NI 43-101 Technical Report
Preliminary Economic Assessment
Authier Lithium Property
Abitibi, Quebec, Canada
for
Glen Eagle Resources Inc.

Respectfully submitted to:
Glen Eagle Resources Inc.
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Prepared by:
Maxime Dupéré, P. Geo., SGS Canada Inc.
Jonathan Gagné, Eng., SGS Canada Inc.
Gaston Gagnon. Eng., SGS Canada Inc.
Florent Baril, Eng., Bumigeme Inc.

SGS Canada Inc.
Mineral Services
Blainville,Qc

BUMIGEME INC.
Montreal, Qc

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1 Summary

1.1 Property Description and Ownership

The Authier property ("Property") is located approximately 45 km northwest of the city of Val d'Or, Province of Quebec. The Property is accessible by the rural road network connecting to the main highway Route 109 located a few kilometres east which links to the south with Route 117, the provincial highway linking Val d'Or and Rouyn-Noranda. The nearest cities, Malartic, Cadillac, Val d'Or and Rouyn-Noranda, are all mining cities with infrastructures and workforce to easily support a mining operation.

The Property consists in one block of map designated claim cells located at the border between the La Motte Township and the Preissac Township, totalling 19 claims covering 653.57 ha. The Property extends 3.4 km in the east-west direction and 3.1 km north-south. From the 19 claims composing the Property, 3 claims were acquired by staking, 15 claims were acquired through two separate purchasing agreements and one claim is held under an option agreement. Glen Eagle is conducting exploration work under valid intervention permits delivered by the Quebec Government, and there is no known environmental liabilities pertaining to the Property. Some of the claims containing mineral resources are subject to mining royalties (please refer to section 4.2 for details).

1.2 History

The Property has been explored in the 1950's and 1960's for volcanic nickel-copper sulfides mineralisation, and later for lithium mineralisation since the late 1960's with the discovery of a significant spodumene-bearing pegmatite intrusion. The Property saw significant amount of exploration work between 1966 and 1980 with delineation drilling programs from 1991 until 1999 with bulk sampling and metallurgical testing programs.

1.3 Geology and Mineralization

The lithium mineralisation outlined on the Property occurs in a spodumene-bearing pegmatite intrusion interpreted to be genetically derived from the late peraluminous monzongranitic pluton of La Motte located north of the pegmatite. The La Motte pluton is located in the Southern Volcanic Zone of the Abitibi Greenstone Belt and is part of the Archean-age syn- to post- tectonic Preissac-Lacorne batholith which intruded along the La Pause anticline in lithologies of the Malartic Group. The Authier pegmatite crosscut the lithologies of the Malartic Group which is mainly composed of mafic to ultramafic metavolcanics and metasediments.

1.4 Exploration

The Glen Eagle 2010-2012 diamond drilling campaign was preceded by prospecting, geochemical sampling and geophysical surveys that covered the Property targeted areas. This work confirmed the presence of several pegmatite occurrences across the Property having a similar geochemical signature to the main Authier pegmatite.

1.5 Drilling

In 2010, Glen Eagle secured the mining rights and completed exploration work as well as 1,905 m of diamond drilling totalling 18 holes targeting the deposit. During 2011, Glen Eagle drilled a total of 4,051 m mainly on the Authier pegmatite deposit and other areas. In 2012, Glen Eagle drilled a total of 3,034 m mainly on the Authier Pegmatite deposit and other areas.

Period	Drill Holes Series	Number of Holes	Metres Drilled	Number of Survey Record	Number of Lithological Record	Number of Assays Record	% Assayed Metres
Historical	AL-XX	21	2,375	90	737	413	25%
	R-93-XX	33	3,700	71	178	258	18%
Glen Eagle Res.	AL-10-XX	18	1,905	73	171	582	41%
	AL-11-XX	27	4,051	93	137	462	17%
	AL-12-XX	24	3,034	84	126	428	21%
Total		123	15,065	411	1,349	2,143	22%

1.6 Mineral Resource Estimate

The mineral resource block model has been interpolated from 3 m long analytical composite data constrained within a 3D wireframe envelop of the mineralized pegmatite defined from drill hole geological interpretation as well as the mineralised intercepts meeting the required criteria. The mineral resource model is defined by block 5 m (east-west) by 5 m (north-south) by 5 m (elevation) in size, located below the bedrock/overburden interface. The deposit covers over 825 m on the EW direction. The average thickness is 25m with a minimum of 4 m and a maximum of 55 m. The deposit is dipping at 40° towards the N. The Authier pegmatite dip increases at depth of 100 m deep in average mostly to the eastern part of the deposit. The pegmatite deposit is modelled to a maximal depth of 220 m below surface. The interpolation of the block grade was performed by ordinary kriging (OK) in multiple passes using anisotropic search ellipsoids increasing in size from one pass to another. The final mineral resources correspond to the estimated blocks located below the bedrock/overburden interface within the modeled envelop and net of un-mineralised material located within the modeled pegmatite envelop. The updated mineral resources were finally classified into measured, indicated and inferred categories using an automated classification process taking into account the availability of a relevant number of nearby composites within a defined search ellipsoid to produce coherent mineral resource categories. The global mineral resources, without the open-pit limitation, using a base case cut-off grade of 0.5% Li_2O are amounting to 2,244,000 tonnes grading 0.95% Li_2O in the measured category, 5,431,000 tonnes grading 0.97 % Li_2O in the indicated category with an additional 1,552,000 tonnes grading 0.96% Li_2O in the inferred category. The next table summarises the global mineral resources for the Authier Project Property with a cut-off grade of 0.5 % Li_2O (base case).

GLOBAL Mineral Resource Estimate - Authier Property				
Cut-off Grade Li_2O (%)	Resources Categories	Tonnes*	Li2O Grade (%)	Li Metal* (tonne)
0.50%	Measured (M)	2,244,000	0.95	8,500
	Indicated (I)	5,431,000	0.97	21,000
	Total (M+I)	**7,675,000**	**0.96**	**29,500**
	Inferred	1,552,000	0.96	5,900

Effective date october 26, 2012

Mineral resources are not mineral reserves and do not have demonstrated economic vialitity

Bulk dansity of 2.71 t/m3 used. *Rounded to the nearest thousand

1.7 Metallurgical Testing

- **Test realized in 1999**

COREM conducted metallurgical testing during 1999 of approximately 40 tonnes of spodumene-bearing pegmatite material sampled from the main mineralised pegmatite intrusion at the Authier property. The metallurgical testing was conducted under the supervision of Bumigeme who was conducting a pre-feasibility study of the Project during that period. The complete metallurgical study conducted in laboratory consisted in a total of 48 tests but only 16 tests returning satisfactory results were reported. The most significant results from the process flowsheet returned a Li_2O concentrate grade ranging from 5.78% to 5.89% with a recovery between 67.52% and 70.19% (tests 33 and 47). The average Li_2O grades of the pegmatitic material from tests 33 and 47 were 1.15% and 1.13% Li_2O respectively. Test number 12, with an average grade of 1.35% Li_2O, produced a Li_2O concentrate grade of 5.96% with a recovery of 75.02%.

- **Tests realized in 2012**

In early fall of 2012, the Company has ordered some mineral processing and metallurgical tests to the SGS Lakefield Laboratory, the report is shown in the Appendix 3. The results of these tests are the base of the study prepared by Bumigeme to develop the metallurgical process involved in this PEA Technical Report. Glen Eagle Resources Inc had mandated Bumigeme Inc a Canadian Engineering consulting firm based in Montreal, working mainly in the mining and metallurgical sector, to develop the metallurgical aspect of his Authier Lithium Project. This mandate is part of the Preliminary Economic Assessment (PEA) compliant with NI 43-101 regulations.

The mandate mainly consists of developing a conventional lithium flotation process plant with a capacity of 2,200 TPD (run of mine), and estimating the capital investment (CAPEX) and operating cost (OPEX) of the concentrator.

Based on the results obtained from the laboratory tests, the design is made to process 803,000 mt per year of run of mine ore at a grade of 0.91% Li_2O. The designed plant will produce approximately 103,000 tonnes of concentrate per year at a grade of 6% Li_2O. The need of capital investment is estimated at $28,353,769. In addition $2,000,000 will be required at the beginning to start the construction of the tailings pond and $3,272,000 for the working capital. It's worth to specify, that the electrical substation is not included in these costs. The concentrator operating cost is estimated at $13.83/t.

The main parameters retained by Bumigeme in their metallurgical section are:

- concentrate grade of 6.0% Li_2O, and;
- overall mill recovery of 85%;
- no mica pre-flotation is considered necessary in the processing.

1.8 Environmental

The actual preliminary environmental report, prepared by DESSAU and GFE Forestry & Exploration Services, for Authier Project didn't return environmental issues. Activities by DESSAU and GFE were performed to determine constraints linked to water and sediments quality and to environmental (physical, biological, human) impact.

1.9 Mining

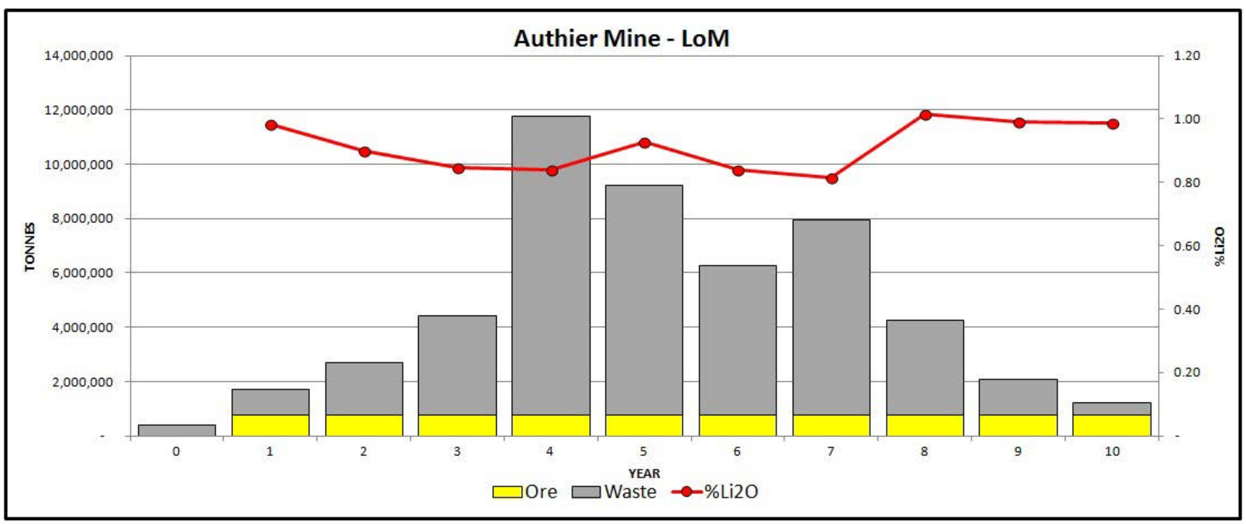
Taking into account the geometry and the depth of the mineralized zone, open-pit mining method has been considered in this study. The near surface resources will be mined by a major open pit, which will have a ten years life following a one year construction and pre production period. The mine plan is based on the Measured, Indicated and Inferred mineral resources contained in the pit design, which was based on a CDN$525/tonne of spodumene concentrate (6% Li_2O) Lerchs-Grossmann optimized pit shell. Open-pit mining will be conducted by the operator and a specialized mining contractor will be assisting the operator during high stripping periods. Surface mining will follow the standard practice of an open-pit operation, with conventional drill and blast, load and haul cycle using a drill/truck/shovel mining fleet. The overburden and waste rock material will be hauled to the overburden and waste disposal areas near the pit. The run-of-mine mineralization will be drilled, blasted and loaded by hydraulic excavators and delivered by large mining trucks to the primary crusher or stockpiles near the crusher.

A preliminary pit optimisation based on well defined parameters was performed to outline the mineral resource demonstrating a prospect for economical extraction. The resulting shell was then used to design a final open-pit including a ramp and safety berms. The mineral resource contained in the final pit is summarized as:

Material type	Density t/m3	Tonnage tonnes	Grade % Li2O	Concentrate tonnes	Strip. Ratio t:t
Measured Resource	2.71	2,360,000	0.90	301,000	5.7
Indicated Resource	2.71	5,120,000	0.92	667,000	
Measured + Indicated Resource	2.71	7,480,000	0.91	968,000	
Inferred Resource	2.71	290,000	0.87	32,000	
Waste - Rock	2.71	39,280,000			
Waste - OVB	2.00	4,970,000			
Total		**52,020,000**			

*Include a 5.0 % mining dilution and a 99.0 % mining recovery

A life-of-mine scenario was developed over 350 days per year, on two 10 hour shifts per day. The mine plan is calling for a daily production of 2,200 tonnes treated per day and an average stripping ratio of 5.7 over 10 years of operation. The following graph is summarizing the production, the waste removal and the average grade over the mine life.

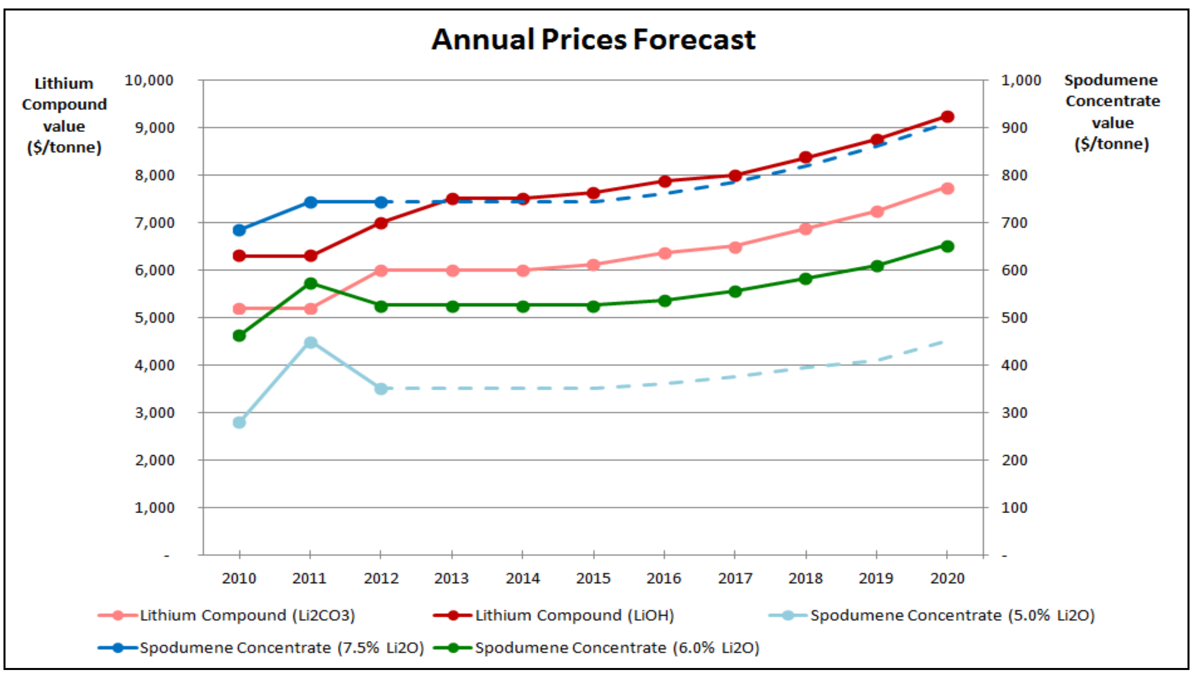


The production was planned over 350 days per year, on two 10 hour shifts per day. The main mining fleet will consists of hydraulic excavators of 6 m^3 bucket capacity, and 37 tonnes off-road trucks. The following Table is listing the main open-pit equipments.

Mining Equipment	Model (refrerences)	Size, or capacity	Quantity
Hydraulic Excavator	Komatsu PC1250LC-8	6.0 m^3	1
Haul Trucks	Komatsu HD325-7	37.0 t	5
Wheel Loader - Secondary	Komatsu WA600-6	6.0 m^3	1
Production Drill	ACopco AC-ROC D55	152 mm	1
Track Dozer	Komatsu D275AX-5	225 kW	1
Grader	Komatsu GD655-5	150 kW	1
Water Truck	-	18,000 l	1
Backhoe - Secondary	Komatsu PC650LC-8	4.0 m^3	1

1.10 Market

As there was no market study available from Glen Eagle, SGS relied on public information's to select a lithium concentrate price to be retained in the base case. Spodumene concentrate prices were available, on public sites for the time being, but for concentrates with grades of 7.5% Li_2O and 5.0% Li_2O only. Long term forecasts for lithium carbonates and hydroxides were also available as shown in pink and red full lines of the next Figure. We therefore applied the same future trend, to the spodumene concentrates of 7.5% and 5.0%, shown in dotted dark and light blue lines, as those of the lithium carbonates and hydroxides known prices, with an extrapolation for the 6.0% Li_2O concentrate, shown in green full line. The next Figure is illustrating the base case spodumene concentrate price of $525/tonne retained in this study.



1.11 Capital and Operating Costs

The capital cost estimate consists of the direct cost for the mine production, the concentrator expenditures and all services. The mine capital costs were reduced by assuming that the majority of the mining fleet will operate under leasing conditions, which greatly reduced the initial capital cost. The following Table summarizes the capital cost of the project.

Capital Cost Summary	M$
Infrastructures including concentrator	35.4
Equipment not under leasing	1.2
Others	6.2
Contingency (15%)*	2.2
Total rounded	45.0
Total paid by Third Party**	4.0
Total paid by Glen Eagle	41.0

**Concentrator contingency is already accounted for into the Infrastructure capital cost.*
***Refers to Section 21*

The unit operation costs have been prepared to include the mine production costs, which are the same for mineralization and waste, the overburden removal, the processing cost and the general and administration cost. There are two hard rock mining costs due to the scenario that includes giving a portion of the waste removal to a mining contractor. During the years 4 to 7, there is a major increase of waste to be removed and it was estimated that it would be more economical to subcontract, instead of increasing the mining fleet for such a short period of time. The estimated cost for the contractor was taken as the owner cost plus 35%. The following Table is summarizing these unit costs.

Unit Operating Cost	**Value**	**Reference**
Hard rock mining by owner	3.20	$/t mined
Hard rock mining by contractor	4.32	$/t mined
Overburden mining	2.50	$/t mined
Processing	13.83	$/t treated
G&A	5.02	$/t treated
Concentrate transport locally	9.72	$/t concentrate
Concentrate transport to US	67.42	$/t concentrate

1.12 Economic Analysis

An economic analysis has been completed with parameters and assumptions that are the base of the cash flow; these main assumptions and parameters are listed below.

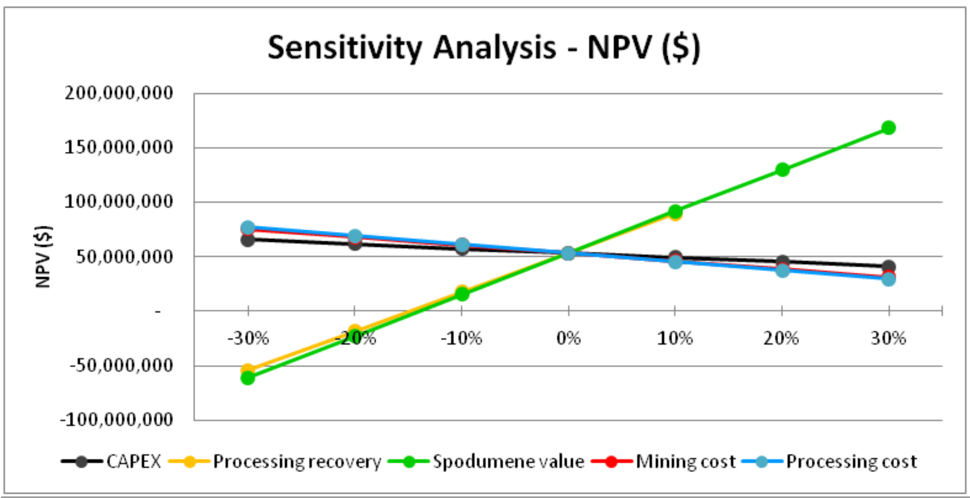
- price of 6.0 % Li_2O spodumene concentre at $525 per tonne;
- processing rate of 2,200 tonnes per day (777,000 tonnes per year);
- constant exchange rate of $1.00 (US$:CDN$);
- discount rate of 6.00 %;
- economical analysis is presented as pre-finance and pre-tax;
- sunk costs and owner's costs are not included in the model;
- 1 year construction period (infrastructures and site preparation);
- 10 years of mining operation;
- initial capital cost will be totally spend during the first year of construction;
- initial capital costs are financed by equity;
- open-pit will be carried out by Glen Eagle and a specialized mining contractor;
- specialized mining contractor will provide his own mining equipments;
- mining and processing expenses will commence in the first year of operation.

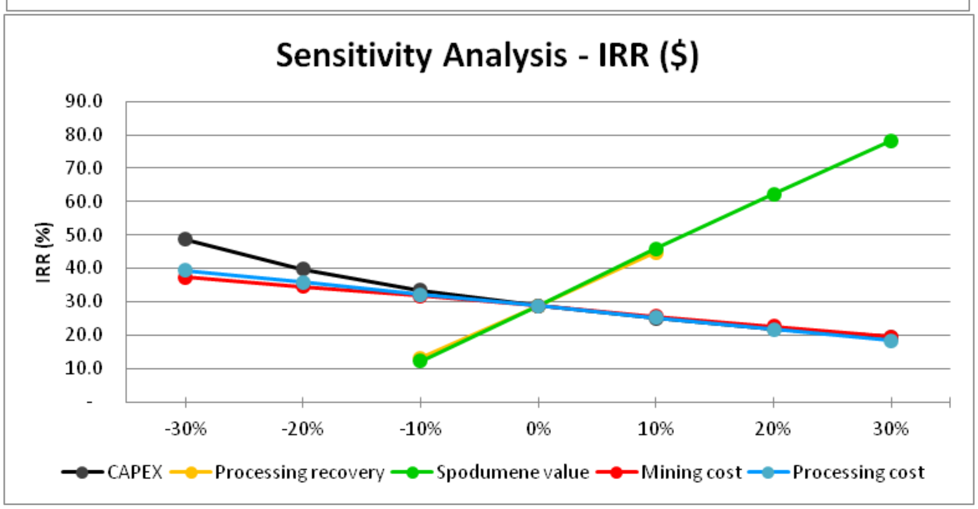
The following Table is showing the economic results of the base case scenario.

Item	Value	Reference
Total Revenues	528.3	M$
Total Operating Costs	364.9	M$
Pre-production Capital Costs	42.1	M$
Sustaining Capital Costs	16.2	M$
Royalty paid	19.8	M$
Salvage value + Working capital recovery	8.8	M$
Undiscounted benefits	94.1	M$
Discounted benefits at 6.00 %	53.3	M$
Internal rate of return	28.7	%
Payback period	1.9	years

A Sensitivity analysis shown in the next Table has been prepared with variations of ±30% for the CAPEX, mill recovery, spodumene value and Opex. The two following Figures are graphs of the NPV and IRR sensitivity for variations of above parameters.

Variation		-30%	-20%	-10%	0%	10%	20%	30%
CAPEX	$	31,500,000	36,000,000	40,500,000	45,000,000	49,500,000	54,000,000	58,500,000
NPV	$	65,700,000	61,600,000	57,500,000	53,300,000	49,300,000	45,200,000	41,100,000
IRR	%	48.6	39.8	33.5	28.7	25.0	21.9	19.4
Processing recovery	%	59.5	68.0	76.5	85.0	93.5		
NPV	$	- 54,300,000	- 18,400,000	17,500,000	53,300,000	89,300,000		
IRR	%			13.1	28.7	44.7		
Spodumene value	$/t conc.	367.5	420.0	472.5	525.0	577.5	630.0	682.5
NPV	$	- 61,000,000	- 23,200,000	15,400,000	53,300,000	91,900,000	129,700,000	168,300,000
IRR	%			12.3	28.7	45.9	62.2	78.3
Mining cost	$/t mined	2.24	2.56	2.88	3.20	3.52	3.84	4.16
NPV	$	75,300,000	68,000,000	60,700,000	53,300,000	46,100,000	38,800,000	31,500,000
IRR	%	37.3	34.5	31.7	28.7	25.7	22.6	19.5
Processing cost	$/t treated	9.68	11.06	12.45	13.83	15.21	16.60	17.98
NPV	$	77,100,000	69,200,000	61,300,000	53,300,000	45,500,000	37,500,000	29,700,000
IRR	%	39.3	35.8	32.2	28.7	25.2	21.7	18.3





The base case scenario supposed that the concentrate production will be sold to the local market (within Abitibi) and to the US market (50% - 50%). To assess the project sensitivity to the potential market, two scenarios were analysed where 100% of the production will be sold to each market respectively. The results are shown in the following Table.

Market	Production sold 100% locally	Production sold 100% to US
NPV (M$)	74.7	32.1
IRR (%)	38.2	19.4
Payback (Years)	1.6	2.3

1.13 Interpretations and Conclusions

- **Geological**

The mineral resources estimate reported in this document is compliant with standards as outlined in the NI 43-101 regulations. The Authier Deposit contains enough resources to justify the launch of a preliminary economic assessment study. The Company recently mandated SGS Geostat and Bumigeme Inc., to conduct a PEA on the Authier Deposit.

The 2010 and 2012 independent sampling programs done by SGS Geostat revealed that the average grade of original assays for Li_2O may vary in the order of 10-15% from one lab to another. Additionally, a bias at a 95% confidence level is observed. SGS Geostat recommends continuing sending duplicate samples to two different laboratories in order to determine the percentage difference between labs. SGS Geostat recommends also continuing the QA/QC protocols.

- **Economical**

The parameters retained in this PEA are including the development of an open-pit mine using standard equipments, at a rate of 2,200 tonnes per day to be process by an on-site concentrator having an estimated recovery of 85%. The final product will be a spodumene concentrate at 6.0% Li_2O. The base case is assuming that 50% of the concentrate will be sold to local market and the rest transported by railway either to the USDA market or to oversea. No contract or purchasing agreements for the concentrate have been negotiated, or signed, at the date of this report. The cash flow analysis prepared from a concentrate price of $525/t, is showing that the Authier Project contains economic Mineral Resource.

1.14 Recommendations

➢ **Geological**

The following budget recommendation is based on a three (3) phase's program. The total estimated budget including Technical services is estimated at $2,542,500. This estimate which is described in the Recommendation Section is purely conceptual and does not include accommodations, meals, transportation and equipment rental costs.

1. SGS Geostat recommends additional infill drilling (Phase 1) to increase the quality of the geological information at depth as well as the overall mineral resources and the geological model, this infill drilling of the Authier Lithium deposit is estimated at a cost of $1,000,000.

2. The Company has available geophysical data, geochemical data and geological surface interpretations of the Authier Lithium property, SGS Geostat recommends (Phase 2) additional exploration, mapping and sampling of its prime exploration targets to increase the geological information and resources on the Property, this additional exploration cost is estimated at $100,000.

3. Following the positive outcome of the exploration campaigns, SGS Geostat recommends additional drilling (Phase 3) of its prime exploration targets to increase the quality of the geological information and the possibility for adding resources on the Property, this exploration drilling of the Authier Lithium property is estimated at $500,000.

Additional QA/QC measures will have to be implemented such as:

- Adding down hole survey measurements (dip, direction of surveyed holes)
- Implementing core orientation surveys on selected future drill targets.
- Continuing core duplicate sampling to a second certified laboratory.
- Sending its homemade reference materials (standards) to a certification laboratory in order to have a certified expected value and standard deviation.

➢ **Mining**

Complete a geotechnical study to optimize the open-pit slopes.
Assess the hydrogeological properties of the open-pit area.

➢ **Processing**

The metallurgical spodumene concentrates recovery; expressed in lithium oxide Li_2O, obtained during the laboratory tests by SGS Minerals Services in Lakefield, Ontario, was good, more than 85%. But, the head grade of the sample was quite high, 1.3% Li_2O, which is above the estimated mill feed grade of the base case at 0.91% Li_2O. Therefore, for future studies, we recommend doing some additional laboratory tests and pilot plant tests involving a more representative mill feed grade, and his effect on the overall recovery. These additional tests should include the followings:

- testing the floatability of the lower grade samples;
- making a flotation lock cycle test;
- testing the efficiencies of more reagents and water from plant location;
- rheological tests consisting of slurry thickening and filtration.

The estimate cost of these lab tests is ± $100,000.

In addition it will be worthy to make a pilot plant flotation campaign. The campaign will require at least 50 tonnes of resources. We estimate the cost of that pilot plant campaign at ± $500,000.

- **Economical**

As mentioned in this study, there was no market study available to prepare this economic assessment. All commercial and marketing values and data used in this study were obtained from public information's and/or from ongoing projects results disclosures.

We therefore recommend a complete marketing study dedicated to the final product of the Authier Project.

2 Introduction

2.1 General and Purpose of the Technical report

This NI 43-101 Technical Report was prepared by SGS Canada Inc. – Geostat ("SGS Geostat") and Bumigeme Inc., for Glen Eagle Glen Eagle Resources Inc. ("Glen Eagle" or "Company"). This Report is a Preliminary Economic Assessment Study (PEA), that incorporates the disclosure of the mineral resources update for the Authier Lithium property ("Property"), submitted on December 18, 2012. This Technical Report was prepared following the guidelines set in NI 43-101 and the Form 43-101F1 Technical Report.

SGS Geostat and Bumigeme understand that this Report will be used by Glen Eagle to support news release disclosing the economic results of the Preliminary Economic Assessment of the Authier Property. The accuracy of the cost estimates in this report is in the rage of ±30%.

2.2 Terms of Reference

This PEA Technical Report of the Authier Lithium property was prepared in collaboration between SGS Canada Inc (SGS Geostat, Blainville) and Bumigeme Inc. of Montreal. The authors for SGS are Mr. Maxime Dupéré P.Geo., Mr. Jonathan Gagné, Eng., and Mr. Gaston Gagnon, Eng.

For Bumigeme Inc., the author is Mr. Florent Baril, Eng.

This technical report was prepared according to the guidelines set under "Form 43-101F1 Technical Report" of NI 43-101 Standards and Disclosure for Mineral Projects. The certificates of qualification of the Qualified Persons responsible for the technical report are found in section 0.

