed through a WHIMS at 5,000 G and 12,000 G. De-sliming was performed with F100 dispersant at pH 11 (with NaOH). Mica was floated using Aero 3030C collector. Spodumene was floated using 630 g/t FA-2 (total) collector with three stages of cleaning. Table 12-15 summarizes the batch results for the August 2017 testwork program.

Table 12-15: August 2017 Metallurgical Testing – Flotation Test Results

Test	Details	Head (% Li_2O)	Conc. Grade (%)		Li Rec. (%)
			Li_2O	Fe_2O_3	
F1	HG, 150 um	1.47	6.22	1.75	83.9
F2	HG, 180 um	1.50	6.45	1.87	86.2
F3	LG, 150 um	0.42	5.52	2.21	66.1
F4	LG, 180 um	0.42	5.36	2.59	49.5
F5	AG, 150 um, diluted	0.97	5.41	2.58	79.8
F6	AG, 150 um, undiluted	0.99	6.32	1.75	82.6
F7	AG, 180 um, diluted	0.99	5.43	2.62	78.1
F8	AG, 180 um, undiluted	1.00	6.31	1.84	85.2

Test results indicated that the LG composite was unable to produce 6.0% Li_2O concentrate while the HG composite achieved the target grade (6.0% Li_2O) and recovery (>80%) with a coarser grind (P_{100} of 180 µm). All concentrates had relatively high iron content ranging from 1.75% to 2.65% Fe_2O_3.

12.5.2 Bench-scale Flotation (October 2017)

A 66 kg sample was prepared from drill cores to provide a sample with representative grade and grain size as identified in the core logging process: coarse, medium and fine. The composite assays are shown in Table 12-16. The composite sample head grade was 1.08% Li_2O. HLS and flotation tests were undertaken on the composite sample.

Table 12-16: Composite assays for the October 2017 test program

Sample	Assay (%)									
	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	MnO
Oct. Comp.	0.51	1.08	74.6	15.4	0.70	0.12	0.17	4.26	3.22	0.08

Figure 12-3 shows the grade-recovery curves for the four tests from the October 2017 testing program. All tests were able to produce greater than 6.0% Li_2O concentrate at roughly 80% recovery. Tests using site water showed slightly better results than tests using tap water.

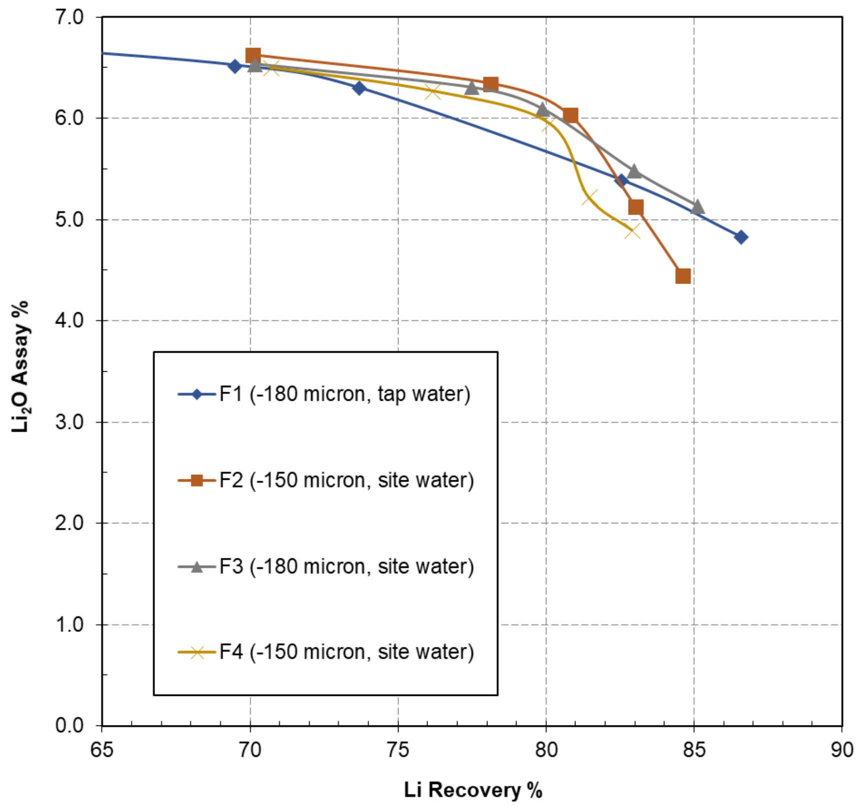


Figure 12-3: Grade-Recovery Curves for the October 2017 Testwork

12.5.3 Heavy Liquid Separation (October 2017)

The aim of the October 2017 HLS testing program was to confirm the results obtained in the 2016 program. Testing examined whether the presence of dilution in the 2016 samples impacted the results. The feed sample used for testing was the same sample as for the October 2017 bench-scale flotation tests (Table 12-17). Samples were stage-crushed and the -6.4 mm / +0.86 mm size fraction was tested. HLS was undertaken using heavy liquids with specific gravity of 2.60, 2.70, 2.80, 2.90, 2.95, 3.00 and 3.10.

There was an improvement in the HLS results as compared to the 2012 testwork program. Using a heavy liquid with sg of 2.95 produced a concentrate grade of 6.16% Li_2O but with low lithium recovery of 13.9% (Table 12-17).

Table 12-17: HLS Combined Sinks Results (October 2017)

HL SG (g/cm^3)	Weight (%)	Assays (%)		Dist. (%)
		Li_2O	Fe_2O_3	Li
3.10	0.8	6.91	1.16	5.0
3.00	1.4	6.66	1.10	8.7
2.95	2.4	6.16	1.08	13.9
2.90	2.8	5.95	1.06	15.4
2.80	8.8	4.56	0.99	37.2
2.70	21.0	3.21	0.88	62.4
2.60	53.3	1.48	0.63	72.7

For upgrading tests (separation at 2.7 g/cm^3), results show lithium recovery of 85% while rejecting more than 69% of the feed mass. If the 2.7 g/cm^3 sinks were combined with the -0.86 mm size fraction, lithium recovery to the flotation feed would be 89.1% with mass rejection of 47%. The process would upgrade flotation feed to 1.82% Li_2O. Although upgrading appears promising, 10.9% lithium loss would be significant. The results indicated that DMS was not a viable option for the project.

12.5.4 Heavy Liquid Separation (December 2017)

The objective of the December 2017 HLS testing program was to test various size fractions. A 120-kg composite sample was prepared from seven drill holes across the Authier deposit. The sample was prepared to reflect the average life-of-mine lithium grade and a representative spodumene grain size across the deposit. The composite assays are shown in Table 12-18.

Table 12-18: Composite Assays for the December 2017 Test Program

Sample	Assay (%)							
	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	CaO	Na_2O	K_2O
Dec. Comp.	0.53	1.14	74.2	15.5	0.69	0.21	4.27	3.22

Historical HLS testwork for the project examined a top crush size of 6.35 mm. Samples for the current study were crushed to a top size of 6.35 mm, 4.75 mm and 3.35 mm. Sub-samples of each were screened at 1,000 µm, 850 µm and 500 µm. As previously, the undersize fractions were not tested. Tests were operated using heavy liquids with sg ranging from 2.6 g/cm^3 to 3.1 g/cm^3. The objective of the HLS testwork was to test finer size fractions to determine if improved separation could be achieved due to increased spodumene liberation.

Results showed that concentrates grading >6.0% Li_2O could be produced using heavy liquid with an sg of 3.0 g/cm^3 with recoveries ranging from ca. 7% - 13% (extrapolation of the data shows the potential for up to ca. 18% lithium recovery).

Tests undertaken using heavy liquid with an sg of 2.7 g/cm^3 showed the potential to upgrade flotation feed. Results showed mass pull ranging from ca. 50% to 62% with 88% to 93% lithium recovery (combined screen undersize and HLS sinks). The grade of the flotation feed stream ranged from 1.6% to 2.0% Li_2O. The testwork results coupled with an economic analysis indicated that DMS was not a viable option for the Authier Project.

12.6 Sayona Québec Pilot Plant Program (2018)

The pilot plant testwork program was undertaken at SGS Canada Inc. facilities from December 2017 to May 2018. SGS received a ca. 5 t sample of drill core from the Authier deposit for testing. The samples were crushed and analysed on a meter-by-meter basis. Two composite samples were prepared to represent 1) Years 0-5 and 2) Years 5+ of operation. The pilot plant feed samples were crushed to 100% passing 3.36 mm. An 80-kg sub-sample from each composite was set aside for batch testing. The testwork program included:

- Feed characterization;
- Grindability tests;
- Batch tests;
- Locked-cycle tests;
- Continuous pilot plant operation;
- Thickening and filtration.

12.6.1 Feed Characterization

Chemical analysis of the two composite pilot plant feed samples are shown in Table 12-19. The head grades of the two composite samples were 1.01% Li_2O and 1.03% Li_2O, respectively. The only significant differences in chemical composition were slightly elevated concentrations of iron and magnesium in Composite 1.

Table 12-19: Chemical Compositions of the Pilot Plant Feed Samples

Analysis	Composite 1 Years 0-5	Composite 2 Years 5+
Li	0.47	0.48
Li_2O	1.01	1.03
SiO_2	73.5	74.9
Al_2O_3	15.6	15.6
Fe_2O_3	0.79	0.56
MgO	0.39	0.10
CaO	0.25	0.17
Na_2O	4.69	4.56
K_2O	2.72	2.95
P_2O_5	0.03	0.02
MnO	0.10	0.09
Cr_2O_3	0.02	0.01
sg	2.71	2.71

Samples of each composite were analyzed by X-ray diffraction (XRD). Results of semi-quantitative mineralogical analysis are shown in Table 12-20. Feldspars (albite and microcline), quartz and spodumene are the major constituents in the samples. The presence of hornblende/ clinochlore and elevated concentrations of biotite in Composite 1 correspond to elevated concentrations of iron and magnesium in the sample (Table 12-19).

Table 12-20: Semi-quantitative XRD Results (Rietveld Analysis)

Mineral	Composite 1 Years 0-5 (wt %)	Composite 2 Years 5+ (wt %)
Albite	36.2	33.9
Quartz	31.1	34.8
Spodumene	11.3	9.7
Microcline	9.6	11.0
Muscovite	4.0	9.3
Hornblende	3.4	-
Biotite	1.6	1.2
Clinochlore	2.7	-
Total	**100**	**100**

A representative crushed (-3.35 mm) sub-sample of each composite was analyzed by QEMSCAN. Each sub-sample was ground to a P_{80} of 425 µm and then screened into five size fractions for characterization: +425 µm, -425/+300 µm, -300/+212 µm, -212/+106 µm, -106 µm. The sample and the five size fractions were analyzed.

Results of spodumene liberation analysis are shown in Table 12-21 and Table 12-22. The amount of free and liberated spodumene was comparable to historical testwork at 66% for Composite 1 and 71% for Composite 2 in the -212 µm/+106 µm size class. A significant portion of the spodumene is associated with quartz and feldspar (31% and 27%, respectively, in the same size class).

Table 12-21: Composite 1 (Years 0-5) Spodumene Liberation Analysis by QEMSCAN

Mineral / Association	Combined	+425 um	-425/+300 um	-300/+212 um	-212/+106 um	-106 um
Free Spodumene	24.5	12.0	18.2	28.2	30.3	60.4
Lib Spodumene	28.0	27.2	28.9	31.4	35.8	12.5
Spd: Quartz/Feldspar	43.9	57.3	48.1	37.0	30.8	24.0
Spd: Phylosilicates	0.11	0.01	0.25	0.02	0.21	0.21
Spd: All Silicates	3.48	3.48	4.62	3.40	2.84	2.76
Complex	0.02	0.01	0.00	0.02	0.06	0.03
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Table 12-22: Composite 2 (Years 5+) Spodumene Liberation Analysis by QEMSCAN

Mineral / Association	Combined	+425 um	-425/+300 um	-300/+212 um	-212/+106 um	-106 um
Free Spodumene	29.7	14.8	24.0	24.6	45.6	64.7
Lib Spodumene	21.8	20.7	23.0	22.5	25.4	19.0
Spd: Quartz/Feldspar	44.5	59.7	48.4	47.3	26.5	15.2
Spd: Phylosilicates	0.18	0.38	0.01	0.03	0.08	0.14
Spd: All Silicates	3.74	4.42	4.47	5.62	2.35	0.96
Complex	0.05	0.01	0.18	0.00	0.00	0.04
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Figure 12-4 shows the optimum grade-recovery curve (perfect separation) for the Composite 1 sample and size fractions. Although spodumene liberation is relatively low (66%) in the -212 / +106 µm size class, the analysis shows the potential to produce a 6.0% Li_2O with high recovery (as a perfect separation would yield 97% recovery). Mineralogical analysis of Composite 2 yielded similar results.

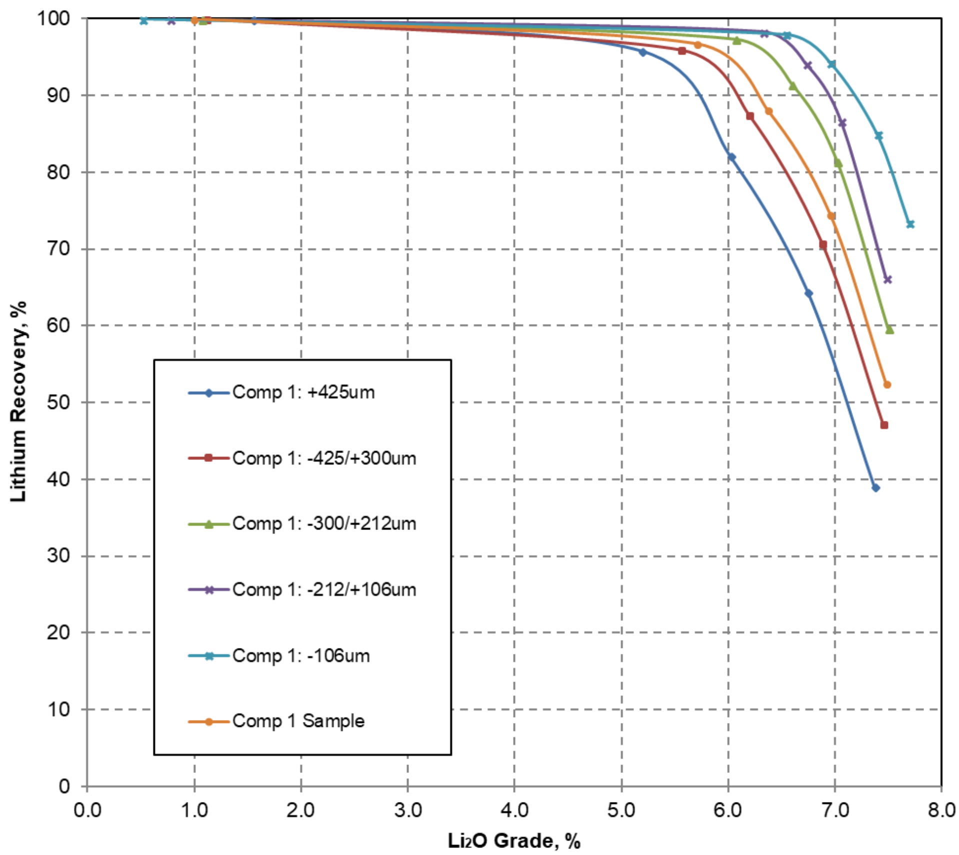


Figure 12-4: Optimum Grade-Recovery Curves for the Composite 1

12.6.2 Grindability

Table 12-23 summarizes the grindability testwork results obtained during the pilot plant program. Six drill core samples were selected for variability grindability testing.

Bond low-energy impact crushing work index (CWI) ranged from 12.1 kWh/t to 19.5 kWh/t (moderately soft to medium range). Bond ball mill work index (BWI) was measured for the six aforementioned samples and for the two composite pilot plant feed samples. Ball mill work index (BWI) ranged from 12.7 kWh/t to 15.8 kWh/t with an average of 14.6 kWh/t ranking the samples as moderately soft to moderately hard. Abrasion index (AI) ranged from 0.806 g to 1.009 g. The material tested is highly abrasive and fell in the 95-98th percentile in the SGS abrasion index database (Figure 12-5).

Table 12-23: Summary of Grindability Results

Sample	Hole no.	CWI (kWh/t)	BWI (kWh/t)	AI (g)
1	AL-17-034 47-49 m	13.0	12.7	0.912
2	AL-17-034 54-56 m	14.7	14.5	0.806
3	AL-17-037 167-171 m	12.1	15.8	0.953
4	AL-17-036 81-83 m	15.8	15.8	1.009
5	AL-17-036 102-104 m	19.5	15.2	1.005
6	AL-17-038 53-54 m	15.0	14.9	0.962
PP1	Composite 1 - Yr 0-5	-	13.7	-
PP2	Composite 2 - Yr 5+	-	14.1	-

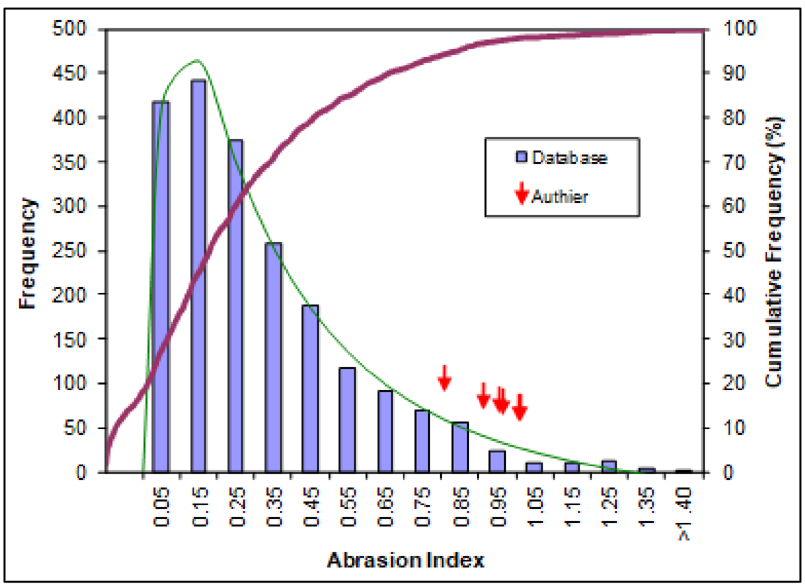


Figure 12-5: Abrasion Index Results Relative to the SGS Database

12.6.3 Bench-scale Flotation Tests

Over forty bench-scale batch tests were operated to confirm and optimize the flowsheet and reagent schemes prior to piloting. Batch tests were undertaken on each composite and included: stage-grinding, magnetic separation (5,000 G and 10,000 G), de-sliming, mica flotation, and spodumene flotation. The batch tests investigated a number of variables (e.g., feed particle size, flowsheet configuration, reagents schemes, spodumene conditioning) to optimize metallurgical performance.

Initial batch tests focused on replicating the results obtained during the October 2017 testwork program. The flowsheet was modified as the testwork program progressed to incorporate successful variations to optimize the flowsheet. The optimized flowsheet that was developed (which was used in tests F37 to F43), is presented in Figure 12-6.

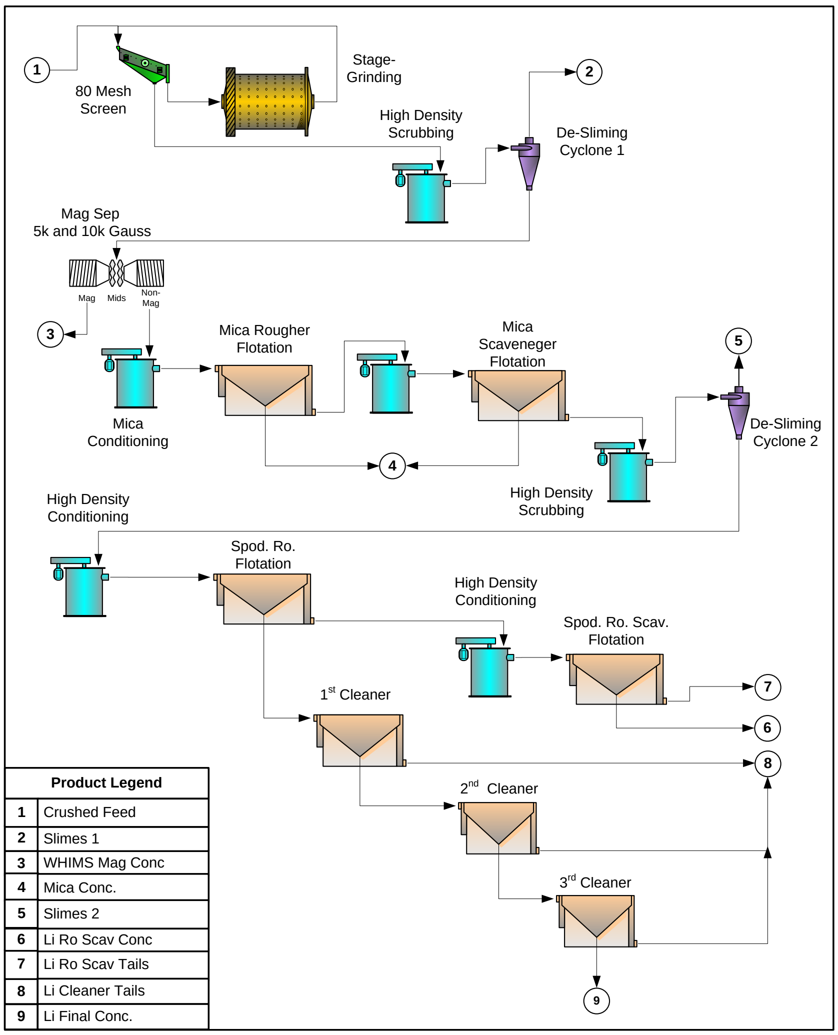


Figure 12-6: Optimized Batch Flowsheet

For the optimized tests, sub-samples of Composite 1 or 2 were stage-ground to 100% passing 180 µm which was selected based on historical testwork, mineralogical data, and batch test results. Samples were initially screened to remove undersize material. The oversize fraction was processed in a 10 kg batch rod mill at about 60% w/w pulp density for a set duration. The ground material was screened to remove the generated undersize fraction. The screen oversize was made up to 10 kg dry equivalent with fresh feed, the grinding-screening procedure was repeated until the weight of the oversize fraction dropped to 4 kg (dry equivalent). At which point a 4 kg batch rod mill was used for grinding. Similarly, a 2 kg mill was used when the oversize weight reached 2 kg (dry equivalent). The grind time was varied in each stage depending on the mass of the remaining oversize fraction. Stage-grinding was complete when almost all the sample was ground to the target size. The ground material was then filtered, dried, homogenized, and split into 2 kg charges for flotation testing.

The stage-ground feed was scrubbed in a Denver D12 4 L flotation cell for 3 min. The scrubbed material was de-slimed in a 12 L cylinder. Water was added to the 11 L mark and solids were allowed to settle for roughly 8 min and then decanted.

The de-slimed material was processed through an Eriez model L-4-20 laboratory-scale Wet High Intensity Magnetic Separator (WHIMS). The material was processed sequentially at 5,000 Gauss and 10,000 Gauss. The magnetic concentrates were filtered, weighted, and submitted for analysis.

The non-magnetic material was transferred to a 4 L Denver D12 flotation cell for mica conditioning. Sodium hydroxide (NaOH) was added to raise the pH to ~10.5 and Armac T (mica collector) and MIBC were added. After 3 min of conditioning, the air was turned on for the mica rougher flotation stage. The mica rougher tailings were reconditioned in the same cell, and after 1 minute the air was turned on for the mica scavenger stage. The mica rougher and scavenger concentrates were filtered, weighted, and submitted for analysis.

The mica scavenger tailings were scrubbed at high density (~65% w/w solids) in the Denver D12 flotation machine for 10 min. The scrubbed material was de-slimed in a 12 L cylinder, allowing the material to settle for roughly 7 min followed by decanting. This procedure was repeated a second time prior to conditioning. The resulting combined slimes were filtered, weighed, and submitted for analysis.

The de-slimed material was conditioned in a 4 L Denver D12 flotation cell at a pulp density of roughly 65% w/w solids. Sylfat FA-2 (spodumene collector) was added and the slurry and conditioned for 5 min. The air to the flotation cell was then opened to commence spodumene rougher flotation. The rougher tailings were decanted to roughly 60% w/w solids and conditioned with additional FA-2 and soda ash (Na_2CO_3) for 3 min prior to the spodumene scavenger flotation stage.

Samples of the concentrate and tailings products from the spodumene scavenger stage were submitted for analysis. The spodumene rougher concentrate was transferred to a 2 L Denver D12 flotation cell for the first stage of cleaning. The 1st cleaner concentrate was placed in a 1 L flotation cell for the 2nd and 3rd cleaner stages. Throughout spodumene flotation stages, soda ash was added to the cell as required to maintain the pulp pH at around 8.5. Tailings from each cleaner stage and the concentrate from the final cleaner stage were submitted for analysis.

Figure 12-7 shows the laboratory WHIMS and Denver D12 flotation machine with the three cell sizes used in the testwork program.

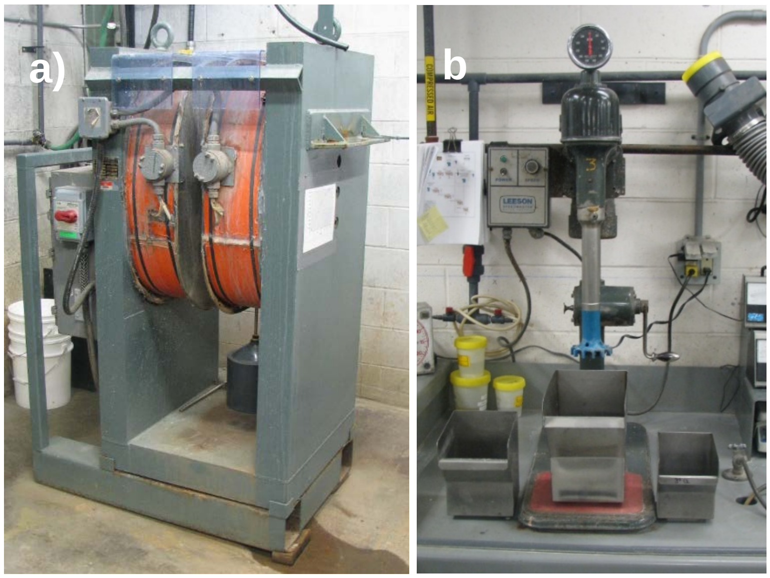


Figure 12-7: Batch Testing Equipment: a) WHIMS, and b) Flotation Machine and Cells

Reagent dosages for the optimized batch tests operated on Composite 1 or Composite 2 are shown in Table 12-24. Armac T dosage ranged from 100 g/t to 110 g/t and FA-2 dosage ranged from 780 g/t to 1,080 g/t. The feed samples for the tests shown in Table 12-24 were stage-ground to 100% passing 180 µm.

Table 12-24: Reagent Dosages for Selected Batch Tests

Feed	Test	Dosage (g/t)					
		NaOH	Na_2CO_3	Armac T	F100	FA-2	Na Silicate
Composite 1	F34	250	300	110	250	1,080	0
	F37	388	150	110	250	1,080	0
	F40	312	125	110	250	780	0
Composite 2	F30	275	175	100	250	1,080	25
	F42	375	162	110	250	980	0
	F43	450	512	110	250	980	0

Figure 12-8 shows the grade-recovery curves for selected batch tests. The results show that 80% lithium recovery was achieved at a concentrate grade of 6.0% Li_2O for both composite samples.

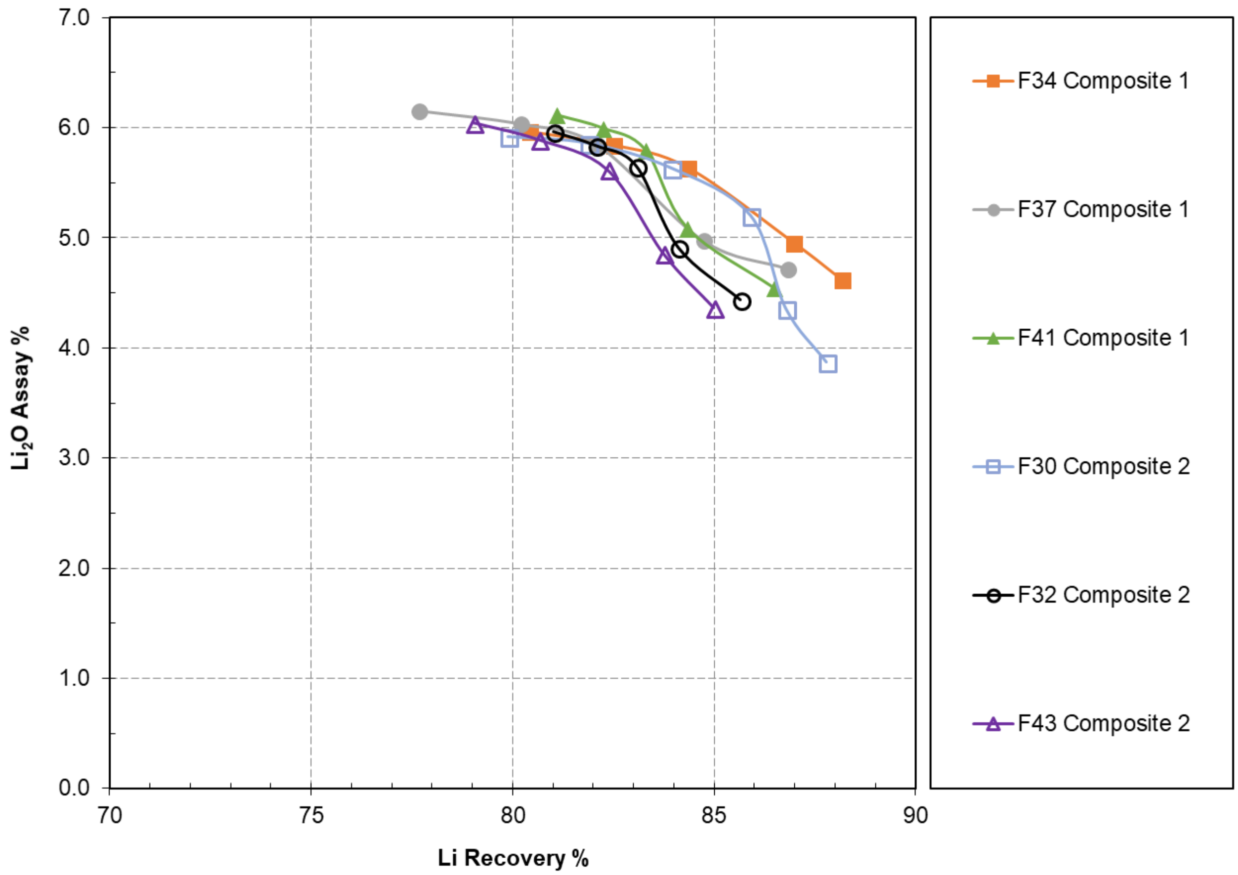


Figure 12-8: Batch Test Grade-Recovery Curves

Table 12-25 shows the detailed results for the optimized batch tests.

Table 12-25: Selected Batch Test Results

Test No.	Combined	Weight		Assays (%)											Distribution (%)									
Objective	Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	CaO	MgO	MnO	P_2O_5	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	CaO	MgO	MnO	P_2O_5	Fe_2O_3
F34	F34 3rd Li Conc.	267.4	13.6	2.77	5.96	63.7	24.7	0.87	0.80	0.38	0.65	0.16	0.05	1.65	80.4	11.7	21.3	4.4	2.3	12.2	21.1	18.7	32.4	24.9
Comp 1	F34 Li 2nd Cl Conc	280.1	14.2	2.71	5.84	63.9	24.5	0.94	0.89	0.37	0.64	0.16	0.05	1.63	82.5	12.3	22.1	4.9	2.7	12.5	21.9	19.2	33.0	25.8
Studying the Effect of Higher Temp in Conditioning	F34 Li 1st Cl Conc.	296.9	15.0	2.62	5.63	64.3	24.2	1.04	1.06	0.36	0.63	0.15	0.05	1.59	84.4	13.1	23.1	5.8	3.4	12.9	22.8	19.8	33.9	26.6
	F34 Li Ro Conc	348	17.7	2.30	4.95	65.6	22.9	1.31	1.62	0.34	0.58	0.14	0.04	1.43	87.0	15.7	25.7	8.6	6.0	14.1	24.6	20.9	36.4	28.1
	F34 Li Ro + Sc Conc	379	19.2	2.14	4.62	66.3	22.3	1.43	1.89	0.33	0.56	0.13	0.04	1.35	88.2	17.3	27.3	10.2	7.7	15.0	25.6	21.5	37.1	28.9
	F34 Mica Conc.	58	2.9	0.26	0.56	58.4	23.6	6.80	2.62	0.22	1.59	0.12	0.03	2.67	1.6	2.3	4.4	7.4	1.6	1.5	11.2	3.1	4.2	8.8
	F34 Mica Ro + Sc Conc.	122.3	6.2	0.23	0.50	63.9	20.0	5.62	3.50	0.30	1.62	0.09	0.05	2.20	3.1	5.4	7.9	12.9	4.6	4.5	24.0	4.7	13.6	15.2
	F34 Li Ro. Tail	1306	66.2	0.02	0.04	78.2	13.3	2.76	5.76	0.17	0.05	0.01	0.01	0.13	2.8	70.2	56.1	67.6	80.4	26.9	7.2	6.2	31.7	9.5
	F34 Li Ro. Sc Tail	1275	64.6	0.01	0.03	78.3	13.3	2.76	5.78	0.17	0.04	0.01	0.01	0.12	1.7	68.7	54.6	66.0	78.7	26.0	6.2	5.6	30.9	8.6
	F34 10A Mags Conc.	36	1.8	0.62	1.32	48.2	17.5	2.05	1.34	1.24	5.60	3.99	0.04	16.14	2.4	1.2	2.0	1.4	0.5	5.3	24.1	61.8	3.4	32.3
	F34 5A Mags Conc.	16	0.8	0.50	1.08	47.9	16.7	1.39	1.59	0.86	3.18	6.07	0.04	20.20	0.9	0.5	0.9	0.4	0.3	1.6	6.2	42.4	1.6	18.2
	F34 Total Slimes	161.2	8.2	0.27	0.58	67.4	15.8	3.16	4.95	2.55	1.02	0.09	0.04	1.64	4.7	7.5	8.2	9.6	8.5	49.2	20.1	6.3	15.0	15.0
	Head (calc.)	1973	100	0.47	1.00	73.7	15.7	2.70	4.74	0.42	0.42	0.12	0.02	0.90	100	100	100	100	100	100	100	100	100	100
	Head (Dir.)			0.47	1.01	73.5	15.6	2.72	4.69	0.25	0.39	0.10	0.03	0.79										
F37	F37 3rd Li Conc.	257.1	13.0	2.86	6.16	63.3	24.6	0.88	0.75	0.36	0.76	0.18	0.05	1.61	77.7	11.1	20.3	4.2	2.0	9.4	23.7	21.3	22.6	24.6
Comp 1	F37 Li 2nd Cl Conc	270.8	13.6	2.80	6.04	63.5	24.4	0.95	0.83	0.35	0.75	0.18	0.05	1.60	80.2	11.8	21.2	4.8	2.4	9.7	24.8	21.9	23.1	25.7
F17 but using Armac T in mica, and higher density in conditioning	F37 Li 1st Cl Conc.	286.3	14.4	2.71	5.84	63.8	24.1	1.05	0.97	0.34	0.74	0.17	0.05	1.57	82.1	12.5	22.2	5.6	2.9	9.9	25.8	22.4	23.7	26.7
	F37 Li Ro Conc	347	17.5	2.31	4.97	65.6	22.6	1.37	1.64	0.31	0.66	0.14	0.04	1.36	84.7	15.6	25.2	8.9	6.0	10.9	27.7	23.0	25.8	28.1
	F37 Li Ro + Sc Conc	375	18.9	2.19	4.72	66.2	22.1	1.47	1.85	0.30	0.63	0.14	0.04	1.31	86.8	17.0	26.7	10.3	7.3	11.5	28.6	23.6	26.8	29.1
	F37 Mica Ro. Conc	42	2.1	0.28	0.60	53.2	18.6	6.09	2.50	0.31	1.67	0.10	0.06	2.18	1.2	1.5	2.5	4.8	1.1	1.3	8.5	1.9	4.6	5.5
	F37 Mica Ro + Sc Conc	79.2	4.0	0.28	0.59	45.2	16.1	4.13	1.78	0.67	3.54	0.86	0.05	7.21	2.3	2.5	4.1	6.1	1.5	5.4	34.0	31.3	6.6	34.0
	F37 Li Ro. Tail	1337	67.4	0.03	0.07	83.0	14.5	2.96	6.16	0.48	0.09	0.01	0.02	0.16	4.3	75.9	62.2	73.8	87.1	65.8	14.7	8.8	52.5	13.0
	F37 Li Ro. Scav. Tail	1309	66.0	0.02	0.03	6.3	1.4	0.24	0.47	0.36	0.06	0.00	0.00	0.08	2.2	5.7	5.8	5.8	6.6	47.2	9.9	2.8	6.5	6.0
	F37 10A Mags	50	2.5	0.62	1.33	88.1	22.6	4.04	5.83	3.06	2.59	1.46	0.04	5.70	3.3	3.0	3.6	3.8	3.1	15.7	15.8	33.9	3.7	17.1
	F37 5A Mags	22	1.1	0.61	1.31	146.6	35.3	7.30	10.93	6.35	2.67	0.18	0.07	4.42	1.4	2.2	2.4	2.9	2.5	13.9	7.0	1.8	2.6	5.7
	F37 Total Slimes	170.8	8.6	0.30	0.64	66.9	15.1	2.76	5.02	3.52	0.82	0.06	0.03	1.15	5.4	7.8	8.3	8.8	9.1	61.1	16.9	4.6	9.0	11.6
	Head (calc.)	1984	100	0.48	1.03	73.7	15.7	2.70	4.77	0.50	0.42	0.11	0.03	0.85	100	100	100	100	100	100	100	100	100	100
	Head (Dir.)			0.47	1.01	73.5	15.6	2.72	4.69	0.25	0.39	0.10	0.03	0.79										
F40	F40 3rd Li Conc.	265.7	13.4	2.80	6.03	64.8	24.8	0.86	0.82	0.30	0.42	0.13	0.06	1.36	78.8	12.0	21.4	4.3	2.3	10.7	13.6	16.0	27.9	20.3
Comp 1	F40 Li 2nd Cl Conc	273.3	13.8	2.76	5.94	65.0	24.6	0.91	0.89	0.30	0.42	0.13	0.06	1.35	79.8	12.3	21.9	4.6	2.6	10.9	13.9	16.3	28.1	20.7
F38 but with 600 g/t Collector Dossage	F40 Li 1st Cl Conc.	287.8	14.5	2.67	5.74	65.3	24.3	0.99	1.04	0.29	0.41	0.13	0.06	1.32	81.3	13.1	22.7	5.4	3.2	11.2	14.5	16.7	28.9	21.3
	F40 Li Ro Conc	336	17.0	2.33	5.03	66.7	23.0	1.24	1.64	0.27	0.38	0.11	0.05	1.19	83.1	15.6	25.1	7.8	5.9	12.3	15.6	17.1	30.6	22.3
	F40 Li Ro + Sc Conc	383	19.3	2.10	4.52	67.7	22.0	1.43	2.03	0.26	0.37	0.10	0.05	1.10	85.2	18.0	27.4	10.2	8.3	13.5	17.1	17.8	32.2	23.7
	F40 Mica Ro. Conc	63	3.2	0.23	0.50	72.3	19.5	6.34	3.97	0.31	1.65	0.08	0.07	1.82	1.5	3.2	4.0	7.5	2.7	2.7	12.7	2.2	7.2	6.5
	F40 Mica Ro + Sc Conc	131.8	6.7	0.23	0.50	44.2	12.7	3.54	2.17	0.42	2.13	0.21	0.04	4.05	3.2	4.1	5.4	8.8	3.0	7.4	34.2	12.6	9.0	30.0
	F40 Li Ro. Tail	1278	64.6	0.03	0.06	77.2	13.1	2.67	5.85	0.31	0.07	0.01	0.02	0.14	3.6	68.6	54.6	64.0	79.8	53.5	11.3	6.8	46.2	9.7
	F40 Li Ro. Sc. Tail	1232	62.2	0.01	0.03	3.3	0.7	0.12	0.25	0.18	0.04	0.00	0.00	0.04	1.6	2.8	3.0	2.8	3.2	30.4	5.3	1.1	3.2	2.8
	F40 10A Mags	50	2.5	0.78	1.67	193.0	48.7	8.96	13.07	3.57	4.73	2.60	0.09	10.48	4.1	6.7	7.9	8.4	7.0	23.9	28.8	60.4	8.1	29.4
	F40 5A Mags	24	1.2	0.77	1.66	345.2	81.5	16.66	25.37	6.29	4.93	0.40	0.15	9.71	2.0	5.9	6.5	7.6	6.6	20.5	14.6	4.6	6.4	13.3
	F40 Total Slimes	183.3	9.3	0.30	0.65	68.3	15.8	3.04	5.04	2.08	0.89	0.07	0.03	1.57	5.9	8.7	9.4	10.4	9.8	51.0	19.9	5.7	9.6	16.1
	Head (calc.)	1980	100	0.48	1.03	72.7	15.5	2.69	4.74	0.38	0.42	0.11	0.03	0.90	100	100	100	100	100	100	100	100	100	100
	Head (Dir.)			0.47	1.01	73.5	15.6	2.72	4.69	0.25	0.39	0.10	0.03	0.79										

Test No.	Combined	Weight		Assays (%)											Distribution (%)									
Objective	Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	CaO	MgO	MnO	P_2O_5	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	CaO	MgO	MnO	P_2O_5	Fe_2O_3
F30	F30 3rd Li Conc (1.5 min)	271.9	13.9	2.75	5.92	64.7	24.3	0.98	0.93	0.23	0.13	0.14	0.04	1.10	79.9	12.1	21.8	4.7	2.8	8.8	18.0	18.4	20.5	22.9
Comp 2	F30 3rd Li Conc.	281.5	14.4	2.72	5.86	64.8	24.2	1.02	0.97	0.23	0.13	0.14	0.04	1.10	81.9	12.5	22.4	5.0	3.1	9.0	18.8	18.8	21.1	23.8
	F30 Li 2nd Cl Conc	300.1	15.3	2.62	5.64	65.2	23.8	1.16	1.12	0.22	0.13	0.13	0.04	1.08	83.9	13.5	23.6	6.1	3.8	9.5	20.5	19.2	21.8	24.9
	F30 Li 1st Cl Conc.	333	17.0	2.42	5.21	66.0	23.1	1.39	1.41	0.22	0.13	0.12	0.04	1.03	85.9	15.1	25.3	8.2	5.3	10.2	22.8	19.5	23.0	26.2
	F30 Li Ro Conc	401	20.5	2.02	4.36	67.7	21.5	1.77	2.01	0.21	0.13	0.10	0.03	0.90	86.8	18.7	28.4	12.5	9.1	11.8	26.3	19.8	25.6	27.7
	F30 Li Ro + Sc Conc	457	23.3	1.80	3.87	68.8	20.6	1.93	2.36	0.20	0.12	0.09	0.03	0.83	87.8	21.6	30.9	15.5	12.1	13.2	28.8	20.6	27.7	29.2
Slight changes to F12	F30 Mica Ro. Conc	46.0	2.3	0.26	0.56	58.2	24.7	7.24	2.44	0.18	0.33	0.10	0.07	2.03	1.3	1.8	3.7	5.9	1.3	1.2	7.7	2.2	6.1	7.1
	F30 Mica Ro + Sc Conc	87	4.4	0.28	0.60	61.2	22.7	6.85	2.82	0.23	0.35	0.10	0.09	1.72	2.6	3.6	6.5	10.4	2.7	2.8	15.6	4.2	14.5	11.4
	F30 Li Ro. Tail	1256	64.2	0.02	0.04	78.4	13.0	2.95	5.48	0.16	0.02	0.01	0.02	0.13	2.4	67.7	53.9	65.3	77.3	28.5	14.8	6.6	47.4	12.5
	F30 Li Ro. Scav. Tail	1201.1	61.4	0.01	0.02	78.5	13.0	2.95	5.50	0.16	0.02	0.01	0.02	0.12	1.4	64.8	51.4	62.3	74.2	27.1	12.2	5.8	45.3	11.0
	F30 10A Mags	29.6	1.5	0.86	1.84	52.8	18.2	1.96	1.78	0.51	1.00	4.52	0.04	16.24	2.7	1.1	1.8	1.0	0.6	2.2	15.1	64.6	2.2	36.8
	F30 5A Mags	16.6	0.8	0.79	1.70	55.4	18.3	1.71	2.00	0.51	0.75	5.76	0.04	12.90	1.4	0.6	1.0	0.5	0.4	1.2	6.3	46.2	1.3	16.4
	Head (calc.)	1958	100	0.48	1.03	74.3	15.5	2.90	4.55	0.36	0.10	0.11	0.03	0.67	100	100	100	100	100	100	100	100	100	100
	Head (Dir.)			0.48	1.03	74.9	15.6	2.95	4.56	0.17	0.10	0.09	0.02	0.56										
F42	F42 3rd Li Conc.	288	14.4	2.77	5.96	64.7	24.1	0.98	1.02	0.21	0.10	0.14	0.04	1.01	81.0	12.7	22.4	4.9	3.3	9.7	13.2	19.7	28.4	22.8
Comp 2	F42 Li 2nd Cl Conc	298	14.9	2.71	5.84	64.9	23.9	1.04	1.11	0.21	0.10	0.14	0.04	1.00	82.1	13.2	23.0	5.4	3.7	10.0	13.9	20.0	28.9	23.4
	F42 Li 1st Cl Conc.	312	15.5	2.63	5.65	65.2	23.6	1.12	1.24	0.21	0.10	0.13	0.04	0.99	83.1	13.8	23.7	6.1	4.3	10.4	14.9	20.3	29.5	24.2
	F42 Li Ro Conc	363	18.1	2.28	4.91	66.8	22.2	1.38	1.79	0.20	0.10	0.12	0.03	0.90	84.1	16.5	26.0	8.7	7.2	11.9	16.7	20.8	30.8	25.4
	F42 Li Ro & Scav Conc	409	20.4	2.06	4.44	67.7	21.3	1.55	2.12	0.21	0.10	0.11	0.03	0.84	85.6	18.9	28.1	11.0	9.6	13.4	19.0	21.5	33.0	26.8
	F42 Mica Ro. Conc	80.8	4.0	0.23	0.50	104.9	21.3	5.49	6.64	0.30	0.18	0.04	0.06	0.58	1.9	5.8	5.5	7.7	5.9	3.9	6.8	1.7	11.3	3.7
F39 but with Composite 2	F42 Mica Ro & Scav. Conc	196	9.7	0.22	0.48	50.2	10.9	2.58	3.03	0.17	0.20	0.18	0.03	1.78	4.4	6.7	6.9	8.8	6.6	5.4	17.9	16.9	13.0	27.2
	F42 Li Ro. Tail	1243	62.0	0.02	0.05	75.5	12.8	2.88	5.31	0.25	0.05	0.01	0.01	0.12	2.7	63.9	51.1	62.2	73.1	49.0	27.3	6.4	30.8	11.3
	F42 Li Ro. Scav. Tail	1198	59.7	0.01	0.02	1.5	0.3	0.07	0.11	0.10	0.01	0.00	0.00	0.01	1.2	1.2	1.3	1.4	1.4	18.4	5.4	0.5	1.4	1.0
	F42 10A Mags	39.1	1.9	0.93	2.00	266.7	65.1	13.69	17.93	4.81	1.49	2.74	0.16	8.45	3.7	7.1	8.2	9.3	7.8	30.1	26.6	52.4	15.2	25.9
	F42 5A Mags	15.8	0.8	0.88	1.89	604.4	141.7	32.19	42.51	11.37	2.80	0.70	0.35	11.11	1.4	6.5	7.2	8.9	7.4	28.7	20.2	5.4	13.6	13.7
	F42 Total Slimes	165.8	8.3	0.30	0.64	68.6	16.0	3.54	4.83	1.78	0.34	0.07	0.04	1.14	5.0	7.7	8.5	10.2	8.9	47.1	25.6	5.9	15.0	14.7
	Head (calc.)	2007	100	0.49	1.06	73.2	15.5	2.86	4.50	0.31	0.11	0.10	0.02	0.64	100	100	100	100	100	100	100	100	100	100
	Head (Dir.)			0.48	1.03	74.9	15.6	2.95	4.56	0.17	0.10	0.09	0.02	0.56										
F43	F43 3rd Li Conc.	278	13.8	2.81	6.05	64.4	24.2	0.92	0.95	0.22	0.10	0.12	0.04	1.04	79.0	12.1	21.6	4.4	2.9	5.9	12.0	16.7	28.6	21.7
Comp 2	F43 Li 2nd Cl Conc	291	14.4	2.74	5.90	64.7	23.9	1.00	1.06	0.22	0.10	0.12	0.04	1.03	80.6	12.7	22.4	5.0	3.4	6.1	12.9	17.1	29.0	22.5
	F43 Li 1st Cl Conc.	312	15.4	2.61	5.62	65.1	23.5	1.12	1.25	0.22	0.11	0.11	0.04	1.01	82.3	13.7	23.5	6.0	4.3	6.5	14.3	17.6	30.0	23.6
	F43 Li Ro Conc	366	18.2	2.26	4.86	66.7	22.1	1.40	1.81	0.21	0.11	0.10	0.03	0.92	83.7	16.5	26.0	8.9	7.3	7.6	16.9	18.5	32.9	25.2
	F43 Li Ro & Sc Conc	414	20.5	2.03	4.37	67.7	21.2	1.58	2.14	0.21	0.11	0.09	0.03	0.87	85.0	18.9	28.2	11.3	9.7	8.5	19.2	18.9	34.1	26.8
	F43 Mica Ro. Conc	82.4	4.1	0.23	0.50	74.3	15.6	4.08	4.68	0.23	0.14	0.03	0.05	0.46	1.9	4.1	4.1	5.8	4.2	1.9	5.1	1.3	10.8	2.8
F41 but with Composite 2	F43 Mica Ro & Sc. Conc	166	8.2	0.22	0.48	44.6	10.1	2.37	2.65	0.17	0.20	0.18	0.03	2.03	3.8	5.0	5.4	6.8	4.8	2.7	14.5	15.2	12.5	25.3
	F43 Li Ro. Tail	1252	62.0	0.02	0.04	77.0	13.0	2.95	5.46	0.27	0.05	0.01	0.01	0.14	2.5	65.1	52.3	64.0	74.6	32.3	25.7	6.7	33.2	12.8
	F43 Li Ro. Sc. Tail	1205	59.7	0.01	0.02	2.4	0.5	0.10	0.17	0.12	0.01	0.00	0.00	0.01	1.2	2.0	2.0	2.1	2.3	13.7	4.8	0.6	2.2	1.1
	F43 10A Mags	50.3	2.5	0.92	1.98	217.1	53.8	11.06	14.29	11.39	1.49	2.25	0.11	7.15	4.7	7.4	8.7	9.6	7.9	55.3	32.5	56.7	13.7	27.0
	F43 5A Mags	27.9	1.4	0.90	1.94	330.6	78.1	17.74	23.28	20.06	1.86	0.35	0.15	6.20	2.5	6.2	7.0	8.6	7.1	54.0	22.5	4.9	10.9	13.0
	F43 Total Slimes	183.3	9.1	0.29	0.62	66.3	15.3	3.36	4.67	3.83	0.34	0.06	0.03	1.02	5.3	8.2	9.0	10.6	9.4	67.7	27.3	5.6	13.1	14.0
	Head (calc.)	2018	100	0.49	1.05	73.4	15.4	2.86	4.54	0.51	0.11	0.10	0.02	0.66	100	100	100	100	100	100	100	100	100	100
	Head (Dir.)			0.48	1.03	74.9	15.6	2.95	4.56	0.17	0.10	0.09	0.02	0.56										

12.6.4 Locked-cycle Tests

A locked-cycle test was performed on each composite sample. The conditions for the tests were based on batch tests F41 and F43. The flowsheet for the locked-cycle tests in shown in Figure 12-9. Feed samples were stage-ground to 100% passing 180 µm. Reagent dosages for the tests are given in Table 12-26. The only differences in the test conditions were the slight increase in Armac T dosage from 110 g/t (Composite 1) to 120 g/t (Composite 2) and the addition of MIBC (10 g/t) ahead of mica flotation for Composite 2.

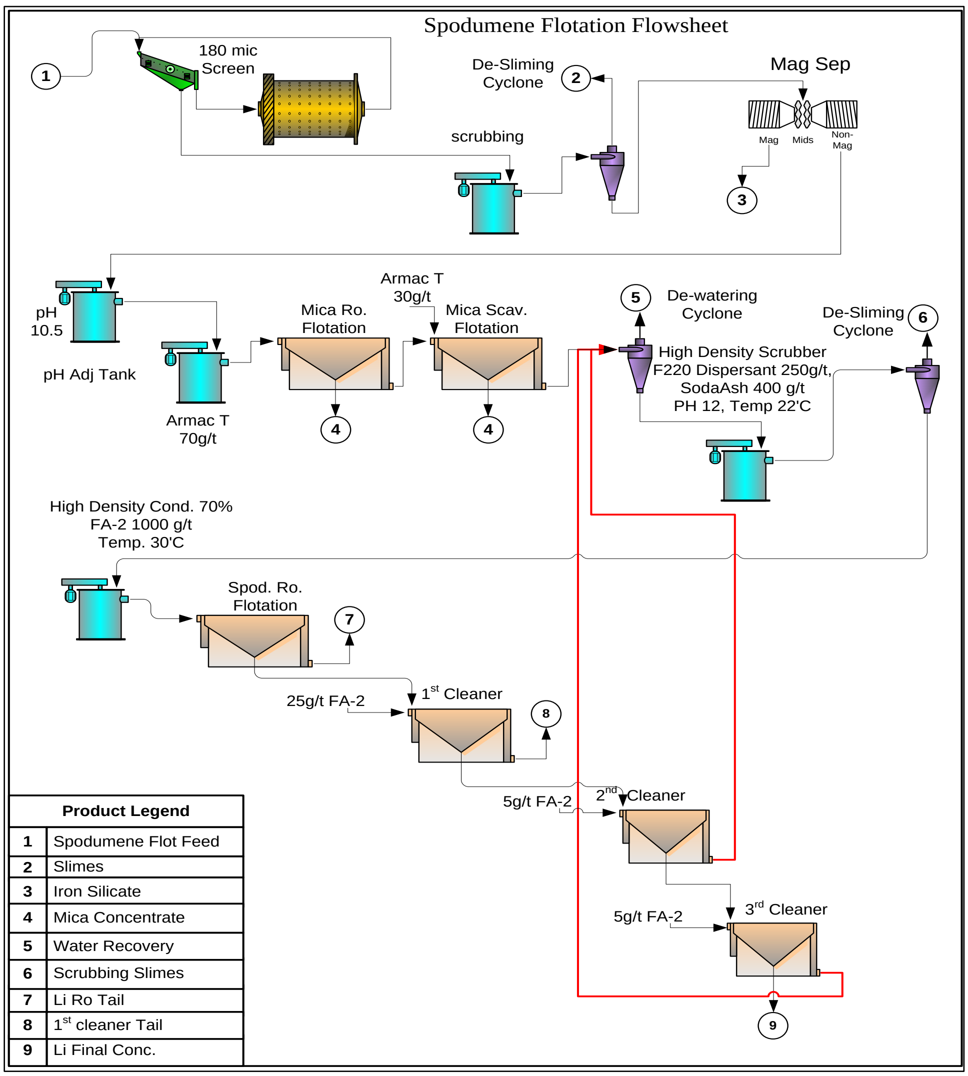


Figure 12-9: Locked-cycle Flowsheet (Composite 1)

Table 12-26: Reagent Dosages for the Locked-cycle Batch Tests

Feed	Dosage (g/t)					
	NaOH	Na_2CO_3	Armac T	MIBC	F100	FA-2
Composite 1	150	600	110	0	250	1035
Composite 2	150	600	120	10	250	1035

Table 12-27 shows the detailed results for the locked-cycle batch tests. Results on Composite 1 and Composite 2 showed an average concentrate grade of 5.85% Li_2O at 84% recovery, and 5.86% Li_2O at 83% recovery, respectively.

Table 12-27: Locked-cycle Test Results

Composite 1 | Projected Balance Cycles B to F																			
Combined	Weight		Assays %									Global Distribution %							
Product	g	%	Li	Li_2O	SiO_2	Al_2O3	K_2O	Na_2O	CaO	MgO	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	CaO	MgO	Fe_2O_3
Li 3rd Cl Conc	1,709.64	14.50	2.72	5.86	63.32	24.52	0.98	0.77	0.42	0.93	1.81	83.8	12.5	22.7	5.3	2.3	14.7	33.6	31.4
Li 1st Cl Tail	417.12	3.54	0.58	1.24	72.89	16.72	3.01	4.57	0.17	0.34	0.57	4.3	3.5	3.8	4.0	3.4	1.5	3.0	2.4
Li Ro Tail	8,106.6	68.74	0.02	0.04	77.74	13.30	2.80	5.76	0.17	0.05	0.05	2.8	72.7	58.5	71.6	83.4	28.5	9.3	4.1
Slime 2	571.56	4.85	0.33	0.72	69.21	15.10	2.71	5.11	2.79	0.60	0.74	3.4	4.6	4.7	4.9	5.2	32.6	7.2	4.3
Mica Ro. Conc.	134.52	1.14	0.26	0.55	57.26	23.79	6.87	2.65	0.24	1.31	2.72	0.6	0.9	1.7	2.9	0.6	0.7	3.7	3.7
Mica Scav. Conc.	304.8	2.58	0.24	0.52	59.19	22.71	6.84	2.61	0.25	1.66	2.66	1.3	2.1	3.8	6.6	1.4	1.6	10.7	8.2
Mag Sep	173.16	1.47	0.57	1.24	42.22	16.90	1.54	0.78	1.34	6.55	21.86	1.8	0.8	1.6	0.8	0.2	4.8	24.0	38.3
Slime 1	376.32	3.19	0.28	0.59	67.52	15.98	3.28	4.92	2.06	1.07	2.00	1.9	2.9	3.3	3.9	3.3	15.9	8.5	7.6
Total Feed	11,793.72	100.00	0.47	1.01	73.50	15.64	2.69	4.74	0.42	0.40	0.84	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Direct Feed			0.47	1.01	73.50	15.60	2.72	4.69	0.25	0.39	0.79								
Combined Slimes		8.04	0.31	0.67	68.54	15.45	2.94	5.03	2.50	0.78	1.24	5.3	7.5	7.9	8.8	8.5	48.4	15.7	11.9
Combined Mag Sep		1.47	0.57	1.23	42.22	16.90	1.54	0.78	1.34	6.55	21.86	1.8	0.8	1.6	0.8	0.2	4.8	24.0	38.3

Composite 2 | Projected Balance Cycles B to F																			
Combined	Weight		Assays %									Global Distribution %							
Product	g	%	Li	Li_2O	SiO_2	Al_2O3	K_2O	Na_2O	CaO	MgO	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	CaO	MgO	Fe_2O_3
Li 3rd Cl Conc	1,701.36	14.59	2.72	5.85	65.07	24.51	0.93	0.93	0.26	0.10	1.09	82.8	12.8	23.0	4.7	3.0	10.5	12.8	24.3
Li 1st Cl Tail	366.72	3.14	0.42	0.90	74.88	15.58	3.08	4.91	0.16	0.10	0.40	2.7	3.2	3.2	3.4	3.4	1.4	2.7	1.9
Li Ro Tail	7,480.08	64.14	0.02	0.05	77.90	13.00	2.88	5.51	0.17	0.06	0.16	3.0	67.4	53.7	63.9	77.7	30.9	31.7	15.2
Slime 2	252.48	2.16	0.41	0.88	66.99	14.79	2.46	4.89	4.85	0.28	0.41	1.9	2.0	2.1	1.8	2.3	29.6	5.3	1.4
Mica Ro. Conc.	396.00	3.40	0.17	0.37	62.94	21.91	7.03	2.93	0.15	0.26	1.42	1.2	2.9	4.8	8.3	2.2	1.4	7.7	7.3
Mica Scav. Conc.	623.88	5.35	0.17	0.37	72.99	15.84	5.17	3.97	0.19	0.15	0.45	1.9	5.3	5.5	9.6	4.7	2.9	7.2	3.6
Mag Sep 10A	115.20	0.99	0.97	2.10	54.76	17.95	2.49	2.04	0.44	1.16	16.39	2.0	0.7	1.1	0.9	0.4	1.2	9.9	24.6
Mag Sep 5A	99.60	0.85	0.84	1.81	56.88	19.55	1.94	2.31	0.54	0.97	9.19	1.5	0.7	1.1	0.6	0.4	1.3	7.2	11.9
Slime 1	627.36	5.38	0.26	0.57	69.78	16.18	3.70	4.96	1.37	0.33	1.19	3.0	5.1	5.6	6.9	5.9	20.7	15.5	9.8
Total Feed	11,662.68	100.00	0.48	1.03	74.08	15.53	2.89	4.55	0.36	0.11	0.66	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Direct Feed			0.48	1.03	74.90	15.60	2.95	4.56	0.17	0.10	0.56								
Combined Slimes		7.54	0.31	0.66	68.98	15.78	3.34	4.94	2.37	0.32	0.97	4.8	7.0	7.7	8.7	8.2	50.3	20.9	11.1
Combined Mag Sep		1.84	0.91	1.96	55.74	18.69	2.23	2.16	0.49	1.07	13.05	3.5	1.4	2.2	1.4	0.9	2.5	17.1	36.6

12.6.5 Continuous Pilot Plant Tests

The concentrator pilot plant was operated by SGS Canada Inc. in Lakefield, Ontario. Pilot plant operation commenced on April 5, 2018, and concluded on April 26, 2018, in a series of thirteen (13) campaigns (operational shifts). Three feed samples were tested: a low-grade commissioning sample, Composite 1 and Composite 2. The commissioning sample was initially fed to the pilot plant to confirm mechanical reliability, robust operating procedures and analytical laboratory capabilities. Once commissioning was complete, the two composite pilot plant samples were processed through the plant. The plant operated for over 100 h and processed over 5 t of feed material.

The flowsheet for continuous pilot plant testing campaign PP06 is shown in Figure 12-10. The circuit was fed at a rate of 50 kg/h of crushed ore (-3.36 mm) to a rod mill (Hazen Quinn 16" x 32") in closed-circuit with a 180 µm vibrating screen. The screen undersize fed a dewatering hydrocyclone.

The cyclone underflow fed the magnetic separation circuit which consisted of an Eriez countercurrent LIMS and a Slon 750II WHIMS unit in series. The magnetic concentrates were combined and sent to tailings. The non-magnetic fraction fed a de-sliming cyclone. The slimes were sent to tailings while the underflow stream fed the mica-conditioning tank (17 L) where sodium hydroxide and Armac T collector were added. The conditioning tank overflowed to feed three Denver A5 (7.7 L) mica rougher cells. The rougher tails fed a second conditioning tank (11 L) where supplemental Armac T was added prior to feeding the three Denver A5 (7.7 L) mica scavenger cells. The mica rougher and scavenger concentrates were combined and sent to tailings.

The mica scavenger tails were de-watered and fed to the attrition scrubber (70 L) where sodium hydroxide and dispersant (Pionera F100) were added prior to de-sliming. The cyclone underflow was collected and thickened (to ~60% w/w solids) and fed to spodumene conditioning. The thickening stage was implemented due to the small-scale of the pilot hydrocyclone (1") which were prone to plugging under the testing conditions.

The slurry was conditioned in a 20 L tank with double impeller with Sylfat FA-2 (spodumene collector) and soda ash. The conditioning tank overflowed to feed four Denver A5 (7.7 L) spodumene rougher cells.

Rougher tailings were discarded. The first and second cleaners were both single Denver D12 machines. Soda ash was added to spodumene flotation cells as required to maintain the pulp pH at 8.5. Final concentrate was collected separately on a shift basis.

Images of the pilot plant grinding circuit, LIMS/WHIMS, mica and spodumene flotation circuits are shown in Figure 12-11 and Figure 12-12.

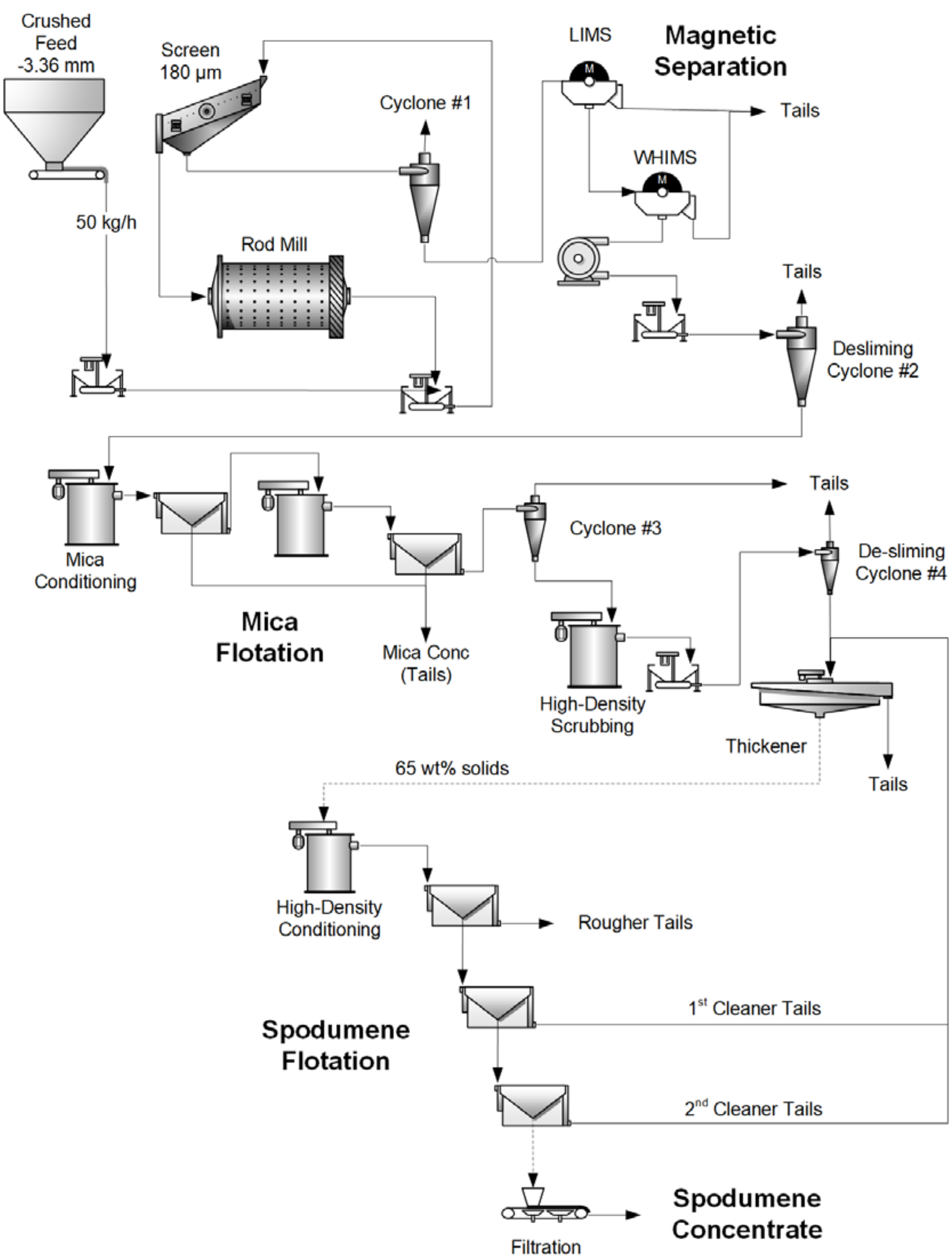


Figure 12-10: Pilot Plant Flowsheet (PP-06)

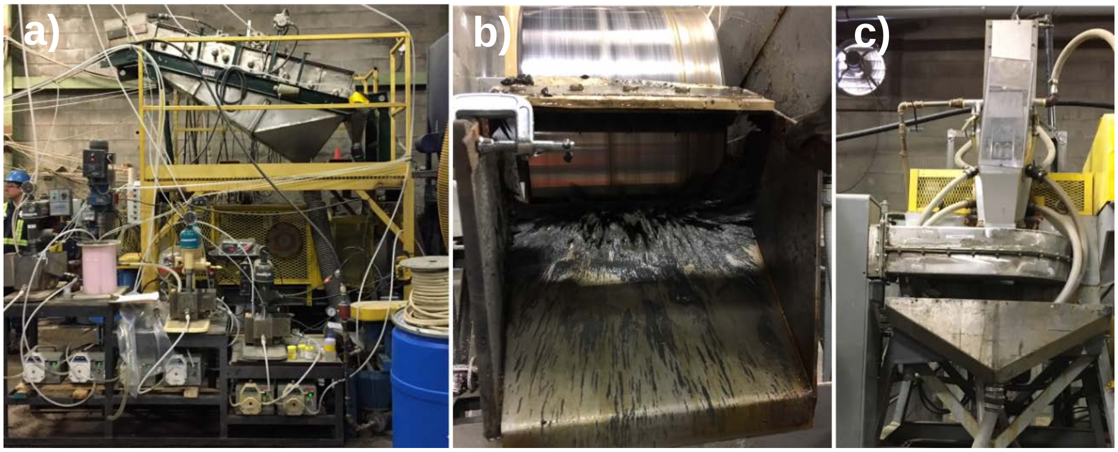


Figure 12-11: Pilot Plant a) Grinding, b) LIMS, and c) WHIMS

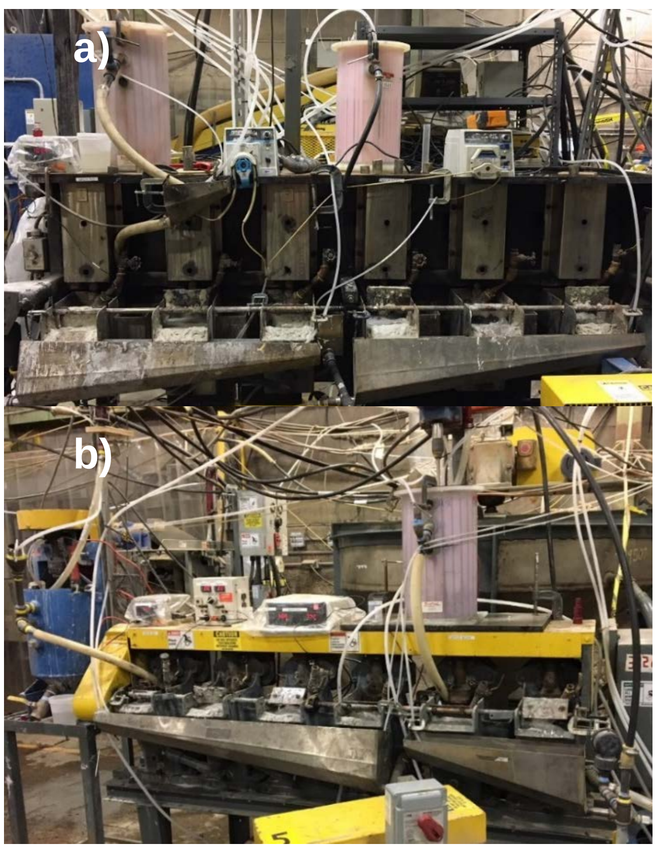


Figure 12-12: Pilot Plant a) Mica and b) Spodumene Flotation Circuits

Reagent dosages for the optimized pilot plant campaigns are shown in Table 12-28. For the optimized conditions, Armac T dosage ranged from 112 g/t to 220 g/t and FA-2 dosage ranged from 656 g/t to 1,106 g/t.

Table 12-28: Reagent Dosages for Selected Pilot Plant Tests

Test	Feed	P_{80} (µm)	Dosage (g/t)				
			Na_2CO_3	Armac T	MIBC	F100	FA-2
PP-11S	Composite 1	188	576	130	21	254	693
PP-11F		188	576	130	21	254	693
PP-12F		189	543	220	21	266	656
PP06	Composite 2	180	402	112	19	242	1,065
PP-07S1		182	600	121	19	264	1,106
PP-07S2		182	600	212	19	264	1,106

Note: NaOH consumption was not measured during pilot plant operation

Pilot plant mass balance data was reconciled using Bilmat software. Reconciled data for the selected campaigns is summarized in Table 12-29. For the optimized flowsheets, pilot plant operation on Composite 1 produced concentrate ranging from 5.9% to 6.0% Li_2O with recoveries ranging from 67% to 71%. Fe_2O_3 content in the spodumene concentrates ranged from 1.70% to 1.89%. For Composite 2, concentrate grade ranged from 5.8% to 6.2% Li_2O with lithium recovery from 73% to 79%. Fe_2O_3 content in the spodumene concentrates ranged from 0.96% to 1.16%.