Information in this report is based on a critical review of the documents and information provided by personnel of Glen Eagle Resources Inc., in particular Mr. Gilles Laverdière, P. Geo., and Director of the Authier Lithium and by Mr. Jean Labrecque, President of Glen Eagle. A complete list of the reports available to the authors is found in the References section of this report.

2.3 Units and Currency

All measurements in this report are presented in meters (m), metric tonnes (tonnes), parts per million (ppm) and percentage (%) unless mentioned otherwise. Monetary units are in Canadian dollars (C$) except when specified in United States dollars (US$). Abbreviations used in this report are listed in Table 2-1.

Table 2-1: List of Abbreviations

km	Kilometres
m	Metres
µm	Micrometres
ha	Hectares
m3	Cubic metres
km/h	Kilometre per hour
%	Percent sign
t/m3	Tonne per cubic metre
$	Dollar sign
°	Degree
°C	Degree Celcius
NSR	Net smelter return
GMR	Gross Metal Royalty
pH	Potential of hydrogen (acidity scale)
Au	Gold
g/t	Gram per metric tonne
ppm	Parts per million
Oz	Ounces (Troy)
NQ	Drill core size (4.8 cm in diameter)
SG	Specific Gravity
NTS	National Topographic System
UTM	Universal Transverse Mercator
NAD	North America Datum

2.4 Disclaimer

It should be understood that the mineral resources which are not mineral reserves do not have demonstrated economic viability. The mineral resources presented in this Technical Report are estimates based on available sampling and on assumptions and parameters available to the author. The comments in this Technical Report reflect the author's from Bumigeme and SGS Geostat best judgement in light of the information available. During the mineral resource estimation process, different assumptions were made. These assumptions were used in order to calculate the modelling cut-off grades and resources cut-off grades following the "reasonable prospect for economic extraction" stated by the NI 43-101 regulation.

An optimised shell was done using economic parameters provided by Jonathan Gagné, Eng. at SGS Geostat. The term *in-pit* refers to the resources within the optimised shell according to the different cut-off grades. The term *in-pit* does not imply that any open pit mining scenarios were made by SGS Geostat. Furthermore, it should not imply that the updated resources stated in this report have demonstrated economic viability.

2.5 Site Visit

Mr. Maxime Dupéré, P. Geo., from SGS, visited the Property on July 30, 2012 for a global outlook of the deposit, plus: drill hole collar locations, review of exploration methodology, sampling procedures, quality control procedures and to conduct an independent check sampling of mineralized drill core intervals selected from drill holes of the Authier Lithium deposit.

Mr. Jonathan Gagné, Eng., also from SGS Geostat visited the property on December 21, 2012 for a global outlook of the site access roads, electricity network, near-by railway, etc.

3 Reliance on Other Experts

The authors of this Technical Report are not qualified to comment on issues related to legal agreements, royalties, permitting, and environmental matters. The authors have relied upon the representations and documentations supplied by the Company's management. The authors assume that the documents, reports and other data listed are substantially accurate and complete in all material aspects. The authors have reviewed the mining titles (claims), their status, the legal agreement and technical data supplied by the Company, and any public sources of relevant technical information.

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 (Figure 4.1). The center of the Property is situated on NTS sheet 32D08 at about UTM 5,361,360 mN, 706,725 mE, NAD83 Zone 17. The Property is accessible by the rural road network connecting to the main highway Route 109 situated a few kilometres east which links Rivière Héva to Amos then Matagami. Highway Route 109 connects at Rivière Héva with the Route 117 which is the provincial highway linking Val d'Or and Rouyn-Noranda.

The only significant mineralisation known on the Property is the spodumene-bearing pegmatite intrusion discovered in the late 1960's and worked by various operators between 1969 and 1999. The spodumene-bearing pegmatite intrusion is located on claims number CDC 2183455, 2194819 and 2116146, and extends at surface between approximately 707,050mE and 707,775mE in the East-West direction, and between 5,359,975 mN and 5,360,275 mN in the North-South direction. The Figure 4-2 shows the location of the mineralised spodumene-bearing pegmatite intrusion (in magenta) in relation with the Property boundaries.

In 1993, Raymor Resources Ltd conducted a 30 tonnes bulk sample of the mineralized pegmatite. The location of the trench where the bulk sample was collected is located at UTM coordinates 707,400 mE, and 5,360,185 mN, NAD83 Zone 17.

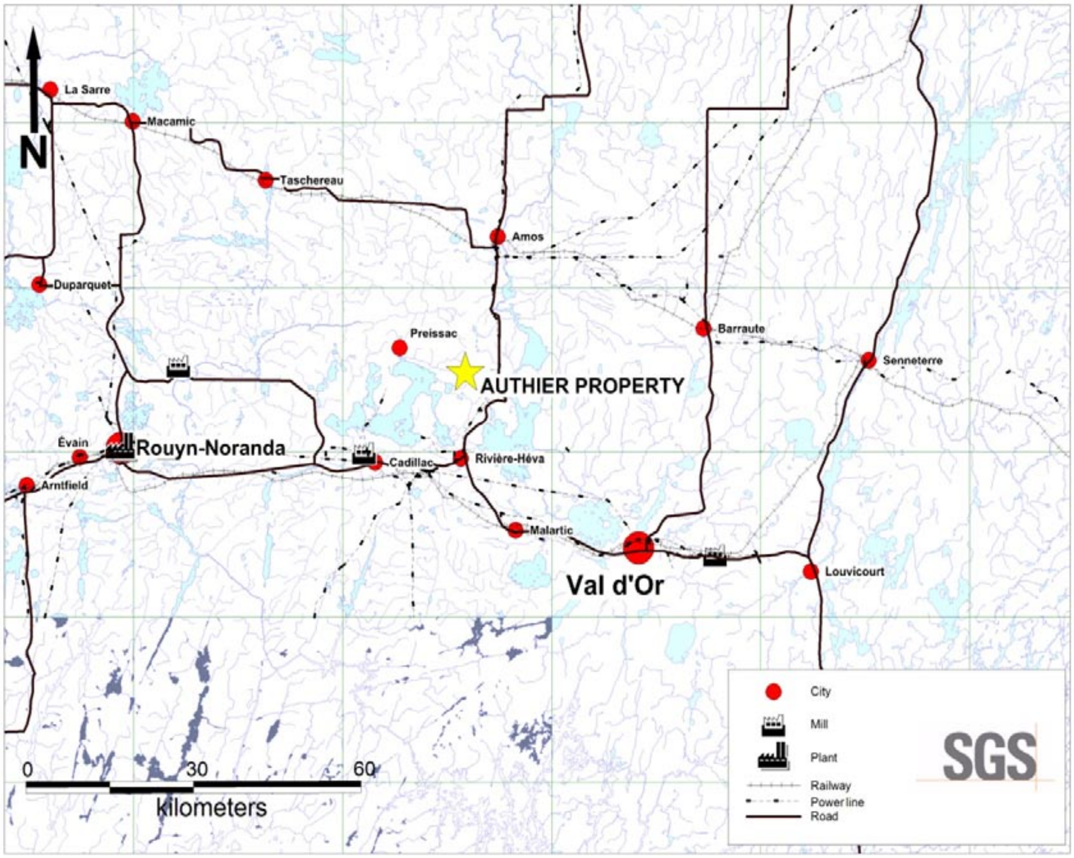


Figure 4-1: Authier Lithium Property Location Map

4.2 Property Ownership and Agreements

As of March 2011, the Property consists in one block totalling 19 claims covering 653.57 ha. The claims are located in the La Motte Township except the two westernmost claims which are located in the Preissac Township. The claims are located over Crown Lands. The Property area extends 3.4 km in the east-west direction and 3.1 km north-south. All claims composing the Property are map designated cells referred as CDC. The Property is adjacent to a protected area reserved for groundwater catchment supply located just the north of the Property, which has been excluded for exploration and mining activities. Figure 4-2 shows the claim map of the Property and a detailed listing of the Authier property claims is included in Table 4-1.

From the 19 claims composing the Property, 3 claims were acquired by staking on November 27, 2009 (CDC 21955725) and July 9, 2010 (CDC 2240226 and 2240227). The other 16 claims were acquired either by a purchasing agreement or are held under an option agreement. The different agreements are detailed below:

One-hundred-percent (100%) interest of 3 claims (CDC 2183454-2183455 and 2194819) was acquired through a purchase agreement concluded with 9187-1400 Québec Inc. on June 8, 2010. The terms and obligations of the purchase agreement are as follow:

- Payment of $20,000 upon signing and payment of $30,000 on the deposit of the file at the regulatory authorities.
- Payment of $25,000 plus 300,000 shares upon acceptance of the transaction by the regulators;
- Payment of $50,000 and 500,000 shares on each first, second and third anniversary of the agreement;
- Payment of $500,000 and one million shares plus a 2% NSR royalty upon completion of a positive feasibility study;
- Required exploration expenditures totalling $2,250,000 distributed over three years.
- Acknowledgment and carry-on of a 1% GMR on the claim CDC2194849 to Globex Mining Entreprises Inc.

Currently, approximately more than 75% of the mineral resources are present inside the 3 claims mentioned above. Seventy-percent (70%) interest of one claim (CDC 2116146) is currently held under an option and joint venture agreement concluded with Royal Nickel Corporation ("Royal Nickel") on September 10, 2010 with the signing of a letter of intent. The claim is subject to a 2% NSR royalty to Jefmar Inc. which half of the royalty can be repurchased for $1,000,000. The terms and obligations of the purchase agreement are as follow:

- Payment of $10,000 upon execution of the agreement;
- Payment of $10,000 on first and second anniversary, and $50,000 on third anniversary of the agreement;
- Exploration work commitment of $100,000 on year one, $150,000 on year two and $200,000 on year three of the agreement;
- Formation of a 70% Glen Eagle and 30% Royal Nickel joint venture agreement at the end of the option period of the agreement.
- 1.5% NSR royalty to Royal Nickel if its interest falls under 10% on CDC2116146.

Currently approximately less than 25% of the estimated mineral resources are present inside the claim (CDC2116146). One-hundred-percent (100%) interest of 12 claims (CDC 2116154-156, 2187651, 2192470-2192471, 2219206-2219209, and 2247100-2247101) is held by Glen Eagle and was acquired through a purchase agreement concluded with Globex Mining Entreprises Inc. ("Globex") on October 5, 2010. The terms and obligations of the purchase agreement are as follow:

- Payment of $25,000 upon signing and 400,000 shares upon acceptance of the transaction by the regulators;
- Payment of $25,000 on the first anniversary of the agreement;
- 2% Gross Metal Royalty to Globex;

Currently, there are no estimated mineral resources reported on theses 12 claims, please refer to the next Figure 4-2.

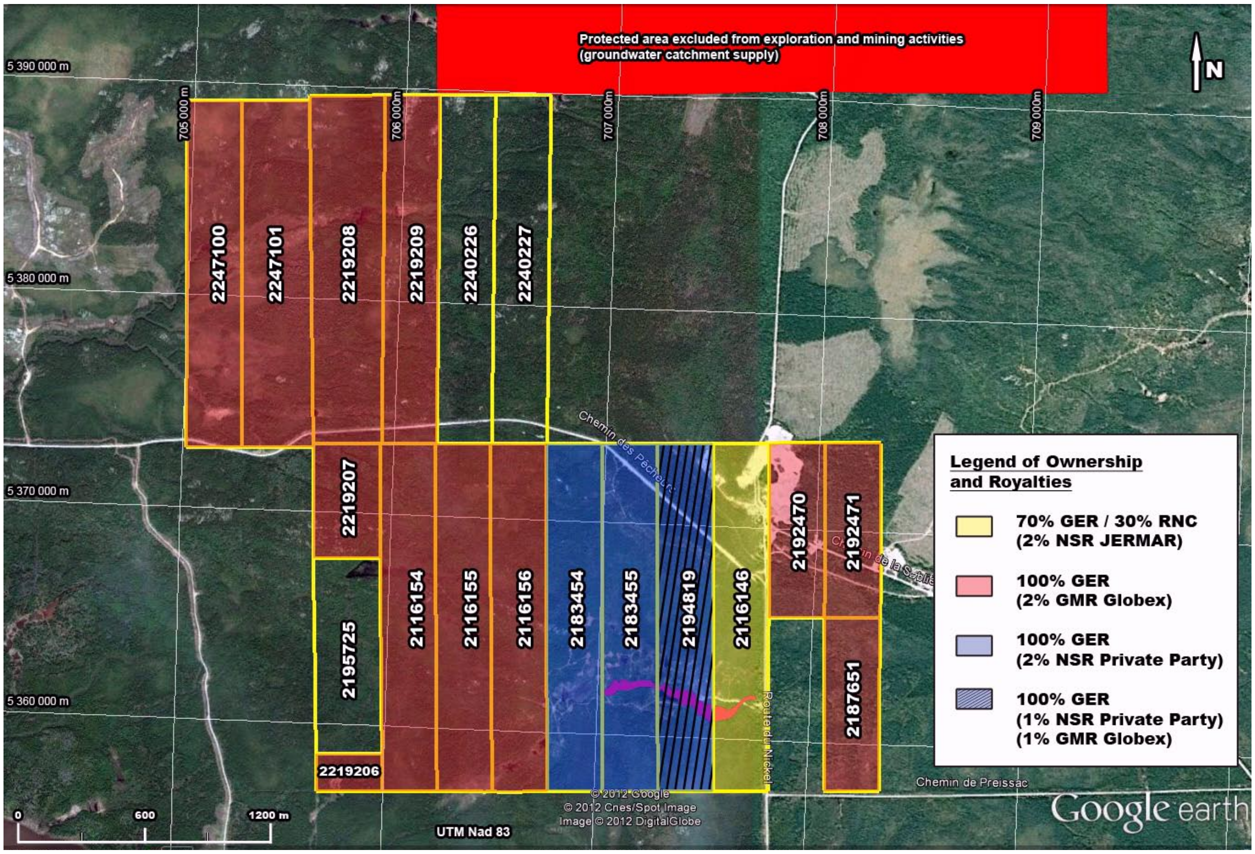


Figure 4-2: Property Mining Titles Location Map

4.3 Royalties Obligations

As described in Section 4.2, the property is subject to three separate royalties defined under different agreements. The first royalty concerns the 3 claims CDC 2183454-2183455 and 2194819 acquired from a private party where a 2% NSR royalty on the CDC 2183454 and CDC 2183455, and a 1% NSR royalty on the CDC 2194819 will be retained by the original owner upon completion of a positive feasibility study. A 1% NSR can be repurchased on the claims CDC 2183454, 2183455 and 2194819 for $1,000,000 leading respectively to a 1%, 1% and 0% on the CDC 2183454, 2183455 and 2194819. The claim CDC 2194819 also holds a 1% GMR to Globex Mining Entreprises Inc. The second royalty relates to claim CDC 2116146 acquired through an option and joint venture agreement with Royal Nickel. The claim is subject to a 2% NSR underlying royalty to Jefmar Inc., which half of the royalty can be repurchased for $1,000,000. Royal Nickel also has the option to transfer its current participation interest to a 1.5% royalty on claim CDC 2116146 if its participation interest falls under 10%. Finally, the third royalty concerns the 12 claims (CDC 2116154-156, 2187651, 2192470-2192471, 2219206-2219209, and 2247100-2247101) acquired from Globex where a 2% Gross Metal Royalty will be retained by Globex upon the completion of the agreement. Currently, there are no estimated mineral resources present inside the 12 claims mentioned above.

Approximately more than 75% of the mineral resources are present inside the 3 claims (CDC 2183454-2183455 and 2194819). About less than 25% of the estimated mineral resources are present inside the claim (CDC2116146). Please refer to Figure 4-2.

4.4 Permits and Environmental Liabilities

Glen Eagle is conducting exploration work under valid forest intervention permit delivered by the provincial Ministère des Ressources Naturelles et de la Faune ("MRNF"). As of the date of this report, the Company confirmed having valid work permits.

There are no environmental liabilities pertaining to the Property, according to the Company management.

Table 4-1: Authier Lithium Property Claim List

NTS	Township	Area (ha)	Title		Entry Date	Expiry Date	Renewals	Exceding Work($)	Work Required ($)	Mining Right ($)	Holder
32D08	LA MOTTE	42,88	CDC	2116154	08/08/2007	07/08/2013	2	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,87	CDC	2116155	08/08/2007	07/08/2013	2	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,86	CDC	2116156	08/08/2007	07/08/2013	2	22323.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,85	CDC	2183454	02/06/2009	01/06/2013	1	162915,15	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,84	CDC	2183455	02/06/2009	01/06/2013	1	294922,99	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	21,39	CDC	2187651	02/09/2009	01/09/2013	1	0.00	500.00	27.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	21,08	CDC	2192470	22/10/2009	21/10/2013	1	0.00	500.00	27.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	21,39	CDC	2192471	22/10/2009	21/10/2013	1	0.00	500.00	27.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,82	CDC	2194819	19/11/2009	18/11/2013	1	205568,34	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	29,03	CDC	2195725	27/11/2009	26/11/2013	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	5,51	CDC	2219206	22/04/2010	21/04/2014	1	0.00	500.00	27.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	17,06	CDC	2219207	22/04/2010	21/04/2014	1	0.00	500.00	27.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	55,96	CDC	2219208	22/04/2010	21/04/2014	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,71	CDC	2219209	22/04/2010	21/04/2014	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,71	CDC	2240226	09/07/2010	08/07/2014	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	42,71	CDC	2240227	09/07/2010	08/07/2014	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	PREISSAC	42,75	CDC	2247100	23/08/2010	22/08/2014	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	PREISSAC	53,77	CDC	2247101	23/08/2010	22/08/2014	1	0.00	1200.00	53.00	100% Glen Eagle Resources (82675)
32D08	LA MOTTE	43,24	CDC	2116146	08/08/2007	07/08/2013	2	195227,98	1200.00	53.00	70% Glen Eagle, 30% Royal Nickel

5 Accessibility, Climate, Local Resources, Infrastructure, Physiography

5.1 Accessibility

The Property is accessible by the rural road network connecting to the main highway Route 109 situated a few kilometres east which links Rivière Héva to Amos then Matagami. Highway Route 109 connects at Rivière Héva with the Route 117 which is the provincial highway linking Val d'Or and Rouyn-Noranda.

5.2 Physiography

The Property is characterised by a relatively flat topography with the exception of the north-eastern area where gently rolling hills occurs due to the presence of a large sand and gravel esker in the area. Outcrops represent approximately 5% of the area. The overburden is relatively thin (up to 2-3 metres) and is characterised by glacial tills and clays. The land is drained westward by small creeks and local grassy swamps occurs in topographic lows. The area is generally covered by forest populated by mixed balsam, spruce and aspen trees. The elevation above sea level ranges from 320 m at the lowest point on the Property to 380 m near the top of the esker, with an average elevation of 350 m.

5.3 Climate

The climatic data used to characterize the sector under study comes from the meteorological station of Val-d'Or, Québec. These observations were carried out during 1961-1991. Exploration work in the area can typically be carried out year-round but soft ground in the areas covered by wet lands creates difficult working conditions from late spring until early winter.

In the Val-d'Or region, the average daily temperature is slightly over the freezing point, i.e. 1.6°C. The average temperature during July reaches 17°C, while the temperature in January falls to -16°C. Precipitation averages 928 mm of water annually in the area. Average monthly precipitation ranges from 48 mm in February to 103 mm in September. Snow falls from October to April, but is much more significant from November to March. The average for these five months is 26 mm using snow to water conversion factor. The pH of the precipitations measured at the Joutel station in 1991 varies from 4.30 in November to 4.78 in June.

The anemometric data collected in Val d'Or between 1961 and 1991 shows that from June to January the southwest winds are dominant, whereas from February to May the winds coming from the northwest are more frequent. In this sector, the winds have an average velocity varying between 11 and 14 km/h.

5.4 Local Resources and Infrastructures

The regional resources regarding labour force, supplies and equipment are sufficient, the area being well served by geological and mining service firms. The cities of Val d'Or and Rouyn-Noranda are regional centers for the Abitibi region and have the necessary infrastructures and workforce to support a mining operation. While there is currently a general shortage of qualified personnel in the mining and exploration sector, the location of the project is favourable in that regard. The area is traditionally a mining area with several operating mines and active exploration companies. All major services are available in Val d'Or and Rouyn-Noranda.

5.5 Surface Rights

All the claims composing the Property are located over Crowns Lands. There is no reason to believe that the Company won't be able to secure the surface rights to construct the infrastructures related to a potential mining operation, including tailings storage and waste disposal areas, and processing plant.

6 History

This section is modified from Karpoff (1994) and includes Regional and Property work conducted up to 2009.

6.1 Regional Government Surveys

A series of geological surveys and geoscientific studies have been conducted by the Quebec Government in the Project area. Geological surveys in the 1955-1959 period then in 1972 (Leuner 1959, and Brett et al. 1976) cover the entire Property area. 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).

6.2 Mineral Exploration Work

Very little exploration work has been conducted on the Property prior to 1966. 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 polarisation surveys followed by 6 drill holes located in the south-western 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 survey 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, and 23 diamond drill holes totalling 2,611.37 metres.

In 1969, the Quebec Department of Natural Resources carried out a series of floatation tests on two drill core composite samples. The bulk sample was composed of split core from drill hole AL-14 (50 metres) and hole Al-19 (38.1 metres). The results confirmed that the material was amenable to concentration by floatation process 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 metres.

In 1980, Société Québecoise d'Exploration Minière ("SOQUEM") completed six diamond drill holes (80-26 to 80-31) totalling 619.96 metres 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 1991, Raymor Resources Ltd ("Raymor") conducted small-scale metallurgical testing of pegmatite rocks mineralised 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" now called "COREM"). Results of the metallurgical testing returned a concentrate grade of 6.3% Li_2O with recovery rate of 72.6%.

In 1993, Raymor conducted additional drilling of 33 holes for a total of 3,699.66 metres 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 tonnes bulk sampling of the pegmatite dyke (completed in 1996).

In 1997, Raymor contracted the CRM to conduct additional metallurgical testing. The tests were conducted on two different samples weighting 18,049 kg (average Li_2O grade of 1.32%) and 12,283 kg (average Li_2O grade of 1.10%) respectively. The results of the metallurgical testing returned for the first sample a concentrate grade of 5.61% Li_2O (after magnetic separation) with a recovery rate of 60.8%. The second sample returned a final concentrate grade of 5.16% with a recovery grade of 58.3%.

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

A Qualified Person has not done sufficient work to classify the above-stated historical mineral resource estimate as current mineral resources. The historical mineral resources uses categories other than the ones set out in the CIM Definition Standards on Mineral Resources and Reserves. The author cannot explain the differences between the categories of the historical estimate and the mineral resources categories defined in the CIM Definition Standards on Mineral Resources and Reserves. The issuer is not treating the historical mineral resource estimate as current mineral resources as defined in section 1.2 and 1.3 of the NI 43-101 Standards of Disclosure for Mineral Projects. The historical estimate should not be relied upon.

In 1999, Raymor concluded an agreement with SOQUEM. The group completed a pre-feasibility study on the Project including additional metallurgical testing. The results of the metallurgical test outlined 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.

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-Lacorne batholith located 40 km northeast of the city of Val d'Or (Corfu 1993, Boily 1995, Mulja et al. 1995a).

The Preissac-Lacorne 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 (serpentinised peridotites, amphibolitic mafic flows) and metasedimentary units (biotite schists derived from greywackes). The Preissac-Lacorne 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 Pegmatites 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 mineralised but mostly barren pegmatite dykes have been mapped in the vicinity of the Preissac-Lacorne batholith. These intrusions cross-cut all the units of the Malartic Group and intrusive lithologies of the batholith except 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 irregular shape dykes often composed 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 a metre in thickness.

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

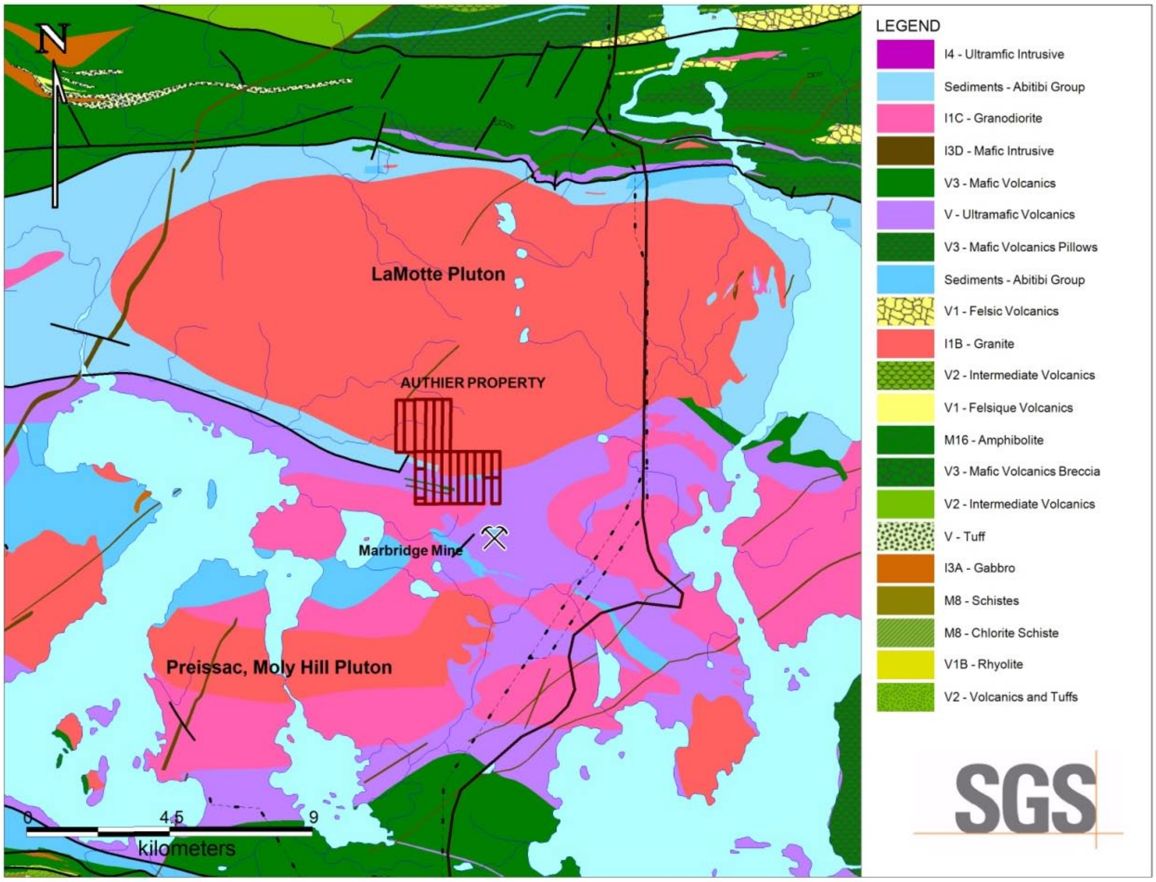


Figure 7-1: Regional Geology Map

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. 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 spodumene-bearing pegmatite which is the focus of the current mineral resource estimate.

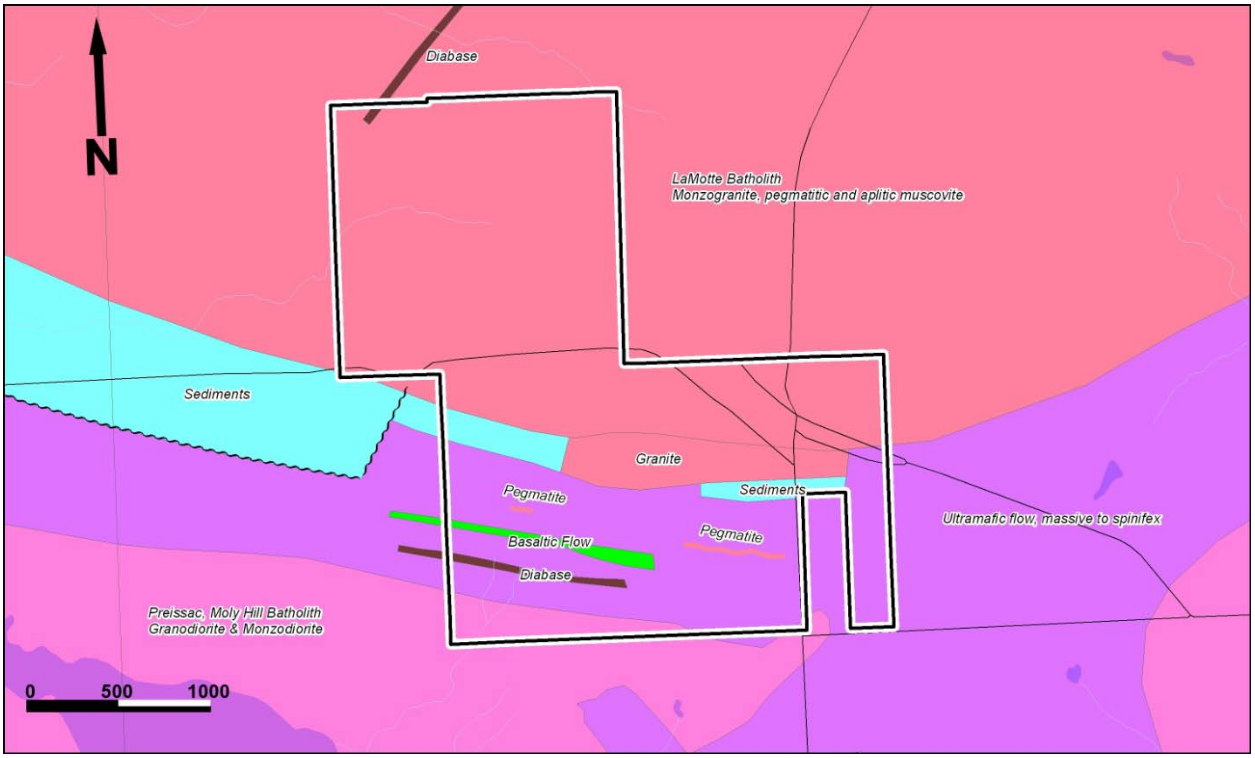


Figure 7-2: Local Geological Map

7.3 Mineralization

The mineralisation observed at the Authier project in the spodumene-bearing pegmatites is principally lithium with trace amount of beryllium, molybdenum, tantalum, niobium, cesium and rubidium. The Table 7-1 details the typical geochemical composition of the Authier pegmatite averaged from the Project database.

Table 7-1: Authier Lithium Property Claim List

	Average	Max	Count
Li_2O (%)	0.68	2.61	1245
BeO (%)	0.014	0.045	134
Cs_2O (%)	0.006	0.032	98
Ta_2O_5 (%)	0.008	0.023	79
Nb_2O_5 (%)	0.011	0.018	79
Rb_2O (%)	0.1	0.22	911
MoS_2 (%)	0.001	0.052	214

The observations made during the logging of the recent drill holes suggest that the main pegmatite at Authier is composed of at least two intrusive phases. The outside border of the pegmatite in contact with the host rocks as been identified as a transition zone or border zone. This transition zone is often significantly less mineralised in spodumene and is characterised 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 characterised by large centimetre-scale spodumene and white feldspar minerals. The core zone hosts the majority of the spodumene mineralisation at Authier. Figure 7-3: is a picture showing 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 which contains local rafts or xenoliths of the amphibolitic host rock which can be a few metres thick and up to 200 metres in length. Based on the information gathered from the drilling, the pegmatite intrusion is more than 700 metres in length and can be up to 60 metres thick. The intrusion is generally oriented east-west, dip to the north at an angle ranging between 35 and 50 degrees and is reaching depth of up the 165 metres below surface.

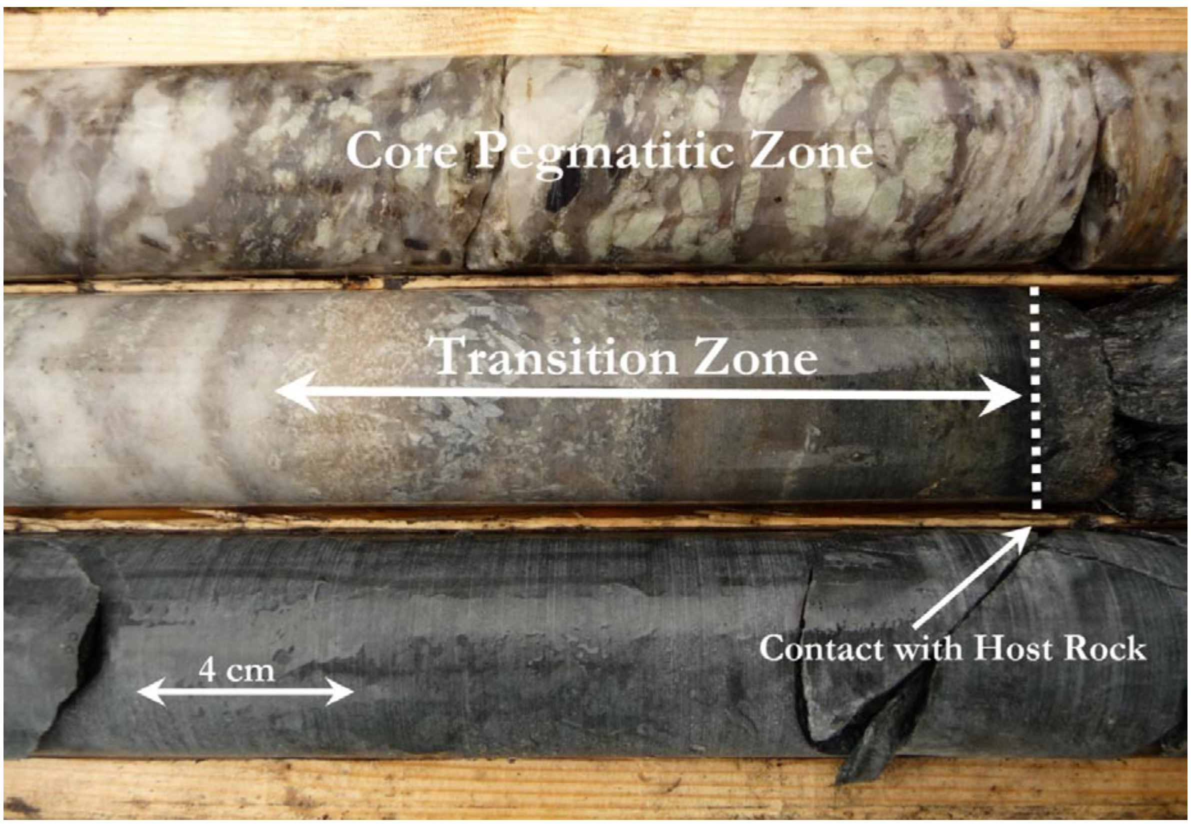


Figure 7-3: Drill Core from Hole Al-10-03 Showing Core and Transition Zones

The interpretation of the mineralisation at Authier is principally based on macroscopic observations made during the logging of the core. The author is not aware that any detailed analysis of the geochemistry of the pegmatite or mineralogical study has been made recently to characterise in more detail the lithologies observed on the Property. SGS Geostat recommends conducting more advanced study on the mineralogy and geochemistry of the mineralised pegmatites occurring on the Property.

8 Deposit Types

The deposit type for the lithium mineralisation 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 terranes 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 localised along deep fault systems which 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, and 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 (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 crystallisation 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 reaches 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 (see Figure 8.1). Rare-element pegmatites are generally considered to be form by primary crystallisation from volatile-rich siliceous melt related to highly differentiate 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).

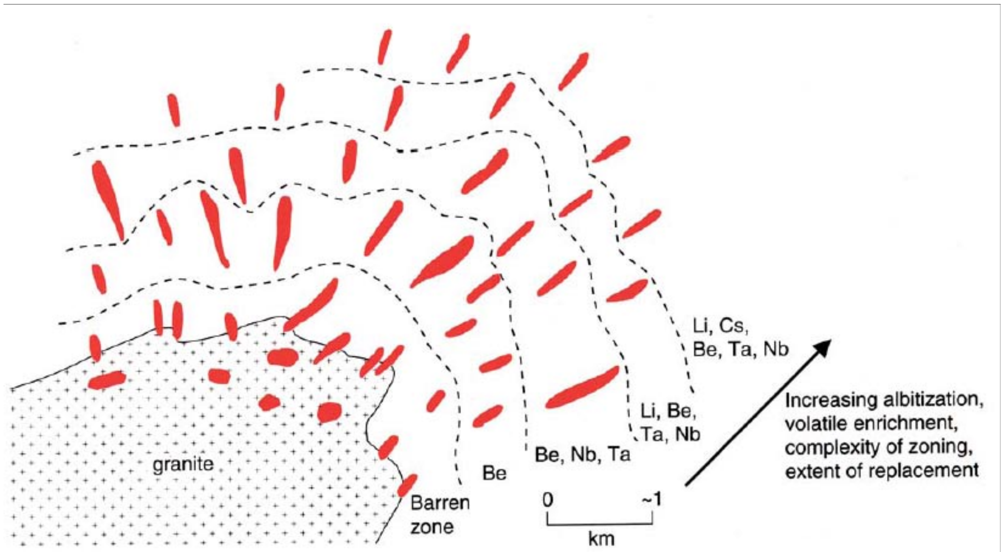


Image from Sinclair 1996 (modified from Trueman and Cerny 1982)

Figure 8-1: Schematic Representation of Regional Zonation of Pegmatites

Another type of deposit model observed in the Property area is nickel-copper sulfides associated with Archean-age ultramafic volcanic lithologies. A good example of this type of deposit, widely occurring around the world as in the Kambalda mining camp in Australia, occurs at the mined-out Marbridge deposits located less than 3 km south of the Property. This type of deposit involves metre-scale thick lenses or sheets of nickel-copper-bearing sulfides mineralisation typically composed of pyrite, pyrrhotite, chalcopyrite, and pentlandite. These lenses of sulfides are generally stratabound within flows of ultramafic volcanic unit referred as komatiite (Guilbert 1986). In the area of the Property, the ultramafic rocks hosting the sulfides are typically serpentinised and occur within a sequence of interbanded felsic volcanic rocks and greywacke sediments (Clark 1965, Graterol 1971).

9 Exploration

The Glen Eagle 2010-2012 diamond drilling campaign was preceded by prospecting geochemical sampling and geophysical surveys that covered the Authier Lithium deposit targeted areas. This work confirmed the presence of several pegmatite occurrences across the Property. Drilling on the Northwest of the Property has been done on pegmatites with a similar geochemical signature to the main Authier pegmatite. A bulk sample was taken for metallurgical studies for which it has not received the results. Please see Figure 7-2.

9.1 Geophysics

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 totalizes 53.5 line-km and was done through the bush without a cut line grid. The lines were read with a GSM-19 overhauser magnetometer, built by the company GEM of Toronto. It was used in walking mode and the locations of the readings were determined by an integrated GPS.

The magnetic measurements were taken continuously along 23 traverse lines for a total of 66,027 readings at every 1.25m. Magnetic diurnal was monitored with a base station and the magnetic readings have been corrected accordingly.

9.2 Geochemistry

In August 2011, a geochemical survey program was undertaken in an effort to discover new spodumene-bearing pegmatites. Eighty-six samples were collected mainly in the northwest part of the property. Four samples were collected on the main pegmatite. The samples 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.

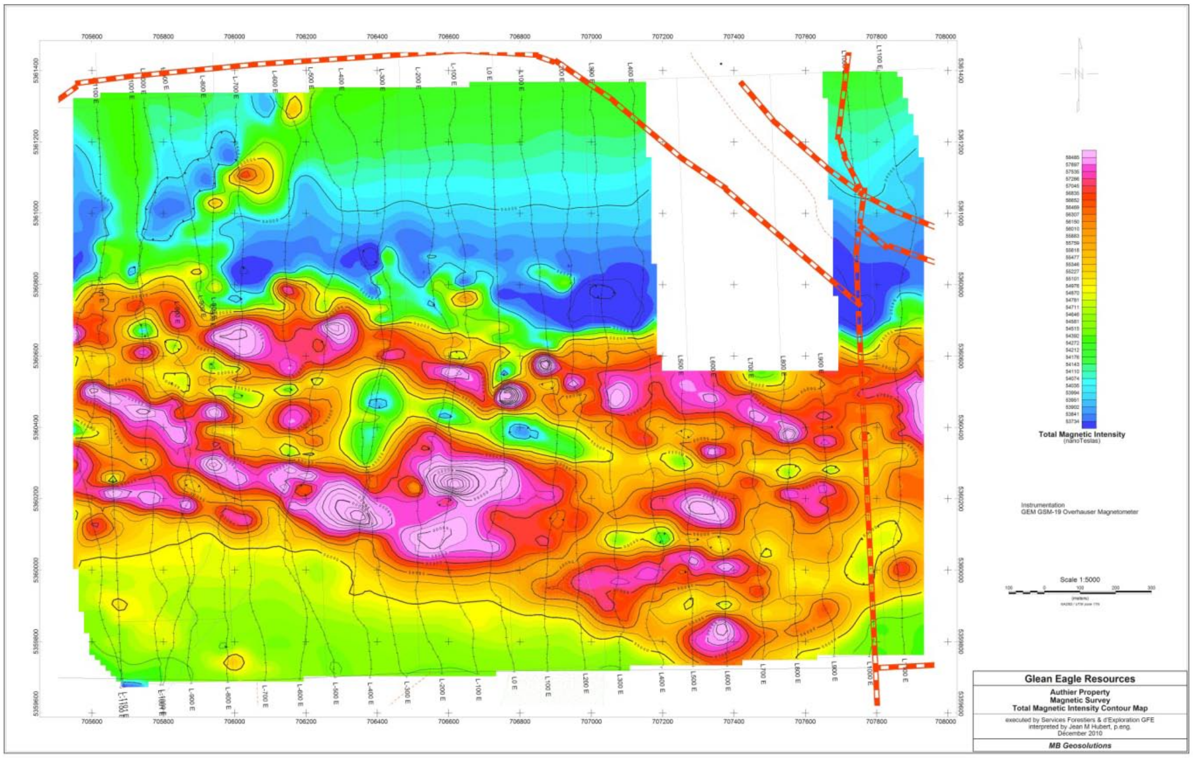


Figure 9-1: 2010 Authier Property Magnetic Survey

10 Drilling

10.1 Drilling Methodologies

This section is based on information provided by Glen Eagle and observations made during the independent verification program conducted at the Property by SGS Geostat on July 30, 2012.

The Company contracted Services Forestiers et d'Exploration GFE ("Services GFE") for the management of the exploration work for the Authier property. Services GFE provided the office, core logging and storage facilities to the Company which are located less than 4 km southeast from the main pegmatite near the town of La Motte.

The evaluation of the geological setting and lithium mineralisation on the Property includes observations and sampling from surface (through mapping and grab samples from trenches) but is principally based on information and sampling from diamond drilling. The drill core logging and sampling was conducted at the Property or at the nearby Service GFE facilities. All samples collected by Glen Eagle were sent to ALS Minerals (Val d'Or, Quebec & Vancouver, BC.) during the course of the 2010 exploration program and to AGAT Laboratories (Mississauga & Sudbury, Ontario) during the course of the 2011-2012 exploration programs. The remaining drill core is currently stored at the nearby Services GFE facilities.

All drill core handling was done on site with logging and sampling processes conducted by employees and contractors of Glen Eagle. The observations of lithology, structure, mineralisation, sample number and location were noted by the geologists and geotechnicians on hardcopy then recorded in a Microsoft Access digital database. Copies of the database are stored on external hard drive for security.

Drill core of NQ size was placed in a wooden core boxes and collected twice a day at the drill site 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 logged and sampling intervals were defined by a geologist. Before sampling, the core was photographed using a digital camera and the core boxes were identified with Box Number, Hole ID, From and To, using aluminum tags. Due to the hardness of the pegmatite units, the recovery of the drill core is generally very good and samples are representative of the mineralisation.

Sampling intervals were determined by the geologist, marked and tagged based on observations of the lithology and mineralisation. The typical sampling length is 1.5 m but can vary according to lithological contact between the mineralised pegmatite and the host rock. In general, at least one host rock sample was collected each side from the contacts with the pegmatite. The drill core samples were split in two halves with one half placed in a new plastic bag along with the sample tag; the other half was replaced in the core box with the second sample tag for reference. The third sample tag was archived on site. The samples were then catalogued and placed in a rice bags or sealed pails for shipping. The sample shipment forms were prepared on site with one copy inserted in one of the shipment bags and one copy kept for reference. The 2010 samples were transported on a regular basis by Glen Eagle's employees or contractors directly to the ALS facilities in Val d'Or.

At the ALS laboratory, the samples shipment is verified and a confirmation of shipment reception is emailed to Glen Eagle's project manager. The 2011-2012 samples were collected on a regular basis by AGAT employees directly at the GFE Val-d'Or facilities and sent to the AGAT Sudbury preparation facilities. At the AGAT laboratory, the samples shipment is verified and a confirmation of shipment reception is emailed to Glen Eagle's project manager A confirmation of shipment reception is emailed to Glen Eagle's project manager. The remaining core samples kept for reference are stored in covered metal racks at Services GFE facilities.

The drill holes in the vicinity of the pegmatite intrusion are generally spaced 25 m to 50 m apart and cover an area 650 m east-west and 300 m north-south. The drill holes are generally oriented N180° and dip between 45° to 70° to a maximum depth of 185 m below surface. The mineralised drill intersection ranges from near true thickness to 85% true thickness. Based on the drill data, the mineralised pegmatite intrusion has been modeled in 3 dimensions. Please refer to section 14for additional details on the modeling and mineral resource estimate of the mineralised pegmatite intrusion.

SGS Geostat validated the exploration processes and core sampling procedures used by Glen Eagle as part of an independent verification program. SGS Geostat found that the 2012 sampling program differed slightly form Glen Eagles established sampling program. The sample intervals were taken according to drill runs and incorporated two lithological units on several occasions.
This action implies a dilution in the assay sample results that should be taken according to the mineralised sections and lithological units.

The Drill hole deviation (dip and azimuth) was measured by a Flexit tool. Measurements are made at the beginning (25 m below surface) and at the end of the hole length. An intermediate measure was done when drill hole length exceeded 150 m. All drill hole casings were left on unless specified and closed by a steel cover. The position and orientation of the drill hole casing is surveyed and the survey values are recorded as the final coordinates and hole orientation in the database by a independent and qualified land surveyor. The Universal Transverse Mercator (UTM), 1983 North American Datum (NAD83) system was used to record position data.

Rock Quality Designation ("RQD") measurements indicate that the hanging wall rock units (mafic volcanics) are very competent. The footwall units, occurring as they do in an active structural zone, has varied RQD measurements ranging from poor to good. Although the finding of 2012 sampling program is significant, SGS Geostat concluded that the drill core handling, logging and sampling protocols are at conventional industry standard and conform to generally accepted best practices. The author considers that the samples quality is good and that the samples are generally representative. Finally, SGS Geostat is confident that the system is appropriate for the collection of data suitable for the estimation of a NI 43-101 compliant mineral resource estimate. SGS Geostat recommends keeping the standard sampling procedures involving the separate sampling of lithological units.

10.2 Historical Drilling

During the years, several exploration companies conducted drilling programs mainly focussing on the Authier Pegmatite. Please see: sub section 6.2. A total of 10,745 metres of historical drilling were done on the Property, see: Figure 10-1: Two historical drill holes were drilled on the western extension of the pegmatite dyke for a total of 190.5 metres. Six diamond drill holes totalling 619.96 metres were done in the central portion of the spodumene-bearing pegmatite.

10.3 Drilling Done by Glen Eagle

From 2010 to 2012, Glen Eagle completed 8,990 m in 69 diamond drill holes on the Authier Property. 7,959 meters were drilled on the Authier deposit; 609 meters (5 DDH) were drilled on the Northwest and 422 metres on the south-southwest 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 mineralised drill intersection ranges 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 which can be a few metres thick and up to 200 metres in length. Based on the information gathered from the drilling, the pegmatite intrusion is more than 700 metres in length and can be up to 60 metres thick. The intrusion is generally oriented east-west, dip to the north at an angle ranging between 35 and 50 degrees and is reaching depth of up the 225 metres below surface. Table 12-1 describes the distribution of the drilling on the Property including historical drilling. Figure 10-1: shows, in plan view, the historical and recent drilling conducted in the vicinity of the main pegmatite intrusion at the Authier property. Please refer to sub section 14.3 for additional information on the geological interpretation of the deposit and examples of section views.

Table 10-1: Summary Description of Drilling done on the Property

Period	Drill Holes Series	Number of Holes	Metres Drilled	Number of Survey Record	Number of Lithological Record	Number of Assays Record	% Assayed Metres
Historical	AL-XX	21	2,375	90	737	413	25%
	R-93-XX	33	3,700	71	178	258	18%
Glen Eagle Res.	AL-10-XX	18	1,905	73	171	582	41%
	AL-11-XX	27	4,051	93	137	462	17%
	AL-12-XX	24	3,034	84	126	428	21%
Total		123	15,065	411	1,349	2,143	22%

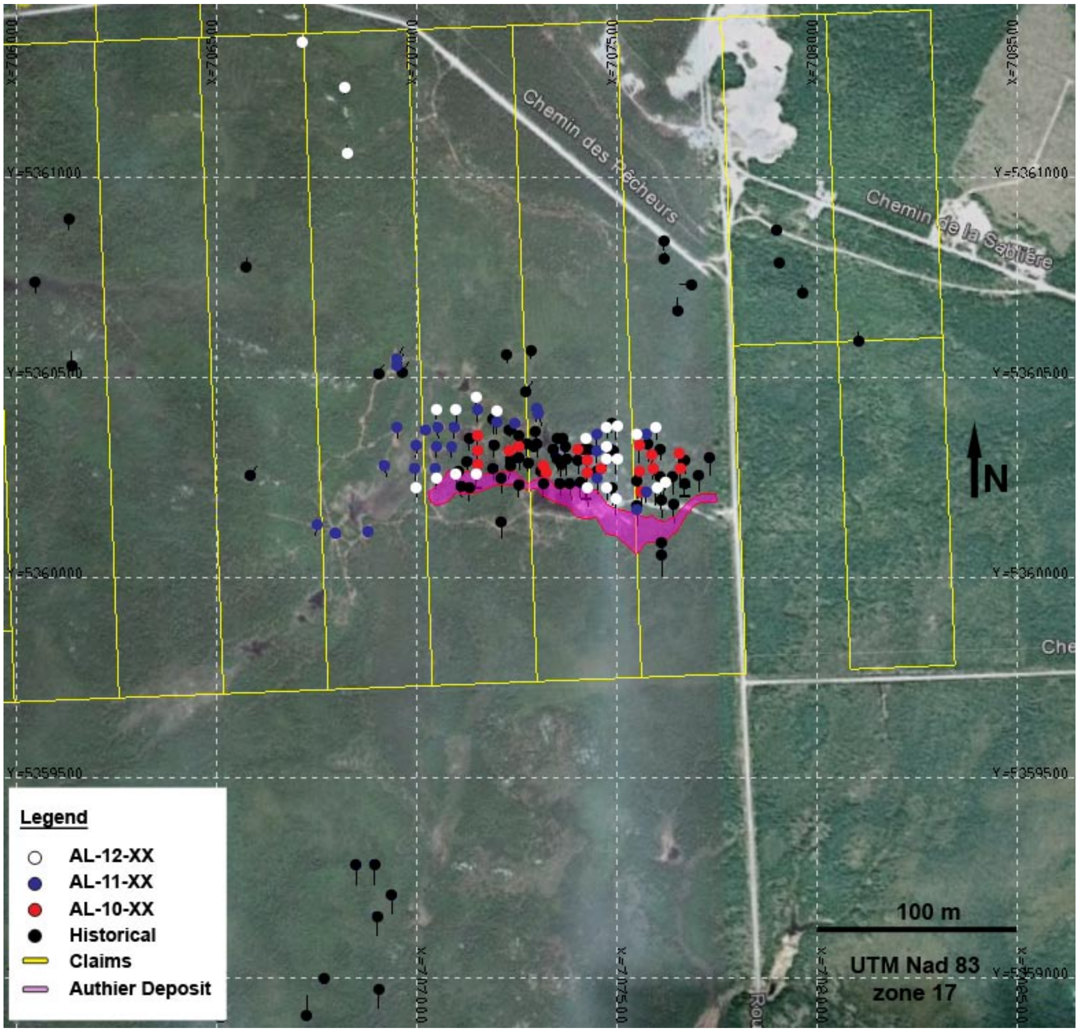


Figure 10-1: Authier Lithium property Diamond Drill Hole Location Map

11 Sample Preparation, Analysis and Security

Drill core samples collected during the 2010 exploration programs are transported directly by Glen Eagle representatives to the ALS laboratory ("ALS") facilities in Val d'Or, Quebec for sample preparation. The submitted samples are pulverized there to respect the specifications of the analytical protocol and then shipped to ALS laboratories in North Vancouver, BC for analysis. Drill core samples collected during the 2011-2012 exploration programs are collected by the employees from AGAT Laboratories ("AGAT") at the GFE Val-d'Or office and transported directly to the AGAT preparation laboratory facilities in Sudbury, Ontario for sample preparation. The submitted samples are pulverized there to respect the specifications of the analytical protocol and then shipped to AGAT laboratories in Mississauga, Ontario for analysis.

11.1 ALS Minerals 2010 Procedures

All 2010 samples received at ALS were processed according to the following procedures. All samples received at ALS are digitally inventoried using bar-code then weighted. Drying is done to samples having excess humidity. Sample material is crushed in a jaw and/or roll crusher to 70% passing 9 mesh. The crushed material is split with a rifle splitter to obtain a 250 g sub-sample which is then pulverised to 85% passing 200 mesh using a single component (flying disk) or a two components (ring and puck) ring mills.

The analyses are conducted at the ALS laboratory located in North Vancouver, BC, which is an accredited laboratory under ISO/IEC 17025 standards. There are two analytical methods used for the samples from the Authier Lithium Deposit. The first analytical method used by ALS is the 38 elements analysis (not including lithium) using lithium metaborate fusion followed by Inductively Coupled Plasma Mass Spectrometry ("ICP-MS") (ALS code ME-MS81). This method uses 0.2 g of the pulverised material and returns different detection limit for each element. The second analytical protocol used at ALS is 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 returns 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 the 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% Li (SGS code ICP90Q). This method uses 20 g of pulp material.

11.2 AGAT Laboratories 2011-2012 Procedures

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

The analyses were conducted at the AGAT laboratory located in Mississauga, Ontario, which is an accredited laboratory under ISO/IEC 17025 standards. 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% Li.

11.3 Quality Assurance and Quality Control Procedure by Glen Eagle

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 in the insertion of reference material, analytical standards and blanks, on a systematic basis with the samples shipped to ALS. The company also sent pulps from selected mineralised intersection to SGS Minerals for re-analysis. SGS Geostat did not visit the ALS or SGS Minerals facilities, or conduct an audit of the laboratories.

11.3.1 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") standards. Both standards are custom made reference materials from mineralised material coming from the main pegmatite intrusion at Authier. In order to evaluate their expected values, both Low-Li and High-Li standards have been analysed 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 is the mineral grade sodium peroxide fusion with ICP-OES finish described in section 11.1. The analytical protocol used at ALS is the ore grade lithium four-acid digestion with ICP-AES finish described in section 11.1.

For the Low-Li standard, the analytical results returned from SGS Minerals for the 15 samples average 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 analysed 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 analysed. 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 are use to determine the expected values (mean value from the 30 samples) and the QAQC warning/failure thresholds (±2 standard deviations and ±3 standard deviations respectively). Table 11-1 shows the results for each standard using both analytical protocols.

Table 11-1: 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 (Li2O %)		
	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

11.4 2010-2012 Reference Materials Results

In 2010, Glen Eagle sent their samples to ALS Minerals of Vancouver and starting in 2011, to AGAT of Mississauga, Ontario. During this period, 31 high-Li and 32 low were inserted to in the sampling procedure. A Graphic representation of RM quality control failures and the labelling results are included in the upcoming graphics. In the next figure, the red lines represent the absolute limits of three times the standard deviations (± 3σ). Out of a total of 63 RM since 2010, 7 RM (11%) have produced results exceeding ±3σ the expected value. Similarly, only 2 RM (3%) have produced results exceeding 10% the expected value. Almost all RMs fall under the 10% difference from the expected RM value.

11.4.1 Z-scores

The Z scores were also calculated and plotted on the next figure. 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.

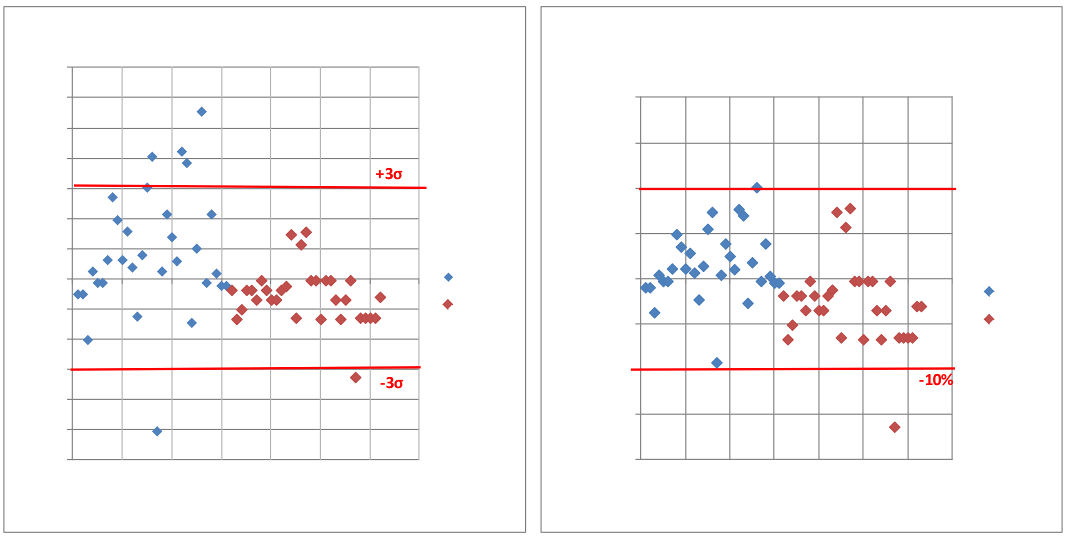


Figure 11-1: RM (Std High, STD Low) Results

11.4.2 ALS Minerals 2010 Reference Materials Results

In 2010, Glen Eagle sent his samples to ALS Minerals of Vancouver. In the next 2 figures, 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 31 RMs, 2 RMs (6%) have produced results exceeding ±3σ the expected value. Additionally, no RM has produced results exceeding 10% the RM expected value. Possible mislabels are included in this analysis.

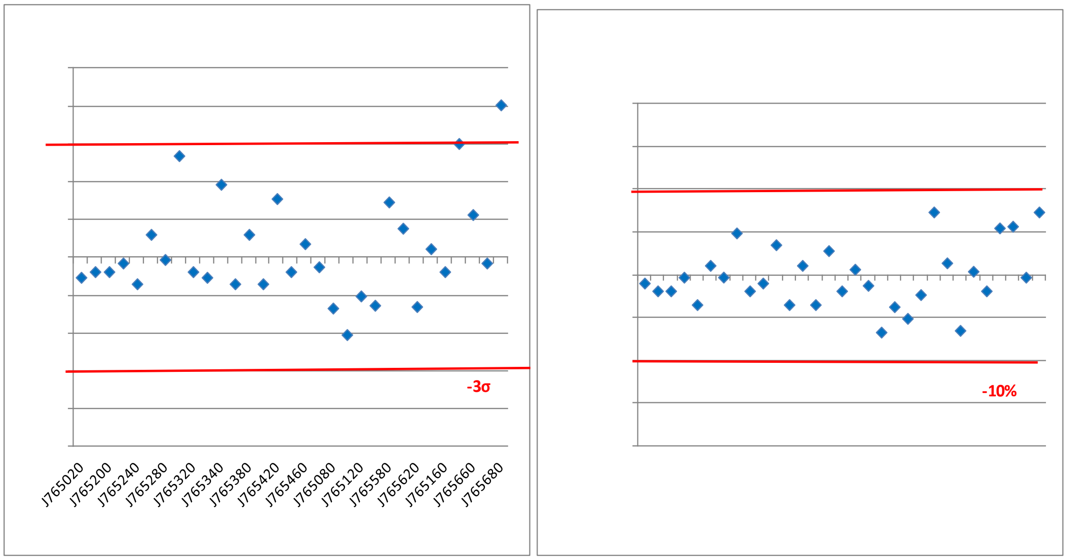


Figure 11-2: ALS 2010 RM Z-Score & Percentage from Expected RM Value

11.4.3 AGAT 2011-2012 Reference Materials Results

Starting 2011, Glen Eagle sent his samples to AGAT Laboratories of Mississauga. In the next figures, 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, 5 RM (15%) have produced results exceeding ±3σ the expected value. Additionally, 2 RMs have produced results exceeding 10% the expected value. Possible mislabels are included in this analysis.