Table 12-29: Selected Pilot Plant Mass Balances

Composite 1 | PP-11S Mass Balance																	
Stream	Mass Pull (%)	Assay (%)															
		Li_2O		SiO_2		Al_2O_3		Fe_2O_3		MgO		CaO		Na_2O		K_2O	
		Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.
Feed	100	1.06	1.08	72.9	73.3	15.7	15.9	0.89	0.82	0.39	0.42	0.30	0.23	4.67	4.69	2.66	2.69
Combined Mag Conc	3.7	2.11	2.11	58.8	58.8	19.5	19.5	6.29	6.42	3.44	3.39	0.69	0.70	2.16	2.16	2.52	2.52
Combined Mica Conc	3.6	0.21	0.21	67.7	67.7	17.9	17.9	0.98	0.98	0.50	0.50	0.22	0.22	3.45	3.45	6.44	6.43
Combined Slimes	14.5	0.69	0.69	70.1	70.0	15.9	15.9	1.60	1.64	0.65	0.64	0.72	0.79	5.02	5.02	3.19	3.18
Spod Ro Tails	65.6	0.20	0.21	76.6	76.6	13.7	13.6	0.27	0.26	0.08	0.07	0.17	0.19	5.54	5.57	2.64	2.61
Spod Cl Conc	12.6	5.95	5.96	62.3	62.3	24.5	24.5	1.70	1.71	0.77	0.76	0.39	0.41	0.80	0.81	1.08	1.08

Streams	Mass Pull (%)	Recovery (%)							
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Combined Mag Conc	3.7	7.3	3.0	4.6	26.2	32.8	8.6	1.7	3.5
Combined Mica Conc	3.6	0.7	3.4	4.1	4.0	4.7	2.7	2.7	8.8
Combined Slimes	14.5	9.4	14.0	14.7	26.3	24.4	35.5	15.6	17.4
Spod Ro Tails	65.6	12.4	68.9	57.0	19.6	13.1	36.8	77.8	65.2
Spod Cl Conc	12.6	70.2	10.7	19.6	24.0	25.0	16.5	2.2	5.1

PP-11F Mass Balance																	
Stream	Mass Pull (%)	Assay (%)															
		Li_2O		SiO_2		Al_2O_3		Fe_2O_3		MgO		CaO		Na_2O		K_2O	
		Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.
Feed	100	1.01	0.99	73.5	72.8	15.5	15.6	0.62	1.10	0.29	0.43	0.27	0.26	4.70	4.69	2.73	2.71
Cyclone #1 O/F	2.9	0.62	0.62	70.4	70.4	16.3	16.3	2.54	2.48	0.76	0.75	0.41	0.41	5.04	5.04	3.30	3.30
Cy#1 U/F	97.1	1.03	0.99	73.6	73.2	15.5	15.6	0.57	0.98	0.27	0.41	0.26	0.25	4.69	4.68	2.71	2.71
Combined Mag Conc	3.6	1.41	1.42	74.4	74.4	16.0	16.0	0.61	0.60	0.22	0.22	0.21	0.21	4.67	4.67	2.32	2.32
Non Mags	93.5	1.01	1.01	73.6	73.5	15.5	15.5	0.56	0.49	0.28	0.28	0.26	0.22	4.69	4.74	2.73	2.66
Cyclone #2 O/F	2.9	0.64	0.65	72.5	72.5	15.7	15.7	0.73	0.72	0.47	0.46	0.32	0.32	5.14	5.14	3.14	3.14
Cy#2 U/F	90.6	1.02	1.01	73.6	73.8	15.5	15.5	0.56	0.61	0.27	0.26	0.26	0.22	4.68	4.70	2.71	2.65
Combined Mica Conc	18.6	0.16	0.16	73.3	73.5	14.8	14.8	0.58	0.54	0.41	0.37	0.19	0.21	4.23	4.22	4.65	4.74
Mica Tail	72.0	1.25	1.27	73.7	74.2	15.7	15.7	0.55	0.46	0.23	0.22	0.28	0.22	4.79	4.79	2.21	2.19
Cyclone #3 O/F	1.9	1.12	1.12	69.5	69.5	16.5	16.5	1.16	1.16	0.68	0.67	0.91	0.95	4.96	4.96	2.54	2.54

PP-11F Mass Balance																	
Stream	Mass Pull (%)	Assay (%)															
		Li_2O		SiO_2		Al_2O_3		Fe_2O_3		MgO		CaO		Na_2O		K_2O	
		Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.
Cy#3 U/F	70.1	1.25	1.29	73.8	74.3	15.6	15.7	0.54	0.53	0.22	0.22	0.26	0.21	4.79	4.83	2.20	2.17
Cyclone #4 O/F	1.4	1.12	1.12	61.7	61.7	14.5	14.5	0.76	0.76	0.66	0.65	5.10	7.16	4.32	4.32	2.02	2.02
Cy#4 U/F	68.7	1.25	1.29	74.1	74.0	15.7	15.8	0.53	0.61	0.21	0.22	0.16	0.22	4.80	4.82	2.21	2.14
Ro Conc	16.2	5.34	5.29	63.3	63.3	23.7	23.6	1.62	1.53	0.78	0.74	0.33	0.35	1.18	1.19	1.42	1.44
Spod Ro Tail	56.5	0.25	0.26	76.6	76.6	13.7	13.7	0.27	0.25	0.08	0.07	0.12	0.18	5.66	5.57	2.46	2.56
1st Cl Conc	13.8	5.73	5.74	62.7	62.7	24.3	24.3	1.69	1.66	0.82	0.79	0.35	0.36	0.91	0.93	1.15	1.17
1st Cl Tail	2.4	3.11	3.14	66.7	66.7	20.7	20.7	1.23	1.24	0.58	0.59	0.22	0.22	2.73	2.72	2.96	2.94
Spod Cl Conc	12.2	5.90	5.91	62.4	62.3	24.5	24.6	1.74	1.72	0.84	0.86	0.36	0.38	0.78	0.76	1.04	1.03
2nd Cl Tail	1.6	4.46	4.49	65.0	65.0	22.4	22.4	1.32	1.33	0.63	0.64	0.26	0.26	1.85	1.84	1.99	1.98
Spod Feed	72.7	1.39	1.25	73.6	73.7	16.0	15.7	0.57	0.56	0.24	0.21	0.17	0.21	4.66	4.64	2.23	2.37
Combined Slimes	9.1	0.81	0.81	69.5	69.5	15.9	15.9	1.41	1.38	0.64	0.63	1.22	1.56	4.94	4.94	2.89	2.89

Streams	Mass Pull (%)	Recovery (%)							
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cyclone #1 O/F	2.9	1.8	2.8	3.1	11.9	7.7	4.5	3.1	3.5
Cy#1 U/F	97.1	98.2	97.2	96.9	88.1	92.3	95.5	96.9	96.5
Combined Mag Conc	3.6	5.0	3.6	3.7	3.5	2.8	2.8	3.6	3.1
Non Mags	93.5	93.2	93.6	93.2	84.6	89.5	92.7	93.3	93.4
Cyclone #2 O/F	2.9	1.8	2.8	2.9	3.3	4.7	3.4	3.1	3.3
Cy#2 U/F	90.6	91.4	90.7	90.3	81.2	84.9	89.3	90.2	90.1
Combined Mica Conc	18.6	2.9	18.6	17.8	17.4	26.5	13.3	16.8	31.7
Mica Tail	72.0	88.4	72.2	72.5	63.8	58.4	76.0	73.4	58.4
Cyclone #3 O/F	1.9	2.1	1.8	2.0	3.5	4.4	6.4	2.0	1.7
Cy#3 U/F	70.1	86.4	70.4	70.6	60.3	54.0	69.6	71.4	56.6
Cyclone #4 O/F	1.4	1.6	1.2	1.3	1.7	3.3	27.4	1.3	1.1
Cy#4 U/F	68.7	84.8	69.2	69.2	58.5	50.7	42.2	70.1	55.6
Ro Conc	16.2	85.5	14.0	24.8	42.1	44.0	20.3	4.1	8.5
Spod Ro Tail	56.5	13.9	58.9	50.0	24.6	15.2	25.6	68.1	50.9
1st Cl Conc	13.8	78.1	11.8	21.6	37.4	39.1	18.2	2.7	5.8
1st Cl Tail	2.4	7.5	2.2	3.2	4.8	4.9	2.0	1.4	2.6
Spod Cl Conc	12.2	70.9	10.3	19.2	33.9	35.5	16.6	2.0	4.7
2nd Cl Tail	1.6	7.2	1.4	2.3	3.4	3.6	1.6	0.6	1.2

Streams	Mass Pull (%)	Recovery (%)							
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
Spod Feed	72.7	99.4	72.8	74.8	66.8	59.2	45.8	72.1	59.4
Combined O/F	9.1	7.3	8.6	9.3	20.5	20.0	41.6	9.6	9.6

PP-12 Mass Balance																	
Stream	Mass Pull (%)	Assay (%)															
		Li_2O		SiO_2		Al_2O_3		Fe_2O_3		MgO		CaO		Na_2O		K_2O	
		Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.
Feed	100	0.99	0.97	73.3	72.9	15.6	15.6	0.80	1.09	0.38	0.43	0.28	0.26	4.64	4.68	2.75	2.77
Cyclone #1 O/F	3.9	0.56	0.56	69.3	69.3	16.3	16.3	2.46	2.41	0.86	0.85	0.43	0.43	5.06	5.06	3.39	3.39
Cy#1 U/F	96.1	1.01	0.99	73.4	73.2	15.5	15.6	0.74	0.96	0.36	0.41	0.27	0.24	4.62	4.64	2.72	2.73
Combined Mag Conc	2.5	2.00	2.00	57.7	57.7	19.6	19.6	7.44	6.93	3.77	3.63	0.75	0.76	2.03	2.03	2.58	2.58
Non Mags	93.5	0.98	0.92	73.8	73.9	15.4	15.4	0.55	0.49	0.27	0.26	0.26	0.21	4.69	4.75	2.72	2.72
Cyclone #2 O/F	2.9	0.56	0.56	72.5	72.5	15.6	15.6	0.63	0.63	0.44	0.44	0.30	0.30	5.17	5.17	3.18	3.18
Cy#2 U/F	90.6	0.99	0.95	73.9	74.0	15.4	15.4	0.55	0.51	0.26	0.24	0.26	0.21	4.67	4.73	2.71	2.69
Combined Mica Conc	16.2	0.15	0.15	72.9	73.0	14.9	14.9	0.69	0.68	0.44	0.43	0.19	0.21	4.18	4.16	4.81	4.81
Mica Tail	74.4	1.18	1.27	74.1	74.6	15.5	15.5	0.52	0.47	0.22	0.20	0.27	0.22	4.78	4.71	2.25	2.20
Cyclone #3 O/F	1.8	1.01	1.01	67.9	67.9	16.3	16.3	1.31	1.31	0.80	0.80	1.39	1.49	4.87	4.87	2.64	2.64
Cy#3 U/F	72.6	1.18	1.27	74.2	74.8	15.5	15.6	0.50	0.60	0.21	0.22	0.25	0.21	4.78	4.75	2.24	2.24
Cyclone #4 O/F	1.6	1.05	1.05	64.9	64.9	15.0	15.0	0.83	0.83	0.69	0.69	3.93	5.06	4.53	4.53	2.24	2.24
Cy#4 U/F	71.0	1.19	1.33	74.5	73.9	15.5	15.5	0.50	0.53	0.20	0.21	0.16	0.22	4.78	4.68	2.24	2.19
Ro Conc	15.5	5.39	5.61	62.8	62.9	23.9	23.9	1.71	1.63	0.79	0.82	0.37	0.38	1.06	1.08	1.41	1.43
Spod Ro Tail	59.8	0.30	0.30	76.8	76.8	13.8	13.9	0.24	0.22	0.08	0.08	0.12	0.18	5.54	5.60	2.48	2.48
1st Cl Conc	13.4	5.76	5.55	62.3	62.4	24.4	24.3	1.80	1.70	0.82	0.81	0.40	0.41	0.82	0.85	1.08	1.09
1st Cl Tail	2.1	3.11	3.12	66.1	66.1	21.0	21.0	1.15	1.15	0.62	0.62	0.18	0.18	2.54	2.52	3.50	3.48
Spod Cl Conc	11.2	5.91	5.89	62.0	61.9	24.5	24.6	1.87	1.90	0.85	0.85	0.42	0.44	0.71	0.69	0.95	0.94
2nd Cl Tail	2.2	4.99	5.03	63.9	63.9	23.5	23.5	1.44	1.46	0.69	0.69	0.26	0.26	1.37	1.35	1.73	1.72
Spod Feed	75.4	1.35	1.25	73.9	73.7	15.9	15.7	0.54	0.56	0.23	0.21	0.17	0.21	4.62	4.64	2.26	2.37
Combined Slimes	10.2	0.71	0.72	69.3	69.3	15.9	15.9	1.49	1.47	0.70	0.70	1.10	1.29	4.98	4.98	3.02	3.02

Streams	Mass Pull (%)	Recovery (%)							
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cyclone #1 O/F	3.9	2.2	3.7	4.1	12.0	8.9	6.1	4.3	4.9
Cy#1 U/F	96.1	97.8	96.3	95.9	88.0	91.1	93.9	95.7	95.1
Combined Mag Conc	2.5	5.1	2.0	3.2	23.5	25.2	6.9	1.1	2.4

Streams	Mass Pull (%)	Recovery (%)							
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
Non Mags	93.5	92.7	94.3	92.7	64.5	65.9	87.1	94.6	92.8
Cyclone #2 O/F	2.9	1.6	2.9	2.9	2.3	3.4	3.1	3.2	3.4
Cy#2 U/F	90.6	91.0	91.4	89.8	62.2	62.5	84.0	91.4	89.4
Combined Mica Conc	16.2	2.4	16.2	15.6	13.9	18.6	10.9	14.6	28.5
Mica Tail	74.4	88.6	75.2	74.2	48.3	43.9	73.1	76.7	60.9
Cyclone #3 O/F	1.8	1.9	1.7	1.9	3.0	3.9	9.1	1.9	1.8
Cy#3 U/F	72.6	86.7	73.5	72.3	45.4	40.0	64.0	74.8	59.2
Cyclone #4 O/F	1.6	1.7	1.4	1.5	1.6	2.8	22.0	1.5	1.3
Cy#4 U/F	71.0	85.1	72.2	70.8	43.7	37.2	41.9	73.3	57.9
Ro Conc	15.5	84.6	13.3	23.8	33.0	32.5	20.5	3.5	8.0
Spod Ro Tail	59.8	18.3	62.7	53.2	17.8	12.2	25.0	71.5	54.0
1st Cl Conc	13.4	77.9	11.4	20.9	29.9	29.0	19.0	2.4	5.3
1st Cl Tail	2.1	6.7	1.9	2.9	3.1	3.5	1.4	1.2	2.7
Spod Cl Conc	11.2	66.8	9.5	17.6	25.9	25.0	17.0	1.7	3.9
2nd Cl Tail	2.2	11.1	1.9	3.3	4.0	4.0	2.1	0.7	1.4
Spod Feed	75.4	102.9	76.0	77.0	50.8	44.7	45.4	75.1	62.0
Combined O/F	10.2	7.4	9.7	10.5	18.9	19.0	40.3	11.0	11.3

Composite 2 | PP-06 Mass Balance															
Stream	Mass Pull (%)	Assay (%)													
		Li_2O		SiO_2		Al_2O_3		Fe_2O_3		CaO		Na_2O		K_2O	
		Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.
Feed	100	1.05	1.08	74.6	74.5	15.5	15.7	0.64	0.58	0.22	0.16	4.53	4.60	2.85	2.88
Combined Mag Conc	2.5	2.28	2.28	64.2	64.2	19.3	19.3	4.82	4.90	0.31	0.31	2.67	2.67	2.32	2.32
Combined Mica Conc	13.6	0.15	0.15	75.9	75.9	14.0	14.0	0.44	0.45	0.11	0.12	4.08	4.07	4.79	4.78
Combined Slimes	11.8	0.75	0.75	70.5	70.5	16.0	16.0	1.52	1.56	0.80	0.91	4.77	4.76	3.46	3.45
Spod Ro Tails	59.8	0.20	0.19	77.5	77.9	13.7	13.4	0.22	0.21	0.12	0.14	5.42	5.49	2.69	2.73
Spod Cl Conc	12.3	6.19	6.08	65.2	64.8	24.4	24.6	1.22	1.30	0.23	0.25	0.88	0.83	1.01	0.96

Streams	Mass Pull (%)	Recovery (%)						
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	CaO	Na_2O	K_2O
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Combined Mag Conc	2.5	5.4	2.1	3.1	18.6	3.4	1.5	2.0
Combined Mica Conc	13.6	2.0	13.8	12.3	9.5	7.0	12.2	22.9

Streams	Mass Pull (%)	Recovery (%)						
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	CaO	Na_2O	K_2O
Combined Slimes	11.8	8.5	11.1	12.2	28.0	42.9	12.4	14.3
Spod Ro Tails	59.8	11.6	62.2	53.1	20.5	33.7	71.5	56.5
Spod Cl Conc	12.3	72.6	10.7	19.3	23.5	13.0	2.4	4.3

PP-07S1 Mass Balance															
Stream	Mass Pull (%)	Assay (%)													
		Li_2O		SiO_2		Al_2O_3		Fe_2O_3		CaO		Na_2O		K_2O	
		Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.	Adj.	Meas.
Feed	100	0.95	1.05	74.4	75.1	15.5	15.7	0.62	0.50	0.24	0.15	4.60	4.57	2.89	2.92
Combined Mag Conc	2.3	1.94	1.94	61.6	61.6	19.4	19.4	6.03	6.35	0.38	0.39	2.58	2.58	2.46	2.46
Combined Mica Conc	13.2	0.18	0.17	73.0	72.9	15.5	15.5	0.54	0.56	0.13	0.14	4.24	4.24	5.24	5.23
Combined Slimes	9.9	0.58	0.58	69.3	69.2	16.0	16.0	1.21	1.27	1.14	1.49	4.93	4.93	3.61	3.61
Cyclone #4 U/F	74.6	1.10	1.23	75.7	75.3	15.4	15.4	0.39	0.38	0.13	0.15	4.68	4.75	2.40	2.41
Flot Feed	79.0	1.24	1.59	75.3	74.5	15.7	16.1	0.42	0.42	0.13	0.15	4.57	4.40	2.39	2.34
Ro Conc	17.0	5.35	5.07	66.1	66.5	23.4	23.1	1.09	1.09	0.22	0.21	1.44	1.59	1.41	1.54
Spod Ro Tails	62.1	0.12	0.10	77.8	78.4	13.5	13.2	0.24	0.28	0.11	0.13	5.42	5.59	2.65	2.67
Cl Tails	4.4	3.60	3.55	69.1	69.0	20.6	20.6	0.96	0.96	0.20	0.20	2.66	2.57	2.17	2.11
Spod Cl Conc	12.5	5.97	5.93	65.1	64.9	24.4	24.4	1.13	1.22	0.22	0.24	1.01	0.97	1.15	1.10

Streams	Mass Pull (%)	Recovery (%)						
		Li_2O	SiO_2	Al_2O_3	Fe_2O_3	CaO	Na_2O	K_2O
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Combined Mag Conc	2.3	4.7	1.9	2.9	22.3	3.7	1.3	2.0
Combined Mica Conc	13.2	2.4	12.9	13.1	11.4	7.2	12.1	23.8
Combined Slimes	9.9	6.1	9.3	10.3	19.3	48.2	10.7	12.4
Cyclone #4 U/F	74.6	86.7	75.9	73.7	47.0	40.9	75.9	61.8
Flot Feed	79.0	103.5	80.0	79.6	53.8	44.6	78.5	65.1
Ro Conc	17.0	95.6	15.1	25.5	29.6	15.6	5.3	8.3
Spod Ro Tails	62.1	7.9	65.0	54.1	24.2	29.0	73.2	56.8
Cl Tails	4.4	16.8	4.1	5.9	6.8	3.7	2.6	3.3
Spod Cl Conc	12.5	78.8	11.0	19.7	22.8	11.9	2.8	5.0

PP-07S2															
Stream	**Mass Pull (%)**	**Assay (%)**													
		Li_2O		**SiO_2**		**Al_2O_3**		**Fe_2O_3**		**CaO**		**Na_2O**		**K_2O**	
		Adj.	**Meas.**	**Adj.**	**Meas.**	**Adj.**	**Meas.**	**Adj.**	**Meas.**	**Adj.**	**Meas.**	**Adj.**	**Meas.**	**Adj.**	**Meas.**
Feed	100	0.99	1.10	74.5	74.2	15.5	15.6	0.58	0.49	0.21	0.15	4.64	4.58	2.80	2.90
Combined Mag Conc	2.4	1.99	1.98	61.9	61.9	19.6	19.6	5.86	6.11	0.39	0.40	2.58	2.58	2.54	2.54
Combined Mica Conc	14.0	0.19	0.19	74.4	74.4	15.2	15.2	0.27	0.28	0.12	0.13	4.55	4.56	4.83	4.79
Combined Slimes	11.8	0.67	0.67	71.6	71.6	16.0	16.0	1.16	1.21	0.68	0.77	4.96	4.97	3.52	3.50
Cyclone #4 U/F	71.7	1.17	1.38	75.4	75.8	15.4	15.5	0.37	0.38	0.14	0.15	4.67	4.65	2.30	2.23
Flot Feed	76.3	1.27	1.51	75.2	74.9	15.6	15.9	0.39	0.43	0.14	0.15	4.59	4.49	2.30	2.28
Ro Conc	17.9	5.08	4.88	67.1	67.4	22.7	22.4	1.07	1.04	0.23	0.23	1.61	1.69	1.44	1.48
Spod Ro Tails	58.4	0.10	0.08	77.6	77.7	13.4	13.1	0.19	0.19	0.12	0.14	5.50	5.71	2.56	2.60
Cl Tails	4.5	2.86	2.84	70.8	70.7	19.0	19.0	0.78	0.78	0.18	0.18	3.25	3.18	2.31	2.29
Spod Cl Conc	13.4	5.83	5.78	65.9	65.7	24.0	24.1	1.16	1.20	0.25	0.26	1.05	1.03	1.15	1.13

Streams	**Mass Pull (%)**	**Recovery (%)**						
		Li_2O	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**CaO**	**Na_2O**	**K_2O**
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Combined Mag Conc	2.4	4.9	2.0	3.1	24.4	4.6	1.3	2.2
Combined Mica Conc	14.0	2.7	14.0	13.7	6.6	8.2	13.7	24.1
Combined Slimes	11.8	8.0	11.4	12.2	23.5	38.6	12.7	14.9
Cyclone #4 U/F	71.7	84.4	72.6	71.0	45.6	48.6	72.3	58.8
Flot Feed	76.3	97.6	77.0	76.6	51.6	52.5	75.5	62.6
Ro Conc	17.9	91.9	16.2	26.3	32.8	19.7	6.2	9.2
Spod Ro Tails	58.4	5.7	60.8	50.3	18.8	32.8	69.2	53.3
Cl Tails	4.5	13.1	4.3	5.6	6.1	3.9	3.2	3.8
Spod Cl Conc	13.4	78.8	11.8	20.7	26.7	15.8	3.0	5.5

Continuous pilot plant operation produced roughly 400 kg of spodumene concentrate. Concentrate from each campaign (operating shift) was individually collected. The spodumene concentrate produced during pilot plant campaign PP-11 was analyzed by QEMSCAN. The mineralogical composition of the concentrate sample is presented in Table 12-30.

Table 12-30: Mineralogical Analysis of PP11 Spodumene Concentrate

Mineral	Composite 1 Years 0-5 (wt %)
Spodumene	77.9
K-Feldspar	7.1
Plagioclase	5.5
Quartz	3.3
Biotite	2.2
Muscovite	1.0
Amphibole/Pyroxine	0.8
Fe-Al Silicate	0.7
Chlorite	0.8
Other	0.7
Total	**100**

12.6.6 Thickening and Filtration

Thickening and filtration testwork was undertaken at Pocock Industrial Inc. in Salt Lake City, Utah during April 2018. Concentrate and tailings samples from SGS pilot plant operation were tested.

The testwork included:

- Materials characterization;
- Flocculant screening, rheology, static and dynamic thickening on the tailings stream;
- Vacuum and pressure filtration testing on the concentrate and tailings streams.

The objective of the testwork was to provide design data for industrial thickening and filtration equipment.

Flocculant screening tests were performed on the tailings sample. Flocculants were evaluated based on their ability to provide the best overall performance with respect to overflow clarity, decantation rates, and underflow viscosity characteristics for the slurry. SNF FO4140 (medium-high molecular weight, 5% charge density, cationic polyacrylamide) was selected as the optimal flocculant. The static and dynamic tests developed a general set of data for tailings thickener design that included optimum flocculent dosage requirement, feed solids concentration as well as the underflow and overflow characteristics.

Static tests showed the tailings stream required an approximate flocculent dosage of 160 g/t SNF FO4140 to produce reasonable overflow clarity and thickening performance. At feed concentrations ranging from 5% to 20% w/w solids, conventional thickener unit area ranged from 0.158 m^2/tpd to 0.193 m^2/tpd to produce the maximum recommended underflow density of 57% w/w solids (based on rheology data). Details of the static test results are shown in Table 12-31.

Table 12-31: Static Thickening: Recommended Conventional Tailings Thickener Operating Parameter Ranges

Flocculant Type and Dosage	Feed Solids (% w/w)	Rise Rate (m^3/m^2h)	Unit Area (m^2/tpd)	Underflow Density (% w/w)
160 g/t SNF FO4140	5.0	28.6	0.174	57
	10	19.6	0.193	
	15	9.8	0.158	
	20	4.8	0.191	

Dynamic tests were performed with 5% w/w solids feed concentration. The tests showed that in-line flocculation showed acceptable flocculation efficiency and settling performance at a flocculent dosage ranging from 160 g/t to 180 g/t SNF FO4140. The recommended maximum design hydraulic loading rate required for effective separation was 6.0 m^3/m^2h (Table 12-32).

Table 12-32: Dynamic Thickening: Recommended High Rate Tailings Thickener Operating Parameter Ranges

Feed Solids (% w/w)	Flocculant	Dosage (g/t)	Design Basis Net Feed Loading (m^3/m^2h)	Predicted TSS (mg/L)	Underflow Density (% w/w)
5.1	SNF FO4140	160 - 180	6.0	150 - 250	57

Vacuum and pressure filtration tests were performed on thickened tailings underflow and as-received (unthickened) spodumene concentrate slurries to establish data for a horizontal belt vacuum filter and standard recessed plate type pressure filter design. Membrane (squeeze) type pressure filters were also evaluated.

Vacuum Filtration

Vacuum filtration tests examined the effect of cake thickness and air-dry time on the production rate and filter cake moisture. A 20 cfm/ft^2 monofilament calendared polypropylene cloth was used for testing and produced a relatively clean filtrate that cleared up within a second. The cloth also provided very good discharging properties once the cake was dried.

Table 12-33 shows a summary of the predicted production rates and operational design parameters for a horizontal vacuum belt filter.

Table 12-33: Vacuum Filtration Data

Sample	Feed Solids (% w/w)	Dry Bulk Cake Dens. (kg/m^3)	Cake Thick. (mm)	Form Time (min)	Dry Time (min)	Cycle Time (min)	Cake Moisture (% w/w)	Predicted Prod. Rate (dry kg/m^{2}h)
Thickened Tailings	57	1269	15	0.31	0.95	1.27	26.8	722
					1.90	2.22	26.0	412
					2.86	3.17	25.1	288
					3.81	1.12	24.3	222
Concentrate	18	1155	15	0.26	0.87	1.13	8.4	739
					1.73	1.99	7.1	417
					2.60	2.86	6.5	291
					3.47	3.73	6.0	223

Under the test conditions, the tailings and concentrate filter cake moisture contents ranged from 24.3% to 26.8% w/w, and from 6.0% to 8.4% w/w, respectively, depending upon the dry time. For both the tailings and concentrate, the moisture contents were above the acceptable limits for the project and as such, the use of vacuum filtration was discounted from consideration for the industrial installation.

Pressure Filtration

Pressure filtration tests examined the effect of cake thickness and air-dry time on production rate and filter cake moisture for the thickened tailings and concentrate (unthickened) samples. The effect of membrane squeezing on cake moisture was also examined. Two operational scenarios were tested; air blow only and light membrane squeeze during air blow followed by a full-pressure membrane squeeze.

During the tests, 80 psig driving force was used for fill and air blow cycles. For the conditions that utilized squeeze, 100 psig squeeze was applied until the last 30 s of air blow when the squeeze pressure was increased to 232 psig to complete the cycle.

An 8-10 cfm/ft^2 mono-multi-polypropylene cloth provided the best performance for both tailing and concentrate samples. It produced a slightly cloudy filtrate that cleared up within a second and provided good discharging properties once the cake was dried.

The squeezing procedure provided only slight improvements in terms of final cake moisture content and as such was not considered for the industrial installations. Table 12-34 shows the pressure filtration data for the air blow only tests. The air dry only produced very good discharge and stacking properties at reasonable dry times (for both samples). Cake cracking did not seem to be a problem for either material at the simulated design conditions. The production rates achieved for the tailing and concentrate tests were high enough to suggest that pressure filtration would be feasible.

Table 12-34: Pressure Filtration Data (Air Blow Only Tests)

Sample	Feed Solids (% w/w)	Dry Bulk Cake Dens. (kg/m^3)	Design Cake Thickness	Sizing Basis (m^3/t)	Design Cake Moisture (%)	Cycle Time (min)	Vol. Prod. Rate (tpd/m^3)	Area Basis Prod. Rate (tpd/m^2)
Thickened Tailings	57	1286	60 mm	0.972	12.2	12	102.9	2.99
					10.2	15	82.3	2.40
					9.2	18	68.6	2.00
					8.5	21	58.8	1.71
Concentrate	18	1573	60 mm	0.794	6.5	12	125.9	3.66
					5.6	15	100.7	2.93
					5.1	18	83.9	2.44
					4.7	21	72.0	2.09

Design air dry cycles for these types of materials are typically in the range of 3 to 5 min. The design options showing air blow durations greater than 3 min were provided to illustrate the relationship between air blow duration and cake moisture. Extended dry times could make these options cost prohibitive. As such, a 3 min dry time (12 min cycle time) was selected for equipment design which yielded moisture contents of 12.2% w/w for the tailings filter cake and 6.5% w/w for the concentrate filter cake.

12.6.7 Summary of 2018 Pilot Plant Testwork Program

The 2018 pilot plant program confirmed the flowsheet and design parameters for the Authier Lithium Project process plant. Testwork confirmed:

- Grind size (P_{80}) of 180 µm
- Magnetic separation was necessary to remove iron-bearing silicate minerals prior to flotation
- Mica flotation using Armac T collector and frother (MIBC) with a rougher-scavenger circuit arrangement
- High-density (>60% solids) conditioning with fatty acid collector was a key process variable required to achieve >75% recovery
- Spodumene scavenger flotation circuit was not required
- Two-stages of spodumene cleaning were needed to a spodumene concentrate grade of 6.0% Li_2O with >75% lithium recovery at a grade
- Spodumene concentrate thickening was not required
- Thickening and filtration testwork confirmed flocculant type/dosage and equipment sizing
- Pressure filtration is necessary for tailings and concentrate to achieve the desired moisture contents

12.7 Sayona Québec Batch Optimization Test Program (2018)

A sub-sample of each of the two pilot plant feed samples (Composite 1 and Composite 2) were tested during the optimization test program undertaken at SGS in 2018. The program included sample preparation, stage-grinding, wet high-intensity magnetic separation, and flotation.

The main objectives of the program were to:

- Study the effect of magnetic field strength for iron-bearing silicate waste rejection
- Determine optimal pulp density during spodumene conditioning
- Test the effect of spodumene collector dosage on concentrate lithium grade and recovery

The lithium grades of Composite 1 and Composite 2 were similar, at 1.03% Li_2O and 1.08% Li_2O, respectively. The iron content in Composite 1 was higher (0.77% Fe_2O_3) than that of Composite 2 (0.46% Fe_2O_3). The metallurgical target was the production of a concentrate grading 6.0% Li_2O with 80% lithium recovery.

The samples were stage-ground to a K_{80} of 180 µm for the beneficiation testwork. Wet high-intensity magnetic separation (WHIMS) testwork on both Composite 1 and Composite 2 indicated that a significant proportion of the iron in the samples could be rejected by magnetic separation. At the maximum magnetic field strength tested (16,000 Gauss), roughly 55% of the iron in

Composite 1 and 45% of the iron in Composite 2 reported to the magnetic concentrate, along with about 12% of the lithium in the samples. To minimize lithium losses (to ca. 5%) while still rejecting a significant proportion of iron (~35-40%), magnetic intensities of 3,000 Gauss and 5,000 Gauss were selected for two-pass WHIMS prior to flotation (magnetic intensities were significantly lower than those previously tested (10,000 Gauss) in the pilot plant testwork program, possibly due to the modified procedure used).

A preliminary flotation test was conducted on Composite 1 using similar conditions to those used in optimized laboratory flotation tests during the pilot plant program. The results of this test were similar to those of the baseline test, producing of a 3rd cleaner concentrate grading 6.03% Li_2O with 77.2% lithium recovery.

Multiple batch tests were undertaken (desliming, magnetic separation, mica flotation, scrubbing) were undertaken (under the same conditions) and the mica tailings streams were homogenized to form a single spodumene flotation feed sample to eliminate variations due to the upstream processes. Lithium losses to the combined slimes, magnetic concentrate, and mica concentrate were similar for both Composite 1 and Composite 2, averaging 6.1%, 3.3%, and 2.6%, respectively.

The spodumene flotation tests on Composite 1 and Composite 2 evaluated the impact of conditioning pulp density and collector dosage on flotation performance. Table 12-35 summarizes the properties of the concentrates from each spodumene flotation test at the stage where on-spec concentrate (>6% Li_2O) was generated.

Table 12-35: Summary of On-Spec Concentrate from the Optimization Testwork Program

Sample	Flotation product	Conditioning Pulp Density (%)	Collector Dosage (g/t)	Assays (%)			Recovery (%)
				Li_2O	SiO_2	Fe_2O_3	Li_2O
Composite 1	Head	-	-	1.03	73.4	0.77	-
	F1-Spod 1 3rd Cl.Conc	55	800	6.07	63.1	1.69	70.3
	F1-Spod 2 2nd Cl.Conc	60	800	6.14	63.1	1.66	74.9
	F1-Spod 3 2nd Cl.Conc	65	800	6.17	63.3	1.68	75.6
	F1-Spod 4 2nd Cl.Conc	50	800	6.05	63.6	1.79	58.0
	F1-Spod 5 1st Cl.Conc	60	900	6.01	63.2	1.65	76.9
	F1-Spod 6 2nd Cl.Conc	60	600	6.05	62.7	1.70	71.9
	F1-Spod 7 3rd Cl.Conc	60	700	5.92	62.9	1.73	64.3
Composite 2	Head	-	-	1.08	74.2	0.46	-
	F2-Spod 1 1st Cl.Conc	55	800	6.13	64.8	1.13	74.3
	F2-Spod 2 1st Cl.Conc	60	800	6.10	64.7	1.08	76.6
	F2-Spod 3 1st Cl.Conc	65	800	6.18	64.5	1.10	79.2
	F2-Spod 4 1st Cl.Conc	50	800	6.07	65.4	1.09	70.8
	F2-Spod 5 2nd Cl.Conc	60	900	6.08	64.6	1.14	75.4
	F2-Spod 6 2nd Cl.Conc	60	600	6.15	64.6	1.18	68.4
	F2-Spod 7 1st Cl.Conc	60	700	6.06	64.7	1.15	74.7

The results for both samples showed a significant improvement in concentrate lithium recovery was obtained when the conditioning pulp density was increased from 50% to 55% solids. Further increases in pulp density resulted in more marginal increases in lithium recovery for both composites (Figure 12-13 and Figure 12-14).

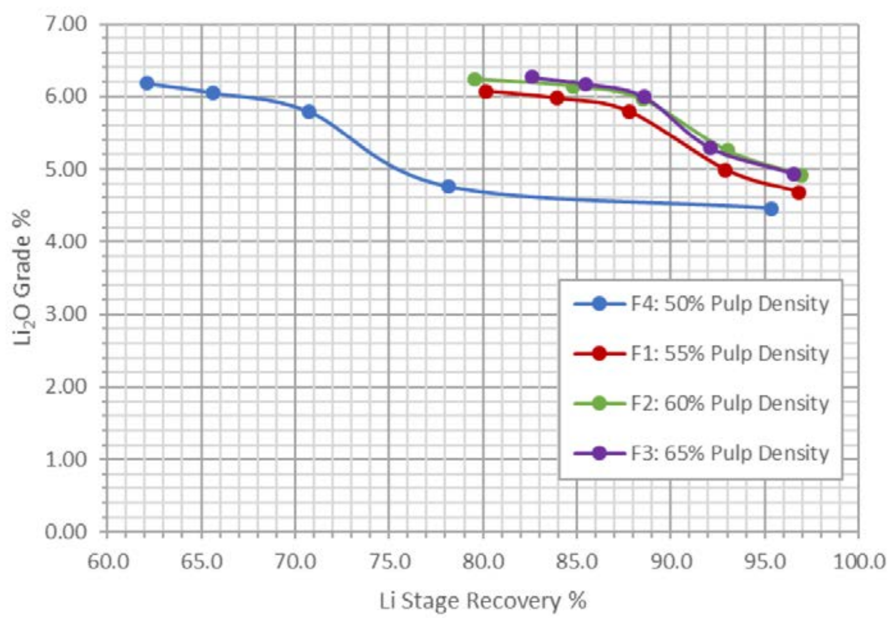


Figure 12-13: Effect of Pulp Density during Spodumene Conditioning (Composite 1)

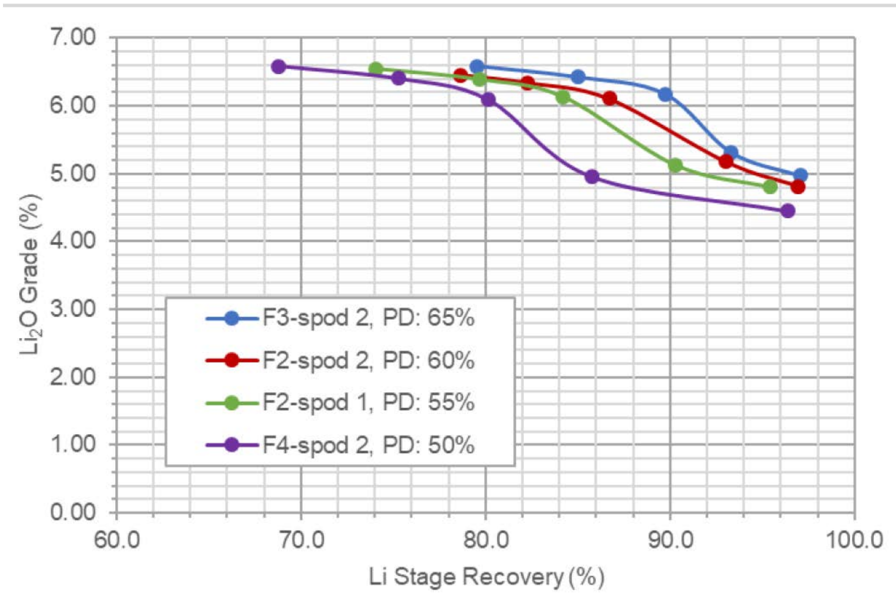


Figure 12-14: Effect of Pulp Density during Spodumene Conditioning (Composite 2)

A conditioning pulp density of 60% w/w solids was selected for the tests evaluating collector dosage; from the results of these tests, it was determined that a spodumene collector dosage of 800 g/t was sufficient to produce a concentrate grading >6% Li_2O with high lithium recovery (>75%) from both composites.

It can be seen from Table 12-35 that on-spec concentrate was typically generated from the 1st cleaner stage in the case of Composite 2. For Composite 1, on-spec concentrate was typically generated from the 2nd cleaner stage; however, in test F1-Spod 2 and F1-Spod 3 (both at near-optimum conditions), the 1st cleaner concentrate graded only slightly below 6% Li_2O (5.96-5.99% Li_2O). These results suggest that possibly the 2nd cleaner stage and definitely the 3rd cleaner flotation stages are not required to produce on-spec concentrate from these samples.

The iron content was higher in the on-spec spodumene concentrates produced in the flotation tests on Composite 1 (average 1.70% Fe_2O_3) compared to those produced in the flotation tests on Composite 2 (average 1.12% Fe_2O_3).

13. MINERAL RESOURCE ESTIMATES

13.1 Introduction

This section reports the results of June 6, 2018 Updated Mineral Resources Estimates ("MRE") for the Authier deposit and Authier North deposit as per the following Table 13-1.

Table 13-1: Authier and Authier North JORC Mineral Resources Estimate (0.55% Li_2O Cut-off Grade) Inclusive of Reserves

Category	Tonnes (Mt)	Grades (% Li_2O)	Contained Li_2O (t)
Measured	6.58	1.02	67,116
Indicated	10.60	1.01	107,060
Measured and Indicated	17.18	1.01	174,176
Inferred	3.76	0.98	36,848

The Mineral Resources have been estimated by Gustavo Delendatti, PhD., for Sayona Québec. Dr. Delendatti is a professional geologist, member of Australian Institute of Geoscientist and has worked in exploration for gold, silver, copper, lead, zinc, tin, lithium and graphite. Dr. Delendatti is an independent consultant, and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which it is undertaking to qualify as an independent Competent Person.

During the MRE process, different assumptions were made. These assumptions were used in order to calculate modelling cut-off grades and resources cut-off grades following the "reasonable prospect for economic extraction". An optimised pit shell prepared by BBA for this DFS was made over the Authier deposit and used for this updated resource estimation.

13.2 Sayona Authier Deposit Database

Sayona Québec conducted the current Mineral Resources Estimate for the Authier deposit using an updated validated database which incorporates diamond drilling programs completed by Sayona in 2016, 2017 and 2018. The database also includes validated historical drilling data from Glen Eagle programs between 2010 and 2012. The Glen Eagle drilling database was also validated by SGS Geostat for the MRE released in November 19, 2013 by Glen Eagle.

The database used to produce the MRE is derived from a total of 225 holes completed at the Authier property, including:

- 81 historical;
- 69 drilled by Glen Eagle's between 2010-2012; and
- 75 drilled by Sayona between 2016 and 2018.

And contains the survey collar location, lithology, and analytical results information.

The database cut-off date is April 24, 2018 (refer to the Data Verification section for a summary information of the records contained in the final drill hole database).

From this database, 199 drill holes were used for the solid modeling and MRE. The MRE is derived from a computerized resource block model. The construction of the block model starts with the modeling of 3D wireframe solids of the mineralization using drill hole Li_2O grade (%) analytical results and lithological data. The solids from the past mineral resources estimation were updated to fit the new data and interpretation were changed in certain sections of the deposit given the new data from the 2018 infill drilling and exploration. The analytical data contained within the wireframe solids was normalized to generate fixed length analytical composites. The composite data was used to interpolate the block grades. Blocks were regularly spaced on a defined grid, filling the 3D selected wireframe solids. An optimized pit shell model was produced by BBA in 2018. The interpolated blocks located below the bedrock/overburden interface, within the optimized pit shell and above a determined cut-off grade constitute the mineral resources. The blocks are then classified based on confidence level using proximity to composites, composite grade variance and mineralized solids geometry.

13.2.1 Analytical Data

There are a total of 5,049 assay intervals in the database used for the current MRE and 2,456 of them are contained inside the mineralized solids. Most of the drill hole intervals defining the mineralized solids have been sampled continuously. Table 13-2 shows the range of Li_2O (%) values from the analytical data.

Table 13-2: Range of Analytical Data Inside Mineralized Solids

	Li_2O (%)
Count	2,456
Mean	0.92
Std. Dev.	0.54
Min	0.00
Median	0.95
Max	2.77

Assays received in Li values were transformed into % Li_2O values using the conversion factor of 2.153. This conversion factor was used upon conversations with Sayona and peers, according to sources such as the ministry of petroleum and mines of British Columbia: http://www.empr.gov.bc.ca/Mining/Geoscience/MINFILE/ProductsDownloads/MINFILEDocumentation/CodingManual/Appendices/Pages/VII.aspx.

The core holes drilled on the project are generally oriented south (163° to 194°), perpendicular to the general orientation of the pegmatite intrusions, and have a weak to moderate deviation toward the west). Their spacing is typically 25 m with larger spacing of 50 m spacing between sections 706750 mE and 707975 mE. The drill hole dips range from 43° to 75° with an average of 50° and the drill hole intercepts range from approximately 70% of true width to near true width of the mineralization.

13.2.2 Mineralized Intervals Data

Mineralized intervals were selected for the modeling of the 3D wireframe. A minimum grade of 0.4% Li_2O over a minimum drill hole interval length of 2 m was generally used as guideline to define the width of mineralized interpretations on sections i.e. polygons. Only Pegmatite intervals were kept even if there were good results either on the footwall or hanging wall of the pegmatite.

13.2.3 Composite Data

Block model grade interpolation is conducted on composited analytical data. A 1.5 m composite calculated length has been selected based on the average thickness of 1 m (2016-2018 1.0 m assay lengths and previous 1.5 m assays lengths. Compositing is conducted within the downhole mineralized intervals that were also used for 3D solid creation. A maximum of 1.5 m and a minimum of 0.5 m were applied to composite creation settings. Calculated length composites are created by finding the appropriate length of the composites to fit entirely the mineralized intervals. The calculated length was established for every interval trying to be as close as possible of the specified 1.5 m lengths. No capping was applied on the analytical composite data.

Table 13-3 shows the statistics of the analytical composites used for the interpolation of the resource block model and Figure 13-1 and Figure 13-2 show the related histogram for Li_2O grade (%). Figure 13-3 and Figure 13-4 display the spatial distribution of the composites in plan and longitudinal view respectively (hole collars are shown as blue circles and sample composites are shown as black diamonds).

Three populations could be outlined and interpreted within the mineralized wireframe and therefore the block model.

High grade lithia correlates with high concentrations of spodumene crystals, mostly earlier mid-to-coarse grain pulses (higher than 0.4 cm length being coarse grain higher than 0.7 cm length) and subordinated mid-to-fine grain spodumene (lower than 0.4 cm length crystals). Zones with lower concentration of spodumene crystals generally shows lower Li_2O grade, mostly mid-to-fine grained and likely related with later pulses of mineralisation.

Lower grade lithium zones are generally crosscutting earlier and higher grade mineralisation pulses within the pegmatite or in the edges of the pegmatite, in the contact zone with the wall rock.

Table 13-3: Statistics for the 1.5-m Composites for Li_2O (%)

	Li_2O (%)
Count	2,488
Mean	0.93
Std. Dev.	0.49
Min	0.00
Median	0.96
Max	2.61

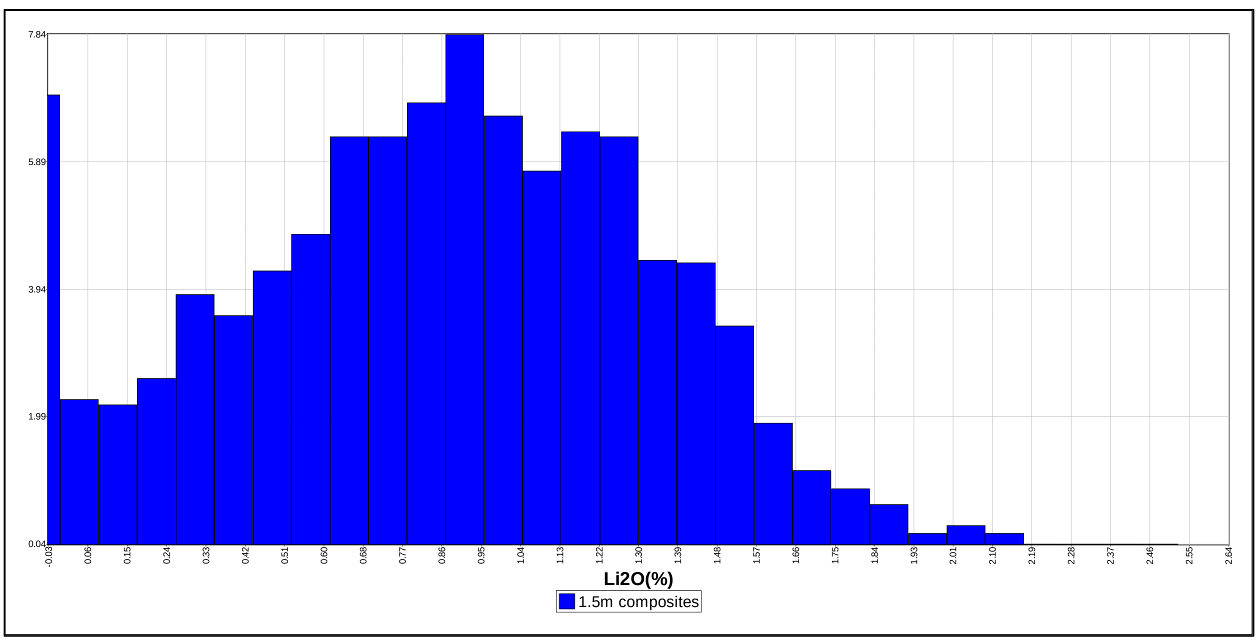


Figure 13-1: Histogram 1.5 m Composites Li_2O (%)

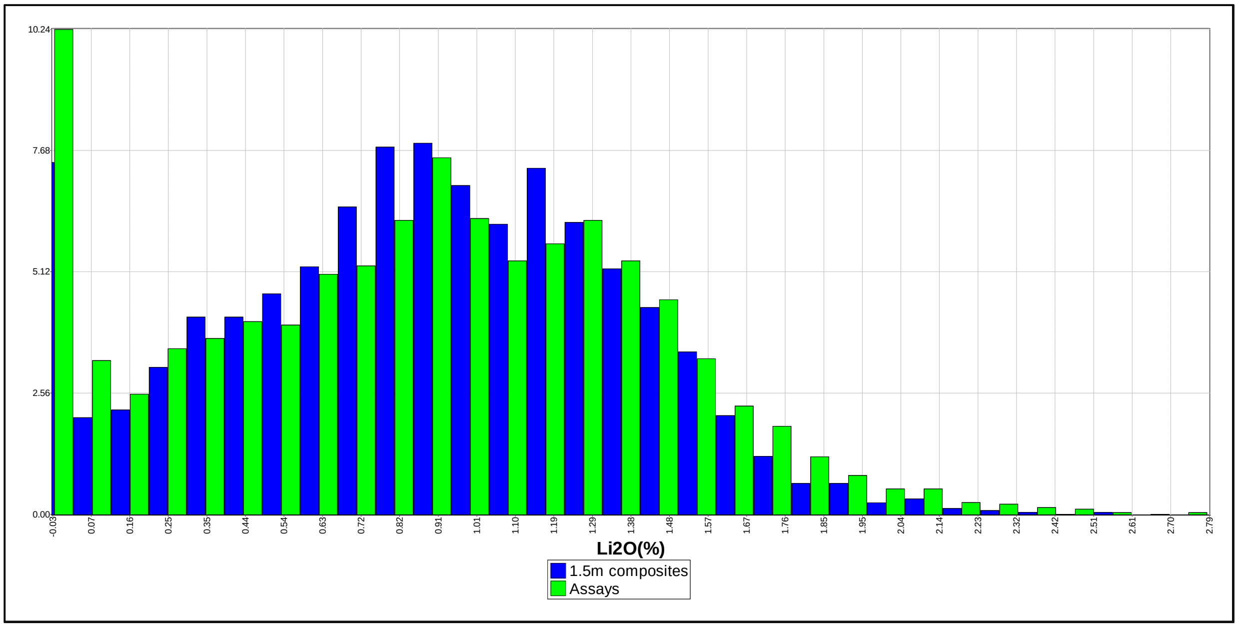


Figure 13-2: Histograms of the Original Samples Compared to the Composites

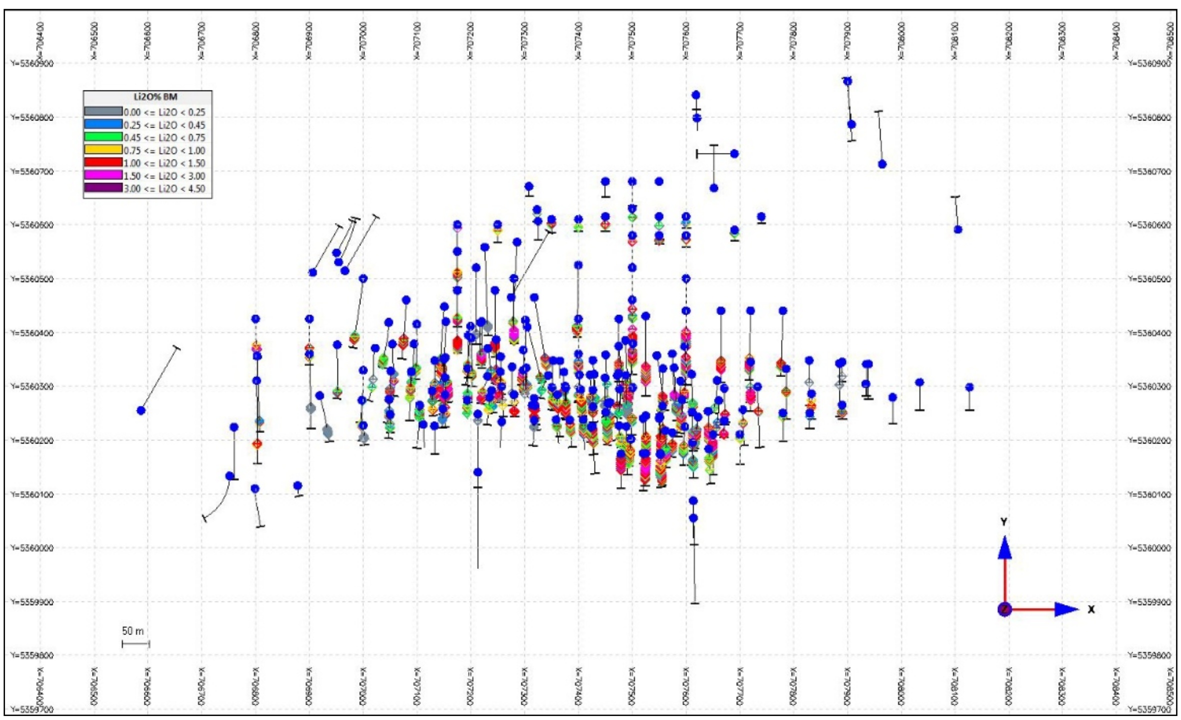


Figure 13-3: Plan View Showing the Spatial Distribution of the Composites

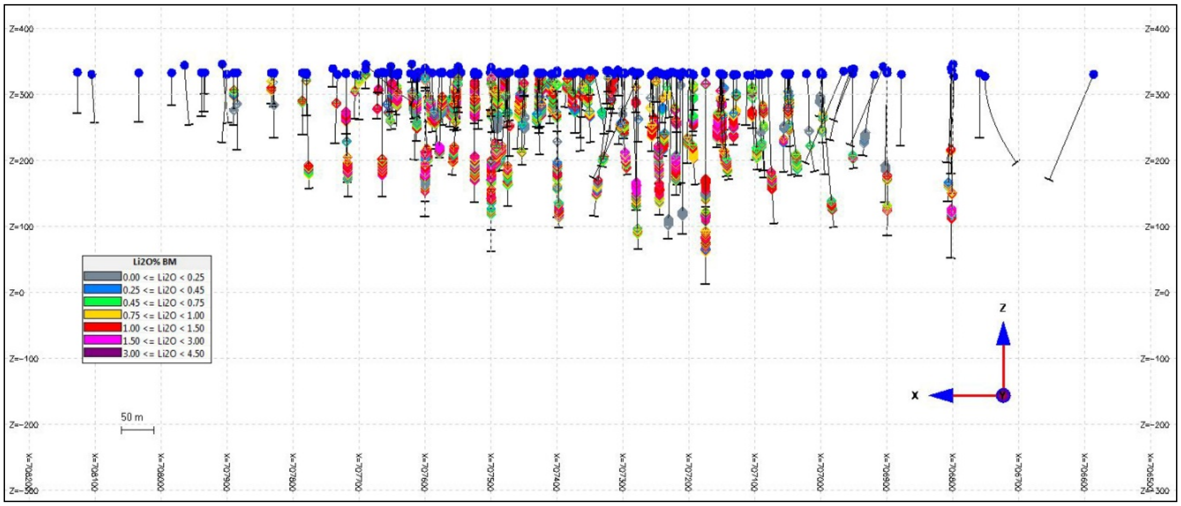


Figure 13-4: Longitudinal View Showing the Distribution of the Composites (Looking North)

13.2.4 Specific Gravity (SG)

An average specific gravity of 2.71 t/m^3 was used to calculate tonnage from the volumetric estimates of the resource block model.

The average specific gravity is derived from the gravity measurements taken from representative mineralized core samples from the 2010 and 2017 drilling periods. See section 11.7.

13.3 Geological Interpretation

The update of the 2017-2018 geological interpretation helped updating the 3D wireframe solids of the mineralization. For the purpose of modeling, sections (looking West) where generated every 25 m, with intermediate sections where it was necessary to tie in the solids. The modeling was first completed on sections to define mineralized polygons using the lithology and analytical data for lithium. A minimum grade of 0.4% Li_2O over a minimum drill hole interval length of 2 m was generally used as guideline to define the width of mineralized interpretations (Figure 13-5 and Figure 13-6). The final 3D wireframe model was constructed by meshing the defined mineralized interpretations based on the geological interpretation. Host rock and internal waste were considered and dealt with during modelling.

The 2017 interpretation continue to show the Main zone, with an orientation of North and a dipping between 30º (Main 2 solid) to 55º (upper part of Main 1 solid), averaging -45° toward the north (Figure 13-7 and Figure 13-8). The solid North is not connected to the Main zone forming a flat shallow dipping structure at about 15º-20º shape.

Local smaller 3D wireframe solids of significant size xenolith material (waste) located inside the Main 3D solid were also modeled. The Main and North 3D solids were cut by the updated 2018 overburden/bedrock contact surface.

13.3.1 Topographic and Overburden/Bedrock Contact Surfaces

All drill hole collars were draped to a topographic LiDAR surface acquired by Sayona in 2016. An overburden/bedrock interface 3D surface has been updated by triangulating the lower intercept of the overburden-coded lithology from the drill hole dataset. This overburden/bedrock contact surface was used to cut the Main and North 3D solids.

Figure 13-9 shows the final 3D wireframe solids in isometric view. The different colors of the 3D solids do not represent any specific parameters and are used to help the visual differentiation.

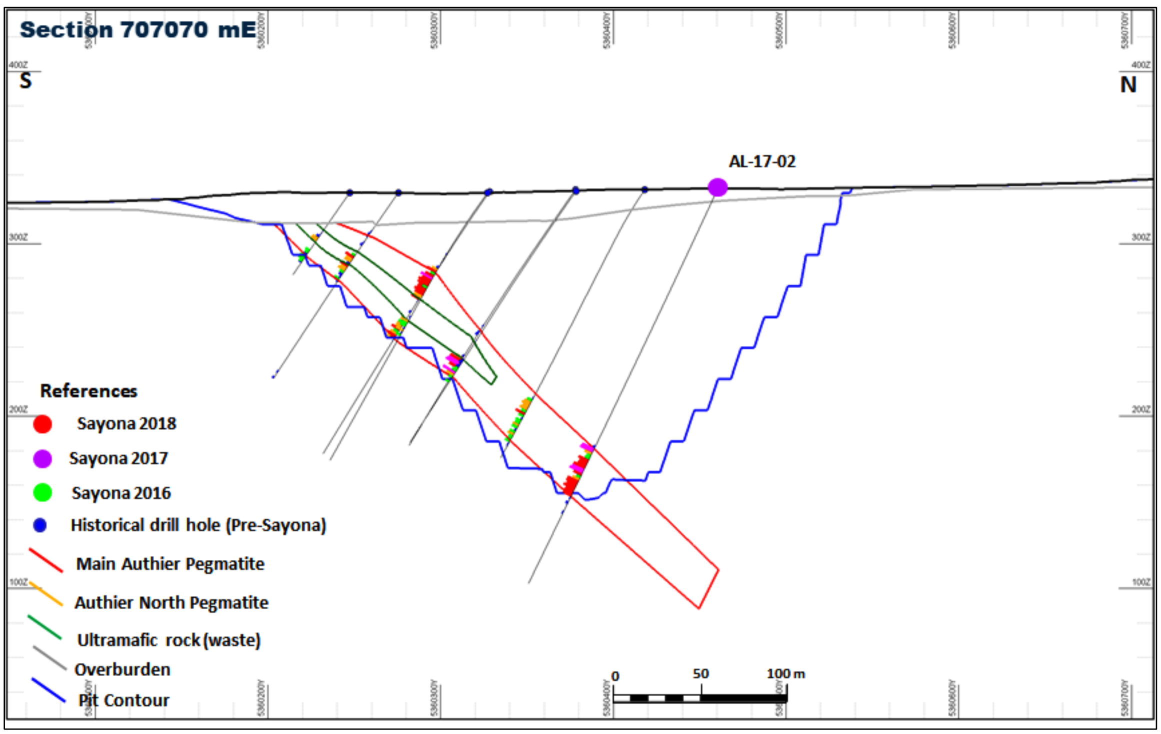


Figure 13-5: Section 707070 mE (Looking West) Showing Interpretation of the Mineralized Solids

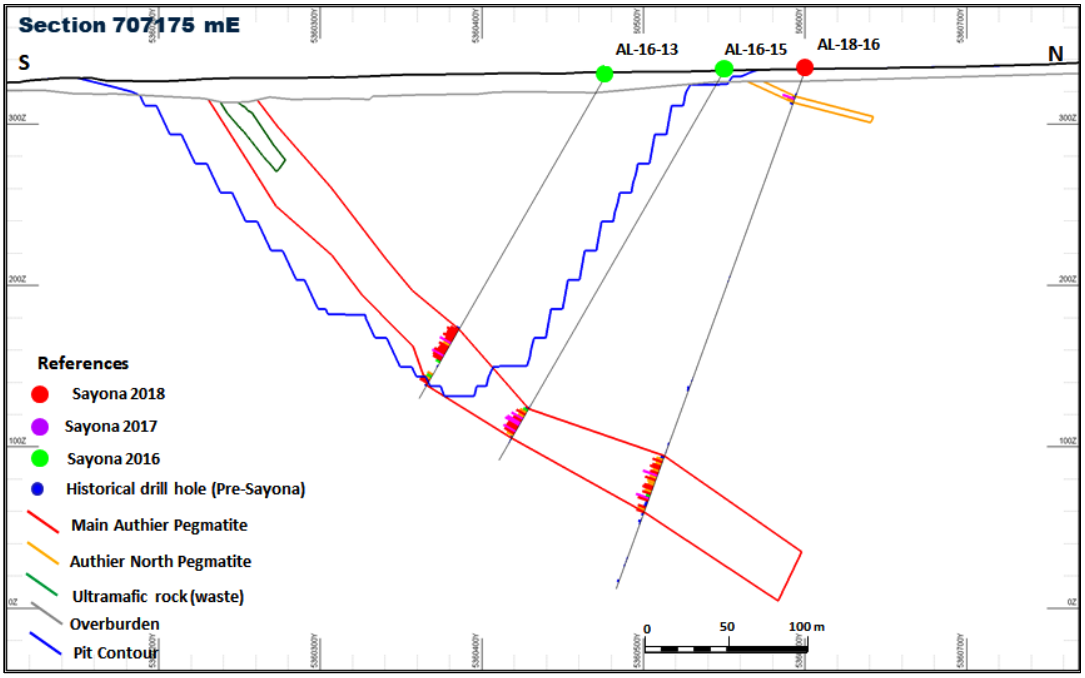


Figure 13-6: Section 707175 mE (Looking West) showing Interpretation of the Mineralized Solids

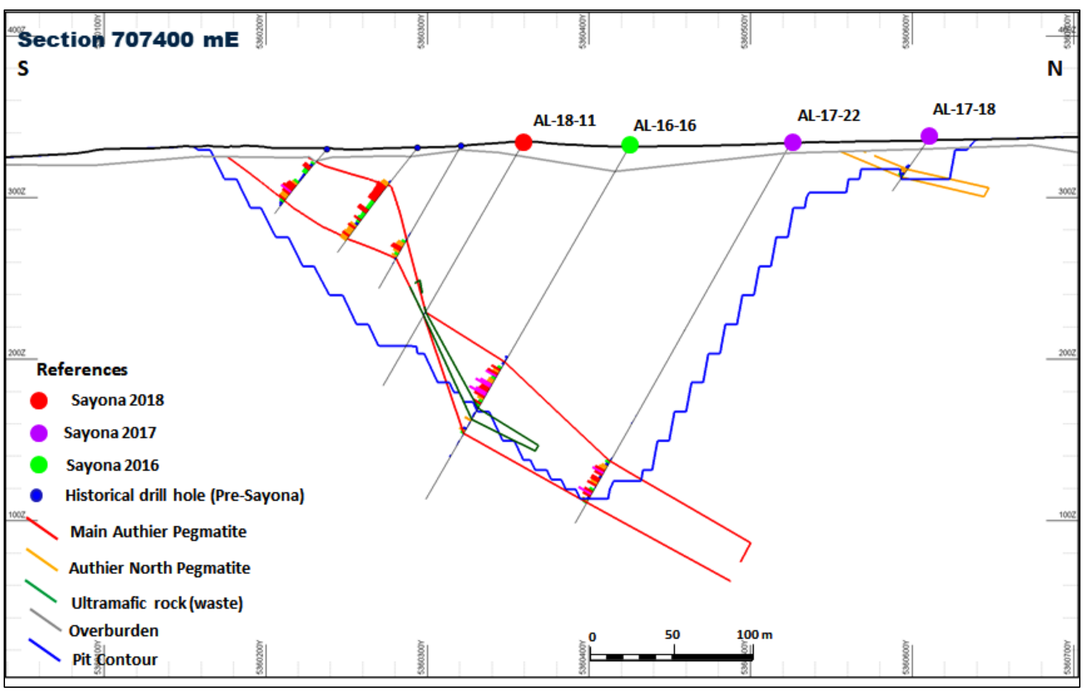


Figure 13-7: Section 707400 mE (Looking West) Showing Interpretation of the Mineralized Solids

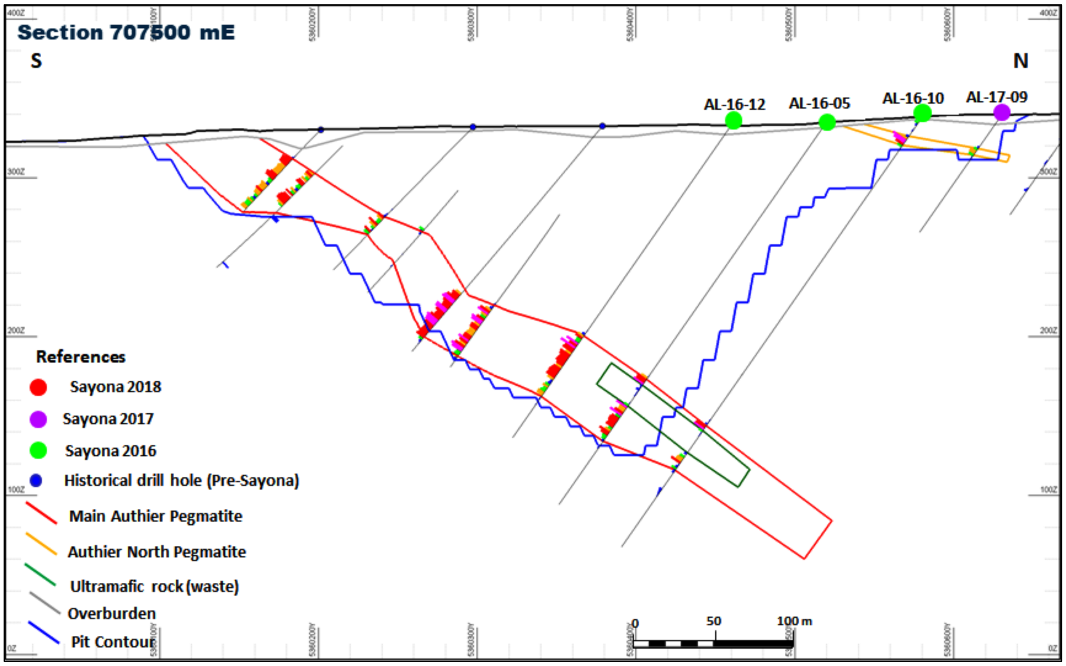


Figure 13-8: Section 707500 mE (Looking West) Showing Interpretation of the Mineralized Solids

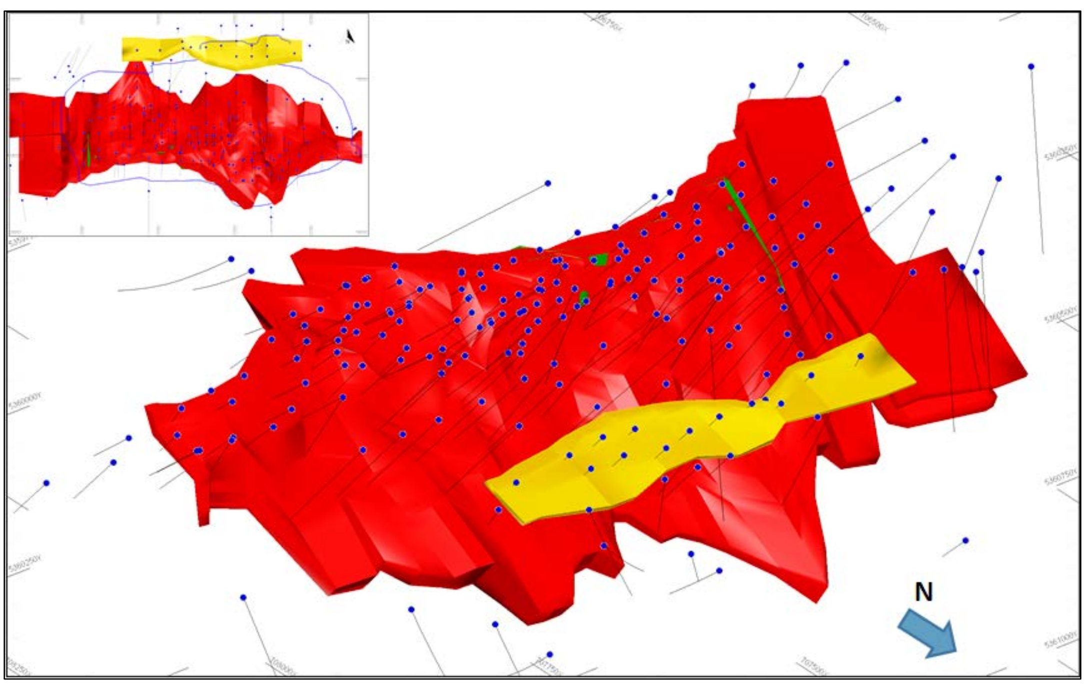


Figure 13-9: Isometric Views of the Final Mineralized Solids (Main Authier Pegmatite in Red and Authier North in Orange)

13.4 Resource Block Modeling

A block size of 3 m (N-S) by 3 m (E-W) by 3 m (vertical) was selected for the resource block model of the project based on drill hole spacing, width and general geometry of mineralization but primarily by the selected SMU from the advanced feasibility study. The 3 m vertical dimension corresponds to the bench height of a potential small open pit mining operation. The 3 m E-W dimension corresponds to about the selected degree of selectivity Sayona wants to achieve during mining. It also accounts for the variable geometry of the mineralization in that direction. The 3 m N-S block dimension accounts for the average minimum width of the mineralization modeled at Authier. The resource block model contains 431,235 blocks located inside the mineralized solids. The Block model was created with block fractions ranging from 0 to 1. Table 13-4 summarizes the parameters of the block model limits.

Table 13-4: Authier Deposit Resource Block Model Parameters

	X	Y	Z
Origin	706,699.5	5359998.5	-51.5
Maximum	707,920.5	5360703.5	347.5
Block Size	3	3	3
Number of Blocks	407	235	133
Length	1,161	398	274

13.4.1 Grade Interpolation Methodology

13.4.1.1 Geostatistical Study 2018

In order to determine the continuity and distribution of the grades (% Li_2O), the 1.5 m composites were submitted to a variographic study. The variographic study helped in the determination of the search ellipse criteria and for the kriging parameters for the block interpolation process used to compare with ID2 and ID3.

The composites show a normal distribution (Figure 13-1) with a relatively low coefficient of variation (standard deviation to the Mean) of 52%. A Variogram was generated for the Main zone orientation (including the Main 2) dip north.

The resulting model variogram for the Main zone can be modelled with the following function (Figure 13-10):

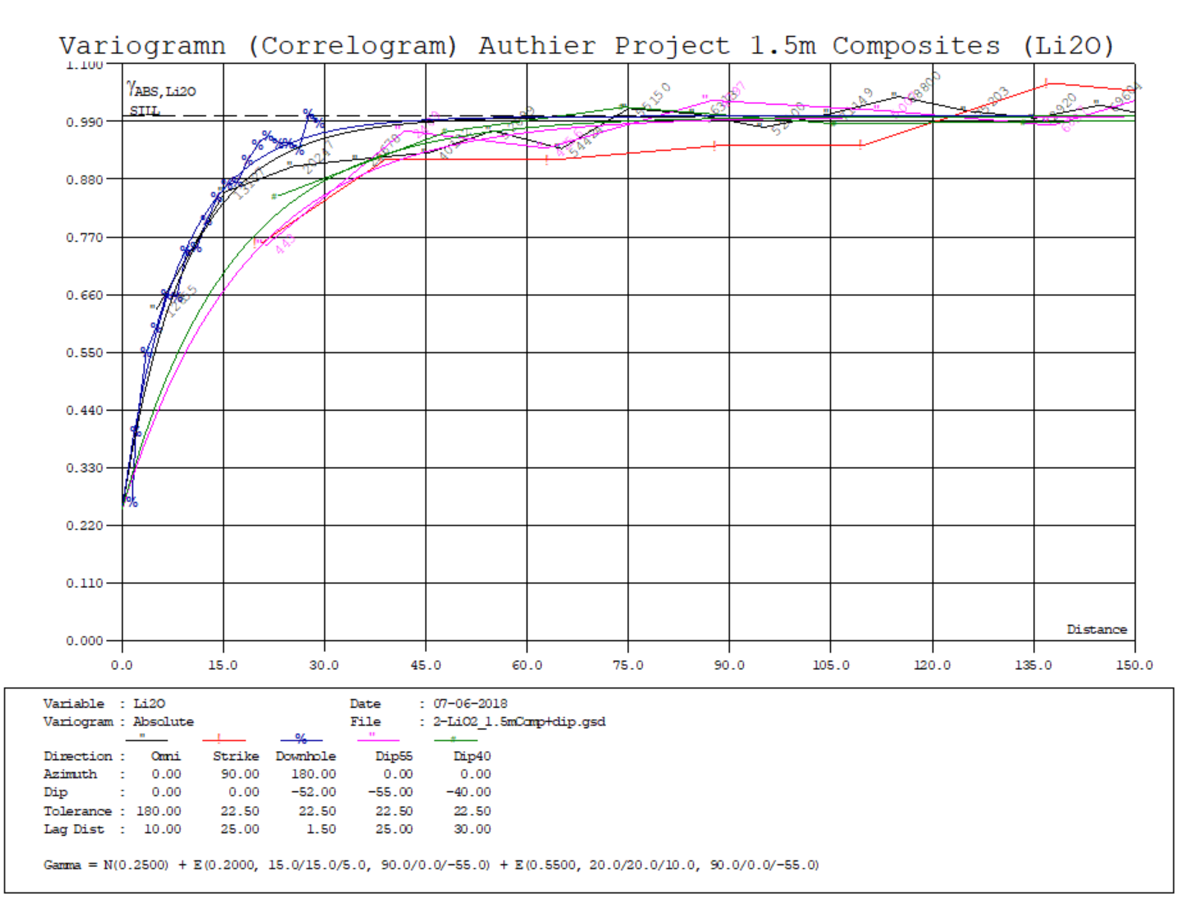


Figure 13-10: Variogram of the 1.5 m Composites for Li_2O Grades

Table 13-5: Authier Resource Variogram Settings

Name	Type	Sill	Longest Range	Median Range	Shortest Range	Az	Dip	Spin
2018_1_5mVario	Nugget	0.25	0	0	0	0	0	0
2018_1_5mVario	Exponential	0.2	15	15	5	90	0	-55
2018_1_5mVario	Exponential	0.55	20	20	10	90	0	-55

Where N represents a nugget effect of 25% and maximum continuity of 60 m is found along both the strike and the dip orientations (-55°). The shortest range is found across the mineralization with a range of 15 m* towards the south and 35° of dip (Table 13-5).

* Exponential component rages are 3 times longer than in this table in reality, i.e., first exponential component: 45 m, 45 m, 15 m. Second component: 60 m, 60 m, 30 m.

The geostatistical study was kept as a tool for estimation purposes but given the normal distribution of the population and relatively low coefficient of variation, the statistic estimation method IDW3 was choose.

13.4.1.2 Block Model Interpolation

The retained grade interpolation for the Authier lithium resource block model is the inverse distance cubed (ID^3) methodology. The interpolation process was conducted using three successive passes with more inclusive search conditions from one pass to the next until most blocks were interpolated.

Variable search ellipse orientations were used to interpolate the blocks. The general dip direction and strike of the mineralized pegmatite were modeled on each section and then interpolated in each block (Figure 13-11). During the interpolation process, the search ellipse was orientated following the interpolation direction (azimuth-dip (dip direction) and spin (strike direction) of each block, hence better representing the dip and orientation of the mineralization.

The first pass was interpolated using a search ellipsoid distance of 50 m (long axis) by 50 m (intermediate axis) and 25 m (short axis) with an average orientation of 90° azimuth (local grid), -55° dip and 0° spin which represents the general geometry of the pegmatites in the deposit. Using search conditions defined by a minimum of seven composites, a maximum of 15 composites and a maximum of two composites per hole (minimum of three holes), 40% of the blocks were estimated. For the second pass, the search distance was twice the search distance of the first pass and composites selection criteria were kept the same as for the first pass. A total of 79% of the blocks were interpolated following the second pass. Finally, the search distance of the third pass was increased to 300 m (long axis) by 300 m (intermediate axis) by 150 m (short axis) and again the same composites selection criteria were applied. The purpose of the last interpolation pass was to interpolate the remaining un-estimated blocks mostly located at the edges of the block model, representing 21% of the blocks.

Figure 13-11 and Figure 13-12 illustrate the three search ellipsoids used for the different interpolation passes. Figure 13-13 to Figure 13-18 show the results of the block model interpolation in isometric view, cross section and bench (202 m RL) views respectively.