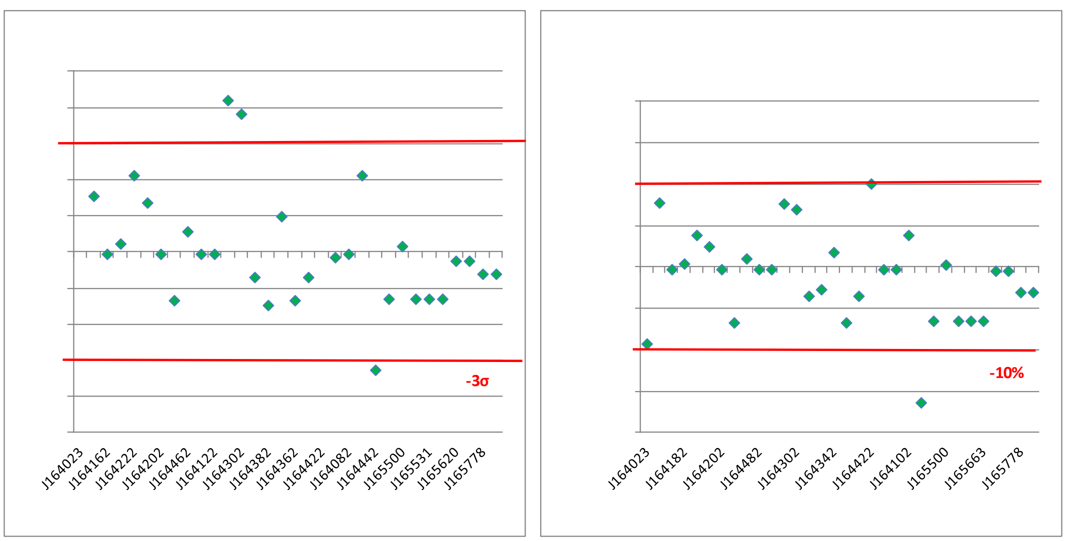


Figure 11-3: AGAT 2011-2012 RM Z-Score & Percentage from Expected RM Value

SGS Geostat is of the opinion that certain RMs have been mislabelled and that the Company must do adequate follow-up on these analyses accompanied with a detailed QAQC report. SGS Geostat has received confirmation that a follow-up is currently being done by Glen Eagle.

12 Data Verification

The digital drill hole database supplied by Glen Eagle was validated for the following fields: collar location, azimuth, dip, hole length, survey data and analytical values. 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 authorised 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 Authier Lithium property final drill hole database includes historical drill hole information and the recent (Glen Eagle) drilling information from the 2010-2012 surface drilling program up to hole number AL-12-24. The database cut-off date is September 20, 2012. Table 12-1 summarizes the data contained in the Property final drill holes database including all relevant data used for the Mineral Resources Estimates. SGS Geostat is of the opinion that the final drill holes database is adequate to support the Mineral Resources Estimates.

Table 12-1: Summary of Final Authier DDH Database, Sept 20, 2012

Period	Number of DDH	Metres drilled	Number of survey records	Number of lithology	Number of Assay records	metres sampled (m)
Historical	81	10745	1497	1248	1456	2356
2010	18	1905	60	171	582	787
2011	27	4051	68	137	462	693
2012	24	3034	57	126	428	640
Total	**150**	**19735**	**419**	**1682**	**2928**	**4476**

12.1 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 the Company 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) mineralised drill core duplicates were collected from holes AL-10-01 and AL-10-11 by SGS. 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 arrived to this conclusion. The 2010 analytical bias showed were not very significant with the duplicate samples returning an average Li_2O value 7.9% higher compare to the original samples.

12.2 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 Lithium 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 30th, 2012 site visit by the author, Maxime Dupéré P.Geo., a total of 38 mineralized core duplicates from the Authier Lithium 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 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% 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 12-1 shows correlation plots for the check data versus the original data. A summary of the statistical analysis conducted on the data is shown in Table 12-2.

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 for shows that the proportion of pairs with an old sample value greater than the new samples 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, results show a clear variability of assay results between laboratories. It is SGS Geostat and the author's opinion that this difference in favor of the 2010 samples for ALS and the one in favor 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.

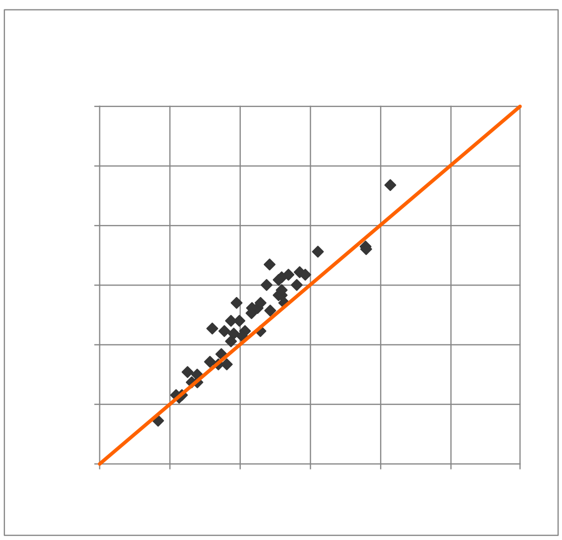


Figure 12-1: Correlation Plot for Independent Check Samples

Table 12-2: 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 39%	15 48%	4 13%
> 1.5%	4	2 50%	2 50%	0 0%

12.3 Twinning of Historical Drill Holes

This part is summarised from the 2010 Authier Lithium technical report.

In order to validate the historical drilling data, SGS Geostat recommended that the Company complete twin holes of selected historical drill holes from the AL-XX and the R-93-XX series. In 2010, following SGS recommendations, Glen Eagle completed 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 mineralised 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 compare to the original mineralised interval of 0.90% Li_2O over 43.28 m intersected in hole R-93-01.

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

Hole AL-10-01 intersected the mineralised 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 compare to the original mineralised interval of 1.25% Li_2O over 49.38 m intersected in hole R-93-25.

Due to localisation difficulties encountered in the field by the Company, 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 mineralised 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 compare to the original mineralised interval of 0.69% Li_2O over 15.24 m intersected in hole AL-18

Hole R-93-12 intersected a mineralised 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 mineralised interval of 0.92% Li_2O over 11.52 m intersected in hole AL-24.

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

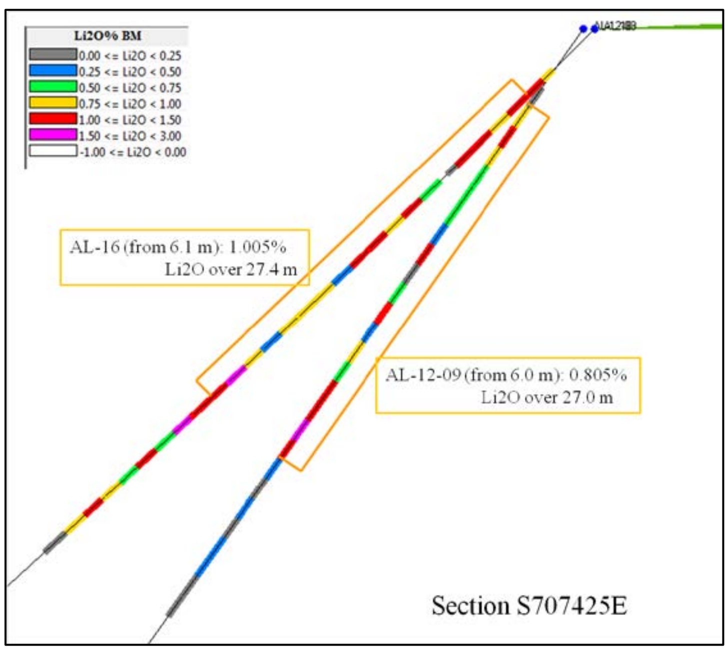


Figure 12-2: Oblique View Showing Results for Twin Holes Al-16 and AL-12-09

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

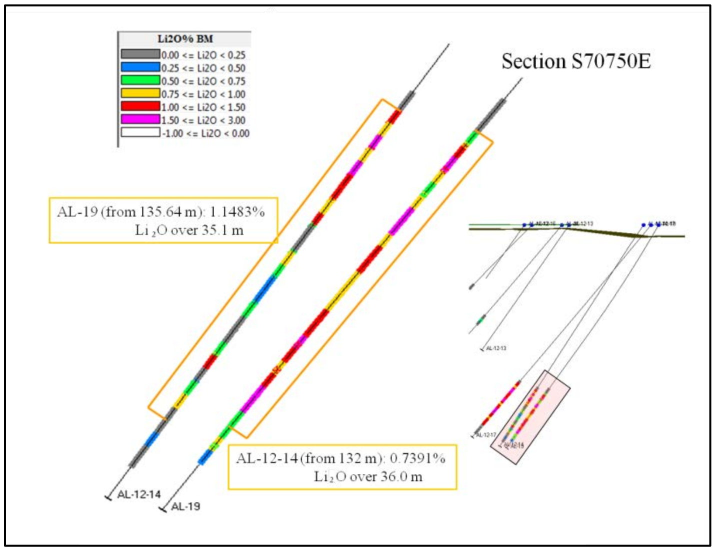


Figure 12-3: Oblique View Showing Results for Twin Holes Al-19 and AL-12-14

Considering the significant 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 mineralised intercepts of which returned Li_2O grade differences in excess of 30% and 40% differences respectively. No systematic analytical bias was outlined. Based on 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 12-3 summarises the overall results of the 2010-2012 twin hole drilling program.

Table 12-3: 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%
Al-10-11	38.55	74.45	35.90	0.87	
R-93-13	7.16	15.24	8.08	1.29	9.82%
Al-10-06	6.55	15.10	8.55	1.17	
R-93-13	31.09	40.84	9.75	1.14	31.63%
Al-10-06	32.70	41.00	8.30	0.83	
R-93-25	76.20	125.58	49.38	1.25	8.11%
Al-10-01	72.00	123.25	51.25	1.35	
AL-18	96.62	111.86	15.24	0.69	54.72%
Al-10-15	81.00	96.40	15.40	1.20	
Al-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 12-1 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 the favor of the duplicate (SGS) Li_2O results, SGS Geostat is in the opinion that the final drill hole database is adequate to support mineral resources estimations.

12.4 Specific Gravity

As part of the 2010 independent data verification program, SGS Geostat conducted specific gravity ("SG") measurements on 38 mineralised core samples collected from drill holes AL-10-01 and AL-10-11. The measurements were performed using the water displacement method (weight in air/volume of water displaced) on representative half core pieces weighting between 0.67 kg and 1.33 kg with an average of 1.15 kg, results average SG value of 2.71 t/m^3 (Table 12-4).

Table 12-4: Specific Gravity Measurements Statistical Parameters

Authier Project - Spodumene pegmatite S.G. (t/m^3)	
Count	38
Mean	2.714
Std Dev	0.007
Relative Std Dev	0.25%
Minimum	2.642
Median	2.714
Maximum	2.813

13 Mineral Processing and Metallurgical Testing

13.1 Metallurgical Tests done in 1999

In 1999, COREM conducted metallurgical testing of approximately 40 tonnes of spodumene-bearing pegmatite material sampled from the main mineralised pegmatite intrusion at the Authier property. The metallurgical testing was conducted under the supervision of Bumigeme who was conducting a pre-feasibility study of the Project during that period.

During the Property visit, the primary author visited the site where the bulk sample was collected. Based on the observation made in the field, the author believes the bulk sample was representative of the general composition of the mineralised pegmatite.

The metallurgical study was originally designed with two phases: an initial gridding and concentration metallurgical study in laboratory followed by a confirmation of the phase one results using a pilot plant. Due to the difficulty to obtain the anticipated results in the first and second phases, a third testing phase conducted in laboratory was necessary. Only the third phase of metallurgical testing is described in the section.

The complete metallurgical study conducted in laboratory consisted in a total of 48 tests but only 16 tests returning satisfactory results were reported. The last process flowsheet test of the metallurgical study, which was outlined as the most successful according to Bumigeme, is shown in Figure 13-1.

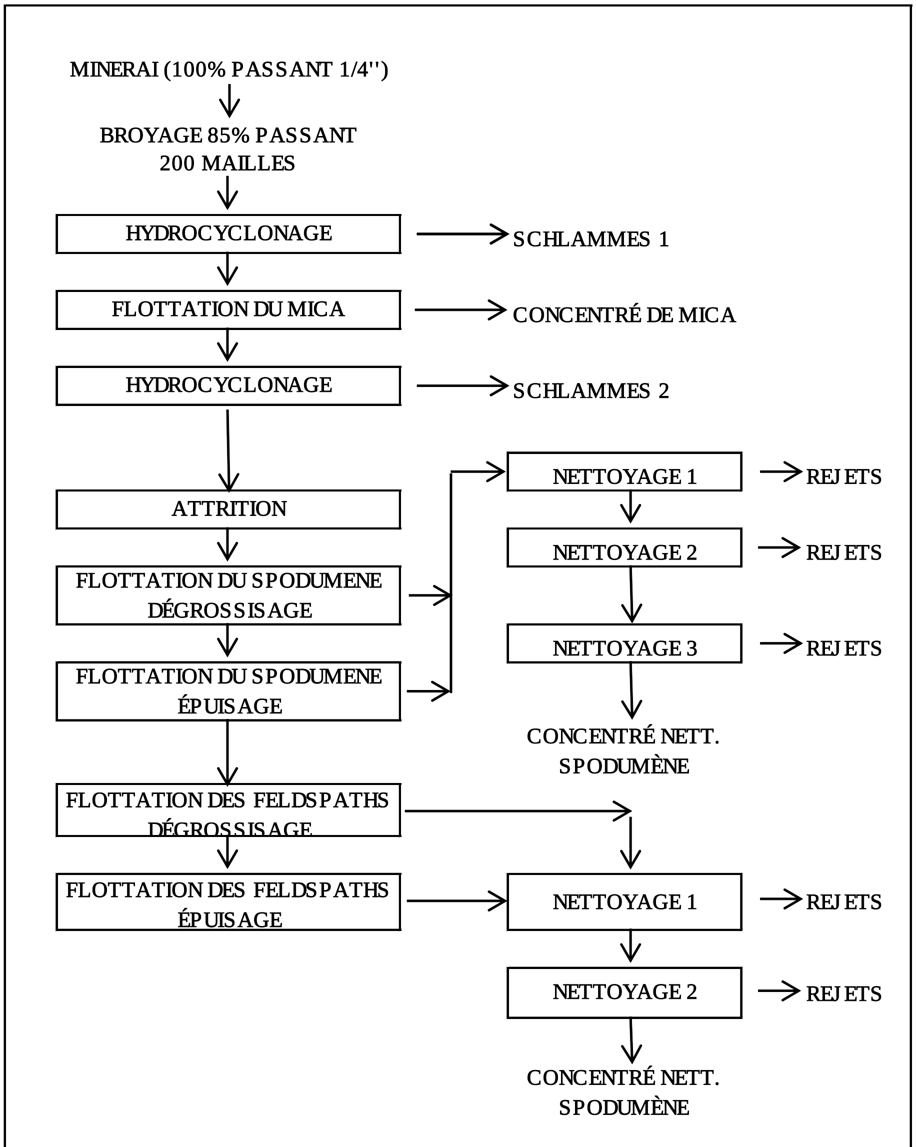


SCHÉMA DE TRAITEMENT - PROCÉDÉ ALTERNATIF NO 2

Figure 13-1: Flowsheet of Alternative Process Number 12 of the 1999 Metallurgical Study

The most significant results from the process flowsheet shown in Figure 13-1 returned a Li_2O concentrate grade ranging from 5.78% to 5.89% with a recovery between 67.52% and 70.19% (tests 33 and 47). The average Li_2O grade of the pegmatitic material from tests 33 and 47 were 1.15% and 1.13% Li_2O respectively. Another test conducted early in the metallurgical study using a different process flowsheet also returned significant results. Test number 12, which sample average grade was 1.35% Li_2O, produced a Li_2O concentrate grade of 5.96% with a recovery of 75.02%.

13.2 Metallurgical Tests done in 2012

This section was prepared by Bumigeme Inc, from Montreal, who was mandated by Glen Eagle to prepare the metallurgical part of the PEA study in collaboration with SGS Geostat.

13.2.1 Test Work Samples

Drill core samples from the Authier property, located 45 km North West of Val d'Or, Quebec were submitted to SGS Mineral Services for metallurgical testwork by Glen Eagle Resources Inc. A total weight of 270 kg was sent to SGS Lakefield in Ontario. One composite sample was prepared from these drill cores. The inventory list of drill core samples is provided in the following Table 13-1. The sample preparation procedure is shown in Figure 13-2.

Table 13-1: Diamond Drill Core Samples

SAMPLE ID	WEIGHT (kg)	SAMPLE ID	WEIGHT (kg)	SAMPLE ID	WEIGHT (kg)
J765656	3,38	J765637	3,68	J165451	4,14
J765664	3,45	J765636	3,42	J165452	3,53
J765666	3,36	J765634	3,22	J165450	4,07
J765658	3,36	J765635	3,5	J165453	3,71
J765659	3,36	J765638	3,45	J164130	3,39
J765660	3,36	J765639	3,45	J164132	3,19
J765661	3,05	J765640	3,45	J164133	3,29
J765662	3,36	J165442	3,56	J164131	3,14
J765665	3,36	J165443	3,86	J164134	3,52
J765663	3,33	J165444	3,86	J164114	3,1
J765667	2,22	J165447	3,74	J164115	3,18
J765657	3,16	J165448	3,73	J164116	3,54
J765655	3,17	J165445	3,68	J164117	3,52
J765654	3,27	J165446	3,68	J165460	3,52
J765652	3,31	J164127	3,64	J165459	3,8
J765649	2,92	J164126	3,2	J165461	3,7
J765653	3,06	J164125	3,53	J164111	3,39
J765651	3,26	J164129	3,13	J164112	2,75
J765650	3,4	J164128	3,29	J164113	3,47
J765646	3,28	J164118	3,58	J164110	3,32
J765648	3,58	J164119	3,15	J165456	3,78
J765645	3,39	J164120	3,15	J165454	3,76
J765644	3,26	J164123	3,26	J165455	3,61
J765647	3,13	J164121	3,57	J165458	3,49
J765642	3,45	J164122	3,57	J165457	3,75
J765643	3,11	J164124	3,27		
J765641	3,33	J165449	3,86		

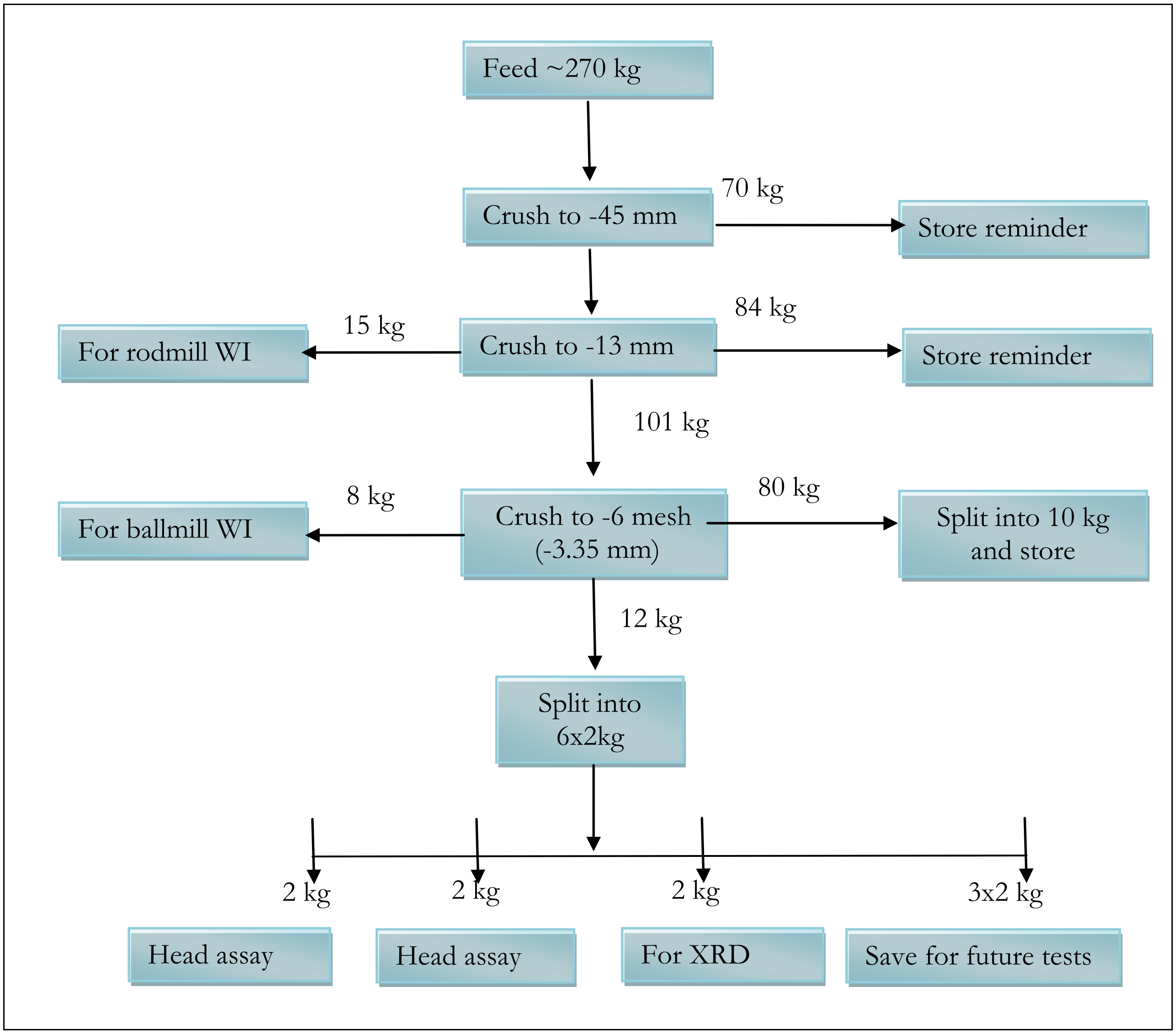


Figure 13-2: Sample Preparation Flowsheet

The results of the two head samples submitted for chemical analyses are presented in Table 13-2.

Table 13-2: Chemical Analysis of Representative Samples

	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI
A	0.57	1.23	74.8	15.8	0.58	0.07	0.17	4.27	3.08	<0.01	0.02	0.1	0.02	<0.01	0.43
B	0.57	1.23	75	15.8	0.59	0.07	0.18	4.16	3.08	<0.01	0.02	0.1	0.02	<0.01	0.36

13.2.2 Mineralogical Analysis

A subsample from the master composite was submitted for XRD determination and the semi-quantitative head composition is shown in Table 13-3. Test details can be found in Appendix 3.

Table 13-3: Head Sample Mineral Composition

Mineral	**Authier Li Deposit Drill Core Sample (wt %)**
Albite	37.2
Quartz	26.5
Microcline	16.2
Spodumene	14.9
Muscovite	4.8
Magnetite	0.3
TOTAL	99.9

The head sample assay show that the composite sample consists of major amounts of albite (37.2%), quartz (26.5%), microcline (16.2%), spodumene (14.9%) and muscovite (4.8%). According to XRD analysis (Table 13-3), the sole Li mineral identified is spodumene.

13.2.3 Grinding Test Work

- **Bond Rod Mill Work Index**

The master composite was submitted for Bond rod mill work index (RWI) determination as per the standard procedure using a product D100 size of -1180 µm. A Bond rod mill work index (RWI) of 12.3 kWh/t (metric) was obtained for the composite (see Appendix 3 for details. The sample is characterized as soft in RWI terms with a percentile of 27%.

- **Bond Ball Mill Work Index**

The master composite was submitted for Bond ball mill work index (BWI) determination as per the standard procedure using a product D100 size of -150 µm. A ball mill work index (BWI) of 15.6 kWh/t (metric) was obtained for the head composite (see Appendix 3 for details). The sample was found to be of medium hardness with a percentile of 64%.

13.2.4 Flotation tests work

The flotation tests were carried out using a Denver D1/D2 flotation cell and D12 flotation machine. All stage grinding was performed in a 2 kg mill. Stainless steel rods were used as the grinding media in all tests, and the closing size for each test is shown in the flotation test details. Scrubbing was performed in a Denver flotation cell at pH of around 11 adjusted with NaOH and in the presence of lignin sulfonate (D618) for 10 minutes. Slime separation after scrubbing was done by settling and decanting in an acrylic container.

A series of batch flotation tests was conducted in various conditions: with mica pre-flotation and without mica pre-flotation; varying the particle sizes (300µm, 210µm, 100µm). Various types of reagents were tested. After passing the concentrate through WHIMS (Wet High Intensity Magnetic Separator), a spodumene concentrate grading 6.09% Li_2O was generated at 88% (global) recovery after two cleaning stages without mica pre-flotation (the summary of the test F8 is indicated in the Table 13-4 below)

Table 13-4: Summary of Test F8

	Weight		Assay (%)		Distribution (%)	
Combined product	**g**	**%**	**Li_2O**	**Fe_2O_3**	**Li**	**Fe_2O_3**
Li Non Mag Final Conc. 15A	305.0	16.13	6.44	1.06	85.0	23.8
Li 4th Cleaner Concentrate	315.8	16.70	6.29	1.58	86.1	36,8
Li 3rd Cleaner Concentrate	323.2	17.10	6.21	1.58	87.0	37.6
Li 2nd Cleaner Concentrate	333.3	17.63	6.09	1.57	88.0	38.6
Li 1st Cleaner Concentrate	347.8	18.40	5.89	1.57	88.8	40.3
Li Rougher Concentrate	353.3	18.69	5.81	1.56	88.9	40.5
Li Rougher Conc. & Scav. Conc.	411.3	21.76	5.07	1.52	90.2	46.3

Flotation kinetic was fairly fast and within one minute, a large portion of the mass, more than 60%, reported to the concentrate. In batch cleaner testing, the following key points have been identified:

- Fatty acid (FA-2) is a suitable spodumene collector as its performance is better than those of its alternatives Aero 704 and 845.

- Stage grinding to a closing size of 150 µm and a P80 around 120 µm was required to achieve Li concentrate with good grade and recovery.
- Scrubbing with lignin sulfonate at pH 11 followed by desliming was also beneficial.
- Implementing two (2) or a maximum of three (3) cleaning stages was a suitable flotation strategy.
- The iron content of the lithium concentrate was high (1.25% Fe_2O_3). It was only possible to remove a portion of the iron in the concentrate (not associated with spodumene as solid solution) using Wet High Intensity (15000gauss) Magnetic Separation.
- Mica pre-flotation was not required to achieve high grade concentrate.

For the metallurgical testing details, see the SGS Minerals Services for metallurgical test-work report enclosed in Appendix 3.

14 Mineral Resource Estimates

14.1 Introduction

This section reports the results of a Mineral Resources Estimates (“MRE”) for the Authier deposit. The Mineral Resources Estimates herein was completed by SGS Geostat and was disclosed in the Company news release dated November 19, 2012.

The mineral resources have been estimated by Maxime Dupéré, P.Geo., for SGS Geostat. Mr. Dupéré is a professional geologist registered with the Ordre des Géologues du Québec and has worked in exploration for gold and diamonds, silver, base metals and iron ore. The author has been involved in mineral resource estimation work on a continuous basis over different mineral deposits since he joined SGS Canada Inc. in 2006. Mr. Dupéré is an independent Qualified Person as per section 1.5 of the NI 43-101 Standards of Disclosure for Mineral Projects with respect to the owner of the mineral titles included in the Project.

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” stated by the NI 43-101 regulation. An optimise shell as made over the Authier deposit. The MRE stated herein do not have demonstrated economic viability.

14.2 Glen Eagle Authier Deposit Database

SGS Geostat conducted the current Mineral Resources Estimates for the Authier deposit using the validated historical and 2010-2012 diamond drill holes from Glen Eagle Authier property. The database used to produce the MRE is derived from a total of 81 historical and 69 recent 2010-2012 surface drill holes, and contains the collar, survey, lithologies, and analytical results. The database cut-off date is September 20, 2012 (refer to the Data Verification section for a summary information of the records contained in the final drill hole database).

From this database, 100 drill holes were considered for the update of the geological interpretation and passing through the mineralized pegmatite: 45 historical (4782 m) 18 in 2010 (1905 m), 20 in 2011 (3104 m) and 17 in 2012 (2290 m). The Table 14-1 summarises the drill hole database crosscutting the Authier deposit.

14.2.1 Analytical Data

There are a total of 2143 assay intervals with an assigned Li_2O, value reported in the database used for the current MRE. The sampling gaps consisting of missing values and or un-assayed material in the assay intervals were considered having the 0% Li_2O grade for the purpose of the block model interpolation process. The grade distribution of the Li_2O assay data is approaching the normal law with more than one grade population observed in the data. There are 1408 assay intervals present and used in the making of the mineralised pegmatite deposit. These assays are present in the mineralised intervals. The Figure 14-1 shows the histogram of the Li_2O assay data used in the resource estimation with the statistics of the Li_2O assay results reported in the drill hole database. Assays containing -1 (for missing values) grade values were not considered in this histogram. The maximum grade is 2.61% while the minimum 0.00% Li_2O (refer to the histogram below). The median is 0.95 % and the mean are 0.94 % with a standard deviation of 0.46%.

Table 14-1: Drill hole Database Summary cross cutting the Authier Deposit

Period	Drill Holes Series	Number of Holes	Metres Drilled	Number of Survey	Number of Litholog	Number of Assays	% Assayed Metres
Historical	AL-XX	20	2271	65	531	333	21%
	R-93-XX	25	2511	71	178	201	22%
Glen Eagle Res.	AL-10-XX	18	1905	42	171	302	22%
	AL-11-XX	20	3104	51	111	275	14%
	AL-12-XX	17	2290	40	84	238	19%
Total		100	12081	269	1075	1349	19%

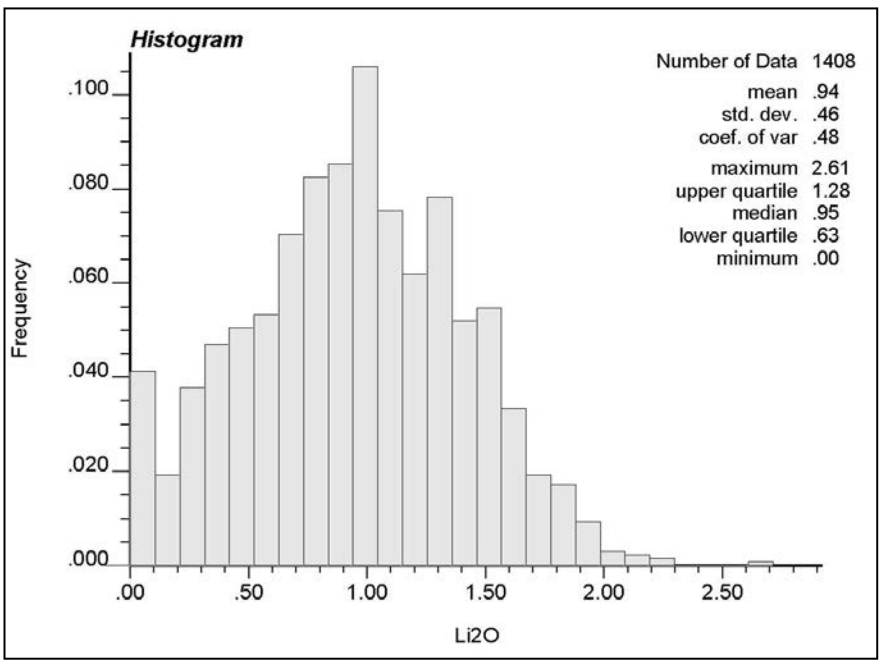


Figure 14-1: Histogram of Li_2O Analytical Data from the Authier Database

14.2.2 Composites

Block model grade interpolation was conducted on composited assay data. A composite length of 3 m was selected to reflect the average assay interval length. Compositing is conducted from the start of each mineralized intercept (Li_2O mineralization) of drill holes. A minimum length of 1.5 m was kept for the composites. The selected length of the composites directly influences the amount of dilution of the model. The longer the composites are, the more likely they will be diluted. The selected length is considered suitable in comparison to the mean length of 2.3 m and the median length of 2.41 m of the assay lengths included in the mineralized intercepts as well as the dimension of the blocks used. Assay capping was not necessary.

A total of 751 composites were generated. The modeled 3D wireframe envelope was used to constrain the grade composites. Please see: Figure showing the spatial distribution of the 3 m composites selected for the estimation of the block model derived from the 3D wireframe envelop.

The Li_2O (%) grades of the composite samples show a distribution approaching the normal law with the presence of low possibly 2 grade populations. In core samples, the maximum grade is 2.17% while the minimum 0.00% Li_2O (refer to the histogram below). The median is 0.97%; the mean is 0.96% with a standard deviation of 0.38%.

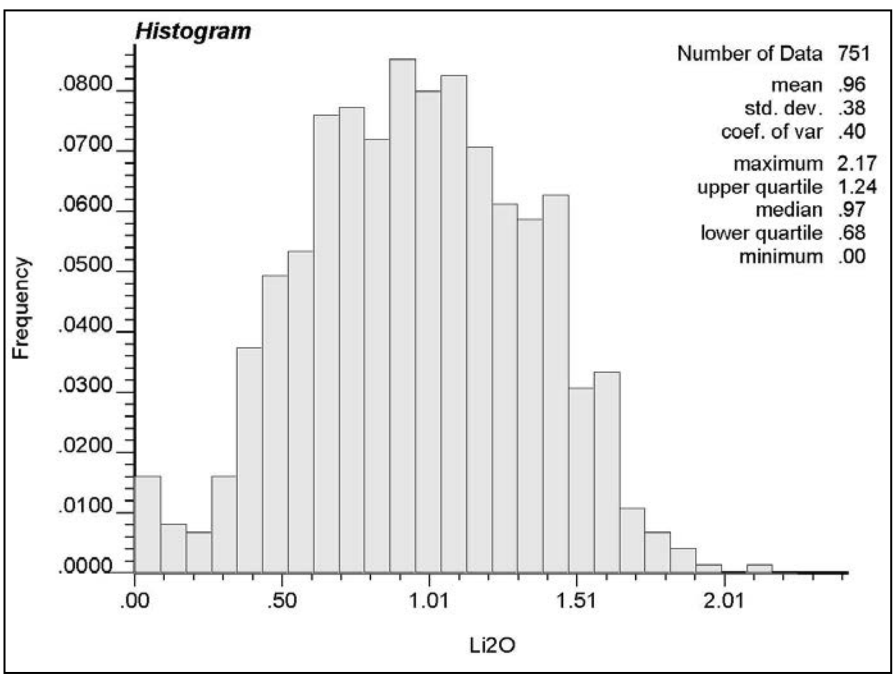


Figure 14-4: Histogram 3 m Composites Li_2O (%)

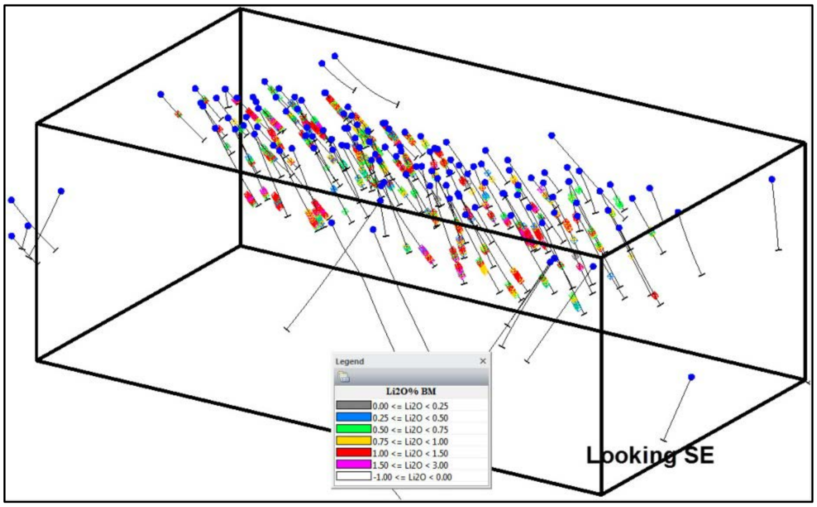


Figure 14-2: Oblique view showing the Spatial Distribution of the Composites

14.2.3 Specific Gravity (SG)

An average specific gravity of 2.71 t/m^3 was used to calculate tonnage from the volumetric estimates of the block model. The average specific gravity is derived from the 38 specific gravity measurements taken from representative mineralized core samples from the 2010 drilling period by SGS. See section 12.4.

14.3 Geological Interpretation and Modeling

SGS Geostat updated the Authier geological model using 37 additional 2011-2012 surface drill holes hitting the deposit. The interpretation was based on the lithological records and the Li_2O grades database. The geological interpretation was made on sections and linked together into a 3D meshed solid. A minimum grade of 0.5% Li_2O over a minimum drill hole interval length of 3-5 m was generally used as guideline to define the width of mineralised interpretations on sections, corresponding to the N-S width of the individual blocks. The same approach was used to define smaller 3D wireframe of significant size un-mineralised pegmatite or amphibolitic xenoliths material (waste envelopes) located within the mineralised pegmatite 3D wireframe envelop. A bedrock-overburden interface 3D surface has been generated by triangulating the lower intercept of the overburden-coded lithology from the drill hole dataset. Figure 14-3 to Figure 14-5 show the contour of the 3D solid in perspective, on section and in level plan views respectively.

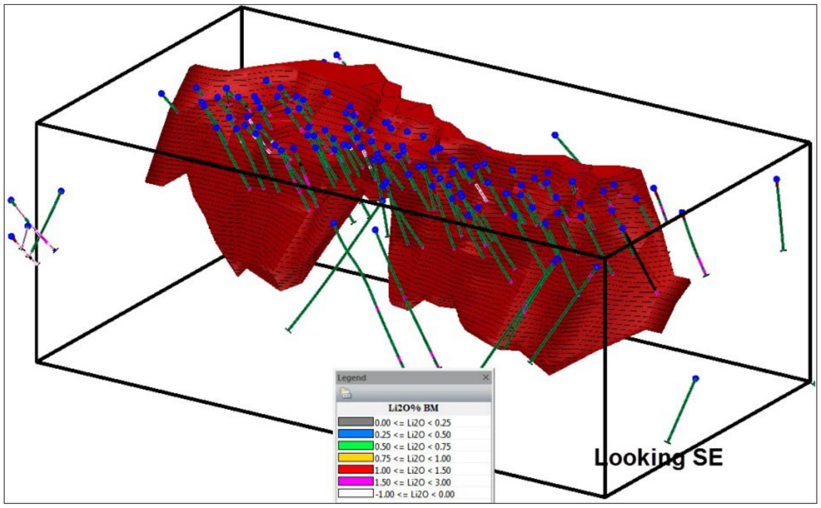


Figure 14-3: Oblique View of the Authier 3D Solid

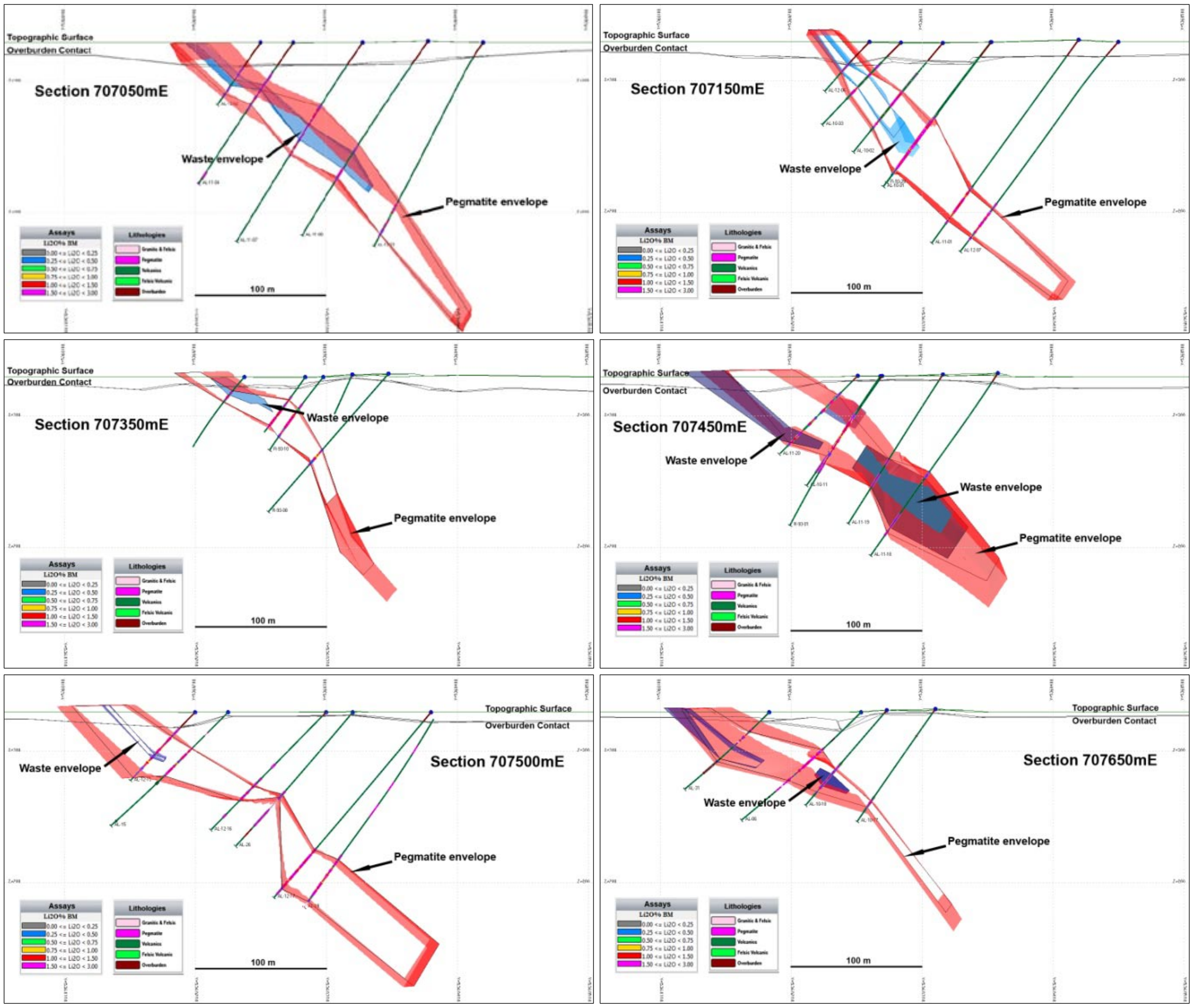


Figure 14-4: Modeled Envelops with Mineralised Intercepts on Sections (looking west)

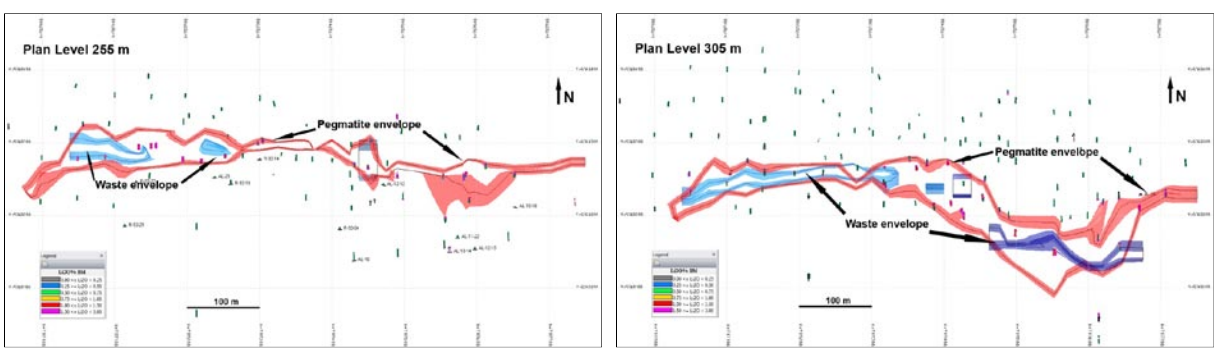


Figure 14-5: Modeled Envelops with Mineralised Intercepts on Level Plan Views

14.4 Spatial Continuity

Experimental curves and fitted models are shown on. As expected, they closely resemble those of the previous study i.e. relative nugget effect is 40% and continuity is better along dip and strike than across.

In September 2012, the spatial continuity of the Li_2O composite grades was assessed by variography. This method was also used previously in the 2010 Authier Resource report by SGS. The Experimental variograms (correlograms), which are the calculated correlation coefficient of grade from composite pairs separated by a given distance for a given direction, have been generated for 3 m Li_2O composite data. The variograms (correlograms) were produced according to various directions in order to identify potential anisotropies in the grade continuity within the mineralised pegmatite envelop (Figure 14-6). Generally, the variography is suggesting some anisotropy at relatively short distance (less than 100-125 m). There are 2 major directions of best continuity, oriented East-West and north corresponding to the general strike orientation and dip direction of the pegmatite deposit. The worst continuity is observed across (drill holes direction) the pegmatite deposit. The nugget effect is moderately low (25%).

As a result, a slight but non negligible difference in the variogram model was observed since the first resource estimation of the deposit in 2010. According to the added information (51 additional DDH, 890 additional analysis records, 263 additional 3m Composite records), the fitted model is the sum of that nugget effect of 25% (vs. 20% in the previous model) plus a short range (local) component of 50% magnitude and a long range (regional) component of 25% magnitude. Ranges of the local component are about 38m along dip, and along strike and 11m across dip+strike (vs. 50m, 15m and 15 m in the previous model) while the ranges of the regional component are about 120m along dip, 120m along strike and 60m across dip+strike (vs. 120m, 60m and 60 m in the previous model). In other words, the long distance continuity is less extensive but more continuous along strike and dip in comparison to the previous study. The Table 14-2 shows the variogram Model used for the resource estimation.

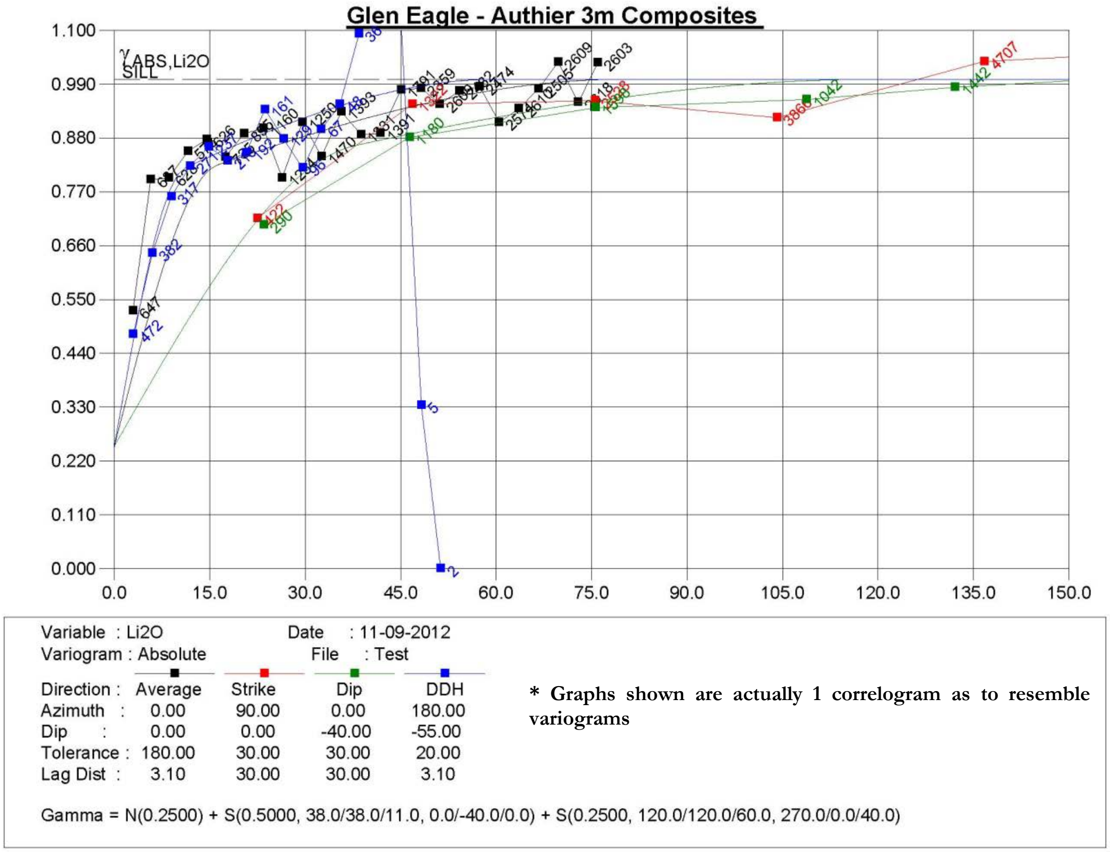


Figure 14-6: Variograms of Li_2O Grade 3m Composites

Table 14-2: Variogram Model of Li_2O 3m Composites

Nugget Effect	Sill(C)	First Spherical Variogram Component						Sill(C)	Second Spherical Variogram Component					
		Ranges(in metres)			Orientation (in degrees)				Ranges(in metres)			Orientation(in degrees)		
		Max	Interm.	Min.	Azimuth	Dip	Spin		Max	Interm.	Min.	Azimuth	Dip	Spin
25%	0.25 25%	38	38	11	270	0	40	0.25 25%	120	120	60	270	0	40

14.4.1 Resource Block Modeling

A block size of 5 m (E-W) by 5 m (N-S) by 5 m (vertical) was selected for the resource block models based on drill hole spacing, width and geometry of the mineralization. The block model is constrained by the 3D wireframe solid. The global block model contains a total of 27,989 blocks.

Table 14-3: Resource Block Model Parameters

Grid NAD 83	X (EAST)	Y (NORTH)	Z (ELEVATION)
Origin:	706,900	5,360,000	340
Size:	5	5	- 5
Discretization:	2	2	2
Starting Coordinate:	706,900	5,360,000	340
Starting Block Indice:	1	1	1
Ending Coordinate:	707,800	5,360,500	0
Ending Block Indice:	181	101	69

14.4.2 Grade Interpolation Methodology

The grade interpolation for the Authier mineral resource block model was completed using the Ordinary Kriging (OK) methodology. Anisotropic search ellipsoids were selected for the grade interpolation process based on the general geometry of the pegmatite intrusion and on the analysis of the spatial continuity of Li_2O grade using variography. Limits are set for the minimum and maximum number of composites used per interpolation pass and restriction are applied on the maximum number of composites used from each hole.

The interpolation of the block grade was performed by ordinary Kriging (OK) in multiple passes using anisotropic search ellipsoids increasing in size from one pass to another. The orientation of each search ellipsoid is identical for each interpolation pass and established at 270° in azimuth, 0° dip and has a 40° spin.

In the first pass, the search ellipsoid dimension was 50 m (long axis) along strike, 50 m (intermediate axis) along dip and 25 m (short axis) perpendicular to the strike and dip. The shape of the ellipsoid corresponds to the general shape of the mineralized body. Search conditions required to estimate each block were defined by a minimum of 5 composites and a maximum of 15 composites with a maximum of 2 composites selected from each drill hole. The first pass resulted in the interpolation of 67% of the blocks.

In the second pass, the search ellipsoid dimension was increased to 100 m by 100 m by 50 m with a reduction of the anisotropy. The search conditions were the same as the first pass interpolation. The second pass resulted in the interpolation of 30% of the blocks. The third interpolation pass estimated the remaining blocks (3% of total). The ellipsoid dimension for the last pass was increased to 200 m by 200 m by 100 m with the same search conditions as defined in the first pass. Please see Table 14-3 and Figure 14-7.

Table 14-4: Search Ellipsoid Parameters

Search Ellipse	Azimuth (°)	Dip (°)	Spin (°)	Major Axis (m)	Medium Axis (m)	Minor Axis (m)
Pass 1	270	0	40	50	50	25
Pass 2	270	0	40	100	100	50
Pass 3	270	0	40	200	200	100

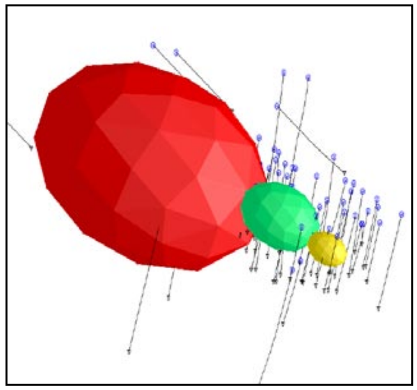


Figure 14-7: Search Ellipsoids used for the Interpolation Process

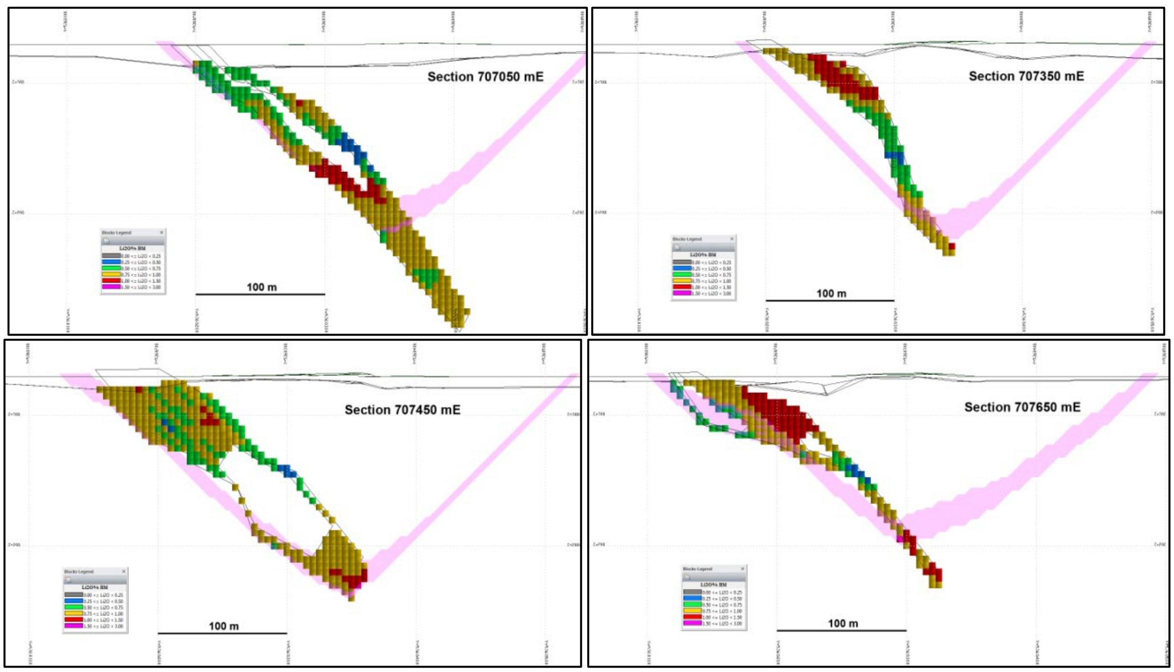


Figure 14-8: Sections Showing Li_2O Block Model Interpolation Results

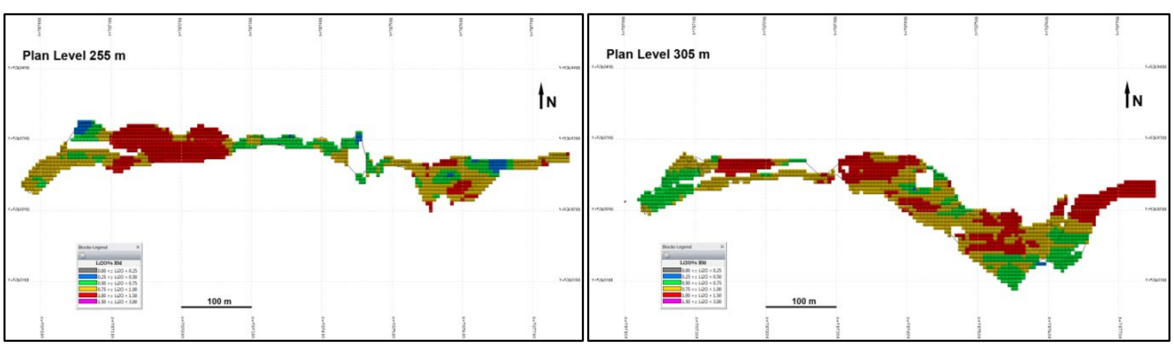


Figure 14-9: Level Plan Views Showing Li_2O Block Model Interpolation Results

14.5 Mineral Resource Classification

The mineral resource classification (MRC) is in accordance with the specifications of the NI 43-101 Policy, namely in Measured, Indicated, and Inferred resources. Classification was done according to the density of drilling and level of confidence of the database information.

Classification is done by a process of automatic classification that selects around each composite a minimum number of composites nearby, from a minimum number of holes inside a research ellipsoid of a given orientation and size. For the Measured category, a first phase of research was carried out with a 35 m by 35 m 35 m ellipsoid (direction, dip and thickness) with a minimum of 7 composites in at least 4 different holes. All blocks within the research ellipse are then categorized as measured to a maximum of 60 % of its maximum radius. The classification of indicated resources step uses the same parameters with a larger research ellipse (twice the size) and a fill to a maximum of 60% of the ellipse radius. The classification of inferred resources corresponds to the remaining part of the non-classified blocks during the first two stages of classification. The mineral resources disclosed were defined using an economic conceptual model.