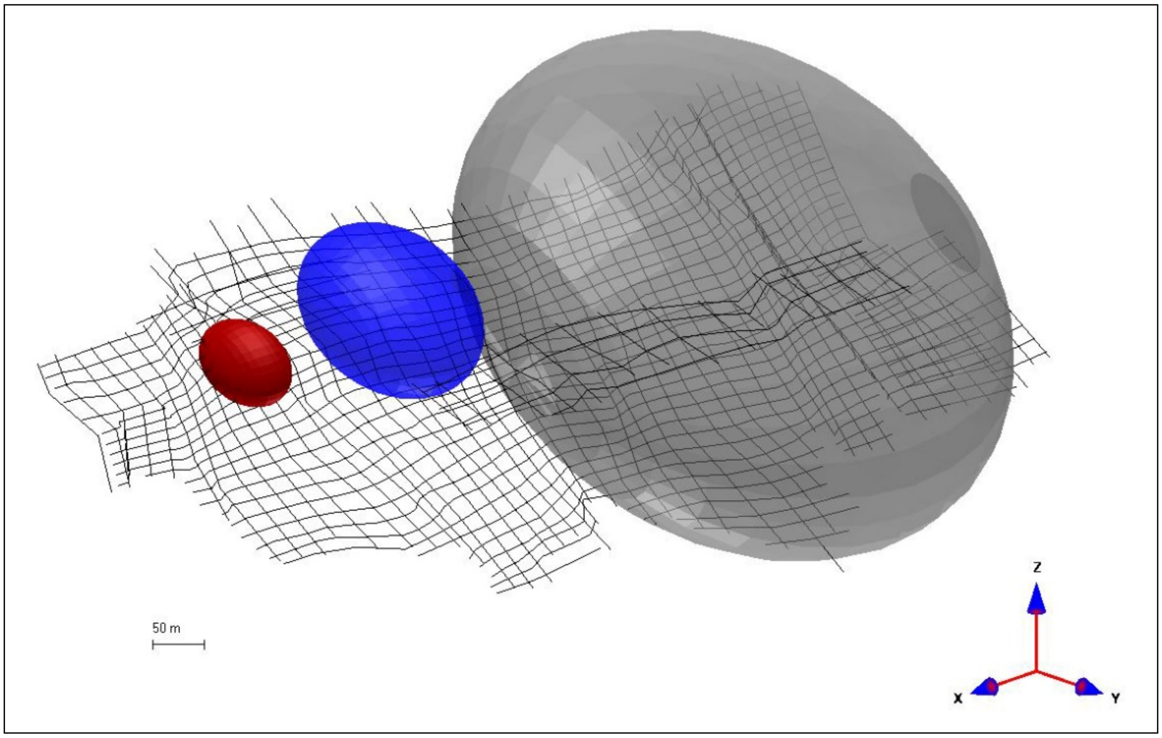


Figure 13-11: View of the 3 Different Search Ellipsoids Used in the Interpolation Process

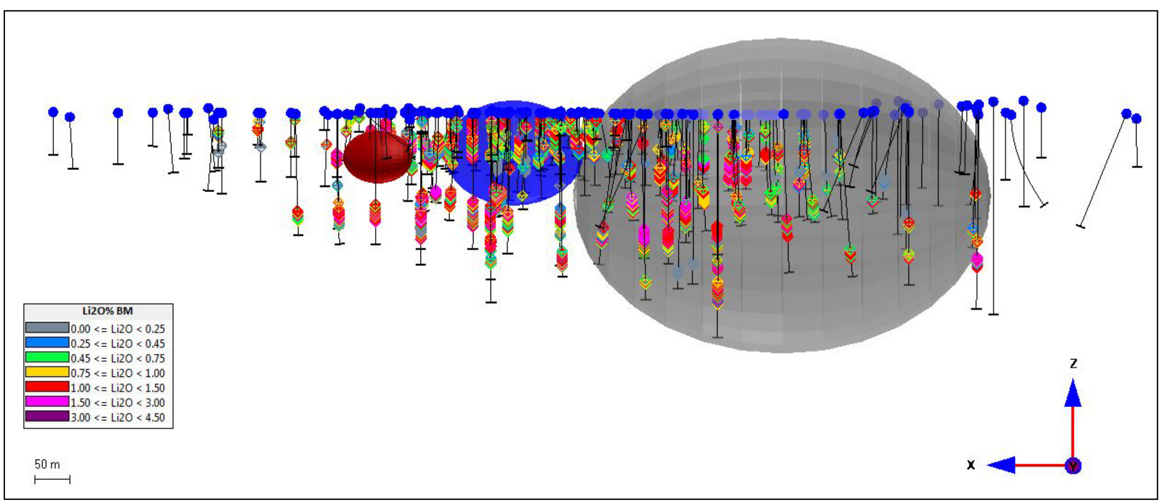


Figure 13-12: Long Section View (Looking South) of the Modeled Orientation of the Ellipse

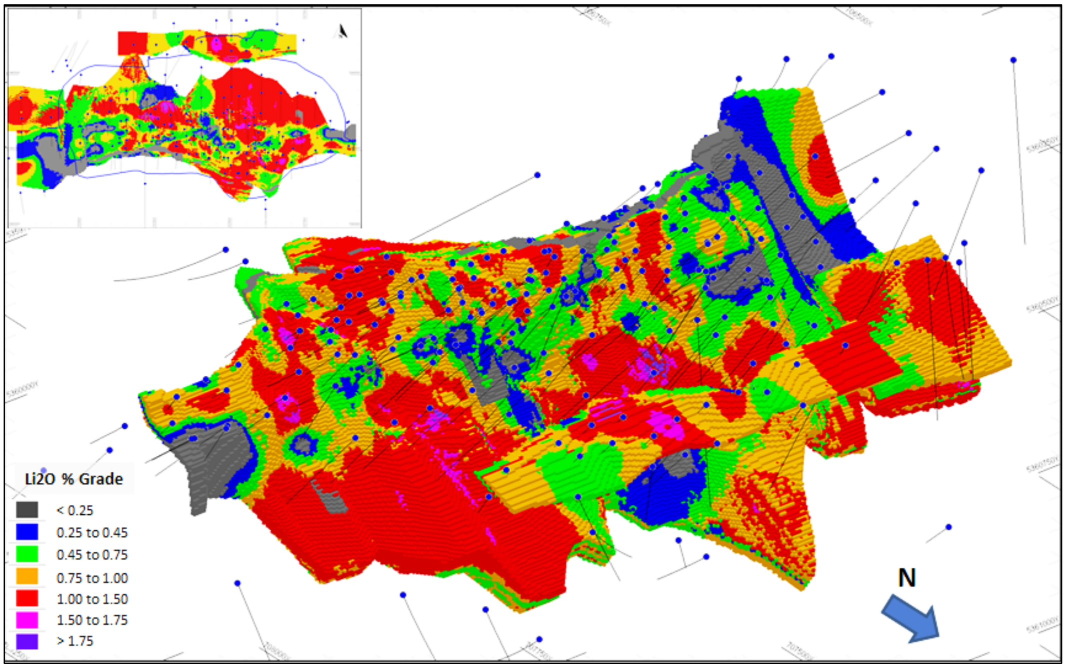


Figure 13-13: Isometric View (Looking Southwest) of the Interpolated Block Model Showing Li_2O Grades (%)

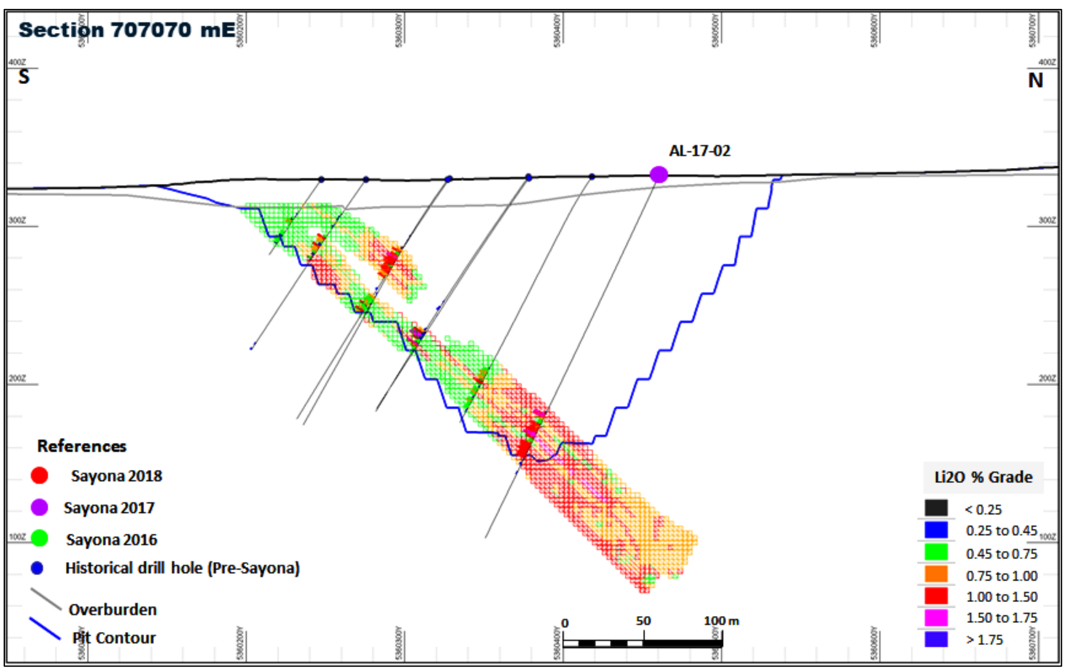


Figure 13-14: Section 707070 mE (Looking West)
Block Model Interpretation Grade Distribution using 0.55% Li_2O Cut-off Grade

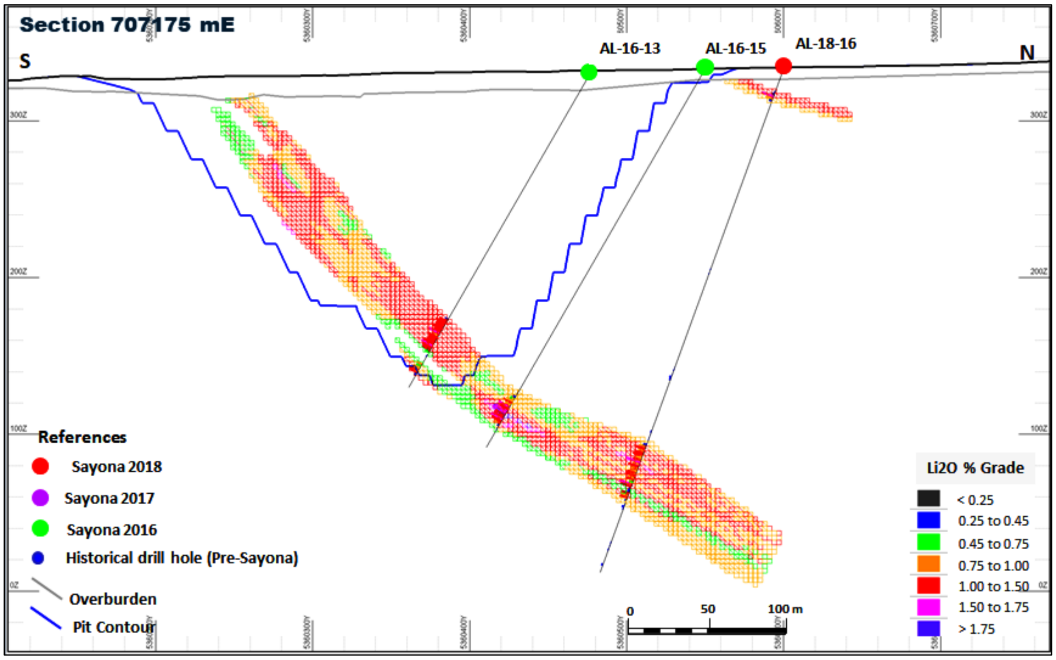


Figure 13-15: Section 707175 mE (Looking West)
Block Model Interpretation Grade Distribution Using 0.55% Li2O Cut-off Grade

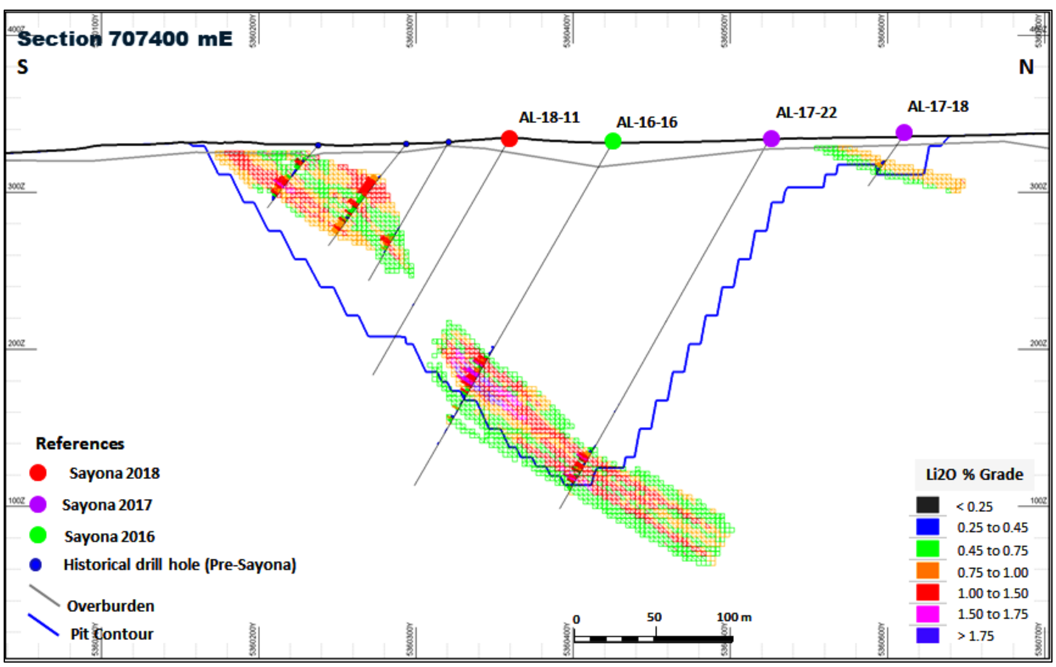


Figure 13-16: Section 707400 mE (Looking West)
Block Model Interpretation Grade Distribution Using 0.55% Li2O Cut-off Grade

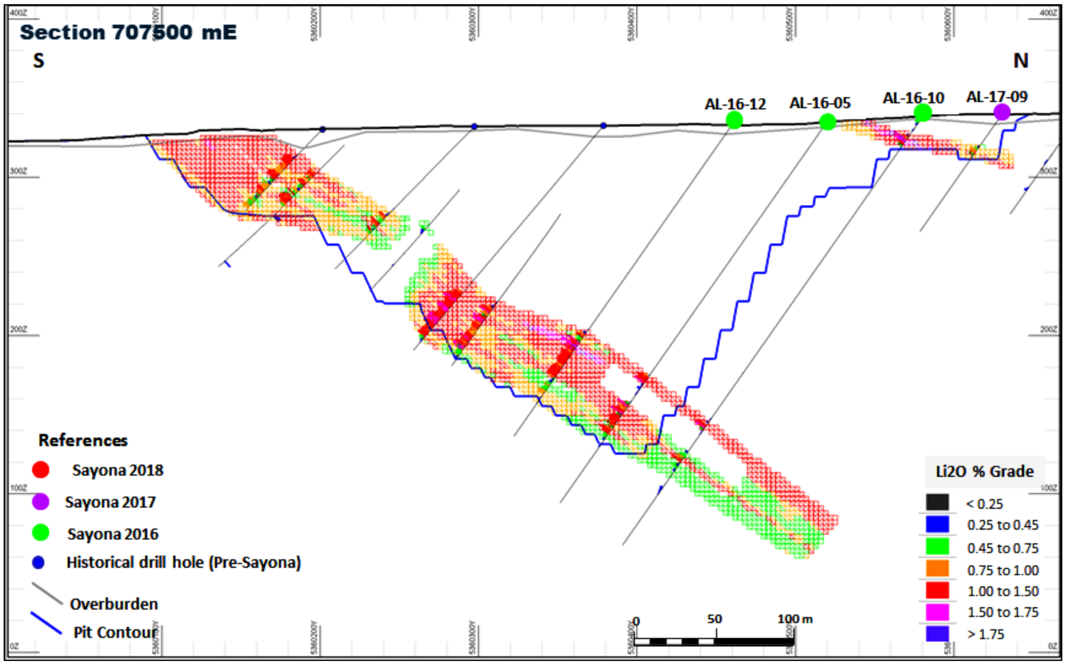


Figure 13-17: Section 707500 mE (Looking West)
Block Model Interpretation Grade Distribution Using 0.55% Li2O Cut-off Grade

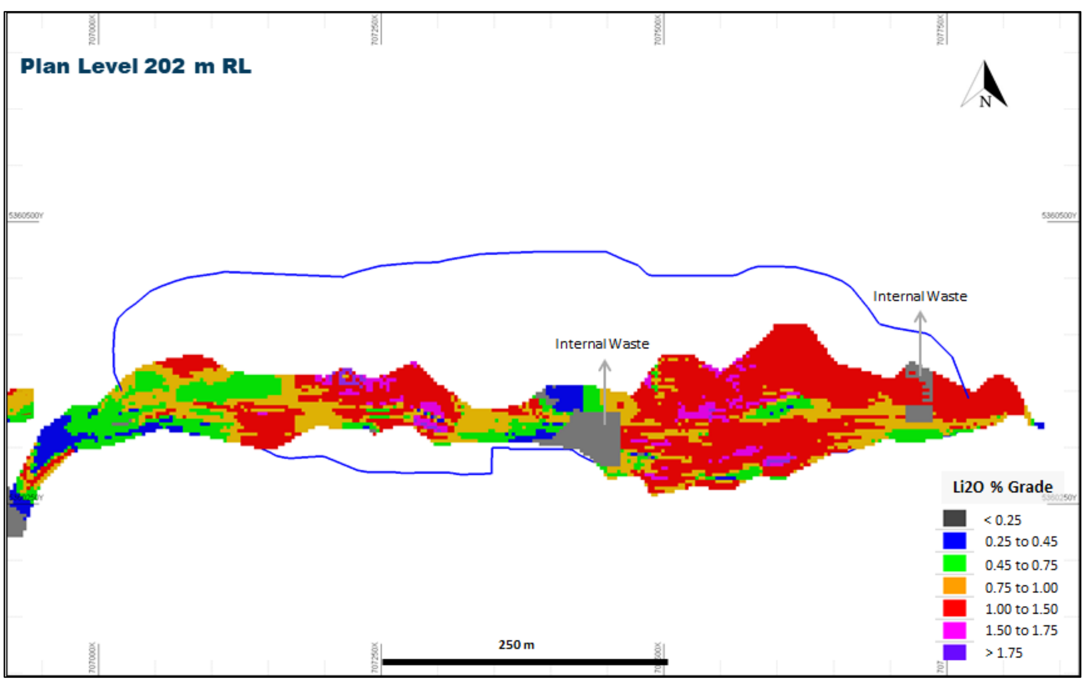


Figure 13-18: Bench View Showing Li_2O Grades (%)
Block Model Interpolation Results for Authier at 202 m RL

13.4.1.3 Statistical Validation of the Interpolation Process

In order to validate the interpolation process, the block model was compared statistically, to the assays and composites. The distribution of the assays, composites and blocks are normal and show a similar average value with decreasing levels of variance (Figure 13-19 to Figure 13-23). The assays and composites have respective averages of 0.92% Li_2O and 0.93% Li_2O with variances of 0.30 and 0.24. The resulting interpolated blocks have and average value of 0.85% Li_2O with a variance of 0.16% (Table 13-6). The block Average value is affected by the 0% values assigned to each waste solid.

Furthermore, the block values were compared to the composites values located inside the interpolated blocks (Figure 13-24). This enables to test for possible over or under evaluation of the grade by the search parameters by testing the local correlation.

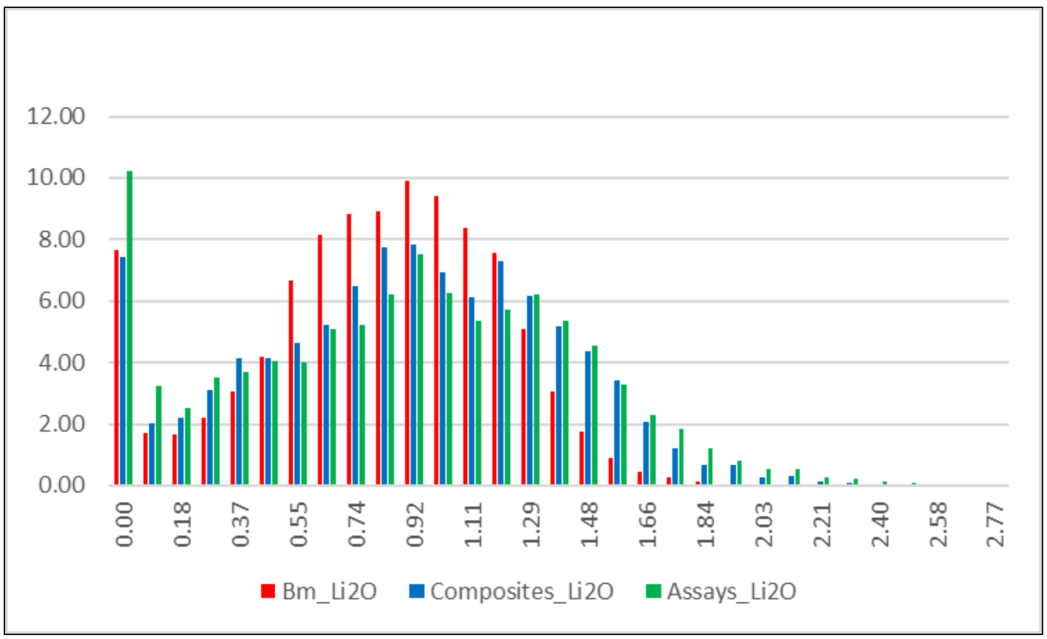


Figure 13-19: Histogram of Blocks vs. Composites vs. Assays

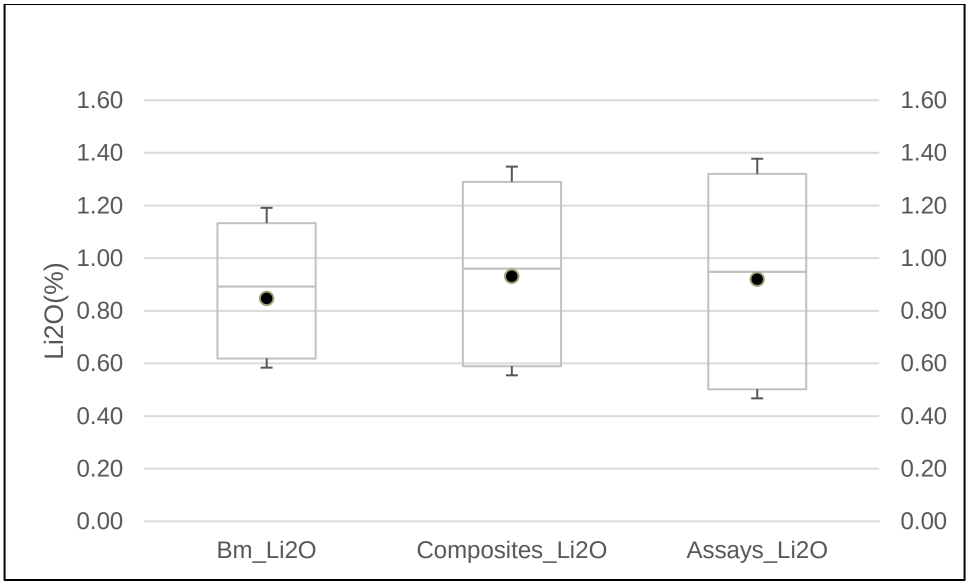


Figure 13-20: Boxplot of Blocks vs. Composites vs. Assays

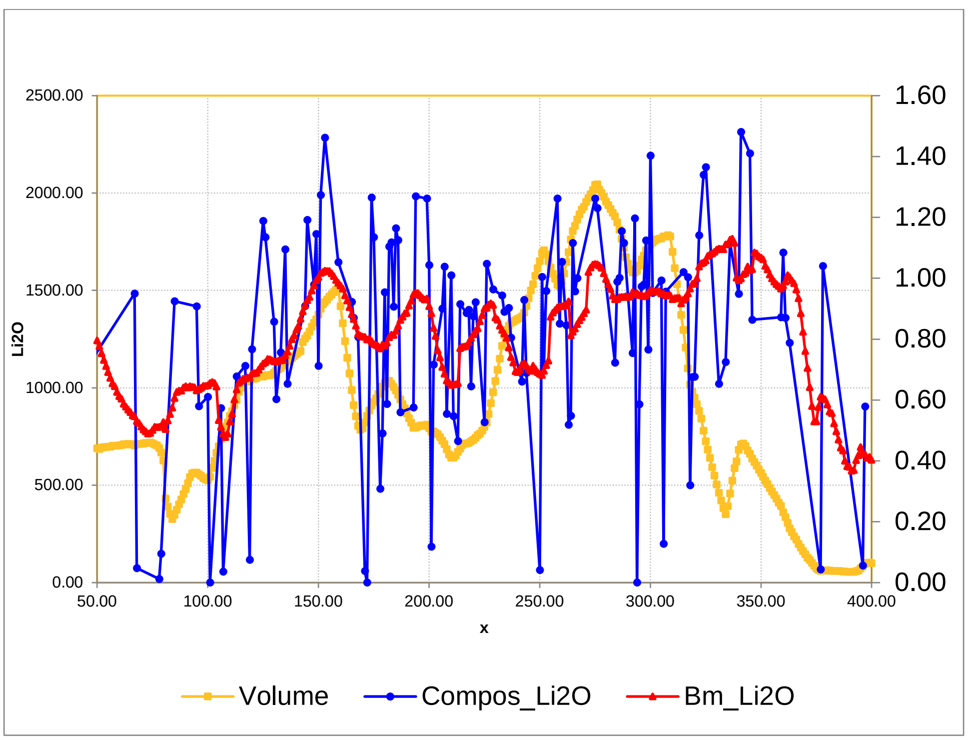


Figure 13-21: Swath Plot (X) of Blocks vs. Composites vs. Volume

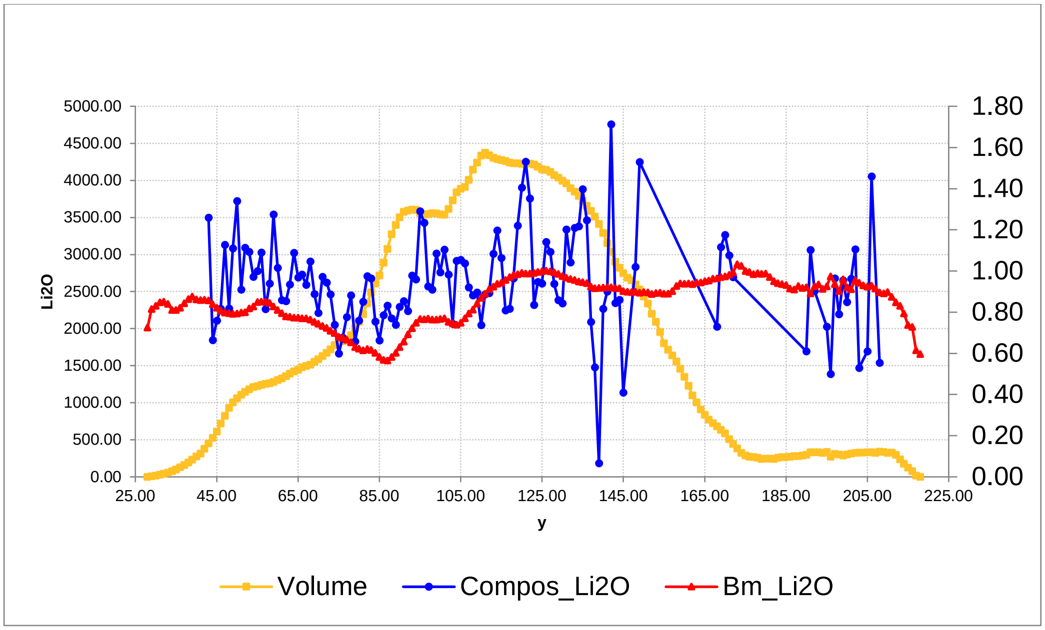


Figure 13-22: Swath Plot (y) of Blocks vs. Composites vs. Volume

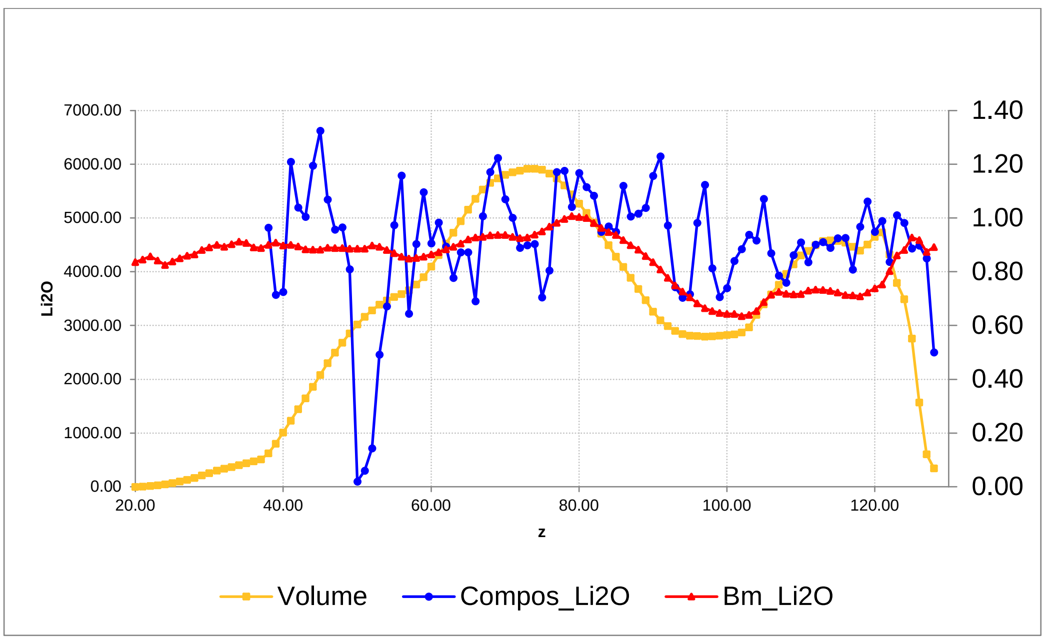


Figure 13-23: Swath Plot (z) of Blocks vs. Composites vs. Volume

Table 13-6: Statistical Comparison of Assay, Composite and Block Data Statistics Report

Statistics	Blocks	Composites	Assays
Min Value	-	-	-
Max Value	2.23	2.61	2.77
Average	0.85	0.93	0.92
Weighted Average Length		0.93	0.91
Variance	0.16	0.24	0.3
Standard Deviation	0.4	0.49	0.55
% Variation	0.47	0.53	0.59
Median	0.89	0.96	0.95
First Quartile	0.62	0.59	0.5
Third Quartile	1.13	1.29	1.32
Count	359,360	2,451	2,804

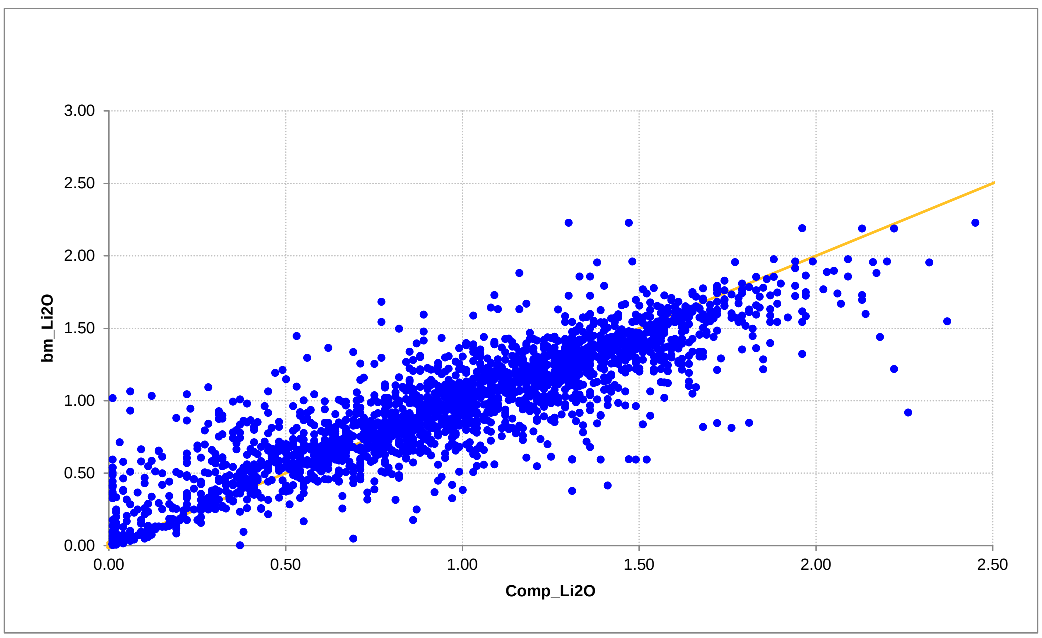


Figure 13-24: Comparison of Block Values vs. Composites Inside Those Blocks

13.5 Mineral Resource Classification

The mineral resources at Authier lithium are classified into Measured, Indicated and Inferred categories. The mineral resource classification follows the JORC 2012 requirements and guidelines and is based on the density of analytical information, the grade variability and spatial continuity of mineralization. The mineral resources were classified in two successive stages: automatic classification followed by manual editing of final classification results.

The first automatic classification stage is focussed on composites (and drill holes) rather than blocks. The classification process focusses around each composite respecting a minimum number of nearby composites from a minimum number of holes located within a search ellipsoid of a given size and orientation. For the Measured resource category, the search ellipsoid was 50 m (strike) by 50 m (dip) by 25 m with a minimum of seven composites in at least three different drill holes (maximum of two composites per hole) An ellipse fill factor of 60% was applied the measured category, i.e., that only 50% of the blocks were tagged as measured within the search ellipse. For the Indicated category, the search ellipsoid was twice the size of the Measured category ellipsoid using the same composites selection criteria. An ellipse fill factor of 85% was applied the Indicated Category All remaining blocks were considered to be in the inferred category.

This automatic classification centred on composites is preferred to the more classical method of classification centred on blocks in a sense that it is limiting significantly the spotted dog effect.

The second classification stage involved the delineation of coherent zones for the Indicated category based on the results of the automatic classification. The objective was to take into account the geological continuity and grade as well as the effect of the open pit design developed in the pre-feasibility study. The second stage consisted of defining a 3D solid on a selected area for the Indicated category.

Figure 13-25 shows an isometric view (looking southwest) of the final block model automatic classification with respective categories (Measured: red; Indicated: yellow; Inferred: blue). Figure 13-26 to Figure 13-29 show the block model automatic classification on different sections and Figure 13-30 in bench at 202 m RL.

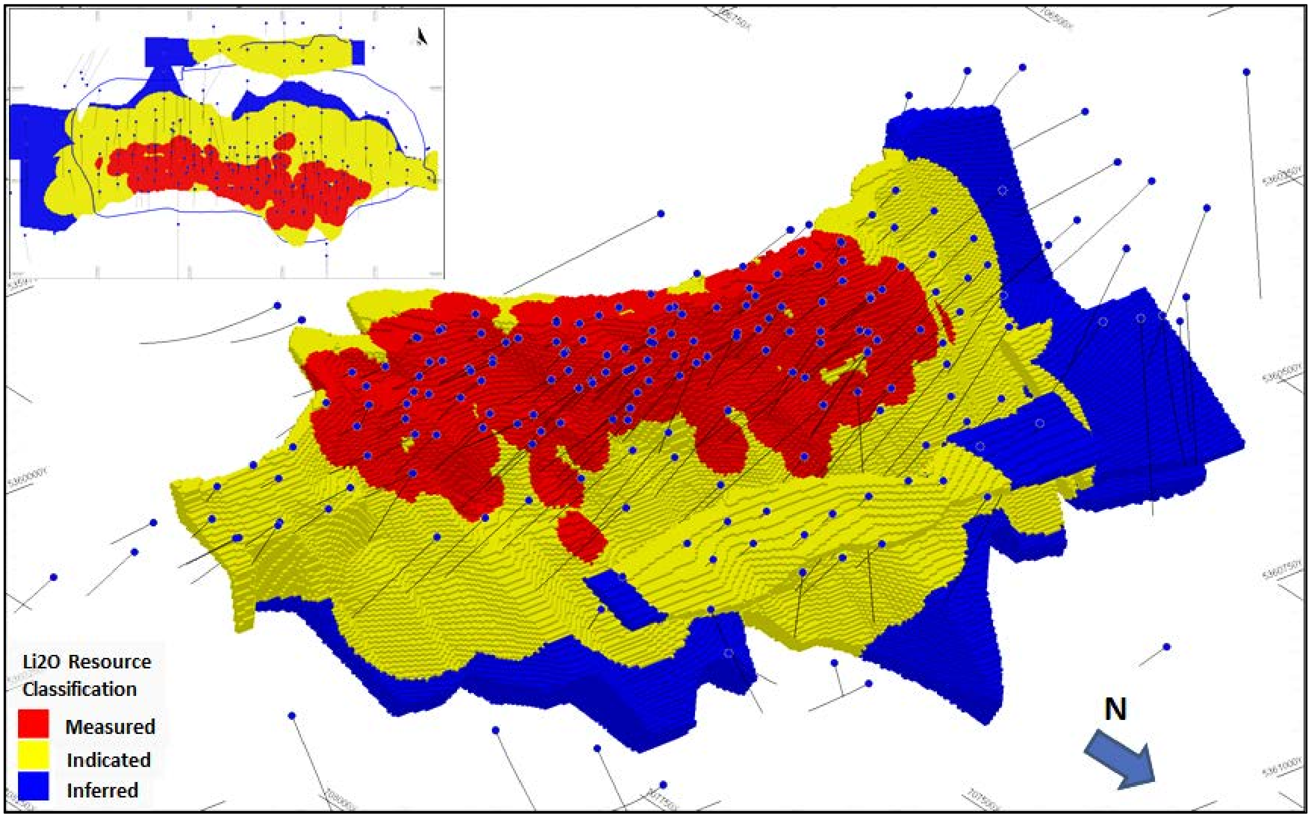


Figure 13-25: Isometric View (Looking Southwest) of the Interpolated Block Model for Authier Showing Li_2O Resource Classification

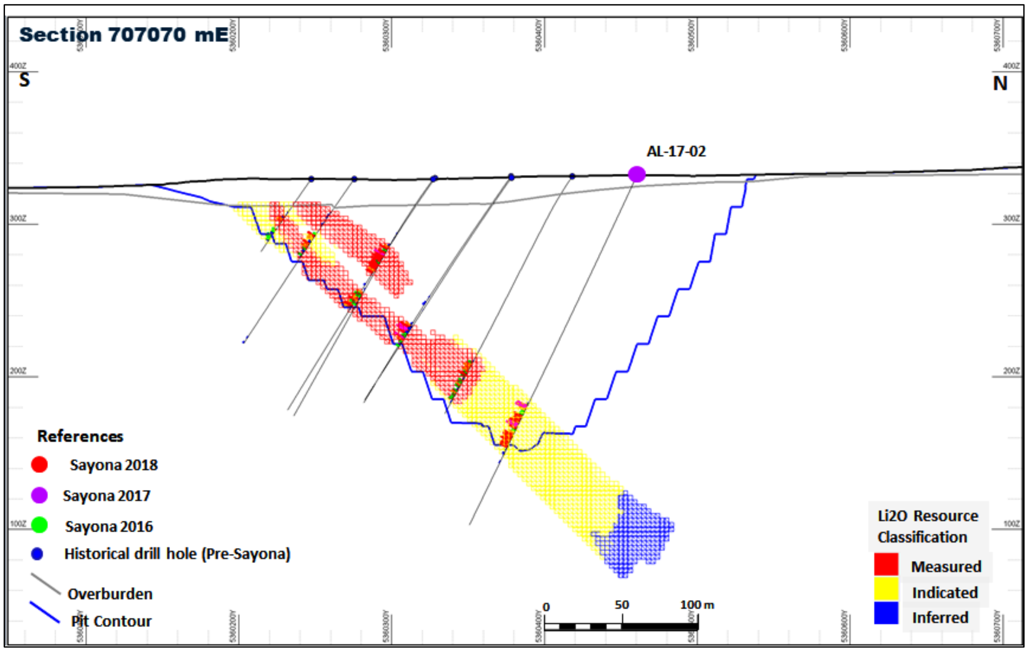


Figure 13-26: Section 707070 mE (Looking West) of the Interpolated Block Model for Authier Showing Li2O Resource Classification Using 0.55% Li2O Cut-off Grade

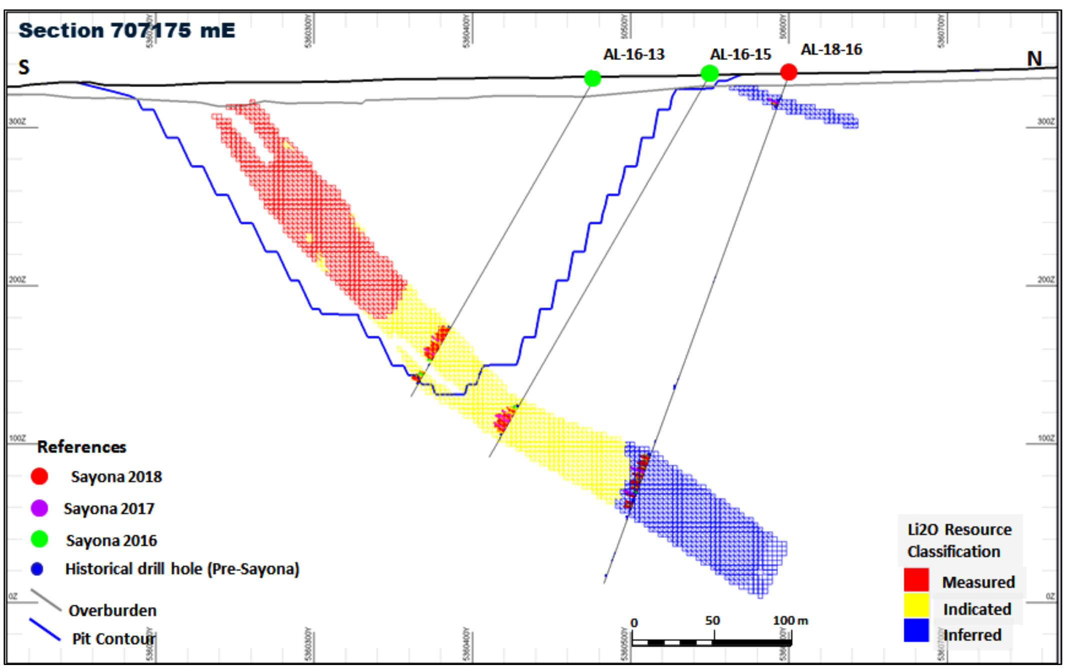


Figure 13-27: Section 707175 mE (Looking West) of the Interpolated Block Model for Authier Showing Li2O Resource Classification Using 0.55% Li2O Cut-off Grade

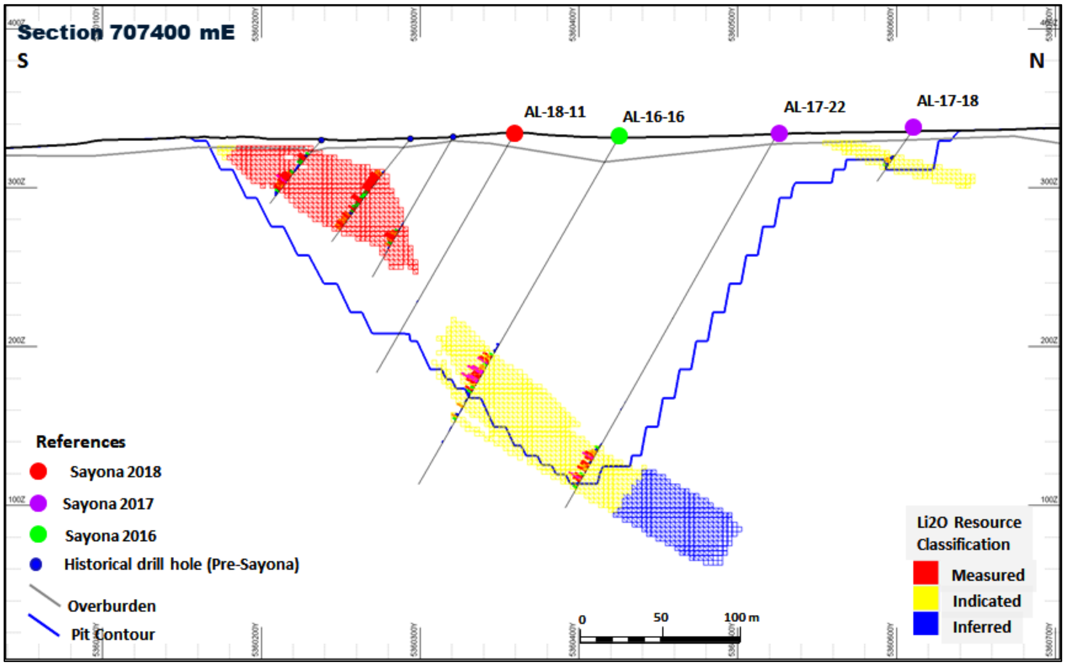


Figure 13-28: Section 707400 mE (Looking West) of the Interpolated Block Model for Authier Showing Li2O Resource Classification Using 0.55% Li2O Cut-off Grade

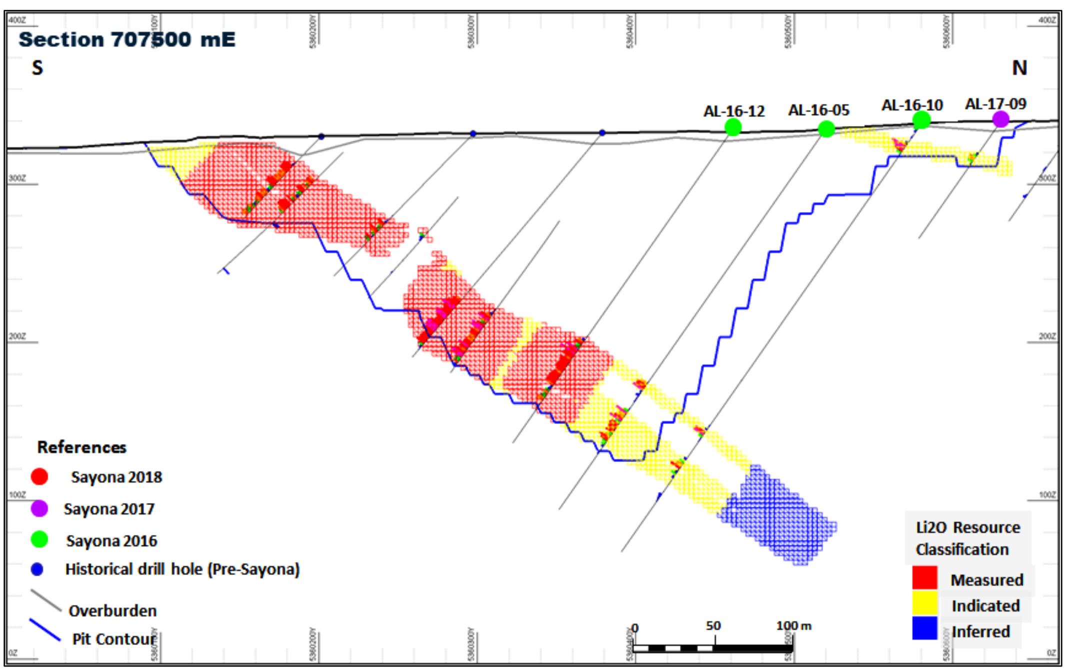


Figure 13-29: Section 707500 mE (Looking West) of the Interpolated Block Model for Authier Showing Li2O Resource Classification Using 0.55% Li2O Cut-off Grade

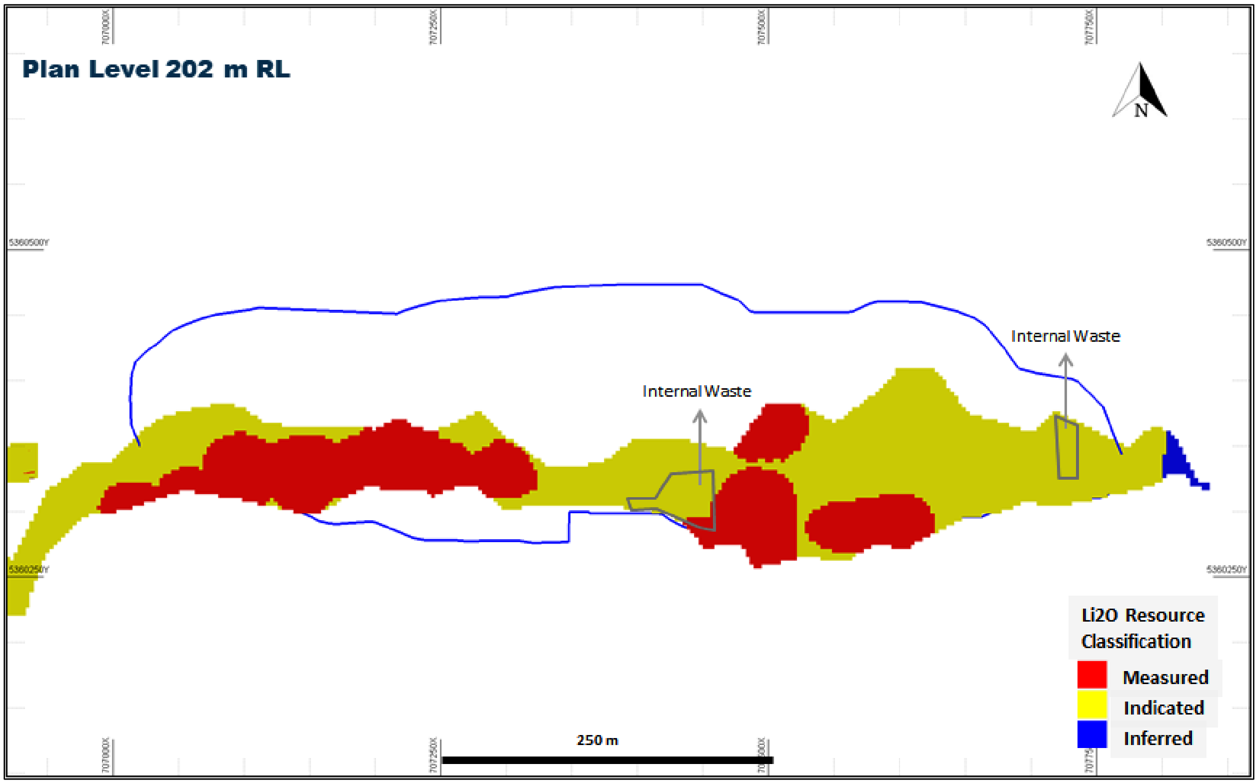


Figure 13-30: Bench View Showing Li_2O Resource Classification for the Authier Interpolated Block Model at 202 m RL

13.6 Mineral Resource Estimation

The updated Mineral Resource for the Authier property is integrated by the Authier pegmatite deposit and Authier North pegmatite deposit. The next Table 13-7 summarizes the global mineral resources for Authier Property for a cut-off grade of 0.55% Li_2O.

Table 13-7: Authier and Authier North JORC Mineral Resources Estimate (0.55% Li_2O cut-off grade) Inclusive of Reserves

Category	Tonnes (Mt)	Grades (% Li_2O)	Contained Li_2O (t)
Measured	6.58	1.02	67,116
Indicated	10.60	1.01	107,060
Measured and Indicated	17.18	1.01	174,176
Inferred	3.76	0.98	36,848

Approximately 98% of the Mineral Resource is contained in the Authier Deposit (main Authier pegmatite). The remaining 2% of the global resource is contained within the Authier North pegmatite.

On April 12, 2018, an independent JORC Mineral Resource (2012) estimate for the Authier pegmatite deposit was reported. The Mineral Resource was revised and optimised during the preparation of this report as part of the pit optimisation process. The updated Li_2O resource was estimated at a higher cut-off grade and with slight differences in the search ellipsoid and interpolation strategies based on the geological and grade continuity of the deposit. The previous JORC Mineral Resource (2012) estimate for the Authier pegmatite deposit is detailed in the following Table 13-8:

Table 13-8: Authier and Authier North JORC Mineral Resources Estimate (0.45% Li2O Cut-off Grade) Inclusive of Reserves (reported on April 12, 2018)

Category	Tonnes (Mt)	Grades (% Li_2O)	Contained Li_2O (t)
Measured	6.09	1.01	61,509
Indicated	11.55	1.04	120,120
Measured and Indicated	17.64	1.03	181,629
Inferred	2.82	0.98	27,636

13.6.1 Sensitivity Analysis

A limited sensitivity analysis was conducted using different estimation methods: Ordinary Kriging (OK), Inverse Distance Squared (ID^2) and Inverse Distance Cubed (ID^3) (Figure 13-31). For this resource estimate the method choose was ID^3. The Sensitivity analysis outlined that the OK mineral resources and grades are affected by smoothing. And that the ID^3 is the one with the most heist average grades. ID^2 and ID^3 are relatively close in terms of tonnage and average grades.

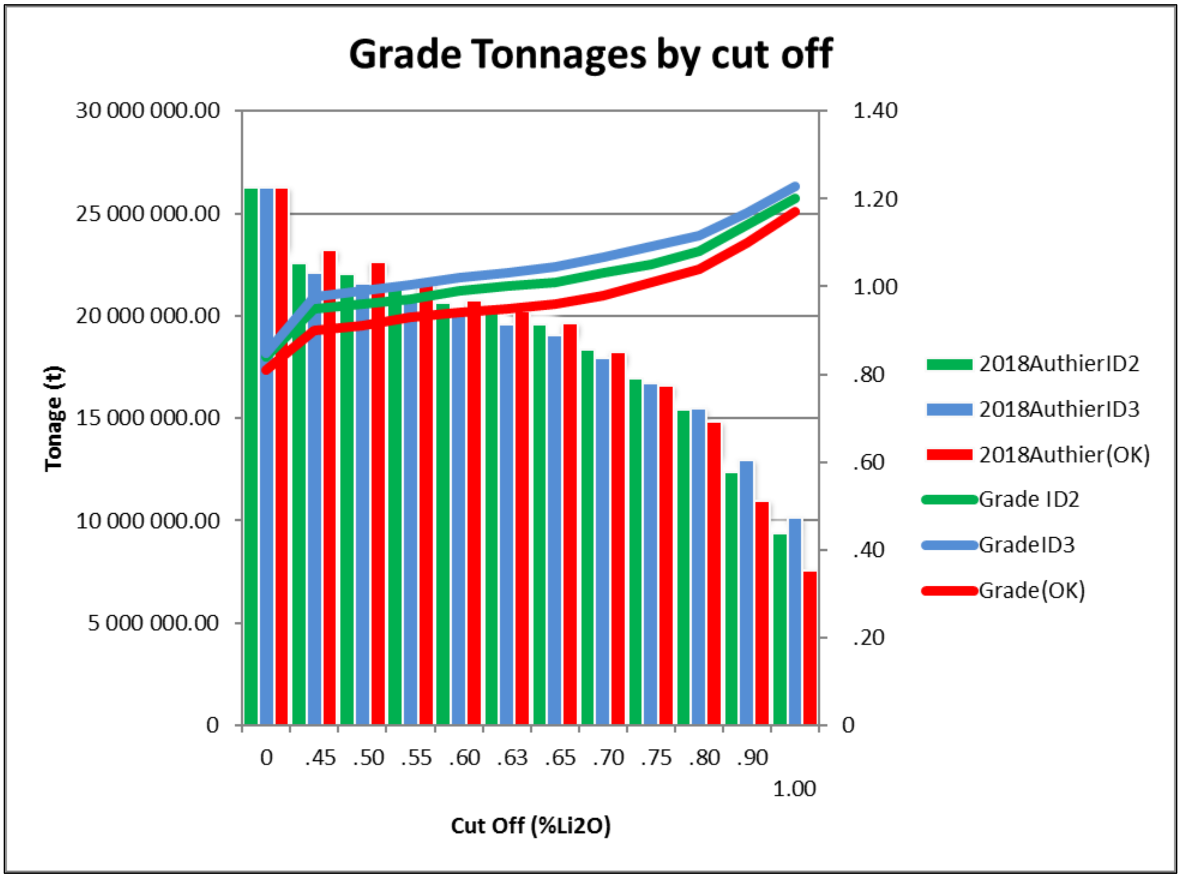


Figure 13-31: Grade Tonnage Curve at Various Cut-off Grades for ID3, ID2 and OK for Authier Resource Project

Drilling by Sayona has shown that the main Authier pegmatite is reasonably predictable in both, grades and geological continuity given the consistency of mineralized widths and grades along the strike extension tested so far.

The resource expansion achieved by Sayona on the Authier mineralized pegmatite has been basically at depth and along strike.

At mid to deep levels (beyond 100 m down surface) Sayona drilling has consistently intercepted mineralized pegmatite returning widths ranging 20 m to 40 m and averages grades equal or higher than 1% Li_2O even in areas untested by previous owners.

The combination of an extensional east - west structural array together with a competent brittle ultramafic metamorphic host rock allowed the placement of a wide single mineralized pegmatite body.

There are a limited number of areas at shallow and deep levels that returned little or no mineralized pegmatite due to faulting interpreted as syn-mineral and post mineral however; the core part of the Authier pegmatite is not significantly affected by such faulting.

This combination of characteristics makes the main Authier mineralized pegmatite predictable in both, geology and grade and allows expanding the mineralisation included in the measured category and expanding to depth the indicated resources categories based in geological continuity.

14. GEOTECHNICAL

The geotechnical investigation includes three complementary studies for the different facilities of the mine. Studies from Hydrogéologie Richelieu (2017 and 2018) gives geotechnical information for the tailings and waste rocks storage facility, a study from Journeaux Associates (2017) treats pit slopes and a study from GHD (2019) provides geotechnical information for building foundations and for water pond construction. This chapter presents only the studies from Hydrogéologie Richelieu and Journeaux Associates.

14.1 Tailings and Waste Rocks Storage Facility

The storage facility for tailings and waste rock is located north and northwest of the Authier open pit. The site is bounded to the north and northeast by Chemin des Pêcheurs. The location is shown in Figure 14-1.

14.1.1 Site Description

The project site covers the tailings and waste rock facility and its immediate vicinity. It is characterized by a relatively flat topography with occasional bedrock outcrops and gently rolling hills (east of the storage facility). The project site is generally covered by forest. Wetlands are present at the center of the site and northeast of the area covered by the proposed storage facility.

14.1.2 Regional Geology

The regional geology associated with the Authier Lithium project site is presented in the following sections.

14.1.2.1 Overburden

The study area is located within the physiographic region identified as the Abitibi Uplands of James Region, as indicated by Physiographic Regions of Canada, Map 1254A (2nd edition). The Abitibi Uplands consist predominantly of quaternary deposits. Quaternary deposits mainly include till plains, moraines, drumlins, sand, silt, and clay plains and old-shore and beach deposits. Part of the Abitibi Uplands is the Cobalt Plain within the Upper Ottawa River Basin. The Cobalt Plain is characterized by low hummocky terrain interrupted by several ridges.

The surficial geology of Canada (Map 195) indicates that the soils in the study area consist predominantly of glacial sediments (Till), glaciolacustrine and lacustrine sediments and undifferentiated bedrock. The glacial till sediments consist largely of silty, sandy, and clayey soils formed by the direct action of glacier ice and may include areas of rock outcrops. The glaciolacustrine and lacustrine sediments consist largely of silt and clay soils with variable thickness and minor stones. The bedrock area consists of bedrock outcrops with alpine and non-alpine settings and may include colluvial deposits, till, and other minor surficial sediments. Bedrock outcrops in the study area were recorded in the Journeaux Assoc's report No. L-17-2035 rev. A (dated December 8th, 2017).

14.1.2.2 Bedrock

Atlas of Canada, Bedrock Geology Map, 3rd edition indicates that the bedrock underlying the region consists of intrusive rocks composed of granodiorite, granite, quartz diorite, and granite gneiss.

With reference to the information provided by Hydrogéologie Richelieu borehole logs (2017 and 2018), the depth to bedrock in the project's general area ranges from 1.0 m to 20.0 m.

14.1.2.3 Groundwater

With reference to the information provided by Hydrogéologie Richelieu borehole logs (2017 and 2018) the reported static groundwater in the general area of the project is at ground level to 2.5 m below the ground surface. Deeper static groundwater is reported in the northeastern portion of the study area where static groundwater depth varies between 6.0 m and 12.0 m below the ground surface.

14.2 Overview of Subsurface Conditions

Hydrogéologie Richelieu borehole records (2017 and 2018) were used as the primary source of information. Four different clusters of boreholes, described as zones 1 to 4 were identified in the study area. The borehole locations are shown in Figure 14-2.

The subsurface stratigraphy encountered in the boreholes can be summarized as follows:

- Zone 1 (PZ-01, PZ-02, PZ-03 and PZ-07)
  - Organic soils; underlain by,
  - Loose sand soils; underlain by,
  - Compact to dense sand soils (in borehole PZ-07R); underlain by,
  - Bedrock.
- Zone 2 (PZ-08 and PZ-09)
  - Organic soils; underlain by,
  - Loose silty sand soils, trace clay; underlain by,

  - Stiff silty clay soils; underlain by,
  - Dense sand and gravel soils, occasional cobbles and boulders; underlain by,
  - Bedrock in boreholes PZ-08R, and PZ-09R.
- Zone 3 (PZ-10, PZ-11, PZ-12, PZ-13, PZ-14 and PZ-16)
  - Organic soils; underlain by,
  - Loose sand to sand and gravel soils, trace to some silt; underlain by,
  - Dense sand and gravel, occasional cobbles and boulders; underlain by,
  - Very dense sand and silt till, some gravel and clay; underlain by;
  - Bedrock.
- Zone 4 (PZ-04, PZ-05 and PZ-06)
  - Loose sand soils; underlain by,
  - Bedrock.

The following sections provide a summary of the conditions encountered in the boreholes.

14.2.1 Organic Soil

A layer of organic soil is present at all borehole locations except boreholes PZ-04R, PZ-05R, and PZ-06R. The organic soil is approximately 100 mm to 800 mm thick.

14.2.2 Sand to Silty Sand

A stratum of sand was found underlying the organic soils in Zone 1 boreholes. This stratum was approximately 10 m to 26 m thick, extending to the inferred bedrock. Based on the N-value obtained from the SPTs, the state of compactness of the sand was interpreted as loose to dense. No N-values from the SPTs were reported on other Zone 1 boreholes. However, the state of compactness of the sand was interpreted as loose in boreholes logs.

A stratum of sand was found underlying the organic soils in Zone 3 boreholes (except PZ-16R). This stratum was approximately 1 m to 14 m thick, extending to the inferred bedrock in boreholes PZ-10Rand PZ-14R. Based on the N-value from the SPTs, the state of compactness of the sand was interpreted as loose to dense.

A stratum of silty sand was found underlying the organic soils in Zone 2 boreholes. This stratum was approximately 0.5 m thick. Based on the N-value from the SPTs, the state of compactness of the silt was interpreted as loose.

14.2.3 Silty Clay

A stratum of silty clay was found underlying the silty sand in Zone 2 boreholes (PZ-08MT and PZ08-R). The stratum was approximately 2.5 m thick. Based on the N-value obtained from the SPTs, the consistency of the silty clay was interpreted as stiff.

14.2.4 Sand and Gravel

A stratum of sand and gravel with occasional boulders was found underlying sand, silty sand, and silty clay soils in Zone 2 and 3 boreholes. This stratum was approximately 1 m to 4.8 m thick. Based on the N-value obtained from the SPTs, the state of compactness of sand and gravel was interpreted as dense.

14.2.5 Sand and Silt Till

A stratum of sand and silt till was found underlying sand and gravel soils in Zone 3 boreholes. This stratum was approximately 1.5 m to 6 m thick. The N-value obtained from the SPTs for this stratum ranged from 49 to 100. Based on this value, the state of compactness of sand and silt till was interpreted as very dense.

14.2.6 Bedrock

Bedrock was encountered in all boreholes at depths ranging from 1.0 m to 20 m. The following information is based on the RQD values reported in borehole logs:

- In Zone 1, the rock mass quality is classified as ranging from very poor to excellent. Of the nine RQD values recorded in Zone 1, there were less than 50% (classified as very poor to poor);
- In Zone 2, the rock mass quality is classified as ranging from poor to good. Of the four RQD values recorded in Zone 2, two were less than 50% (classified as very poor to poor);
- In Zone 3, the rock mass quality is classified as ranging from very poor to good. Of the 18 RQD values recorded in Zone 3, seven were less than 50% (classified as very poor to poor);
- In Zone 4, based on the RQD values the rock mass quality is classified as fair to excellent.

In general, these RQD values were seen in the upper 6.0 m to 7.0 m where frequent fractured zones were reported in borehole logs.

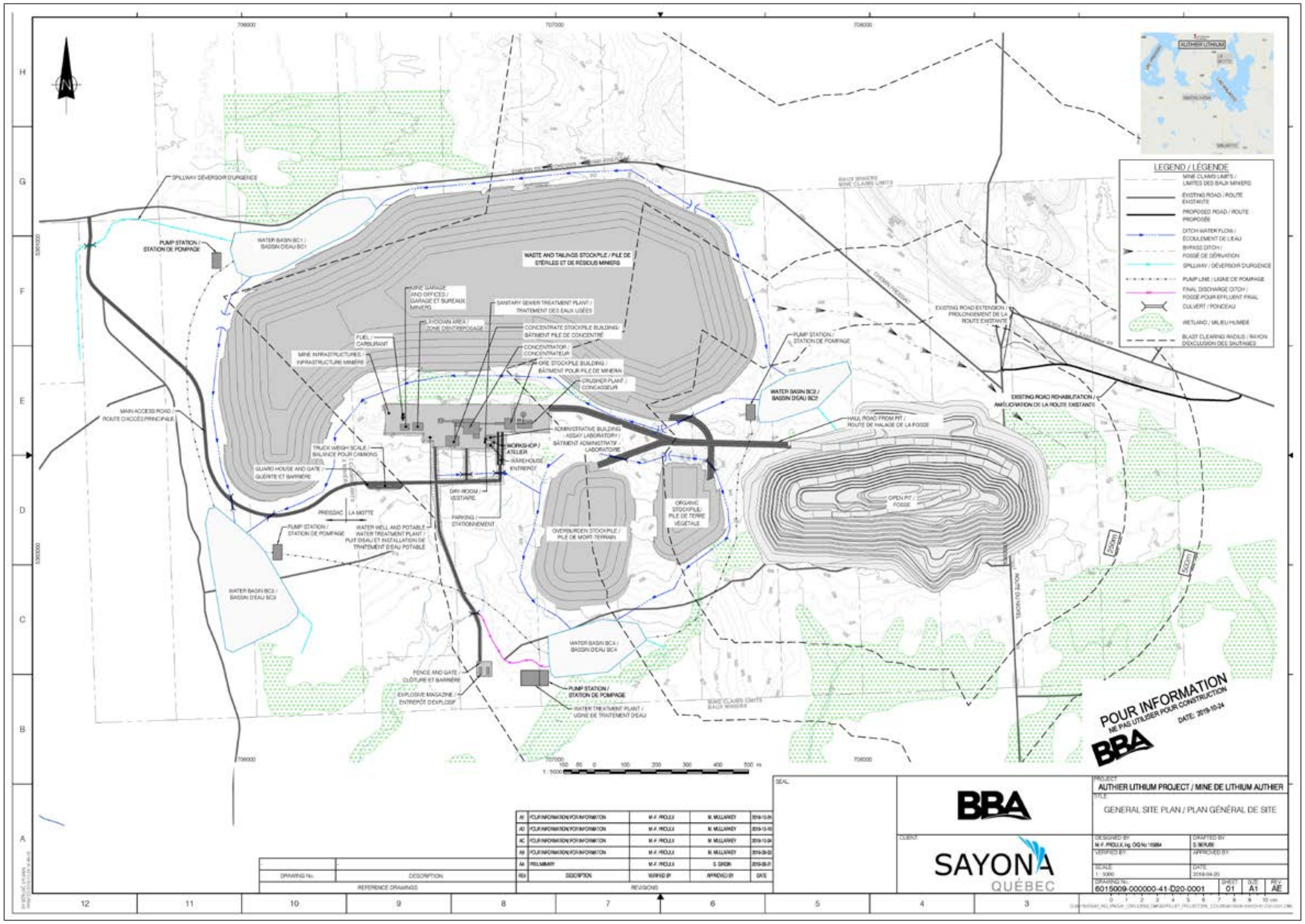


Figure 14-1: Location of the Tailings and Waste Rocks Storage Facility

14.3 Geotechnical Design Criteria

14.3.1 Materials and Parameters

14.3.1.1 Tailings

The tailings are filtered and transported to the storage facility for disposal. Filtered tailings density is 2.29 t/m^3, with a water content of 12%. A degree of saturation of 85% is assumed for the filtered tailings.

14.3.1.2 Waste Rock

Rock material that does not contain enough mineralized material to be economically processed is deposited in the storage facility. The swell factor for the waste rock material is 1.3 and the waste rock density in the pile is 2.90 t/m^3. A grain size distribution of 0 to 750 mm is assumed for the waste rock material.

14.3.1.3 Foundation Soils

The foundation conditions were assessed based on the geological and geotechnical information presented in the Hydrogéologie Richelieu study of 2017 and 2018 in the form of borehole records.

Foundation conditions on the west side of the storage facility consisted of approximately 1.0 m thick overburden soils (predominantly characterized by sandy soils) underlain by fractured bedrock (over a thickness of approximately 7.0 m). Overburden soils generally increase in thickness from west to east along cross section A.

Foundation conditions on the east side of the storage facility consisted of approximately 13.0 m thick overburden soils (predominantly characterized by sandy soils) underlain by fractured bedrock (over a thickness of approximately 8.0 m).

14.3.1.4 Material Strength Parameters

The material unit weights and effective strength parameters used in the stability analysis are provided in Table 14-1.

The shear strength of the waste rock is based on a function that defines the variation of shear strength with normal stress. The shear strength of rock materials typically reduces at higher stresses due to the crushing of particle contact points within the material and a reduction in material dilatancy. Shear strength is also related to the density and durability of the material and the particle size distribution. The strength function representative of lower shear strength rockfill (Leps, 1970) was selected based on the assumption that waste rocks may be composed of average density particles.

Table 14-1: Materials Strength Parameters

Material	Total unit weight $\gamma\ (kN/m^3)$	Dry unit weight $\gamma_d\ (kN/m^3)$	Water content (%)	Friction angle[3] φ °	Cohesion $c\ (MPa)$
Tailings	16.7	14.7[1]	10	30	-
Waste rock	21.9[2]	-	-	39	-
Foundation soils (loose sand soils)	19.0	17.0	-	28	-
Fractured Bedrock [4]	25.0	-	-	40	0.01

Note(s):
(1) Dry density of 1.5 t/m³ was used for compacted tailings.
(2) In situ density of 2.9 t/m³ and a swell factor of 30% were used for waste rock.
(3) The friction angle of the tailings is representative of a material compacted by multiple passes of heavy trucks.
Higher friction angle is used for foundation soils for post-deposition i.e., long term analysis ($\varphi = 32°$).
Average density rockfill particles, and confining normal stress of approximately $\sigma'_n = 2MPa$ are assumed to determine the friction angle of the waste rock (Leps, 1970)
(4) Material parameters for fractured bedrock and foundation soils were assumed based on experience with similar materials.

14.4 Pit Slopes

14.4.1 Summary

Sayona Québec retained Journeaux Associates (Journeaux Assoc.), a division of Lab Journeaux Inc., to assist with the pit slope design for the Authier Lithium project. The aim of the open pit mine design is to provide an optimal excavation configuration in the context of safety, ore recovery and financial return. The objective of the slope study was to meet these criteria and create a slope design that establishes walls that will be stable for the life of the open pit and beyond mine closure.

The Authier Lithium project involves an open pit excavation striking 1000 m long in the east-west direction and 600 m wide in the north-south direction, extending to a maximum depth of about 200 m.

The objective of the study was to review the existing structural geology model developed by Sayona Québec and based upon geological data collected from exploration core oriented boreholes drilled in 2016 and 2017. It should be noted that the drilled boreholes were mainly for exploration purposes, therefore they were drilled mainly perpendicular to the tabular lithium-rich pegmatite ore body dipping towards the north and not towards the pit walls.

In addition, most boreholes (26) were localized mainly on the central north side of the pit and drilled away from the pit wall limit with just a few (3) boreholes drilled on the south side. In the design of pit wall slopes, the structural geology features encountered in the exploration boreholes are assumed to be dominant in the region and projected behind the pit wall limits. It is recommended that additional geotechnical oriented boreholes be drilled to confirm the structural geology model, before the final design of the pit wall slopes.

Laboratory direct shear tests on rock joints were conducted on a number of rock cores selected from exploration boreholes drilled in November, 2017.

The existing structural geology data and the results obtained from the laboratory direct shear tests served to review the existing designed pit slopes and to recommend optimum stable economical pit slopes.

A preliminary structural analysis of the oriented structures within the Authier deposit has been undertaken for the DFS, expanding on some of the early-stage and limited geotechnical work completed for the February 2017 pre-feasibility study. Very little geotechnical information was gathered at the Authier property in previous drilling campaigns prior to Sayona Québec's ownership.

14.4.2 Method of Work

Eighteen (18) exploration core oriented boreholes were drilled in 2016 and 31 exploration boreholes were drilled in 2017. These boreholes were logged by Sayona Québec and the relevant structural geology data were reported in a database that was transferred to Journeaux Assoc.

To conduct the current review of the existing structural geology model, the following available data, supplied by Sayona Québec, were consulted and studied:

- Pre-feasibility study prepared by SGS Canada Inc., November 23, 2016;
- A report summarizing the results of the unconfined compression strength (UCS) tests done at Université École Polytechnique, in Montréal. Québec;
- Geological drilling database 2016 and 2017;
- A partial collection of photos of the rock cores recovered, but no borehole camera logging of the exploration boreholes;
- A recent structural geology summary based on oriented core data processing and interpretation prepared by the Sayona Québec geologist (Gustavo Delendatti) presenting stereonets, which summarize the major families of joints encountered in the recovered rock cores.

In December 2017, a site visit was made by Journeaux Assoc. personnel to inspect a zone of predominant bedrock within the mine lease and locate visible rock outcrops for the purpose of locating a potential mine building site. Due to high snow cover, it was difficult to observe any structural geologic features on bedrock outcrops.

During the site visit, rock core samples stored from three boreholes in the core shack were selected by Journeaux Assoc. personnel for laboratory direct shear testing. Table 14-2 lists the 2017 exploration borehole sections, from which samples were selected by Journeaux Assoc. for laboratory testing.

Table 14-2: List of the 2017 Exploration Boreholes from which Samples were Selected for Laboratory Direct Shear Tests

Borehole ID	Sample JA ID	Depth along Borehole (m)	Dip (°)	Azimuth (°)	Location Along Pit
AL-17-22	DS # 1	75.4	-60	180	North Wall Middle of Pit
AL-17-22	DS # 2	75.5	-60	180	
AL-17-22	DS # 3	175.3	-60	180	
AL-17-22	DS # 4	175.5	-60	180	
AL-17-22	DS # 5	176.8	-60	180	
AL-17-14	DS # 6	76.7	-55	180	North East Corner of Pit
AL-17-30	DS # 7	11.0	-45	180	South East Corner of Pit
AL-17-30	DS # 8	11.7	-45	180	

14.4.3 Groundwater Level

From summer 2017 to spring 2018, Richelieu Hydrogéologie Inc. carried out a hydrogeological investigation.

Twenty-four (24) boreholes were drilled with piezometer installations in all of them. The location of the hydrology boreholes is shown in Figure 14-2.

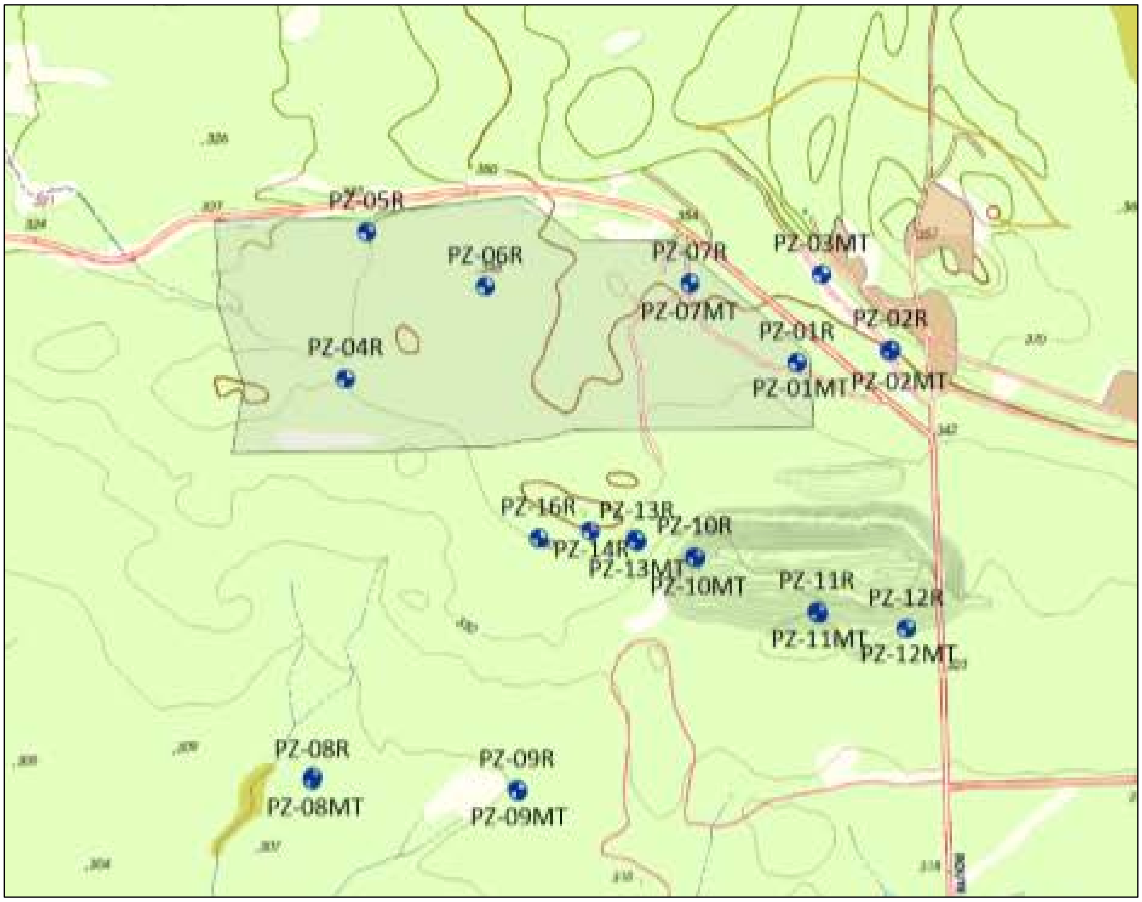


Figure 14-2: Location Plan of the Drilled Hydrology Boreholes

As observed in Figure 14-2, ten hydrology boreholes are close and six within the proposed pit limits. Table 14-3 presents the water levels measured in ten hydrology boreholes drilled near the proposed pit limits.

Table 14-3: List of the Hydrology Boreholes within the Pit Limits

Borehole ID	Date of Borehole Drilled dd/mm/year	Date of Water Level Measured dd/mm/year	Water Level Depth (m)	Ground Elevation (m)	Water Level Elevation (m)
PZ-10MT	11/07/2017	21/06/2018	0.28	330.93	330.65
PZ-10R	07/07/2017	21/06/2018	1.27	331.00	329.73
PZ-11MT	06/07/2017	20/06/2018	0.71	330.28	329.57
PZ-11R	05/07/2017	20/06/2018	0.74	330.08	329.34

Borehole ID	Date of Borehole Drilled dd/mm/year	Date of Water Level Measured dd/mm/year	Water Level Depth (m)	Ground Elevation (m)	Water Level Elevation (m)
PZ-12MT	04/07/2017	02/05/2018	0.21	326.91	326.70
PZ-12R	03/07/2017	02/05/2018	-0.58	326.79	327.37
PZ-13MT	19/07/2017	27/06/2018	2.01	337.41	335.40
PZ-13R	18/07/2017	27/06/2018	2.07	337.11	335.04
PZ-14R	17/07/2017	27/06/2018	1.81	349.43	347.62
PZ-16R	23/04/2018	28/06/2018	3.12	343.42	340.30

The hydrogeological investigation carried out by Richelieu Hydrogéologie Inc. reports water levels between -0.6 m and 3.1 m from ground level in the area of the open pit or between elevations of 326 m to 347 m. These measurements indicate that the water level is quite high, thus it will affect the stability of the existing overburden and the pit walls during excavation of the proposed open pit.

During excavation, both the surface water run-off and groundwater will be draining into the pit.

To prevent surface water draining into the open pit, a peripheral shallow ditch, combined with low berms, can be used to redirect drainage water away from the pit (Figure 14-3).

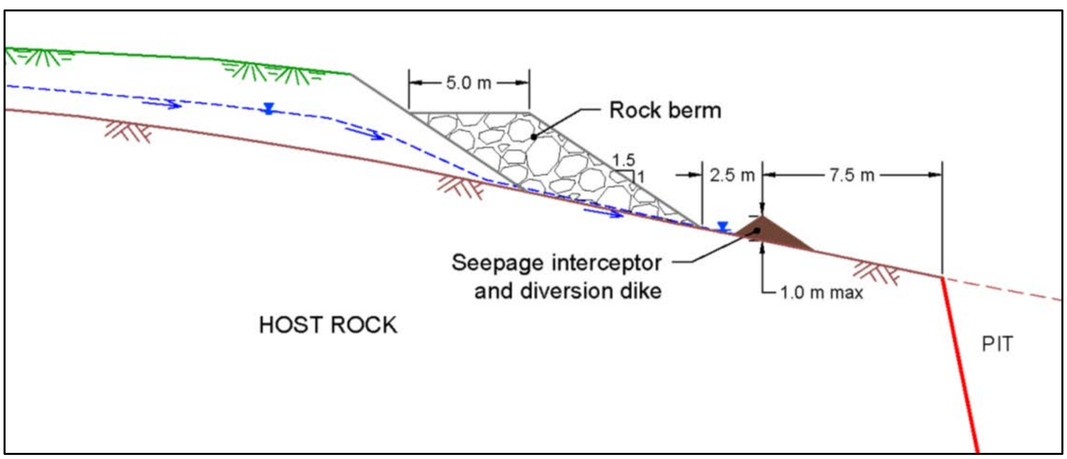


Figure 14-3: Typical Cross-section Showing a Berm for Diverting Water

As the pit is deepened and its level falls below the water level, the groundwater must be properly managed. Permeable zones, such as fault zones, will be critical locations for water flow. A discussion on site dewatering is presented in Section 14.4.7.4. The pit slope design, presented in this report, is based on dry bedrock conditions (dry pit walls).

14.4.4 Overburden

The overburden covering the ore body is mainly composed of an organic top layer followed by a layer of fine to coarse brown and grey sand with a depth varying between 1 m and 19 m along the 1,000-m long pit. Figure 14-4 presents the depths of the overburden soil layer as encountered in the exploration boreholes drilled in 2016 and 2017.

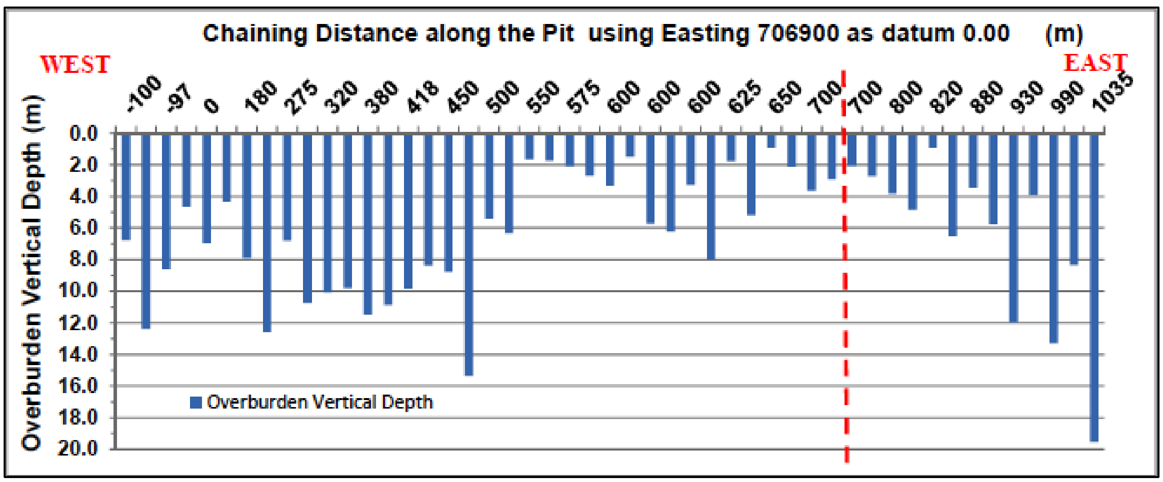


Figure 14-4: Depth to Bedrock Based on Exploration Boreholes Drilled in 2016 and 2017

As illustrated in Figure 14-4, deep overburden is encountered at both the west and east extremities of the pit. The overburden varies between 5 m and 15 m for the initial 700 m length of the pit starting on its west side. On the east end, the depth of the overburden increases progressively, west towards east, from 5 m to 20 m at its extreme east end.

The overburden at the center of the pit is relatively shallow, varying up to about 8 m at one location.

No geotechnical information was available at the time of writing this report for characterization of the overburden since most core drilled boreholes were for bedrock exploration only.

Limited information on the overburden, i.e. description and standard penetration N-Values, is presented on the hydrogeology program borehole logs. Borehole PZ-10R, located within the proposed open pit limits, indicates that the overburden depth is about 12.5 m and it is composed of a loose 0.5 m organic layer, followed by a 3 m compact brown and grey sand layer, with grey sand extending for an additional 9 m, where it changes consistency and becomes loose before reaching the bedrock at a depth of 12.5 m below the ground surface.

The deep overburden and the high water level will negatively affect the stability of the overburden slopes once the proposed open pit is excavated. Hence, the overburden at the perimeter of the pit should be excavated using a minimum slope of 4H: 1V. To attain a steeper excavation slope, the water table should be lowered either by excavating a deep collection ditch all around the pit or lowering the water table mechanically by installing pumping wells. A minimum step back of about 10 m between the pit excavation limit and the toe of the overburden slope should be respected.

14.4.5 Bed Rock Description

At the Authier property, the main lithium bearing pegmatite rich strikes east-west and dips 40 to 50 degrees to the north. The dominant host rock is an ultramafic metamorphic amphibolite mainly composed of amphibole and plagioclase feldspars with little or no quartz. As well, the amphibolite is slightly foliated as shown in Figure 14-5.

The developed structural geology model is based upon the structural geologic joints encountered in twelve (12) oriented boreholes drilled in 2016 and ten (10) oriented boreholes drilled in 2017. All structural geologic features, i.e., measurement of the inclination of angle alpha and beta of joints and fractures, were logged by Sayona personnel and transferred to Journeaux Assoc.

Figure 14-5: Foliation Observed on Amphibolite Hosting Rock

14.4.5.1 Laboratory Testing

14.4.5.1.1 Uniaxial Compression Strength (UCS) Tests

Sayona Québec tested twelve (12) rock samples in Uniaxial Compression Strength (UCS) as part of the drilling program completed in 2016. All UCS tests were carried out at Université École Polytechnique on amphibolite and pegmatite cores recovered from the exploration boreholes, drilled perpendicular to the main body and according to ASTM D-7012-14. The UCS results are presented in Table 14-4.