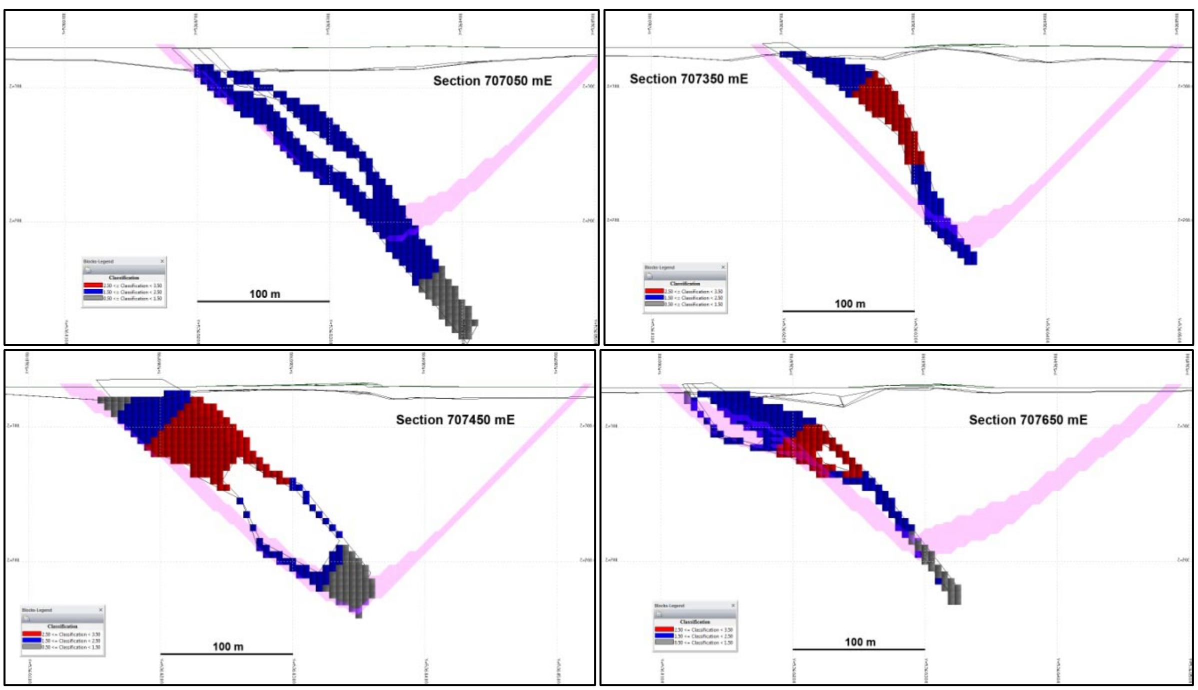


Figure 14-10: Sections Showing Li_2O Block Model Interpolation Results

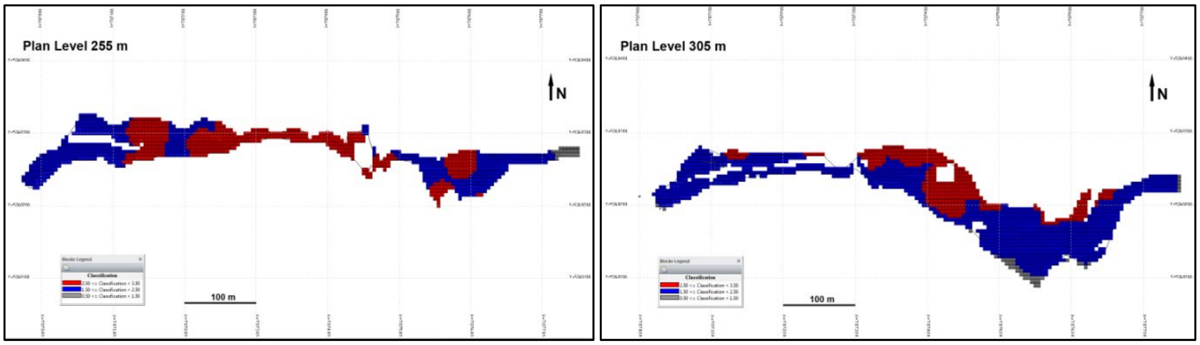


Figure 14-11: Level Plan Views Showing Li_2O Block Model Interpolation Results

14.6 Mineral Resource Estimation

The updated mineral resource for the Authier property, consisting entirely of the Authier mineralised Pegmatite deposit is as follows. The global mineral resources, without the open-pit limitation, using a base case cut-off grade of 0.5% Li_2O are totaling 2,244,000 tonnes grading 0.95% Li_2O in the measured category, 5,431,000 tonnes grading 0.97 % Li_2O in the indicated category with an additional 1,552,000 tonnes grading 0.96% Li_2O in the inferred category. The next Table 14-5 summarises the global mineral resources for Authier Property for a cut-off grade of 0.5 % Li_2O (base case), and also the resources using a 0.8% Li_2O cut-off grade.

Table 14-5: Global Mineral Resources of the Property

GLOBAL Mineral Resource Estimate - Authier Property				
Cut-off Grade Li_2O (%)	**Resources Categories**	**Tonnes***	**Li_2O Grade (%)**	**Li Metal** (tonne)**
0.5%	Measured (M)	2,244,000	0.95	8,500
	Indicated (I)	5,431,000	0.97	21,000
	TOTAL (M+I)	**7,675,000**	**0.96**	**29,500**
	Inferred	1,552,000	0.96	5,900

Effective date october 26th, 2012.
Mineral resources are not mineral reserves and do not have demonstrated economic viability.
Bulk density of 2.71 t/m3 used. * Rounded to the nearest thousand.

GLOBAL Mineral Resource Estimate - Authier Property				
Cut-off Grade Li_2O (%)	**Resources Categories**	**Tonnes***	**Li_2O Grade (%)**	**Li Metal** (tonne)**
0.8%	Measured (M)	1,621,000	1.05	6,800
	Indicated (I)	4,470,000	1.02	18,200
	TOTAL (M+I)	**6,091,000**	**1.03**	**25,000**
	Inferred	1,416,000	0.98	5,500

Effective date october 26th, 2012.
Mineral resources are not mineral reserves and do not have demonstrated economic viability.
Bulk density of 2.71 t/m3 used. * Rounded to the nearest thousand.

In-Pit Resources

The final in-pit resources are those shown at item 16.4 of the Section 16: Mining Methods.

15 Mineral Reserve Estimate

There are no mineral reserve estimates present in this report.

16 Mining Methods

16.1 Introduction

Taking into account the geometry and the depth of the mineralized zone, open-pit mining method has been considered in this study. The near surface resources will be mined by a major open pit, which will have a ten years life following a one year construction and pre production period. The mine plan is based on the Measured, Indicated and Inferred mineral resources contained in the pit design, which was based on a $525/tonne of spodumene concentrate (6% Li_2O) Lerchs-Grossmann optimized pit shell. Open-pit mining will be conducted by the operator and a specialized mining contractor will be assisting the operator during high stripping periods. Surface mining will follow the standard practice of an open-pit operation, with conventional drill and blast, load and haul cycle using a drill/truck/shovel mining fleet. The overburden and waste rock material will be hauled to the overburden and waste disposal areas near the pit. The run-of-mine mineralization will be drilled, blasted and loaded by hydraulic excavators and delivered by large mining trucks to the primary crusher or stockpiles near the crusher.

16.2 Overall Pit Slope Angle

Since the required geotechnical data are not available for determining the pit slope angle, SGS Geostat utilized a fixed 50° slope in rock and a 25° slope in overburden material. These values are based on the results of a study performed by Hoek and Bray (1974) which has the purpose of reasonably predicting the angle at which a slope is considered stable by analysing various mining projects.

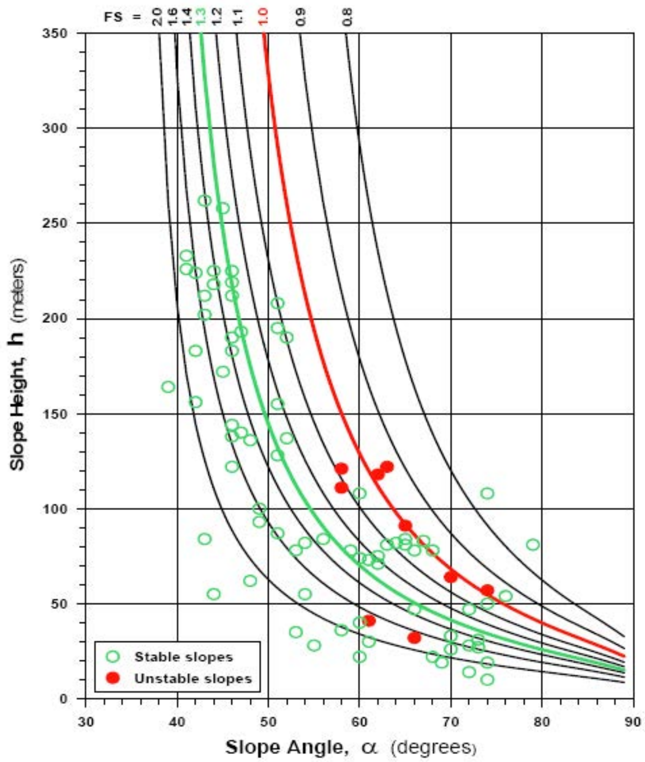


Figure 16-1: Cases of Rock Slope With Stable and Failed Conditions Distinguished[1]

16.3 Pit Optimization Procedure and Parameters

In order to develop an optimal engineered pit design for the Authier deposit, an optimized pit shell was first prepared using the Lerchs-Grossman 3D routine in Gems Whittle ("LG 3D"). The basic optimization principle of the algorithm operates on a net value calculation for each block in the model, in other words revenue from sales less total operating cost; mining, processing, and general and administration costs.

In accordance with the guidelines of the NI 43-101 and the Canadian Institute of Mine Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves, blocks classified in the Measured, Indicated and Inferred categories are allowed to drive the pit optimizer for a Preliminary Economic Assessment Study.

[1] Steffen, O. K. H., Contreras, L. F., Terbrugge, P. J., Venter, J., A Risk Evaluation Approach for Pit Slope Design, 2008

For the initial optimization, the required parameters were selected by SGS to evaluate the most economic open-pit profile. Although these parameters are not necessarily final, a reasonable degree of accuracy is required, since the analysis is an iterative process. The economic and operating parameters used in the initial optimization are given in the next Table.

Table 16-1: Open-pit Optimization Parameters

Slope Angle - Rock	*deg.*	50.00
Slope Angle - OVB	*deg.*	25.00
Waste Mining Cost	*Cdn$/tonne*	3.24
Ore Mining Cost	*Cdn$/tonne*	3.49
Mining Recovery	*%*	99.00
Mining Dilution	*%*	5.00
Processing cost	*Cdn$/t treated*	15.00
G&A cost	*Cdn$/t treated*	4.86
Transport cost	*Cdn$/t treated*	6.77
Ore Mining Increment	*Cdn$/t treated*	0.25
Total Ore Based Cost	***Cdn$/t treated***	**26.88**
Processing recovery	*%*	85.00
Spodumene concentrate grade	*%Li2O*	6.00
Concentrate purety	*%*	100.00
Concentration ratio		7.84
Selling Price	*US$/tonne*	525.00
Exchange Rate	*Cdn$:US$*	1:1
*In-situ Li2O value** *	*$/10kg*	74.38
Resulting cut-off grade	***%Li2O***	**0.38**

Note: The economic parameters used at the time of the pit optimization do not necessarily confirm those stated in the economic model.

The marginal cut-off grade or milling cut-off grade (CoG) is used to classify the material inside the pit limits as in-pit resource or waste. Since the material is located inside the pit, the marginal cut-off grade excludes the mining cost and corresponds to the grade required to cover the costs of processing, G&A, and other costs related to transport ore material. The marginal cut-off is calculated as follow:

$$\textbf{Resulting Marginal CoG} = \frac{\textbf{Total Ore Based Cost x } (1 + \%\textbf{Mining Dilution})}{\textbf{Resulting Selling Price x Processing Recovery}}$$

16.4 Pit Optimization Results

Using the above input parameters, a total of 25 pit shells were created for prices ranging from $205 to $525/tonne of spodumene concentrate. The following table shows the sensitivity of pit size to concentrate prices. Shell #25, which contains approximately 8.65 Mt of resource grading at 0.91 %Li_2O, represents the maximum pit size assuming an average long-term concentrate price of $525/tonne.

Table 16-2: Optimization Results

Shell #	Rev Factor	Resource tonnes	Waste tonnes	Total tonnes	Strip Ratio t:t	Grade %Li2O
1	0.39	9,860	9,980	19,840	1.0	1.16
2	0.40	11,972	12,078	24,050	1.0	1.15
3	0.45	125,711	233,597	359,308	1.9	1.15
4	0.48	704,261	914,680	1,618,941	1.3	1.02
5	0.50	1,250,064	1,613,718	2,863,782	1.3	0.99
6	0.53	1,589,518	1,843,925	3,433,443	1.2	0.96
7	0.55	2,611,048	5,000,223	7,611,271	1.9	0.97
8	0.58	2,910,712	5,529,362	8,440,074	1.9	0.96
9	0.60	3,441,725	7,069,794	10,511,519	2.1	0.95
10	0.63	3,655,468	7,311,052	10,966,520	2.0	0.94
11	0.65	3,981,541	8,355,281	12,336,822	2.1	0.93
12	0.68	4,187,889	8,942,207	13,130,096	2.1	0.93
13	0.70	4,348,813	9,418,854	13,767,667	2.2	0.92
14	0.73	4,515,723	10,071,732	14,587,455	2.2	0.92
15	0.75	5,533,380	16,856,157	22,389,537	3.1	0.92
16	0.78	5,710,854	17,676,638	23,387,492	3.1	0.92
17	0.80	5,911,569	19,185,618	25,097,187	3.3	0.92
18	0.83	6,279,897	21,594,594	27,874,491	3.4	0.91
19	0.85	7,044,373	27,763,133	34,807,506	3.9	0.91
20	0.88	7,130,997	28,538,256	35,669,253	4.0	0.91
21	0.90	7,623,276	33,299,937	40,923,213	4.4	0.91
22	0.93	7,845,470	35,404,487	43,249,957	4.5	0.91
23	0.95	8,106,399	38,185,865	46,292,264	4.7	0.91
24	0.98	8,301,479	40,187,248	48,488,727	4.8	0.91
25	**1.00**	**8,349,017**	**40,806,865**	**49,155,882**	**4.9**	**0.91**

In order to select an optimal pit shell for developing a life-of-mine scenario, discounted cash flow analyses were performed taking into account the sequence of mining for all the nested pit shells (25 shells created previously) using a fixed concentrate value of $525/tonne. Preliminary assumptions made in order to perform those analyses are:

- Capital expenditure: $45,000,000
- Mill throughput: 2,200 t per day, 350 days per year (770,000 t per year)
- Discount rate: 6.00 %

Note: The economic parameters used at the time of the pit optimization do not necessarily confirm those stated in the economic model.

Three different mining scenarios were used for the analyses: best case, specified case, and worst case. The best case (mining by layers) and worst case (mining bench per bench) were combined to produce a realistic mining scenario (specified case). This realistic case assumes three mining phases (using shells 4, 6 and 9 for pushback's). The results of the optimization are summarized below:

Table 16-3: Optimization Results

Shell #	NPV-best $ disc	NPV-spec $ disc	NPV-worst $ disc	Resource tonnes	Waste tonnes	Grade %Li2O	Strip. Ratio t:t	Length years	IRR-best %	IRR-spec %	IRR-worst %
1	- 44,482,784	- 44,482,784	- 44,482,784	9,867	9,973	1.15	1.0	0.0	-	-	-
2	- 44,375,087	- 44,375,087	- 44,375,087	11,981	12,069	1.15	1.0	0.0	-	-	-
3	- 38,815,684	- 38,815,684	- 38,815,684	126,156	233,154	1.15	1.8	0.2	-	-	-
4	- 16,838,444	- 16,838,444	- 16,838,444	717,467	901,490	1.02	1.3	0.9	-	-	-
5	1,547,961	1,474,769	1,424,336	1,280,939	1,582,862	0.98	1.2	1.7	8.1	7.8	7.7
6	10,813,280	10,631,214	10,484,666	1,637,206	1,796,257	0.95	1.1	2.1	22.1	21.0	20.3
7	39,297,232	38,711,160	38,015,031	2,675,701	4,935,590	0.97	1.8	3.5	43.9	40.8	37.2
8	45,472,666	44,709,259	43,849,937	2,963,604	5,476,493	0.96	1.8	3.8	47.0	43.1	38.9
9	56,555,417	55,242,698	54,076,586	3,502,409	7,009,134	0.95	2.0	4.5	50.3	44.6	39.5
10	59,481,309	57,932,169	56,562,219	3,687,062	7,279,481	0.94	2.0	4.8	51.2	45.1	39.4
11	64,844,017	62,939,791	61,249,680	4,011,966	8,324,885	0.93	2.1	5.2	52.3	45.6	39.1
12	67,794,933	65,633,162	63,708,810	4,213,885	8,916,240	0.93	2.1	5.5	52.8	45.6	38.6
13	69,734,531	67,297,478	65,139,922	4,369,994	9,397,702	0.92	2.2	5.7	53.2	45.5	38.0
14	71,640,624	68,900,269	66,510,729	4,531,389	10,056,098	0.92	2.2	5.9	53.5	45.4	37.6
15	83,116,933	78,556,877	74,793,865	5,552,969	16,836,608	0.92	3.0	7.2	54.2	43.4	33.6
16	84,655,206	79,630,057	75,583,017	5,728,460	17,659,070	0.92	3.1	7.4	54.3	42.9	32.7
17	86,466,872	80,830,240	76,522,868	5,928,970	19,168,257	0.92	3.2	7.7	54.4	42.3	31.9
18	89,053,396	82,106,786	77,297,078	6,300,390	21,574,143	0.91	3.4	8.2	54.5	41.0	30.3
19	93,832,270	83,729,934	77,929,352	7,066,134	27,741,421	0.91	3.9	9.2	54.7	38.0	27.0
20	94,283,111	83,815,245	77,914,486	7,152,117	28,517,186	0.91	4.0	9.3	54.7	37.7	26.6
21	96,343,432	83,374,068	76,856,736	7,639,121	33,284,143	0.91	4.4	9.9	54.8	35.5	24.6
22	97,039,709	82,920,503	76,129,055	7,857,955	35,392,054	0.91	4.5	10.2	54.8	34.5	23.7
23	97,554,306	81,907,969	74,772,783	8,114,848	38,177,468	0.91	4.7	10.5	54.8	33.1	22.5
24	97,762,356	80,878,241	73,494,340	8,308,663	40,180,117	0.91	4.8	10.8	54.8	32.2	21.7
25	97,766,520	80,484,479	73,010,491	8,355,178	40,800,757	0.91	4.9	10.9	54.8	31.9	21.5

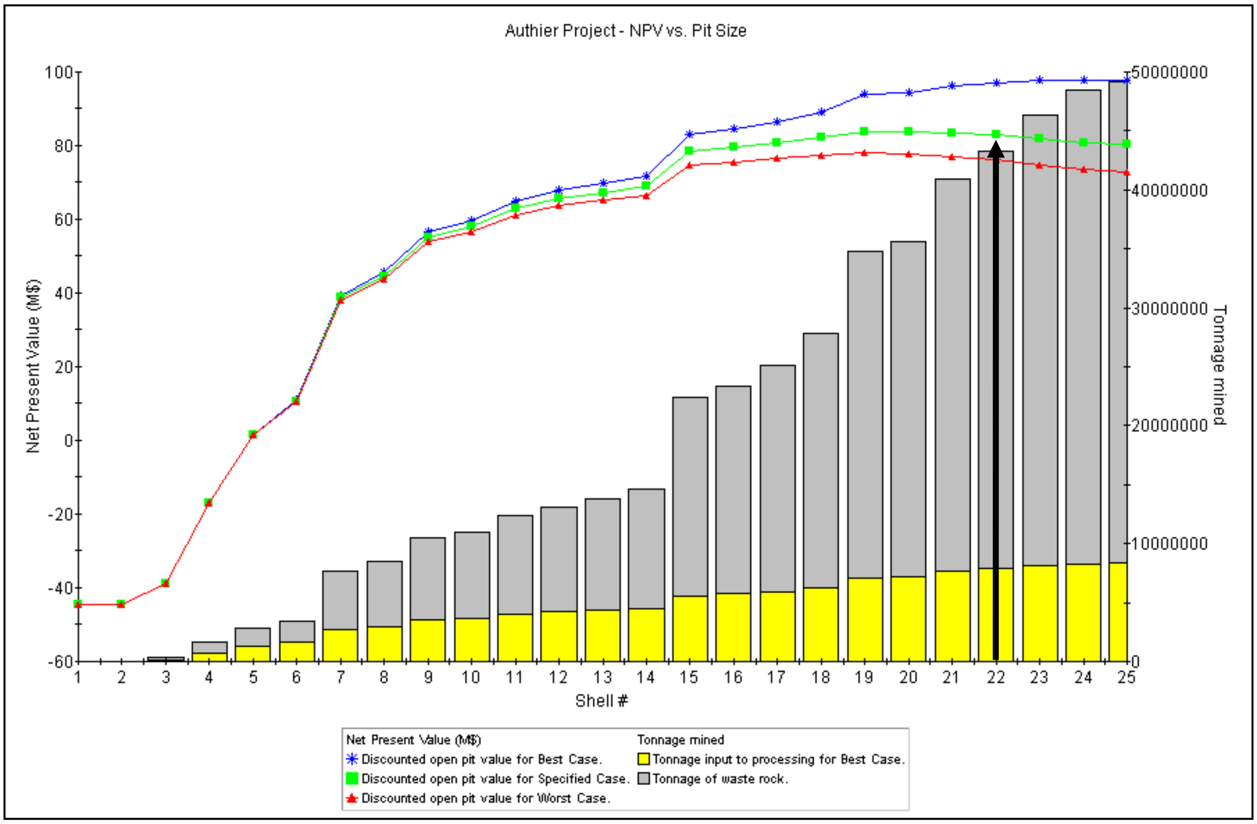


Figure 16-2: Pit Tonnage versus NPV ($) showing Optimal Shell

Shell #25 is the optimal shell as defined previously. All other shells (1 to 24) are smaller in size and are used to evaluate the net present value of the project if mining would stop at this specific shell rather than at the optimal one. SGS and Glen Eagle Resources strategy was to select a shell that provide an interesting NPV but which allows at least 10 years of mining operation. Based on this constraint, the shell #22 was selected.

Considering that this study is a PEA, no grade optimization scenarios were made in this report. Future studies should analyse several scenarios to quantify the variation of the Project's NPV when raising the mining CoG; and thus indirectly raising the stripping ratio and the daily mining rate.

Therefore, the selected shell (#22) has an in-situ content of 7.86 Mt of resource, at an average grade of 0.91 % Li_2O (mill feed grade) and a strip ratio of approximately 4.5:1. The following figure show pit #22.

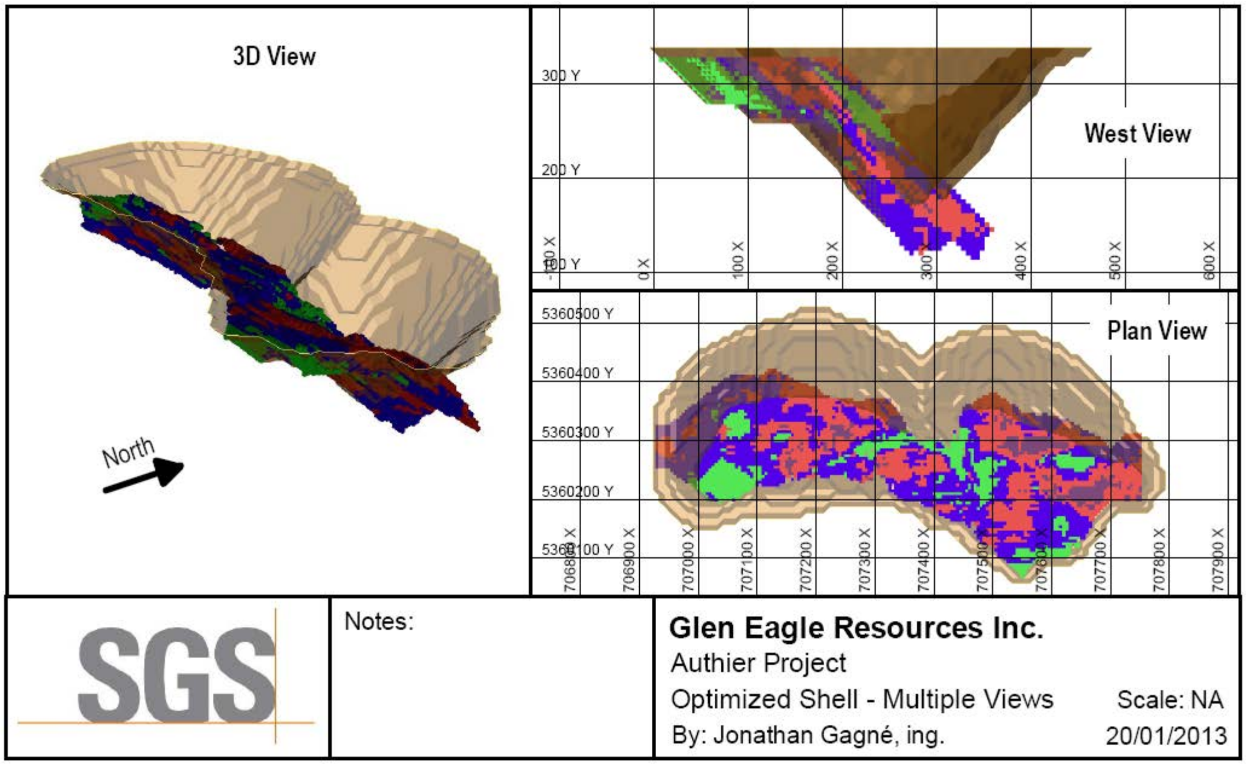


Figure 16-3: Multiple Views of the Optimized Shell

16.5 Ultimate Pit

16.5.1 Pit Design Parameters

Using the base case shell (#22) as reference, an open-pit including a ramp and safety berms was designed to develop a more realistic mining scenario. The new designed pit will account for the additional waste material coming from the addition of a ramp to the base case shell. The design parameters used are defined as:

Overall slope angle:	50° in rock and 25° in overburden
Face angle:	85°
Bench height:	5 m
Safety berm:	17 m width (1 safety berm at each 20 m vertically)
Ramp grade:	12.0 % (single lane) and 10.0 % (double lanes)
Ramp width:	15.0 m (single lane) and 19.80 m (double lanes)

16.5.2 Ultimate Pit Design

The next Figures are showing views of the designed open-pit with his dimensions.

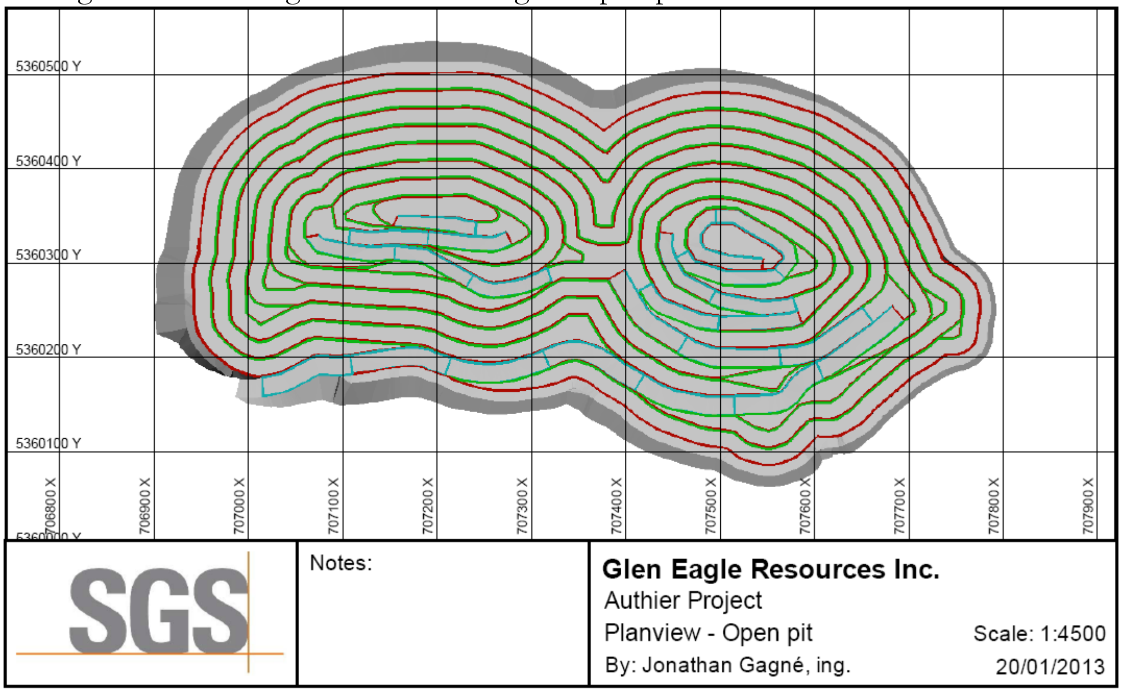


Figure 16-4: Plan View of Open-Pit

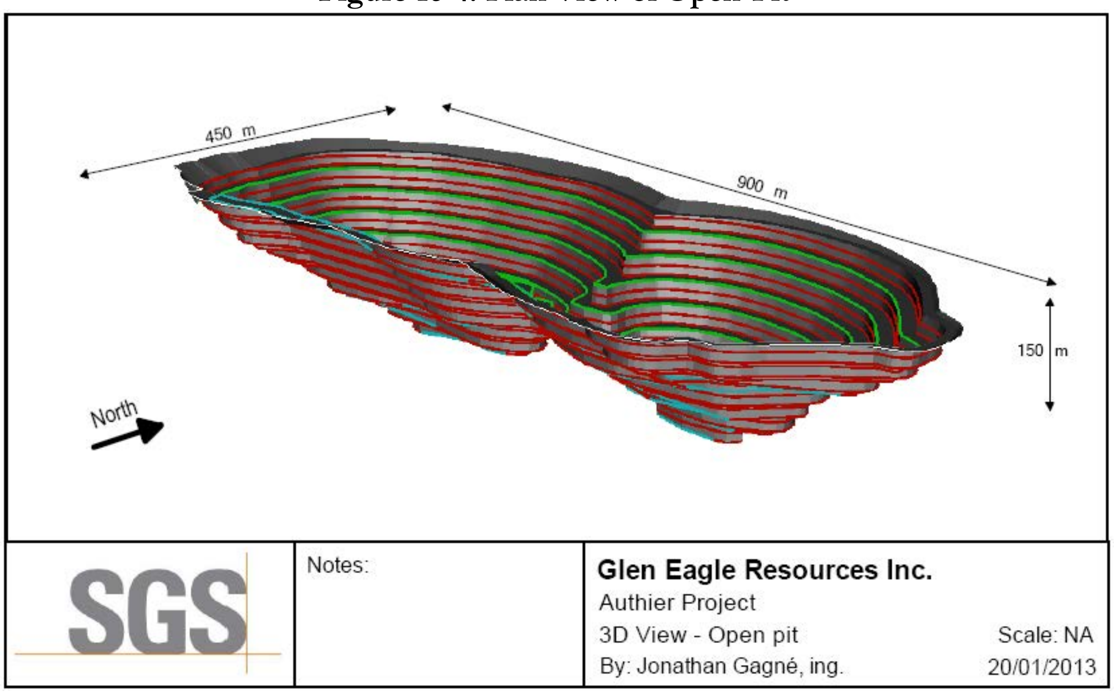


Figure 16-5: Open-Pit 3D View

16.5.3 Resource Contained within Pit Design

The ultimate pit design results are presented in next Table.