Table 14-4: Summary of Uniaxial Compression Strength (UCS) Tests Results

Sample ID	Borehole ID	Rock Type	Depth		Unit Weight (kN/m^3)	UCS (MPa)
			From (m)	To (m)		
1	AL-16-04	Pegmatite	122.3	122.5	25.7	173.41*
2	AL-16-10	Amphibolite	35.3	35.6	29.3	190.47*
3	AL-16-11	Amphibolite	181.3	181.5	26.0	136.46
4	AL-16-12	Amphibolite	222.6	222.8	28.4	223.79
5	AL-16-13	Pegmatite	221.3	221.5	26.3	208.49
6	AL-16-14	Amphibolite	22.6	22.8	27.0	242.52
7	AL-16-15	Pegmatite	102.0	102.2	25.6	127.31
8	AL-16-15	Amphibolite	65.2	65.4	28.7	226.59
9	AL-16-15	Amphibolite	230.0	230.2	28.3	159.93
10	AL-16-16	Amphibolite	174.0	174.2	27.1	236.14
11	AL-16-18	Amphibolite	231.7	231.9	25.7	218.75
12	AL-16-18	Amphibolite	216.7	216.9	25.7	227.33

* The strength reported is extrapolated as the scale used during the first two tests was not sufficient to reach the rupture strength of the rock.

The results of the tests show that the rock UCS varies between 127 MPa and 242 MPa, with an average value of 185 MPa. These values indicate that the rock has a very high compressive strength.

14.4.5.1.2 Direct Shear Tests on Rock Joints

The joints tested were recovered from host rock core samples drilled in 2017. The tested joints are clean and the joint roughness varied between smooth planar and wavy rough surface. The joints with a rough profile result in high shear strength compared to smooth planar joints, which exhibited lower shear strength.

Table 14-5 summarizes the results of seven direct shear tests done by Journeaux Assoc. on clean joints of amphibolite host rock core samples selected during the site visit of Journeaux Assoc. personnel in November 2017.

Table 14-5: Summary of Direct Shear Test Results on Clean Dry Host Rock Joints

Borehole ID	Sample JA ID	Vertical Depth	Depth			
			Normal Stress (Mpa)	Angle of Friction	Normal Stress (Mpa)	Angle of Friction
AL-17-22	DS #1	65	0.38	59	0.75	45
AL-17-22	DS #2	65	Large Joint Plane could not be tested			
AL-17-22	DS #3	152	0.53	54	1.07	51
AL-17-22	DS #4	152	0.4	53	0.8	51
AL-17-22	DS #5	153	0.61	56	1.21	39
AL-17-14	DS #6	63	0.62	68	-	-
AL-17-30	DS #7	8	0.51	70	1.02	64
AL-17-30	DS #8	8	0.53	60	1.07	41

Initial testing of the intact rough joints resulted in friction angles varying between 53° and 70° when the joint surface is tight and intact. When re-testing and subjecting the same sample to double the normal load applied to the slide plane in the first test, the angle of friction of joints is much lower at 39° and 64°; such decrease in shear strength being due to loss during the initial shear test of roughness or shearing of asperities in wavy rough joint planes. The seven tested rock samples indicated that the shear strength of the joints is quite high. A typical plot of the normal stress versus the shear stress obtained in the direct shear test done on a sample recovered from AL-17-22 at vertical depth of 65 m is shown in Figure 14-6. The full direct shear test results are presented in Appendix A of the full report of Journeaux Assoc.

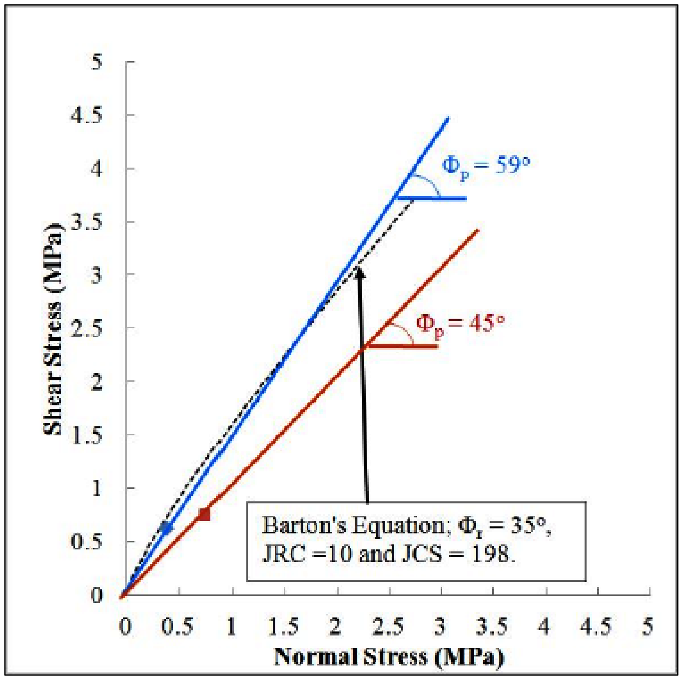


Figure 14-6: Typical Direct Shear Test Results Obtained on a Sample Recovered from AL-17-22 at Vertical Depth of 65 m

14.4.5.2 Rock Mass Properties

14.4.5.2.1 Rock Quality Designation (RQD)

Rock quality designation (RQD) measures the total length of solid pieces of fresh, slightly weathered and moderately weathered core longer than 100 mm (4 in.) against the total length of the indicated core run expressed as a percentage; RQD values rate the quality of the rock on a scale from 0 to 100%. A RQD of 90-100% rates the rock as being of excellent quality. Table 14-6 presents the RQD ranges and corresponding rock quality classification.

Table 14-6: Correlation Between RQD and Rock Mass Quality (after Singh and Goel, 1999)

S. N°	RQD (%)	Rock Quality
1	< 25	Very Poor
2	25-50	Poor
3	50-75	Fair
4	75-90	Very Good
5	90-100	Excellent

Figure 14-7 presents the RQD values versus the corresponding elevation as reported in the database transmitted to Journeaux Assoc. by Sayona.

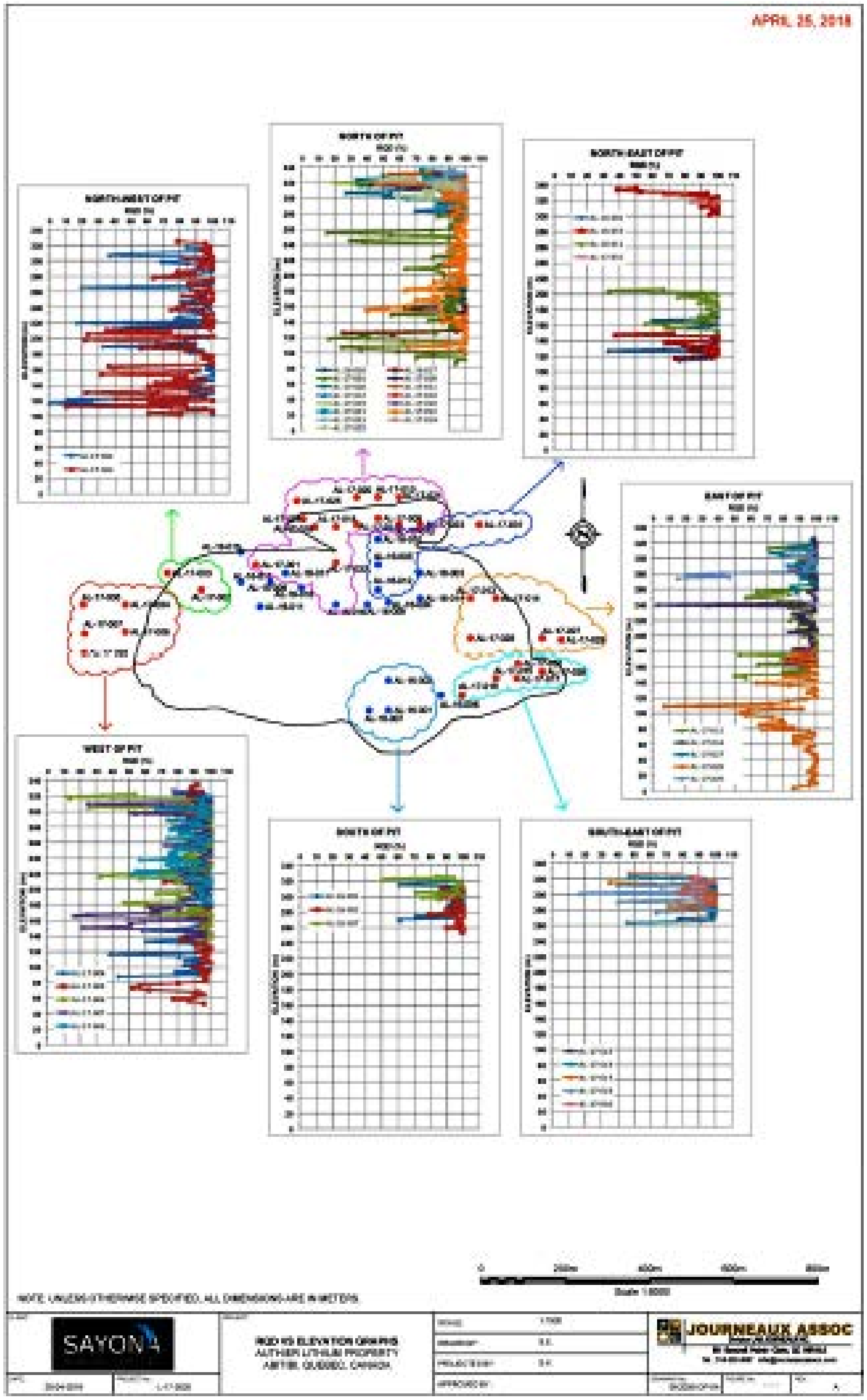


Figure 14-7: RQD Values vs Elevation for Exploration Boreholes

Most RQD values are greater than 75%, which indicates that the rock is in general of good to excellent quality. At certain localized zones, the RQD values dropped to less than 25%, which indicates that the rock is of poor quality. These zones of low RQD values may be faults or heavily fractured seams.

It should be emphasized that it was impossible to review or evaluate the reported RQD(s) since the set of photos of the rock cores recovered from the boreholes drilled in 2016 and transmitted to Journeaux Assoc. was not complete; various sections were missing. In addition, the full set of photos of the rock cores recovered from the boreholes drilled in 2017 was not transmitted to Journeaux Assoc.

14.4.5.2.2 Rock Mass Rating (RMR)

The Rock Mass Rating (RMR) classification system developed by Bieniawski (1989) was used to estimate the strength of the rock mass. The RMR system incorporates UCS, RQD, joint spacing, joint condition and groundwater condition. Each of these parameters is assigned a rating value and the sum of these yields the RMR, which varies between 0 and 100. Based on the available data and inspection of certain collected joints, the RMR 1989 of the hosting VVM mafic rock is 84, classifying the rock mass as very good quality with an angle of internal friction of rock mass greater than 45°, similar to the laboratory tests carried out on the previously sheared and destructed texture of virgin joint plane.

14.4.5.2.3 Rock Mass Strength

Hoek-Brown Failure Criterion

The evaluation of the shear strength of intact core samples or of distinct discontinuities (joints) is achievable by conducting well-established laboratory tests on recovered rock core samples. However, the estimation of the shear strength of the rock mass is more difficult, as it should account for the combined influences of intact rock, discontinuity, shear strength, the frequency and the continuity of fractures. According to standard practice, the Hoek-Brown failure criterion provides an empirical approach for evaluation of the shear strength of the rock mass. To apply this failure criterion, two (2) Hoek-Brown material constants for jointed rock mass (m and s) should be defined. Values of m and s vary with rock mass quality and lithology. These values are calculated using correlation of the values obtained on intact rock, the geological strength index (GSI) and the degree of disturbance factor (D), which measures the damage of the rock mass due to blasting.

As no extensive laboratory testing program was done at this stage of the project, the Hoek-Brown material constants were estimated based on available published tables and charts in the literature.

Table 14-7 summarizes the suggested Hoek-Brown parameters.

Table 14-7: Values of Hoek-Brown Parameters (Wyllie et al., 2004)

Hosting Rock Type	mi intact rock	S Intact rock	Geological Strength Index	Disturbance Factor (D)
Amphibolite (metamorphic slightly foliated)	26	1	55-75	1

14.4.5.3 Structural Geology within the Authier Property

The proposed pit walls follow the general orientation outline of the east-west trending tabular pegmatite body. The bedding planes (schistosity) in the open pit area are quite consistent, dipping between 45° and 50° towards the north.

The structural geology critical to the pit walls stability was evaluated from logging 4,190 structural features such as joints, veins, and faults, encountered in rock cores recovered in the drilled oriented boreholes. The structural data served to plot stereonets identifying prominent pole concentrations. These pole concentrations are grouped to identify the joints controlling the stability of the open pit walls.

The structural geology model was prepared by Sayona Québec and the stereonets were transmitted to Journeaux Assoc. to recommend stable and economical pit slopes. Figure 14-8 is the stereonet presenting the major families of joints controlling the stability of the pit walls.

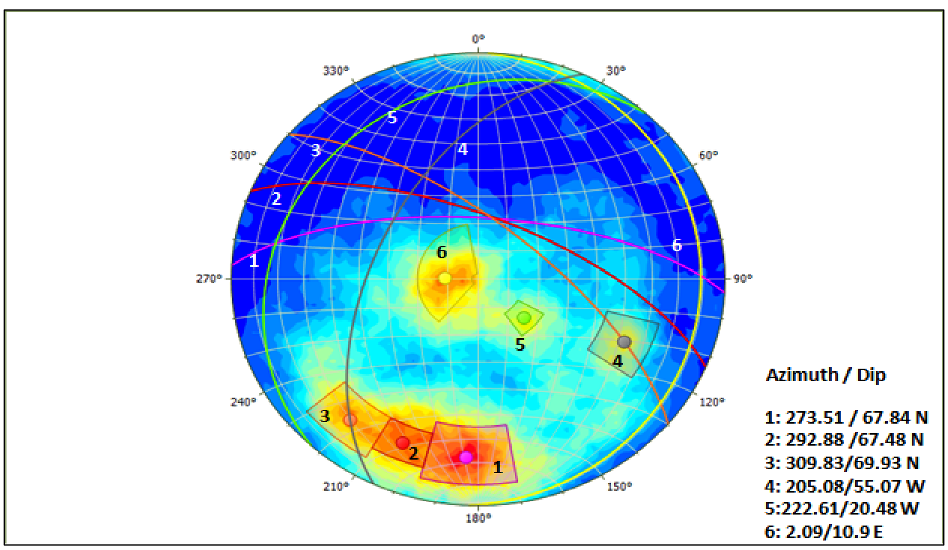


Figure 14-8: Stereonet Plot Showing the Major Families of Joints Encountered

As can be observed in the stereonet, all logged structural data can be reduced into six (6) critical joints. These joints can be grouped into three (3) design major families of joints based upon the dip direction.

Table 14-8 summarizes the design major families of joints developed in the structural geology model.

Table 14-8: Major Families of Joints

JOINT ID	Azimuth (°)	Dip Angle (°)	Notes
J1	274-310	68-70	Joint striking East South East – West North West and dipping 70° towards the North
J2	205-222	20-55	Joint striking North East – South West and dipping 20°-55° towards North West
J3	2	10	Near horizontal joint striking North – South and dipping 10° towards the East

From these structural geologic model features, the south wall is expected to be the most prone to failures as bedding planes and major joint families (J1 and J3) are dipping parallel to the wall and sliding on these planes are most probable.

On the opposite, north pit wall, these discontinuities dip at 45° into the wall and are less prone to initiate large scale shear failures unless some major fault zone behind the wall and dipping parallel or at a steeper angle could lead to a major rock slide. This should be investigated as the pit is deepened.

Faults are another critical feature controlling the stability of the pit walls. The information about the faults within the project area was retrieved from SIGEOM. Figure 14-9 presents a map with the locations and orientations of faults in the Authier property region.

However, some photos of major rock cores recovered in the drilled boreholes show zones of broken rock, which could probably indicate localized zones of faults. Figure 14-10 shows typical rock cores representing probable fault zones within the proposed pit limits.

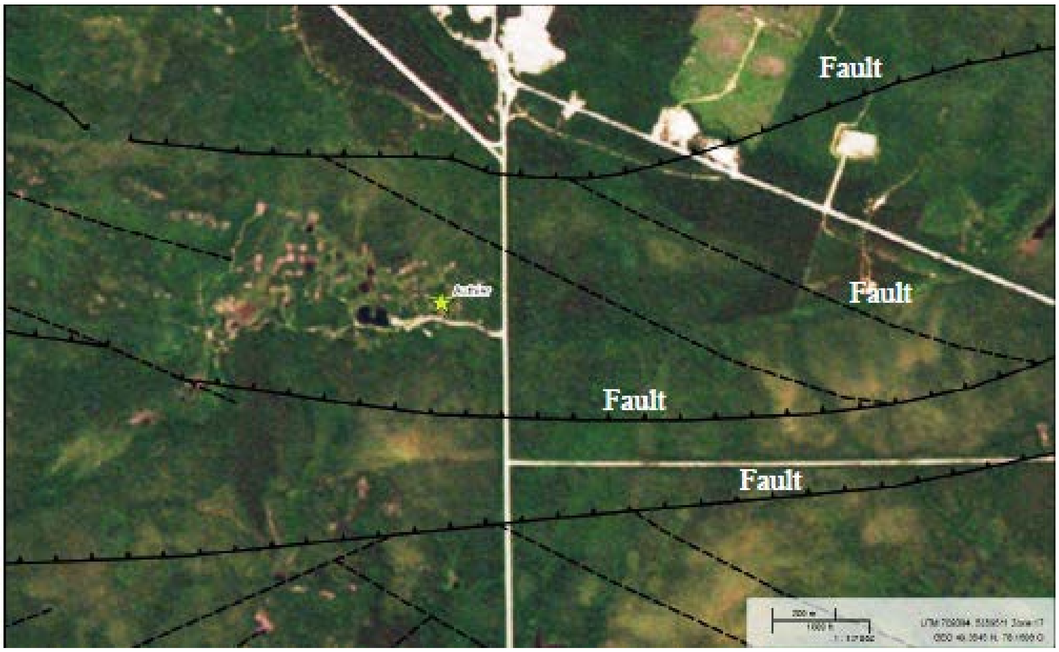


Figure 14-9: Faults Within the Authier Property

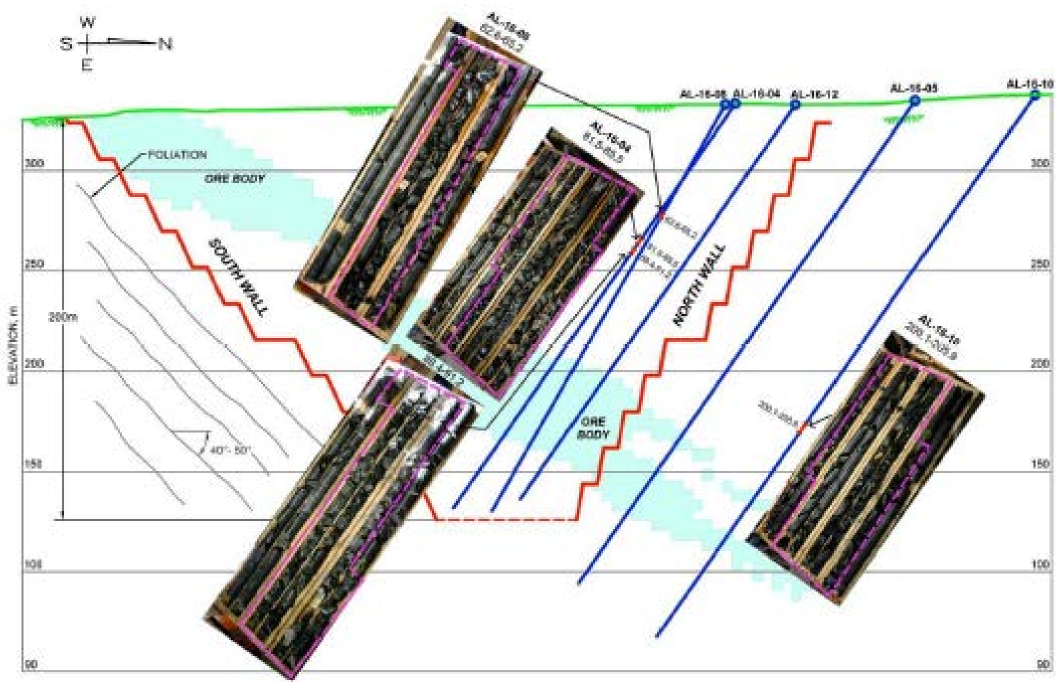


Figure 14-10: Broken Rock Cores Probable Fault Zones

Most of the fractured rock cores are localized in the central north side, the northwest end, and the northeast end of the pit. According to Sayona Québec, the drilling program schedule was very limited; therefore, no details on the orientation and the extent of these fractured zones are fully developed at this stage.

It should be emphasized that these fractured zones are critical to the stability of the pit walls, especially if these faults are dipping into the pit. As the open pit excavation reaches these fractured zones, consolidation of the rock face using anchors will be necessary to ensure the stability of the pit walls excavated at the economic recommended design slopes.

14.4.6 Pit Wall Stability Analysis

The major bedding and the steepest discontinuity plane are reported to dip towards the north within the rock formation of the project. This structural geologic logging is the controlling feature in open pit slope design.

The kinematic, i.e. local failure, stability of pit slopes was analyzed using Rockplane and Swedge (software by Rocscience) to account for major planar failures and more limited local wedge type rock falls, respectively.

In general, there are two (2) typical failure mechanisms for the Authier pit slopes:

- Plane shear failure, deep or shallow, affected by frost penetration;
- Local wedge failure.

14.4.6.1 Planar Shear Failure

Local planar failure will occur when a continuous discontinuity daylights, i.e. intersects, the bench face, creating a relatively thin knife-edged triangular block of rock that may slide if, as a rule of thumb, the inclination of the major geologic defect is steeper than the peak friction angle of the shear plane. Usually, with seasonal freeze-thaw conditions, these planes of weakness can deteriorate and lead to sliding of the block. Figure 14-11 shows a typical planar failure. The average shear strength properties ($\Phi p \sim 60°$) of the more common joint planes established in the laboratory, along with the predominance of steeply dipping joints, confirm that local failures will occur when a joint plane daylights into the pit face.

Figure 14-11: Planar Failure in a Rock Wall

Larger deep planar failures involving many benches of the high slope are wall conditions due to the fact that a major continuous discontinuity feature must be located deep behind the pit slope and well beyond the maximum depth of frost penetration. In addition, any steeply dipping major discontinuities or shear zones are critical and may require anchoring to maintain stability.

14.4.6.2 Wedge Failure

Local wedge failures will occur when two opposing dipping joints meet on a steeply inclined angle towards the pit. In this case, the intersecting line between the two joint surfaces creates a rock wedge that can slide if the angle of the intersecting line is steeper than the peak friction angle. Figure 14-12 shows a schematic of wedge failure where two intersecting joints form a wedge along a sloping face. Figure 14-13 shows a typical wedge failure.

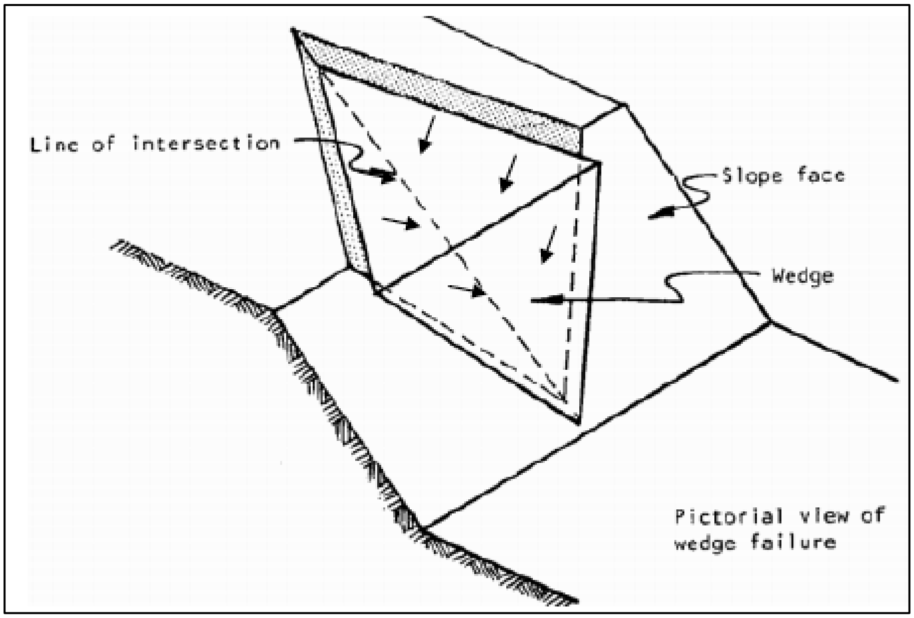


Figure 14-12: Schematic of Two Intersecting Joints Causing Wedge Failure

Figure 14-13: Wedge Failure Within Pre-shear of Rock Face

Within the rock surrounding the Authier deposit, many different weakness plane dips and orientation may exist; therefore, a variety of wedges may occur within the pit walls. Careful monitoring for such features is necessary to prevent slides. However, because of the dominant structural geology interpreted to generally dip northward and northwest, the south pit wall is expected to be most prone to wedge failure.

14.4.6.3 Summary of Pit Slope Analysis

Planar shear failures and wedge failures are the two failure mechanisms that are expected to occur locally within the open pit. Both planar and wedge failures are controlled by the shear strength properties of the joint planes, which were noted to be usually rough and wavy in the rock cores. Common rough joint planes within the amphibolite host rock formations will control most of the planar and wedge failures at the pit wall interface. Lab tests indicate that shear strength within the rough joint planes are most probable in the range of 53° to 70°. The majority of joints dip north or northwest and some may daylight within the south pit wall. Joints in the surface zone within the frost depth can slide, particularly if the pit walls are not dry.

Weaker smooth joint planes are at risk of sliding if the joint planes daylight in the pre-sheared excavation slope at even low angles.

14.4.6.4 Pit Slope Design Recommendations

As most of the lithium-rich pegmatite will be mined, the final pit walls will be mostly parallel to the formation dip and the mafic volcanic amphibolite bedrock. It is recommended that initially the north wall, where most critical joints and bedding planes dip at about 50° into the wall, is designed using an overall slope of 59° and this for the total depth of the pit of about 200 m.

For the south wall, where most critical joints and bedding planes daylight into the pit at about 50°, the slopes should be designed for an overall pit slope of 48° over the total depth of the pit of 200 m. A typical north-south cross section showing the bedding planes, the major families of joints encountered, and the recommended design slope parameters is presented in Figure 14-14.

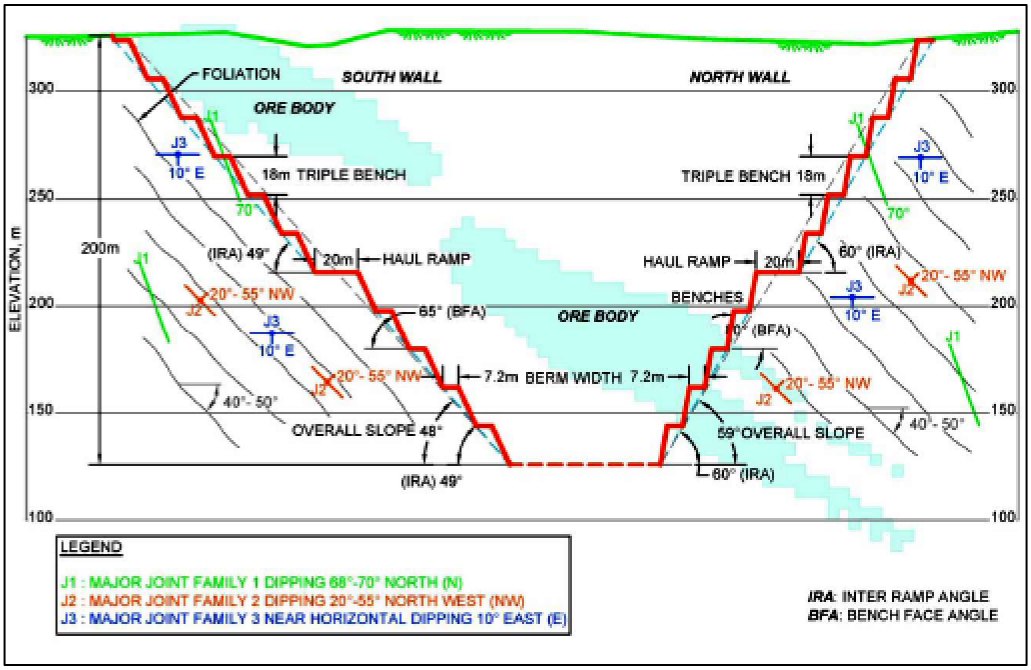


Figure 14-14: Typical North South Cross-section of the Pit Walls

Table 14-9 presents the ultimate pit slope design recommendations for the north and south walls. A drawing with the recommended design slope parameters and the locations where the recommended slopes are applicable on the north side and the south side of the proposed pit is presented in Appendix B of the Journeaux Assoc. report.

Table 14-9: Ultimate Slope Design Recommendations

Wall	Wall Azimuth (°)	Maximum Vertical Bench Separation (m)	Bench Face Angle (°)	Inter-ramp Angle (°)	Catch Bench (m)	Overall Pit Slope (°)
North Wall	25° (NE) – 335° (NW)	18	80	60	7.2	59
South Wall	155° (SE) – 205° (NW)	18	65	49	7.2	48

It should be clearly emphasized that the above recommended slopes are strictly limited to the north and south walls of the central part of the proposed open pit. Due to limited available data concerning the orientation of the fractured zones, observed mainly in the northwest corner and the east / southeast end of the pit, the slopes at the pit extremities should be reviewed after conducting additional detailed geotechnical investigations in these localized areas.

Based upon how the slopes perform during the summer and winter periods of the initial years of operation, the slopes can be adjusted, if necessary.

To determine whether steeper slopes can be used for the ultimate pit slope, it will be necessary to inspect and review the structural geology of the exposed bedrock of the excavation so a more refined structural plan can be prepared showing the strike and dip of any daylighting shear zones or faults that might have a serious impact on pit slopes. These inspections should be done for the first few lifts and then, thereafter, on a yearly basis. Depending upon the conditions of the pit walls, the stability analysis can be repeated for the deeper parts of the pit using any new structural information that is exposed so that the ultimate pit slopes can be confirmed as designed or adjusted as required.

14.4.6.5 Overburden Design Recommendations

For stability of the pit overburden, a 10 m catch bench (berm) at the overburden/bedrock contact and a 14° slope (4H:1V) are recommended (Figure 14-15). This geometry may be presently incorporated in the pit design for all areas, regardless of the thickness of overburden, and adjusted during operations as more information will be available relative to soil type, condition and overburden depth as well as ground water level and drainage.

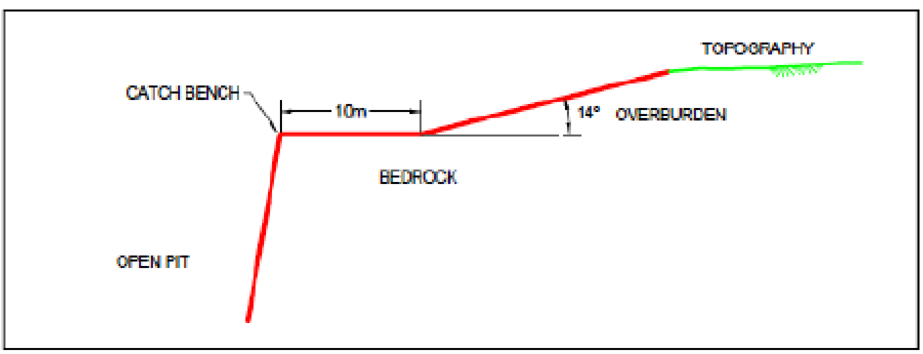


Figure 14-15: Typical Overburden Set-back Bench Geometry

14.4.7 Discussion

14.4.7.1 Structural Geology and Pit Designs

The dominant structural geology of the bedrock formation is controlled by a solid, massive mafic amphibolite host rock, which contains pegmatite intrusion rich in the lithium and trending in a general east-west direction. This orientation, with predominant north and west dipping joints, defines the principal structural weakness in the formation and will control the main instability planes affecting mainly the south wall.

14.4.7.1.1 Effect of Main Structural Features on Pit Wall Stability

The risks of instability are greatest in the south pit wall, with the foliation also dipping at 45° or more and near the pit slope; foliation mass sliding can occur if secondary joints dip at low angles 50° into the pit.

Pre-shearing techniques are recommended since mass excavation with large powder factors lead to considerable back-break, resulting in the opening of joint planes and other discontinuities, ultimately leading to unstable rock sections along the wall. As well, detonation of charges in the nearest rows parallel to the pit walls is preferable since a detonation towards the open pit area will damage the structural geology of the pit wall face.

Flatter dipping intersecting joint systems isolate large rock volumes resting on low dipping sliding surfaces, which control the risk of instability and rock falls. This instability depends upon the surface roughness of the joint surface and the relative strength or hardness of the corrugation along the sliding surfaces. Naturally, water seeping along the unfavourable joint planes behind the pit wall weakens the contact material and increases the risk of instability.

The presence of these unfavourable structural conditions must be searched for and identified as the pit is deepened, and their destabilizing effects minimized by appropriate and strategic drilling of horizontal relief and drainage holes.

Exposed faults during the open pit excavation, if any, should be photographed and brought to the attention of the designer for stability consideration. Naturally, if the fault is cutting the wall perpendicularly to the pit face, it will not cause serious slope failure issues, it drains well and drainage holes may be required to lower the water table to release pressure on the pit wall and eliminate heavy ice pressure during the freezing season. Faults oriented parallel or near parallel to the pit and dipping towards the pit can cause significant planar or wedge failures.

14.4.7.2 Degradation of Host Rock

Certain rock types are prone to degradation when exposed to air and/or water, as well as freezing and thawing cycles or changes in confining stress. Generally, the massive solid amphibolite host rock is not expected to be sensitive to these elements.

14.4.7.3 Minimizing Slope Instability by Pre-shearing Blasting

Mining of the ore body should be carried out using pre-shearing techniques, following generally acceptable MTQ procedures. This involves closely spaced (~1 m) inclined pre-shear holes drilled parallel to the bench face angle at 10V:1H and taken at about 1 m or 1.5 m below the bench level. Blast holes on bench levels should be no longer than 12 m if deviation of the drill hole is to be minimized.

In addition, the protection of the bench shoulder should be a serious objective, and blasting holes within 1.5 m of the shoulder must be avoided. A continuous production hole should be drilled parallel to the bench face to minimize damage due to back break and opening of discontinuities or predominant joints.

To avoid isolated rock falls rebounding onto lower parts of the slope, the horizontal berm should have a 3% slope towards the pit wall. During removal of the blast rock, the pre-sheared rock face must be carefully scaled and all wedge zones removed, if considered at risk to slide. At this time, the finished bench surface should be properly sloped downwards (10%) towards the pre-shearing blast holes. Initially, this will minimize falling boulders from bouncing from the ledge and falling to lower benches. Figure 14-16 shows a photograph of a pre-sheared rock face over three (3) 15-m high benches.

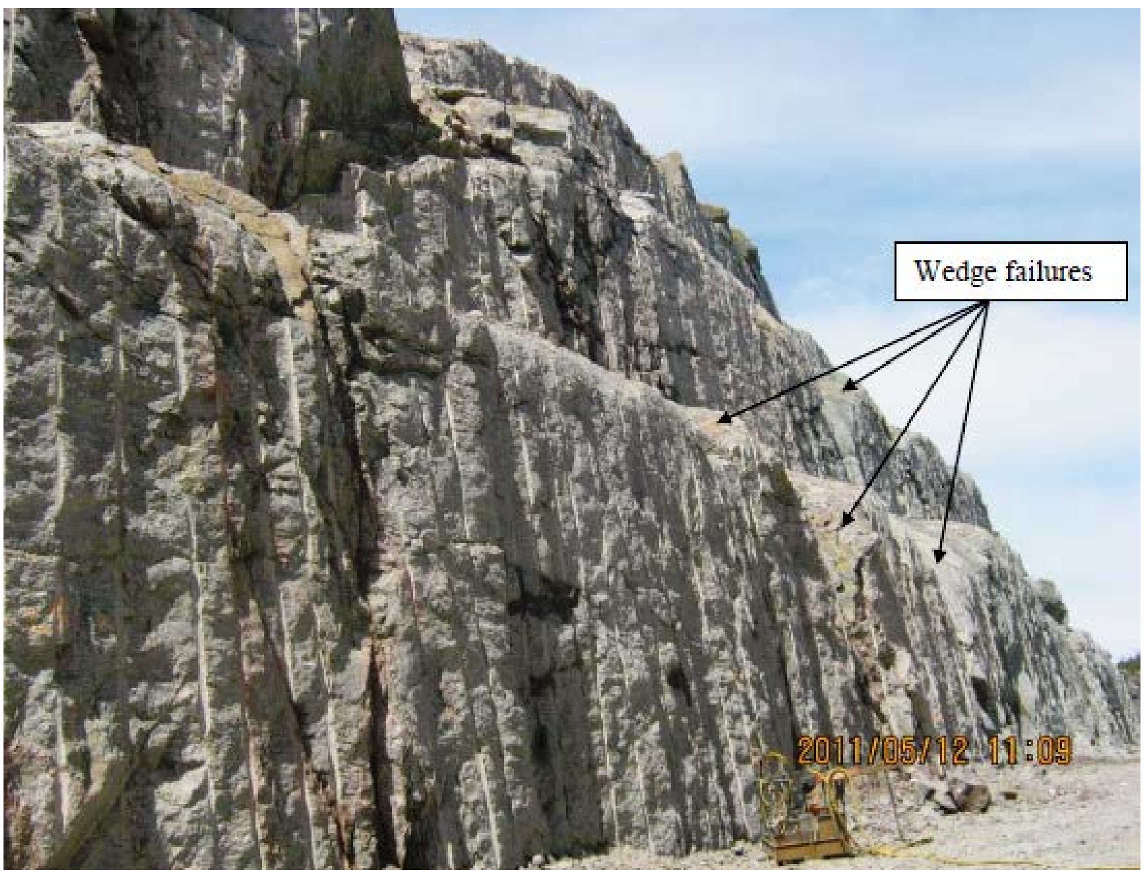


Figure 14-16: Pre-shearing Blast Over Three (3) 15-m High Benches

14.4.7.4 Excavation Control and Scaling

During the excavation of the pit wall, careful removal of loose blocks or potentially unstable material should be done along the final face of the slopes using shovels (Figure 14-17). This will reduce future clean-up costs and reduce rockfall hazards for roadways. Scaling is usually an effective technique for stabilizing slopes for periods between two and ten years, depending upon site condition. It is not a permanent solution, but it is relatively inexpensive and may be used as a short-term strategy for the beginning of the mine life.

Figure 14-17: Scaling Methods; (left) Using an Excavator, (right) Using a Lift and Steel Bars to Pry Loose Rocks

14.4.7.5 Detailed Pit Slope Inspections and Slope Monitoring

The bedrock along the walls of the open pit will be subject to seasonal freezing and thawing cycles and could also be subject to some local lateral displacement of the rock face, particularly in areas where seepage waters appear on the face. Such movements could indicate some signs of slope instability. To help identify any movements, it is prudent to install some lateral movement detection devices along the haul road. These are usually spaced every 200 m at both the 50 m and 150 m pit levels. Regular monitoring of well-secured and anchored pins, installed a minimum of 10 m into the pit walls, will establish whether lateral movements are occurring. These movement measurements will help in identifying what stabilization measures may be required. Detailed mapping of the structural geology should be done for the first few lifts and then thereafter on a yearly basis as the pit is excavated. From these observations the lateral displacement monitor can be positioned as well as any horizontal drainage holes in seepage zones.

14.4.8 Conclusions and Recommendations

The examination of the bedrock from the exploration boreholes and the related structural geology described in literature leads to the following conclusions:

- The Authier deposit is oriented east-west and is bounded within a massive mafic amphibolite host rock, which is mainly composed of amphibole and plagioclase feldspars, with little or no quartz;
- There are three dominant joint systems controlling the stability of the pit walls as follows:
  - J1: Major joint family 1 dipping 68°-70° toward the north (N);
  - J2: Major joint family 2 dipping 20°-55° toward the north west (NW);
  - J3: Major joint family 3 near horizontal dipping 10° towards the east (E).
- Except for local fault zones, the analysis of rock cores indicates excellent rock quality with RQD values between 90% and 100%, and generally a recovery of 100%;
- Pre-splitting or pre-shearing techniques should be used for shaping the bench faces of the permanent ultimate pit slopes;
- The preliminary stability analysis indicates stable slopes for a 200 m deep pit with an overall pit slope (OPS) of 59° (north wall) and 48°(south wall) and inter-ramp angles (IRA's) of 60° to 49° with 7.2 m wide catch benches with face angles (BFA's) of 80° and 65° over 18 m high triple benches. The pit slope design parameters provided are based on dry pit wall conditions;
- Based upon the proposed design conditions and the limited depth of the open pit (200 m), large scale slip circle failure is not expected;
- Local plane and wedge shear failures are expected to occur locally around the pit, particularly where steeply dipping discontinuities intersect the bench faces at obtuse or oblique angles to the slopes. These can be safely controlled by properly scaling and monitoring the pit walls as the excavation is deepened;
- Regular inspections of the structural geology along the pit wall are required for the first few lifts and then thereafter on a yearly basis to validate the structural model deduced from the cores;
- Monitoring of movements in the pit walls is required at widely spaced intervals on at least two pit levels (e.g., 50 m and 150 m levels);
- The south wall of the pit is more exposed to plane and wedge sliding, while the north wall of the pit is more stable as the major bedding planes are dipping towards the north and the major joints sets dip towards the north and the west.

15. ORE RESERVES ESTIMATE

15.1 Introduction

The ore reserves estimate was completed by BBA Inc. in June 2018 and is based on the May 2018 block model prepared by Dr. Gustavo Delendatti. This is the same model that was used to report the mineral resources presented in Chapter 13 of this report. Reporting of mineral resources and ore reserves has been carried out in accordance with JORC reporting standards per ASX Listing Rule 5.

The ore reserves have not changed for this updated definitive feasibility study (UDFS) relative to the DFS that was prepared by BBA with an effective date of February 2019. The only changes made for the mine design for the UDFS were an update of the life-of-mine (LOM) production plan with a revised processing capacity of 2,600 tonnes-per-day (actual 2,418 tonnes-per-day as average feed rate over 365 days with 93% availability) as well as new waste and overburden piles. The mine equipment fleet, workforce and cost estimate was updated based on the revised mine plan.

15.2 Resource Block Model

The resource model for the project was provided to BBA by Sayona via a web link. The resource model was supplied in a file called “2018AuthierID3.csv”. The model was supplied with the 3D wireframes used to define the different lithological zones in a total of 12 DXF files.

Dr. Gustavo Delendatti prepared several resource estimates using different interpolation methods. BBA was instructed to use the block model interpolated using the inverse distance cubed method. The file provided contained only blocks that were included in the three main pegmatite zones. The resource estimate considers a parent block size of 3 m x 3 m x 3 m. The resource model considers a constant pegmatite density of 2.71 t/m³.

15.3 Topography Data

Sayona provided BBA with a LiDAR topographic survey completed in 2016 by Geoposition arpenteurs géomètres (file: ‘20161108_Courbes_Geoposition_La_Motte_NAD83_MTM10.dwg’).

Topographic contours were provided at 0.5 m intervals for the project site in the UTM NAD 83 coordinate system. This surface was used as the reference datum for the ore reserves estimate.

15.4 Mining Block Model

Based on the resource model described above, BBA created a mining block model to be used for mine design and planning purposes. Host rock (non-pegmatite) and overburden material was added to the resource model and sub-celled along the boundaries of the different material contacts. Material densities of 1.90 t/m^3, and 2.90 t/m^3 were used for overburden and host rock material, respectively.

The sub-celled model was then regularized to the parent block size of 3 m x 3 m x 3 m. Based on the scale of the proposed operation and type of loading equipment, an SMU of 3 m x 3 m x 3 m is considered acceptable. Lithium grades and block densities were averaged during the regularization. Resource classification was conserved from the resource model while the block material type was coded based on majority. The final mining block model is called “310.dm” created from the Deswik project “Projet Authier v7.dcf”. The model was then exported to MineSight for mine planning as “505.csv”.

15.5 Modifying Factors

For the conversion of mineral resources to ore reserves, it is necessary to consider and apply a variety of *modifying factors*. Those that are applicable to the project are discussed in detail below.

15.5.1 Model Recoveries

The metallurgical recovery was determined from the results of metallurgical test work performed on samples from the Authier Lithium deposit to produce a lithium concentrate of 6% Li_2O. A constant recovery of 80% has been applied for the pit optimization. Additional metallurgical test results have been received during the course of the DFS and the metallurgical recovery was reduced to 78% for the financial analysis. BBA did not run pit optimization at 78% recovery. Further information on the calculation of metallurgical recovery can be found in Section 12.

15.5.2 Mill Cut-off Grade Calculation

The breakeven cut-off grade (COG) is calculated considering costs for processing, G&A, and other costs related to concentrate production and transport. Table 15-1 presents the parameters used to determine the mill COG. Based on a lithium concentrate selling price of $780 CAD per tonne ($600 USD at an exchange rate of $1.30 CAD / $1.00 USD), the COG would be 0.27% Li_2O. However, due to metallurgical recovery limitations, an artificially elevated COG of 0.55% Li_2O was selected based on iterative analysis. Note that the costs presented in Table 15-1 are preliminary. The final DFS costs are presented in Chapter 25.

Table 15-1: COG calculation parameters

Parameter	Units	Value
Metallurgical Recovery	%	78
Gross 6% Li_2O Price	USD/t con	600
Selling Cost	USD/t con	60
Royalties	USD/t con	12.00
Net 6% Li_2O Selling Price	USD/t	600
Con Grade	%	6
Exchange Rate	USD/CAD	0.77
Processing Cost	CAD/t (milled)	18.85
Tailing Transportation Cost	CAD/t (milled)	0.46
Ore Re-handling	CAD/t (milled)	0.10
G&A Cost	CAD/t (milled)	6.00
Calculated Cut-Off Grade	% Li_2O	0.27
Final Elevated Cut-Off Grade	**% Li_2O**	**0.55**

15.6 Dilution and Ore Loss

Lack of selectivity along the ore / waste contacts resulting from the use of large scale mining equipment, in combination with other operational factors, typically results in additional ore losses and dilution from what is included in the resource estimation.

A detailed dilution model was developed by BBA and coded into the mining block model. This was then used throughout the mine planning process. This section provides a summary of the methodology used and the full description is presented in the technical memorandum “6015003-000000-4M-ERA-0001-RAA (Dilution & Ore Loss Calculation).docx”.

The following presents the process for creating the mining model:

- The addition of waste and air zones to the resource model;
- Regularization of the resource block model to the mining SMU (3 m x 3 m x 3 m);
- Application of dilution and ore loss factors to blocks above the cut-off grade (0.55% Li2O).

The dilution and ore loss was estimated on a block-by-block basis taking into consideration the type of rock and the lithium oxide grade of the analyzed block as well as those of the neighbouring blocks.

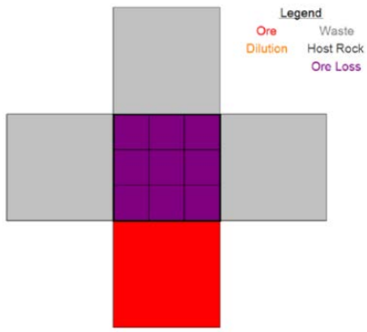
To determine whether a block would be fully or partially considered as dilution or ore loss, it was important to know the type of material surrounding the block. Every block in the block model was given an I, J and K index that corresponds to its position in the block model relative to the origin. For the Authier Project, the ore will be mined in 3-metre high flitches, which corresponds to the height of each block. Given this mining practice, it was assumed that the operation will be able to effectively mine each flitch and thus the material above or below a block will not be influenced. The information of the blocks in the I+1, I-1, J+1 and J-1 positions were interrogated and based on this information, hypotheses regarding dilution and ore loss can be evaluated.

The first hypothesis considered was that any block of ore that was surrounded by 3 or 4 waste blocks, or sandwiched between 2 waste blocks, would be considered as complete ore loss.

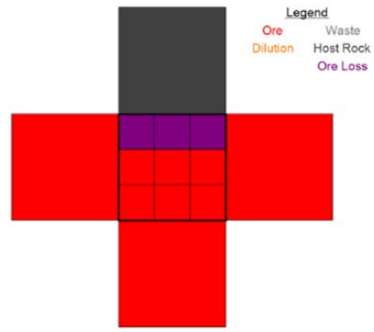
The second hypothesis considered relates to the sensitivity of the metallurgical process to contaminants. In order to avoid negatively impacting the metallurgical recovery of the lithium oxide, it is important to minimize the amount of host rock sent to the mill. Therefore, any ore blocks that neighbour a host rock block would have a 1-metre thick skin considered as ore loss along the edge. A 1-metre thick skin was considered since this volume would be the equivalent of mining an ore block to a reasonable working face angle of 56°.

The third hypothesis considered was if a waste block was surrounded by three (3) or four (4) ore blocks, the block would be considered as full dilution. Additionally, if a waste block was sandwiched between two (2) ore blocks that had no ore loss, then the block would also be considered as full dilution.

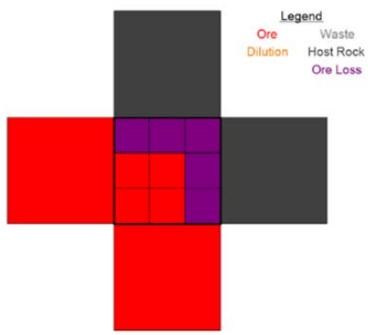
The fourth and final hypothesis considered if a waste block touched one (1) or two (2) ore blocks with no ore loss, a 1-metre thick skin is considered as dilution along the edge. This final step completed the process of preparing a mining block model. These hypotheses are presented graphically in Figure 15-1.



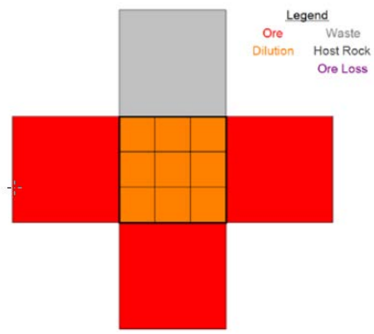
Plan view of complete ore loss of an ore block



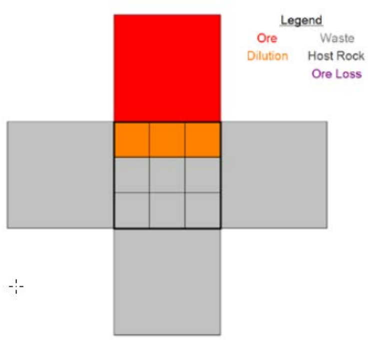
Plan view of partial ore loss when an ore block is surrounded by one host rock block



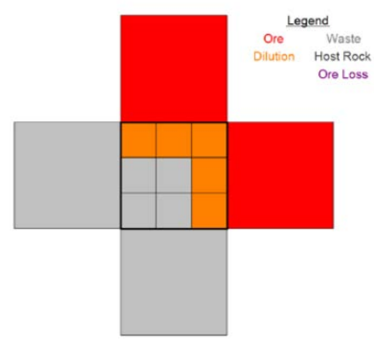
Plan view of ore loss when an ore block is surrounded by two host rock blocks



Plan view of full dilution of a waste block



Plan view of partial dilution when a waste block is surrounded by one ore block



Plan view of partial dilution when a waste block is surrounded by two ore blocks

Figure 15-1: Hypotheses regarding dilution and ore loss

The dilution and ore losses calculated in the final pit design are presented in Table 15-2 below. It should be noted that no considerations were made for any operational errors such as material being trucked to the wrong destination. An additional ore loss factor could be added to account for such losses.

Table 15-2: Dilution and ore loss in the final pit design

Parameter	Tonnes in final pit (Mt)	Factor (%)	Grade (% Li_2O)
Dilution	0.5	4.4	0.43
Ore Loss	0.8	6.7	0.88

15.7 Pit Optimization Inputs

Pit optimization for the DFS was completed using the Pseudoflow command in Deswik CAD. Inferred resources were not considered as potential run of mine (ROM) feed for the pit optimization.

The input parameters used for the DFS pit optimization are presented in Table 15-3 and detailed in the Mine Design Criteria document prepared by BBA (6015003-00000-4M-EDC-0001-R0.pdf). Note that the selling prices, costs and technical parameters used were based on the best available information at the time of the study, including costs from the 2017 Updated Pre-feasibility Study and geotechnical information from Journeaux Assoc.'s report (2018).

It should be noted that the pit shells were not recreated in this updated study, however, the discounted cash flows were updated in Table 15-4 based on the revised mine lives that consider a mill throughput of 2,600 tonnes per day.

Table 15-3: DFS pit optimization parameters for the Authier lithium deposit

Parameter	Value	Units	Notes
Canadian dollar exchange rate	1.30	CAD/USD	J. Gagné (Sayona) 2018/02/08
Mining Costs			
Overburden	3.00	CAD/t mined	
Ore	3.10	CAD/t mined	Based on PFS update average mining cost (3.14$/t moved)
Waste	2.95	CAD/t mined	
Inc. Haul Cost per Bench	0.014	CAD/t/bench	BBA estimate: 6 m bench & CAT 775
Ref. Bench	320	m elev.	BBA estimate
Ore Re-handling	0.10	CAD/t milled	BBA estimate: CAT 988. Assume 50% of ore re-handled
Processing and Other Costs			

Parameter	Value	Units	Notes
Process Cost	18.85	CAD/t milled	From updated PFS
Tailing Transportation Cost	0.46	CAD/t milled	BBA Estimate: CAT 775
G&A Cost	6.00	CAD/t milled	J. Gagné (Sayona) 2018/02/08
Transportation Cost (Mine to Port)	60.00	CAD/t conc.	J. Gagné (Sayona) 2018/04/06. For transport and ship loading
Metallurgical Recovery	80	percent	PFS Update
Revenue Model & Economics			
Spodumene Selling Price	600	USD/t conc.	J. Gagné (Sayona) (2018-04/05). 6.0% Li_2O FOB port
Royalty	12.00	USD/t conc.	J. Gagné (Sayona)
Discount Rate (%)	8.00	%	J. Gagné (Sayona)
Pit Slopes			
Overall Pit Slope - South Wall	50	degrees	Journeaux (adjusted for ramp)
Overall Pit Slope - North Wall	50	degrees	
Other Parameters			
Pit Optimization Boundaries	None		
Setback from infrastructure	-	m	J. Gagné (Sayona)
Setback from watercourse	-	m	J. Gagné (Sayona)
Dilution and Ore Loss	Included		Estimated on a block-by-block basis

The optimized parameters do not necessarily correspond with the final design parameters used in the DFS.

The geotechnical consultant, Journeaux Assoc., provided recommendations for the bench face angle (BFA), inter-ramp angle (IRA), and catch bench width. These recommendations were used to calculate the overall pit slope for each sector, and adjusted to include the assumed ramp placement.

15.8 Pit Optimization Results

The optimiser estimates a ‘best’ case and a ‘worst’ case discounted value. The best case requires that each shell be mined sequentially while the worst case mines the deposit on a bench-by-bench basis. The best case is generally impracticable as shell increments can be very small and therefore not minable by themselves. The worst case is always achievable but gives much lower values. In practice, a compromise between the two cases is generally achieved by staging the pit using suitable cutbacks. An average of the best and worst discounted values (DCFBEST) has been calculated and used as a measure to compare optimisation results. A discount rate of 8% and ROM feed rate of 0.88 Mt per year have been used in this analysis.

The values returned by the optimizer do not include capital investments and are only used as a relative indicator of the sensitivity of the project to changes in costs, etc. Mine designs based on the shells will typically add 10% extra waste with some potential loss of ore. This is due to the requirement of taking into account the minimum mining width, access requirements and other practical mining constraints.

The revenue factor of 0.80 pit shell was selected by Sayona as the final pit limits. This selection was based on several factors including:

- ROM feed (>12 Mt);
- Feed grade (>1.00% Li_2O);
- Total waste tonnes (<84 Mt);
- DCF_{AVG} NPV at or near maximum value.

Table 15-4: DFS pit optimization results

Revenue Factor Shell	ROM Feed	Grade	Waste	Strip Ratio	Mine Life	DCF_{BEST}	DCF_{WORST}	DCF_{AVG}
	(Mt)	(% Li_2O)	(Mt)	n/a	(Yrs)	(M$)	(M$)	(M$)
0.25	0.0	1.32	0.0	1.04	0.0	0.8	0.8	0.8
0.30	0.3	1.22	0.2	0.63	0.3	23.5	23.5	23.5
0.35	0.9	1.13	1.1	1.22	1.1	63.6	63.5	63.6
0.40	1.7	1.05	2.2	1.27	2.0	103.5	102.6	103.1
0.45	2.3	1.01	3.3	1.45	2.6	124.5	122.8	123.6
0.50	3.3	1.01	8.1	2.42	3.8	164.3	160.5	162.4
0.55	6.4	1.01	30.2	4.70	7.3	245.6	233.4	239.5
0.60	10.0	1.02	61.1	6.11	11.3	307.0	273.2	290.1
0.65	10.9	1.01	68.6	6.30	12.3	318.5	276.7	297.6
0.70	11.3	1.01	72.2	6.40	12.8	322.6	276.4	299.5
0.75	12.0	1.01	80.0	6.64	13.6	328.6	272.6	300.6
0.80	**12.3**	**1.00**	**83.5**	**6.76**	**14.0**	**330.5**	**270.4**	**300.5**
0.85	12.5	1.00	84.8	6.81	14.1	331.1	269.1	300.1
0.90	12.6	1.00	87.0	6.89	14.3	331.7	266.9	299.3
0.95	12.7	1.00	88.7	6.96	14.4	331.9	265.1	298.5
1.00	12.9	1.00	91.0	7.06	14.6	332.1	262.4	297.3
1.05	13.0	1.00	92.5	7.14	14.7	332.1	260.6	296.3
1.10	13.1	1.00	94.9	7.26	14.8	331.8	257.4	294.6

Revenue Factor Shell	ROM Feed	Grade	Waste	Strip Ratio	Mine Life	DCF_{BEST}	DCF_{WORST}	DCF_{AVG}
	(Mt)	(% Li_2O)	(Mt)	n/a	(Yrs)	(M$)	(M$)	(M$)
1.15	13.1	1.00	96.2	7.32	14.9	331.6	255.8	293.7
1.20	13.3	1.00	98.8	7.45	15.0	331.0	251.7	291.4
1.25	13.3	1.00	100.9	7.56	15.1	330.5	248.7	289.6
1.30	13.4	1.00	102.0	7.62	15.2	330.2	247.3	288.7
1.35	13.5	1.00	103.6	7.71	15.2	329.7	244.8	287.3
1.40	13.5	0.99	105.3	7.79	15.3	329.2	242.1	285.6
1.45	13.6	0.99	105.9	7.81	15.4	329.0	241.1	285.1
1.50	13.6	0.99	107.5	7.90	15.4	328.4	238.8	283.6

These results are presented graphically in Figure 15-2.

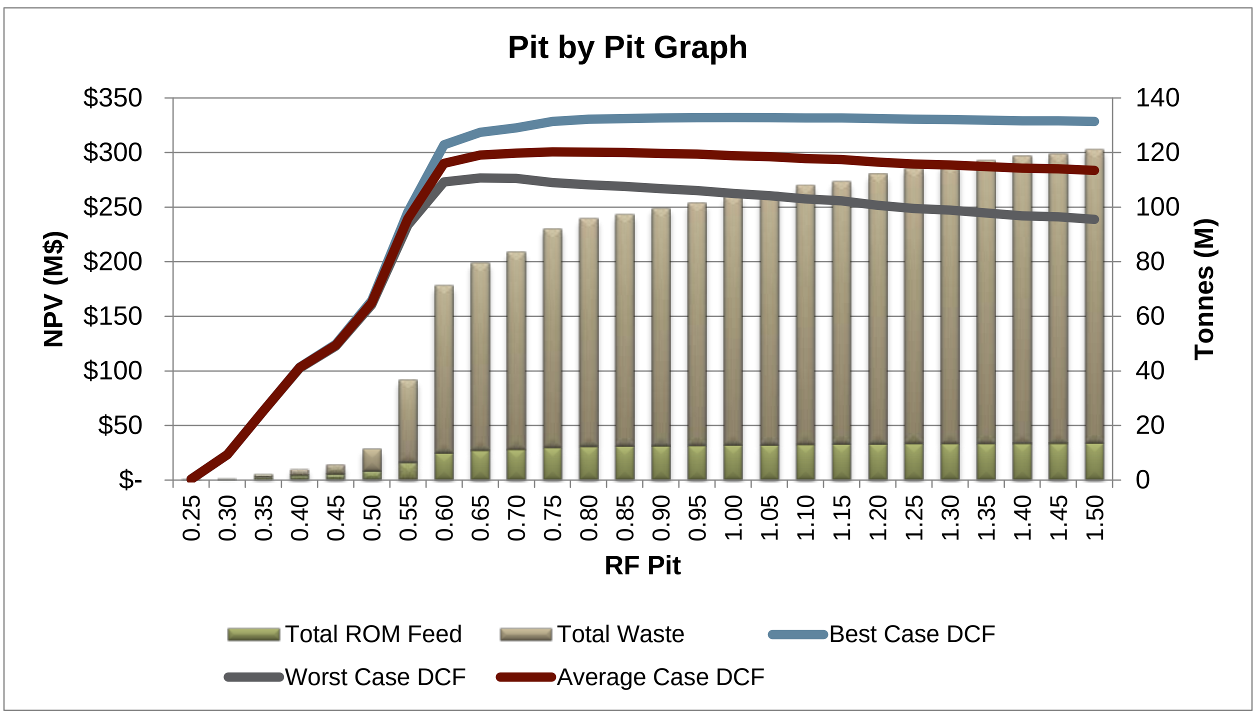


Figure 15-2: DFS pit optimization results

With the exception of selling prices, BBA did not perform a sensitivity analysis on other parameters. It is recommended that pit optimization sensitivity be conducted on the following parameters:

- Metallurgical recovery;
- Overall pit slopes;
- Dilution and ore loss.

15.9 Mine Design

15.9.1 Open Pit Geotechnical Parameters

The geotechnical requirements for the UDFS pit design were prepared by Journeaux Assoc. with their recommendations provided in a report entitled “Open Pit Slope Design Authier Lithium Project Feasibility Study”. Recommendations were provided for the overall slope angle (OSA), inter ramp angle (IRA), bench face angle (BFA) and catch bench width. For design purposes, the following IRA, BFA and triple-bench arrangement were retained, and are summarized by sector in Table 15-5.

Table 15-5: Pit design geotechnical parameters

Pit Slope Sector	OSA	IRA	BFA	Catch Bench Width (m)
North	59°	60°	80°	7.2
South	48°	49°	65°	7.2
Overburden	-	-	14.0°	-

Journeaux recommended overall pit slopes of 59 degrees and 48 degrees for the north wall and south wall, respectively. It represents an increase of 3-4% compared to the UPFS report.

BBA recommends that further geotechnical work be undertaken prior to advancing to the next stage of the project.

An illustration of the different slope zones is presented in Figure 15-3. Journeaux did not specify the parameters for the transition zone. BBA has assumed that the values for the transition zone are between the north and south wall values.

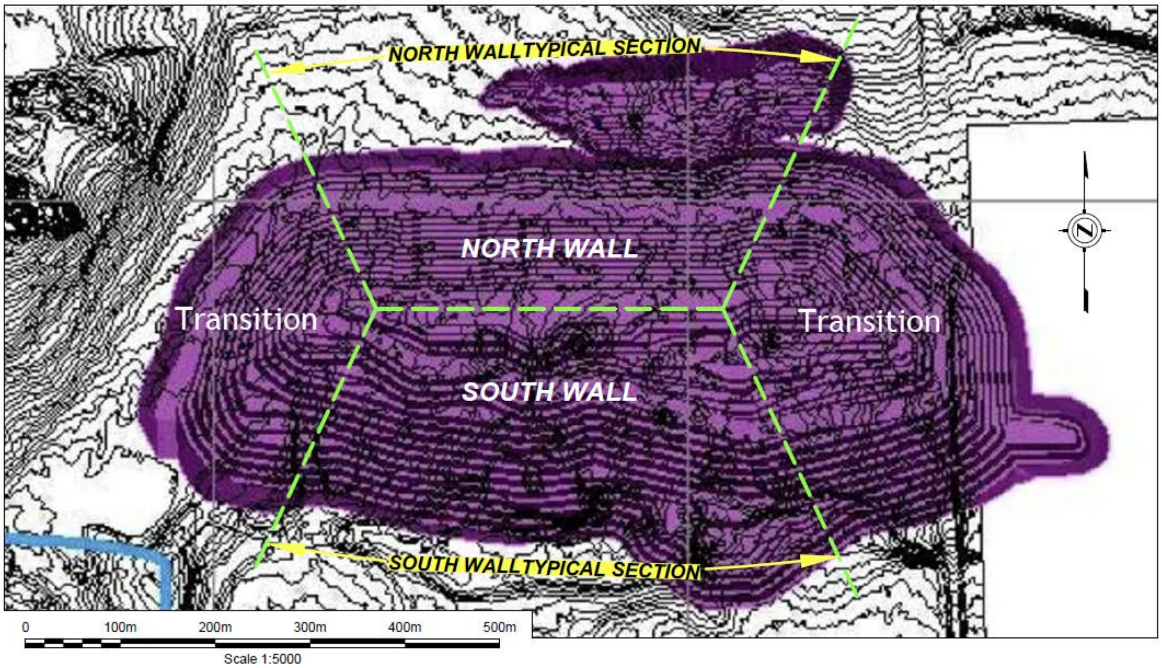


Figure 15-3: DFS pit slope design sectors

15.9.2 Pit Design Parameters

The detailed mine design was carried out using the selected pit shell as a guide. The proposed pit design includes the practical geometry required in a mine, including pit access and haulage ramps to all pit benches, pit slope designs, benching configurations, smoothed pit walls and catch benches. The major design parameters used are described in Table 15-6 and Table 15-7.

Table 15-6: Pit design parameters

Item	Value			Units
	North Wall	South Wall	Transition	
Overburden				
Berm Width	0			m
Bench Face Angle (BFA)	4H : 1V			-
Set back at the bedrock/OB contact	10			m
Rock				
Bench Height	6	6	6	m
Benching Arrangement	Triple	Triple	Triple	m
Berm Width	7.2	7.2	7.2	m

Item	Value			Units
	North Wall	South Wall	Transition	
Inter-Ramp Angle (IRA)	60	49	54.5	degrees
Bench Face Angle (BFA)	80	65	72.5	degrees
Overall Slope Angle (OSA)	59	48	53.5	degrees
Maximum Stack Height	N/A	N/A	N/A	m

Table 15-7: In pit haul roads

Item	Value	Units	Notes
Road Width (dual lane)	23	m	Based on Cat 775G
Road Width (single lane)	17	m	Bottom benches
Max. no. of benches at single lane	8	n/a	Based on 6 m bench height
Maximum Grade - Overburden	10	%	
Maximum Grade - Hard Rock	10 to 12	%	12% for the last benches

The design outlines a pit of ~1,000 m in length (east-west), an average of 600 m width (north-south) and down to a final pit depth of 200 m. The maximum planned total material movement including waste, stockpile reclaim, and ore to the run-of-mine ("ROM") pad is 14.7 Mtpa.

Figure 15-4 presents a plan view of the ultimate Authier Lithium Pit.

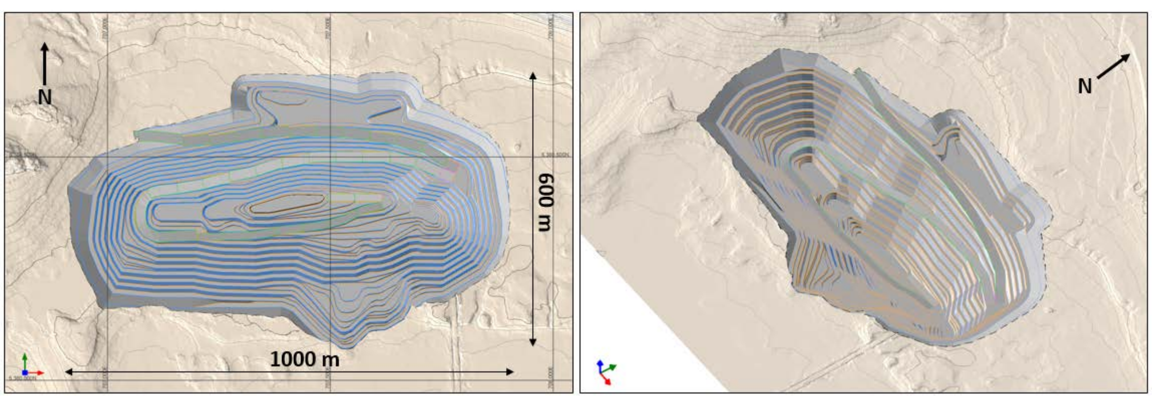


Figure 15-4: Ultimate Authier lithium pit plan view (ROM pad shown to the west of pit)

15.10 Ore Reserves

The ore reserves for the Authier Lithium project are based on the results of the 2018 DFS. Ore reserves are based on measured and indicated mineral resources contained within the final pit design following the application of modifying factors. Reserves are reported as ROM feed tonnes at the crusher above a cut-off grade of 0.55% Li_2O. Ore dilution and ore loss factors are incorporated into the ore reserves. Details related to the life-of-mine plan and additional mining related works are presented in Chapter 16 of this report. The ore reserves, presented in Table 15-8 were prepared by Isabelle Leblanc, P.Eng. of BBA Inc., as the competent person. The effective date of the ore reserves is October 2019 and the reference point is the primary crusher feed.

Table 15-8: Authier lithium project ore reserves

Ore Reserves	**Tonnage (Mt)**	**Grade (% Li_2O)**
Proved	6.1	0.99
Probable	6.0	1.02
Total Proved & Probable	**12.1**	**1.00**

16. MINING METHOD

16.1 Mining Phases

The goal of phasing is to prioritize the mining of more profitable resources (higher grade, lower strip ratio, etc.) as well as to defer waste stripping requirements so as to improve project economics. The pushbacks were selected for the pit optimization series based on contained ore tonnes and minimum pushback width. A total of 5 phases were designed within the final pit limits

The physicals by phase are shown in Table 16-1 below.

Table 16-1: DFS mining phase physicals

Physicals	Units	Phases					Total
		1	2	3	4	5	
Total In Pit	(Mt)	6.3	10.0	2.1	26.7	50.6	95.6
Waste Rock	(Mt)	3.8	7.0	1.4	22.9	43.1	78.1
Overburden	(Mt)	0.4	1.8	0.4	1.2	1.6	5.4
Total ROM Feed (≥ 0.55% Li_2O)	(Mt)	2.0	1.3	0.3	2.6	5.9	12.1
	(% Li_2O)	0.98	1.02	0.88	1.01	1.01	1.00
Strip Ratio	($t_{waste+OB}$: t_{ROM})	2.06	7.00	5.70	9.22	7.61	6.9

The pit phase designs are presented in the following figures.

Phase 1 was located on the south portion of the pit to access the shallow pegmatite rock of the main zone. Phase 1 presents a low strip ratio of 2.06 tonnes of waste and overburden per tonne of ore.

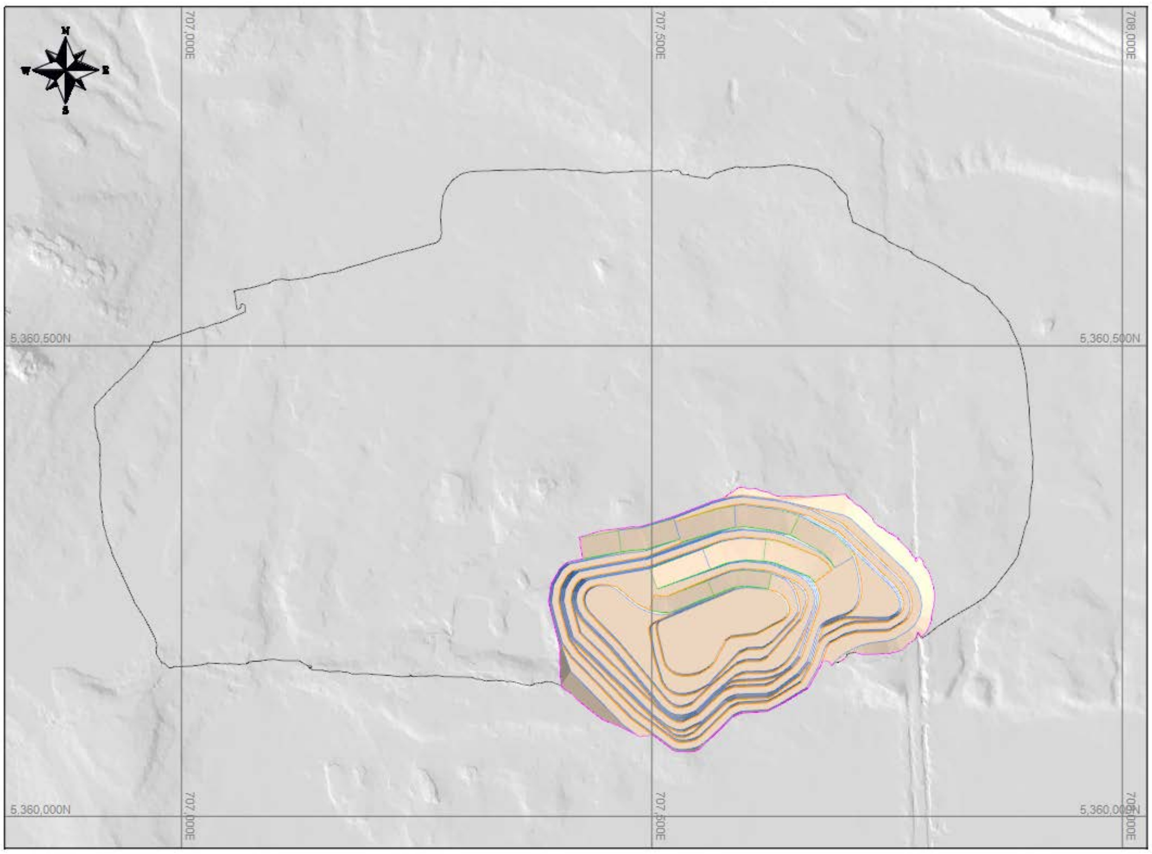


Figure 16-1: Pit Phase 1 design

The pit extends to the west for Phase 2 and another access ramp is developed. The strip ratio rises to 7.00 as the pit reaches an area of increased overburden thickness.

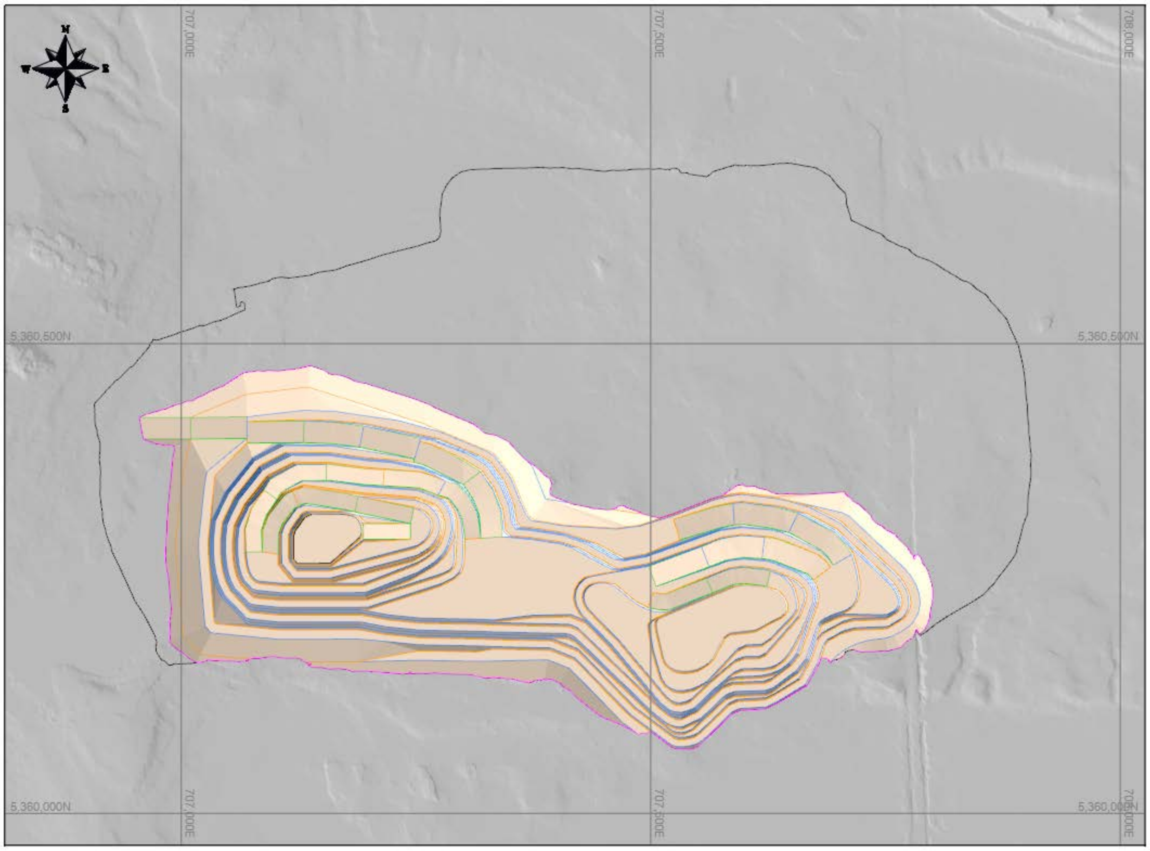


Figure 16-2: Pit Phase 2 design

A temporary access ramp will be developed for Phase 3, which consists of mining of the north zone. This phase will be mined at a lower rate to ensure that the lower grade material of the north zone is blended with the higher grade ore from the main zone.

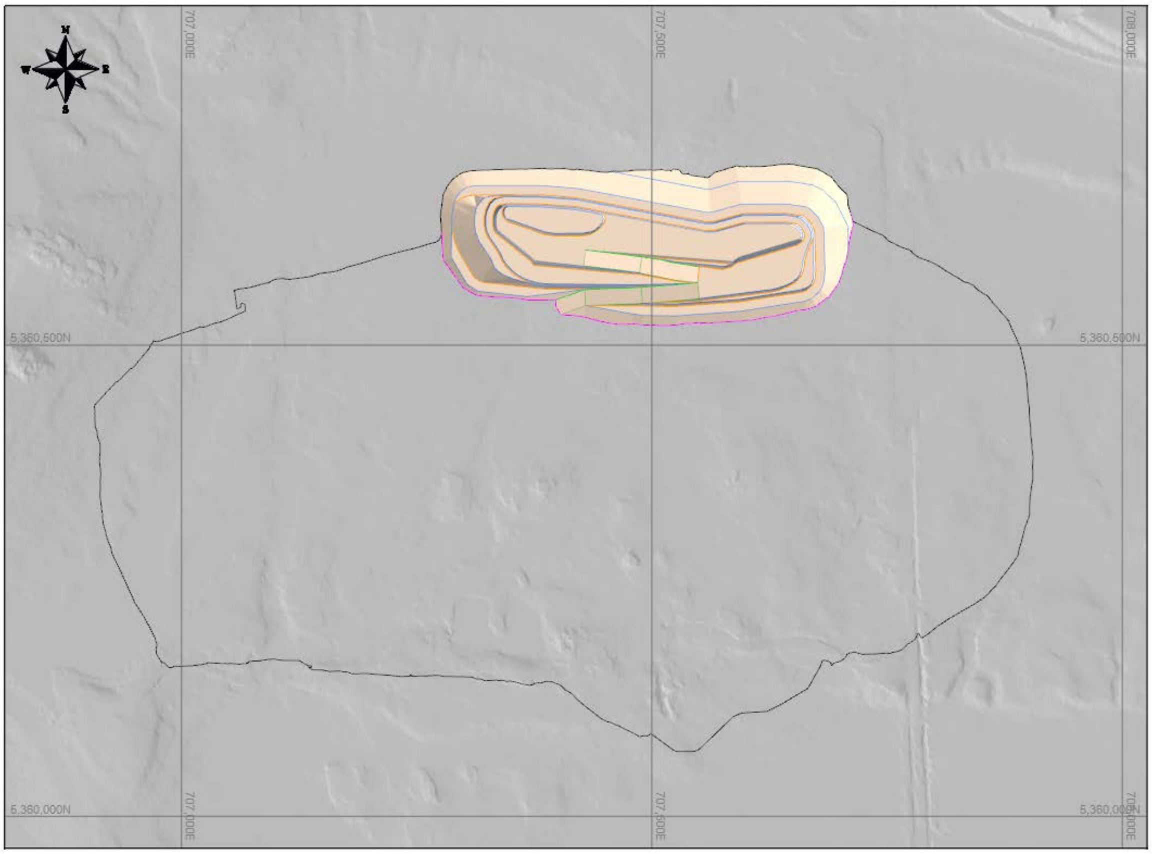


Figure 16-3: Pit Phase 3 design

Phase 4 is a pushback of Phase 1. A temporary access ramp is developed to mine the first benches until Phase 2 ramp access can be utilized to reach the bottom the pit. Phase 4 presents the highest strip ratio (9.22 tonnes of waste and overburden per tonne of ore) of all phases.

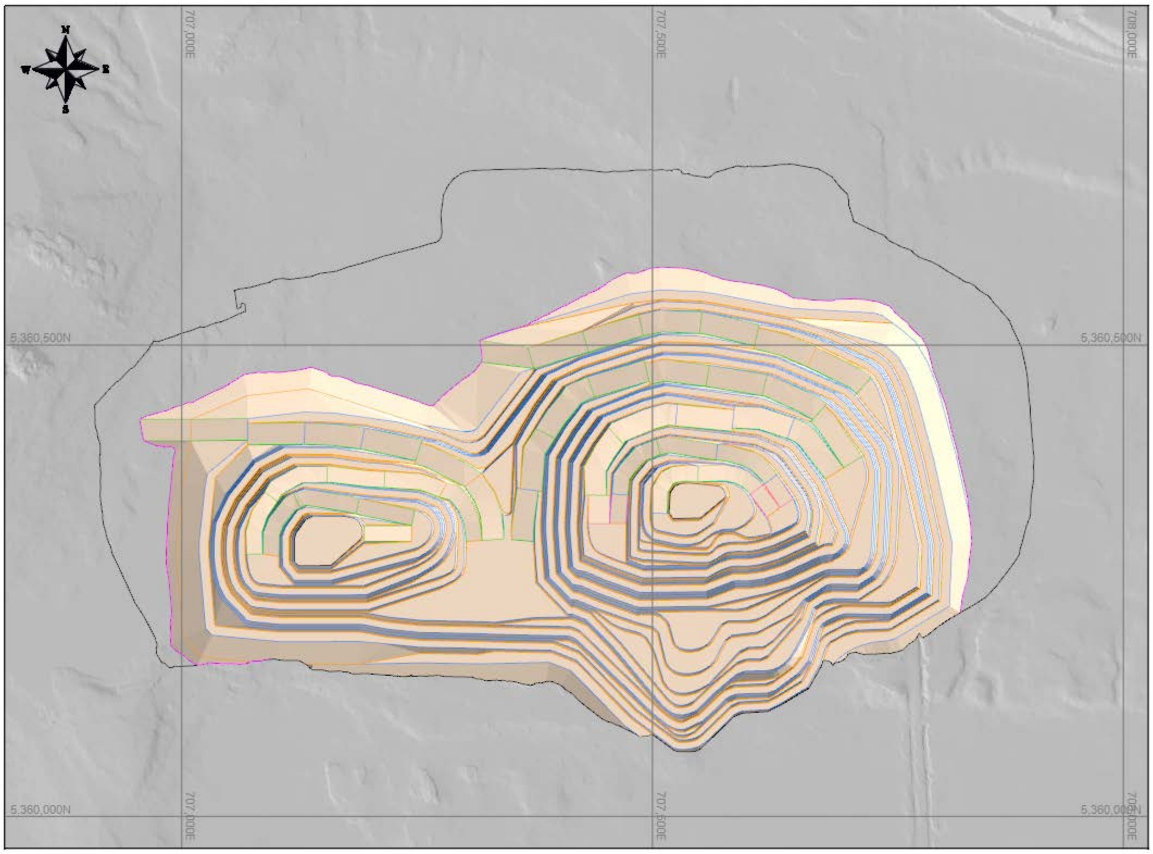


Figure 16-4: Pit Phase 4 design

Phase 5 completes the pushback sequence. The final ramp is located on the north wall to minimize the haulage distances and facilitate the mining of the last pushback. A single lane ramp is used for the bottom benches in order to minimize the life-of-mine strip ratio.

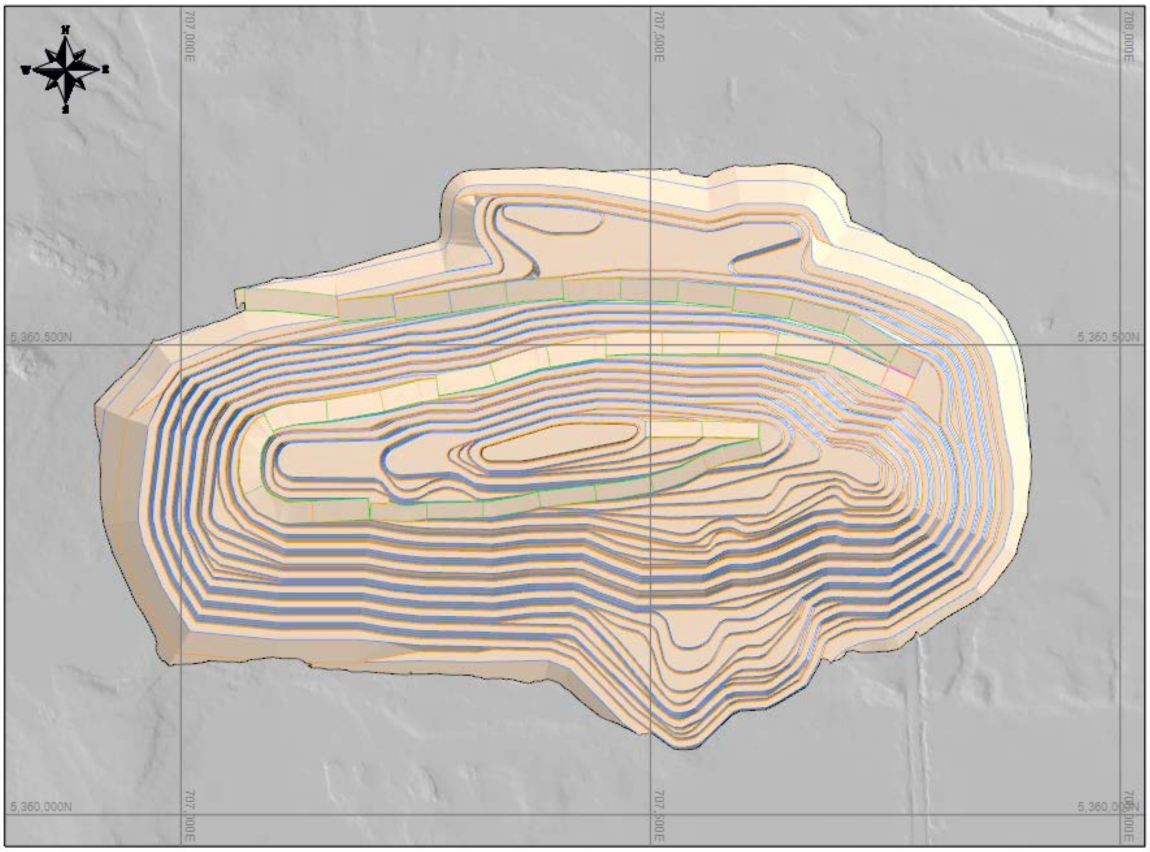


Figure 16-5: Pit Phase 5 and ultimate pit design

16.2 Mine Infrastructure

16.2.1 Dewatering

The hydrogeological study, completed in 2018 by Richelieu Hydrogéologie Inc., demonstrated that the mining activities will not affect the quality of the water.

Dewatering applies to the management of groundwater that, if not diverted from the pit or pumped from it, would impede mining operations or add to operating costs, notably for access to ore, blasting, and wear and tear on machinery. Dewatering requirements for the project were estimated by Technosub, a supplier of mine dewatering equipment. The pumping system has been designed to take four times the average water inflow (surface and underground combined) estimated in the hydrogeological report.

16.2.2 ROM Pad and Ore Stockpile

Mill feed will be delivered to the ROM pad located adjacent to the primary crusher. The ROM pad will be used to blend and harmonize the feed of ore into the primary crusher. As such, it will be used as a short-term or working stockpile only. For the purposes of mine planning, 75% of ore exiting the pit and destined for the crusher was considered to be re-handled on the ROM pad.

16.2.3 Haul Roads

Mining haul roads have been designed to accommodate 2-way traffic for 60 t class haul trucks. The Komatsu HD605-8 was the reference unit used to size the haul roads. Roads will incorporate drainage ditches as well as a safety berm when a drop of more than 3 m exists beyond the road edge. Single lane haul routes are proposed in some locations (ex. last benches of phases or the final pit). Table 16-2 lists the specified haul road dimensions used for the UDFS.

Table 16-2: Road design parameters

Parameters	Units	Dual Lane	Single Lane
Reference Haul Truck	-	HD605-8	HD605-8
Operating Width	m	5.7	5.7
Running Surface Multiplier	factor	3.0	2.0
Running Surface Width	m	17.0	11.5
Tire Diameter	m	2.7	2.7
Berm Height : Tire Ratio	ratio	0.5	0.5
Berm Height	m	1.3	1.3
Berm slope xH:1V Ratio	ratio	1.3H:1.0V	1.3H:1.0V
Berm Width (Top)	m	0.5	0.5
Berm Width (Bottom)	m	4.0	4.0
No. of Berms - Surface Road	number	2	2
No. of Berms - Pit Ramp	number	1	1
No. of Berms - Pit Slot	number	0	0
Ditch Depth	m	0.75	0.50
Ditch slope xH:1V Ratio	ratio	1.0H:1.0V	1.0H:1.0V
Ditch Width (Bottom)	m	0.5	0.5
Ditch Width (Top)	m	2.0	1.5
No. of Ditches - Surface Road	number	0	0

Parameters	Units	Dual Lane	Single Lane
No. of Ditches - Pit Ramp	number	1	1
No. of Ditches - Pit Slot	number	2	2
Overall Width - Surface Road	m	25.0	19.5
Overall Width - Pit Ramp	m	23.0	17.0
Overall Width - Pit Slot	m	21.0	14.5
Maximum Grade - Permanent Road	%	10.0	10.0
Maximum Grade - Temporary Road	%	12.0	12.0
Haul Road Drainage Crossfall	%	2.0	2.0

16.2.4 Explosives Storage

Two explosives magazines will be brought on-site by the explosives provider. One will house priming explosives, such as caps and detonating cord, and the other will hold a small amount of explosives and boosters.

The magazines are to be strategically located in a fenced and gated area on the southwest corner of the Authier Lithium property, so as to meet provincial and federal explosives regulations. A gravel road from the MIA will be built to access this area. As the proposed main supplier of explosives is located in close proximity to the mine, magazine capacities will be kept at a minimum.

Explosives will be delivered to the mine site by the selected supplier and stored until use, in accordance with provincial and federal regulations. Bulk emulsion will be handled and loaded into the holes by the blasting contractor (i.e., no bulk emulsion storage facilities are required on-site). Site layout is presented in Chapter 18 of this report.

16.3 LOM Planning

A life-of-mine ("LOM") plan with a 2,600 tonnes per day ore processing capacity was completed for the Authier UDFS using MineSight Schedule Optimizer ("MSSO"). Details are presented below.

16.3.1 Strategy & Constraints

The following constraints and objectives were considered during the development of the LOM plan:

- Mill ramp up of 3 months or 1 quarter (first year mill feed 772 kt);
- Maximum annual mill feed of 883 kt/y;

- No long-term stockpile;
- Maximum ROM mining rate of 883 kt/y;
- Maximum mining rate 14 Mt/y;
- Bench sinking rate limit of 12 benches/y or 72 m/y vertical advance (per phase);
- Target mill feed grade ≥0.8% Li_2O;
- Pre-production (6 months) maximum tonnage of 794 kt;
- Mine planning strategy: maximize NPV.

16.3.2 Results

The ROM feed contained in the final pit is sufficient for a mine life of 13.8 years.

Due to the phase designs, very little waste material is mined to supply the mill in the first two years. This strategy keeps the mining activities to a minimum allowing the operation to improve its mining practices, equipment needs, and consequently keeps mine operating costs low.

The overall pit has a variable strip ratio. The annual productivity gradually increases to 12 Mt in Year 5. As productivity stabilizes with a maximum of 14 Mt in Year 6 and Year 7 with the deepening of the pit, more ore becomes accessible and the annual productivity gradually decreases from Year 8 to the end of the mine life.

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Table 16-3: LOM plan

Physicals	Units	Pre-Prod	Production														LOM Total
		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	
Total Moved (Expit + Rehandle)	**(kt)**	**794**	**3,599**	**3,958**	**6,625**	**9,617**	**12,582**	**14,657**	**14,662**	**12,440**	**9,623**	**6,636**	**3,183**	**2,396**	**2,116**	**1,718**	**104,608**
Total Expit	**(kt)**	**794**	**3,020**	**3,296**	**5,964**	**8,956**	**11,920**	**13,995**	**14,000**	**11,778**	**8,961**	**5,974**	**2,522**	**1,734**	**1,454**	**1,168**	**95,536**
Expit Waste Rock	(kt)	538	2,068	1,391	4,142	7,322	10,429	11,672	13,098	10,895	8,078	5,092	1,639	852	572	435	78,223
Expit Overburden	(kt)	257	180	1,022	939	751	609	1,440	20	0	0	0	0	0	0	0	5,217
Expit Ore to Mill	(kt)	0	193	221	221	221	221	221	221	221	221	221	221	221	221	183	3,024
Expit Ore to ROMpad	(kt)	0	579	662	662	662	662	662	662	662	662	662	662	662	662	550	9,072
Expit Ore	(kt)	0	772	883	883	883	883	883	883	883	883	883	883	883	883	733	12,096
Expit Ore Grade	(% Li_2O)	0.00	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	1.00
Stripping Ratio	**(t_{waste}:t_{RoM})**	**0.0**	**2.9**	**2.7**	**5.8**	**9.1**	**12.5**	**14.9**	**14.9**	**12.3**	**9.2**	**5.8**	**1.9**	**1.0**	**0.6**	**0.6**	**6.9**
Total Stockpile Rehandling	**(kt)**	**0**	**579**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**662**	**550**	**9,072**
Stockpile to Mill	(kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stockpile to Mill Grade	(% Li_2O)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
ROMpad to Mill	(kt)	0	579	662	662	662	662	662	662	662	662	662	662	662	662	550	9,072
ROMpad to Mill Grade	(% Li_2O)	0.00	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	1.00
Crusher Feed	**(kt)**	**0**	**772**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**883**	**733**	**12,096**
Crusher Feed Grade	(% Li_2O)	0.00	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	1.00

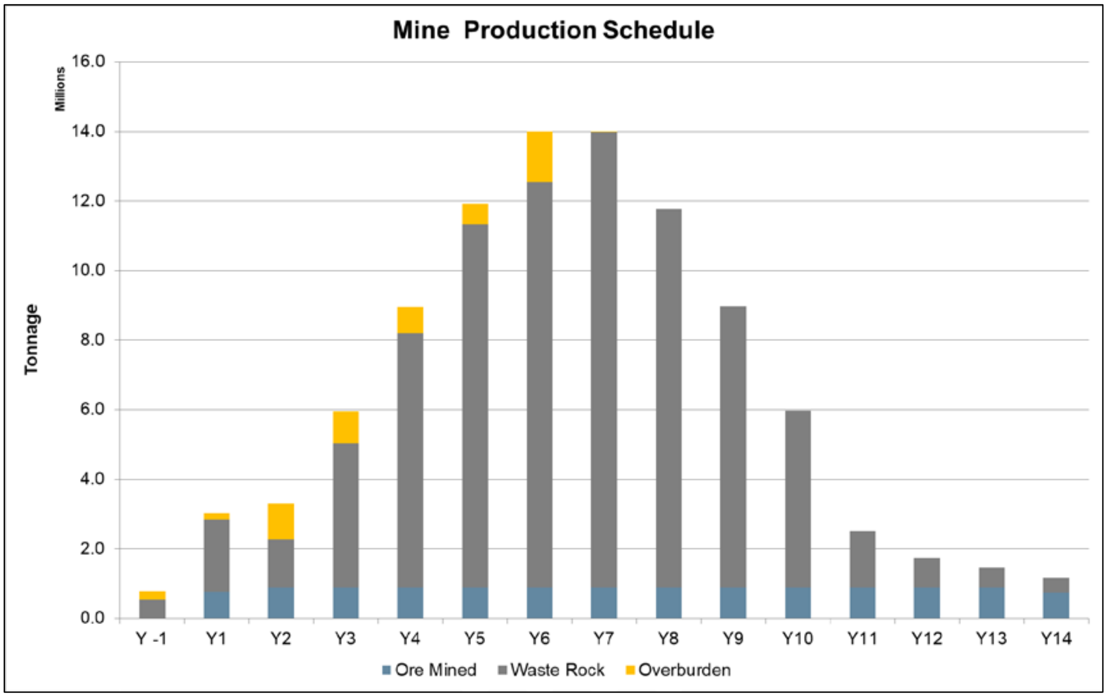


Figure 16-6: Authier mine plan

16.4 Mine Operations

16.4.1 Roster

The mine will operate 365 days per year with two 12-hour shifts per day until Year 10, and then will only operate at 12 hours per day. The mine equipment operators and mechanics will work on a seven-working-day / seven-rest-day schedule. All other employees will work a regular 40-hour work week. The manpower requirements for the entire mine life, including the mine operations related staff are presented in Chapter 23.

16.4.2 Drilling

Drilling and blasting operations represent a crucial process when developing and sustaining a hard-rock mining operation. The performance and efficiency of this primary rock fragmentation process can heavily impact the ore dilution / loss parameters and other downstream activities such as loading, hauling and material handling, as well as crushing and grinding.

Blast fragmentation curves were developed based on rock characterization, types of explosives, blast patterns and powder factor. A crusher feed size of 530 mm (P_{80}) with a particle top size of 600 mm was targeted.

Drilling activities will be executed by the mine's drilling equipment fleet, consisting of a maximum of three down-the-hole (DTH) drill rigs. All hard rock material will be drilled with 5" diameter holes. Production blasts will occur on 6 m bench heights for ore and 9m for waste with triple benching arrangement. The drill hole patterns in ore and waste material are presented in Table 16-4.

Table 16-4: Drilling ore and waste patterns

Drill Pattern		Ore	Waste
Bench Height	m	6	9
Hole Diameter	in.	5	5
Hole Diameter	mm	127	127
Burden	m	4.00	4.50
Spacing	m	4.00	4.50
Sub-Drill	m	0.80	0.80

Pre-split drill holes will be drilled every 1.50 m along the final pit walls to improve the pit wall quality.

16.4.3 Blasting

Blasting will be executed under contract with an explosive company that will supply the blasting materials and technology, as well as the equipment to store and deliver the explosive products. Production drill holes will be loaded with a bulk emulsion explosive, whereas pre-split drill holes will be loaded with a continuous packaged emulsion to produce uniform fracturing or "splitting" at the final wall location.

In areas with ore material, the blasts will be detonated with an electronic blasting system. Electronic detonators offer greater flexibility and precision for the blast sequence, which can in turn improve rock fragmentation and diggability, and better control the blast movement. In areas that are completely waste material, the blasts will be detonated with pyrotechnic detonators as they are less costly.

Based on the drilling patterns listed above and blast fragmentation curves for host rock and pegmatite by using emulsion blasting agent with an average density (in the hole) of 1.15 g/cm^3, the powder factor will vary from 0.21 to 0.26 kg of explosives per tonne of rock. All work associated to blasting activities will be carried out by the explosive supplier. The proposed arrangement would need 2-3 blasts per week.

16.4.4 Loading

Loading activities will be carried out by the mine's loading equipment fleet. A maximum of four 9.4 t-capacity hydraulic backhoe excavators and two 10.7 t-capacity production wheel loader will be required. This equipment is compatible with the hauling fleet.

The excavators will be used to load all ore from the pit, the overburden material, and some waste rock. The excavators can selectively mine the ore material to better control dilution and ore losses.

The wheel loader will be used to load waste material and reclaim material from the various ore stockpiles. In order to ensure a fairly constant ore feed grade to the mill, it was assumed that a certain percentage of ore exiting the pit will be stockpiled and reclaimed on a short-term basis. The wheel loader can also be dispatched much more quickly to different areas of the operation.

16.4.5 Hauling

Hauling activities will be carried out by the mine's hauling equipment fleet. A maximum of 12 60 t-capacity rigid haul trucks and three 40 t-capacity articulated haul trucks will be required throughout the mine life. The rigid haul truck will be used to haul all hard rock material and the articulated trucks will be used to haul overburden and tailings.

The ore will be mainly hauled to short-term stockpiles located near the crusher or directly to the crusher. The waste rock and tailings material generated by the mill will be transported and stockpiled on the waste stockpile according to a co-deposition plan. The overburden will be hauled to the overburden stockpile.

The hauling equipment fleet requirements were estimated based on the quantities of material to be transported in a given period and the representative haul cycle times. The haul cycle times and corresponding fuel consumptions were estimated with the MS Haulage simulation software.

16.4.6 Auxiliary

The auxiliary equipment fleet will consist of a variety of support equipment.

A 354 hp bulldozer will be required on the waste stockpile for the hard rock material, and one 210 hp bulldozer will be required for the tailings on the waste stockpile and the overburden stockpile. A maximum of two 14 ft moldboard motor graders will be required for preparing and grading the haul roads. A 50 t auxiliary excavator will also be required for pit wall scaling and other secondary work around the pit (pit dewatering activities, ditches, breaking oversize rocks, etc.). The operation will also need a water / sand spreader for watering the roads in the summer months for dust suppression and spreading sand for better traction in the winter. Finally, tower lights, an equipment transporter, a fuel and lube truck, and pick-up trucks will be needed.

All mine equipment requirements over the mine life are presented in Table 16-5.

Table 16-5: Mine equipment requirements over the LOM

Item	H -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Haul Truck - 60t	1	2	2	5	8	11	12	12	12	11	8	8	5	5	5
Haul Truck - 40t (ADT)	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3
Excavator - 90t	1	1	1	2	3	3	4	4	3	2	2	2	1	1	1
Loader - 10t	-	1	1	1	1	2	2	2	2	2	2	2	2	1	1
DTH Drill	1	1	1	2	2	2	3	3	3	2	2	2	1	1	1
Large Dozer	-	-	1	1	1	1	1	1	1	1	1	1	1	-	-
Medium Dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Auxiliary Excavator	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor Grader	1	1	1	1	2	2	2	2	2	2	1	1	1	1	1
Water Truck / Sand Spreader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Equipment Transporter	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel & Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant	3	4	5	7	8	8	10	10	9	7	7	7	5	4	4
Service Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pick-Up Trucks	6	7	7	7	7	7	7	7	7	7	7	5	5	5	5

16.4.7 Equipment Availability and Efficiency

Table 16-6 presents the operational parameters used to estimate the effective utilization of the primary equipment selected.

Table 16-6: Operational parameters

Description	Units	Haul Truck	Articulated Truck	Excavator	Production Loader	Drill
		HD605-8	HM400-5	PC800LC8	WA600-8	Sandvik DI550
Down Time						
Average Mechanical Availability	%	85.0	85.0	85.0	85.0	80.0
Operating Delays						
Shift Change	min/shift	15	15	15	15	15
Pre-Start Delays	min/shift	15	15	15	15	15
Meal Breaks	min/shift	60	60	60	60	60
Fueling, Lube & Service	min/shift	15	15	15	15	15
Sub-Total Operating Delays	hr/shift	1.8	1.8	1.8	1.8	1.8
Utilization (after Op. Delays)	%	90	90	90	90	90
Total Operating Delays	**hr/shift**	**2.8**	**2.8**	**2.8**	**2.8**	**2.8**
Use of Availability	**%**	**76.9**	**76.9**	**76.9**	**76.9**	**76.9**
Operating Efficiency						
Tramming / equipment relocation(1)	%	2	2	10	15	10
Queuing / Hang Time(2)	%	15	15	20	0	0
Other Operating Delays	min/hr	3	3	5	10	6
Operator Efficiency before (1) and (2)	%	95.0	95.0	91.7	83.3	90.0
Total Operating Efficiency	**%**	**78.0**	**78.0**	**61.7**	**68.3**	**80.0**
Hours Breakdown						
Calendar Time (CT)	hr/yr	8,760	8,760	8,760	8,760	8,760
Scheduled Time	hr/yr	8,760	8,760	8,760	8,760	8,760
Available Time (AT)	hr/yr	7,446	7,446	7,446	7,446	7,008
Utilized Time (UT)	hr/yr	5,632	5,632	5,632	5,632	5,295
Operating Time (OT)	**hr/yr**	**4,393**	**4,393**	**3,473**	**3,848**	**4,236**
Scheduled as % of Calendar	%	100.0	100.0	100.0	100.0	100.0
Availability	**%**	**85.0**	**85.0**	**85.0**	**85.0**	**80.0**
Use of Availability - UA	**%**	**75.6**	**75.6**	**75.6**	**75.6**	**75.6**
Operating Efficiency - OE	**%**	**78.0**	**78.0**	**61.7**	**68.3**	**80.0**
Effective Utilization - EU	%	50.1	50.1	39.6	43.9	48.4

16.4.8 Mine Dewatering

The mine dewatering system will consist of high head, auto-priming, and diesel driven pumps with corresponding piping. The models of pumps and the piping will vary over time.

16.4.9 Mine Maintenance

As the operation is located in the Abitibi region of Québec, which is serviced by many equipment suppliers, the maintenance expected to be carried out at the site are only regular preventive maintenances and tire changes. All major equipment repairs will be contracted out to the equipment supplier's maintenance team or, if need be, sent out to the supplier's workshop.

16.4.10 Mine Technical Services

The mine technical services team will consist of the engineering team headed by a chief engineer and supported by mining engineer, mining technicians, and a geologist supported by geology technicians.

17. RECOVERY METHODS

17.1 Introduction

The Authier Lithium concentrator is designed to process 2,600 tpd. The ROM ore will contain an average of 1.00% Li_2O throughout the life of the mine. Iron content in the first 5 years of operation is expected to be slightly higher than in the life of mine average, due to the presence of elevated concentrations of iron-bearing silicate minerals (e.g., hornblende and chlorite. Overall lithium recovery is expected to be 78% with an average spodumene concentrate grade of 6.0% Li_2O. The annual spodumene concentrate production will be roughly 112,700 tonnes (LOM average).

17.2 Process Flowsheet Selection

The proposed processing facility for the Authier Lithium project is based upon a configuration of unit operations including:

- Crushing;
- Grinding;
- Magnetic separation;
- Mica flotation;
- Spodumene flotation;
- Tailings thickening and filtration (for dry stacking disposal); and
- Concentrate filtration, storage and transportation.

A simplified process flowsheet is shown in Figure 17-1.

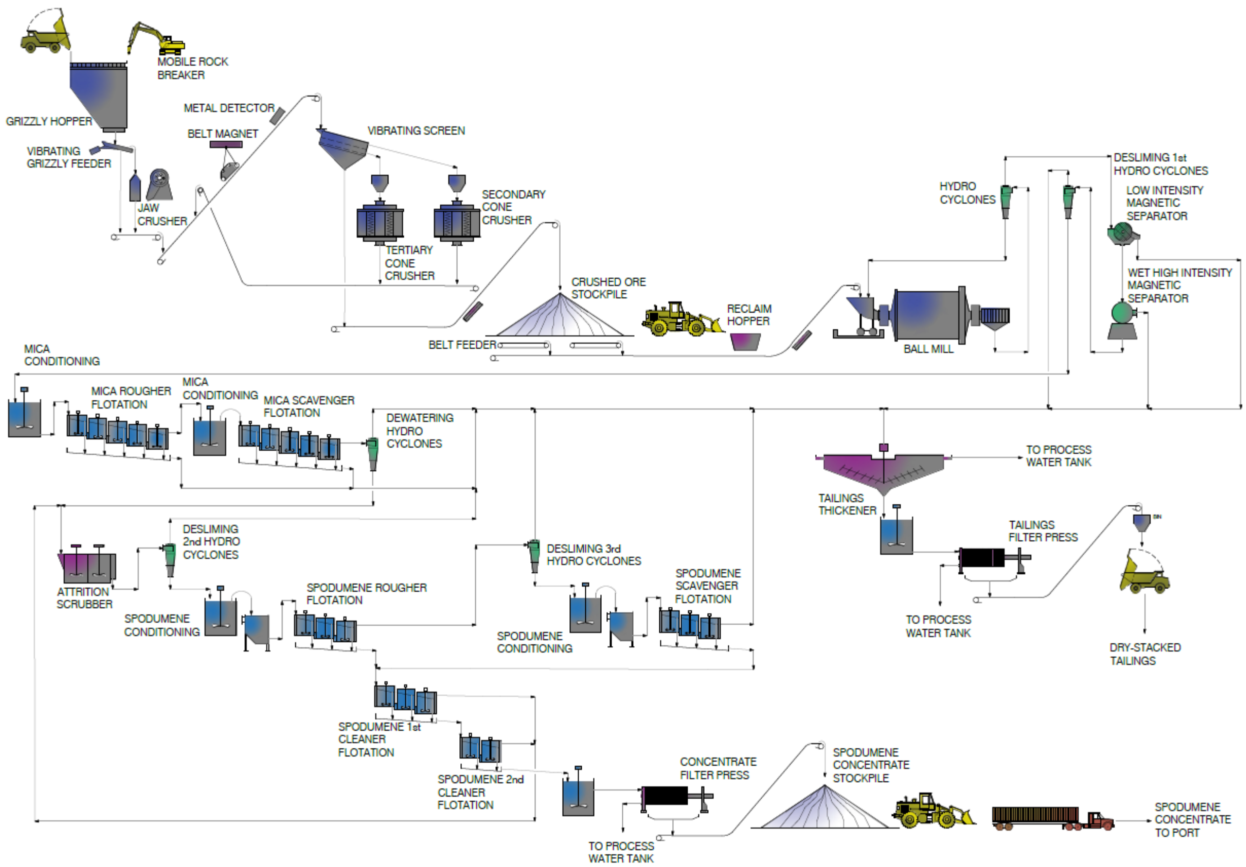


Figure 17-1: Simplified Process Flowsheet

A detailed description of the flowsheet is provided in the following sections.

17.3 Process Design Criteria and Material Balance

Mass and water balances for the flowsheet are based on an annual throughput of 882,570 t of ROM ore per year (average daily throughput of 2,400 tpd) as presented in Table 17-3. The process design criteria and mass and water balances were developed based on mineralogical analyses, results of historical metallurgical test work programs and equipment vendor recommendations.

The following sections present the process design criteria and material balances.

17.3.1 General

Table 17-1 presents the site conditions.

Table 17-1: Site Conditions

Parameter	Units	Value	Comments
Location		45 km northwest of Val D'Or, Québec	
Nominal latitude	UTM m E	706,725	Nominal
Nominal longitude	UTM m N	5,361,360	Nominal
Elevation	mASL	1,100	Nominal
Temperature - max	°C	25	Public information
Temperature - min	°C	-21	Public information
Daily temperature range - max	°C	20	
Daily temperature range - nom	°C	15	
Expected indoor temperature	°C	5-30	
Annual rainfall	mm/yr	930	Public information
No. of rain days	day/yr	115	>0.2 mm
Annual snowfall	Cm	288	
No. of snow days	day/yr	101	>0.2 mm

17.3.2 Ore Characteristics

Table 17-2 presents the basic ore characteristics for the Authier Lithium project.

Table 17-2: Process Design Criteria – Ore Characteristics (Mass Balance Assumptions)

Parameter	Units	Value
Ore type		Pegmatite
Mining method		Open pit, drill and blast
ROM dilution	%	< 2
Feed grade:		
Min. (cut-off)	% Li_2O	0.55
Ore SG		2.71
Moisture	%w/w	3
Nominal ROM mineralogy		
▪ Spodumene	%	13.1
▪ Quartz	%	33.1
▪ Plagioclase (Albite)	%	31.2

Parameter	Units	Value
▪ K-Feldspar (Microcline)	%	9.6
▪ Muscovite	%	9.0
ROM head grade		
▪ Li_2O	%	1.00
▪ Li	%	0.46
Bulk density (crushed ore)	t/m^3	1.6-1.8
Angle of repose	Deg	37
UCS	Mpa	85
Crushing work index (CWi)	kWh/t	15
Bond abrasion index (Ai)	G	0.98
Bond ball mill work index (BWi)	kWh/t	15.1

17.3.3 Operation Design Parameters

Table 17-3 presents the operating design parameters for the Authier Lithium project.

Table 17-3: Operation Design Parameters

Parameters	Units	Value
Crushing		
Annual throughput (design)	tpy	882,570
Operating days per year	d	365
Number of operating shifts per day	No.	2
Shift duration	h	10
Plant utilization	%	50
Operating hours per year	h	4,406
Daily production - average	tpd	2,418
Design feed rate	tph	200
Processing Plant		
Annual throughput (design)	tpy	882,570
Operating days per year	d	365
Number of operating shifts per day	No.	2
Shift duration	h	12
Plant utilization	%	93
Operating hours per year	h	8,147
Daily production - average	tpd	2,418
Design feed rate	tph	108

17.3.4 Material Balance

METSIM was used to simulate the process flowsheet and to determine the material balance for a 2,600 tpd processing plant. The model is based on the mineralogical composition of the feed and mineral recovery factors at various stages of the process. The recovery factors were developed from bench scale tests and confirmed during pilot plant operation. The overall material balance is presented in the Table 17-4.

Table 17-4: Mass and Water Balance

Grinding Area (6015009-000000-49-D10-0002)

Stream Number	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	100
Stream Name	**Crushed Ore**	**Mill Feed**	**Cyclone UF**	**PW Mill Feed**	**Ball Mill Discharge**	**PW Mill Discharge**	**Cyclone Feed**	**Cyclone OF**	**LIMS Mags**	**Mag Sep Feed**	**WHIMS Mags**	**No mags**	**Comb Mag+Slimes**	**PW Desl Cyc Feed**	**Desl Cyc Feed**	**Desl Cyc UF**	**Desl Cyc OF**	**PW to Grinding**
Solids (tph)	200	108	269		377		377	108	0.00	108	2.68	106	9.72		106	98.6	7.04	
Water (m³/h)	6.19	3.35	97.0	77.3	178	171	348	251	0.00	251	1.94	249	326	173	423	98.6	324	421
Pulp (m³/h)	80.2	43.4	197		317		489	292	0.00	292	2.83	289	330		463	135	327	
% solids (wt%)	97.0	97.0	73.5		68.0		52.0	30.1	0.00	30.1	58.0	29.8	2.90		20.0	50.0	2.13	
Solids density (t/m³)	2.70	2.70	2.70		2.70		2.70	2.70	0.00	2.70	3.02	2.70	2.77		2.70	2.70	2.69	
Slurry density (t/m³)	2.57	2.57	1.86	1.00	1.75	1.00	1.49	1.23	0.00	1.23	1.63	1.23	1.02	1.00	1.14	1.46	1.01	1.00
Li_2O assay (%)	1.00%	1.00%	1.00%	0.00%	1.00%	0.00%	1.00%	1.00%	0.00	1.00%	2.02%	0.97%	1.09%	0.00%	0.97%	0.99%	0.73%	0.00%
Li_2O distribution (%)	2.00	1.08	2.69		3.77		3.77	1.08	0.00	1.08	0.054	1.03	0.11		1.03	0.98	0.051	

Mica Flotation Area (6015009-000000-49-D10-0003)

Stream Number	22	24	25	26	27	28	29	30	31	33	34	35	36	101
Stream Name	**Desl Cyc UF**	**PW Mica Cond**	**Mica Flot Feed**	**Mica Ro Con**	**PW Ro Con**	**Mica Ro Tail**	**Mica Scav Feed**	**Mica Scav Con**	**Mica Scav Tail**	**Dewater Cyc Feed**	**Dewater Cyc OF**	**Dewater Cyc UF**	**Comb Mica + Sl**	**PW to Mica Flotation**
Solids (tph)	98.6		98.6	9.74		88.9	88.9	6.55	82.3	82.3	0.00	82.3	16.3	
Water (m³/h)	98.6	296	394	29.2	0.00	365	365	15.3	350	350	295	54.9	340	296
Pulp (m³/h)	135		432	32.7		399	399	17.8	381	381	296	85.6	346	
% solids (wt%)	50.0		20.0	25.0		19.6	19.6	30.0	19.0	19.0	0.00	60.0	4.58	
Solids density (t/m³)	2.70		2.70	2.83		2.68	2.68	2.63	2.69	2.69	0.00	2.69	2.75	
Slurry density (t/m³)	1.46	1.00	1.14	1.19	0.00	1.14	1.14	1.23	1.13	1.13	1.00	1.60	1.03	1.00
Li_2O assay (%)	0.99%	0.00%	0.99%	0.17%	0.00%	1.08%	1.08%	0.082%	1.16%	1.16%	0.00%	1.16%	0.13%	0.00%
Li_2O distribution (%)	0.98		0.98	0.016		0.96	0.96	0.005	0.96	0.96	0.00	0.96	0.022	

Table 17-4: Mass and Water Balance (cont'd)

Spodumene Flotation Area (6015009-000000-49-D10-0004)

Stream Number	35	37	38	39	40	41	42	43	44	45	47	49	50	51	53	54	55	56	102
Stream Name	Dewater Cyc UF	Attrition Discharge	PW Attrition Discharge	Desl Cyc Feed	Desl Cyc OF	Desl Cyc UF	PW Ro Feed Tank	Spod Ro Feed	Spod Ro/Scav Con	Spod Ro/Scav Tail	Spod Ro/Scav Tail + Sl	Spod Cl Feed	Spod 1st Cl Tail	Spod 1st Cl Con	Spod 2nd Cl Feed	Spodumene Concentrate	Spod 2nd Cl Tail	Spod Cl Recl Tail	PW to Spodumene Flotation
Solids (tph)	82.3	86.7		86.7	5.15	81.5		81.5	18.4	63.1	68.2	18.4	2.52	15.9	15.9	14.1	1.85	4.37	
Water (m³/h)	54.9	72.4	130	202	148	54.4	136	190	73.8	116	264	73.8	10.1	63.7	63.7	56.3	7.40	17.5	266
Pulp (m³/h)	85.6	105		235	150	84.8		221	80.2	141	291	80.2	11.0	69.2	69.2	61.1	8.09	19.1	
% solids (wt%)	60.0	54.5		30.0	3.37	60.0		30.0	20.0	35.1	20.5	20.0	20.0	20.0	20.0	20.0	20.0	20.0	
Solids density (t/m³)	2.69	2.69		2.69	2.67	2.69		2.69	2.92	2.63	2.63	2.92	2.71	2.96	2.96	2.99	2.73	2.72	
Slurry density (t/m³)	1.60	1.52	1.00	1.23	1.02	1.60	1.00	1.23	1.15	1.28	1.14	1.15	1.14	1.15	1.15	1.15	1.14	1.14	1.00
Li_2O assay (%)	1.16%	1.18%	0.00%	1.18%	0.73%	1.21%	0.00%	1.21%	4.94%	0.12%	0.16%	4.94%	1.34%	5.51%	5.51%	6.00%	1.75%	1.51%	0.00%
Li_2O distribution (%)	0.96	1.02		1.02	0.038	0.98		0.98	0.91	0.074	0.11	0.91	0.034	0.88	0.88	0.84	0.032	0.066	

Spodumene Concrete Dewatering Area (6015009-000000-49-D10-0005)

Stream Number	54	61	62
Stream Name	Spodumene Concentrate	Final Concentrate	Filter Press Filtrate
Solids (tph)	14.1	14.1	0.00
Water (m³/h)	56.3	0.98	55.3
Pulp (m³/h)	61.1	5.69	
% solids (wt%)	20.0	93.5	
Solids density (t/m³)	2.99	2.99	0.00
Slurry density (t/m³)	1.15	2.65	1.00
Li_2O assay (%)	6.00%	6.00%	0.00%
Li_2O distribution (%)	0.84	0.84	

Table 17-4: Mass and Water Balance (cont'd)

Tailings Dewatering Area (6015009-000000-49-D10-0006)

Stream Number	19	36	47	64	65	66	67	68
Stream Name	Comb Mag+Slimes	Comb Mica + Sl	Spo Ro Tail + Sl	Tails Filtrate	Tails Thkr Feed	Tails Thkr UF	Tails Thkr OF	Tails Filter Cake
Solids (tph)	9.72	16.3	68.2	0.00	94.3	94.3	0.00	94.3
Water (m³/h)	326	340	264	58.3	930	71.1	859	12.9
Pulp (m³/h)	330	346	291		967	107		48.2
% solids (wt%)	2.90	4.58	20.5		9.20	57.0		88.0
Solids density (t/m³)	2.77	2.75	2.63	0.00	2.67	2.67	0.00	2.67
Slurry density (t/m³)	1.02	1.03	1.14	1.00	1.06	1.55	1.00	2.22
Li_2O assay (%)	1.09%	0.13%	0.16%	0.00%	0.25%	0.25%	0.00%	0.25%
Li_2O distribution (%)	0.11	0.022	0.11		0.24	0.24		0.24

(Water Balance (6015009-000000-49-D10-0009)

Stream Number	62	64	67	100	101	102	103	104	105	106	107
Stream Name	Press Filter Filtrate	Tails Filtrate	Tails Thickener OF	PW to Grinding	PW to Mica Flotation	PW to Spodumene Flotation	Make-Up Water	Gland Seal Water	Fresh Water Required	Total Filtrate to Process Water	FW for Reagents
Water (m³/h)	55.3	58.3	859	421	296	266	6.42	21.0	10.5	114	4.06

Figure 17-2 presents a schematic of the plant water balance showing the distribution and types of water used in the process (i.e., fresh water, recycled process water, and filtered process water). The source of fresh water is not shown in the schematic, but is a water collection basin (BC2), described in Section 19. The main water losses are water contained in the final spodumene concentrate and tailings.

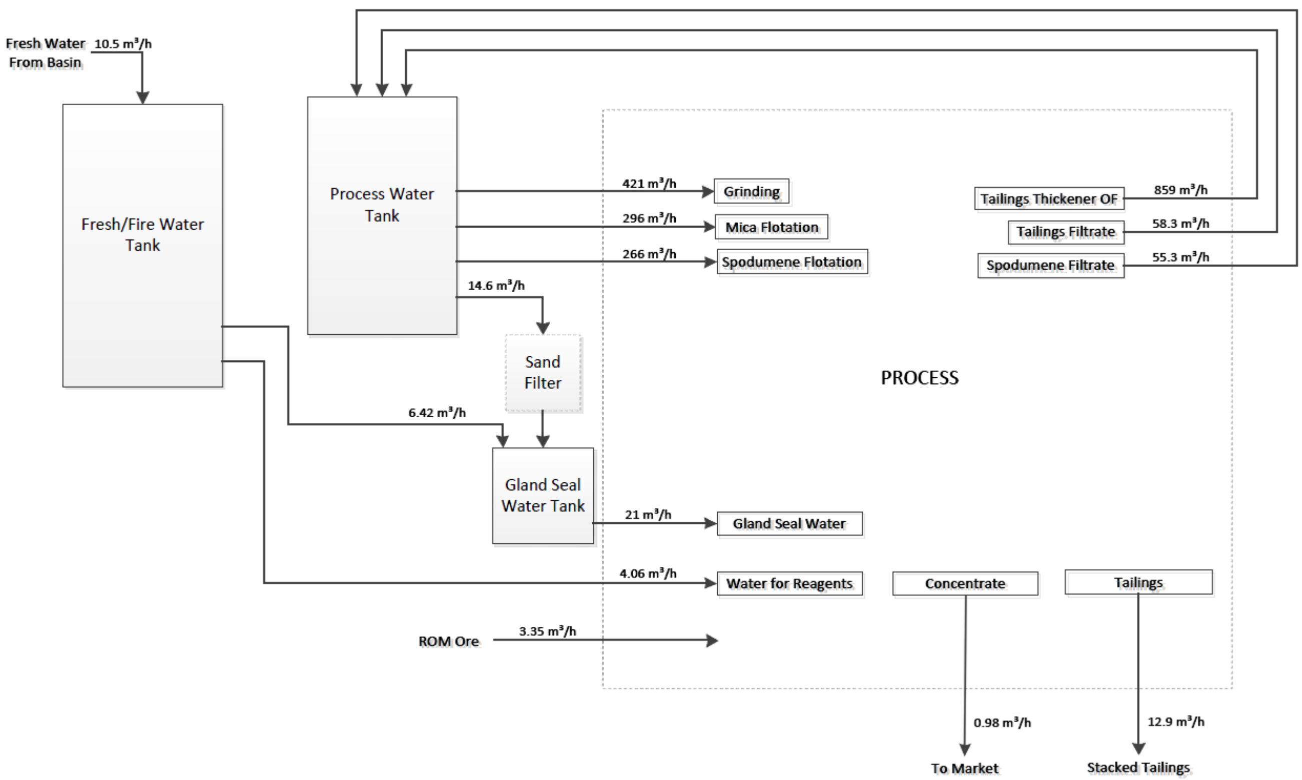


Figure 17-2: Water Balance Schematic

17.4 Process Description

17.4.1 Major Equipment List

Table 17-5 provides a list of the major process equipment.

Table 17-5: Major Equipment List

Item Description	Number of units	Make/Model/Type
3100 Crushing		
ROM hopper	1	130 t capacity, 2 trucks
Stationary grizzly	1	307
Primary crusher	1	Terex MPS MJ47 jaw crusher
Primary screen	1	Terex MPS MHS8203 horizontal screen
Secondary crusher	1	Terex MPS MC450X S/F cone crusher
Tertiary crusher	1	Terex MPS MC450X S/F cone crusher
3300 Milling & Mag Separation		
Ball mill	1	Outotec 4.00 m dia. x 6.35 m length, 1,550 kW
Mill cyclones	3	Multotec
LIMS	1	Outotec SLon® CTN-1224
WHIMS	1	Outotec Slon® 3000
Desliming 1st cyclones	3	Multotec
3400/3500 Mica and Spodumene flotation		
Mica flotation conditioning tanks	2	14.7 m3
Mica rougher flotation cells	5	Metso RCS 10
Mica scavenger flotation cells	6	Metso RCS 10
Dewatering cyclones	2	Multotec
Attrition scrubber	1	Mining Equipment VMIX-1500/90/315S
Desliming 2nd cyclones	2	Multotec
High density conditioning tanks & motors	2	Outotec Oktop®, 14.9 kW
Spodumene rougher flotation cells	4	Metso RCS 10
Spodumene scavenger flotation cells	4	Metso RCS 10
Spodumene 1st cleaner flotation cells	2	Metso RCS 5
Spodumene 2nd cleaner flotation cells	3	Metso RCS 5
Spodumene concentrate filter press	1	Diemme GHT1500.F12, 51 plates, 1.5 m x 1.5 m
3600 Tailings and Dewatering		
Tailings thickener	1	FLS 16 m dia.
Tailings filter press	1	Diemme GHT2500.F22, 101 plates, 2.5 m x 2.5 m

17.4.2 Crushing Circuit

ROM ore will be fed by truck to a 130 t capacity crusher feed hopper fitted with a stationary grizzly with 600 mm openings. The crushing plant will operate on 2 shifts per day of 8 hours. The average daily throughput will be 2,400 tpd. The crusher feed hopper discharges at a rate of 200 tph onto a vibrating grizzly with a 100 mm aperture. The oversized material is fed to the primary jaw crusher. The crushed grizzly undersize material discharges onto the primary crusher conveyor belt.

Material is transferred from the primary crusher conveyor onto the primary screen feed conveyor, which feeds a triple deck vibrating screen with 165 mm (protective screen), 32 mm and 13 mm openings, respectively. The upper and middle deck oversized material discharges to the secondary crusher feed hopper. The bottom deck screen oversized material discharges to the tertiary crusher feed hopper.

The secondary and tertiary crusher products will discharge onto the cone crusher discharge conveyor and then onto the screen transfer conveyor. The cone crusher discharges are combined with the primary crusher material and transferred to the triple deck screen. The screen undersize material (-13 mm) is conveyed to a covered crushed ore stockpile. The stockpile will have a total capacity of 35 hours. A front-end loader will be used to manage the stockpile. The crushed ore from the stockpile will be conveyed to the ball mill feed chute. The ball mill will be located in the process plant building.

17.4.3 Milling Circuit

Feed to the milling circuit will be controlled by a variable speed belt conveyor from the covered stockpile. The milling circuit is designed and sized to grind crushed ore (feed size of 80% passing 8,000 µm) to a product size of 80% passing 180 µm. The nominal throughput will be 2,600 tpd at 93% availability or 108 tph. The ball mill will have the flexibility to grind coarser or finer by adjusting the ball charge in the mill to suit the liberation characteristics of ore fed to the plant.

The mill is a conventional ball mill equipped with a trommel screen and operated in closed-circuit with a cluster of hydrocyclones. The cyclone underflow is returned to the mill feed chute. The 13 ft diameter by 21 ft length, 2,280 HP mill is sized in an effort to minimize slurry residence time and fines production, and will have a circulating load of 250%. The mill discharge pulp density is controlled at 68% w/w solids by adjusting the water addition to the mill.

Ball mill media will be stored in a metal storage pit. A ball bucket and overhead crane will be used to feed the balls to the ball feed hopper on a periodic basis.

The ball mill cyclone overflow, with a P_{80} of 180 µm, flows by gravity to the magnetic separator pump box and on to magnetic separation. The targeted solids density for the magnetic circuit feed is 30%. If the solids density is too high, water can be added to the magnetic separator pump box.

17.4.4 Magnetic Separation Circuit

The ball mill cyclone overflow, at about 30% w/w solids, is fed to the low intensity magnetic separator (LIMS). The purpose of the LIMS is to remove iron fines from ball mill media wear. The pegmatite ore is very abrasive and the LIMS unit prevents the clogging or plugging of the wet high intensity magnetic separator (WHIMS) intended to facilitate a smoother and uninterrupted operation with minimal downtime for maintenance. The purpose of the WHIMS unit is to remove iron-bearing silicate (paramagnetic) minerals such as amphiboles, biotite, and chlorite. These iron-bearing minerals are detrimental to the flotation process as they float along with the spodumene, resulting in lower grade concentrate. The LIMS is operated at about 1,800 Gauss, while the WHIMS is operated at about 10,000 G. Any highly magnetic material that is not removed by the LIMS unit will eventually clog the WHIMS unit.

The magnetic concentrate from the LIMS flows by gravity to a pump box which feeds the tailings thickener. The non-magnetic portion flows by gravity to the WHIMS. As the feed slurry passes through the rotating matrix of the WHIMS, it is exposed to a high intensity magnetic field causing the paramagnetic materials to attach to the matrix. As the matrix rotates outside the magnetic field the attached paramagnetic material is flushed from the matrix with water. The flushed magnetic material flows by gravity to the magnetic concentrate pump box. The non-magnetic fraction from the WHIMS flows by gravity to the de-sliming 1st cyclone pump box and is then pumped to the de-sliming 1st cyclones prior to the mica flotation circuit. On average, about 2.5% of the WHIMS feed is rejected to the magnetic concentrate. The magnetic concentrate will contain about 5% of the total lithium and 45% of the total iron in the ROM ore. The amount of magnetic concentrate removed and the lithium loss to magnetic concentrate will vary and is largely dependent upon the level of iron bearing minerals present in the ROM ore.

17.4.5 Flotation Circuit

The flotation circuits consist of mica flotation followed by spodumene flotation.

Mica flotation will remove mica, mainly muscovite, from the spodumene flotation feed, thereby minimizing contamination by mica in the final concentrate. The mica concentrate will contain mainly mica, quartz, and feldspars. The mass pull is expected to be about 15%, with about 2% of the total lithium reporting to the mica concentrate. About 10% of the total iron is also removed with the mica concentrate. The mica flotation tailings feed the spodumene flotation circuit. Mica flotation not only minimizes mica contamination of the final spodumene concentrate, it also upgrades the feed to the spodumene flotation circuit, allowing for a higher final spodumene concentrate grade.

The feed to the mica flotation circuit consists of de-slimed slurry from the 1st de-sliming cyclone underflow, which is diluted from 50% solids to 20% solids in the mica flotation conditioning tank. The 1st de-sliming cyclone treats the non-magnetic fraction from the WHIMS unit. The de-slimed slurry flows by gravity to the mica flotation conditioning tank. Mica collector and NaOH are added to the conditioning tank. The conditioning tank has an effective volume of about 15 m^3 with a slurry residence time of about 2 min. Flotation tank cells in a rougher-scavenger arrangement are used in the mica flotation circuit. The combined mica concentrate from the rougher and scavenger stages is sent to the tailings thickener, while the mica flotation tails are sent to the dewatering cyclones and pumped to the attrition scrubber. The attrition scrubber consists of four rectangular cells providing a total effective volume of 28 m^3. Each cell has an impeller powered by a 90 kW motor, for a total mixing power of about 12 kW/m^3. The scrubber discharge is de-slimed by the second de-sliming cyclones and fed by gravity to a first high-density conditioning tank at 60% solids, where a fatty acid collector is added. High intensity conditioning of the slurry is a key element for successful spodumene flotation. Effective conditioning requires a slurry density of at least 60% solids, intense mixing of the slurry, and a residence time of 10 min. Typically, the mixing intensity is in the range of 2.5 to 8 kW/m^3 of effective slurry volume. Ideally, the slurry and collector should be added close to the impeller blades for efficient conditioning.

The conditioning tank will be an Outotec Oktop® conditioning tank or similar type conditioning tank with an effective slurry volume of 20 m^3. The impeller will be powered by a 15 kW motor to achieve a maximum effective mixing power of 8 kW/m^3. A mixing power of 8 kW/m^3 may not always be necessary and, therefore, a variable frequency drive (VFD) will allow adjusting the mixing power while minimizing power consumption and equipment wear. The conditioned slurry then flows by gravity to the dilution pump box where the slurry is diluted to 30% solids prior to rougher flotation. The feed to the spodumene flotation circuit is expected to be roughly 80 tph after losses to slimes, magnetic separation, and mica pre-flotation. The tailings from the rougher stage are de-slimed by the third de-sliming cyclones and fed by gravity to a second high density conditioning tank at 60% solids. The conditioned slurry is then diluted to 30% solids in a dilution pump box prior to scavenger flotation. The total mass pull in the rougher and scavenger flotation circuit is expected to range between 20-23% of the feed mass to the rougher cells with over 80% of the total lithium reporting to the rougher/scavenger concentrate. The rougher/scavenger concentrate is expected to contain about 5% Li_2O.

The rougher and scavenger concentrates are combined and progress to the 1st and 2nd cleaner stages. The cleaner tails are returned to the attrition scrubber. The 2nd cleaner concentrate is the final spodumene concentrate. The mass balance is based on a final spodumene concentrate production rate of 14.1 tph or 314 tpd at 93% availability at a concentrate grade of 6.0% Li_2O. The iron content of the concentrate will be about 1%.

17.4.6 Concentrate and Tailings Handling

The final spodumene concentrate at 20-35% solids will be pumped to the concentrate filtration holding tank. The slurry will be pressure filtered to a moisture content of 6.5%. The filter cake discharges onto a conveyor belt that leads to a covered concentrate stockpile. The plant tailings are thickened in a 16 m diameter thickener from about 10% solids to about 55-60% solids, before being fed to a pressure filter that produces a cake with 12% moisture content, which is conveyed to a truck loading station.

17.4.7 Reagents and Consumables

Reagents used in the process include collectors for mica and spodumene flotation, dispersant, soda ash, sodium hydroxide, frother and flocculant. Consumption rates are estimated based on lab-scale testwork and pilot plant trials. Table 17-6 and Sodium hydroxide is delivered in liquid form in tanker trucks at a concentration of 50%. It is used for pH control in the mica flotation and de-sliming circuits.

The mica collector is delivered in solid form in 1,000 kg bulk bags. It is dissolved and added to the mica flotation conditioning tanks.

The spodumene collector (fatty acid) is delivered in a liquid bulk tanker truck. It is added to the high-density conditioning tanks prior to rougher and scavenger flotation.

Dispersant is delivered in solid form in 600 kg bulk bags. Dispersant is added to the attrition scrubber prior to de-sliming.

The frother (MIBC) is delivered in liquid form in 1,000 kg totes. It is added to the mica flotation cells.

Soda ash (58% Na_2O) is delivered as a dry solid in bulk tanker trucks. Soda ash is added as a solution for pH control.

Flocculant is delivered in solid form in 750 kg bulk bags. It is used to improve solid/liquid separation in the tailings thickener.

Table 17-7 list all reagents, media, areas of usage and their purpose.

All process reagents are contained in a separate area within the process plant to prevent contamination of surrounding areas in case of a spill. Safety showers are provided in the different reagent mixing and utilization areas in the event of contact with the reagents. Grinding media will be stored in pits located indoors and close to the ball mill.

The main consumables for processing the ore are the grinding media, liners for the ball mill and crushers, and the reagents used in the concentrator. Other consumables include cyclone parts, LIMS and WHIMS spare parts, screen media, and spare parts for the high-density conditioning tank.

Table 17-6: Reagents Consumptions

Reagent	Area	Use	Consumption (tpa)	Inventory on site (t)
Sodium hydroxide (5%)	Mica conditioning, high density scrubbing	pH control	265	45
Soda ash (5%)	High density conditioning, rougher and cleaner flotation	pH control	485	45
Dispersant (Lignosulfonate, 10%)	High density scrubbing and conditioning	Fine particle dispersion	221	9
Frother (MIBC)	Mica flotation	Frother	18	2
Mica Collector	Mica flotation	Surface-active agent	106	5
Spodumene Collector (Fatty acid)	Rougher and cleaner flotation	Surface-active agent	883	39
Flocculant (Cationic polyacrylamide, 0.5%)	Tailings thickener	Flocculate solids to assist in solid/liquid separation	159	7

Sodium hydroxide is delivered in liquid form in tanker trucks at a concentration of 50%. It is used for pH control in the mica flotation and de-sliming circuits.

The mica collector is delivered in solid form in 1,000 kg bulk bags. It is dissolved and added to the mica flotation conditioning tanks.

The spodumene collector (fatty acid) is delivered in a liquid bulk tanker truck. It is added to the high-density conditioning tanks prior to rougher and scavenger flotation.

Dispersant is delivered in solid form in 600 kg bulk bags. Dispersant is added to the attrition scrubber prior to de-sliming.

The frother (MIBC) is delivered in liquid form in 1,000 kg totes. It is added to the mica flotation cells.

Soda ash (58% Na_2O) is delivered as a dry solid in bulk tanker trucks. Soda ash is added as a solution for pH control.

Flocculant is delivered in solid form in 750 kg bulk bags. It is used to improve solid/liquid separation in the tailings thickener.

Table 17-7: Overview of Grinding Media and Liner Consumption

Media	Area	Liner sets per year	Consumption (tpa)
Balls (50 mm diameter)	Ball mill	N/A	1,093
Ball mill liners	Ball Mill	5	N/A
Jaw crusher liners	Crushing	19	N/A
Secondary crusher liners	Crushing	12	N/A
Tertiary crusher liners	Crushing	12	N/A

17.4.8 Power and Fuel Requirement

Site power includes electrical power to operate process equipment in the crushing circuit, concentrator building, and tailings stacking operation. Heating, ventilation, and air conditioning (HVAC) will also operate using electricity. No fuel is used at site other than for vehicles and mining operation.

17.4.9 Personnel

The labour for the processing plant includes salaried operating staff, maintenance personnel, hourly operating technicians, and laboratory technicians. The crushing plant will be operating two ten-hour shifts per day and the concentrator will be operating two twelve-hour shifts per day. Each employee will work 1,920 hours per year and the concentrator will operate 8,147 hours per year. For some positions, a little over 4 employees will be required to cover all plant operating time. There will be floating operators who will replace during vacation time.

Front-end loader operators to handle ore and concentrate stockpiles and dry stacking operations will be part of the mine operating crew and are not included in the processing plant operating costs. Table 17-8 gives a breakdown of the processing labour.

Table 17-8: Breakdown of Processing Personnel

Role/Position	Peak No.	Salary/hourly
Maintenance		
▪ Superintendent	1	Salary
▪ Foreman & planner	1	Salary
▪ Mechanic	13	Hourly
▪ Welder	7	Hourly
▪ Apprentice	7	Hourly
Total	**29**	

Role/Position	Peak No.	Salary/hourly
Processing		
▪ Mill/process superintendent	1	Salary
▪ Metallurgist	1	Salary
▪ Shift supervisor	3	Hourly
▪ Processing technicians		
o Control room operator	4	Hourly
o Crushing operator	4	Hourly
o Grinding operator	4	Hourly
o Flotation operator	4	Hourly
o Support	1	Hourly
Electrician	1	Hourly (5 day wk)
Instrument mechanic	1	Hourly (5 day wk)
Pipe fitter	2	Hourly (4-4 day shift)
Millwright	2	Hourly (4-4 day shift)
Total	**28**	

17.5 Crushing and Concentrator Layout

Figure 17-3 shows the general arrangement of the crushing circuit. Figure 17-4 is a front view of the concentrator building, with the crushed ore feed conveyor to the right. Figure 17-5 is a plan view of the concentrator showing the layout and general arrangement of process equipment.

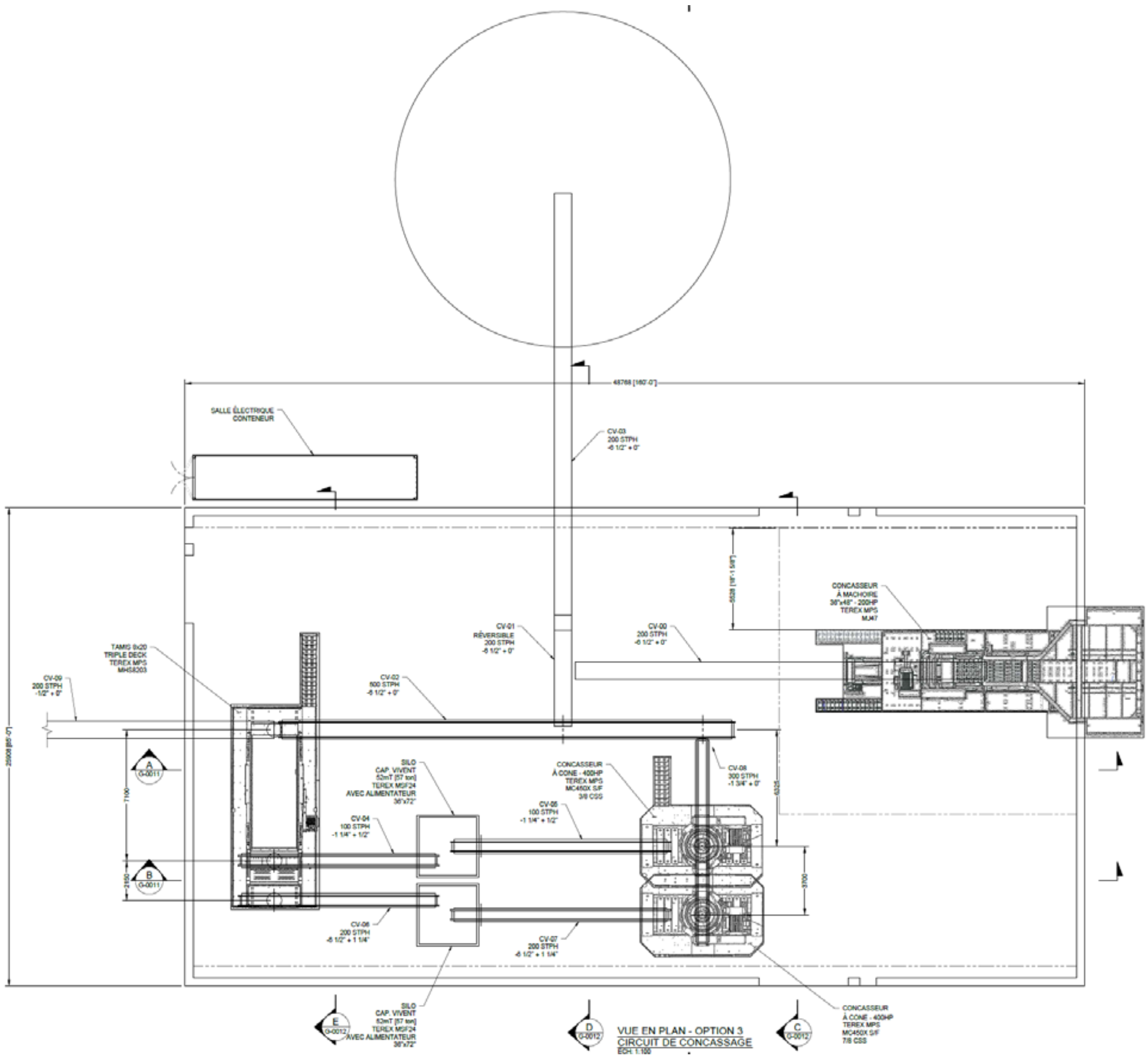


Figure 17-3: General Arrangement of the Crushing Circuit

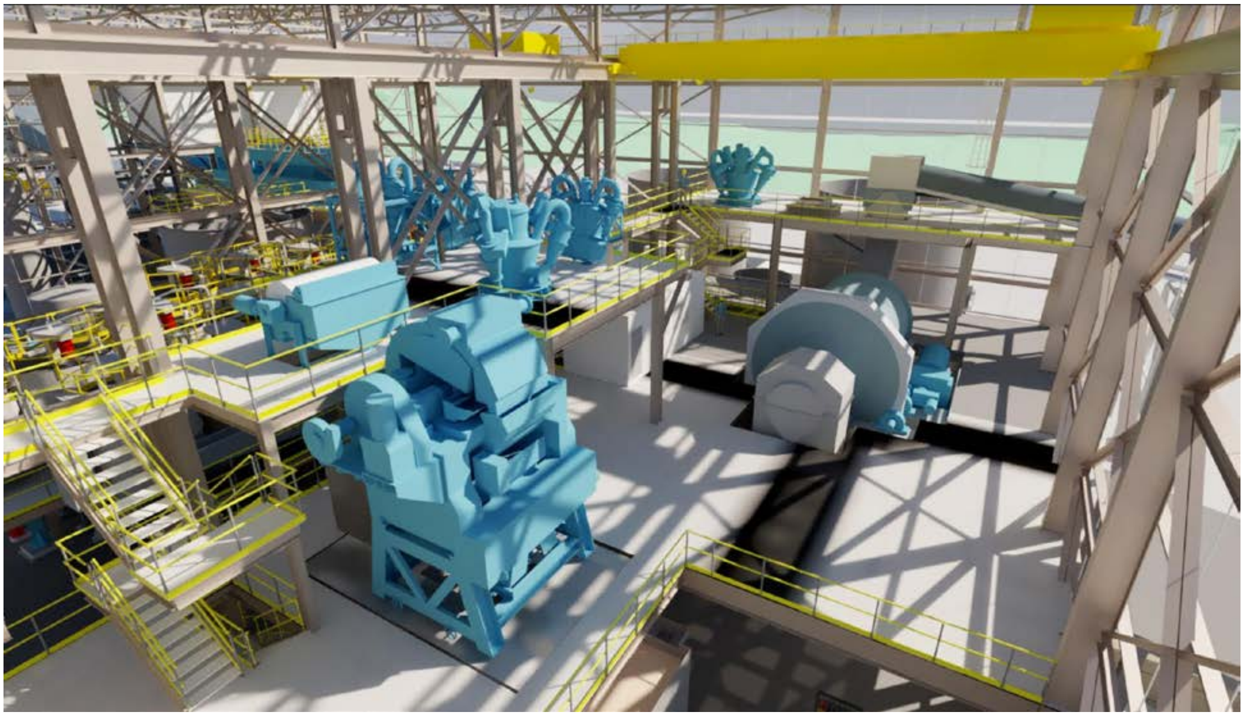


Figure 17-4: Concentrator Grinding Bay

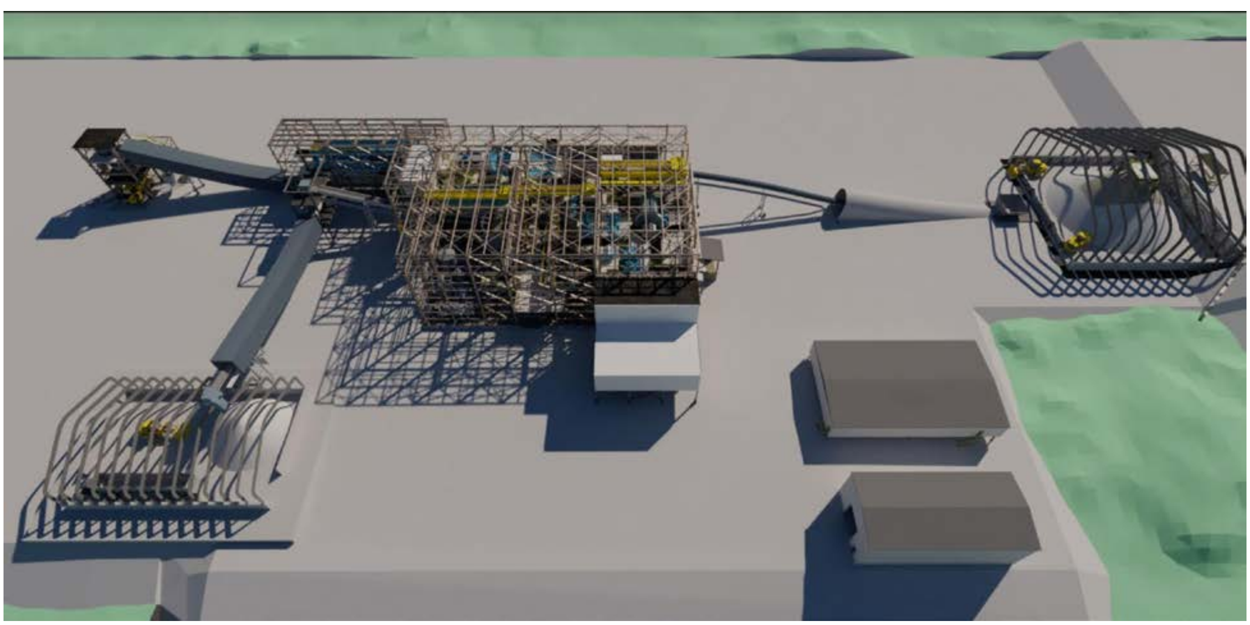


Figure 17-5: Concentrator and Storage Buildings

18. MINING INFRASTRUCTURES

18.1 Tailings and Waste Rocks Management

The following standards and regulations were used for the design of the Co-disposition Storage Facility and water retention structures:

- Directive 019 specific to the mining industry in Québec;
- Metal and Diamond Mining Effluent Regulations ("MDMER") in Canada;
- Loi sur la sécurité des barrage (The Dam Safety Law applied in Québec) ("LSB") and the associated regulation ("RSB");
- The Dam Safety Guideline produced by the Canadian Dam Association (2007);
- Manuel de conception des ponceaux (MTQ, 2004);
- Règlement sur la santé et la sécurité du travail dans les mines, Loi sur la santé et la sécurité du travail - Québec (2014) (Québec health and safety regulations);
- The Québec and/or the Canadian Legal framework applied to the environment and water sectors.

18.1.1 Co-disposition Storage Facility

18.1.1.1 Co-disposition Storage Facility Location

During the execution of the feasibility study a number of location options were studied by others; the chosen location was then optimized by BBA, moving the facility approximately 300 m to the west of the original site selection to create a buffer between the new location and a perceived sensitive esker;

The final layout is provided in Figure 18-1.

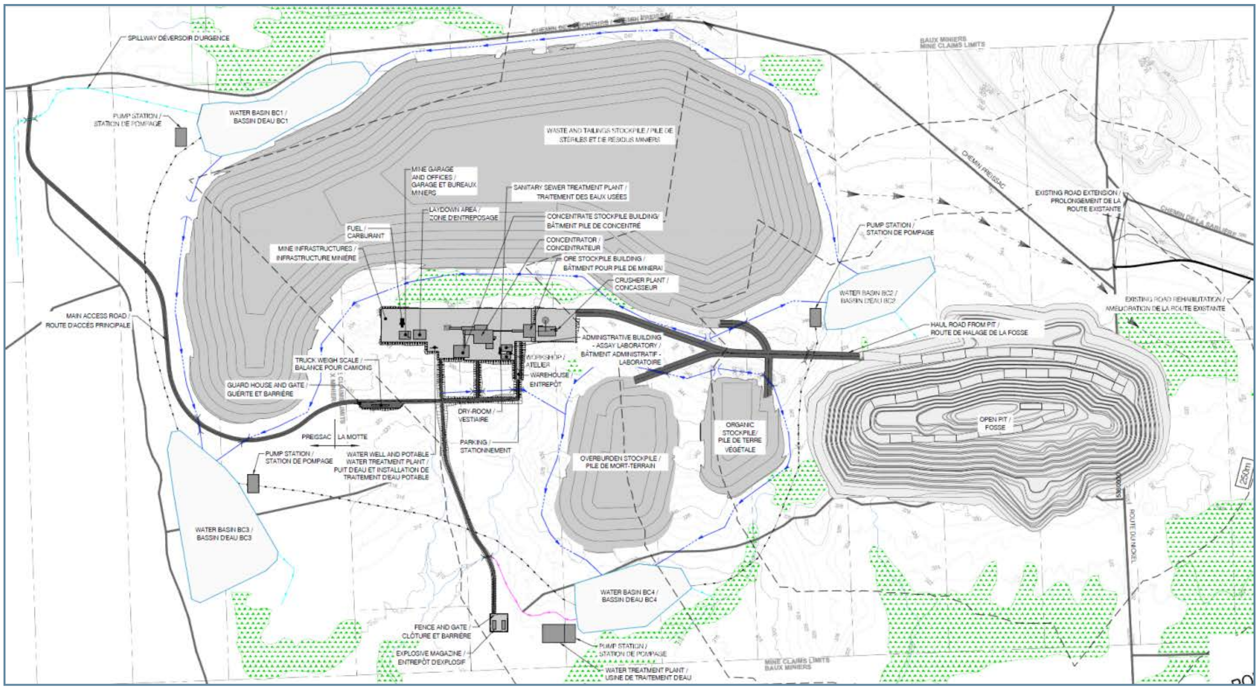


Figure 18-1: Site Layout

The original selection of the proposed location has been defined as per the following steps:

1. Analysis of site characteristics: based on aerial photos, LIDAR information and regional land use information. This includes the identification of existing infrastructure such as: electric lines, roads, forestry domains, and natural water bodies;
2. Volumetric compliance for tailings and waste placement: the targeted volume was around 6.6 Mm³ of tailings and 34 Mm³ of waste rock;
3. Preliminary analysis of the environmental and social constraints of the selected Co-disposition storage facility footprints.

18.1.1.2 Co-disposition Storage Facility

Sayona Mining has opted for a co-disposal method to store tailings produced at the concentrator and waste rocks from the mine. The co-disposal strategy consists of using waste rocks to construct peripheral berms and peripheral roads and confining filtered tailings into waste rock cells.

During the 15-year lifespan of the open pit mine (including preproduction), a total of 34 Mm³ of waste rock and 6.6 Mm³ of tailings will be generated for a total of 40.6 Mm³. The quantity of tailings has been calculated by subtracting the average yearly spodumene production (data from the PFS (Sayona, 2017)) from the concentrator's direct ore feed which is based on the life of the mine (Sayona, 2019). The deposition will take place during two different phases to reduce CAPEX associated with water management facilities.

The first phase of co-disposition starts from pre-production to Year 4 and will be contained in the eastern portion of the storage facility footprint, as shown in Figure 18-2. Phase 1 is contained within a watershed configured using 3 perimeter ditches, one of those being temporary, thus avoiding construction of collection basin BC1 and BC3 and associated ditches. In the subsequent phase, the co-disposal storage facility will be extended to the west to reach its full footprint, as shown in Figure 18-2.

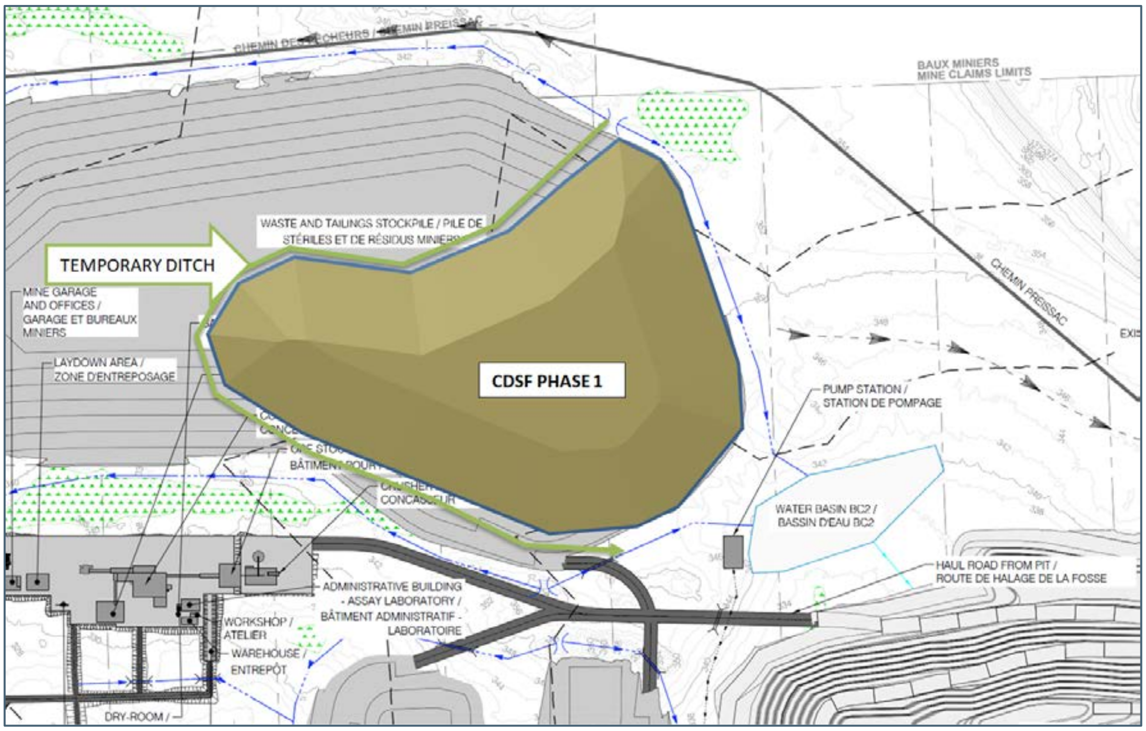


Figure 18-2: Waste Pile Footprint for Phase 1

A total of 1.8 Mm³ of tailings and 6.7 Mm³ of waste rocks will be co-disposed in Phase 1. Considering that the pre-production (Y-0) will only produce waste rocks, this material will be used to prepare the waste rock cells within which tailings will be deposited. In the subsequent years, the tailings will be disposed inside the waste rocks cells.

During the second phase, the co-disposal storage facility of Phase 1 will be expanded to the west. The goal is to reach the full footprint of the co-disposal storage facility in order to facilitate tailings and waste rock management. Waste rock will be used to create cells that will contain the filtered tailings. Optimization of this facility remains possible and should be considered in detailed engineering.

18.1.1.3 Tailings and Waste Rocks Storage Facility Design

The typical cross section of tailings and waste rocks is presented in Figure 18-3. Berms will be built in order to confine tailings within surrounding waste.

Waste rock (i.e., the deposition strategy) is planned to have sufficient available space in the cells to manage upcoming tailings). Tailings will be transported by truck from the concentrator to the co-disposal storage facility.