Table 16-4: Pit Design Results

Material type	Density t/m3	Tonnage tonnes	Grade % Li2O	Concentrate tonnes	Strip. Ratio t:t
Measured Resource	2.71	2,360,000	0.90	301,000	5.7
Indicated Resource	2.71	5,120,000	0.92	667,000	
Measured + Indicated Resource	2.71	7,480,000	0.91	968,000	
Inferred Resource	2.71	290,000	0.87	32,000	
Waste - Rock	2.71	39,280,000			
Waste - OVB	2.00	4,970,000			
Total		**52,020,000**			

*Include a 5.0 % mining dilution and a 99.0 % mining recovery

16.6 Mine Development and Production Schedule

The mine development used a number of push-backs, or phases, designed to meet the following objectives:

- Enable the mining of high grade mineralization as early as possible.
- Effectively reduce stripping ratio in the initial mining stage.
- Balance the stripping ratio over the period of the mine life.
- Maintain a minimum mining width between two working phases.

16.6.1 Pushback Width

In order to have a safe operation, a minimum mining width has to be respected when introducing a pushback into an operating pit. An appropriate mining width was determined based on:

- Komatsu PC-1250 hydraulic shovel and HD-325 mining truck;
- 20 m allowance for equipment.

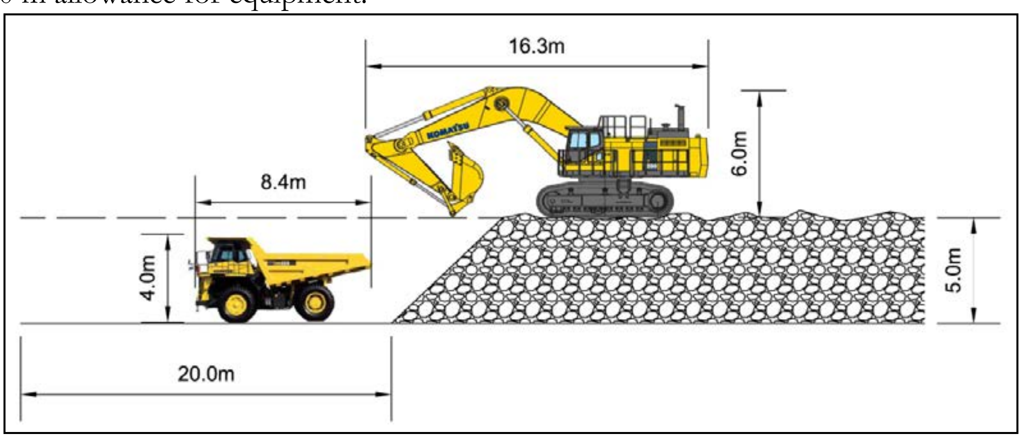


Figure 16-6: Minimum Push-back Width

16.6.2 Mine Development

Four minable phases are proposed to develop the ultimate pit. Each phase or pushback is designed with at least a minimum mining width of about 20 m to accommodate the mining equipment that will operate on each working bench. Figure 16-7 and Figure 16-8 illustrate these pushbacks.

- Phase/Pushback #1

At the beginning of the Project, the mining activities will be concentrated around phase #1 since the shell defined by this phase gives the higher grade achievable near surface and a low waste-to-ore stripping ratio. Prioritizing the mining in this section of the deposit will maximize revenue at the beginning of the Project, thus maximizing the net present value (NPV).[2]

- Phase/Pushback #2

Phase #2 generally expands of Phase #1. A constant difference of 40 metres has been kept during development of the mining plan (LoM) to limit the number of benches mined simultaneously. This constraint has also the effect to limit the variation of stripping ratios from years to years.[3]

- Phase/Pushback #3

Phase #3 generally expands of Phase #1 and 2. A constant difference of 40 metres has been kept during development of the mining plan (LoM) to limit the number of benches mined simultaneously. This constraint has also the effect to limit the variation of stripping ratios from years to years.[4]

- Phase/Pushback #4 (Optimal Pit Design)

Phase #4 includes the mining of the rest of the mineralized material.

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[2] Considering that this study is at a PEA level and has for main purpose to evaluate the economic potential of the deposit, no final pit design was completed for phase #1

[3] Considering that this study is at a PEA level and has for main purpose to evaluate the economic potential of the deposit, no final pit design was completed for phase #2

[4] Considering that this study is at a PEA level and has for main purpose to evaluate the economic potential of the deposit, no final pit design was completed for phase #3

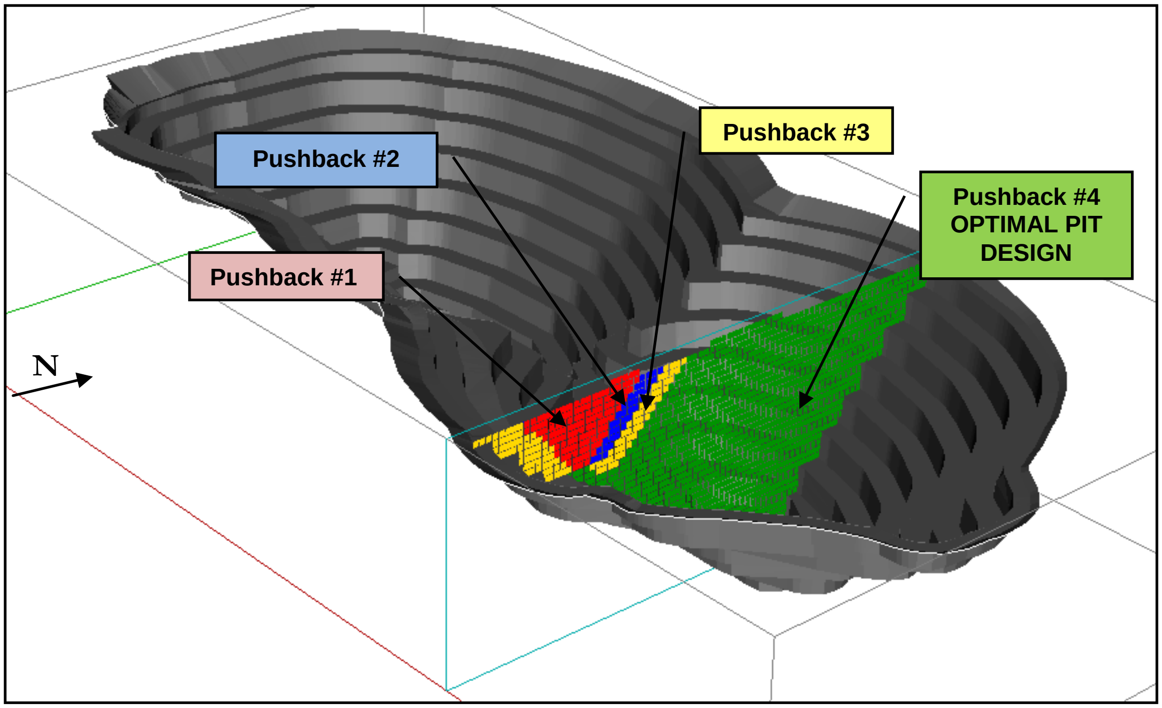

Figure 16-7: Typical Section View of Pushback's 1, 2, 3 and 4 (Optimal Pit Design)

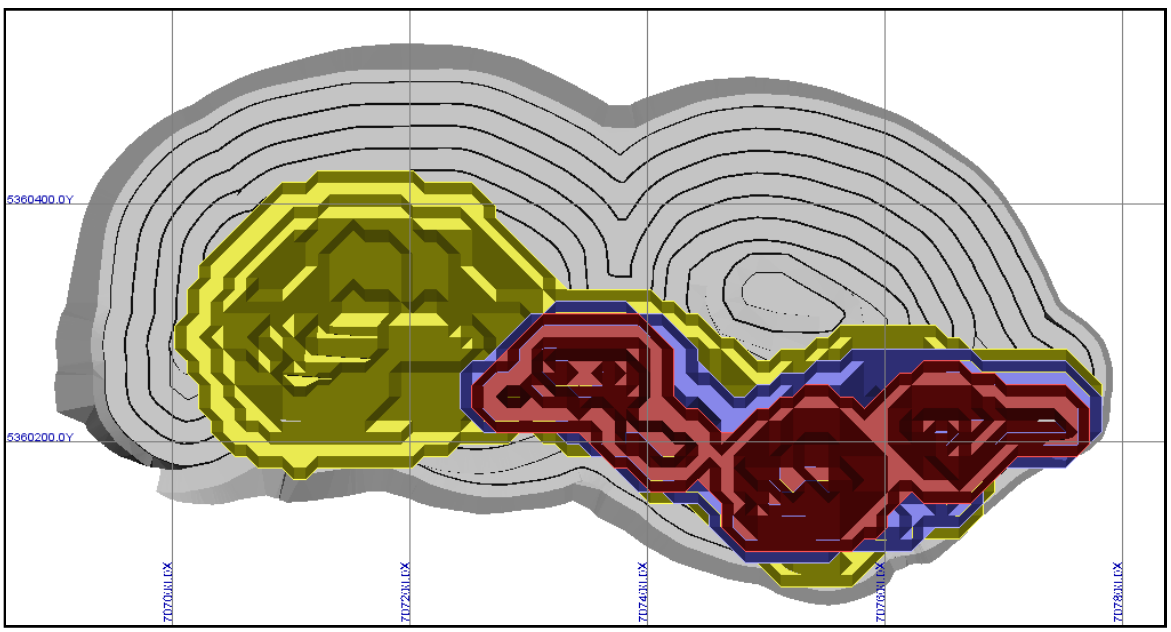

Figure 16-8: Plan View of the Optimal Open-Pit

16.6.3 Production Schedule

A mine production schedule was prepared for the development and the operation of the Project. The mining production schedule for the open pit is based on a pre-stripping period of approximately 12 months. The results of the developed schedule are summarized in Table 16-5 and presented by Figure 16-9 and Figure 16-10. Key findings include:

- potential Project life of 10 years, with mill operating at full capacity of 2,200 tpd;
- potential mill feed over the Project life of 7,77 Mt at 0.91 %Li_2O;
- potential spodumene concentrates production of 1.0 Mt;
- an important spike in stripping ratio during Year 4 and 5.

Table 16-5: Mine Development and Production Data

Year	Resource	Waste - Rock	Waste - OVB	Total	Strip Ratio	Input Grade	Concentrate
	tonnes	tonnes	tonnes	tonnes	t:t	%Li2O	tonnes
0	-	200,000	200,000	400,000	-	-	-
1	770,000	526,393	405,911	1,702,304	1.7	0.99	108,446
2	770,000	827,841	1,089,556	2,687,397	2.5	0.90	98,847
3	770,000	2,351,889	1,292,710	4,414,599	4.7	0.85	93,322
4	770,000	9,009,301	1,982,569	11,761,870	14.3	0.84	92,386
5	770,000	8,457,829	-	9,227,829	11.0	0.93	102,084
6	770,000	5,504,066	-	6,274,066	7.2	0.84	92,540
7	770,000	7,162,696	-	7,932,696	9.3	0.81	89,557
8	770,000	3,485,843	-	4,255,843	4.5	1.01	111,616
9	770,000	1,312,947	-	2,082,947	1.7	0.99	108,919
10	770,000	437,897	-	1,207,897	0.5	0.99	108,600
Total	**7,700,000**	**39,276,702**	**4,970,746**	**51,947,448**	**5.7**	**0.91**	**1,006,317**

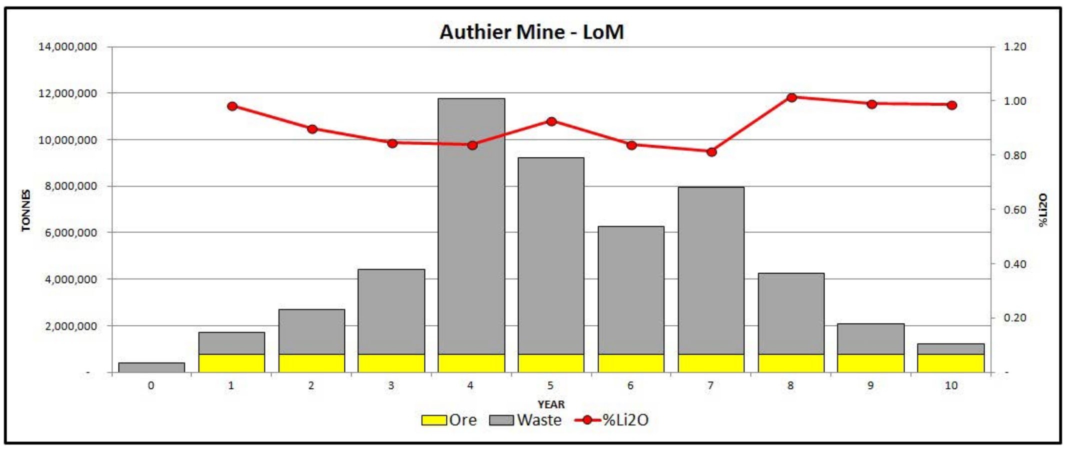

Figure 16-9: Mill Feed Grade (Li_2O), Resources and Waste Production

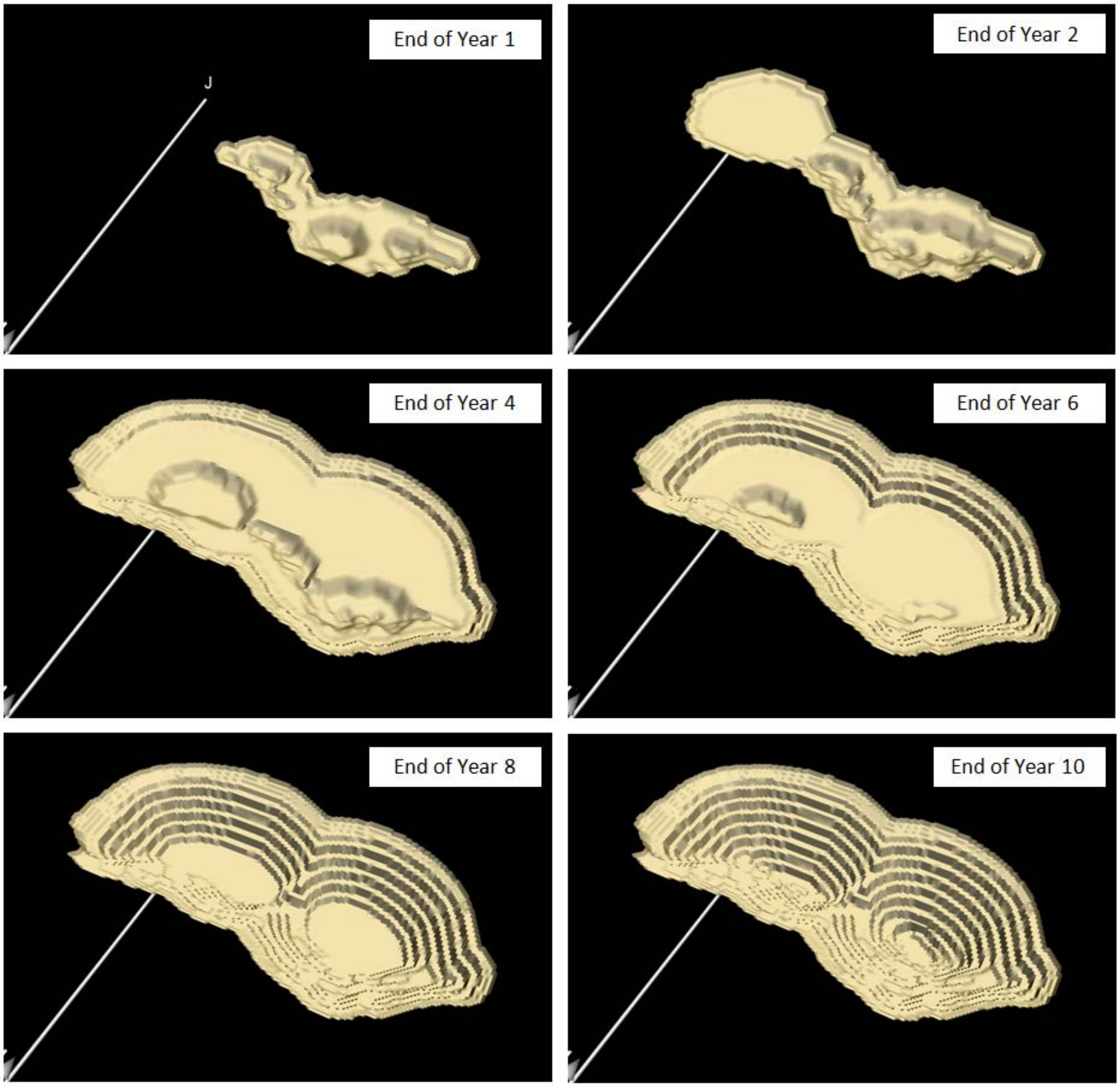

Figure 16-10: 3D Views of Years' End Pit

16.7 Pit Dewatering

The progressive deepening of the open pit will result in increasing water infiltration from precipitation (rain and snow) and groundwater inflow. The maximum depth of the pit will be reached at the end of the mine life and will be around 150 m under topography. As the pit deepens and increases in footprint, it will be necessary to control water inflow through the construction of an in-pit dewatering systems such as drainage ditches, sumps, water pipes and pumps.

16.8 Mine Operations

The proposed schedule for the open-pit operations is based on two 10-hour shifts per day, 7 days a week and 350 days per year. The mining operations at Glen Eagle will use conventional mining methods, including drilling and blasting sequences, loading with hydraulic excavators and hauling with off-road mining trucks. The selection of two 10-hour shifts is based mainly on the fact that this schedule allows spare time for equipment check-ups and small maintenance.

16.8.1 Drilling

Blast holes drilling will be performed by diesel drilling units. All holes will be drilled with a diameter of 5.0 inch (12.7 cm), on a 4.0 m x 4.0 m pattern. Due to the overall average low dip of the mineralized zones, between 35° to 50°, the working bench height is planned at 5.0 m. The next Table is a summary of the parameters selected for the blast hole drilling.

Table 16-6: Drilling Parameters & Assumptions

Parameters	Units	Ore &Waste
Holes Depth	m	6.0
Penetration Rate	m/min	0.55
Grade Control Sampling Time	min	2.0
Move and Align Time	min	2.0
Total Time per Hole	min	14.9
Holes per Hour	holes	4.0
Average Drilling Rate	m/h	24.1

The total number of drills required for the maximum production is estimated at two units, as shown in the following Table, for a daily tonnage of 27,950 tonnes per day. As the owner daily maximum tonnage production is 12,500 tonnes, one drill is sufficient.

Table 16-7: Drilling Data

DRILLING		
Maximum Daily Tonnage Year-4 (Ore + Waste)	**27,950**	**Tonnes**
Burden	4.0	m
Spacing	4.0	m
Depth (bench height)	5.0	m
Sub-drilling	1.0	m
Rock density (average)	2.7	t/m3
Tonnes per holes	216.8	t/hole
Tonnes per drilled meter	36.1	t/m
Required tonnes per day	27,950	tpd
Required meters per day	773.5	mpd
Required holes per day	128.9	holes
Average drilling rate	24.1	m/h
Required drilling time per day	32.1	hours
Drills required, based on 17.0 h/day per drill	1.89, rounded to 2	

16.8.2 Blasting

This operation will be under a subcontract with an explosive contractor (supplier) that will take control of all blasting operations, explosive and detonators supply and loading and connecting the blasts. The project will benefit from being close to major suppliers with blasting supplies and experience. The regular powder factor is estimated at 0.30 kg/tonne, both in ore and waste, using emulsion type explosive having a density of 1.28 g/cm3. The following Table summarizes the parameters retain for the production estimations.

Table 16-8: Blasting Parameters

BLASTING		
Maximum Daily Tonnage (ore + waste)	**27,950**	**tonnes**
Tonnes per hole	216.8	tonnes
Holes per day	129.0	unit
Hole depth	6.0	m
Collar	2.00	m
Active column	4.00	m
Hole diameter	5.0	in
Hole diameter	12.7	cm
Emulsion density	1.28	g/cc
Qty emulsion per hole	65	kg
Qty emulsion per day	8,367	kg
Powder factor	0.30	kg/tonne

Emulsion-type explosive was selected due to his good water resistance and to better overall performance than other explosives, like the Amex products. Non electric detonators, like the Nonel type are recommended. Blasting results will determine if electronic detonators should be preferred.

16.8.3 Fleet Requirements

The mining fleet selection was done after evaluating two scenarios. The first scenario was to have a mining fleet large enough for the whole production. The second scenario was to have a smaller mining fleet for the mine of life, and having a mining contractor for the four years during which the waste removal is much higher than the average. For reference, please see the table of item 16.6.3 illustrating the surge of waste removal during the years 4 to 8. These two scenarios were cost estimated for comparison using heavy equipment benchmarks and in house database. The recommendation is to retain the second scenario, i.e. the one calling for the hiring of a mining contractor during the years 4 to 8.

The loading and trucking selection was done by giving priority to the fuel consumption which represents the most important consumable item. The final selection of the main mining fleet consists of hydraulic excavator of 6.0 m3 bucket capacity (Komatsu PC 1250LC, or equivalent) and off-road trucks of 37 tonnes loading capacity (Komatsu HD-325, or equivalent). The permanent fleet include one hydraulic excavator and 5 trucks having an annual capacity of 4,375,000 tpy. The maximum tonnage to be mined by contractors is amounting to 15,715,000 tonnes, or an average of 3,930,000 tpy, during years 4, 5, 6 and 7. The list of the mining fleet and the support equipment is given in the following Table.

Table 16-9: Mining Fleet and Equipments

Mining Equipment	Model (refrerences)	Size, or capacity	Quantity
Hydraulic Excavator	Komatsu PC1250LC-8	6.0 m^3	1
Haul Trucks	Komatsu HD325-7	37.0 t	5
Wheel Loader - Seconda	Komatsu WA600-6	6.0 m^3	1
Production Drill	ACopco AC-ROC D55	152 mm	1
Track Dozer	Komatsu D275AX-5	225 kW	1
Grader	Komatsu GD655-5	150 kW	1
Water Truck	-	18,000 l	1
Backhoe - Secondary	KomatsuPC650LC-8	4.0 m^3	1

16.8.4 Manpower Requirements

The manpower required for the operation consists of the staff and hourly personnel. The hourly personnel are based on two 10-hour shifts, 7 days per week. This schedule means that 3 operators are required to operate one mining fleet equipments, scheduled at 20 hours per day. Every man works two weeks in row and is off the third week, this represents 34 weeks of work or 2,380 hours per year, for 3 men this amounts to 7,140 hours per year, and this equates a mining equipment scheduled on two 10-hour shift during 350 days for a total of 7,000 hours, or 7,300 hours if scheduled on 365 days per year.

The estimation of the required manpower for the regular production, from year-2 to the end of the project is shown below. This estimation does not take in account the manpower that will be needed by the mining contractor, neither the manpower during the construction period.

Table 16-10: Owner Manpower

Personnel	Staff	Hourly
Mine Management	33	
Mine Operation		42
Mine Maintenance		18
Concentrator Management	6	
Concentrator Operation		30
Concentrator Maintenance		5
Sub-total	39	95
Total	**134**	

17 Recovery Methods

Introduction

This item has been prepared by Bumigeme Inc from Montreal as mandated by Glen Eagle to realize the development of a conventional lithium flotation process plant to process 803,000 tonnes per year of mineral, producing around 103,000 tonnes of spodumene concentrate at a grade of 6.0% Li_2O. A process flow diagram is done based on laboratory metallurgical tests prepared by SGS Minerals Services at Lakefield. Capital and operating costs for 2,200 metric tons per day plant are estimated.

17.1 Mineral Processing

This section describes the designed process based on the results of metallurgical tests conducted by SGS Lakefield. The mineral processing plant consists of a crushing and grinding circuit followed by a desliming circuit; a flotation circuit followed by a concentrate dewatering and drying circuit. A block flow diagram is provided (Figure 17-1), and the material balance is enclosed in Appendix-2.

17.1.1 Process Description and Design Criteria

The plant is designed to process 803,000 tonnes of ore (run of mine) per year and to produce approximately 103,000 tonnes per year of spodumene concentrate at a grade of 6% Li_2O.

Table 17-1: Process Design Criteria

Designation	Average value	Design Value	Unit
Scheduled operating days per year		365	d
Equipment availability - Crushing - Others		 67 92	 % %
Plant capacity	2,200	2,400	tpd
Plant feed analysis			
Li_2O	0.91		%
Moisture (assumed)	3		%
Plant recovery			
Flotation	85		%
Annual production			
Spodumene concentrate	103,000		tpy
Concentrate grade	6.00		% Li_2O

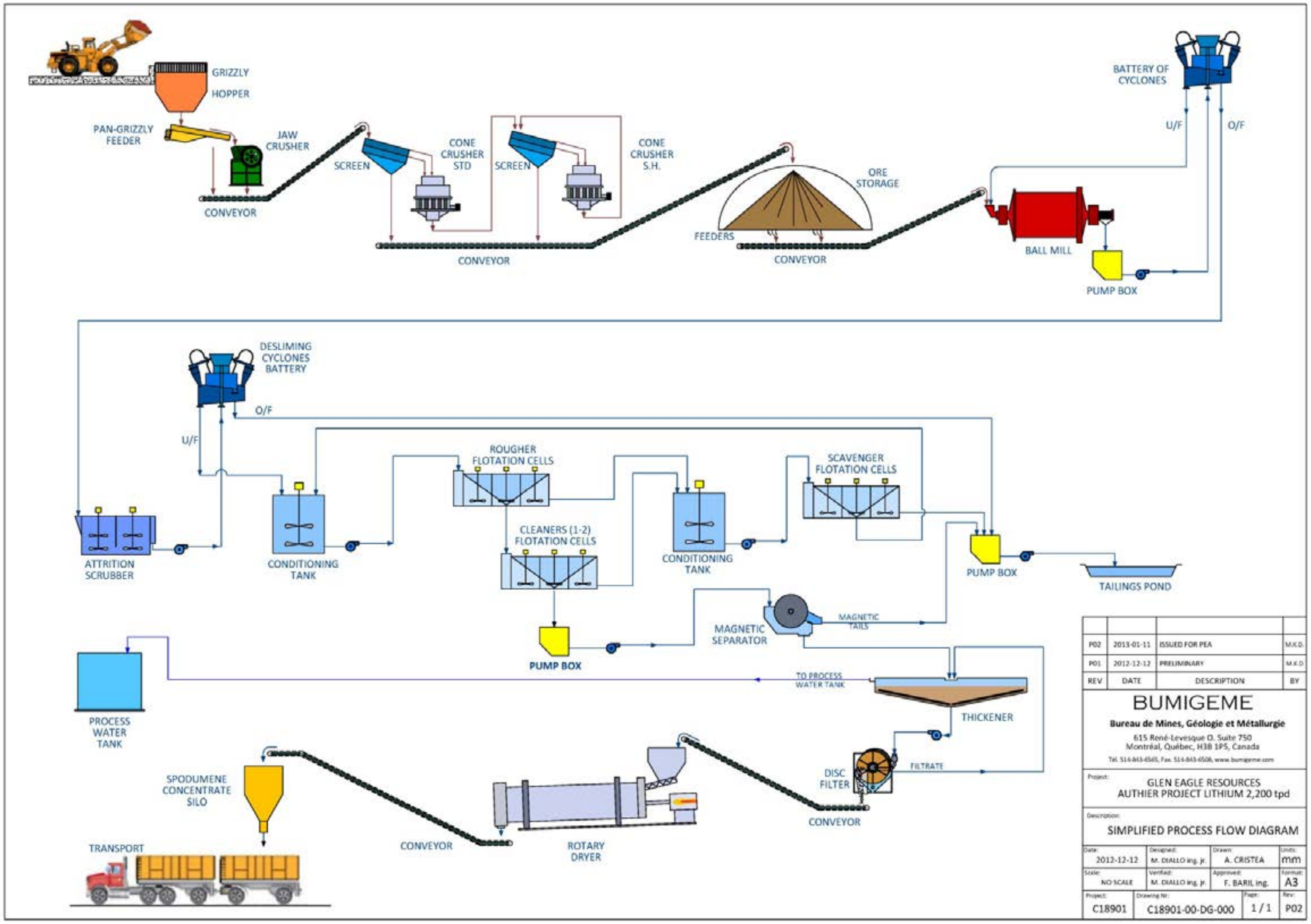

Figure 17-1: Process Flow Diagram

17.1.2 Crushing

The ore will be first scalped by 600mm x 600mm grizzly and feed to a 50 mt hopper. A Pan-grizzly feeder is then used to feed the plus 200mm to a 1100mm x 860mm Jaw crusher (160 tph), which reduces it to minus 200mm. The Pan-grizzly feeder undersize and jaw product are combined and conveyed to a first double deck screen. The oversize of the two decks feed a ROC1600 STD cone crusher (secondary crushing stage). The standard cone crusher product is then passed through a second double deck screen. The oversize of the second double deck screen is conveyed to a ROC1600 SH cone crusher (tertiary crushing stage) before being recycled to the second double deck screen. The second double deck screen undersize (< 6mm) is conveyed to a 10500 mt ore storage consisting of a tunnelled dome with two apron feeders.

17.1.3 Grinding

The grinding circuit consists of a 3.8m x 7.5m ball mill of 100 TPH of capacity in closed circuit with a battery of 3 x 20” gMAX20-3140 Krebs type cyclones. The circulating load is 300%. The grinding is designed to achieve the particle size for the ongoing flotation process (P80=150μm)

17.1.4 Scrubbing and Desliming

Attrition scrubbing and desliming are required to eliminate the fine particles (-35 microns) that otherwise affect the flotation recovery and increase reagent consumption. The circuit consists of a 2 x AS108VBH-2 Westpro type attrition scrubber and a battery of 3 x 15” gMAX15-3117 Krebs type desliming cyclones. The cyclones’ undersize is fed to the flotation circuit, while the overflow is pumped to the tailings pond.

17.1.5 Flotation and Magnetic Separation

Based on the best results of the laboratory tests, the flotation circuit consists of a bank of 5 x FL100 Westpro type rougher flotation cells, a bank of 2 x FL100 Westpro type scavenger cells and two stages of 3 x FL100 Westpro type cleaner cells. The scavenger concentrate is recycled to a 40 m3 rougher conditioning tank. The two cleaner’s tails are recycled to a 15 m3 scavenger conditioning tank. The concentrate obtained after the second cleaning stage is passed through a High Intensity Magnetic Separator. A magnetic intensity of about 15000 gauss will be sufficient.