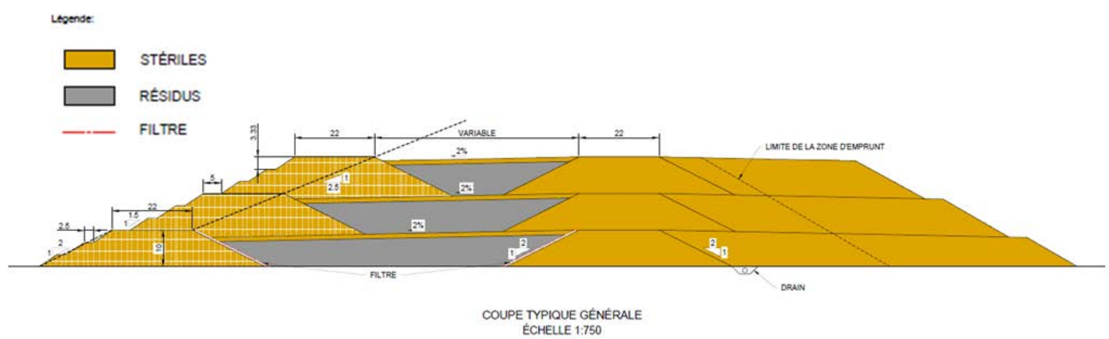


Figure 18-3: General Cross Section of the Tailings and Waste Rocks Facility

All the waste rocks and filtered tailings will be contained in this co-disposal storage facility. It was designed with the following parameters:

- Final overall slope angle: 2.5H:1V;
- Bench slope angle: 2H:1V;
- Bench height: 10 m;
- Ramp width: 22 m;
- Access ramp slope: 10%;
- Dry tailings density: 1.6 t/m^3;
- Waste rock and tailings are considered non-potentially acid generative and non-leachable;
- In-situ waste rock density: 2.3 t/m^3;
- This pile has a footprint of approximately 105 ha, and a maximum height of ±70 m.

Table 18-1 summarizes the total volumes of waste rocks and filtered tailings to manage and the associated capacity of the co-disposal storage facility for the 15-year life of the mine. Waste rocks quantities were obtained from information based on the life of the mine and mining plans.

Table 18-1: Summary of the Tailings and Waste Rocks Site Capacity

Parameter	Quantity
Total tonnage to manage	88.75 Mt
Total volume to manage	40.59 Mm^3
Waste rock volume	34.01 Mm^3
Tailings volume	6.58 Mm^3

18.1.1.4 Stability Analysis for CDSF and Related Infrastructure

Stability analysis has been performed in both static and Pseudo static states for the facility and includes:

- Critical CDSF sections;
- Overburden piles.

At this stage of the project the stability analysis has not included basin dikes and critical road section. Foundation parameters have been estimated at this phase.

18.1.1.5 Waste and Tailings Handling Methodology

Based on BBA past experience with projects of this size and the transportation distance of the waste and tailings, the handling of waste and tailings is to be conducted using trucks from the filter plant to the CDSF. The Capex and Opex related to the transportation and disposal of waste and tailings has been included in the mining portion of this report.

18.1.2 Water Management

18.1.2.1 Basins and Ditches

The design criterion applying to the storage capacity of the basins is the following: The water management basins must be able to manage a 24h rainfall with a recurrence of 1,000 years combined with a 100 years recurrence snowmelt, as per Directive 019 (MDDELCC, 2012), given that the waste and tailings are not acid generating and not leachable.

For water management basins where retained structures are considered, an emergency spillway and channel must be able to safely discharge the most severe flooding event. This is considered to be the Probable Maximum Flood (PMF) as specified in the Directive 019; Freeboard requirements are as stipulated by Directive 019 (section 2.9.3.1) and the CDA guidelines (section 6.4). Since there might be vulnerable infrastructures and environments directly downstream of the proposed dikes, the dikes must be designed to have a normal freeboard of at least 1.5 m

(Directive 019), measured between the dam crest and the maximum water level during the design event

The design criteria applying to the ditches of the CDSF are presented below and are based on a design rainfall of a 100 year recurrence as per Directive 019. The discharge was increased by 18% to take into account the impact of climate change:

- Minimum depth 1.0 m;
- Minimum base width 1.0 m;
- Minimum freeboard m 1.0 m;
- Minimum longitudinal slope 0.001 m/m;
- Minimum velocity 0.5 m/s;
- Lateral slopes are defined according to the natural terrain;
- Riprap has to be defined according to water velocities.

18.1.2.2 Water Management Strategy

The general water management strategy developed for the Sayona projects aims at:

- Divert off site all non-contaminated water from non-perturbed areas surrounding the site;
- Managing by draining, conveying and containing runoff from surface infrastructure from the mill and CDSF areas as well as underground water;
- Recycling a maximum of the mine site water from runoff, process, and groundwater for water supply purposes;
- For Total Suspended Solids (“TSS”) sedimentation, retain water in ponds prior to treatment or release to the environment;
- Treat all contaminated water before to release it to the environment;
- Minimizing the footprint in the first 4 years to reduce the basin requirements to Basins BC2 and BC4, in the initial phase;
- In order to reduce the required treatment capacity, the spillway for BC2 will overflow into the pit, in case of emergency. Sayona Québec will put appropriate protocols in place to monitor and control a potential overflow.

18.1.2.3 Project Watersheds

The project watersheds have been delineated in order to perform the design of ditches and basins. Figures 18-4 and 18-5 show the watersheds of the mine site in natural and developed conditions. Topographic information was gathered from *Données Québec* which gives access to LiDAR information at a resolution of 1 m.

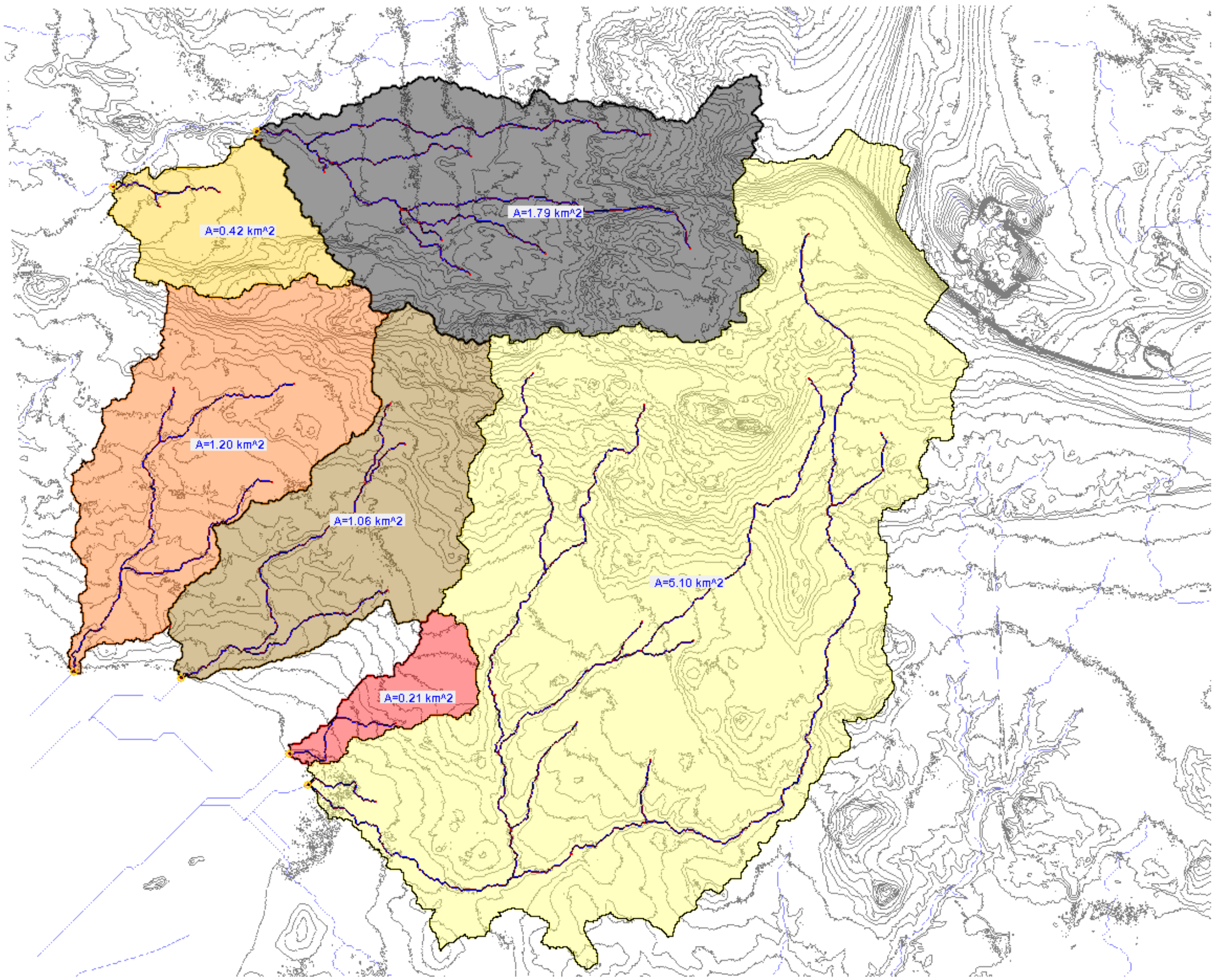


Figure 18-4: Project Watersheds in Undeveloped Conditions

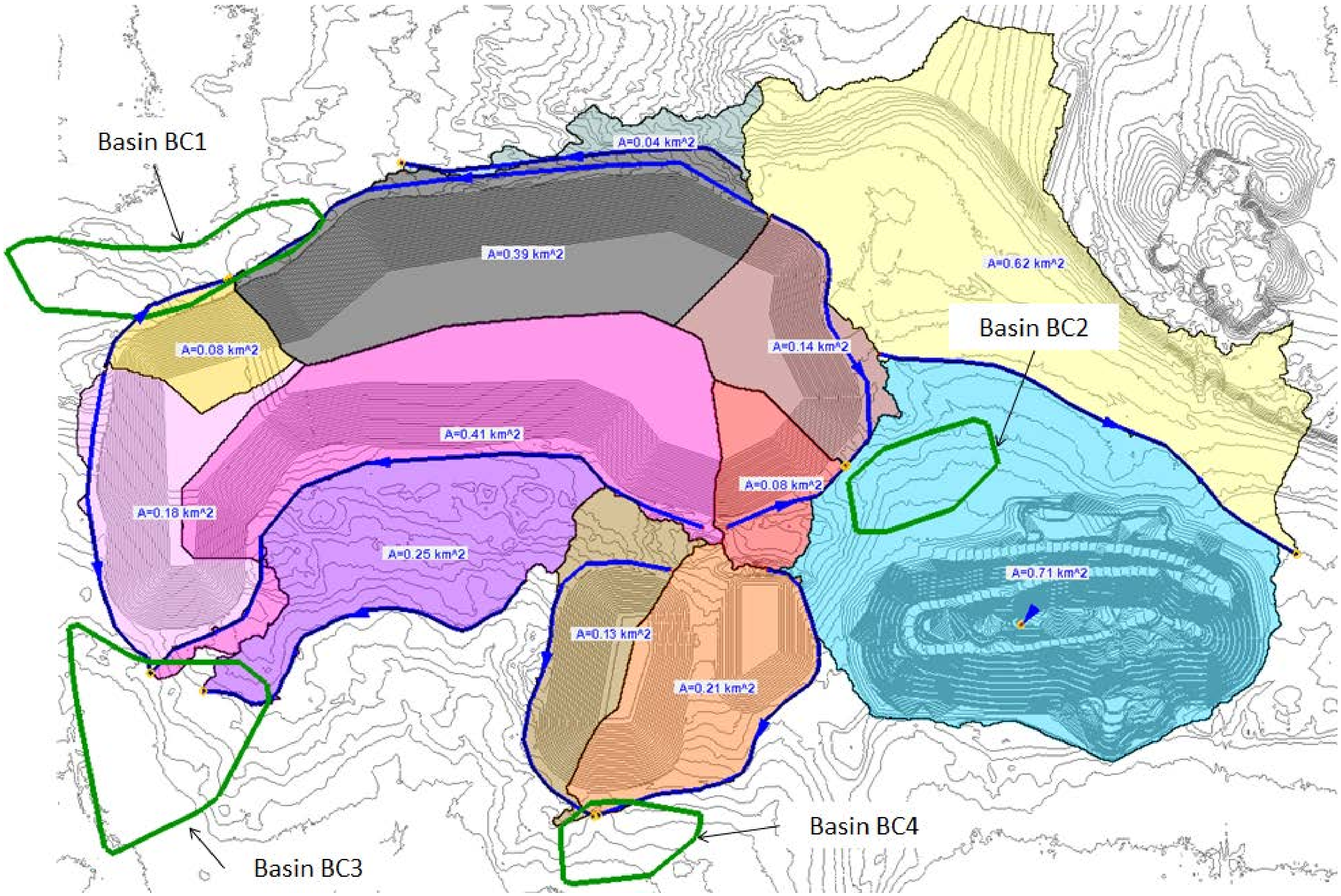


Figure 18-5: Project Watersheds in Developed Conditions

18.1.2.4 Operational Water Balance and Flux Diagrams

An operational water balance was performed for the normal hydrological conditions. The following parameters where considered:

- Total annual precipitations are 904 mm with 651 mm of rainfall and 253 mm of snowfall (SNC-Lavalin, 2018);
- It is assumed that the snowmelt occurs from mid-April to mid-May;
- The total annual lake evaporation is 460 mm (SNC-Lavalin, 2018);
- The potential evapotranspiration (ETP) is 364 mm (SNC-Lavalin, 2018). It is assumed that the stockpiles and the mine pit have respective evapotranspiration rate of 70% and 50% of the ETP;
- It is assumed that the ice cover of the basins is 1m thick and forms from mid-December to mid-April. The ice melts from mid-April to mid-May;
- The groundwater infiltration rate into the mine pit is 55 m³/h (SNC-Lavalin, 2018);
- Water is pumped from basin BC2 to the concentrator at a rate of 10.5 m³/h (communication from Sayona). It was assumed that the mill has a rate of utilization of 95%.

The resulting flow diagram and the main outcomes of the water balance are presented in Table 18-2: and Figure 18-6.

Table 18-2: Main Outputs of the Operational Water Balance

Parameter	Value (m³)
Yearly volume of water to be treated	2 088 959
Yearly volume to be sent to the mill	87 381
Yearly volume of water released to the effluent	2 088 959
Fresh water needs	0 (1)
The water that needs to be sent to the mill (10.5 m³/h) is largely compensated by the hydrological and groundwater inflows. Therefore, the water balance shows that there is no need for fresh water.	

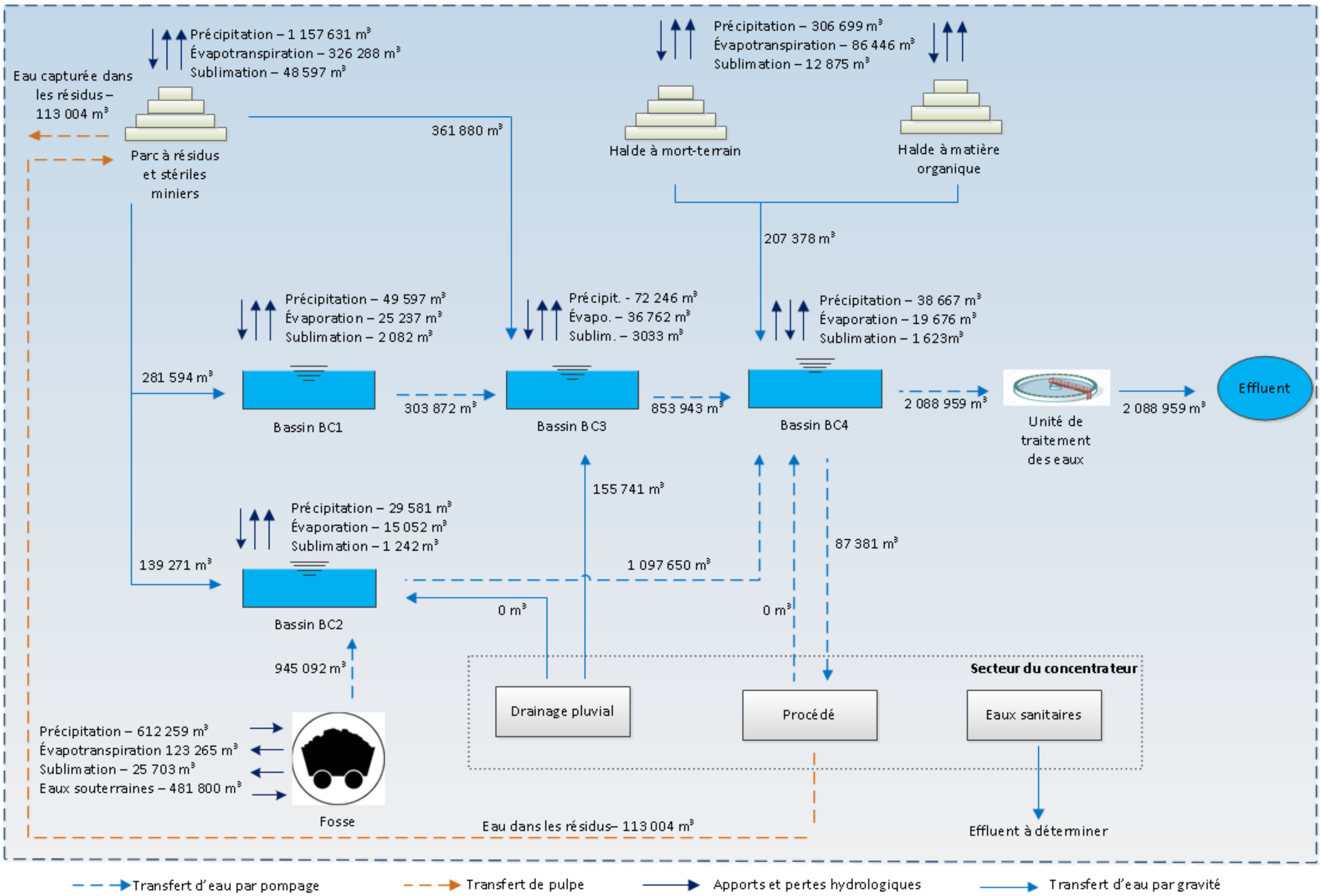


Figure 18-6: Water Balance for Normal Precipitations at Year 7

18.1.2.5 Basins Sizing and Design

Based on the design criteria (Guide 019) and the water management approach previously described, the environmental design flood was established.

Two basins, BC2 and BC4 will be required from the outset having a total storage capacity of 92,700 m^3 for the first 4 years of operation namely 32,600 m^3 for BC2 and 60,100 m^3 for basin BC4. These basins will ensure compliance up to year 4 of operations. Thereafter, the total capacity will need to be increased to 278,800 m^3 by the addition of two basins: BC1, with a capacity of 67,600 m^3 and BC3 with a capacity of 118,500 m^3. Basin capacity has taken into consideration the operation of a water treatment plant used to treat TSS and having a capacity of 0.33 m^3/s. The full treatment facility will only be required as of year 4, and only half of the capacity should be budgeted at the outset of operations.

Basin volumes will be attained partially through excavation and partially to the construction of dams. Dam height has been limited to roughly 4.0 m. Table 18-3 provides crest elevations for each basin as well as elevation for each associated spillway.

Table 18-3: Crest Elevations

Basin designation	Basin volume (m^3)	Crest elevation (m)	Spillway elevation (m)	Freeboard (m)
BC1	67,600	327	325	2
BC2	32,600	337	335	2
BC3	118,500	312	310	2
BC4	60,100	315	313	2

18.1.2.6 Design of the Ditches Surrounding the CDSF

A cross-section was calculated for the most critical ditch, i.e. the ditch with the largest watershed. This cross-section was then generalized to the entire site. The hydrotechnical parameters of the ditch are presented in Table 18-4.

Table 18-4: Typical Cross-section to be used for the Mine Site Ditches

Average slope	Discharge	Roughness coefficient	Base width	Lateral slope	Water depth	Velocity	Total depth (1)
[m/m]	[m^3/s]	[s/m$^{1/3}$]	[m]	[H:1V]	[m]	[m/s]	[m]
0.023	10.5	0.040	2	2	0.98	2.71	1.3

(1) Including a 0.3 m freeboard

18.1.2.7 Waste Water Treatment

All waste products coming for the Authier mine are considered to be non-acid generating and non-leaching, as such a conventional sedimentation and physio-chemical treatment approach was retained for the treatment of TSS. A water treatment facility will be required for this project. The water treatment plant is designed to handle the volume associated with the snow melt, while the design rain event will be stored in the various ponds. The design treatment capacity will be 0.33 m^3/s assuming 24 hr operation, with 90% availability. However, the treatments capacities for phase 1 can be reduced to 0.15 m^3/s. Modular systems have been considered. It is assumed that the water treatment plant will not operate during the winter months. The proposed treatment system is composed by:

- Pre-settling basins;
- A polymer dosing unit;
- A coagulant and pH adjustment units;
- Filtering membranes bags (Geotube® or similar);
- And a polishing pond.

18.1.2.8 Assessment of the Risk of Climate Change

In general, consequences of Climate changes are a new risk that needs to be addressed in water management plans and for the design of the water management infrastructure (e.g., basins and ditches). Mitigation measures and adaptation measures have to be considered.

For the Authier Lithium project, the risk was analyzed based on available scientific data including recommendations put forward by the OURANOS consortium for the province of Québec. According to the simulations performed by OURANOS (www.ouranos.ca/portraitsclimatiques) for the Abitibi region, assuming Val d'Or as a reference station, the projections (2041-2070 horizons) of climate changes in terms of temperatures increase and precipitations are based on 'high level of greenhouse gas emissions' scenario (50th percentile) and shown in Figure 18-4.

Table 18-5: OURANOS Projections for Temperature and Precipitation

Mean Temperature	Projected variation (ºC)	Relative variation in Temperature	Mean Precipitation	Projected variation (mm)	Relative variation (%)
Annual	+3.2 (02.0)	260	Annual	+85 (900)	+ 9.4
Winter	+3.8 (-14.0)	73	Winter	+30 (161)	+18.6
Spring	+2.6 (01.4)	285	Spring	+32 (188)	+17.1
Summer	+3.1 (16.3)	119	Summer	-05 (295)	-15.3
Autumn	+2.9 (04.2)	169	Autumn	+25 (261)	+9.6

Note: variation is relative to the reference period 1981-2010

For the Authier Lithium project, the design for water collecting ditches has assumed an increase by 18% of the Intensity Duration-Frequency values that are available for the Amos weather station (Environment Canada). In order to manage the risk of an increase in runoff water volumes, the water treatment design capacity was increased by 10%. Also, in order to manage the risk, the mine pit was considered as a buffer in case of an extreme precipitation event beyond the design criteria. It is understood that during extreme events, the operations (in the pit) will be temporarily stopped.

Given that the Life of Mine (LOM) is about 14 years, the aforementioned design parameters are considered reasonable to cope with the risk of climate change.

18.2 Pads and Roads

18.2.1 Site Preparation and Pad

General site preparation will consist of clearing, grubbing, topsoil and overburden removal, rock excavation, backfilling and surface leveling for all site infrastructures. The needs for the plant site area and at the CDSF area were estimated based on rudimentary air photo interpretation. For the CDSF access road, between the plant site and the CDSF site, the needs were estimated based information provided by Sayona.

Clearing and grubbing will be done in and around all infrastructure areas in order. Topsoil and overburden will be removed to provide a stable sub-base for roads and pads. A general overview of the Authier site showing the property limits, processing plant site, CDSF, waste piles, and site infrastructure can be found in the general arrangement plan in Figure 18-1. Site drainage will be achieved with the excavation of drainage ditches at the extremity of the processing plant site pad, on the side of the roads and around the sedimentation ponds. A frost depth of 2.8 m is considered for building foundations not sitting on bedrock and for the underground piping networks.

The granular pads accommodate the following structures:

- Concentrator building
- Concentrate stockpile building
- Ore stockpile building
- Assay laboratory
- Sewage treatment plant
- Potable water treatment plant
- Administrative building
- Dry-room
- Warehouse

- Crusher plant
- Workshop
- Fuel storage
- Mine garage and offices
- A laydown area
- Explosive magazine

18.2.2 Site Road, and Site Security

The site entrance is located on Chemin des Pêcheurs to the North-West of the property. The main access road has a total width of 10 m and is approximately 1.9 km long, and accessing the employee parking and process plant. The processing plant site access is controlled by an access gate located approximately 1.5 km m away from the Chemin des Pêcheurs entrance. A separate entrance is designated to the employee parking in order to segregate light from regular vehicle traffic for safety purposes.

On-site roads consist of mostly of heavy-duty traffic haul roads for access between the mine infrastructures, process plant and open pit areas as well as the waste/tailings, overburden and topsoil stockpiles. The total width of haul roads is 20 m and total length is approximately 1.2 km. A 600 m long and 7 m wide light-vehicle traffic service road also goes from the process plant to the explosive magazine.

19. PROJECT / SITE INFRASTRUCTURE

The project infrastructure comprises the offsite infrastructure and the onsite infrastructure. Offsite infrastructure includes the access road, the power supply line and substation, and the communication line. Onsite private infrastructure includes circulation infrastructure, buildings and all associated utilities.

19.1 Offsite Infrastructure

19.1.1 Road Relocations

The proposed pit location and blast exclusion zone of 250 m will result in the closure of approximately 800 m of the Route du nickel, which runs north-south through the property (Figure 19-1).

To stay outside of the 250 m blast exclusion zone, a new portion of road (approximately 150 m) will be constructed to connect Route du Nickel and Chemin de la Sablière. Allowances have also been made for appropriate signage and warning lights to be placed on this road.

These roads are currently constructed to a gravel surface and any new works will need to be undertaken to the same standards as required by the local authority. The Route du nickel is part of the emergency access road to Preissac as it is the only access to the village if the Preissac Bridge becomes unusable. Thus, the road construction must therefore be completed prior to excavation of pit.

19.1.2 Road Improvements

The proposed mill location will connect to Route 109 via Chemin des Pêcheurs, Chemin Preissac (4.7 km), and Chemin de la Sablière. These segments of road will be upgraded to all-weather road to allow concentrate shipping and the receipt of reagents and other supplies. The current road will be replaced by a new, upgraded gravel road. A new portion of road will be constructed to directly connect the intersection of Chemin Preissac / Chemin du Nickel and Chemin de la Sablière.

19.1.3 Power Supply

A 25 kV power line from the Hydro-Québec grid network is located 7 km away and to the south of the mine site. The line has sufficient capacity to feed the mine site. Bringing power to site will require the modification of an existing mono-phase line (2.5 km) to the intersection of Route du Nickel and Chemin St-Luc and a further extension to the Sayona sub-station located in proximity to the process plant. Hydro-Québec will install the line to the Sayona sub-station located in proximity to the process plant.

The overhead power line and associated infrastructure will be built to support a 7.0 MVA load and will be connected to main substation.

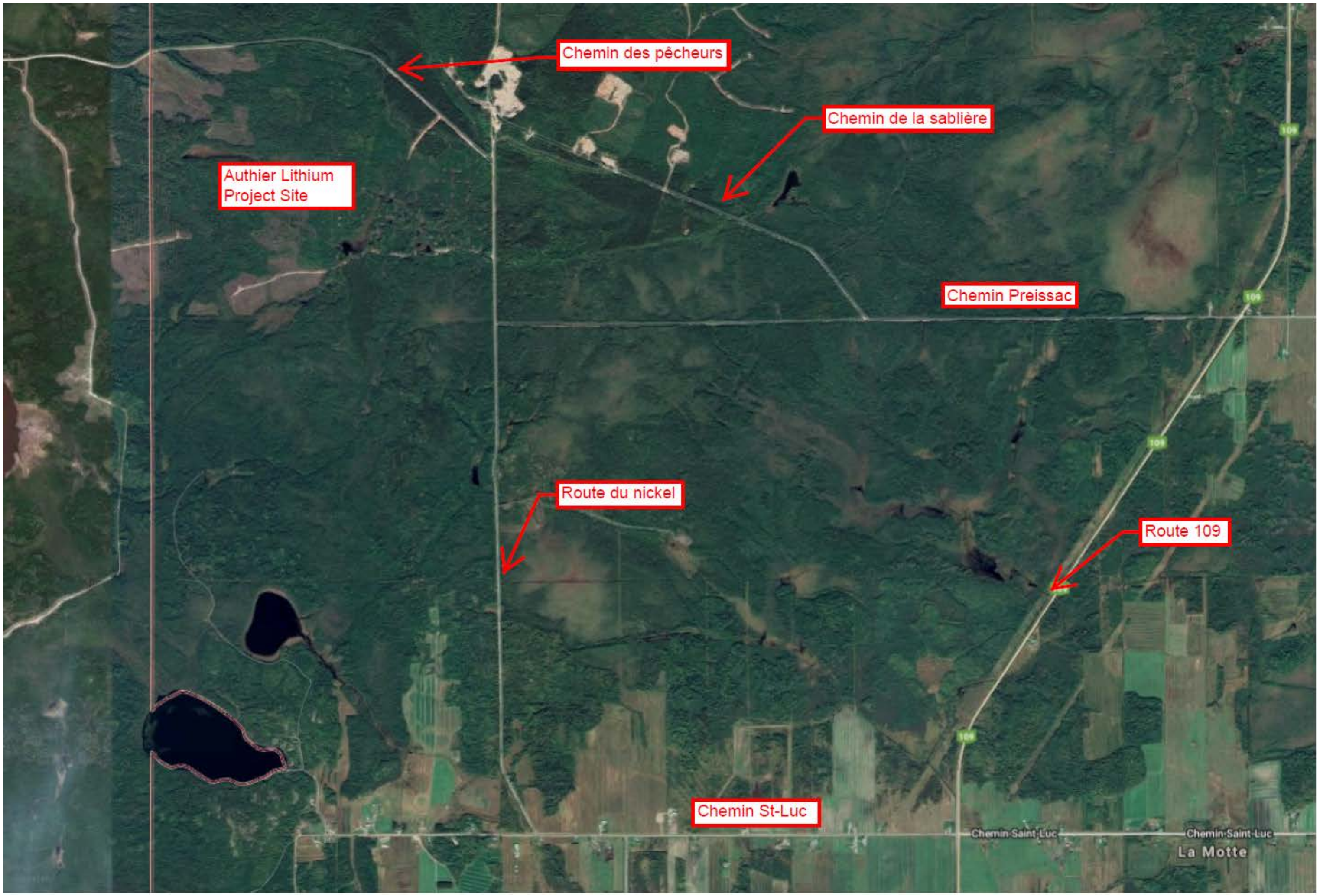


Figure 19-1: Road Network in the Vicinity of the Authier Lithium Property

19.2 Site Layout

A preliminary site layout has been prepared, which considers the operational requirements for the site, light and heavy vehicle traffic flows, site access, pit access, water management, environment infrastructure locations and stockpiles. Figure 19-2 shows the overall site layout and offers a close-up overhead view of the mine industrial area and process plant, including the crushing areas.

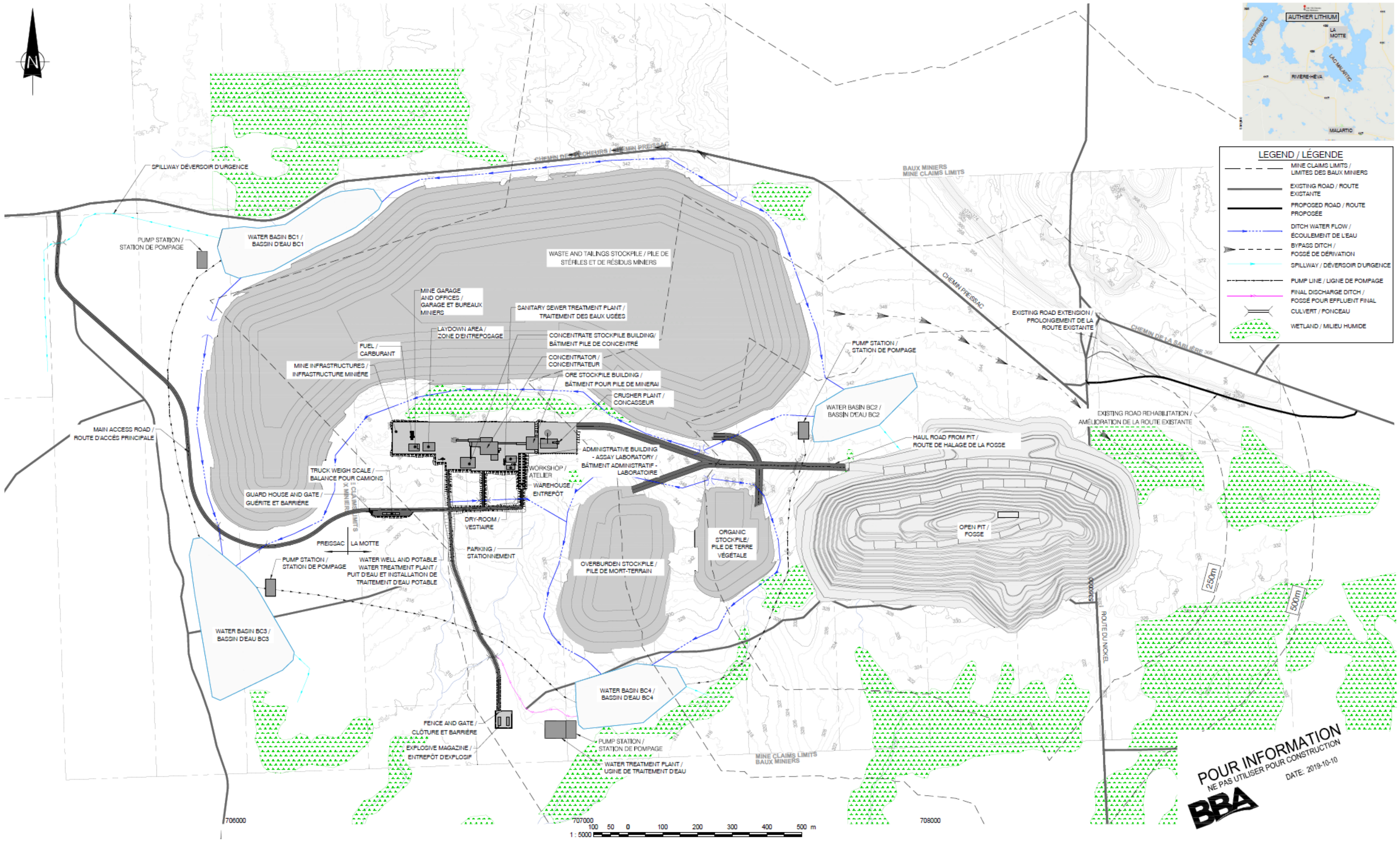


Figure 19-2: Authier Lithium Project Site Layout

19.3 Onsite Access Road, Haul Roads and Internal Roads

Access to the site will be via a road that runs north-south and connects to the Chemin des Pêcheurs. The access road is on the north-western portion of the property and is located in the municipality of Preissac. The site road will be 10 m wide and lead to a mine security and access point.

The main access road will provide access to the employee vehicle parking area, after the main security gate and truck weigh scale. Traffic gates will be installed at strategic points to control the circulation for safety issues. Gates will be installed on the site access roads to temporarily prevent traffic from entering the property or leaving the industrial site. Traffic gates will be closed prior to blasting and standard operating procedures will be developed to sweep the road. Vehicular traffic is to be kept at least 250 m from the pit during blasting.

All roads and circulation areas are defined based on standard engineering practices. However, they will have to be designed according to the subgrade conditions and the different vehicle load types, once this information becomes available. At the time of writing the UDFS, geotechnical data was not available for all site infrastructure areas. It is expected that select rock materials from the overburden will, over the course of time, be available to be sourced and crushed on site to produce a suitably blended gravel to be used for construction of all site roads and hardstand gravel pavements.

The mine plan allows for three months of pre-production, during which insufficient waste rock will be available to complete all roads and pads as shown.

19.3.1 Haul Roads

Heavy vehicle (HV) haul roads will be laid out to provide access to the active pit, the waste dump area, the ore stockpile laydown pad and the mine industrial area (MIA); the haul road to the MIA runs along the north side of the process area. These are two-way roads, 20 m wide with a geometry accommodating mining dump trucks.

Light vehicles (LV) access to the pit and ROM dump area will be via the concentrator pad and will share the HV haul roads along with the heavy vehicles. Driving and communication standard operating procedures will be developed to manage HV / LV interaction on HV haul roads.

19.3.2 Internal LV Roads and Car parking

Internal light vehicle (LV) roads will be constructed prior to the commencement of operations. Two-way LV roads will be constructed with a 7 m wide gravel surface.

One light vehicle car park for 75 vehicles will be provided adjacent to the administration building at the process plant.

The explosive magazine storage area will only be accessible via the main access road, 700 m after the security gate. It consists of a single lane road suitable for LV traffic.

The bulk of the mine infrastructures are proposed to be located in the MIA to the west of the process plant. The MIA has an overall approximate area of 13 ha, once fully developed.

19.4 Mill and Associated Buildings and Services

19.4.1 Crushing and Process Plant

The primary crusher is located at 800 m and the process plant at 800 m to 1,000 m from the pit to ensure an adequate perimeter from the pit (a departure from the pre-feasibility study which had the processing area just outside of a 250 m limit from the pit's perimeter). While the previous layout was more compact, it was chosen without sufficient geotechnical information and did not respect certain minimum distances that must be observed to protect personnel and equipment from errant fly rock resulting from blasting.

The primary jaw crusher was located close to the plant and placed in a manner that takes best advantage of the local topography. Where possible, the process buildings were placed on known outcroppings of rock.

Crushing infrastructure comprises: the primary jaw crusher, screening, and secondary and tertiary cone crushing. After crushing and before the process plant, a covered, crushed ore stockpile with total storage capacity of 35 h of feed will be built. Ore from the stockpile will be fed continuously to the mill via two reclaim belt feeders. The crushed ore stockpile will be managed using mobile equipment to push material into the reclaim feeders or feed a hopper located outside of the ore stockpile structure. The option to maintain stockpiles outside of this structure will also exist to increase flexibility during periods when the crusher is down for maintenance.

The process plant is constructed in three bays. The first bay holds the ball mill, classification and magnetic separation unit operations. The second bay is dedicated to flotation and the third bay will be used for concentrate production and reagent storage. To minimize the size of the process building, the tailings filter press electrical was located outside of the building as was the tailings thickener, process and HVAC rooms, which will be located immediately adjacent to the process building in elevated, pre-fab buildings.

19.4.2 Mine Industrial Area (MIA) Infrastructure

The mine industrial area (MIA) infrastructure will include the following elements:

- Temporary construction management facility;
- General earthworks, drainage for the temporary construction management facility, process plant area, including crushing, ROM stockpile and MIA; earthworks for overburden dumps will be undertaken by the mining fleet;
- Administration facility;
- Assay laboratory;
- Personnel changing area (dry);
- Workshop, tire change, warehouse and storage areas;
- Explosive magazine storage;
- Fuel, lube and oil storage facility; and
- Reticulated services, including power, lighting and communications; raw water and clean water for fire protection, process water and potable water; potable water treatment plant; sewage collection, treatment and disposal.

19.4.3 Temporary Construction Management Facility

An area of approximately 1 ha will be provided as a hardstand area for the establishment of a construction management building and car park. The building will be a pre-engineered re-locatable type structure with temporary services (tank and pump for potable water delivered from offsite, self-contained waste water collection facility for pump out and disposal offsite, temporary communications facility and temporary one-phase power line for construction power). The facility will cater for 120-140 persons. It is expected that this facility, including services and servicing requirements, will be sourced under a lease/service type Contract. Construction contractors for the process plant and the MIA buildings and services will be required to supply similar facilities for their management purposes and workforce requirements. At the completion of construction, these facilities will be reallocated to operation or removed and the disturbed areas rehabilitated in accordance with the site environmental requirements.

19.4.4 General Earthworks Including Hardstands and Laydowns

The process plant and ROM area, including the area for the administration buildings, laboratory, other ancillary buildings and car park is approximately 11.5 ha in size and the MIA area is approximately 1.5 ha. At the commencement of construction these areas will be cleared of vegetation and topsoil, and graded to peripheral ditches, which will direct runoff from these areas to site collection ponds.

All trafficked areas (pads) will be designed with gravel pavements suitable for the foundation soils and the classes of vehicles using them. When suitable, overburden from the pit will be used for the majority of the large volume of fill required to create the ROM storage pad and dump area. A laydown area will be constructed with a graveled surface adjacent to the process plant for use during construction. At the completion of construction, the gravel surface of the laydown area will become a part of the operational site. It is expected that waste rock will be available over the course of time to be crushed and screened on site and of suitable quality for construction of all site roads and hardstand gravel pavements.

19.4.5 Administration Facility

The proposed administration building will be located to the south of the process plant, and will be a light construction modular building with steel cladding and roofing. This building will be sized for a workforce of 35 persons and will include offices for the various departments, first aid room, washrooms (M/F), communications and store room, dining room, and meeting rooms. The building will be compliant with the relevant Québec and Canadian Building Codes.

A changing area (dry) will be annexed to the administration building.

Part of the administration building will be built as part of the early works program and will serve as the construction office during the construction period.

19.4.6 Assay Laboratory

A light-construction modular building, similar in construction to the administration building, will house the assay laboratory, which will be staffed 24/7 by contractors and will have dedicated areas for sample preparation, pulp storage, a weighing room, wet chemistry lab space, reagent storage, a room dedicated to the ICP, meeting rooms, office space, wash rooms (M/F), electrical and other utility rooms.

The site plan shows the administration building and assay laboratory as being co-located and may be built as one single structure, but will likely be built as two separate buildings.

19.4.7 Cold Storage Warehouse

Adjacent and to the south of the process plant, the cold storage warehouse will be a Mega-Dome type structure with a floor space of approximately 400 m^2, which will include an office and receiving/issues area.

19.4.8 Workshop

A small workshop will be located next to the cold storage warehouse and will be used primarily to service the process plant.

19.4.9 Mine Security and Access Point

A guard house and gate will be erected at the entrance to the industrial area, along the main access road. This area will also be the site of the weigh station, which will weigh incoming and outgoing concentrate transport trucks and bulk chemical supplies. The guard house will be a serviced, pre-fabricated building, similar in construction to a mobile home.

19.4.10 Maintenance Garage

The proposed single bay maintenance garage will be a pre-engineered, steel-framed and steel-clad structure, sized to suit an unloaded Caterpillar 775G off-highway haul truck with the tray down. Access will be drive in and reverse out, and the workshop will include an office area, a tool crib, a first aid room, meals room and washroom facilities (M/F). Light servicing of both the HV and LV vehicles will be performed in this workshop, with heavier servicing performed at a service centre outside of the mining site. The garage will be equipped with a 10-t overhead travelling crane.

An additional uncovered bay will be provided at the southern end of the main workshop building for a tire change area. A 15-bay car park will be provided adjacent to the maintenance garage for the personnel working in this area.

Fenced hardstand areas will be built adjacent to the maintenance garage for large and bulky items, which will not be detrimentally affected by being stored outside, e.g. HV tires.

19.4.11 Fuel, Lube and Oil Storage Facility

An external bunded fuel facility is proposed to hold two 50,000 L diesel storage tanks, a 10,000 L gasoline storage tank as well as bulk lubricant and coolant supplies, which will be moved into the maintenance workshop as required. All tanks and piping will be of steel construction. The diesel supply will be fitted with high flow reticulation to the HV refuelling bay and both diesel and gasoline with low flow reticulation to a LV fuel dispenser. These quantities are deemed sufficient for more than a week of supply at peak operations (Years 5 to 8). A dedicated, self-bunded, semi-trailer sized bay will be provided for fuel and bulk lube deliveries. A fuel truck will be used for fuelling the dozers and shovels.

19.4.12 HV and LV Wash Down Facilities

There will be no HV or LV wash down facilities at site.

19.5 Water Utilities

19.5.1 Raw Water/Potable Water

Raw water, which will be treated and used for potable water, washrooms and emergency showers, is proposed to be supplied from one or two well(s) located on site. Raw water from the well will be pumped directly to a packaged potable water treatment plant (PWTP), which will produce clean water for the site.

Potable water will be distributed from the treatment plant to the administrative building and the MIA in underground PVC piping installed below frost depth.

19.5.2 Fire Water

Fire water for the mine site will be drawn initially from the fresh water tank (636 m^3) located on the north side of the process plant; if more supply is needed then BC2 will be used as a storage capacity. The fire water pumping system will consist of both an electric delivery pump to supply firefighting water to buildings throughout the mine site at the required pressure and flows, and a diesel driven electric start pump that will start in the event that power is unavailable to the electric pump or it fails to start with a set time of a fire demand being registered. An electric “jockey” pump will be used to maintain pressure in the fire mains. The maximum fire water requirement has been estimated at 268 m^3/h over a 2-hour period, with full replenishment required within 8 h. Water will be supplied to the fresh water tank from one of the water basins (BC2).

Fire water will be distributed from the tank to the administrative building, the crusher area and the MIA in underground PVC piping installed below frost depth.

19.5.3 Process Water

Process water will be stored in a storage tank (442 m^3) also located on the north side of the process plant. It will be fed primarily from the overflow water from the tailings thickener, the tailings filtrate and spodumene filtrate. It will be topped up with fresh water from the fresh water tank.

Anti-scalant will be added to the process water to reduce fouling of pipelines, spray nozzles and screen apertures. The estimated plant make-up water requirement is estimated to be approximately 187 m^3/d.

19.5.4 Sewage

Sewage and domestic waste water generated in the occupied areas of the process plant and MIA will be collected in underground PVC piping installed below frost depth and directed to a package sewage treatment plant (STP) located slightly to the west of the administration building. Clean effluent from the plant will be discharged into the process area peripheral ditch. Solid waste produced by the STP will be collected on a regular basis by a local cartage contractor and disposed of at a local authority sewage treatment farm.

19.6 Electrical Utilities

19.6.1 Electrical Room

There will be two electrical rooms. The main electrical room will house the MV motor control centre, LV switchgear and the LV Motor Control Centres (MCCs). It will also house the mill variable frequency drive (VFD), the standalone and wall mount VFDs sized over 30 HP; those sized under 30 HP will be installed in the MCC's.

The other electrical room will be installed in the crushing area.

19.6.2 Power Supply – Incoming

The process plant will be supplied by a 25 kV overhead power line coming from the Hydro-Québec distribution power grid. The connection point from the Hydro-Québec installation will be on the property and in close proximity to the process plant. Negotiation with Hydro-Québec is ongoing for the installation schedule and cost. The power demand required for the project was evaluated to be 7.0 MVA.

The 25 kV overhead power line will come into the site through a set of disconnect switches, recloser and metering unit as required by Hydro Québec, constituting a connection point to the utility network. Another section of overhead power line from the connection point going to the process area will feed a 7.5/10 MVA power transformer (25/4.16 kV) via a second set of disconnect switches, recloser and underground cable, to the electrical room that houses the medium and low-voltage distribution transformers and equipment.

Engineering and environmental studies for the power line will be carried out by Hydro-Québec in 2019-2020.

19.6.3 Power Distribution

The incoming power line voltage will be stepped down to 4.16 kV and then to 600 V. The main electrical room will supply power to the equipment located in the process plant. The secondary crushing area will be supplied through a 4.16 kV cable from the main electrical room. The cable will be hooked to wooden poles as an overhead power distribution line. In case of an outage, an emergency diesel generator of 500 KW will feed the critical loads, which will all be connected to one MCC in the main electrical room. To minimize the generator size, the process critical loads will be fed in sequence for a required minimum time to ensure the integrity of the process during the outage. A procedure describing the energization steps, priorities and sequences must be developed and put into place by Sayona Québec according to need; other critical loads such as emergency lighting and PLCs will be continuously fed.

The mine garage and administration office areas will be supplied by the 25 kV power line through pole mounted transformers, adequately sized for the needs of these two areas. A single-phase power line will be extended to feed the explosive magazine.

19.7 Communication Utilities

19.7.1 Communications Incoming

An optical fibre ground wire (OPGW) will be connected at the intersection of Route 109 and Chemin St. Luc, and will be installed on the new power line to the site. An allowance has been made for bringing the OPGW to the substation location and from the substation to the proposed communications station.

19.7.2 Communications – Site Wide

A factored allowance was made in this DFS for a site wide communications system. No details have been developed around its components or implementation. Cell phone coverage is available at site.

A site wide radio system will be installed for the mining operation and emergency response.

19.8 Explosives Magazines

Two explosives magazines will be brought on site by the explosives provider. One will house priming explosives, such as caps and detonating cord, and the other a small amount of explosives and boosters.

The magazines are to be strategically located in a fenced and gated area on the southwest corner of the Authier Lithium property, so as to meet provincial and federal explosives regulations. A gravel road from the MIA will be built to access this area. As the proposed main supplier of explosives is located in close proximity to the mine, magazine capacities will be kept at a minimum.

19.9 Waste Management

Mine site waste including general, green and regulated waste will be collected, recycled where applicable and disposed of according to its type.

19.9.1 General, Green and Regulated Waste

Domestic and general waste will be disposed of by licensed contractors, most likely at a local authority operated facility. Green waste will be recycled and utilised in regeneration works, where practicable and feasible.

Regulated waste will be disposed of by licensed contractors as per statutory requirements.

19.10 Surface Water Management

The water management infrastructure is composed of a clean water diversion ditch; non-clean water collection ditches that surround the projected deposition areas as waste rock and tailings dump, overburden dump, top soil dump; servicing areas; non-clean water retention basins; pumping stations and conveyance pipelines and a water treatment plant (WTP).

19.10.1 Design Criteria

The design criteria for surface water management are based on Directive 019 for the mining industry (MDDELCC, March 2012). For the Authier project, all the surface water collection basins, pumping stations and treated water outfall are designed to manage a spring surface runoff based on a 1 in 100-year snowpack depth melting over a 30-day period combined with a 1 in 1,000-year, 24-hour rainfall event, as per the Directive 019 requirements. The tailings and waste rock are assumed to be non-potential acid generating (Non-PAG) and non-leachable.

Furthermore, runoff water and underground water collected in the open pit will be transferred to a water collection basin that will provide sufficient residence time to allow for sedimentation of suspended material.

Water collection ditches are designed based on 1 in 100-year rainfall criteria.

Emergency spillways and freeboards have been considered as per Directive 019. I n general the heights of basins dykes have been limited to a maximum of 3.8 m in order to reduce any risk and consequences in case of dike failures.

19.10.2 Water Management Footprint

The mine water management plan (WMP) addresses the management of runoff water that has been in contact with the mine site as well as the clean water that actually flows through the project site. WMP included industrial area, overburden stockpiles and tailings and waste rocks storage facility runoff water. Runoff water and underground water from the open pit are also collected. The domestic water is collected and an appropriate treatment system is to be provided.

For the preparation of the WMP, priority has been given to the minimization of the impacted areas that generate non-clean water in order to reduce the water volumes that will be managed. On the other hand, reclaim of non-clean water is prioritized in order to maximize the reutilization ratio.

The WMP mitigates the volume of contact water inflows to be managed on site by diverting clean water into the environment.

19.10.3 Water Management Facilities by Project Phases

The development of the WMP for the Authier project is divided in two distinct phases (Phases 1 and 2). For each phase, the water management infrastructures (i.e., basins and pumping requirements) are sized based on the required volume of surface runoff to manage. The volume varies according to the catchment area of the tailings storage facility.

For Phase 1 of the project (pre-production to year 4 of the exploitation), two water collection basins (BC-2 and BC-4), located in strategically selected areas, are required to manage the surface runoff water on the Authier mine site and dewatering from the pit. For Phase 2, with the expansion of the tailings and waste rocks storage facility, two additional water collection basins (BC-1 and BC-3) will be added. The basins, ditches, pumping station and pipelines for the entire project are illustrated in Figure 19-2. For both phases, the pumping stations will be designed with sufficient redundancy and flexibility for maintenance.

Primary settling will take place in BC-2, BC-1 and BC-3 before being transferred into the BC-4 collection basin. For Phase 1, a water treatment plant (WTP) will be installed. For Phase 2, the WTP will be upgraded to allow for higher capacity.

BC-4 basin also collects water from the overburden area and runoff from the industrial area. A sufficient residence time is expected. It will facilitate suspended solid sedimentation. A final pumping station will transfer the treated water from the Polishing Basin (PB), inside BC-4, to the environment through the effluent EF ditch.

The WTP will treat contaminants such as suspended solids and heavy metals to comply with current regulations (Directive 019 and MMERD). Treated water will be discharged into the PB before the final discharge into the environment.

19.10.4 Pumping System and WTP Capacity

A total of four pumping stations are required over the life of the project. Phase 1 requires two pumping stations for BC-2, BC-4 and for the PB, while Phase 2 requires two additional pumping stations for BC-1 and BC-3. A dewatering pumping system is to be installed in the pit to basin BC-2 for Phase 1, the required capacity of the WTP is 600 m^3/h (one treatment unit) and for Phase 2, 1,200 m^3/h (two treatment units). In terms of the pumping stations, as pumping will not take place in winter, electric submersible pumps can be used for all the basins. The pumps will be installed on a floating structure and should be recovered before winter.

19.10.5 Treatment Technology

The main issue with the non-clean water is the presence of suspended solids. In general the removal of such material is achieved by gravity settlement in large ponds with sufficient residence time. However, the treatment will be based on a physico-chemical process where chemical products such as coagulant and flocculant will be added. The electrical charge of the suspended solids will be neutralized and large flocs will be formed which readily settle.

A water treatment plant will be located close to basin BC-4. The treated water will be sent to a polishing basin where quality control is to be completed prior to water being released to the effluent. Recirculation or backflow from polishing basin to BC3 is to be performed in case of non-conformity.

Note that the polishing basin (PB) will be designed in later engineering stages; the PB is part of the basin BC-4.

The process may generate sludge. Characterization of the sludge is therefore necessary to determine whether the sludge can be safely disposed in the co-disposal storage facility or sent out to a specialized landfill.

20. MARKETING

20.1 Lithium Market

Since its discovery, and for most of its history, lithium has been mostly known for its pharmaceutical application in treating mental health disorders. The number of applications has grown and now covers a diverse range of consuming segments including consumer electronics, transport, energy storage and construction. Since the early 2000s however, the demand patterns for lithium-based product applications have undergone a major shift. At the time, construction and household applications including ceramics, glass ceramics and greases, made up almost 50% of overall demand. Since then, partly due to the anticipated electric vehicle 'revolution', rise of consumer electronics (i.e., rechargeable batteries which previously represented less than 5% of demand), and increasing penetration of renewable energy, rechargeable batteries have surged to now account for nearly 50% of overall lithium consumption.

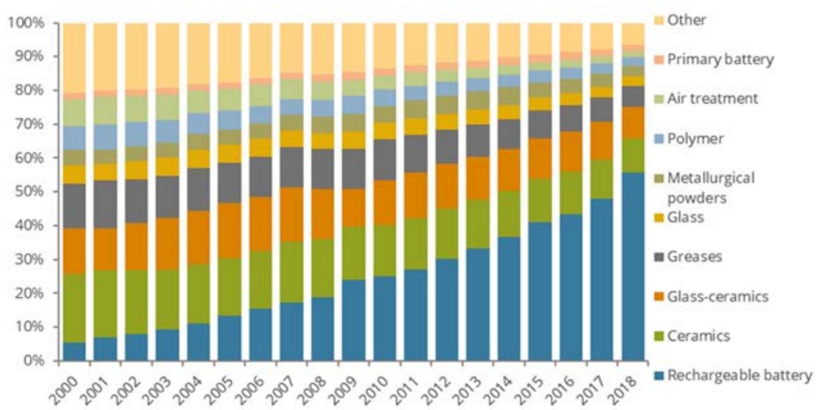


Figure 20-1: Lithium Consumption by Product Applications (Source: Roskill, Lithium Outlook to 2028)

20.1.1 Lithium Chemical Saleable Products

As part of the lithium extraction and processing value chain there are essentially three main saleable products from various sources; a mineral concentrate (typically a spodumene concentrate), lithium carbonate and lithium hydroxide. Lithium carbonate and hydroxide are typically produced in two grades, technical and battery grade.

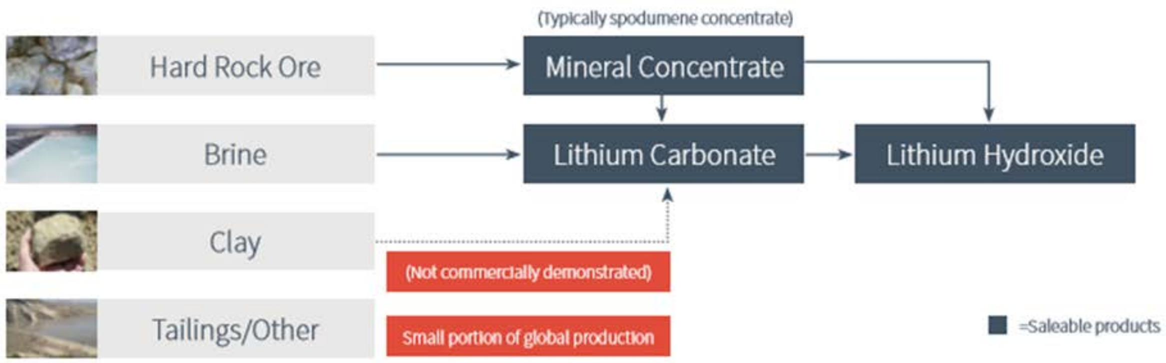


Figure 20-2: Lithium Saleable Products
(Source: Hatch)

Lithium hydroxide (LiOH) is an inorganic chemical available in anhydrous or monohydrate form. It is usually produced by evaporative crystallization following the reaction of dissolved lithium species with sodium hydroxide. Lithium hydroxide is a sensitive chemical that will react with carbon dioxide present in air to form lithium carbonate, which places extra requirements on shipping to maintain product quality. Lithium hydroxide has typically traded at a premium compared with lithium carbonate for supply into the battery chemicals market, as it is often preferred for the production of nickel rich cathodes.

Lithium carbonate (Li2CO3) – is an inorganic chemical with a relatively low solubility in water which has been exploited in order to extract lithium from water and brines. Lithium carbonate is typically produced by the reaction of lithium in solution with sodium carbonate followed by several washing and polishing stages. Lithium carbonate is the precursor that is most frequently used to produce other lithium compounds and products; for this reason various forms of lithium production are often quoted as lithium carbonate equivalent (LCE).

Mineral concentrate (spodumene) is the most abundant lithium bearing mineral found in economic deposits. It has a high lithium content (~3.7%) aluminosilicate mineral occurring as crystals in granites and pegmatites often intermixed with quartz. The high lithium content, combined with typical associations (i.e., quartz), means that spodumene deposits typically allow for high lithium recovery and concentrate grade through gravity concentration and/or flotation.

Other lithium chemical products include: lithium chlorite, lithium fluoride, lithium metal, butyllithium and other derivatives.

The Authier Lithium project being developed by Sayona Québec is targeting production of a spodumene concentrate eligible for offtake agreements to supply downstream refiners for conversion to a lithium carbonate or hydroxide chemical product to be used as an input into product application manufacturing.

20.1.2 Downstream Lithium Consumption by Product Type

The growth of the rechargeable batteries market has translated into a current downstream product mix whereby battery grade chemicals account for the majority of demand (56%) vs. technical grade (32%), with carbonate holding the largest share.

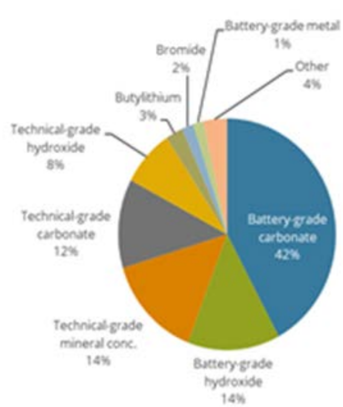


Figure 20-3: 2018 Consumption of Lithium by Product Type (% on t LCE basis)
(Source: Roskill, Lithium Outlook to 2028)

The lithium chemical demand profile is mostly driven by the prevailing battery chemistries being produced in the market. Currently, lithium nickel cobalt aluminum oxide (NCA) and lithium iron phosphate (LFP) are the most produced battery chemistries. Both are proven chemistries with the NCA being adopted by the Tesla Model S. Driving demand for the LFP chemistry is its adoption by e-buses and dominant market position in China.

Lithium nickel manganese cobalt oxide (NMC) is anticipated to become the preferred chemistry for application across the automotive industry as its proponents seek to improve performance, reduce cobalt dependence and lower costs. The expected shift away from LFP towards NCA and NMC through to 2030 provides a basis to drive an increase in demand for lithium hydroxide as enhanced battery performance increases the number of potential applications. Both lithium hydroxide and carbonate are set to experience robust demand growth over the coming decades as penetration of electric vehicles rises.

**Table 20-1: Battery Chemistry Summary Table
(Source: Battery University, McKinsey Research)**

Type	Materials	Automotive Applications	Other Applications	Specific Energy Wh/kg	Cycle Time	Specific Power	Cost $/kwh
LCO	Lithium Cobalt Oxide		Portable Electronics, Laptops	150-200	50–100	Low	Low
LMO	Lithium Manganese Oxide	Nissan Leaf, BMW I3	Power Trains	100–150	300–700	Mid	High
LFP	Lithium Iron Phosphate		Ebikes, E-tools, Ebuses	90–120	1000–2000	Low	Very High
NCA	Lithium Nickel Cobalt Aluminum	Tesla Model S	Consumer Electronics, Grid Storage, Power Trains	200–260	500	N/A	Mid
NMC	Nickel Manganese Cobalt Oxide (NMC 111, NMC 532, 622, NMC 811)	Nissan Leaf, BMW I3, Tesla Model 3	Consumer Electronics	150-220	1000-2000	High	Mid

In line with the anticipated surge in demand for rechargeable batteries as electric vehicles penetration rises and displaces internal combustion vehicles as the preferred means of mass transportation, Hatch estimates that lithium hydroxide and carbonate will experience a rapid increase in demand through to 2030.

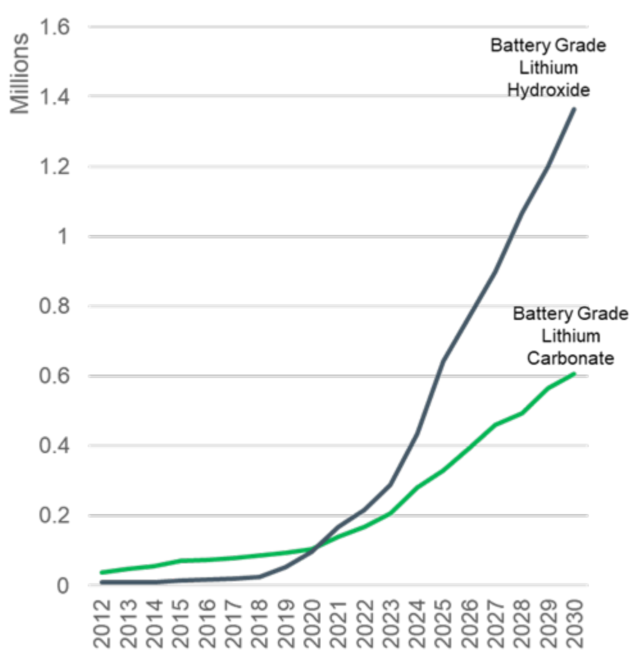


**Figure 20-4: Future Demand Outlook for Battery Grade Lithium Chemicals (LCE t)
(Source: Hatch Estimates)**

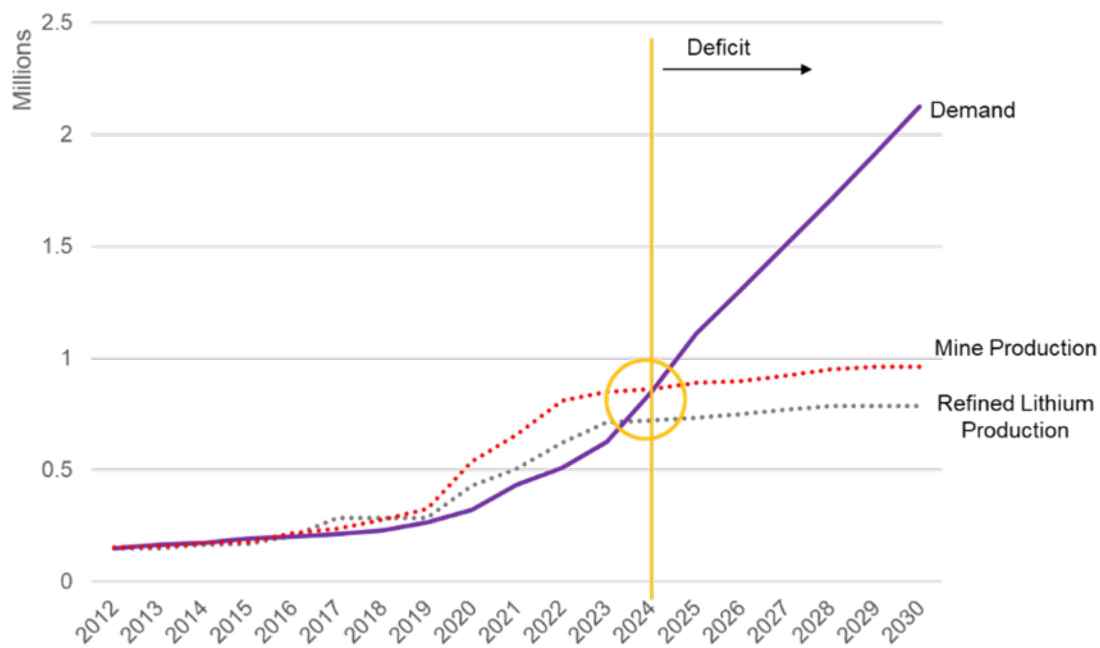


Figure 20-5: Forecast Demand and Supply of Lithium Products and Refining Plant Availability (LCE t) (Source: Hatch Estimates)

In spite of the anticipated increase in demand, Hatch estimates it will be outstripped by the growth in supply over the next five years as new mines, spurred by recent high prices, enter production. The future mine production and refined lithium production outlook, shown above, are estimated based on identified projects in the pipeline, which are factored based on an estimated likelihood of entering production. Future refined lithium production appears to be lower than mine production mainly due to a lack of information on refining projects in China. It can be expected that China will continue to build new refining facilities in the future and this will close out any gaps between the above computed future demand and supply forecasts. Hence deficit or surplus is evaluated from the intersection of demand and mine production.

20.2 Lithium Pricing

There are no terminal markets for spodumene or lithium refined products yet. In comparison, non-ferrous metals (copper, zinc, nickel, tin, lead, aluminium), ferrous metals (steel scrap, billet, hot rolled coils), minor metals (cobalt, molybdenum) and precious metals (gold, silver, platinum group metals) are traded regularly on terminal markets such as LME, COMEX or SHFE and secondary markets such as India, Dubai, Japan, Singapore and Hong Kong. A key function of a terminal market for any metal is to provide a transparent, unbiased discovery of prices. These reference prices are based on genuine trading sessions undertaken in the exchange. The prices then become the basis for commercial negotiations and settlements by producers and financial institutions. Due to lack of terminal markets for spodumene and lithium, the only alternative is to rely on prices published by data sources or industry news sources. Such sources are independent collectors and sellers of information with no vested interests in the commercial transactions, therefore the data can be considered independent and impartial. The pricing data is collected regularly, usually daily, based on surveys of market participants completed by as broad a cross-section of the market as possible, including distributors, traders, producers, end-users or brokers. The raw pricing data is then reviewed and normalized for technical specifications, freight rates and time differences. The normalization reflects the prevailing value in the market and facilitates like to like comparison of prices from different origins at a given market. The main sources for pricing data are Fastmarkets MB (previously Metal Bulletin), Benchmark Intelligence, CRU, Argus Media, Roskill and potentially up to 20 other organizations. To ensure price data collection complies with best practices, some of the larger data collection sources engage independent consultancies to review policies and processes. For the purposes of this report we have used Fastmarkets MB pricing data.

LME have announced that it is working on launching cash settled lithium futures contracts. These contracts will be based on transparent and robust prices for lithium. In June 2019, it announced that it will be partnering with Fastmarkets MB to create a set of transparent benchmark prices for lithium which would be acceptable for the wider industry. Lithium producers like Albemarle are reported to be using Fastmarkets MB reference prices for their commercial contracts.

Table 20-2: Fastmarkets MB Reported Prices of Spodumene and Refined Lithium Products

#	Product	Grade	Price Basis	Type
1	Spodumene	5-6% Li2O	CIF China, $/kg	Spot
2	Lithium Carbonate	99.5% Li3CO3 min, battery grade	CIF China, Japan, Korea, $/kg	Spot, contract
3	Lithium Carbonate	99.5% Li3CO3 min, battery grade	DDP Europe, USA, $/kg	Spot, contract
4	Lithium Carbonate Index	99.5% Li3CO3 min, battery grade	EXW China, ¥/t	Spot
4	Lithium Carbonate Spot Prices	99.5% Li3CO3 min, battery grade	EXW China, ¥/t	Spot
5	Lithium Hydroxide Monohydrate	56.5%. Li.OH.H2O min, battery grade	CIF China, Japan, Korea, $/kg	Spot, contract
6	Lithium Hydroxide Monohydrate	56.5%. Li.OH.H2O min, battery grade	DDP Europe, USA, $/kg	Spot, contract
7	Lithium Hydroxide Monohydrate	56.5%. Li.OH.H2O min, battery grade	EXW China, ¥/t	Spot

As can been seen in the above table, the price data is either spot or contracts. For a vast majority spodumene volumes are transacted under offtake agreements where prices are negotiated and settled principal to principal. Offtake agreements are usually agreed for five years with a fixed price through the term of the agreement.

20.2.1 Historical Prices of Spodumene

Spodumene prices have had three distinct phases recently. From 2012 to the end of 2015, prices increased by 1.7x to $600/t from $375/t as demand for lithium from battery manufacturers took off. This is the very early part of the lithium price cycle when demand outstripped supply. As the longer term view of lithium demand became more visible, it encouraged investments in new project development.

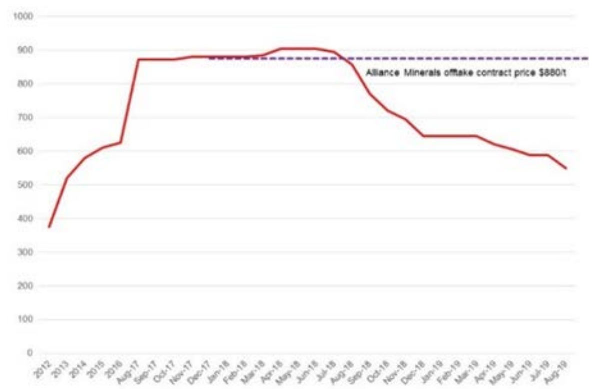


Figure 20-6: Spodumene Prices FOB Australia $/t
(Source: Core Lithium Investor Presentations, Fastmarkets MB)

From 2016 to early 2018, prices stayed relatively steady between $800-$900/t. During this period many new projects which were commissioned based their offtake contracts on these high prices. Demand remained strong relative to supply supporting prices.

From early 2018 to August 2019, prices declined by 36% from $900/t in January 2019 to $540/t in August 2019. The price decline was driven by significant increase in lithium carbonate supply and the balance shifted to excess supply globally. Alongside this shift in demand supply balance, many spodumene producers found converters under offtake agreements failing to pick up cargoes. These distress cargoes were then sold in the spot markets at prices significantly lower than offtake agreement prices. This trend also contributed to the sharp decline in prices during this period.

Until 2016, there was almost no spot market for lithium. The spot market was marginal, illiquid and largely ignored by the industry, therefore there was near convergence between spot prices and contract prices of spodumene. Since 2016, as more Chinese converter capacity was commissioned, activity in the spot market increased. Spot market prices started to diverge away from offtake contract prices. Spodumene producers started to seek greater exposure to the spot market and gain value from price upswings. The downside of this new development is when spot prices go below offtake prices; then there can be instances of commercial non-performance by converters. This then accelerates the divergence of spot prices vs. offtake contract prices.

The current transformation of the pricing of spodumene is a natural progression that many other metals have undergone in the past. Iron ore is a classic example. For several decades, iron ore was priced after extended 'annual mating' negotiations between the three largest iron ore producers and Japanese and European steel mills. That model of pricing has now been completely abandoned and, in its place, iron ore is sold based on price indices reported by Fastmarkets MB or Platts. The model has been in place for some years and it is quickly maturing to different price indices reflecting quality differentials. We believe that the spodumene industry pricing will eventually transition to a model similar to iron ore. The spodumene spot market and offtake contract market will eventually merge and operate on prices which reflect the prevailing market conditions. Similar to other metals, it will be China which will be the epicenter shaping spodumene prices.

20.3 Long Term Offtake Contracts Between Spodumene Producers and Converters

The spodumene industry is now undergoing a slow transformation in the way the material is priced between producers and converters. In the past, there were long term offtake agreements signed between producers and converters, or producers and traders, which would in turn supply to converters. These agreed prices were usually a good barometer for future prices for spodumene; however, in the recent year or so, long agreed contract prices have come under pressure. The latest agreements between Altura Mining and Chinese converters have prices linked to prevailing lithium carbonate prices with a pre-agreed price band between floor price: $550/t to ceiling price: $950/t. The floor price reflects the operating cost and some margins for the miner. In January 2019, Alliance Minerals Assets amended its lithium offtake agreements from a long term agreed price $880/t (FOB Australia, 6% Li_2O) in 2018 to a price indexed to lithium carbonate prices with a minimum floor price of $680/t and a ceiling price $1080/t. The prices for the current quarter are based on the average prices of the preceding quarter

Table 20-3: Recently Concluded Offtake Contracts and Prices (Source: Investor Presentations)

#	Producer	Customer	Volumes (kt)	Years	Specifications	Prices
1	Altura Mining, Pilgangoora, Aus	Shandong Ruifu, China	175	June 2019 – June 2024	Average 6.1% Li_2O, 1.04% Fe_2O_3, 0.63% Mica	Floor Price: $550/t Ceiling Price: $950/t
		Guangdong Weihua, China	25	August 2019- August 2024		Prices indexed to lithium carbonate and other process factors.
2	Sigma Lithium, Brazil	Mitsui Japan	330	2020-2024		Prevailing Market Prices
3	Core Lithium, Australia	Sichuan Yahua, China	75	Nov 2020- Nov 2023	Average 5.5% Li_2O	Prevailing Market Prices
		Shandong Ruifu, China	TBC	TBC	Average 5.5% Li_2O	Prevailing Market Prices
4	Alliance Mineral Assets, Australia	Jiangxi Bao Jiang Lithium Industrial, China	100 (min.80) 140(min.100)	2019 2020-2022	Average 6% Li_2O	Floor Price: $ 680/t Ceiling Price: $1080/t Prices for current quarter = average of preceding quarter If not agreed, higher of $880/t or floating price applies

20.4 Forecast Prices by Analysts

The table below assembles the available information from a number of analysts and producers who have forecasted long term prices of spodumene. Most analysts do not clearly describe the point of pricing but it can be assumed this refers to FOB Australia prices given that bulk of spodumene produced globally is exported from Australia. There is a wide variation in outlook of spodumene price forecasts among the analysts. The rapid decline in prices from $900/t in early 2018 to $580/t in August 2019 could explain why the forecasts may still be lagging the market conditions; however, there is some agreement among analysts that the high prices of spodumene witnessed by the market between 2016 and H1 2018 is an aberration and that future demand supply balance outlook will not be able to support such high prices. In principle Hatch agrees with the premise of this argument. In the long term, cost pressures on electrical vehicle manufacturing and battery manufacturing, alongside removal of government subsidies, could cascade up the supply chain to put pressure on spodumene prices.

Table 20-4: Forecast Prices Spodumene by Analysts (6% Li_2O)
(Source: Various analyst reports, Core Lithium Investor Presentations, in USD)

#	Description	2019	2020	2021	2022	2023	LT
1	Analyst 1	660	708	685	636	612	550
2	Analyst 2	831	733	660	562	589	700
3	Analyst 3	644	569	508	534	545	n/a
4	Core Lithium, BMI	750	700	720	800	675	650

20.5 Price Forecasts for Spodumene

20.5.1 Methodology

The methodology described below takes a departure from making informed and valid conclusions from offtake contract prices and analyst forecasts. The offtake prices and analyst prices are used to guide the final forecast numbers, and to validate if they are comparable or widely divergent. The following steps were undertaken in developing the spodumene price forecasts in this report.

- Begin with forecast lithium carbonate (battery grade) prices in China;
- Deduct typical converter margins;
- Calculate lithium carbonate production costs;
- Deduct typical converter conversion costs;
- Calculate spodumene costs;
- Divide spodumene costs by 7.75[1] to give guidance to spodumene prices in CIF China USD/t;
- Adjust guidance prices with lithium demand supply balance factor;
- Deduct freight rates to calculate FOB Canada prices.

20.5.2 Spodumene Prices

Table 20-5: Hatch Forecast Prices (real) of Spodumene Concentrate (6% Li_2O) FOB Canada
(Source: Hatch Estimates, USD)

	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Base Case	609	545	564	558	544	713	712	730	736	718	718	675	675	675	675	675	675
High Case	670	599	620	641	740	820	818	839	847	861	861	810	810	810	810	810	810
Low Case	579	518	535	530	579	642	640	657	663	646	646	608	608	608	608	608	608

[1] 100 kt spodumene (6%) contains 6,000 t of Li_2O which can yield 12,910 t of recovered lithium carbonate. Therefore, 1 t of lithium carbonate required 7.72 t of spodumene

The prices shown in the table show the prices computed from the methodology described above. Based on the demand supply balance of the lithium we expect prices to respond to excess supply between 2020 and 2022 before it starts increasing. As the market moves into an expected deficit from 2024, prices will improve to $713/t in 2024, further improving to $736/t by 2027. After 2027 we expect the improved pricing will incentive more supply into the market which will then soften the prices to a longer-term average of $675/t.

21. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

21.1 Environmental and Social Studies

Environmental baseline studies were conducted in 2012 by Dessau. In 2017 and 2018, a literature review was completed followed by site surveys mainly by Richelieu Hydrogéologie and SNC-Lavalin. Results will be presented in the following sections:

- Physical Environment in Section 21.1.1;
- Biological Environment in Section 21.1.2;
- Social Environment in Section 21.1.3.

21.1.1 Physical Environment

21.1.1.1 Topography

The topography of Authier Property is relatively flat. The average elevation is 350 m, varying from 320 to 390 m. The most elevated area is located in the northeast and consists of small hills due to the presence of an esker. The topography is presented in Figure 21-1. The elevation at the planned mine site operation varies between 334 m and 328 m above sea level.

On a regional scale, the crest of the Esker of St-Mathieu-Berry overhangs the surrounding ground by approximately 50 to 60 m, with a general down slope in a north direction except for its southern extension, just north of the mining property, which has a down slope in a south, south-west and south-east direction (Figure 21-2).

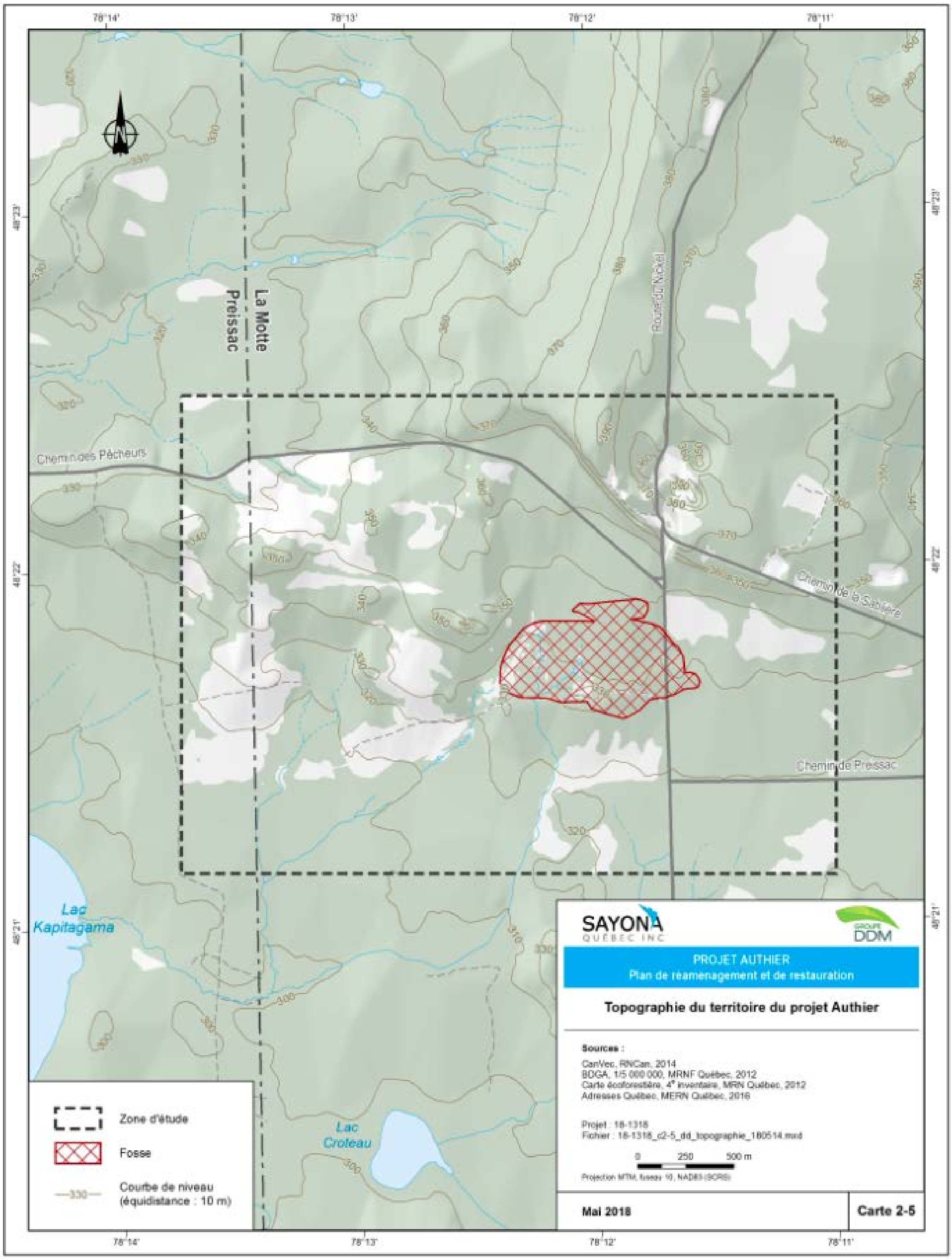


Figure 21-1: Topography of the Project Area

21.1.1.2 Local Bedrock Geology

The local geology is well described previously in this report. For the environmental considerations, it is emphasized here that local intrusive rocks are oriented east-west, suggesting a trend of local structural weaknesses oriented east-west. These intrusive rocks correspond to numerous small pegmatites, which intruded into the mafic and ultramafic rock and are generally composed of quartz monzonite including the larger spodumene-bearing pegmatite.

The pegmatites are described, in the boreholes database and in the literature, as medium to coarse-grained rocks.

Within the property limits, fractures were observed in only three boreholes over the 53 boreholes documented in the borehole database while no major faults were identified. On a preliminary basis, the bedrock is considered relatively unfractured.

Based on 75 boreholes, drilled and surveyed in the property limits, the bedrock elevation varies between 308 and 353 m for an average of 324 m.

21.1.1.3 Local Overburden Geology

The surficial geology is shown in Figure 21-2. In the project area, the three main geological features are small and large bedrock outcrops, the Esker of St-Mathieu-Berry and glacial lacustrine sediments.

Clayey material (clay, silty clay, silt) originating from the Lake Barlow-Ojibway is present inside a radius of 10 km, south of the property limits. These clayey materials were deposited in areas where the elevation (bedrock or till) was under 320 m, directly over the bedrock or over a basal till covering the bedrock. Pockets of organic materials are also encountered, and organic top soils are thin.

Outcrops represent approximately 5% of the area. However, over this the bedrock is only covered by a thin layer of soil in one third of the Northern claims.

Scientific papers, sponsored by the Geological Society of Canada (Bolduc et al., 2004[1]), describe the geological and hydrogeological settings of the Esker of St-Mathieu-Berry. It is made up of glaciofluvial sand and gravel with a core of gravel and pebbles, deposited directly over the bedrock. It has a cross-section form of a bell and of a longitudinal crest extending over 25 km on a south-north orientation, with its southern limit starting in the north-east corner of the

[1] Bolduc, A.M., Riverin, M.-N., Fallara, F., Paradis, S.J. et Lefebvre, R. 2004. Modélisation 3D du segment sud de l'esker de Saint-Mathieu – Berry près d'Amos, Abitibi. Congrès quadriennal de l'AQQUA, Québec 2004.

property. The topography slightly follows the bedrock topography with thickness being relatively constant. On a south to north direction, the crest and bedrock topography rise over about 1 km, reaching a sort of plateau over about 4 km, then sloping down over about 8 km, and finally slightly undulating northward.

The crest of the Esker of St-Mathieu-Berry overhangs the surrounding ground by 20 m to 30 m. Sand and gravel pits are exploited both in the northern and in the southern portions of the esker. The glacial lacustrine sediments, deposited during the glacial regression, cover the shoulders of the esker in the northern portion, the basal till and/or the bedrock. They are composed of sand, silty sand, gravel, and eventually boulders. They have not been compacted and contain some clay.

The thick basal till, observed in the south-west corner of the property, is described as continuous with an average thickness over 1 m and a content of less than 30% of fine particles (silt and clay).

A total of ten water wells are located in a radius of 5 km from the center of the ultimate pit and listed in the Québec Hydrogeological Information System (SIH, Système d'information hydrogéologique). The closest well is located at 3 km from the ultimate pit. The overburden thickness varies regionally (radius of 5 km) with an average of 8.8 m (Richelieu Hydrogéologie, 2018[2]).

[2] Richelieu Hydrogéologie Inc., 2018. Projet de lithium Authier de Sayona Québec Étude hydrogéologique et évaluation des effets du projet sur l'environnement.

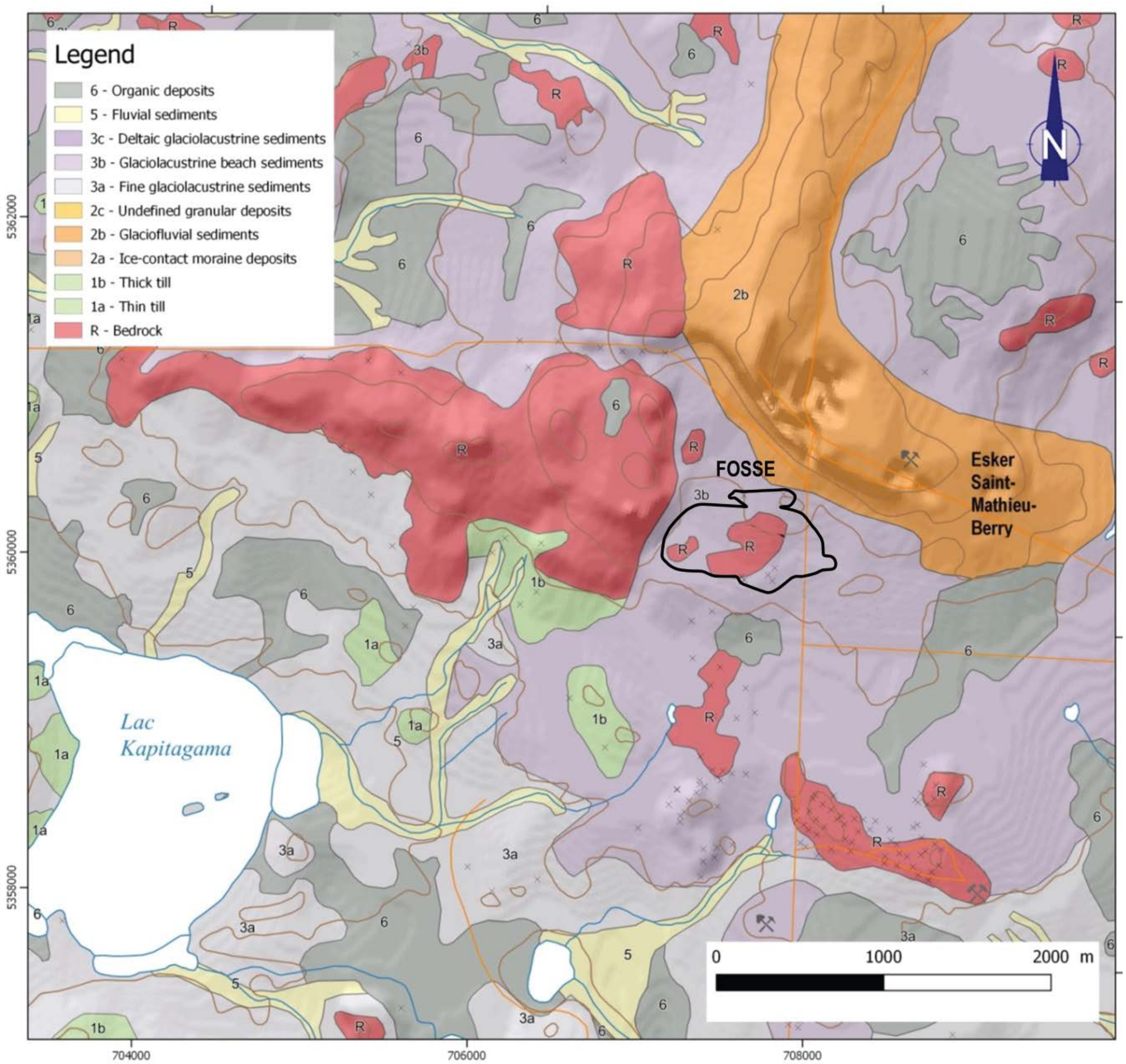


Figure 21-2: Local Overburden Geology

21.1.1.4 Hydrology

Watersheds

The Authier project is close to the water division of two important watersheds that divide the province of Québec: the Harricana River which reports to James Bay, and the Kinojevis River which reports to the St-Lawrence River. The Authier project is in the Kinojevis watershed. The Authier Property is on two sub-watersheds: Kapitagama Lake watershed and Croteau Lake watershed. There are no significant bodies of water or streams close to the future mine site, other than small streams and ponds. A detailed characterization of bodies of water and streams was completed in October 2017 and May 2018. The surface area of the sub-watersheds has been calculated and is presented in Table 21-1 and shown on Figure 21-3.

Table 21-1: Surface Areas and Map of the Sub-watersheds

Watersheds ID	Area
102	46 ha of 763.9 ha
104	109 m^2 of 544.7 ha
Effluent (in 104)	30 ha of 101 ha (effluent) of 544.7 ha (Watershed 104)
105	77.85 ha of 2,361.8 ha

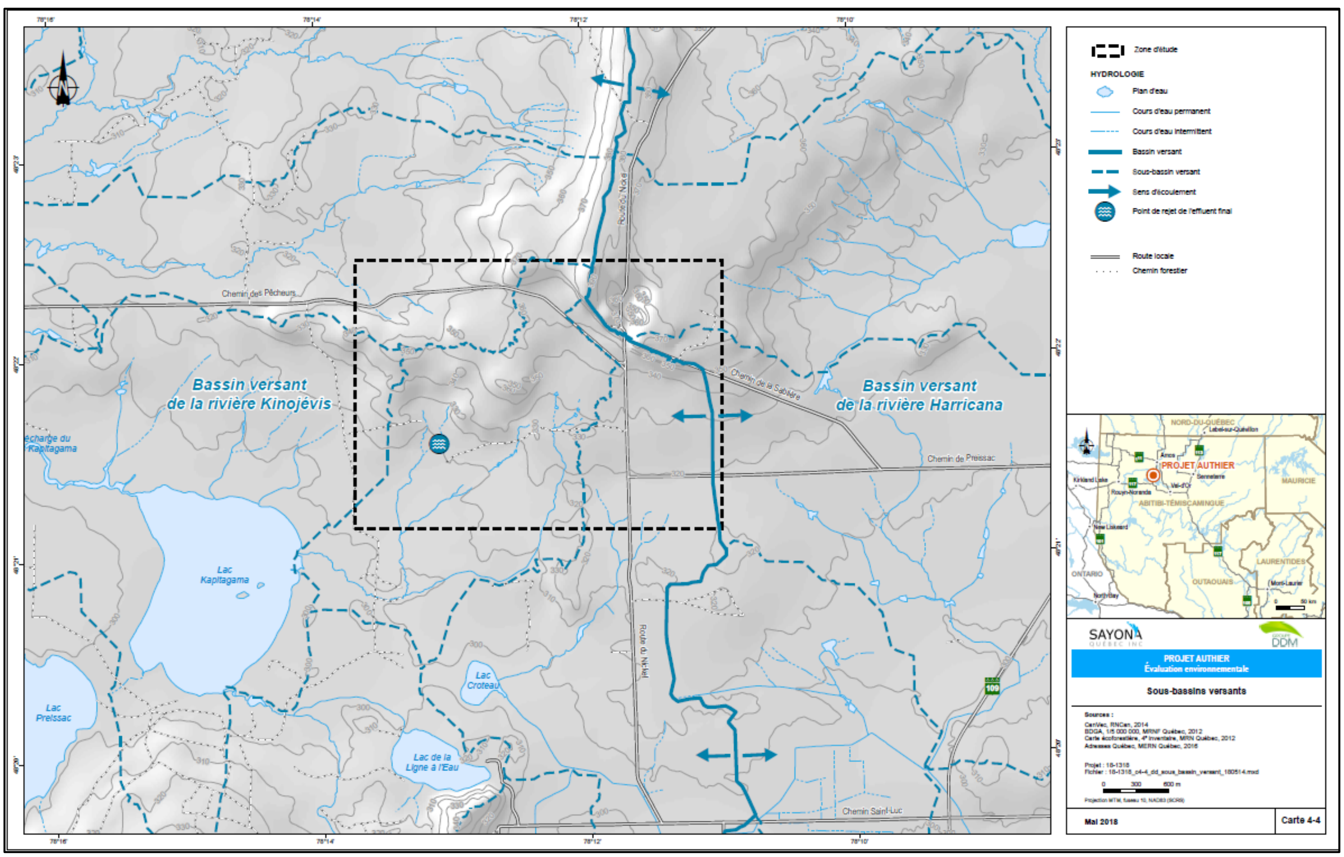


Figure 21-3: Sub-watersheds

Rainfall IDF Curves

The rainfall intensity was obtained from the reconstituted intensity-duration-frequency (IDF) curves for the Authier project site. Table 21-2 shows the rainfall IDF curves for the Authier project.

Table 21-2: Rainfall IDF Curves for the Authier Project

Duration	Return Period (year)						
	2	5	10	25	50	100	1,000
	Intensity [mm]						
5 min	6.4	9.3	11.2	13.8	15.6	17.5	23.7
10 min	9.7	14.5	17.6	21.7	24.7	27.7	37.6
15 min	11.6	17.5	21.4	26.4	30.2	33.9	46.2
30 min	15.0	24.3	30.5	38.5	44.4	50.3	69.9
1 h	18.3	28.9	36.4	45.9	53.0	60.1	83.6
2 h	24.2	35.6	43.3	52.9	60.2	67.3	91.1
6 h	34.1	49.9	60.4	73.7	83.7	93.5	126.3
12 h	40.0	56.0	66.7	80.3	90.4	100.4	133.6
24 h	45.3	65.1	78.4	95.2	107.7	120.1	161.3

Flood Flows

Flood flows at the final effluent discharge point of the Authier project have been calculated by the rational method and are presented in the Table 21-3.

Table 21-3: Flood Flows at the Final Effluent Discharge Point of the Authier Project

Return Period [years]	Flood Flow [m³/s]
	Rational Method
2	0.8
5	1.4
10	1.8
20	2.7
50	3.5
100	4.2

Low Flow Rates

The specific low flowrates calculated by the Centre d'Expertise Hydrique du Québec (CEHQ, 2014) for the Harricana River station in Amos are used for the project and are presented in Table 21-4. The low water flow is then estimated by multiplying the specific flows by the area of the watershed under study.

Table 21-4: Specific Low Flow Rates - Harricana River in Amos (3,724 km²)

Period	Type of low flow	Return Period [year]	Duration [day]	Flow [m³/s]	Specific Flow [l/s/km²]
Summer	Q2,7	2	7	16.51	4.43
Summer	Q10,7	10	7	12.88	3.46
Summer	Q5,30	5	30	15.22	4.09
Annual	Q2,7	2	7	25.66	6.89
Annual	Q10,7	10	7	15.72	4.22
Annual	Q5,30	5	30	20.96	5.63

The application of the basin transfer method, with the specific flows calculated by the CEHQ for the Harricana River in Amos (Table 21-4), makes it possible to obtain the low water flow values for the final effluent point. These values are presented in Table 21-5.

Table 21-5: Low Water Flows - Final Effluent Discharge Point of the Authier Project

Period	Type of low flow	Return Period [year]	Duration [day]	Flow [l/s]	Specific Flow [l/s/km²]
Summer	Q2,7	2	7	4.5	4.43
Summer	Q10,7	10	7	3.5	3.46
Summer	Q5,30	5	30	4.1	4.09
Annual	Q2,7	2	7	7.0	6.89
Annual	Q10,7	10	7	4.3	4.22
Annual	Q5,30	5	30	5.7	5.63

According to Table 21-5, it is found that flowrates are low. However, given the very large difference in area between the Harricana River watershed at Amos (3,724 km²) and the final effluent discharge point (1 km²), it is to be expected that results are overestimated. During summer and winter low flows, it is likely that the base flow of the small stream at the final effluent discharge point will dry out and be zero.

21.1.1.5 Hydrogeology

This section is based on the report prepared by Richelieu Hydrogéologie inc., 2018[3].

Hydrogeological Study Works

A hydrogeological study was conducted by Richelieu Hydrogéologie inc. in order to describe the hydrogeological context of the Authier property and to evaluate the effects of the project to the environment. The study started in December 2016 and currently includes the installation of 27 observation wells (piezometers), groundwater sampling campaigns, the achievement of variable head permeability tests and tracer profile testing as well as groundwater level surveys. From data collected during the surveys and data coming from several public information (climate, geology, etc.), the actual groundwater flow conditions of the Authier property were simulated.

Table 21-6 presents the coordinates and results obtained from the piezometers. Figure 21-4 shows the location of the piezometers.

[3] Richelieu Hydrogéologie Inc., 2018. Projet de lithium Authier de Sayona Québec Étude hydrogéologique et évaluation des effets du projet sur l'environnement

Table 21-6: Coordinates, Depth, and Hydraulic Conductivity (from Richelieu Hydrogéologie, 2018[3])

Piezometer ID	UTM NAD83 Zone 17 East	UTM NAD83 Zone 17 North	Depth (m)	Hydraulic Conductivity (m/s)
PZ-01MT	707350.0	5360949.0	11.58	4.94×10^{-05}
PZ-01R	707350.0	5360949.0	18.53	5.51×10^{-05}
PZ-02MT	707620	5360981	18.56	1.95×10^{-07}
PZ-02R	707623	5360986	23.50	3.81×10^{-08}
PZ-03MT	707423	5361205	26.79	1.88×10^{-06}
PZ-04R	706038.3	5360902.1	9.14	5.40×10^{-07}
PZ-05R	706100.5	5361328.2	8.23	6.59×10^{-06}
PZ-06R	706445.2	5361170.2	8.23	1.87×10^{-06}
PZ-07MT	707038.9	5361179.7	8.66	7.36×10^{-06}
PZ-07R	707038.9	5361179.7	16.06	1.69×10^{-06}
PZ-08MT	705942	5359746	6.21	6.53×10^{-06}
PZ-08R	705941	5359737	10.41	6.20×10^{-07}
PZ-09MT	706539	5359705	7.59	
PZ-09R	706537	5359709	13.22	
PZ-10MT	707056	5360382	15.17	1.47×10^{-06}
PZ-10R	707053	5360385	19.05	1.80×10^{-08}
PZ-11MT	707413	5360221	5.27	4.77×10^{-06}
PZ-11R	707409	5360227	14.38	2.27×10^{-08}
PZ-12MT	707668	5360177	3.81	2.26×10^{-06}
PZ-12R	707668	5360177	13.19	2.27×10^{-08}
PZ-13MT	706882	5360437	6.06	2.91×10^{-08}
PZ-13R	706885	5360430	10.75	7.31×10^{-07}
PZ-14R	706750	5360460	6.99	3.59×10^{-06}
PZ-15R	707780	5359751	20.66	1.46×10^{-07}
PZ-16R	706600	5360440	14.97	4.04×10^{-07}
PZ-17MT	708817	5360600	18.02	3.06×10^{-07}
PZ-18MT	708530	5360724	18.20	

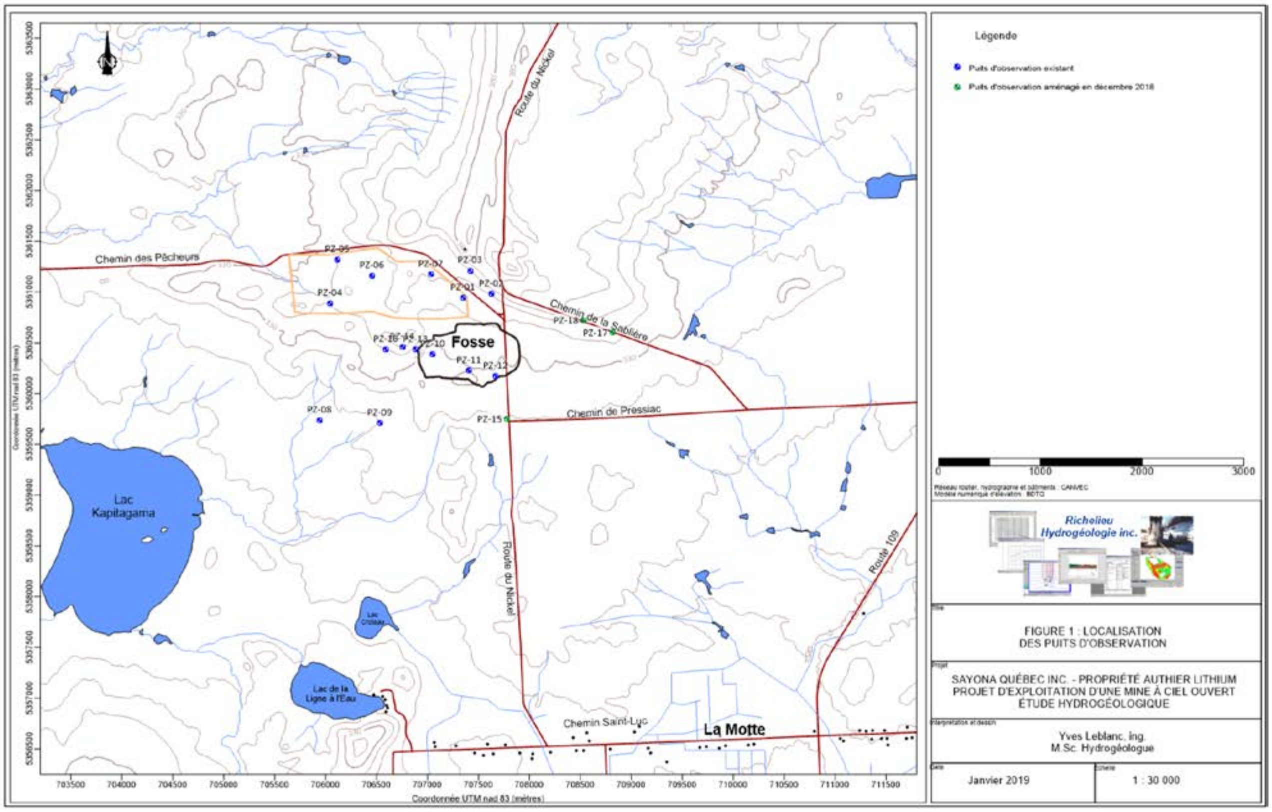


Figure 21-4: Location of the Piezometers (from Richelieu Hydrogéologie, 2018[3])

Hydrostratigraphic Units

The hydrostratigraphic units identified at the Authier property are the following:

- Bedrock, a regional aquifer of a standard to low permeability;
- Glacial till, an aquitard discontinuously covering the bedrock;
- Fluvio-glacial sand and gravel (esker), a highly permeable aquifer, covering the till;
- Glacio-lacustrine sand (aquifer) and silt (aquitard), covering the till unit and partly the fluvioglacial unit; and
- Organic layer, a thin and discontinuous aquitard.

The most permeable part of the bedrock is located in its upper 150 m, after which it becomes impermeable. Its mean hydraulic conductivity is in the order of magnitude of 10^{-7} m/sec. The unconsolidated sediments unit has a mean measured hydraulic conductivity to the order of magnitude of 10^{-6} m/sec.

Groundwater Flow Direction

Following the water level surveys that were done for all piezometers installed on the site property, the following observations could be made:

- The groundwater level in the area of the property is of the order of 329 m;
- On the entire site, it varies from 305 m in the south-west part, to 345 m in its northern part, which gives a general direction of flow towards the southwest under a horizontal hydraulic gradient of 0.02;
- The calculated vertical hydraulic gradients indicate in some well nests an upward flow direction (low) and, for other well nests, a downward flow direction (low). The distribution of these hydraulic gradients is randomly observed and, in general, the vertical gradient is low with average of 0.07.

Figure 21-5 shows the baseline conditions of groundwater flow direction in the project area. On this figure, it is possible to observe that the regional flow is similar to the topography of the land. The piezometric surface conforms to the topographic surface. Groundwater level is found at a mean depth of 1.8 m below ground surface, except for the esker where groundwater level is found at more than 9 m deep. The groundwater flow is in the SW direction through the property.

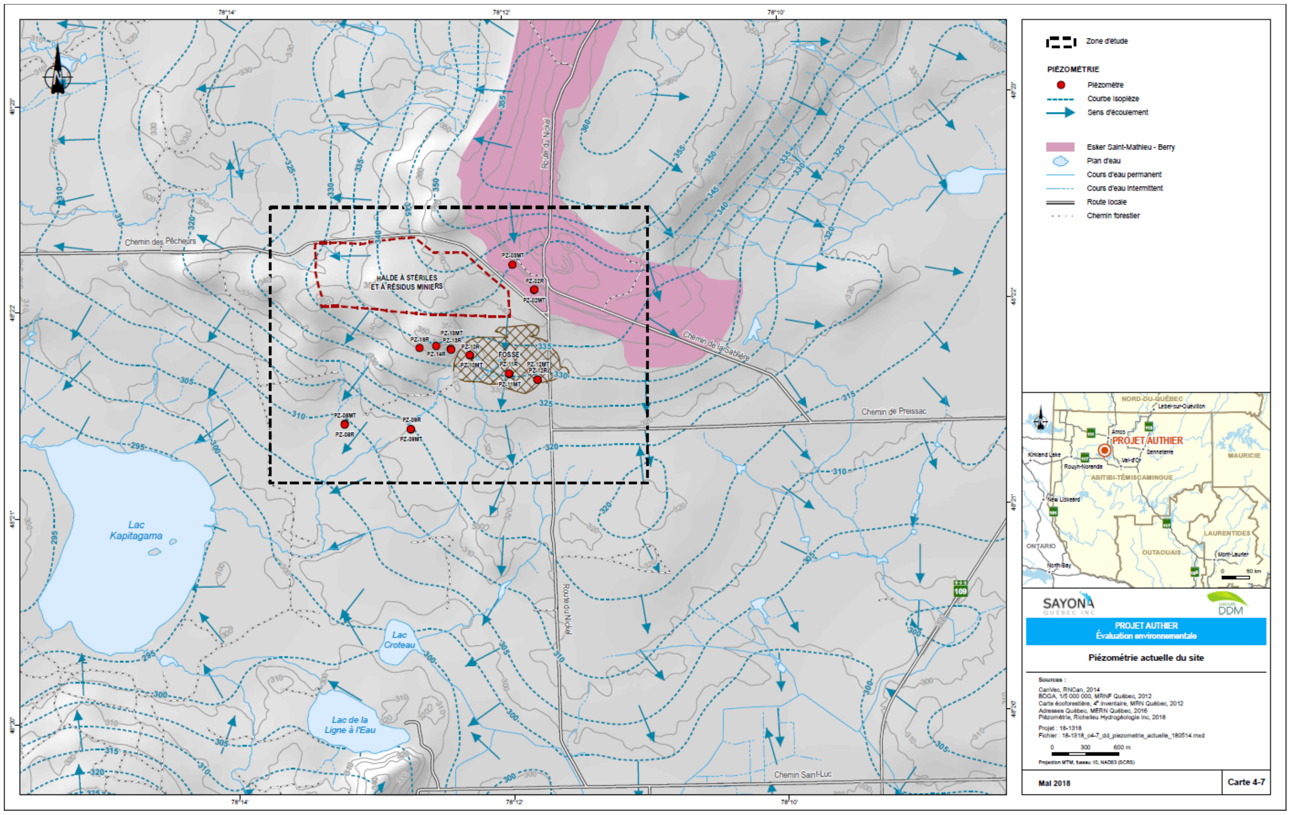


Figure 21-5: Baseline Groundwater Flow Direction (Sayona Québec, May 2018[4])

Groundwater Recharge, Vulnerability and Quality

Groundwater recharge is achieved into the esker and bedrock outcrops, while it discharges into local brooks and wetlands. The recharge has been evaluated from 0 mm to 420 mm/year, as a function of the sol texture and ground slope.

The DRASTIC index vulnerability was estimated between 60 and 200 at the Authier property, which represents low to high vulnerability to surface activities.

An inventory of the groundwater users revealed that the closest well is at a distance of 3.2 km from the projected open pit. Residential wells are located on the road Ligne à l'Eau and St-Luc.

[4] Sayona Québec inc., 2018. Évaluation environnementale Projet Authier La Motte, Québec, Canada. Document for public consultation beginning on May 18, 2018.

The mean depth of these wells is 75.8 m. The groundwater types are calcium-bicarbonate and sodium-potassium. The pH is neutral to alkaline and the mineral charge is quite low (mean electrical conductivity of 148 µS/cm). There is no difference between the groundwater quality from bedrock and from the unconsolidated deposits.

Anticipated Effects of the Projects on the Groundwater

When the open pit will reach its maximum depth (200 m), the area of influence would get 1,000 m to 2,100 m. The filling of the open pit will be achieved in about 25 years by both surface and groundwater inflow.

During the mine life, the groundwater flow from beneath the tailings and slag will be directed towards the pit, then at natural flow, it will be directed towards south-west direction.

The effects of mine dewatering on residential wells are negligible since those wells are located at more than twice the radius of influence of the open pit. Their groundwater quality should neither be affected since the open pits will act like a drain, intercepting all possibly contaminated groundwater.

The effect of the project on the environment would be, in the worst-case scenario, a reduced groundwater outflow to the local surface water network and to the wetlands. A reduced flow of brooks or drying of wetlands could then occur into the area of influence (about 6 km^2).

The southern part of the St-Mathieu-Berry esker is enclosed into the area of influence of the mine. This area would have a local drawdown of 0.5 m to 6.5 m in a surface of about 1.9 km^2. This part of esker is not connected to the main part of the esker which is being tapped by the drinking facilities of the city of Amos and also by the Eska water bottling society. Both portions of the esker are separated by a bedrock lump.

In the esker, the groundwater generally flows towards the north, except in the project area where it is heading south and south-east and to the Harricana River watershed. The southern portion of the esker, located in the project area, is located in a different watershed than the remainder of the esker. However, it is recommended to avoid any construction in the north as the esker is a source of drinkable and commercial water for Amos as well as others. The esker is a concern for the stakeholders. However, because it is located at a lower altitude than the esker and isolated from it by a bedrock, the Authier project will not threaten, in any way and under any circumstances, the water quality of this esker.

21.1.1.6 Water Quality

Underground Water Quality

During the sampling campaigns (2017-2019), 14 - 27 wells were sampled (refer to Figure 21-4). Samples collected were analyzed for a variety of parameters including metals, nutrients, major anions and cations, volatile compounds, polycyclic aromatic hydrocarbons and C_{10}-C_{50} petroleum hydrocarbons. Analytical results show that:

- Underground waters are lightly loaded in minerals. Electric conductivity varies from 34 to 259 µS/cm;
- Underground water pH varies from neutral to alkaline;
- Comparing to drinking water standards, a few excesses are observed for mercury (n = 4) and nickel (n = 2). Esthetical water standards are exceeding for iron (n = 9), manganese (n = 11) and sulfur (n = 6). Surface water standards is exceeded for copper (n = 1), mercury (n = 4) and aluminium (n = 1);
- C_{10}-C_{50} petroleum hydrocarbons, MAH and PAH are generally below limit of detection.

In order to follow up the groundwater in conformity to the government's directive 019 on the mining industry, it is recommended to proceed with the bi-annual groundwater quality and groundwater level survey started in 2017. Those surveys should last for the entire mine-life and 5 years after the mining activities ceases.

Surface Water Quality

Surface water has been sampled during the fall 2017 and spring 2018 campaigns as part of the fish habitat study in the streams that are likely to be impacted by the project. This study aim to verify the presence of fish in the targeted streams, to identify all streams located within the core study area, and to determine the surface water quality in the targeted streams.

Sampling of the surface water was conducted in five locations, i.e. four stations in the core study area and one outside the extended study area, along the main stream draining the core study area. The locations are shown in Figure 21-6. Samples were taken according to standard methods for both preparation, and handling and storage.

The analysis of surface water samples included pH and alkalinity, SS and turbidity, nutrients (phosphorus and nitrogen), dissolved oxygen, major ions and conductivity, metals, cyanides and coliforms.

The results were compared to the provincial criteria for the protection of aquatic life for chronic toxicity established by the “Ministère de l’Environnement et de la Lutte Contre les Changements Climatiques” (MELCC) and the Canadian aquatic life protection guidelines for long term exposure of the “Canadian Council of Ministers of the Environment” (CCME). Laboratory analysis revealed that the water sampled is acid, with low to medium alkalinity, relatively limpid (except at two stations), and slightly mineralized. Exceedances of criteria for water quality were observed at least at one station for the following parameters: total phosphorus, dissolved oxygen, aluminum, iron, manganese, and nickel.

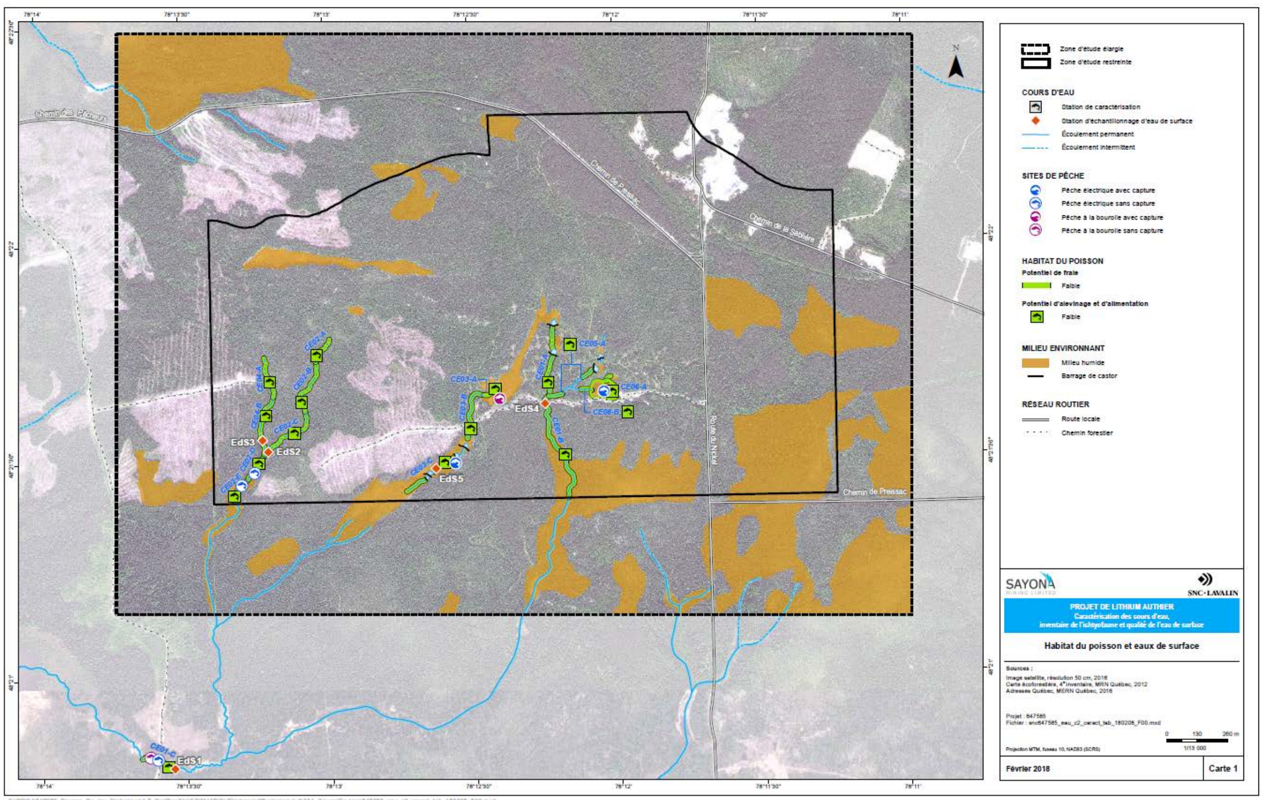


Figure 21-6: Surface Water Sampling Locations (from SNC-Lavalin, 2018a[5])

5 SNC-LAVALIN, 2018a. Caractérisation des cours d’eau, inventaire ichtyologique et qualité des eaux de surface. Projet Authier. Rapport présenté à Sayona Québec.

21.1.2 Biological Environment

21.1.2.1 Vegetation and Wetlands

A field visit was carried out in August 2012. Although the entire area was not visited, the field visit validated the information from the forestry map study area, which is dominated by a mix of coniferous trees. Also, the study carried out by SNC-Lavalin in 2017 and completed in February 2018 was used to confirm by photo-interpretation the forest stand.

The relevant forest is made up of old and mature forest stands. Both categories were identified on forestry maps provided by the MFFP. Old forests in the area are estimated to be 90 years old, while the mature forests are approximately 70 years of age. On the Authier property, at least 52% (a total of 483 hectares: 480 hectares of mature forest and 3 hectares old forest) of the area is covered by relevant forest. It is also important to note that 85.3 ha of the study area is totally or partially cut.

The wetlands were characterized on the 30 August, 1 September and 8 November 2017 as well as on the 22 and 23 June 2018. A final characterization will be completed in 2019 (1 to 8 August). A total of 44 wetlands were characterized in 2017 and 2018 and at least 14 more will be visited in 2019. Wetlands present in the study area cover a total of 599 ha. Bogs and swamps are the main wetland classes characterized during the field surveys. Only a few bogs were located near the project area. These bogs did not reveal any major particularities.

Terrestrial Vegetation

The terrestrial vegetation (675 ha) consists mainly of mixed (295.4 ha) and coniferous (273.5 ha) forest stands. Hardwood stands are scarce (2.2 ha). Together, forest areas cover more than 80% of the study area. It should be noted that 85.3 ha of the study area have been totally or partially cut.

Stands of fir and white spruce, mixed with white birch dominate the forest landscape of the site. Other sites are occupied by black spruce, jack pine and larch, often in the company of white birch or trembling aspen. Due to the abundance of balsam fir, the spruce budworm is the main factor in forest dynamics, although fire is also important. Table 21-7 presents a breakdown of the surface area of terrestrial vegetation on the project site.

Table 21-7: Surface Area of Terrestrial Vegetation on the Project Site

Terrestrial Vegetation	Surface Area (ha)	Surface Area in the Project Site (%)
Forested Environments		
Medium hardwood (30 to 70 years)	2.2	0.3
Mixed young	138.2	16.4
Medium young (30 to 70 years)	157.2	18.7
Softwood young	123.7	14.7
Medium Softwood (30 to 70 years)	149.8	17.8
Regeneration	18.3	2.2
All Forested Environments	**589.4**	**70.1**
Disturbed Forested Environments		
Total Cut	67.9	8.1
Cut by Bands	17.4	2.1
All Disturbed Forested Environments	**85.3**	**10.2**
Total of all Forested Environments	**674.7**	**80.3**

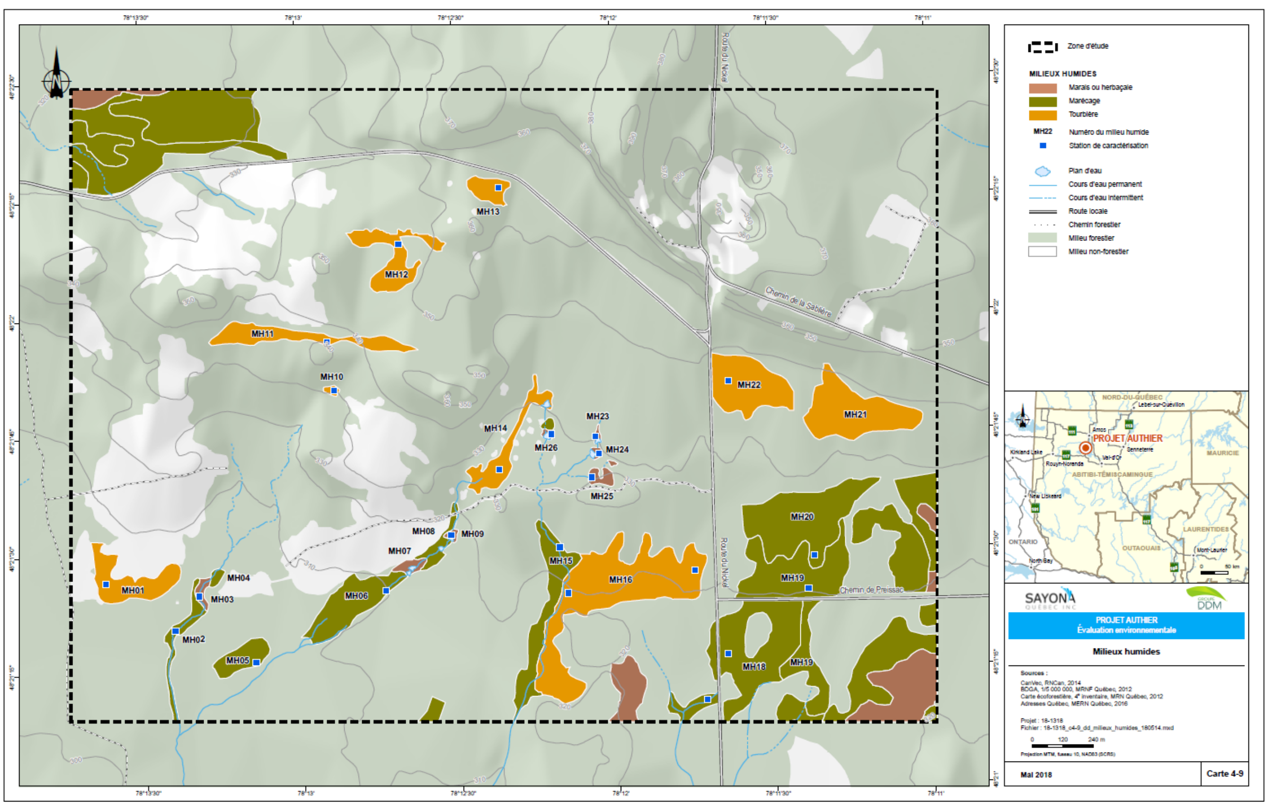


Figure 21-7: Wetlands Type and Location (from SNC-Lavalin, 2018b[6])

21.1.2.2 Terrestrial and Avian Fauna

In 2012, information regarding the fauna species present on the property was obtained from four different databases: the CDPNQ (Centre de Données sur le Patrimoine Naturel du Québec), the AARQ (Association des Aménagistes Régionaux du Québec), the AONQ (Atlas des Oiseaux Nicheurs du Québec) and the ÉPOQ (Étude des populations d'oiseaux du Québec). Data from inventories carried out by the MFFP (Ministère de la Faune, des Forêts et des Parcs) were also used. The information collected from all these sources was essential in preparing the fieldwork.

SNC-Lavalin carried out a field inventory for snakes, salamanders and anurans in the summer of 2017 and in the spring of 2018. Bird surveys were conducted by SNC-Lavalin in 2017 and by Groupe DDM in 2019. A bat inventory was also completed by SNC-Lavalin in 2017 (27 June to 17 July). The inventory period included the calving and lactation periods for Québec bat populations. Finally, a small mammal and rodent inventory was conducted from the 24 to the 28 August 2017 by SNC-Lavalin.

During the fieldwork for reptiles and amphibians, one species of salamander, five species of anurans and one species of reptiles were observed. All the species sampled are considered fairly common in Québec. No species at risk were observed.

A total of 27 small mammals belonging to 4 species were caught at the trapping stations. The Masked Shrew was the main species with 21 individuals captured. No species of special concern was caught.

The bat inventory confirmed the presence of 4 species of chiropterans. All four species are listed as present in the region of Abitibi-Témiscamingue. Three of the 4 species observed are at risk and are described hereafter.

A total of 66 bird species were observed during the inventories. Nesting was confirmed for 2 species (Sharp-tailed Grouse and Cedar Waxwing). Species at risk observed are described hereafter.

21.1.2.3 Fish and Fish Habitat

In 2012, visual characterizations were made for five ponds and one stream in the Preissac Lake watershed. Results showed that each pond was populated by beavers and contained beaver dams. The substrate of each pond was made of sand and organic material. The footprint of the ponds varied from 503 m^2 to 10.120 m^2. A total of thirteen sticklebacks were found in two of the five ponds.

An ichthyological fauna inventory has been completed during the characterization of bodies of water and streams during summer 2017. The main objectives of this study were to characterize fish habitat in the streams that are likely to be impacted by the project, to verify the presence of fish in the targeted streams, to identify all streams located within the core study area, and to determine the surface water quality in the targeted streams.

Two biologists carried out the biophysical characterization of streams (potential fish habitat) and the fish survey on August 30 and September 1, 2017. Fieldwork was limited to streams located in the core study area, with the exception of one for which a characterization station was positioned outside the extended study area. Stream characterization was conducted via characterizations at spot locations, whereas the fish survey was performed wherever possible using electrofishing and bait trap. Fish habitat quality was then assessed for two habitat categories (spawning and nursery/foraging), based on the observer's judgement using habitat characteristics and fishing results.

Results indicated that spawning and nursery/foraging habitats are of low quality in streams of the core study area, due among other things to physicochemical conditions. Only one fish species was captured (i.e., brook stickleback).

21.1.2.4 Benthic Community

The benthic community of the different stations sampled in 2012 is mostly composed of nematodes, annelids, insect larvae and mollusks. Results are showing between 4 and 34 different species with a variation of the number following the sampling stations.

21.1.2.5 Endangered Wildlife

The CDPNQ, MFFP, and COSEWIC (Committee on the Status of Endangered Wildlife in Canada) databases were consulted to identify any endangered species potentially present on the property. It is important to mention that the absence of a species from a database or a field survey does not mean that the species is absent from the area of interest.

Three at risk bat species were observed in the study area. The Hoary and Silver-haired bats are likely to be designated threatened or vulnerable in Québec (MFFP, 2019). They have no status at the federal level. The Little brown bat is considered endangered and is listed in Appendix 1 of the Species at Risk Act in Canada.

Twelve avian species at risk are considered potential nesters in the study area (Table 21-8). This group includes species considered at risk in Canada (Government of Canada, 2019) and Québec (MFFP, 2019).

Table 21-8: Avian Species at Risk Considered as Frequenting or Likely to Frequent the Study Area During the Nesting Season (Adapted from Sayona Québec, August 2019)

Species	Presence in the study area	Status of the species	
		Québec[1]	Canada[2]
Haliaeetus leucocephalus	Nil	Vulnerable	Not at Risk
Falco peregrinus	Nil	Vulnerable	Special concern
Coturnicops noveboracensis	Nil	Vulnerable	Special concern
Asio flammeus	Nil	SDTV[3]	Special concern
Chordeiles minor	Observed	SDTV	Threatened
Antrostomus vociferus	Possible	SDTV	Threatened
Contopus cooperi	Observed	SDTV	Threatened
Contopus virens	Observed	None	Special concern
Hirundo rustica	Possible	None	Threatened
Cardellina canadensis	Possible	SDTV	Threatened
Dolichonyx oryzivorus	Possible	None	Threatened
Euphagus carolinus	Possible	SDTV	Special concern

(1) Ministère des Forêts, de la Faune et des Parcs (2019).

(2) Gouvernement du Canada (2019).

(3) Species likely to be designated threatened or vulnerable.

21.1.3 Social Environment

21.1.3.1 Population

The Authier project site is located in La Motte in the administrative region of Abitibi-Témiscamingue. The property is accessible by a rural road network (Preissac Road and Nickel Road) connecting to Route 109 located a few kilometers east of the site (approximately 5 km). Route 109 connects Rivière-Héva with Amos then Matagami; then joins Route 117 at Rivière-Héva. The project is located approximately 35 km south of the Abitibiwinni community of Pikogan.

The Abitibiwinni (Community of Pikogan) are the Algonquins of northern Abitibi. Today, Abitibiwinni is one of nine Algonquin communities in Québec. The community of residence of Abitibiwinni is known as Pikogan, a reserve established in 1956, 3 km north of the city of Amos. The reserve is bordered by the Harricana River and is enclosed in the city of Amos without being part of it or part of any Regional County Municipality (RCM) since it is a First Nation reserve.

The Authier mine project area is at the heart of the ancestral Abitibiwinni Aki territory, which the Abitibiwinni has never yielded. Community members continue to frequent this territory, including traditional hunting, fishing and picking activities. The community lives approximately 35 km north of the Authier mine project site and 3 km north of Amos, on the west bank of the Harricana River. Municipalities near the Authier project site include: La Motte, Saint-Mathieu d'Harricana, Rivière-Héva, Preissac, and Amos. Table 21-9 presents a comparative socio-economic profile between Pikogan and these municipalities.

Table 21-9: Comparative Socio-Economic Profile Between Pikogan, La Motte, Saint-Mathieu-d'Harricana, Rivière-Héva, Preissac, Berry, Amos and the Abitibi MRC (Adapted from Sayona Québec, May 2018)

Categories	Pikogan	La Motte	Saint-Mathieu-d'Harricana	Rivière-Héva	Preissac	Berry	Amos	MRC d'Abitibi
Population (2016)	538	453	735	1 420	835	538	12 823	24,639
Population (2011)	538	457	696	1 433	786	625	12 671	24,354
Variation of population 2011-2016 (%)	0.0	-0.9	6.2	-1.0	6.2	-13.9	1.2	1.2
Men	250	260	380	735	450	290	6 300	12,380
Women	285	200	355	680	390	245	6 525	12,260
Area km^2	1	176.9	106.71	426.17	428.26	577.33	430.29	7,679.36
Density of population	539.7	2.6	6.9	3.3	1.9	0.9	29.8	3.2
Mean age	30.8	43.2	39.1	41.9	45.2	37.6	42.7	41.6
Spoken language at home	French (325), English (80), Native language (55)	French (450), English (0)	French (735), English (5)	French (1,390), English (20)	French (825), English (5)	French (535), English (0)	French (12,455), English (70), Native language (10), Others (40)	French (11,995), English (75), Native language (35)
Employment rate	41.7 %	50.7 %	72.4 %	66.9 %	58.9 %	62.8 %	60.6 %	59.9 %
Unemployment rate	16.7 %	12.5 %	3.4 %	6 %	9.4 %	15.4 %	7.4 %	8.5 %
Number of workers in the mining industry	10	40	25	250	80	20	340	845

21.1.3.2 Stakeholder Mapping

Stakeholder identification was completed in 2017 using a mapping of the study area and a series of interviews with community stakeholders. The project is located on the territory of the municipality of La Motte and on the territory recognized in the agreement signed between the Government of Québec and the Abitibiwinni First Nation[6]. Thus, these two communities were therefore targeted first for information and consultation meetings. The list of stakeholders was then completed by identifying the individuals or groups that could be directly or indirectly affected by the Authier project.

[6] https://francophonie.sqrc.gouv.qc.ca/VoirDocEntentes/AfficherDoc.asp?cleDoc=117107105120244139201191157180540762121062061 39

The main Community/Regional Stakeholders (non-exhaustive list) are as follows:

- Abitibiwinni First Nation;
- Municipality of La Motte;
- Municipality of Saint-Mathieu-d'Harricana;
- City of Amos;
- Municipality of Berry;
- Municipality of Rivière-Héva;
- Municipality of Preissac;
- Regional County Municipality of Abitibi;
- Société de l'eau souterraine d'Abitibi-Témiscamingue (SESAT);
- Groupe de recherche sur l'eau souterraine (GRES UQAT);
- Organisme de bassin versant du Témiscamingue (OBVT);
- Organisme de bassin versant Abitibi-Jamésie (OBVAJ);
- Eska.

A Community Relations Program has been developed to approach and engage local stakeholders. This program included information sessions and consultations with municipalities, land users, First Nation community, non-governmental environmental organizations and recreational associations. Consultation and community engagement efforts that have been deployed throughout the project development allowed Sayona to outline stakeholders' main preoccupations and expectations. The objective of this program is to provide baseline information to address some of the communities' concerns and take them into consideration in the permitting process and in the design of the operation phase. The involvement of stakeholders will continue throughout the various project stages. A public consultation process is currently underway and will be completed during fall 2018.

21.1.3.3 Land Uses

The proposed mine site is entirely located on a forestry sector of public tenure which is not regulated by agreement. The main authorized uses for this forested area are production and harvesting of trees, outdoor activities and agriculture.

In the project area, the activities found are as follows:

- Lumbering;
- Mining activities;
- Exploitation of eskers and moraines;

- Agricultural production;
- Recreational (trails, campsites, ski resorts, etc.) and residential activities (residences, motels, cottages);
- Ecological reserves; and
- Hunting, fishing and trapping activities.

21.2 Impacts and Mitigation Measures

The project will create temporary and permanent modifications to the mine site. During the environmental assessment process, project activities, that may directly or indirectly affect the environmental (physical and biological) and social (human) components, have been identified. These activities could be conducted during one or all of the 3 phases of the project: construction, operation and closure (and restoration).

During the construction phase, the activities that might impact the environments are:

- Site preparation (excavation, stripping, backfilling, blasting and management of the overburden);
- Construction of temporary and permanent infrastructure and facilities;
- Runoff water, drinking water and wastewater management;
- Management of hazardous and residual materials and fuels;
- The use and maintenance of heavy machinery and vehicles;
- Hiring and presence of workers; and
- The purchase of goods and services.

During the exploitation phase, the activities that might impact the environments are:

- Site preparation (excavation, stripping, backfilling, blasting and management of the overburden);
- Construction of temporary and permanent infrastructure and facilities;
- Runoff water, drinking water and wastewater management;
- Management of hazardous and residual materials and fuels;
- The use and maintenance of heavy machinery and vehicles;
- The presence of infrastructures and buildings;
- Hiring and presence of workers;
- The purchase of goods and services;
- Mining, storage and processing of ore; and
- Restoration of the site and revegetation of the waste rock and tailings pile.

During the closure and restoration phase, the activities that might impact the environments are:

- Runoff water, drinking water and wastewater management;
- Management of hazardous and residual materials and fuels;
- The use and maintenance of heavy machinery and vehicles;
- The presence of infrastructures and buildings;
- Hiring and presence of workers;
- The purchase of goods and services;
- Restoration of the site and revegetation of the waste rock and tailings pile;
- Stopping the purchase of goods and services;
- Layoff of workers; and
- The remains of the site.

21.2.1 Impacts and Mitigation Measures on the Physical Environment

21.2.1.1 Air Quality

Air emission modeling will be conducted during 2018-2019 and Sayona Québec will put in place a dust management plan to limit most of the possible nuisance.

The following activities will be included in the future mitigation measures that will be implemented:

- During the summertime, water is used to control dust on mining site roads and on Preissac road;
- Establishing a procedure to ensure that the blasting activities are adequately controlled;
- Progressive cover of the filtered tailings with waste rocks in the waste rock pile and tailing storage facility;
- Asphalt overlay of the last 300 meters of Preissac road (before the intersection with Highway 109).

In general, Sayona Québec will implement a complaint management protocol to allow citizens to express their concerns if the mining activities generate dissatisfaction.

21.2.1.2 Noise

Mining activities will generate soundscape modifications. Given the size and remoteness of the project, the soundscape should not be altered and the citizens should remain unbothered. However, the soundscape will be locally altered and may disturb some territory users. A noise modelling is currently conducted and results will be available in the fall 2018. Depending on the results obtained, Sayona Québec would put in place the following mitigation measures:

- Blasting activities are prohibited during evenings, weekends and at night;
- In-pit drilling activities will be done mainly during the day;
- The primary crusher (jaw crusher), secondary crushers and sieves will be placed in a building therefore limiting the noise;
- The trucks will have a white noise backup alarm (multi-frequency sound);
- Heavy machinery and vehicles will have functional and efficient mufflers;
- Noise control devices will be installed on pneumatic and hydraulic hammers.

21.2.1.3 Soils

On-site activities may affect soil quality. Sayona Québec will implement the following mitigation measures to limit the unwanted effects:

- A procedure in the event of an oil, hazardous waste or hazardous material spill;
- Employee training to make sure that they are aware of the above procedure and that they know how to react in the event of a spill;
- Soil characterization in 2019 (baseline) and at the end of the mining operations and, if necessary, decontamination.

21.2.1.4 Hydrology

Watersheds will be affected by mining operations. The only way to limit unwanted effects is to develop the project so that it has the smallest possible footprint and to avoid, as much as possible, any infringement on permanent watercourse. Infrastructure location was based on this idea.

21.2.1.5 Surface Water Quality

To reduce unwanted effects on surface waters, the mitigation measures presented below will be implemented:

- Deforestation is limited to the required surfaces only;

- Establishment of required works during construction to avoid the transport of suspended matter towards watercourses;
- Establish a procedure for accidental oil, hazardous waste or hazardous material spills;
- Build ditches and collecting basins to collect potentially contaminated waters;
- Construct a treatment system capable of ensuring the discharge of compliant effluents and aiming to respect, as far as possible, the effluent discharge objectives (EDO) that will be fixed by the MELCC;
- Employee training to make sure that they are aware of the procedure and that they know how to react in the event of a spill;
- Monitor water quality in the basins before its discharge into the receiving environment;
- Restore progressively the waste rock pile and the tailing storage facility to promote the implementation of vegetation cover;
- Limit the flow rate at the final effluent to avoid erosion of the receiving watercourse banks;
- Use emulsion type explosives. The loading of explosives into boreholes will be done with a tank truck, thus limiting possible spills;
- Prioritize the use of abrasives rather than ice melters during winter.

By applying all these mitigation measures, the water that will be discharged into the natural environment is expected to be harm free for the environment.

21.2.1.6 Hydrogeology

Dewatering the pit will cause localized groundwater drawdown during the mine operation period. This drawdown will not affect any water users, therefore no mitigation measures is required.

21.2.1.7 Underground Water Quality

During the life of the Authier Project, mining activities will have no effect on the underground water quality and the following mitigation measures will be implemented:

- A procedure in the event of an oil, hazardous waste or hazardous material spill;
- Employee training to ensure that they are aware of the above procedure and that they know how to react in the event of a spill;
- Appropriate management of residual materials, hazardous materials and fuels.

21.2.2 Impacts and Mitigation Measures on the Biological Environment

21.2.2.1 Terrestrial Vegetation

The following mitigation measures will be adopted to reduce negative effects of activities on the terrestrial vegetation. All measures will apply to the construction, operation, reclamation and closure phases when relevant:

- Construction areas will be adequately delimited to minimize the size of terrestrial vegetation affected by construction work;
- Affected sites will be vegetated with indigenous species after the work is completed to restore natural conditions as quickly as possible and avoid erosion;
- Temporary protection materials for sites being revegetated will be used to promote rapid vegetation growth;
- The use of surface layers already affected by exploration work will be prioritized for heavy machinery circulation and the development of temporary storage sites for construction materials;
- The use of abrasives rather than ice melters will be prioritized during winter;
- Water will be used to control dust on the mining site roads during summertime;
- The use of herbicide to control vegetation growth will be prohibited. Mechanical and manual methods will be prioritized when vegetation growth control will be necessary;
- An erosion and vegetation monitoring program will be implement at sites likely to be affected and, if necessary, stabilization measures will be applied;
- The stored overburden will be used for the gradual reclamation of the waste rock pile and tailings storage facility.

21.2.2.2 Wetlands

The following mitigation measures will be implemented to reduce negative effects of activities on wetlands. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant:

- Construction areas will be adequately delimited to preserve the wetland areas affected by construction work;
- Culverts will be installed in areas where a road crosses wetlands to ensure that surface water circulates freely;
- Drainage ditches near wetlands will be shallow and designed to limit the drawdown of the phreatic surface;

- The use of surface layers already affected by exploration work will be prioritized for heavy machinery circulation and the development of temporary storage sites for construction materials;
- Construction areas will be adequately delimited to avoid wetlands;
- The use of abrasives rather than ice melters will be prioritized in winter;
- Water will be used to control dust on mining site roads during summertime;
- A prevention and intervention plan in the event of an accidental spill or hazardous material leak will be developed to preserve the integrity of wetlands;
- Adequate training will be provided to employees to ensure that a quick and safe response occurs in the event of a hazardous material spill;
- Standards for the storage and handling of harmful substances will be respected and staff will be adequately trained.

Finally, a compensation plan should need to be developed to offset losses of wetlands under the Act respecting the conservation of wetlands and bodies of water.

21.2.2.3 Ichthyofauna

The following mitigation measures will be implemented to reduce negative effects of activities on ichthyofauna and its habitat. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant:

- Affected sites will be vegetated with indigenous species after the work is completed to restore natural conditions as quickly as possible and avoid erosion;
- Construction areas will be adequately delimited to avoid fish habitat;
- Culverts will be installed in areas where a road crosses wetlands and watercourses to ensure that surface water circulates freely;
- All potentially contaminated waters will be treated if needed before being sent back into the aquatic environment;
- The use of abrasives rather than ice melters will be prioritize during winter;
- Water will be used to control dust on the mining site roads during summertime;
- Maintenance areas will be designed to preserve the aquatic environment in the event of an accidental spill of hazardous material;
- A prevention and intervention plan in the event of an accidental spill or hazardous material leak will be developed to preserve the integrity of the ichthyofauna habitat;
- Adequate training will be provided to employees to ensure that a quick and safe response occurs in the event of hazardous material spill;

- Standards for the storage and handling of harmful substances will be respected and staff will be adequately trained;
- Waste will be eliminated properly;
- Revegetation of disturbed sites will be carried out after the work is completed to limit erosion;
- A riparian strip with a width of at least 30 m will be preserved on the banks of watercourses and waterbodies (Directive 019) to protect the aquatic environment;
- Rehabilitate final effluent (stream that will receive runoff water from the waste rock and tailings pile) based on its original conditions (bank, slope, dimension, vegetation and particle size).

21.2.2.4 Herpetofauna

The following mitigation measures will be implemented to reduce negative effects of activities on the herpetofauna. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant:

- Construction areas will be adequately delimited to minimize the surface area affected by construction work;
- Temporary protection materials for sites being revegetated will be used to promote rapid vegetation growth;
- All potentially contaminated waters will be treated if needed before being sent back into the aquatic environment;
- Revegetation of disturbed sites will be carried out after the work is completed to restore natural conditions as quickly as possible and provide new habitats for herpetofauna;
- The use of surface layers already affected by exploration work will be prioritized for heavy machinery circulation and the development of temporary storage sites for construction materials;
- The use of abrasives rather than ice melters will be prioritized during winter;
- Water will be used to control dust on the mining site roads during summertime;
- A prevention and intervention plan in the event of an accidental spill or hazardous material leak will be developed to preserve the integrity of the herpetofauna habitat;
- Adequate training will be provided to employees to ensure that a quick and safe response occurs in the event of a hazardous material spill;
- Standards for the storage and handling of harmful substances will be respected and staff will be adequately trained;

- A riparian strip with a width of at least 30 m will be preserved on the banks of watercourses and waterbodies (Directive 019) to protect the aquatic environment and to reserve a space allowing species to move;
- The circulation of heavy machinery and vehicles will be limited to predetermined areas to reduce affected areas;
- The waste rock pile and tailings storage facility will be revegetated with indigenous species when operating conditions allows it, to provide new habitats for herpetofauna as quickly as possible.

21.2.2.5 Chiropterofauna

The following mitigation measures will be implemented to reduce negative effects of activities on the chiropterofauna. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant:

- Construction areas will be adequately delimited to minimize the area of forest stands affected by construction work;
- Revegetation of disturbed sites with indigenous species will be carried out after the work is completed to restore natural conditions as quickly as possible
- Temporary protection materials for sites being revegetated will be used to promote rapid vegetation growth;
- Low luminosity light bulbs will be used to reduce lighting distance;
- Illuminated sites will be confined were they are really needed;
- Timers and motion detectors will be favored to limit unnecessary artificial lighting;
- Construction materials for infrastructure with high STC5 rates will be used when possible;
- Deforestation will be limited to pre-established sections;
- Heavy machinery and vehicles will have functional and efficient mufflers;
- Noise control devices will be installed on pneumatic and hydraulic hammers;
- Trucks will have a white noise backup alarm (multi-frequency sound);
- Fixed motorized equipment such as generators will be cased to soundproof them;
- Equipment will be installed as far as possible from the sensitive receiving environment;
- Regular maintenance on all equipment will be done, especially exhaust systems;
- The loudest work will be done during the day if possible;
- Limit circulation of heavy machinery and vehicles to predetermined locations (for example, service roads and work areas) to reduce the area of affected zones.

21.2.2.6 Small mammals

The following mitigation measures will be implemented to reduce negative effects of activities on small mammals. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant:

- Construction areas will be adequately delimited to minimize the size of vegetation areas affected by construction work;
- Revegetation of disturbed sites with indigenous species will be carried out after the work is completed to restore natural conditions as quickly as possible;
- Temporary protection materials for sites being revegetated will be used to promote rapid vegetation growth;
- An erosion and vegetation monitoring program will be put in place at sites likely to be affected and, if necessary, stabilization measures will be applied;
- A prevention and intervention plan in the event of an accidental spill or hazardous material leak will be developed to preserve the integrity of the aquatic environment;
- Adequate training will be provided to employees to ensure that a quick and safe response occurs in the event of a hazardous material spill;
- The use of surface layers already affected by exploration work will be prioritized for heavy machinery circulation and the development of temporary storage sites for construction materials;
- Limit circulation of heavy machinery and vehicles to predetermined locations (for example, service roads and work areas) to reduce the size of affected areas;
- The use of abrasives rather than ice metlers will be prioritize during winter;
- Water will be used to control the dust on mining site roads during summertime;
- The waste rock pile and tailings storage facility will be revegetated with indigenous species when operating conditions allows it, to provide new habitats for small mammals as quickly as possible.

21.2.2.7 Avifauna

The following mitigation measures will be implemented to reduce negative effects of activities on avifauna. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant:

- Construction areas will be adequately delimited to minimize the size of vegetation areas affected by construction work;

- Revegetation of disturbed sites with indigenous species will be carried out after the work is completed to restore natural conditions as quickly as possible;
- Temporary protection materials for sites being revegetated will be used to promote rapid vegetation growth;
- Low luminosity light bulbs will be used to reduce lighting distance;
- Illuminated sites will be confined were they are really needed;
- Timers and movement detectors will be installed to limit unnecessary artificial lighting;
- Heavy machinery and vehicles will have functional and efficient mufflers;
- Noise control devices will be installed on pneumatic and hydraulic hammers;
- Fixed motorized equipment such as generators will be cased to soundproof them;
- Equipment will be installed as far as possible from sensitive receiving environment;
- Regular maintenance on all equipment will be done, especially exhaust systems;
- Circulation of heavy machinery and vehicles will be limited to predetermined areas to reduce affected surface area;
- Revegetation of disturbed sites with indigenous species will be carried out after the work is completed to restore natural conditions as quickly as possible and recreate a new habitat for the avifauna;
- Deforestation will be undertaken, if possible, outside the bird breeding season;
- Heavy machinery will be directed for traffic only on the planned deforestation sites and the limits beyond which the circulation of heavy machinery and vehicles is prohibited will be clearly identified;
- The use of surface layers already affected by exploration work will be prioritized for heavy machinery circulation and the development of temporary storage sites for construction materials;
- Overburden and native species will be used to naturalize the waste rock and tailings pile;
- The waste rock pile and tailings storage facility will be revegetated with indigenous species when operating conditions allows it, to provide new habitats for the avifauna as quickly as possible.
- Native plant species (herbaceous, shrubs and trees) will be used during site restoration.

21.2.2.8 Species of Interest

The described mitigation measures will be implemented to reduce negative effects of activities on species of interest according to their taxonomic group. All of these measures will apply to any of the construction, operation, reclamation and closure phases when relevant.

21.2.3 Impacts and Mitigation Measures on the Human Environment

Among the mitigation measures considered to reduce or eliminate the undesirable effects of the project and to ensure maximum local benefits, a human environment monitoring program will be put in place jointly with stakeholders as part of the work of the Liaison Committee and Monitoring Committee. Before activities start, the comments and concerns of stakeholders will continue to be documented and discussed to ensure a rigorous follow-up of the issues and to provide immediate responses.

21.2.3.1 Use of Land and Resources

Hunting camp owners expressed interest in continuing discussions with Sayona Québec regarding the development of the project and on the measures that could be taken if the project were to cause drawbacks. Some mentioned their desire to receive help in securing their access to the site by opening a new path if the project goes forward.

For snowmobile and ATV trails, it will be necessary to evaluate with the members of the concerned groups or clubs if their course is disturbed by the activities of the project and, if necessary, to see if it would be possible to modify certain affected routes.

21.2.3.2 Employment and Economy

Employment creation in this region is expected by the community, Sayona Québec has committed to favor employing local population if qualifications are deemed equivalent to ensure direct social and economic benefits for the local population. Sayona Québec also committed in giving subcontracting contracts to local companies, particularly for construction, deforestation or transport, which will further stimulate the economy and direct benefits to the local economy. This commitment was made before the La Motte Community as well as the Abitibiwinni First Nation.

For this purpose, Sayona Québec initiated the creation of a local business register that also contains their contact information. This will facilitate local recruitment.

21.2.3.3 Cultural and Archaeological Heritage

No mitigation measures or specific maximization is planned for the cultural and archaeological heritage, except if, during mining activities a cultural or archaeological site is discovered. In this case, the managers will have to report it to the site supervisor and, if necessary, work will cease at this site until an evaluation is completed by archaeologists. The public will be informed.

21.2.3.4 Well-being and Landscape

With respect to social welfare measures, the human environment monitoring program will include measures to ensure continuity in the relationship between Sayona Québec and stakeholders. The committee responsible for implementing the program will be able to define, in collaboration with the communities involved, measures to counter the undesirable effects of the project on social well-being, or to identify and amplify its benefits.

With regard to the landscape component, Sayona Québec has also committed to discuss with stakeholders to identify possible measures to take advantage of the site, once restored.

21.3 Monitoring Program

During the mine site future operations, a monitoring program needs to be implemented with some instrumentation (e.g. groundwater monitoring wells, surface water monitoring stations, etc.). The environmental monitoring program aims to ensure compliance with the environmental laws and regulations and the conditions of the certificates of authorization that will be issued by the MELCC or conditions of the mining lease issued by the MERN.

The monitoring program will also consist of ensuring compliance with the commitments that Sayona Québec has made during the various meetings with stakeholders and public consultations. A document outlining all mitigation commitments, mitigation measures and conditions will be prepared, and Sayona Québec will ensure that all of these commitments, measures and conditions are met.

The monitoring program will be used to continue the environmental monitoring of the site after its rehabilitation and closure. Following the reclamation work, environmental monitoring of surface and groundwater will be carried out according to the requirements of Directive 019 of the MELCC.

21.3.1 Environmental Monitoring

In order to follow the evolution of the effects induced by the project implementation, Sayona Québec proposes to implement various environmental monitoring programs during the operation and after the closure and restoration of the site.

21.3.1.1 Groundwater Monitoring

Piezometers are already installed on the site and monitoring of groundwater quality has been done since 2017. Some piezometers are equipped with water level probes and measurements are done continuously. This monitoring will continue during construction, operation and after the closure of the site. Piezometers will be added before construction outside the affected areas, as many of the piezometers currently installed will have to be destroyed (those in the footprint of the pit or the dump for example). The monitoring program will comply with the requirements of Directive 019.

Table 21-10 presents the piezometers on the site, the proposed sampling frequency and the parameters that will be analyzed. The exact location of the piezometers that will be installed prior to the construction of the site will be presented in as part of the process of applying for permits for these works to the MELCC. The follow-up will be done according to the modalities requested by the MELCC under the conditions of the certificates of authorization. After restoration, monitoring will continue for a minimum of 5 years.

Table 21-10: Groundwater Monitoring

	# Piezometers	Water Quality
Piezometers	PZ-01R, PZ-01MT, PZ-02R, PZ-02MT, PZ-03MT, PZ-04R, PZ-05R, PZ-06R, PZ-07R, PZ-07MT, PZ-08MT, PZ-08R, PZ-08MT, PZ-09MT, PZ-09R, PZ-09MT, PZ-10R, PZ-10MT, PZ-11R, PZ-11MT, PZ-12R, PZ-12MT, PZ-13R, PZ-13MT, PZ-14R, PZ-15R, PZ-16R, PZ-17MT, PZ-18MT, 08010004 (MDDELCC)	PZ-01R, PZ-01MT, PZ-02R, PZ-02MT, PZ-03MT, PZ-04R, PZ-05R, PZ-06R, PZ-07R, PZ-07MT, PZ-08MT, PZ-08R, PZ-08MT, PZ-09MT, PZ-09R, PZ-09MT, PZ-10R, PZ-10MT, PZ-11R, PZ-11MT, PZ-12R, PZ-12MT, PZ-13R, PZ-13MT, PZ-14R, PZ-15R, PZ-16R, PZ-17MT, PZ-18MT
Sampling Frequency	Minimally twice a year: in spring and summer to represent periods of high and low water levels. Many wells are instrumented for continuous monitoring.	Minimally twice a year: in spring and summer to represent periods of high and low water levels.
Analyzed Parameters	n/a	Metals: Arsenic, copper, iron, nickel, lead, zinc, cyanides, Hydrocarbons: C10-C50 Physico-chemistry: pH, electrical conductivity (in situ) Major ions: Ca^{2+}, HCO_3^-, K^+, Mg^{2+}, Na^+, SO_4^{2-}) Radioactivity: Ra226

21.3.1.2 Surface Water Quality Monitoring

The stream CE02 flows into Lake Kapitagama about 4 km from the site. The monitoring of this final effluent will comply with the requirements of Directive 019 on the mining industry (March 2012 version) and with the requirements of the Metal and Diamond Mining Effluent Regulation (MMER), at the Federal level.

Monitoring will be carried out as soon as the final effluent is discharged and will continue for 5 years after closure (Directive 019). The parameters monitored will be at least those of Directive 019 and the MMER, but Sayona Québec intends to present a more comprehensive program in relation to the environmental objectives that will be presented for the project.

21.3.1.3 Fish Fauna Population Monitoring

Only the federal government requires monitoring of the biological environment, which is a requirement of the MMER.

Monitoring will be carried out on the effluent stream of the mine effluent and a plan will have to be developed before the start of the construction phase. Sayona Québec will submit its monitoring program to the authorities for approval (sections 19 and 23 of Schedule 5 of the MMER).

The objective of the monitoring program will be to assess the maintenance of fish fauna populations and habitat conditions. In addition to the study of fish fauna, the program will include:

- Collection of sediment samples in the same areas as those used for monitoring the fish fauna. Sediment samples will be taken to determine particle size distribution and total organic carbon content;
- Sampling to document benthic invertebrate communities (living at the bottom of the water). This information is used as an indicator of the quality of the fish habitat. Comparing benthic invertebrate communities exposed to treated mine effluents (exposed area) with other communities that are not exposed (reference area) will qualify the effects.

21.4 Permitting Requirements

In accordance with Québec's Mining Act and Environmental Quality Act, permits are required in order to build and operate the mine. A mining lease is required from the Ministry of Energy and Natural Ressources (MENR). In the Environmental Quality Act, the trigger to produce and Environmental study is the capacity of extraction and treatment. If this capacity stays below 2,000 tpd, the company has to obtain a certificate of authorization from the MELCC.

From a federal point of view, no Environmental Impact Assessment is required as long as none of the physical activities (SOR/2012-147) would trigger the federal process.

Furthermore, some other permit and authorization will be required in connection with the mining activities.

21.4.1 Mining Lease

The mining lease is required to extract ore under the Mining Act. The application must be accompanied by, among other things, an approved closure and rehabilitation plan and a scoping and market study on processing in Québec.

The deliverance of the mining lease is conditional on obtaining the approbation of the closure plan. According to the Quality Environmental Act a certificate of authorization is also required for construction and operation of the mine. A public consultation must also be part of the legal obligation and should last at least two months and include public open doors in the municipality where the project is located.

21.4.2 Certificate of Authorization (Government Decree)

The global certificate of authorization frames the environmental component of the project, in respect to the regulation respecting the environmental assessment and review of certain projects (CQLR, cQ2, r23.1). The projects listed in Schedule 1 are subject to the environmental impact assessment and review procedure under the Environment Quality Act (article 31.1). Therefore, Schedule 1 includes the establishment of a mine whose maximum daily capacity is equal to or greater than 2000 metric tons.

In order to obtain it, an application must be filed to the MELCC. Before delivering any permit and the government decree, the Québec government has to consult the First Nations impacted by the project.

21.4.3 Other Authorizations

21.4.3.1 Other Requirements

Other permits or leases will have to be obtained depending on planned development activities at the site. Also, depending on RCM or municipal legislation, some permits may also be required from the RCM or the municipality.

21.4.3.2 Provincial Government

On the provincial side, these other authorizations or approvals could be required:

- Depollution attestation under the *LQE*, art. 31.11 (see *Règlement sur les attestations d'assainissement en milieu industriel*) – emissions of a metal ore mining establishment with a mining capacity greater than 2,000,000 metric tons per year of ore or mine tailing processing capacity greater than 50,000 metric tons per year (operations involving ore beneficiation are included in ore processing operations);
- Authorization under *LQE*, art. 32 – establish waterworks or install devices for waste water treatment;
- Authorization under *LQE*, art. 48 – install atmospheric depollution equipment;

- Authorization under *Loi sur les espèces menaces ou vulnérables*, art. 17 – activity carried out in threatened/vulnerable plant species habitat;
- Wildlife management permit under *Loi sur la conservation et la mise en valeur de la faune*, art. 26 – disturbance to beaver dam, eggs, nest or den;
- Authorization under *Loi sur la conservation et la mise en valeur de la faune*, art. 128.6 – activity carried out in wildlife habitat pursuant to *Règlement sur les habitats fauniques*;
- Approval under *Loi sur les mines*, art. 241 – tailings and waste storage and concentrator site;
- Authorization under *Loi sur les mines*, art. 232.2 – land rehabilitation and restoration work;
- Lease under *Règlement sur la vente, la location et l'octroi de droits immobiliers sur les terres du domaine de l'État*, art. 39 – occupation of Crown land;
- Forestry permit under *Loi sur l'aménagement durable du territoire forestier*, art. 73 – forest development activities (related to timber felling, construction of infrastructure) by mining rights holder;
- Certificate of authorization (in accordance with *LQE*, art. 22) under *Règlement sur les carrières et sablières*, art. 2 – pit or quarry operation;
- Approvals under *Loi sur le régime des eaux*, arts. 57 and 71 – construction/ maintenance of reservoirs for storage of water from waterbodies/ watercourses and construction/maintenance of dams and other water-retaining works respectively;
- Authorization under *LQE*, art. 46 s) (*Règlement sur le captage des eaux souterraines*, art. 31) – wells (groundwater extraction for industrial water supply) if collection exceeds 75 m^3 per day;
- Permits under *Règlement d'application de la Loi sur les explosifs*, arts. 3, 4 and 6 respectively – possess, purchase, store and transport of explosives;
- Approval under *Loi sur le Bâtiment* (*Code de Construction*), art. 8.08 – installation of petroleum equipment (storage of petroleum products);
- Certificate of conformity under *Loi sur le Bâtiment* (*Code de Construction*), art. 8.12 – installation of high-risk petroleum equipment;
- Maintaining a register, certificate of conformity and permit under *Loi sur le Bâtiment* (*Code de sécurité*), arts. 114, 115 and 120 respectively – installation and operation of petroleum equipment (including high-risk petroleum equipment).

21.4.3.3 Federal Government

Based on available information, the federal government will not be involved in the permitting process. The Authier Project could require these federal authorizations or approval:

- License and permit under Explosives Act, art. 7(1) – license for explosives factory and permit for transportation of explosives;
- Approval under Transportation of Dangerous Goods Act, 1992, art. 7 – Emergency response assistance plan to import, offer to transport, handle or transport dangerous goods;
- Agreement with competent minister or permit under Species at Risk Act, art. 73 – Activity affecting a listed wildlife species, any part of its critical habitat or the residences of its individuals;
- A possible authorization from Fisheries and Oceans Canada under Fisheries Act paragraph 35(2) Authorizations, because *no person shall carry on any work, undertaking or activity that results in serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery.*
- Metal and Diamond Mining Effluent Regulations (SOR/2002-222) of the Fisheries Act, including the Environmental Effects Monitoring Studies, Analytical Requirements for Metal Mining Effluent, Annual Report Summarizing Effluent Monitoring Results.

21.5 Waste Rock, Ore and Tailings Characterization

21.5.1 Geochemical Characterization

Sayona Québec conducted a geochemical characterization study of ore, waste rock and tailings samples.

The geochemical program allows the classification of waste rock and tailings according to provincial authority's regulations standard for acid mine drainage and leachability, and identify any chemical that could potentially affect the surface or groundwater quality.

No evidence of sulfides has been observed in the ore or in the waste rock. Volcanic ultramafic and mafic rocks are the principal lithological unit that hosts the deposit and is likely to end up in waste rock stockpile. The geochemical characterization of this unit, the others minor units (peridotite, schist, diorite, barren pegmatite) and the ore (spodumene-bearing pegmatite) was done using static tests. Samples of drill core were tested following the characterization program guidelines proposed by the MDDELCC (*Directive 019 on the Mining Industry*, 2012). Some additional tests were completed to identify elements that could be leached under different conditions.

A total of three ore samples and 52 waste rock samples were collected and tested. These samples were selected based on geological cross-sections through the deposit in order to select samples that will represent the vertical and spatial variability of the lithological rock units. A total of two samples of concentrator tailings have also been tested. Samples were collected from metallurgical testing and are representative of the final tailings.

The potential of waste rock to generate acid rock drainage was evaluated through Modified Acid Base Accounting. Results showed that waste rock, ore and tailings samples were not acid generating. Almost all of them contained less than 0.3% total sulphur content, which is below the threshold (*Directive 019*). Only one exceeded this criterion with a content of 0.303% total sulphur.

Samples of waste rock, ore and tailings have also been tested for their metal leaching potential. Static tests under *Directive 019* that is used to characterize the metal leaching potential of rock materials consists of trace metals analysis combined with short term leaching test Toxicity Characteristic Leaching Procedure Test (TCLP – EPA Method 1311). Another short-term leaching test was conducted to characterize the metal leaching potentials of both waste rock, and ore and tailings: the Synthetic Precipitation Leaching Procedure test (SPLP – EPA Method 1312). This test simulates natural acid-rain-type conditions of water pH of 4.2 using sulphuric acid and nitric acid.

According with definition of Québec's *Directive 019*, the waste rock and the ore could be classify as leachable for copper and nickel in accordance with TCLP test results. Although, the TCLP test uses an organic acid as the leaching solution, it is not representative of the leaching conditions that would be at the Authier site. Results from SPLP indicated that the waste rock, the ore and the tailings did not show exceedances of any parameters, except in copper for one sample of ore.

21.5.2 Considerations for Tailings and Waste Rock Management

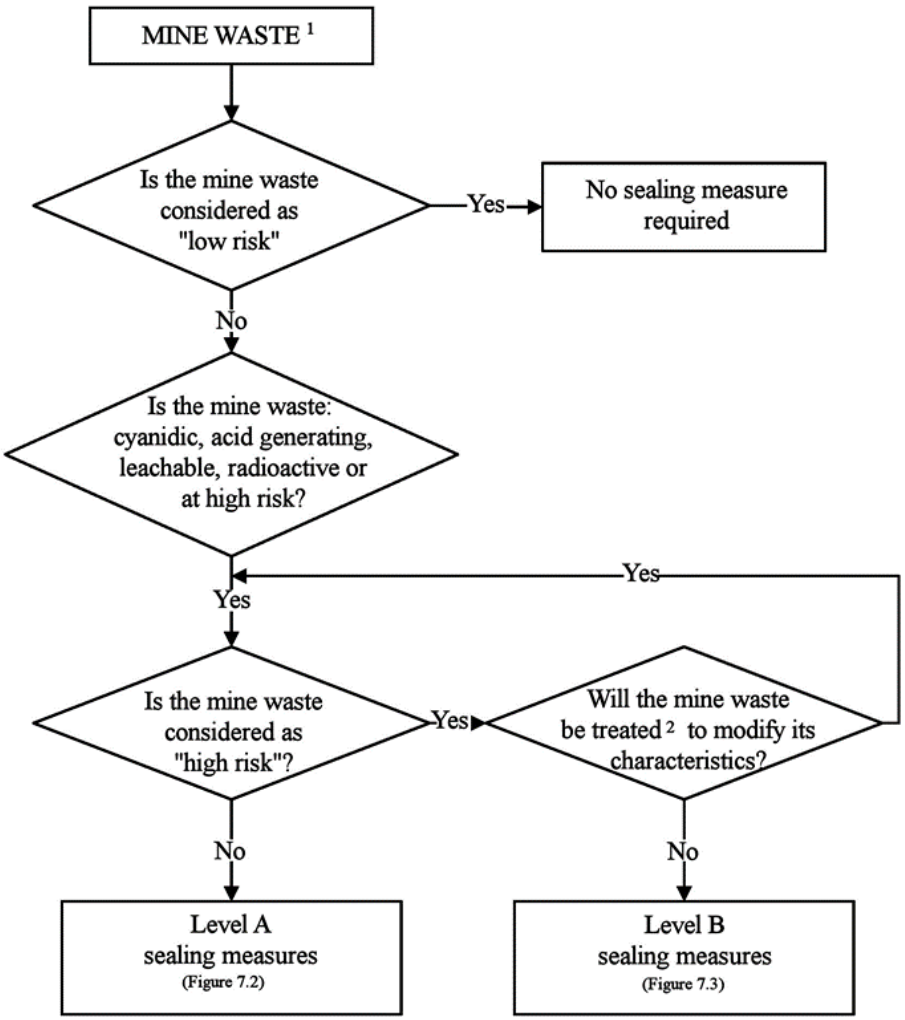
The disposal of tailings and waste rock must comply with the guidelines of Directive 019. The Directive 019 is a tool used by Regulators to analyze mining projects requiring the issuance of a Certificate of Authorization as per the Environmental Quality Act[7] . Directive 019 specifies, among other things, the level of protection required in the construction of a WMF (Waste Management Facility) based on the level of hazardousness of the mining waste to be disposed of.

Appendix I of Directive 019 defines mine waste as any substance, solid or liquid, rejected during the processes of extraction (mining), preparation, enrichment and separation of ore, including the mud and dust produced during treatment or purification of waste water and air emissions[8]. Based on this definition, it is considered that mine waste includes both the tailings produced by milling of the ore and the waste rock extracted by mining methods to access the ore.

Appendix II of Directive 019 describes the characteristics of eight different types of mine waste. Directive 019 specifies that "low risk" mine waste do not require any particular confinement and that "high risk" mine waste be confined inside an impoundment built with Level B protective measures. All the other types of waste have to be confined with Level A protective measures.

[7] Environmental Quality Act, L.R.Q. c. Q-2
[8] Non-official translation of the definition of mine waste (Directive 019, Appendix I)

Figure 21-8 below summarizes the criteria used to determine the level of protective measures required for a given type of mine waste.



[1] Mine waste: includes both the tailings and the waste rock (as per Appendix II of the Directive).

[2] Treatment: process aiming at reducing the hazardousness of the mine waste.

Figure 21-8: Decision Flowsheet to Determine the Level of Required Protective Measures (Translation of Figure 2.3 of Directive 019, March 2012 version)

In respect with requirements stated above, and in order to minimize the footprint of the project, tailings and waste rock will be co-disposed on a unique pile located close to the pit. Tailings will be filtered and transported by truck on the pile. This deposition technique will ensure that no dyke will be necessary which eliminate the risk of any dyke failure. Also, the co-disposition technics allow a progressive restoration, a more efficient water management in operation and at closure and a better dust control.

Furthermore, the tailings and waste rock will be co-dispose in two phases in order to have a more efficient water management. For phase 1, the collection basin #2, the sedimentation basin and the ditches associated with them will be constructed to collect contact waters from the co-disposal facility. In phase 2, due to the extension to the west of the pile, the collection basins #1 and 3 and their ditches will need to be constructed (see Figure 18-2 and 18-3). Also, all clean water will be diverted from the site.

21.6 Closure and Reclamation Plan

A rehabilitation and closure plan is a requirement under the *Loi sur les mines*. It must be approved before the mining lease is issued, and a financial guarantee to fully implement the plan must be provided in three payments in the first 2 years following the approval of the plan. The closure plan has been submitted in May 2018.

Progressive reclamation would be encouraged during the mining operation and will involve activities to reclaim, where possible, some parts of the waste rock and tailings stacking areas, exhausted borrow pits, etc.

Rehabilitation would involve all activities after mining operations in accordance with the approved plan. Finally, monitoring would ensure that rehabilitation has been done successfully. Once all these steps are completed to the satisfaction of the MERN, the land could be returned to the Crown.

21.6.1 Overview

In accordance with the *Mining Act* (L.R.Q., chapter M-13.1, article 232.6) requirements a detailed closure plan has been submitted to the MERN on May, 18, 2018. Rehabilitation will be integrated early into the operation of the Project in order to reduce the mine’s footprint. The closure plan includes the following activities:

- Rehabilitate the waste rock and tailings pile by covering the slope and benches with overburden and vegetation;
- Remove from the site all surface and buried pipelines;
- Remove buildings and other structures;
- Rehabilitate and secure the open pit;
- Assess surface workings;
- Reclaim any civil engineering works;

- Remove machinery, equipment and storage tanks; and,
- Complete any other work necessary for final rehabilitation and closure.

21.6.2 Post-Closure Monitoring

The detailed post-closure monitoring program will be developed once the project has been well defined and the specific rehabilitation activities are known. It will be conducted for at least 5 years after the final activities are completed. It will include the following aspects:

- Monitoring of final effluent and surface water quality (6 times a year for a minimum of 5 years);
- Stability of vegetation (once a year for 5 years);
- Inspection for slope of the open pit, waste rock and tailings pile, ditches, etc. (once a year for 5 years);
- Soils stability (control of erosion) (once a year for 5 years); and,
- Monitoring of groundwater quality (minimum of 3 wells, twice a year for 5 years).

All the analysis parameters and the number of samplings per year will be established in respect with de Directive 019.

22. TRANSPORT, LOGISTICS AND PORT INFRASTRUCTURE

22.1 Introduction

As with the majority of mining operations, not just within Abitibi region, but globally, the logistical challenges of getting the product from producer to consumer are a key consideration for this project.

The base objective of the logistics component of the proposed Authier Lithium project is to transport roughly 115 000 t per annum of spodumene concentrate at 6% Li_2O to a selected port facility. The product will then be loaded onto bulk carrier vessels in, most likely, 15,000 t to 25,000 t consignments for export to the purchaser.

22.2 Location

The mine is located in La Motte approximately 590 km from Montreal and can be reached by travelling via Highway 15, 117 and 109 North (Figure 22-1).

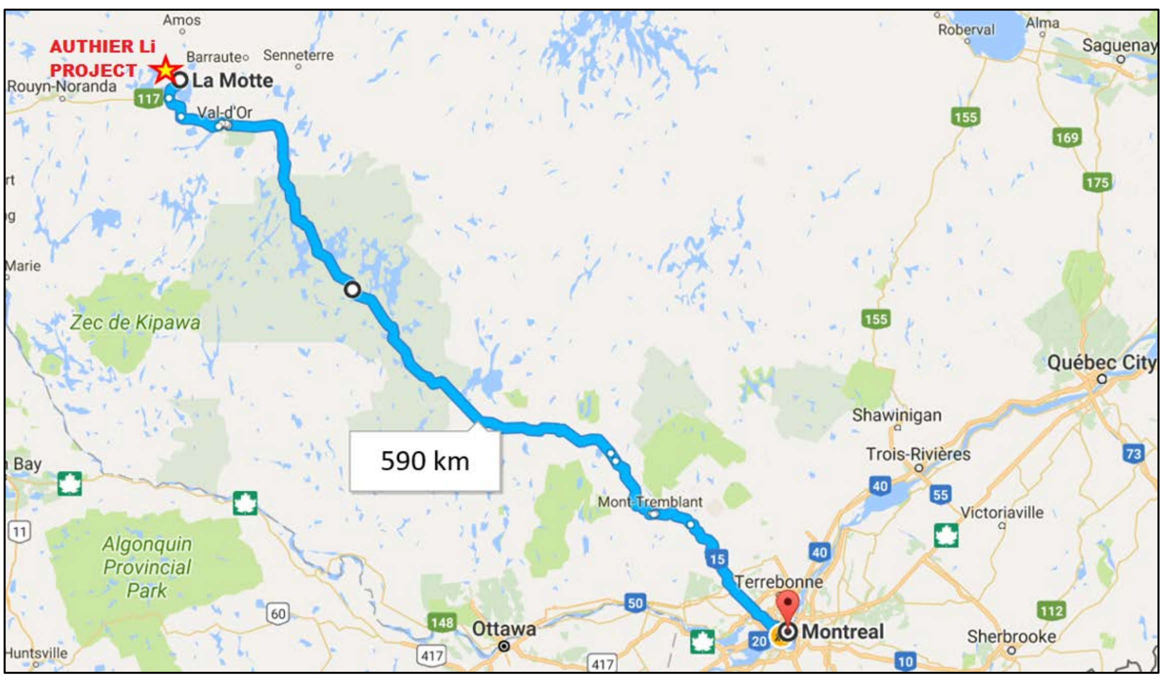


Figure 22-1- Project Location Relative to the Port of Montréal

22.3 Mine Site Manipulations

A front-end loader will be used to load the 40-t trailers in the covered concentrate storage area. Trucks will come to the site on a daily basis and will bring the trailers to their final destination. The loader will be supplied and maintained by Sayona.

A weighbridge will be used to ensure that the payload is within acceptable limits as prescribed by Québec regulations. The weighbridge will be supplied, installed and maintained by Sayona.

22.4 Logistics Study

In 2018, Sayona commissioned a transport and logistics company to assess the feasibility of transporting the spodumene concentrate from the stockpile at the mine site to side of ship (Free on Board) of a bulk carrier vessel located at a port facility. The study assessed two options:

- A combination of bulk-trucks and bulk-railcars; and
- The use of bulk-trucks for transport from site to port.

The study also assessed three port options: Montréal, Contrecoeur and Trois-Rivières (refer to (Figure 22-2).

Figure 22-2– Location of Prospective Ports Considered in Study

22.4.1 Truck and Train to Port

The execution plan of this option is defined as follows:

1. Loading operation on site (by Sayona);
2. Covering of the trailers;
3. Weighing each truck with a certified scale;
4. Transport from site to a railway siding;
5. Uncovering of trucks;
6. Unloading of concentrate into appropriate storage facility;
7. Loading the concentrate into railcars;
8. Covering the railcars;
9. Transport from railway siding to port;
10. Uncovering of railcars;
11. Unloading of concentrate into appropriate storage facility.

Various railway service providers owning a railway siding in Abitibi were approached: Groupe Mirault (Val-d'Or), Dumas (Cadillac), Groupe Octium (Malartic) and CK Logistic (Rouyn-Noranda). Canadian National (CN) was also contacted to provide analysis support. Groupe Mirault was retained due to their all-in service package coordinating all aspects of road and rail transport to their closed shed in Val-D'Or.

Thus, Mirault would provide the trucks to transport the concentrate from the mine to their shed, load the leased railcars on their return from the port, install the cover on the railcars and monitor the rotation of the traffic with CN rail. The terminal is served by CN five days a week.

The table below illustrates the number of standard railcars that will be required to meet the demand of roughly 115,000 t per annum product delivered from the railway siding shed to the selected port storage shed.

Table 22-1: Railcars Analysis

Item	Unit	Value
Yearly tonnage	t	115,000
Railcar tonnage	t	96
Cars to be shipped	Unit per year	1,198
Operating weeks		50
Cars per week		24
Operating days	day	350

The rail option will be solely based on the service of CN Rail. They will control the flow of loaded and empty railcars.

If the whole traffic would move by rail, the mine would be fully dependent on the performance of CN Rail. On a national basis, CN Rail service has been affected by several issues and we would therefore not recommend entrusting 100% of the tonnage to the rail system.

22.4.2 Truck to Port

The execution plan for this option is defined as follows:

1. Loading operation on site (by Sayona);
2. Covering of the trailers;
3. Weight each truck with a certified scale;
4. Transport form site to port;
5. Uncovering of trailers;
6. Unloading of concentrate into appropriate storage place.

All road transport operations will be performed in accordance under Québec Ministry of Transport regulation. This regulation involves a thaw period, during which reduction of load is mandatory. During periods of thaw, the road is 30% to 70% more fragile than usual. A single overloaded vehicle can cause substantial damage. However, a slight reduction in load significantly decreases damage to the pavement. Therefore, load restrictions are imposed on heavy vehicles during periods of thaw. These restrictions generally range from 8% to 20%. Thaw periods are determined after monitoring the progress of thaw on the pavement. This is done using sensors across the Québec road network. Weather forecasts are also considered.

It was decided to aim for a turnkey service instead of Sayona owning its own transport fleet. The justifications for contracting a turnkey service are part of the following:

- Availability of drivers (hiring and retention);
- Administration and management of human resources;
- Costs of infrastructures and mechanics to maintain fleet;
- Costs for accident-incident-violation of regulations;
- Other costs (e.g., inventories, management, insurances);
- Necessity of maintaining a 24-hour service call line for road service;
- CAPEX of approximately $4.0 M.

Various contractors were requested to submit a bid to provide a turnkey service for the transportation of the concentrate from the mine to ports. Service Nolitrex Inc. proposal was retained due to their experience, their enthusiasm to participate in the project and their economic proposal. Nolitrex would to take care of the entire logistic from the mine to the port by providing truck, driver and B-train 40 t trailers, maintaining the fleet, taking care of the administration and fees, etc. and would charge Sayona on a cost per tonne basis.

The table below illustrates the numbers of standard B-train configuration that will be required to meet the demand of roughly 115,000 t per annum product delivered from the mine to the selected port storage shed.

Table 22-2: Trucking Analysis

Item	Unit	Value
Round trip – Assuming Port of Montréal	km	1,182
Annual tonnage	t	115,000
Monthly tonnage	t	9,583
Payload	t	38.8
Delivery windows	Hour	7 am to 5 pm

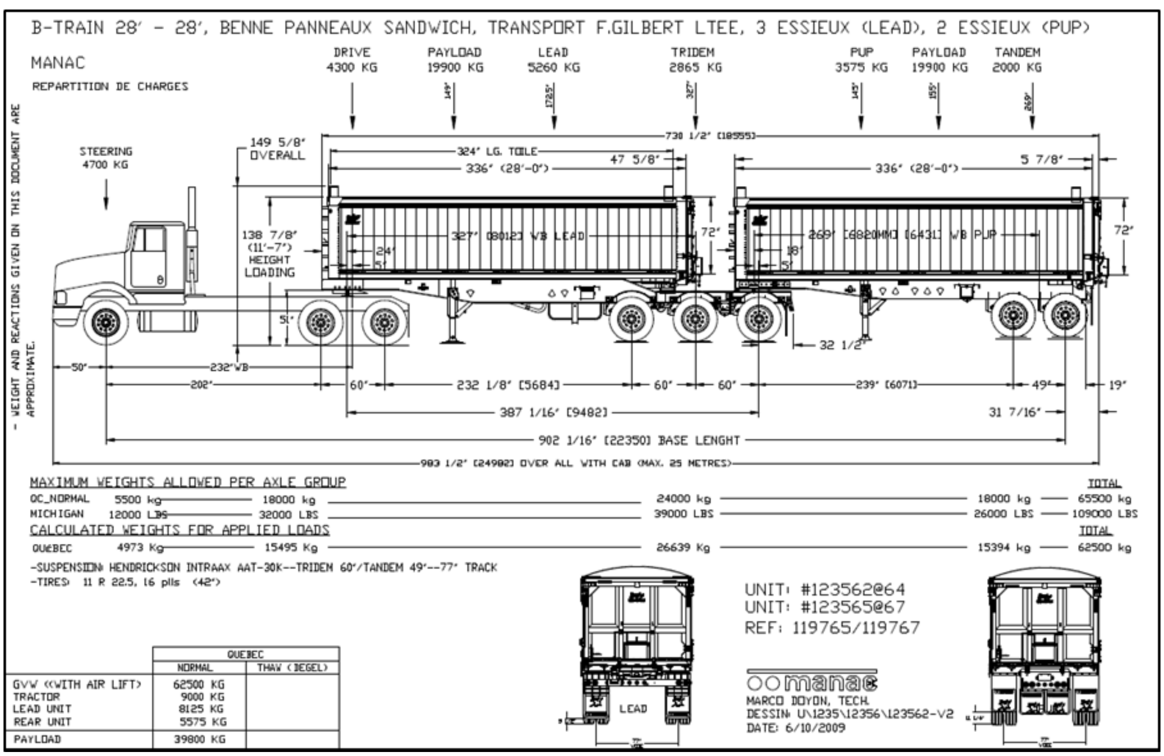


Figure 22-3 Typical B-train Trailer Configuration

22.4.3 Various Ports Options

Three ports were visited to determine their capability to receive the concentrate, to store it and to load it into the vessel. The following table summarizes the key findings for each option:

Table 22-3: Summary of Key Findings for each of the Proposed Port Options

Port	Operator	Pro	Cons
Montréal (590 km from site)	Logistec	▪ Is closest to the mine ▪ Most efficient bulk crane for loading operation ▪ Lowest price ▪ No congestion at bulk terminal ▪ Dedicated storage space available	▪ Rush hour traffic ▪ Major road and rail project may affect traffic
Contrecoeur (638 km from site)	Logistec	▪ No congestion ▪ Easy access apart from tunnel issue ▪ No congestion (dedicated bulk port)	▪ Major LH Lafontaine tunnel renovation project will affect flow of traffic to port of Contrecoeur ▪ Lack of covered space unless long term contract signed for construction of additional space - Highest price
Trois-Rivières (698 km from site)	Somavrac	▪ Already handling similar traffic ▪ No congestion	▪ Potential lack of space due to new traffics and limited land available for construction of new sheds ▪ Short rail reducing tonnage per railcar ▪ Furthest port from mine site

Transport from Authier mine to all three ports will be affected by the spring weight limitation (across 3 zones).

22.4.4 Costs Analysis

All costs received are presented in Table 22-4. The transport analysis was undertaken in 2018 and a 3% cost escalation was applied for the feasibility study update.

Table 22-4: Summary of Transportation Costs Received from Service Providers

	Unit	Main Transport Method					
		Train	Train	Train	Truck	Truck	Truck
Port		**Mtl**	**Contrec.**	**Trois-Riv.**	**Mtl**	**Contrec.**	**Trois-Riv.**
Transport Cost	$/t wet	5183	54.92	54.92	49.17	54.17	53.83
Port Cost	$/t wet	21.52	21.52	27.50	15.13	15.13	16.50
Total Cost	**$/t wet**	**73.35**	**74.38**	**82.42**	**64.30**	**69.30**	**70.33**

22.4.5 Selected Transport Option

The study concludes that the most optimal scenario would be to use a third-party transport contactor to haul from the mine to the port of Montreal using a fleet of B-trains trailers and trucks (owner operators). The B-trains to be on leased contract with owner operator. A well-managed dedicated truck fleet would allow Sayona to have complete control on the tonnage to be shipped according to production. It will also allow:

- Cost control and provisional projection;
- Flexibility to ship its concentrate as required to fulfill the confirmed orders;
- Ease of bulk handling (loading and unloading) / no intermediate handling between mine and port.

22.5 Port Operations

22.5.1 Stockpiling Operation

Product will be dumped in or in proximity to a covered storage shed. Stockpile management in the shed will be by a front-end loader and will be under the responsibility of the port logistic company.

Figure 22-4: Typical Storage Shed

22.5.2 Loading Operations

Material will be brought from the shed to the side of the vessel using a front-end loader and trucks. Concentrate will be loaded into the vessel using a mobile harbor crane (Figure 22-5). These operations will be under the responsibility of the port logistic company.

Figure 22-5: Typical Mobile Harbor Crane

23. HUMAN RESOURCES

23.1 Summary

The Authier Lithium project is located in the municipality of La Motte, Québec. The territory of the municipality covers an area of 176.9 km^2. The population of La Motte was 455 individuals in 2016. A portion of the project territory extends into the neighboring municipality of Presissac.

The Project is located on the territory of the Abitibiwinni First Nation community of Pikogan. The community is approximately 34 km north of the Project and 3 km north of the Town of Amos, located on the west bank of the Harricana River. The Pikogan community covers 2.77 km^2. In 2016, its population was 854 individuals, 319 of whom lived off the reserve. Most of the Pikogan population speak Anishnabe, followed by French and English. Some members of the community also speak Cree.

The Project will benefit from the local human resources and services in the Abitibi region. The biggest cities in proximity to the Project are Amos (13,000 people), Val-d'Or (26,000 people) and Rouyn-Noranda (42,000 people). The region is rich in mineral deposits and is one of the leading mining regions of Québec. The region has a long history of mining activities with a skilled workforce. The main cities are well equipped with all necessary amenities including hospital, emergency services, schools, sports centres, food, lodging, wireless, and wireline telecommunications.

Figure 23-1: Major Cities Surrounding the Authier Project

Table 23-1 presents the Project workforce requirements over the mine life. Year -1 is a fractional year, which explains the reduced workforce. There is a slight reduction in the manpower required in Year 1 due to lower planned production. The total mine workforce is 93 in the first year of operation and reaches a peak of 176 individuals by Year 6.

Table 23-1: Project Workforce

Year		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mining																
Mine superintendent	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine shift foreman	Staff	2	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Trainer / Health & Safety	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Clerk	Staff	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Truck Operator		5	11	14	20	32	44	56	48	48	42	32	17	13	12	12
Excavator Operator		2	6	6	9	12	15	17	17	14	11	8	5	4	4	4
Drill Operator		2	4	3	4	6	6	7	9	8	6	4	2	2	2	2
Auxiliary Fleet Operator		7	11	14	14	17	17	17	17	17	17	14	7	7	6	6
General Labourers		4	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Sub-Total – Mining		**24**	**42**	**47**	**57**	**77**	**92**	**107**	**101**	**97**	**86**	**68**	**37**	**32**	**30**	**30**
Mine Maintenance																
Maintenance Superintendent	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanic		9	10	12	15	19	24	28	26	25	22	17	10	7	6	5
Sub-Total - Mine Maintenance		**10**	**11**	**13**	**16**	**20**	**25**	**29**	**27**	**26**	**23**	**18**	**11**	**8**	**7**	**6**
Processing																
Processing superintendent	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shift supervisor	Staff	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Plant metallurgist	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Process technicians		6	17	17	17	17	17	17	17	17	17	17	17	17	17	17
Sub-Total - Processing		**9**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**	**22**
Processing - Maintenance																
Maintenance Planner	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Electrician		0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Instrument Mechanic		0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pipe Fitter		0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Millwright		0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Sub-Total - Mill Maintenance		**1**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**
Technical Services																
Mining Engineer	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mining Technician / Surveyor	Staff	1	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Geologist	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geology Technician	Staff	1	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Environment Coordinator	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Sub-Total - Technical Services		**5**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**5**	**5**	**5**	**5**
Admin & Others																
Secretary	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Purchasing manager / storekeeper	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
HR responsible / Public relations	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Accounting manager	Staff	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Sub-Total - Admin & Others		**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
Total		**54**	**93**	**100**	**113**	**137**	**157**	**176**	**168**	**163**	**149**	**126**	**86**	**78**	**75**	**74**

24. CAPITAL COST ESTIMATE

Not applicable.

25. OPERATING COST ESTIMATE

Not applicable.

26. ECONOMIC ANALYSIS

26.1 Summary

An economic, life-of-mine cash flow model of the project has been constructed using the production mining and processing production schedules developed for the feasibility study. The key outcomes of the economic evaluation for 100% of the project, before any financing costs, are presented in Table 26-1.

Table 26-1: Economic Analysis Summary

Authier Lithium Project Highlights		
Description	**Unit**	**Value**
Average Annual Ore Feed to the Plant	t	874,594
Average Annual Grade to the Plant	% Li_2O	1.0
Annual Average Spodumene Production - Dry (6% Li_2O)	t	114,116
Li_2O Recovery	%	78
Life of Mine (LOM)	year	13.8
LOM Strip Ratio	waste to ore	6.90
Average Spodumene Price	USD/t	693
LOM Operating Costs (mine gate) – Excluding Royalties	CAD million	631
LOM Transport and Logistics Costs (mine to port)	CAD million	109
Royalties purchase	CAD million	3.0
Development Capital Costs	CAD million	120
LOM Capital Costs	CAD million	211
Royalties	CAD million	20.4
Total Net Revenue	CAD million	1,412
Total Project EBITDA	CAD million	461
Average Life of Mine Cash Costs (mine gate) – Excluding Royalties	CAD/t	400
Average Life of Mine Cash Costs (FOB Port of Montreal) – Excluding Royalties	CAD/t	469
Average Life of Mine Cash Costs (FOB Port of Montreal) – Excluding Royalties	USD/t	356
Net Present Value (real terms @ 8% discount rate)	**CAD million**	**216**
Pre-tax Internal Rate of Return	**%**	**33.9**
Project Payback Period (After Start of Production)	year	2.7
Exchange Rate	CAD:USD	0.76

26.2 Principal Assumptions

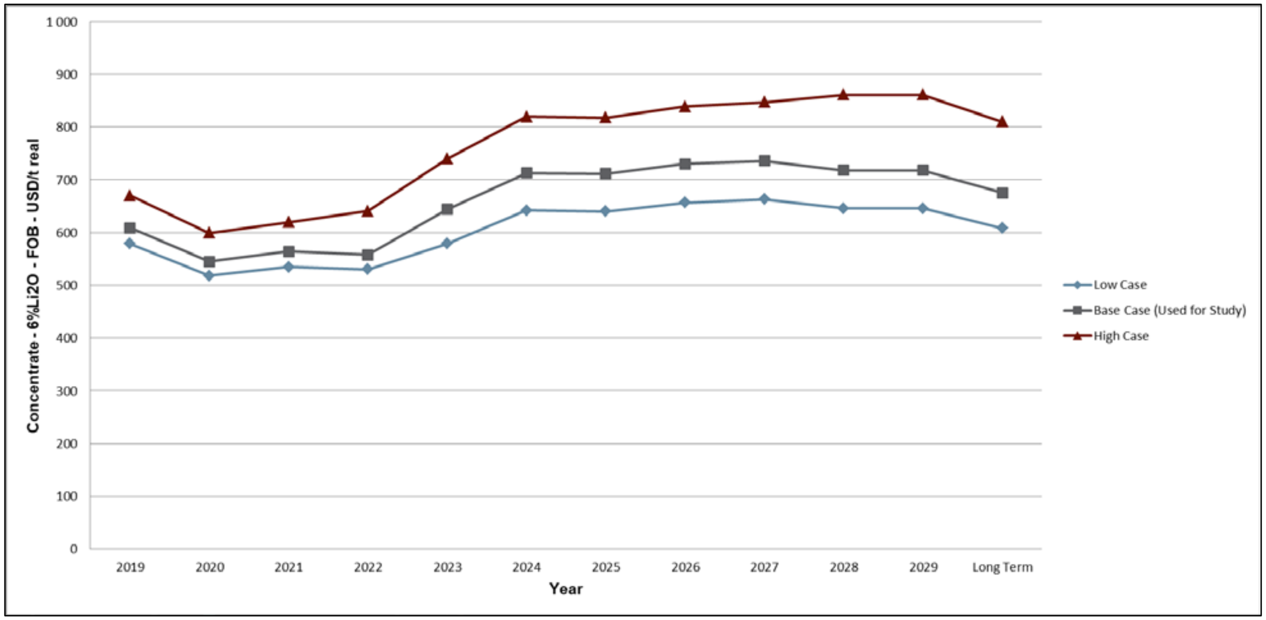
The following table provides the principal assumptions utilized in the financial model.

Table 26-2: Key Financial Model Assumptions

Assumption	Value
Exchange Rate (CAD:USD)	0.76
Discount Rate	8%
Pre-Production Duration	4 months
First Production Quarter	Q1 2023
Ramp up Period	3 months
Transport Losses	0.50%
Fuel Price	0.971 $/L
Concentrate Humidity (%wt./wt.)	6.5%

26.3 Pricing

Concentrate pricing is a key assumption within the financial model; for this study Hatch developed spodumene price forecasts using an internal methodology which was then validated against off take and analyst prices. The following graph and table detail the analyst pricing forecasts used to generate the average price forecast for the purposes of the financial model.



	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	Long Term
Low Case	579	518	535	530	579	642	640	657	663	646	646	608
Base Case (Used for Study)	609	545	564	558	644	713	712	730	736	718	718	675
High Case	670	599	620	641	740	820	818	839	847	861	861	810

Figure 26-1: Pricing Forecast Chart (USD, Real)

26.4 Cash Flow Forecasts

The following figure and table summarize the cash flow forecasts from the financial model.

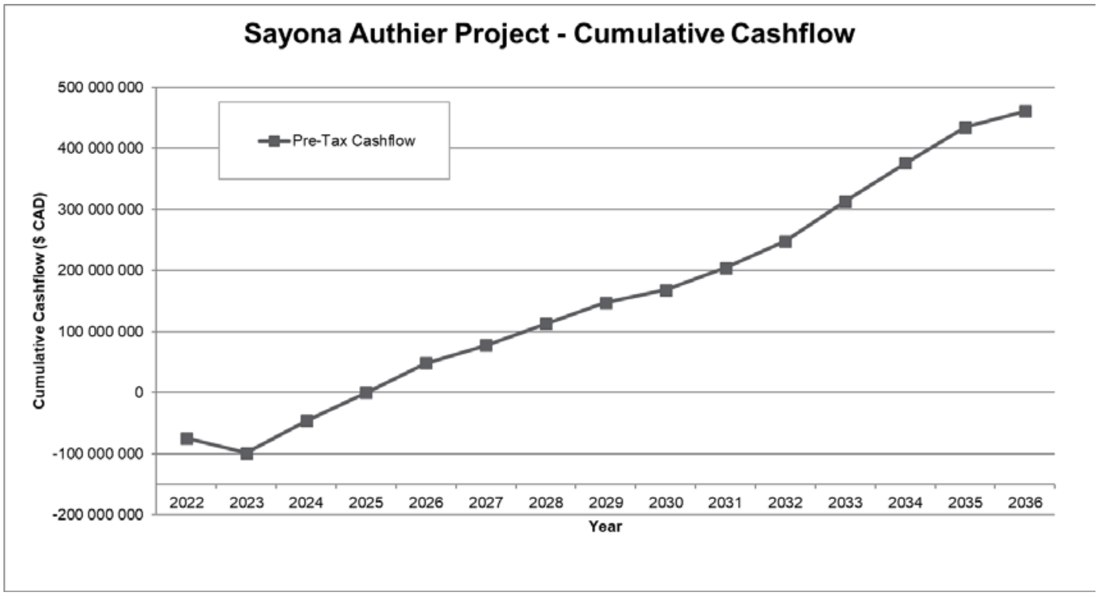


Figure 26-2: Non-discounted Pre-tax Cumulative Cashflow

Table 26-3: Annual Cash Flow Forecasts

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	LOM Total / Average
Discounted Years	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	
Calender Year	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	
Mine Production Summary																
Total Moved (Expit + Rehandle) (t)	**794 399**	**3 599 408**	**3 957 890**	**6 625 448**	**9 617 475**	**12 582 355**	**14 657 049**	**14 661 927**	**12 439 533**	**9 622 584**	**6 636 305**	**3 183 456**	**2 396 198**	**2 116 354**	**1 717 698**	**104 608 081**
Total Expit (t)	**794 399**	**3 020 220**	**3 295 961**	**5 963 521**	**8 955 547**	**11 920 428**	**13 995 121**	**14 000 000**	**11 777 606**	**8 960 657**	**5 974 378**	**2 521 528**	**1 734 271**	**1 454 426**	**1 168 068**	**95 536 131**
Expit Waste Rock (t)	537 591	2 067 574	1 391 299	4 141 719	7 322 475	10 429 138	11 672 493	13 097 818	10 895 036	8 078 087	5 091 808	1 638 958	851 701	571 856	435 228	**78 222 781**
Expit Overburden (t)	256 808	180 395	1 022 091	939 232	750 502	608 720	1 440 059	19 612	-	-	-	-	-	-	-	**5 217 417**
Expit Ore to Mill (t)	-	193 063	220 643	220 643	220 643	220 643	220 642	220 643	220 643	220 643	220 643	220 643	220 643	220 643	183 210	**3 023 983**
Expit Ore to ROMpad (t)	-	579 188	661 929	661 928	661 928	661 928	661 927	661 928	661 928	661 928	661 928	661 928	661 928	661 928	549 630	**9 071 950**
Expit Ore (t)	-	772 251	882 572	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	732 840	**12 095 933**
Expit Ore Grade (% Li_2O)	-	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	**1.00**
Stripping Ratio	**-**	**2.91**	**2.73**	**5.76**	**9.15**	**12.51**	**14.86**	**14.86**	**12.34**	**9.15**	**5.77**	**1.86**	**0.97**	**0.65**	**0.59**	**6.90**
Total Stockpile Rehandling	**-**	**579 188**	**661 929**	**661 928**	**661 928**	**661 928**	**661 927**	**661 928**	**661 928**	**661 928**	**661 928**	**661 928**	**661 928**	**661 928**	**549 630**	**9 071 950**
ROM pad to Mill (t)	-	579 188	661 929	661 928	661 928	661 928	661 927	661 928	661 928	661 928	661 928	661 928	661 928	661 928	549 630	**9 071 950**
ROM pad to Mill Grade (% Li_2O)	-	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	**1.00**
Ore sent to Crusher (t)	-	772 251	882 572	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	732 840	**12 095 933**
Crusher Feed Grade (% Li_2O)	-	1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	**1.00**
Process Plant Production Summary																
Crusher Feed (t - Dry)		772 251	882 572	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	882 570	732 840	**12 095 933**
Crusher Feed Grade (% Li_2O)		1.03	0.95	0.95	1.05	0.91	1.06	1.04	0.90	1.03	1.02	1.15	1.07	1.00	0.86	**1.00**
Spodumene Concentrate Grade (%Li_20)		6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	**6.0**
Li_2O Recovery (%)		78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	**78.0**
Spodumene Concentrate (t - Dry)		103 408	109 056	108 451	120 839	104 699	121 872	119 769	103 340	118 032	116 871	132 023	122 521	115 031	82 351	**1 578 264**
Spodumene Concentrate (t - Wet)		110 597	116 638	115 990	129 240	111 978	130 344	128 096	110 524	126 237	124 996	141 201	131 039	123 027	88 076	**1 687 983**
Tailings (t - Dry)		668 843	773 516	774 119	761 731	777 871	760 698	762 801	779 230	764 538	765 699	750 547	760 049	767 539	650 489	**10 517 669**
Tailings (t - Wet)		760 049	878 995	879 681	865 603	883 944	864 430	866 819	885 489	868 794	870 113	852 894	863 692	872 204	739 192	**11 951 896**
Revenue																
Exchange Rate (USD:CAD)		0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	**0.76**
Spodumene Concentrate Price (USD/t)		644	713	712	730	736	718	718	675	675	675	675	675	675	675	**693**
Spodumene Concentrate Price (CAD/t)		847	938	937	961	968	945	945	888	888	888	888	888	888	888	**911**
Spodumene Concentrate Sales ($)[1]		87 186 654	101 800 572	101 093 452	115 489 114	100 885 940	114 561 198	112 584 749	91 323 368	104 306 472	103 280 871	116 671 186	108 273 922	101 654 578	72 775 116	**1 431 887 192**
Royalty Deduction ($)		1 098 363	1 656 572	1 300 252	1 180 255	1 805 248	1 953 237	1 971 015	985 246	1 021 467	1 114 788	2 466 658	1 622 956	1 430 996	764 128	**20 371 179**
Total Revenue ($)	**-**	**86 088 291**	**100 144 001**	**99 793 200**	**114 308 859**	**99 080 692**	**112 607 961**	**110 613 734**	**90 338 122**	**103 285 005**	**102 166 082**	**114 204 528**	**106 650 966**	**100 223 583**	**72 010 988**	**1 411 516 012**
Operating Expenditures																
Open Pit Mining ($)	-	11 091 899	13 494 854	17 112 808	24 861 706	32 053 388	38 046 728	36 068 709	34 999 679	30 494 636	23 791 934	13 080 126	10 226 674	9 004 104	7 923 476	**302 250 721**
Mineral Processing ($)		13 676 565	15 630 350	15 630 315	15 630 315	15 630 315	15 630 314	15 630 315	15 630 315	15 630 315	15 630 315	15 630 315	15 630 315	15 630 315	12 978 594	**214 218 971**
On Site Laboratory		813 002	929 145	929 143	929 143	929 143	929 143	929 143	929 143	929 143	929 143	929 143	929 143	929 143	771 512	**12 734 232**
Process Plant Mobile Equipment ($)	-	899 382	899 382	899 382	899 382	899 382	899 382	899 382	899 382	899 382	899 382	899 382	899 382	899 382	823 754	**12 515 722**
Water Treatment ($)		1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	1 148 927	**16 084 984**
Tailings and Water Management ($)		386 126	441 286	441 285	441 285	441 285	441 285	441 285	441 285	441 285	441 285	441 285	441 285	441 285	366 420	**6 047 966**
General and Administration (G&A) ($)		4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	4 789 673	**67 055 418**
Total Onsite Operating Costs ($)	**-**	**32 805 575**	**37 333 617**	**40 951 533**	**48 700 431**	**55 892 112**	**61 885 453**	**59 907 434**	**58 838 404**	**54 333 361**	**47 630 659**	**36 918 851**	**34 065 399**	**32 842 829**	**28 802 356**	**630 908 015**
Product Transport and Logistics Costs ($)[2]		7 111 702.90	7 500 153	7 458 517	8 310 509	7 200 494	8 381 516	8 236 915	7 107 021	8 117 400	8 037 585	9 079 654	8 426 157	7 911 023	5 663 548	**108 542 196**
Total Operating and Shipping Costs ($)	**-**	**39 917 278**	**44 833 771**	**48 410 050**	**57 010 940**	**63 092 606**	**70 266 969**	**68 144 349**	**65 945 425**	**62 450 761**	**55 668 244**	**45 998 505**	**42 491 556**	**40 753 852**	**34 465 904**	**739 450 210**
Capital Expenditures	Pre-Production							Sustaining								
Mine Preproduction ($)	3 246 946		-	-	-	-	-	-	-	-	-	-	-			**3 246 946**
Mine Equipment- Financed ($)	912 173	1 817 450	1 800 496	3 237 816	4 111 996	4 763 330	6 206 422	5 374 484	4 052 627	3 880 140	2 928 617	2 024 731	1 276 397	100 147	-	**42 486 823**
Mine Equipment- Purchased ($)	1 281 667	258 000	103 000	248 668	103 000	-	206 000	1 198 556	-	-	-	-	401 556	-	-	**3 800 447**
Process Plant Mobile Equipment ($)	-	1 156 500	-	-	-	-	-	1 156 500.0	-	-	-	-	-	-	-	**2 313 000**
Process Plant and Infrastructure ($)	69 013 150	46 008 766														**115 021 916**
Remaining Site Preparation Activities ($)		9 780 400	-	-	-	-	-	-	-	-	-	-	-	-	-	**9 780 400**
Building Capital Rental ($)	-	1 022 027	768 445	768 445	768 445	768 445	563 301									**4 659 107**
Tailings and Water Management Infrastructure ($)		-	-	-	10 438 200	-	-	-	-	-	-	-	-	-	-	**10 438 200**
Water Treatment Plant - Capital Rental ($)		218 400	218 400	218 400	218 400	584 650	-	-	-	-	-	-	-	-	-	**1 458 250**
Wetland Compensation ($)		271 466	271 466	271 466	264 242	294 640	259 760	168 240	161 200	248 000	330 840	333 372	333 372	444 972	418 484	**4 071 520**
Royalties buyback ($)		3 000 000														**3 000 000**
Salvage Value ($)																**-**
Reclamation and Closure ($)	-														10 875 166	**10 875 166**
Total Capital Costs ($)	**74 453 935.75**	**63 533 010**	**3 161 807**	**4 744 795**	**15 904 283**	**6 411 065**	**7 235 482**	**7 897 780**	**4 213 827**	**4 128 140**	**3 259 457**	**2 358 103**	**2 011 325**	**545 119**	**11 293 650**	**211 151 776**
Working Capital ($M)		6 652 879.62			(6 652 880)											**-**
Pre-Tax Cash Flow																
Pre-Tax Cash Flow ($M)	(74 453 936)	(24 014 876)	52 148 423	46 638 356	48 046 516	29 577 020	35 105 510	34 571 606	20 178 870	36 706 104	43 238 382	65 847 921	62 148 085	58 924 612	26 251 434	**460 914 026**
Cumulative Pre-Tax Cash Flow ($)	(74 453 936)	(98 468 811)	(46 320 389)	317 967	48 364 483	77 941 503	113 047 013	147 618 619	167 797 489	204 503 593	247 741 975	313 589 895	375 737 980	434 662 593	460 914 026	

26.5 Net Present Value - Internal Rate of Return and Payback Period

The following financial metrics have been calculated from the financial model:

1. Non-Discounted Cashflow (pre-tax) – CAD460.9M;
2. Net Present Value at 8% discount rate (pre-tax) – CAD215.6M;
3. Internal rate of return (pre-tax) – 33.9%;
4. Simple payback period (pre-tax) – 4.0 years (after start of project); and
5. Payback period (pre-tax) – 2.7 years (after start of production).

26.6 Taxes

The taxation schedule and resulting after-tax financial analysis has not been evaluated during this study. All values presented in this report are on a pre-tax basis.

26.7 Royalties

Royalties have been calculated in accordance with the tables presented in section 4.3 and assumed the purchase of some of the NSR (refer to section 24.8.2), resulting in a weighted average royalty rate of 1.53% of Net Sales Revenue (NSR).

26.8 Sensitivity Analysis

A sensitivity analysis has been performed on key financial model inputs. The results of the sensitivity analysis are presented in the following tables and figures, and are on a pre-tax basis.

Table 26-4: Sensitivity Tables (CAD M)

Total CAPEX	Variance	-30%	-20%	-10%	**Base Case**	10%	20%	30%
	Initial Capital Cost (M$)	$148 M	$169 M	$190 M	**$211 M**	$232 M	$253 M	$274 M
	Cumulative Cash Flow (M$)	$524 M	$503 M	$482 M	**$461 M**	$440 M	$419 M	$398 M
	Pre-tax IRR (%)	54.6%	46.0%	39.3%	**33.9%**	29.5%	25.8%	22.7%
	Pre-tax NPV (8%) (M$)	**$269 M**	**$251 M**	**$233 M**	**$216 M**	**$180 M**	**$180 M**	**$163 M**

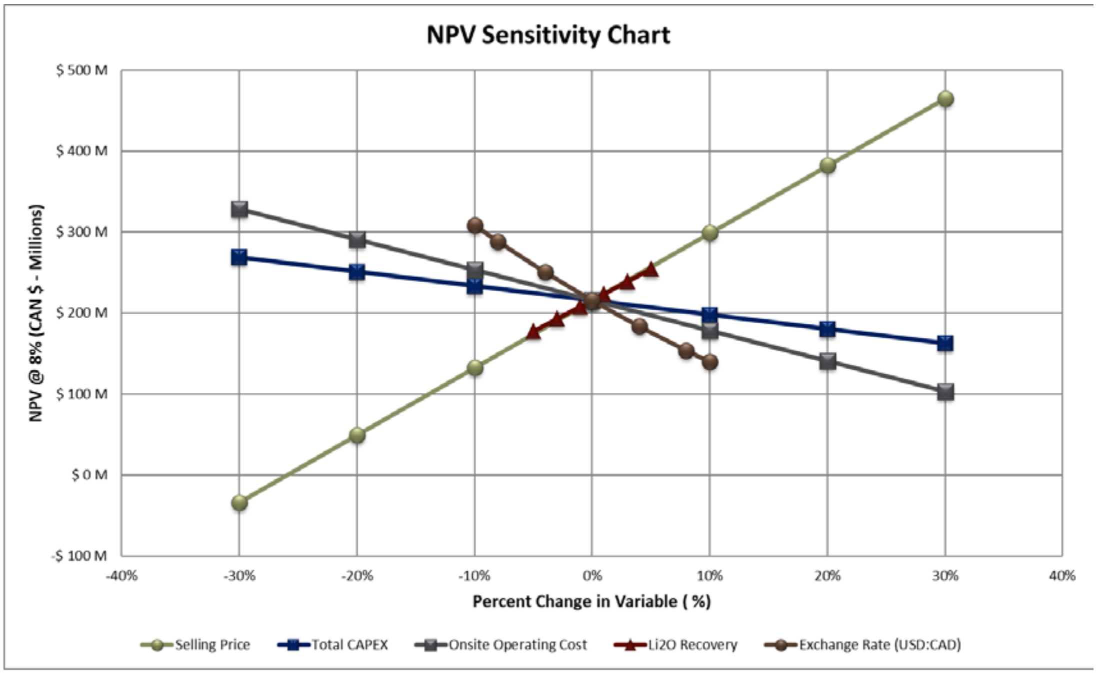
Onsite Operating Cost	Variance	-30%	-20%	-10%	**Base Case**	10%	20%	30%
	Operating Cost ($/t conc.)	279.82	319.80	359.77	**399.75**	439.72	479.70	519.67
	Operating Cost (M$)	$442 M	$505 M	$568 M	**$631 M**	$694 M	$757 M	$820 M
	Cumulative Cash Flow (M$)	$650 M	$587 M	$524 M	**$461 M**	$398 M	$335 M	$272 M
	Pre-tax IRR (%)	47.0%	42.7%	38.3%	**33.9%**	29.3%	24.6%	19.7%
	Pre-tax NPV (8%) (M$)	**$328 M**	**$291 M**	**$253 M**	**$216 M**	**$178 M**	**$140 M**	**$103 M**

Selling Price of Li_2O Concentrate (USD)	Variance	-30%	-20%	-10%	**Base Case**	10%	20%	30%
	Conc. Price ($US/t)	339	443	561	**693**	838	997	1170
	Cumulative Cash Flow (M$)	$37 M	$179 M	$320 M	**$461 M**	$602 M	$743 M	$884 M
	Pre-tax IRR (%)	0.4%	12.7%	23.6%	**33.9%**	43.9%	53.8%	63.7%
	Pre-tax NPV (8%) (M$)	**-$34 M**	**$49 M**	**$132 M**	**$216 M**	**$299 M**	**$382 M**	**$465 M**

Li_2O Recovery	Variance	-5%	-3%	-1%	**Base Case**	1%	3%	5%
	Recovery	74%	76%	77%	**78%**	79%	80%	82%
	Cumulative Cash Flow (M$)	$396 M	$422 M	$448 M	**$461 M**	$474 M	$500 M	$526 M
	Pre-tax IRR (%)	29.2%	31.1%	33.0%	**33.9%**	34.8%	36.7%	38.5%
	Pre-tax NPV (8%) (M$)	**$177 M**	**$193 M**	**$208 M**	**$216 M**	**$223 M**	**$239 M**	**$254 M**

Exchange Rate (USD:CAD)	Variance	-10%	-8%	-4%	**Base Case**	4%	8%	10%
	Exchange Rate	0.72	0.74	0.75	**0.76**	0.77	0.78	0.80
	Cumulative Cash Flow (M$)	$618 M	$584 M	$520 M	**$461 M**	$407 M	$356 M	$333 M
	Pre-tax IRR (%)	45.0%	42.6%	38.1%	**33.9%**	30.0%	26.3%	24.5%
	Pre-tax NPV (8%) (M$)	**$308 M**	**$288 M**	**$250 M**	**$216 M**	**$184 M**	**$154 M**	**$140 M**

Discount Rate	Variance	0	4	6	**8**	10	12
	Pre-tax NPV (8%) (M$)	**$461 M**	**$313 M**	**$260 M**	**$216 M**	**$179 M**	**$149 M**



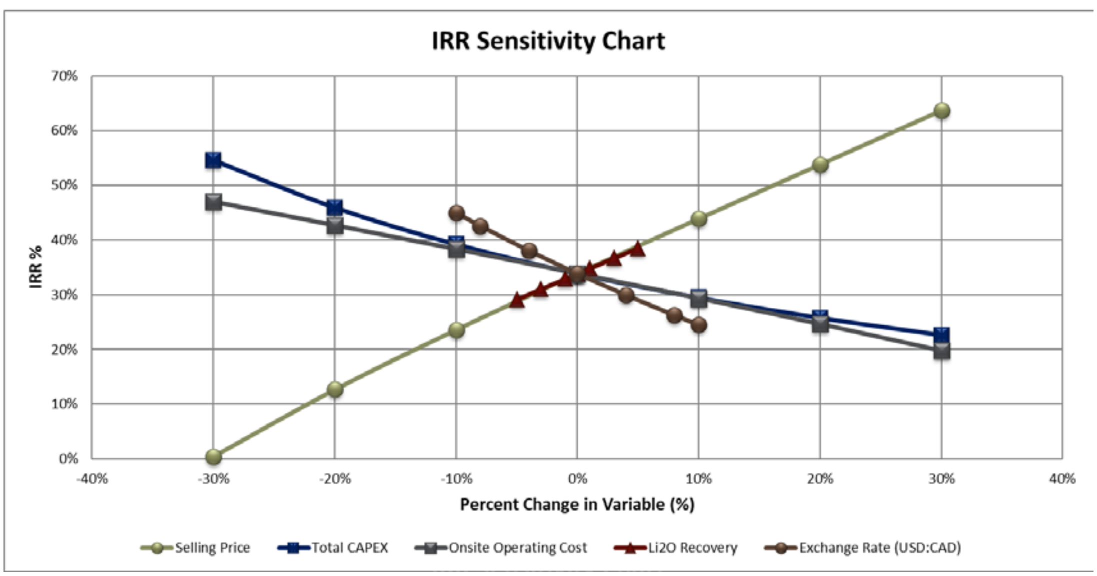


Figure 26-3: Sensitivity Charts

28. RISK AND OPPORTUNITY

A project risk assessment was undertaken to assess the strengths and weaknesses of the technical and commercial viability of delivering the Authier business plan as outlined in the DFS. As with any mining project, there are risks associated with the development, commissioning and operation of a mine. The main risk areas include:

- Financial
- Organizational
- Resource
- Geology
- Mining
- Processing
- Environmental
- Design
- Construction
- Legal
- Community
- Transportation
- Sales (securing off-take or sales contracts)
- Technological

A high-level project risk assessment has been completed. The risk assessment identifies risks, impact category and a mitigation plan. The likelihood, impact, controls and measures were developed for the identified risks. The assessment is necessarily subjective and qualitative.

The following tables summarize the risks and opportunities identified during the DFS. Many risks are common to all mining projects and can be managed through proper planning, engineering and management. The risk and opportunities registers should be reviewed and updated at each stage of the project to reduce uncertainties and de-risk the project.

Table 28-1: Project Risk Assessment

Risk Details				Mitigation
Risk	**Category**	**Item**	**Impact Category**	**Actions**
1.1	Financial	Lack of supplier support/availability	Financial	Appropriate commercial terms and establishment of contracts
1.2	Financial	Delay in product payment affect cash flow	Financial	On-going management of production schedules, forecast cash flows
1.3	Financial	Exchange rate fluctuations	Financial	Appropriate hedging/sales strategy
1.4	Financial	Insurance adequacy	Financial	Appropriate insurance advice/risk assessment
1.5	Financial	Variability in market price of products	Financial	Appropriate sales strategy including contracting arrangements
1.6	Financial	Operational and financial model errors	Financial	Peer review
1.7	Financial	Fuel and freight price increases	Financial	Long terms contracts
1.8	Financial	Supplier cost increases	Financial	Long terms contracts
1.9	Financial	Lack of working capital prior to revenue	Financial	Appropriate debt, equity and working capital funding
1.10	Financial	Revenue variation due to inaccurate resource model	Financial	Appropriate modelling and QA/QC
2.1	Organization	Inadequate ERP system	Schedule	Dedicated personnel to create systems of work
2.2	Organization	Loss, or recruitment of key staff / operators	Schedule	Acceptable site conditions, appropriate remuneration and good working environment
2.3	Organization	Lack of appropriate human resources	Schedule	Acceptable site conditions and remuneration, good working environment
2.4	Organization	Lack of overall project plan	Schedule	Adequate management team, on-going development of project plan
2.5	Organization	Inadequate OHS systems	OHS	Dedicated OHS personnel, management support and systems creation
2.6	Organization	Personnel unable to reach site due to weather	Schedule	All weather access to site
2.7	Organization	Fatigue	OHS	Travel policy, fatigue management plan
3.1	Resource	Resource - reserve reconciliation	Reserve	Higher drilling density - ore control
3.2	Resource	Ore grade	Reserve	Higher drilling density - ore control
3.3	Resource	Dilution	Reserve	Higher drilling density - ore control

Risk Details				Mitigation
Risk	**Category**	**Item**	**Impact Category**	**Actions**
4.1	Geology	Ore grade	Reserve	Higher drilling density - ore control
4.2	Geology	Ore width - geometry	Reserve	Higher drilling density - ore control
4.3	Geology	Structure	Reserve	Higher drilling density - geotechnical control
5.1	Mining	Mining rate limit vs processing capacity it will be very difficult to make up for lost production in the event of unplanned downtime due to ore mining rate limitations.	Financial	Proper planning of downtime to minimize downtime of mining and plant activities
5.2	Mining	Pit Slope Stability Geotechnical uncertainty related to pit slope stability may impact the mine design and operational requirements	Schedule	Review geotechnical report and implement a slope monitoring and assessment program
5.3	Mining	Co-deposition of tails & waste rock risk to mill production if the co-deposition of waste and tails materials is more difficult than expected.	Operations	Develop detailed waste management operations plan. Ensure synchronization between tails deposition plan and mine plan allows for a minimum of flexibility
5.4	Mining	Selective mining requirements in ore with respect to dilution management. Selective mining at ore/waste contact may prove more complex than expected, resulting in increased production costs and possible risk to final product quality.	Processing	Put in place well defined grade control plan with an emphasis on ore selectivity requirements.
5.5	Mining	Geological uncertainty with respect to waste intrusions in pegmatite. Further work required to define and understand the impact of the waste intrusions in the pegmatite. May impact ore reserves.	Reserve	Perform additional infill drilling where intrusions have been identified to better define the interpretation of these zones. Review core logs and update geological interpretation using implicit modelling.
5.6	Mining	Availability of experienced technical personnel for a lean management team. Currently there is a shortage of skilled technical staff in the region. Attracting skilled labour may be more costly and difficult than anticipated.	HR	Begin filling key staff positions as early as possible. Ensure that the right people are hired (avoid the "warm body in a seat" approach).

Risk Details				Mitigation
Risk	Category	Item	Impact Category	Actions
5.7	Mining	Availability of skilled mine operations personnel. Currently there is a shortage of skilled operations personnel in the Abitibi region. Attracting skilled labour may be more costly and difficult than currently anticipated.	HR	Evaluate options of contract mining for operations to mitigate this risk. Hire key operations management / supervisory roles as early as possible.
6.1	Processing	Variance in expected concentrate grade (Li or Fe) and/or recovery	Financial	Continued testing and diligent equipment selection, close communication with the mine/geologists concerning feed characteristics / dilution
6.2	Processing	Adequate supply of reagents	Project delay	Timely discussions with distributors / manufacturers and reagent testing from various sources
6.3	Processing	Adequate supply of parts	Financial	Inventory management system, sourced from local suppliers, procurement systems
6.4	Processing	Mechanical failures, low availability	Financial	Preventative maintenance program and available spare parts, training
6.5	Processing	Delays in commissioning	Schedule	Operational readiness program
6.6	Processing	Slower than anticipated ramp-up due to lack of skilled operators	Schedule	Training programs, begin recruitment early, appropriate and timely commissioning and start-up planning
6.7	Processing	Overgrinding and high lithium losses to slime	Financial	Operator training on crushing/grinding circuits, equipment selection
6.8	Processing	Process water quality	Operations	Difficult to assess the impact of water recycling within the plant. Implement a system to monitor process water quality
6.9	Processing	Availability for 'fresh' water to feed the plant	Operations	Review availability and sources of water (throughout the year) to be used to feed the plant
6.10	Processing	Tailings and concentrate moisture content	Operations	Training programs: overgrinding, water control, proper filter operation
6.11	Processing	Plant water balance	Operations	Review plant water balance prior to commencement of detailed engineering based on water requirements for specific vendor equipment
6.12	Processing	In-plant maintenance work	Operations	Perform a detailed layout review ensuring adequate space for maintenance / operation work
6.13	Processing	On-site reagent storage	Operations	Review quantities of each reagent on-site / in storage during detailed design
6.14	Processing	Concentrate / tailings filtration	Operations	Review filtration area layout / operability

Risk Details				Mitigation
Risk	**Category**	**Item**	**Impact Category**	**Actions**
7.1	Environment	Spring freshet requires temporary water storage in the pit and may affect productivity	Environment	Identify locations in the pit with enough capacity to accommodate the precipitation. Increase pumping capacity from the BC-2 to BS but with partial treatment (as WTP capacity is limited). Explore direct pumping from the pit to the collecting basin (BS) with treatment plant
7.2	Environment	Process water purging resulting in fine suspended solids in the water collection and treatment system which do not settle and may contain other contaminants (e.g., reagent)	Environment	The purge water can be transported out of the site by truck tanker. The purge water can be sent to an intermediate collecting basin and the particles can be partially sedimented at this basin. Additional treatment modules can be added to the current WTPs
7.3	Environment	There might be other contaminants in the mine water due to leaching (such as heavy metal)	Environment	Co-disposal will reduce infiltration and reduce leaching. Additional treatment modules can be added to the current WTPs
7.4	Environment	Other contaminants in the mine water over discharge limit due to explosives (ammonium nitrate)	Environment	Appropriate explosive management and best practice in blasting and appropriate water treatment
8.1	Design	Lack of geotechnical information resulting in an underestimation of foundation requirements for mill building and other foundation-sensitive elements.	Environment	Comprehensive geotechnical drilling program and confirmation of site selection
8.2	Design	Lack of geotechnical information resulting in an underestimation of road and pad construction requirements.	Environment	Comprehensive geotechnical drilling program and confirmation of site selection
8.3	Design	Off-site road work	Environment	Review assumptions for earthworks and design
8.4	Design	Control systems	Financial	An allowance was included for control systems. Detailed design and reviews required.
9.1	Construction	Delay in site works due to weather	Schedule	Project execution plan built on appropriate activities for the given season
9.2	Construction	Damage to equipment during move	Financial	Oversight of equipment delivery required as part of procurement activities
9.3	Construction	Construction defects	Financial	Oversight of equipment delivery required as part of procurement activities
9.4	Construction	Execution does not achieve objectives	Schedule	Planning must be a pro-active and on-going activity

Risk Details				Mitigation
Risk	**Category**	**Item**	**Impact Category**	**Actions**
9.5	Construction	Geotechnical conditions unfavourable on site.	Financial	Comprehensive geotechnical drilling program and confirmation of site selection
9.6	Construction	Capital cost escalation during execution.	Financial	Continue to refine the budget as part of basic and detailed engineering
9.7	Construction	Inadequate allowances made for the minimal road construction required for start of mining operations	Financial	Complete road construction as early as possible and transition as quickly as possible from contractor to mining personnel to complete road construction to optimize costs
10.1	Legal	Mining lease not yet granted	Schedule	Follow the appropriate permitting procedures
10.2	Legal	Delays in obtaining permits and CoA	Schedule	Proper planning of permitting activities
11.1	Community	Issues related to Indigenous relations	Social	Continuous discussions with communities and signing of agreement
11.2	Community	Community issues (e.g., noise, roads.)	Social	Continue community and stakeholder consultation program
12.2	Transport	Port labour disruptions / work stoppages	Financial	Follow labour relations closely. Determine options for alternate transportation methods / routes.
12.2	Transport	Material spill during transport	Financial	Appropriate response and clean-up procedures in place prior to shipments. Close contact with shipping companies.
12.3	Transport	Delays in shipment	Financial	Detailed communication plan with the transportation provider
12.4	Transport	Traffic congestion / safety onsite	Safety	Perform a traffic study and review site layout based on anticipated vehicular/pedestrian traffic
13.1	Sales	Securing sales contracts for concentrate	Financial	Continue approaching North American and Chinese converters to test Authier concentrates and discuss product concentrate specifications. Sign MOU’s and discuss contract specifications to convert into Binding Offtake Contracts (BOC). Potentially engage with trading houses with existing connections to multiple end-users to de-risk the demand side, and potentially extend coverage to lithium chemicals for a future integrated refinery project.
13.2	Sales	Securing suitable pricing arrangements in the sales contract with downside protection	Financial	Complete a study assessing the different forms of contractual pricing arrangements that could agree in a BOA, i.e., cap/collar, escalator/de-escalator clauses
13.3	Sales	Transport logistics and costs associated with selling concentrate overseas	Financial	Re-assess logistics and costs of transporting material to overseas converters during the detailed engineering phase

Risk Details				Mitigation
Risk	**Category**	**Item**	**Impact Category**	**Actions**
13.4	Sales	Potential to value add concentrates in country into lithium carbonate/hydroxide	Financial	Complete metallurgical testing and PFS/DFS on production of lithium carbonate/hydroxide.
14.1	Technological	Delays in project admissibility (receivability)	Approval	Conduct a HAZID study (hazard identification) to document all hazards and mitigation measures and include the information in the environmental assessment report (EIA).
14.2	Technological	Delays in project admissibility (receivability)	Approval	Prepare a preliminary emergency response plan. Consult with the appropriate parties to document their comments, recommendations.
14.3	Technological	Inadequate design / operational problems	Design	Conduct a HAZOP (hazard and operability) when P&IDs are 75% complete (prior to detailed execution)

Table 28-2: Project Opportunities

Opportunity Details		
Risk	**Category**	**Item**
1.1	Financial	Assess the impacts of various financing scenarios
2.1	Organization	Begin planning to build a strong owner's team for the detailed engineering phase
3.1	Resource	Potentially increase the size of the Mineral Resource by testing extensions of known mineralization along strike at both of the Authier pegmatites, as well as by conversion of Inferred Mineral Resources to Reserves
4.1	Geology	Infill definition drilling within the main resource zone where the mineralisation is not well defined and is currently treated as waste
4.2	Geology	Increase the size of the Mineral Resource at depth by testing the deep extensions of the known mineralisation, especially those located on the west portion of the deposit
5.1	Mining	Assess the impact of high-grading during the first 3 years of operation
5.2	Mining	Assess the option of varying the number of cutbacks
5.3	Mining	Perform a cost trade-off to assess the used and/or larger mining equipment
6.1	Processing	Test various process reagents to ensure the most cost-effective option is selected
6.2	Processing	Evaluate the use of screens in the grinding circuit to minimize overgrinding to optimize energy usage and minimize lithium losses to the slimes
6.3	Processing	Evaluate the effect of water quality on processing
6.4	Processing	Evaluate opportunities to minimize water usage (e.g., low flow gland seal water pumps)
6.5	Processing	Optimize filtration to minimize fresh water usage
7.1	Environment	Assess the possibility of optimizing location of the waste dump in relation to the Saint-Mathieu-Berry esker
7.2	Environment	Complete geochemical characterisation of the waste rock and tailings to confirm whether they are suitable to be used as civil construction materials
7.3	Environment	Optimize water management and design/construct basins and treatment facilities
8.1	Construction	Continue focusing on delivery of turn-key packages from local contractors
8.2	Construction	Optimize excavation/backfill by using existing
8.3	Construction	Develop strategies to maximize use of waste rock as construction materials
9.1	Community	Continue to increase visibility of Sayona in the local community
10.1	Transport	Explore various transportation options
11.1	Sales	Continue approaching North American and Chinese converters to test Authier concentrates and discuss product concentrate specifications. Discuss contract specifications to convert into Binding Offtake Contracts (BOC)

29. ADJACENT PROPERTIES

29.1 General

The area surrounding the Property, which is located between Val-d'Or, Amos and Malartic, is well known for mineral exploration activity, especially for gold, copper and zinc. The Authier property is surrounded by several exploration properties owned by various companies.

The most relevant mineral property in proximity (27 km east) to the Authier Project is the Québec Lithium property owned by North American Lithium Inc. The Québec Lithium property hosts a lithium deposit occurring in a series of spodumene-bearing pegmatite dykes that share strong similarities with the mineralized pegmatite intrusion observed at the Authier property. The pegmatite intrusions at the Québec Lithium property are dipping steeply and are oriented approximately N310°. The reported thicknesses of the dykes range from less than one metre to several tens of metres with a strike extent of several hundreds of metres.

On December 6, 2011, Canada Lithium Corp. announced an updated National Instrument 43-101 compliant Mineral Resource estimate prepared by AMC Mining Consultants. Resources presented were 33,239,000 t at 1.19% Li_2O in the Measured and Indicated category and 13,757,000 at 1.21% Li_2O in the Inferred category, based on a 0.8% Li_2O cut-off grade. This update was ordered due to the fact that Canada Lithium qualified persons were unable to verify the resource estimates from the previous feasibility study (February 28, 2011).

On June 13, 2011, Canada Lithium announced a new proven and probable mineral reserve estimate contained in an updated Feasibility Study prepared by BBA Inc. The estimate was based on 80% ore recovery, a waste dilution factor of 20% at 0.05% lithium oxide and a cut-off grade of 0.6%. The new Ore Reserve estimate is amounting to 17,064,000 t at 1.17% Li_2O in the Proven and Probable categories and based on a cut-off grade of 0.6% Li_2O.

Construction of the mine and process plant began in August 2011 and the production commenced in 2013. At the time, the company had difficulties meeting production targets due to mine dilution and processing issues and decided to cease production in early 2014.

The project was acquired by Jilin Jien Nickel Industry Company in 2016 who announced plans for a phased restart, with a concentrator start-up planned for Q3 2017 and export of concentrate produced to China, followed by modifications to the Lithium Carbonate plant for a restart in early 2018.

The technical data provided in this section was previously taken from the Canada Lithium and the North American Lithium website as of the date of this report. The author of this section was unable to verify any of the information provided under this section. The information and analytical results published by Canada Lithium are not necessarily indicative of the mineralization on Sayona Québec's Authier Lithium property that is the subject of this Technical Report.

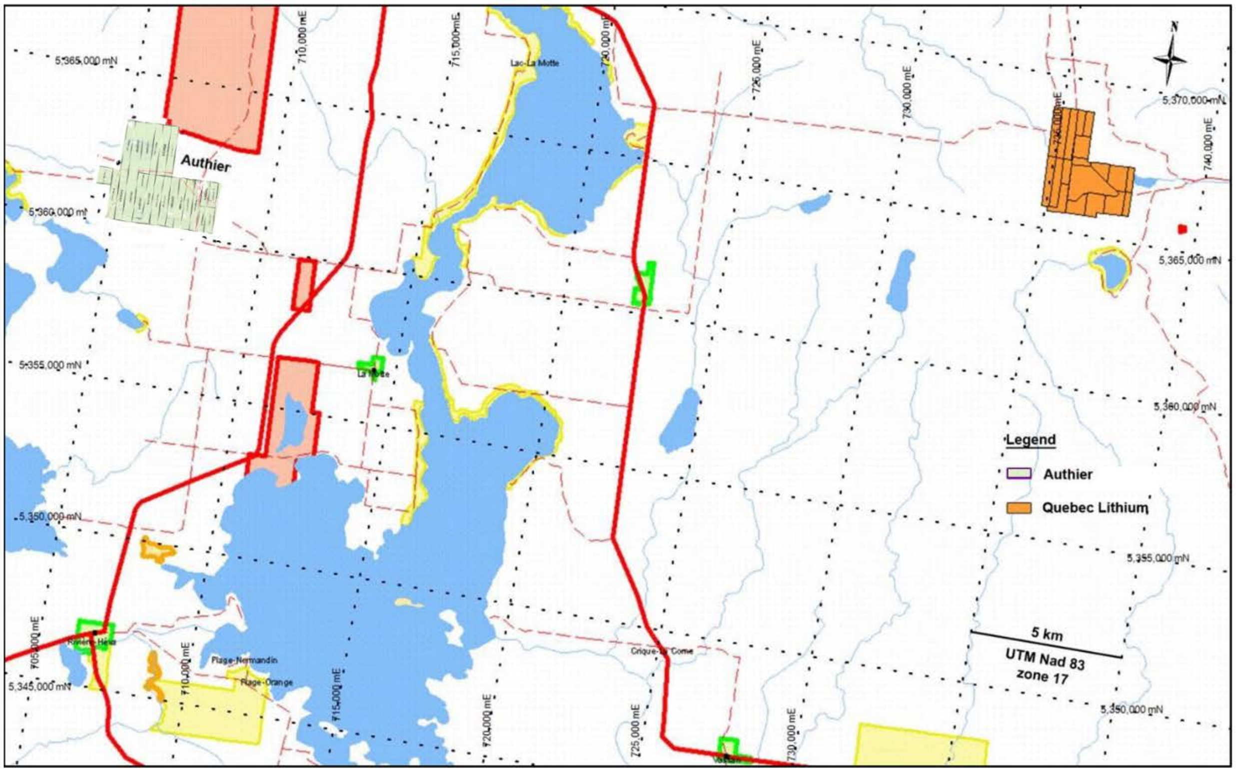


Figure 29-1: Adjacent Properties Map

30. OTHER RELEVANT DATA AND INFORMATION

This section has been included for potential future conversion of this study into a National Instrument 43-101 Technical Report. It is intentionally blank for the purposes of this Definitive Feasibility Study Update.

31. CONCLUSIONS AND RECOMMENDATIONS

31.1 General

The Updated Definitive Feasibility Study (UDFS) includes the expanded JORC resource (2018 DFS), results from a number of technical optimization programs, and realignment of pricing to reflect recent industry forecasts. The UDFS confirms the technical and financial viability of constructing a simple, low-strip ratio, open-cut mining operation and processing facility producing spodumene concentrate. The positive study demonstrates the opportunity to create substantial long-term sustainable shareholder value at a low capital cost.

Given the technical feasibility and positive economic results of the UDFS, it is recommended to continue the work necessary to support a decision to fund and develop the project.

31.2 Recommendations

Based on the results of the UDFS, the following conclusions and recommendations are made:

31.2.1 Exploration and Geology

- Potentially increase the size of the Mineral Resource by testing extensions of known mineralization along strike at both the main Authier pegmatite and Authier North pegmatite, as well as by conversion of Inferred Mineral Resources to Reserves;
- Infill definition drilling within the main resource zone where the mineralization is not well defined and is currently treated as waste;
- Increase the size of the Mineral Resource at depth by testing the deep extensions of the known mineralization, especially those located on the west portion of the deposit.

31.2.2 Mining

- Assess the impact of high-grading during the first 3 years of operation;
- Assess the option of varying the numbers of cutbacks;
- Perform pit optimization sensibility on overall pit slopes, metallurgical recovery and dilution/ore loss;
- Conduct an additional geotechnical assessment to confirm the recommended pit slopes prior to advancing to the next stage of the project.

31.2.3 Waste Dumps and Tailings Management

- Further optimization of the location of the waste dump in relation to the Saint-Mathieu-Berry esker;
- Complete geochemical characterization for the waste rock and tailings to confirm whether they are acceptable as off-site civil construction materials;
- Optimization of the water management plan and design/construction of the water basins and treatment plant.

31.2.4 Environment and Social

- Inform and involve stakeholders as the project advances;
- Continue evaluating the impacts of the project on the environment;
- Design of mitigation measures, if required, to control dust, noise, etc.;
- Increase visibility of Sayona in the region with a local office in La Motte.

31.2.5 Metallurgy and Processing

Testwork Programs

The successful feasibility testwork undertaken in 2018 allowed for optimization of the process flowsheet. Further testwork is recommended to increase our understanding of the process:

1. Comminution / classification:
    a. Further evaluate the use of vibrating screens rather than hydrocyclones in the ball mill circuit to minimize fines production.
2. Flotation (bench-scale tests):
    a. Continue to explore the impact of feed iron content and the relationship between iron removal and lithium losses in magnetic separation;
    b. Test alternate flotation reagents (from various suppliers).
3. De-watering:
    a. Perform vendor testing to confirm thickener and filter sizing.
4. Dilution:
    a. Further testing of contact/transition zone ore to determine recovery / quality issues (previous metallurgical testwork programs have shown the impact of the presence of mine “dilution” on concentrate grade and recovery. The mine currently plans to reject contact zone material).

Processing and Recovery

Further work is recommended related to the processing plant as follows:

1. Crushing:
   a. Confirm modular crushing arrangement and review maintenance plan / equipment required.
2. Flotation:
   a. Review circuit sizing and layout optimization with equipment suppliers;
   b. Plant water balance needs to be refined prior to detailed engineering.
3. Plant layout:
   a. Further work is recommended to perform a detailed plant layout review;
   b. Examine the placement of the concentrate storage building and tailings disposal as related to site traffic.

31.2.6 Infrastructure

During the UDFS, the following elements have been modified or relocated:

1. Waste and tailings piles were modified and relocated further from the Saint-Mathieu-Berry esker.
2. Main access road was changed and enters the site from the north-west of the property in the municipality of Preissac.

The company has committed not to displace any material from the adjacent esker for plant construction. A conceptual site layout plan was developed which includes water management and treatment facilities, traffic management, and infrastructure. All major buildings were located on existing out-crops easily visible from the LIDAR surveys. Preliminary geotechnical studies were undertaken in 2018 after completion of the DFS. Final plant lay-out and water management basin dimensions will be optimized during detailed engineering.

The following recommendations are made related to project infrastructure:

1. Site layout:
   a. Further work is recommended to optimize the site layout and footprint;
   b. All road and pad construction can be appropriately scheduled to maximize the use of mine waste rock from the pit. There is a possibility of using crushing equipment to produce aggregate for the civil construction to lower costs;
   c. Examine extending the plant site by back-filling with waste rock;

   d. Examine a strategy for waste pile management and perimeter ditch construction to be performed by mining operations;
   e. Optimization of the use of waste rock for construction of internal roads and infrastructure areas;
   f. The locations of process and infrastructure areas require further examination and optimization with respect to existing surface levels, prevailing geotechnical conditions, and operational considerations.
2. Geotechnical:
   a. An infrastructure geotechnical investigation is required for design of the plant and water management basins;
   b. Further investigations should include testing in the proposed location(s) of the process and non-process infrastructure (per above), with recommendations on foundation design, soil stability and permeability for water management and water level for excavation provided from the geotechnical consultant (including recommended design factors).
3. Survey:
   a. Further ground-feature surveys are needed for the proposed infrastructure areas including off-site roads and proposed intersection locations.
4. Water management:
   a. Water management (e.g., location of ditches, catchment basin size and water treatment plant location and size) will be optimized during the detailed engineering phase. Basin size must be appropriately dimensioned to include fire water reserve and process make-up water.
5. General infrastructure:
   a. All recommended service infrastructure work should be focused on developing turn-key packages from local contractors to reduce the overall cost. UDFS costs are based on preliminary proposals from local contractors. Further negotiations during the detailed engineering phase with local contractors will allow for cost optimization.

31.2.7 Project Execution

A project execution strategy has been included in the UDFS with a clear separation between detailed engineering and an owner-driven Project Construction Management (PCM) team. The flexibility of a small flexible construction team fits well with the size and scope of the Authier project. Such an approach typically accelerates the construction and start-up timeline and simplifies the construction contract administration process. The UDFS is based on such a contracting strategy with a significant reduction in the overall construction time and on the construction indirect costs.

31.2.8 Financial

- Assess impacts of different financing scenarios;
- Tracking of exchange rate and spodumene concentrate price forecasts.

32. REFERENCES

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