17.1.6 Concentrate Dewatering and Drying

The final concentrate is first pumped to a 6m diameter thickener and flocculents are added. The underflow of the thickener at around 45 % solids is then pumped to a 2.7m x 6 rotary disks filter to bring the moisture in the concentrate to around 15%. The dewatered concentrate is fed to a 1.8m x 10.7m rotary dryer to reduce the water to less than 5%.

17.1.7 Concentrate Storage and Load-Out

The dried spodumene concentrate is directed to a 1,000mt silo incorporated with a truck load out facility. In case of long time truck unavailability, there is a possibility to bypass the concentrate silo to a stock pile.

17.1.8 Reagents

Reagents consist of the lignosulfonate Marasperse D618 used to reduce viscosity and enhance desliming stage before flotation; the fatty acid (FA-2) as the best spodumene collector found during laboratory testing; caustic soda and soda ash for the pH adjustment purpose. A flocculent is used during the thickening stage. A small amount of sulphuric acid can be used to break the froth and improve magnetic separation efficiency.

17.1.9 Infrastructures and Services

Services will include process water; clean water and air facilities; the laboratory; reagents and polymer handling and preparation; fuel storage and distribution.

17.1.10 Tailings

- **Tailings Disposal Infrastructures**

The tailings disposal infrastructures will consist of one (1) tailings disposal facility and a network of pumps and connecting pipes. Tailings from the concentrator will be routed to the tailings disposal facility via pipelines. The tailings disposal facility will cover a surface area of approximately 0.6 km^2, sufficient to hold the 7.0 Mm^3 of tailings that will be generated by the concentrator.

Details concerning the assessment of the seismic risk, geotechnical study, tailings properties and geochemistry and protection of underground water related to the design of the tailings disposal infrastructure should be covered in more details, in the next studies (pre-feasibility or feasibility studies).

- **Tailings Management**

The volume of tailings generated by the concentrator is estimated at 7.0 Mm^3. The capital costs estimate of the Project includes only Year 1 and Year 2 of the berm construction. Consequently, the berm raise required to hold all the tailings generated by the Project will be managed as an operating cost over the mine life.

For this PEA, it is assumed that the ground underlying the tailings disposal infrastructures is competent. Consequently, the tailings disposal option retained for the Authier Project did not include any geomembrane. It was assumed that the tailings will be deposited directly on the bedrock.

17.2 Capital Cost Estimate

A scoping level capital cost estimate of the process plant (accuracy of +/- 30%) is summarized in Table 2 below. The costs are based on equipments obtained from equipment suppliers. The main equipments list is enclosed in Appendix 2.

Table 17-2: Capital Cost Estimate

Delivered equipment	**$**
Reception, crushing and storage	3 609 228
Grinding and desliming	2 089 500
Spodumene flotation & Magnetic separation	1 208 025
Decantation, filtration, and storage	1 183 350
Total 1	**8 090 103**
Services	
Laboratory	600 000
Reagents and polymer	300 000
Fuel and air	120 000
Others	200 000
Total 2	**1 241 000**
Installation	3 133 961
Instrumentation & Control	2 089 307
Piping	2 491 097
Electricity	2 410 739
Buildings	3 021 331
Yard improvement	436 266
Total 3	**13 582 701**
Total direct cost	**22 913 804**
Indirect Capital Cost	
EPCM	2 749 657
Contingency	2 690 308
Total indirect cost	**5 439 965**
Capital investment costs	**28 353 769**

This fixed capital investment cost does not include the electrical substation. The tailings pond investment needed for the first operating years is estimated at $2,000,000 and the working capital needed for plant start-up (first three operating months) is estimated at $3,272,000.

17.3 Operating Cost Estimate

The plant operating costs are summarized in the next Table 17-3.

Table 17-3: Plant Operating Costs (Opex)

			Annual Cost	Unit Cost
N°	**Reagents and consumables**		**$ /year**	**$/t**
1	Caustic soda (NaOH)		98 859	0.12
1	Soda Ash		20 446	0.03
1	Fatty Acid		1 096 095	1.37
1	F-100 (D-618)		1 191 371	1.48
1	Floculant		17 503	0.02
1	Liners consumption (set)		35 000	0.04
1	Balls consumption		515 392	0.64
Total 1			**2 974 667**	**3.70**
	Operating labor			
1	Superintendant	125 000	125 000	
4	Shift Bosses	90 000	360 000	
4	Crushing operators	72 000	288 000	
4	Grinding and desliming operators	72 000	288 000	
4	Flotation, Mag-separation & thickening operators	72 000	288 000	
4	Filtration and drying operators	72 000	288 000	
4	Truck loading and other tasks laborers	65 000	260 000	
	Laboratory			
1	Chief chemist	89 000	89 000	
3	Analysis technicians	72 000	216 000	
4	Samplers	65 000	260 000	
4	Mechanics	72 000	288 000	
1	Electrician	72 000	72 000	
3	Drivers	55 000	165 000	
Total 2			**2 987 000**	**3.72**
	Utilities			
1	Electricity		1 228 911	1.53
1	Combustible (Fuel)		1 186 531	1.48
1	Maintenance and repairs (% FCI)		1 984 764	2.47
1	Operating supplies (% Main&Rep)		297 715	0.37
1	Laboratory analysis and charges		448 050	0.56
Total 3			**5 145 970**	**6.41**
		Total Operating Cost	**11 107 637**	**13.83**

17.4 Recommendations

The lithium (Li_2O) recovery obtained during the laboratory tests was good (more than 85%). But the head grade of the used sample was quite high (~1.23% Li_2O). According to the mining plan, the run of mine grade will be around 0.91% of Li_2O.

Therefore, for future studies, we recommend doing some additional laboratory tests and pilot plant tests to point out the lower grade effect on the recovery. These additional tests will include the following:

- Testing the floatability of the low grade ore
- Making a flotation lock cycle test
- Testing the efficiencies of more reagents and water from plant location.
- Rheological tests (slurries thickening and filtration)

We roughly estimate these lab tests cost at $100,000. In addition it will be worthy to make a pilot plant flotation campaign. The campaign will require at least 50 mt of ore. We estimate the cost of that pilot plant campaign at $500,000.

18 Project Infrastructures

18.1 Glen Eagle General Site Plan

The Authier Project will require the construction of several components and facilities which will all be located at the mine site. The following section describes the major components of the required constructions and the general layout is shown in the next Figure18-1.

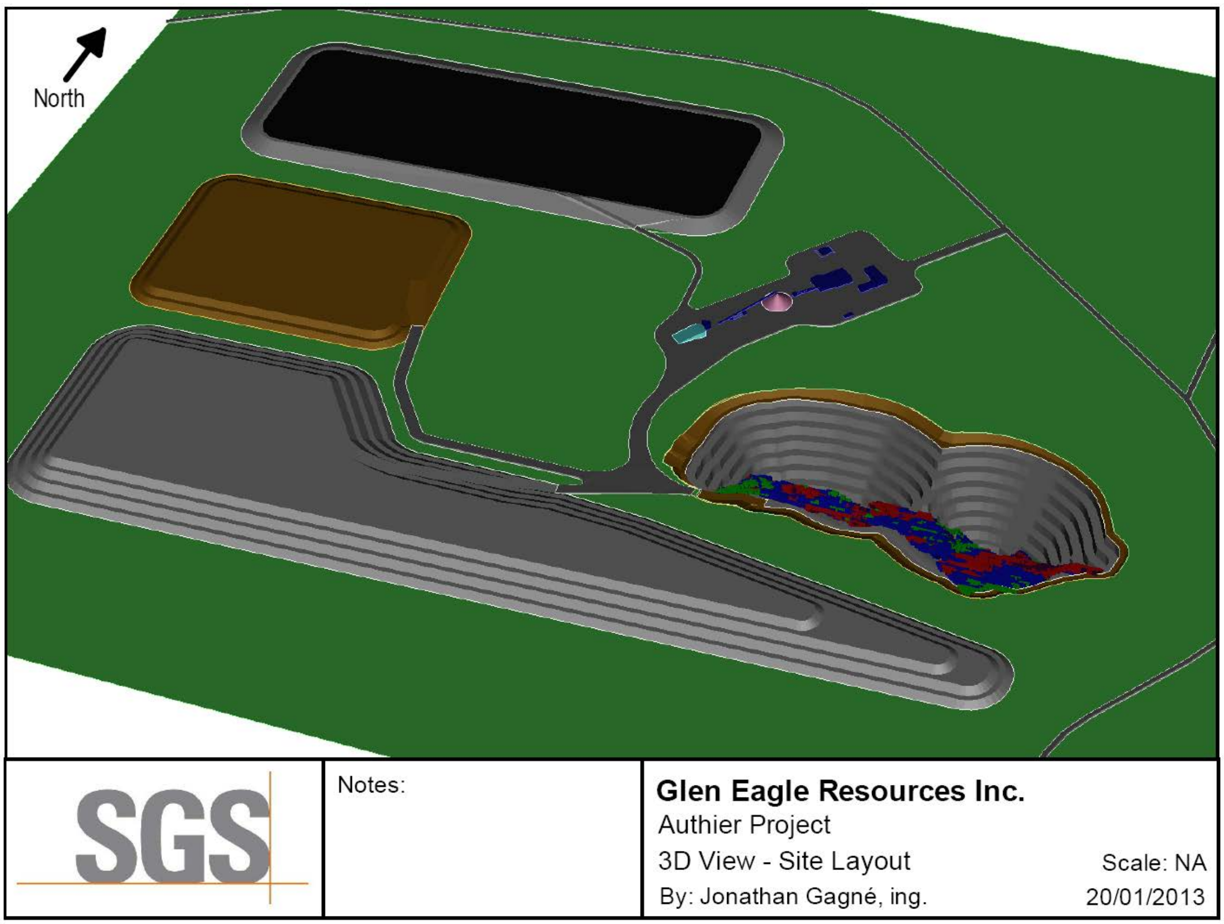

Figure 18-1: 3D View of Site Layout of Authier Project

18.2 Listing of On-Site Major Infrastructures

- **Processing Facilities**

This is described in section 17.

- **Main Building - Offices and Mechanical Shop**

The proposed main building will be a pre-engineered building with steel cladding and roofing, adjacent to the process building, warehouse, etc. The building will be a large structure that will includes:

- Offices for each department (administration, engineering, geology, etc.)
- First aid facility
- Welding shop
- Secondary warehouse
- Vehicles repair shop for heavy and light vehicles, a wash-bay and a 20 t overhead crane
- Electrical shop
- Mine dry

The service complex will have a footprint of ±1,800 m^2.

- **Electrical Power and Substation**

The electrical power will easily be available from Hydro-Quebec, as described in Section 18.3 below. The estimated power demand for the project is estimated to be 5.0 MW as summarized below.

Table 18-1: Estimated Project Power Requirement

Area	Power - MW
Crushing	0.76
Processing	2.69
Infrastructure	1.10
Subtotal	**4.55**
Security Factor - 10%	0.45
Total	**5.00**

The power will be delivered to the substation from a 25 kV overhead power line to a new substation. The voltage will be first stepped down to the selected voltage of the crushing and grinding installations, most likely to 4.16 kV, and then to the standard lower voltages.

- **Emergency Generator**

The emergency power will be provided by a diesel generator, mainly to reduce the down time in the concentrator following power failure, and also to have a minimum of lighting, heating and communication services available. The emergency estimated power need is 750 kW.

- **Warehouse/Laydown Yard**

A ±1,000 m^2 building will be used as main warehouse. The building will be a pre-engineered steel frame building covered by cladding and roofing panel. The building will be assembled on site. An outside area, adjacent to the warehouse building will be reserved to store large size consumables, such as drill rods, steel products, etc.

- **Fuel Farm**

The storage capacity at the mine site will be ±75,000 litres for fuel, in one saddle mounted tank, and ±5,000 litres tank for gasoline, these quantities are sufficient for one week production at the peak capacity, i.e., at year-5 . The fuels tank will have facilities to refuel the mine and support mobile equipments. These tanks will be field-erected steel tanks, within proper containment.

- **Explosives Magazines**

Two explosives magazines will be installed on site. One of these magazines will house priming explosives products such as caps and detonating cord while the second will house all packages explosives and boosters. The magazines will be bermed and strategically disposed to meet provincial and federal explosive regulations. As the main explosives suppliers are at close distances from the mine site, the magazines will be of reduced capacities.

- **On-Site Roads**

On-site gravel roads will be built to service the processing plant and all mine services. Haul roads needed to give access to resources and waste piles will be built according to recommended specifications for the off-road 37 tonnes trucks. The main roads to be built are shown on the next Figure.

- **Tailings Management Facilities**

Please refer to Section 17 for complete description.

➢ **Waste Rock Stockpile**

A waste rock material stockpile will be erected at proximity of the mine entrance/exit. This stockpile will be composed of rock material that does not contain enough mineralized material to be economically processed. It will have a volume of approximately 19.0 Mm3 (average area of 475,000 m2 and 40 m height) and will be strategically located to minimize hauling distances, and thus the size of the mining fleet. The pile will be deserved by a 10 % access ramp and will have an overall slope angle of 32 degrees. Refer to next Figure for a visual sketch of this waste pile.

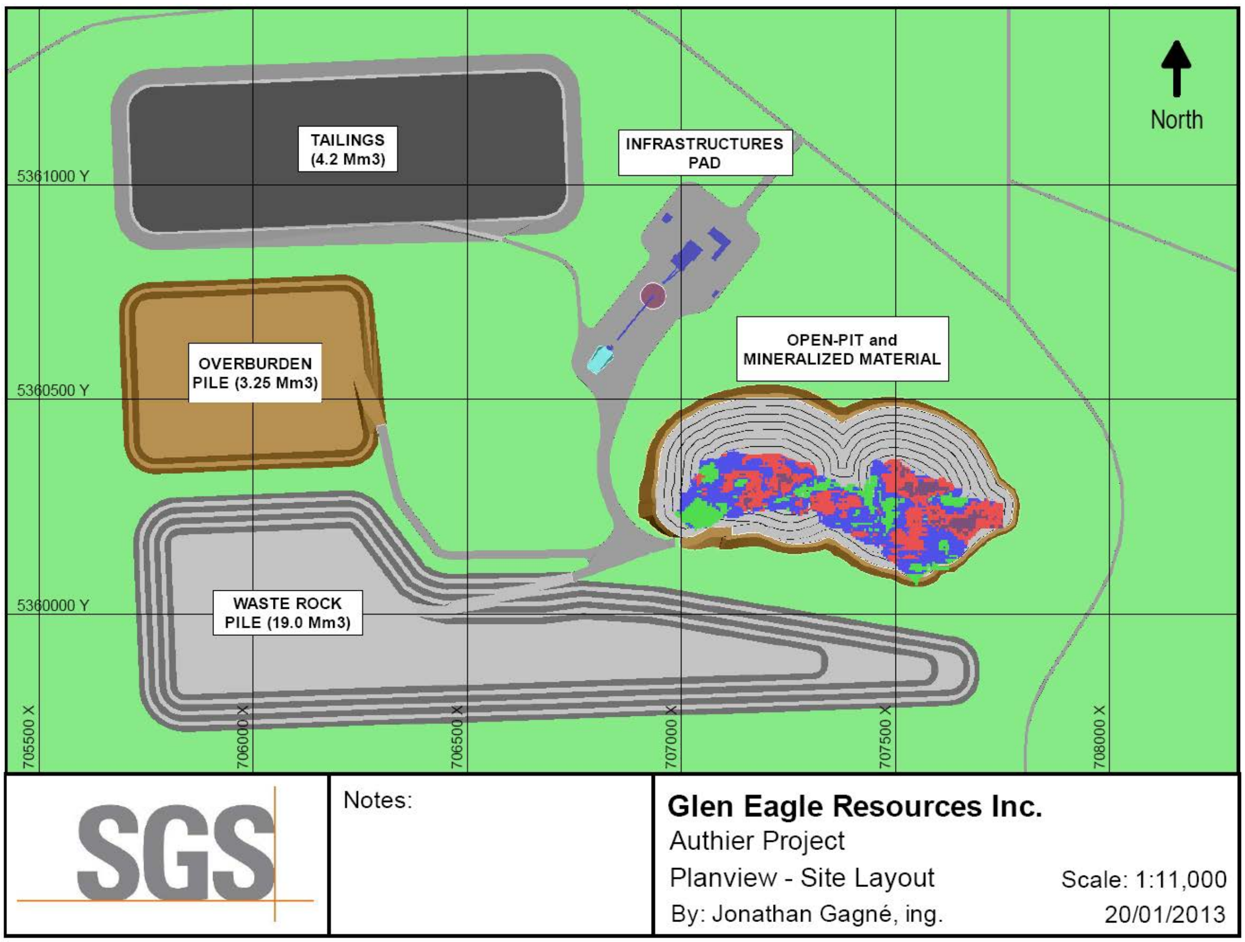

Figure 18-2: Authier Project Site Layout Plan View

➢ **Overburden Stockpile**

An overburden material stockpile will be erected at West of the property. This stockpile will be primary composed of top soil material that need to be remove in the first operational years in order to reach hard rock material containing mineralization. It will have a volume of approximately 3.25 Mm3 (average area of 216,000 m2 and 15 m height). The pile will be deserved by a 10 % access ramp and will have an overall slope angle of 20 degrees. Refer to previous Figure for a visual sketch of this waste pile.

- **Water Supply**

No detailed investigations into the water requirements and supply sources have been carried out. It is envisaged that water for processing would be obtained from several sources and would be treated prior to use in the process. Primary water sources would be from pit dewatering, collection of surface runoff in natural or artificial structures, existing ponds, reclaim water from the TMF and other sources. Studies on the water supply balance and remedial measures will need to be conducted as part of the next developing stages, either prefeasibility or feasibility studies.

18.3 Listing of off-Site Major Infrastructures

The major off-site infrastructures are defined as:

- **Electrical Network**

At this time, there are no electrical lines which can connect Glen Eagle facilities to Hydro-Québec power distribution network. Based on a preliminary evaluation from Hydro-Québec, a total of 2.7 km of new network power line needs to be installed; the following workings have to be done in order to supply power to Glen Eagle mine site:

- Modify a section of the line on the Saint-Luc road over 1.5 km.
- Extend the connection over 1.2 km by replacing more than 20 existing poles in order to support new power lines.
- Installed 18 new posts for the exclusive use of Glen Eagle.

For the site power distribution at the Authier project, an electrical substation located near the processing building has to be installed (refer to previous Section).

- **Off-site Road**

The Glen Eagle property is connected to 109 Road with secondary gravel road by a local road. One of these access roads (as presented in yellow on Figure 18-3) will have to be upgraded to allow concentrate shipping to various destinations, such as local market and/or railway infrastructures. This actual gravel road offers a good foundation and few sections will have to be levelled and expanded

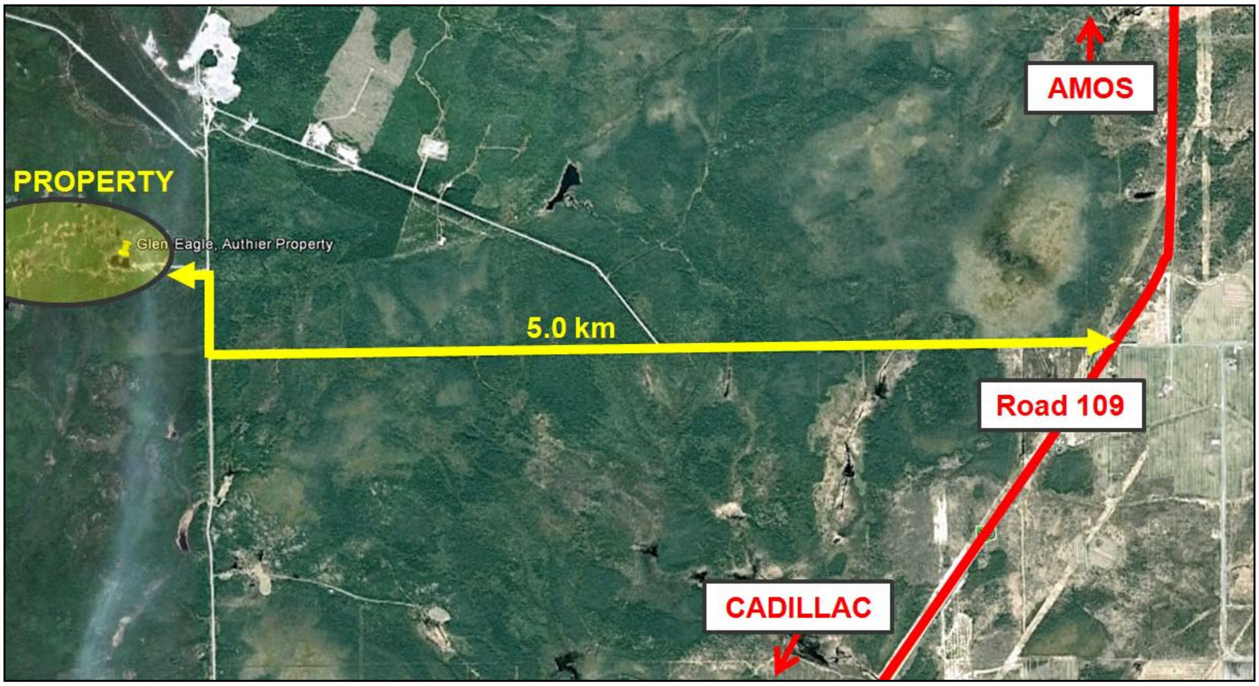

Figure 18-3: Authier Property and Local Access Roads

- **Railway infrastructures**

The concentrate production that will be ship by railway to the US market needs to pass by a storage and railway cars loading facilities. SGS discuss with a local contractor, J&R Dumas Inc., who his actually responsible to coordinate all concentrate shipping activities of Agnico-Eagle Mines zinc concentrate to a consumer located in Valleyfield, Qc. This contractor own two sideways and a concentrate storage facilities, all located in Cadillac, (refer to next Figure) and who has demonstrated an interest to provide Glen Eagle Resources with the same service that they are offering to Agnico-Eagle Mines. The actual sideway can be used but a new storage facility (a prefabricated steel building or a dome), capable to store 1,500-2,000 tonnes of Glen Eagle concentrate, needs to be built.

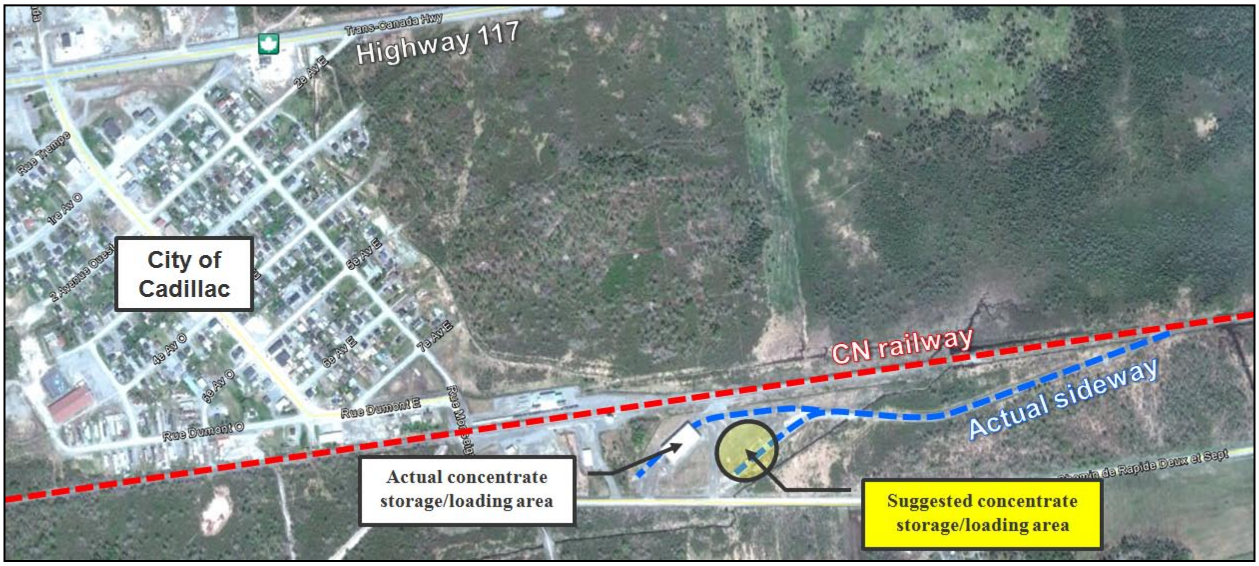

Figure 18-4: Proposed Local Railway Storage & Loading Concentrate Facilities

Figure 18-5: Example of Storage & Loading Concentrate Facilities

19 Market Studies and Contracts

19.1 Introduction

Lithium is the lightest of all metals with a density of 0.543 g/cm^3, this is almost 5 times lighter than aluminum that has a density of 2.69 g/cm^3, considered a lightweight metal. Lithium characteristics, mainly its high specific heat capacity and its high electro chemical potential make it a product for many applications in several industries.

19.2 Demand

The most important growth in demand is coming from the Li-ion battery market developed in the first decade of the 21st century. Lithium consumption is mainly in carbonate, ±45% of total, followed by technical-grade spodumene concentrates, ±16%, and lithium hydroxide, ±17%. Technical-grade spodumene concentrates are mainly used in glass and ceramic markets.

19.3 Automotive Batteries

The greatest expansion for the lithium market is certainly the one for the automotive batteries. In an extract from the 4th Lithium Supply and Market Conference held in Buenos Aires, Argentina, it is mentioned that during 2011, only 4 models from 4 manufacturers, of lithium-ion equipped automobiles were offered for an estimated total of 65,000 cars sold. For 2012, at least 12 models will be available for sale, including those from known manufacturers like Ford, GM, Honda, Toyota, Nissan, VW and BMW. There is no doubt that the automotive batteries market is on the verge of a major increase.

19.4 Base Case Spodumene Concentrate Price

As there was no market study available from Glen Eagle, SGS relied on public information's to select a lithium concentrate price to be retained in the base case. Spodumene concentrate prices were available, on public sites for the time being, but for concentrates with grades of 7.5% Li_2O and 5.0% Li_2O only. Long term forecasts for lithium carbonates and hydroxides were also available as shown in pink and red full lines of the next Figure. We therefore applied the same future trend, to the spodumene concentrates of 7.5% and 5.0%, shown in dotted dark and light blue lines, as those of the lithium carbonates and hydroxides known prices, with an extrapolation for the 6.0% Li_2O concentrate, shown in green full line. The next Figure is illustrating the base case spodumene concentrate price of $525/tonne retained in this study.

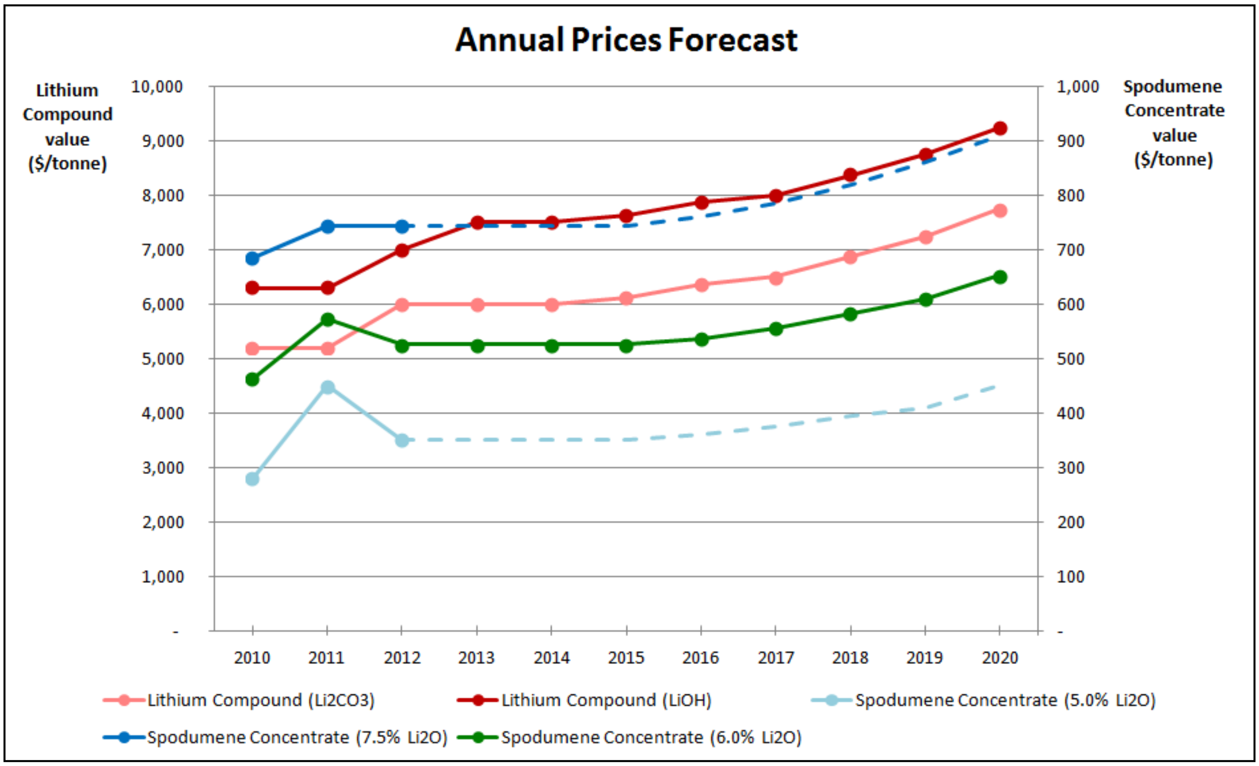

Figure 19-1: Spodumene Concentrate Forecast Price ($/t)

20 Environmental Studies, Permitting and Social or Community Impact

Glen Eagle Resources mandated Dessau and GFE Forestry & Exploration Services to make an environmental study of the type "Baseline" for Authier property. The Preliminary environmental study was delivered in December 10th of 2012 for the Authier Property and is entitled: « Réalisation d'une étude environnementale préliminaire sur la propriété Authier ». The following is a summary of the preliminary report. Activities by DESSAU and GFE were performed to determine constraints linked to water and sediments quality and to environmental (physical, biological, human) impact. This preliminary report is based on public bibliographical data and field work (summary inventories, sampling and photographic).

20.1 Property Location

The Authier property is located in Abitibi-Témiscamingue, about 45 kilometres northwest from Val d'Or and 15 kilometres from Rivière-Héva. The property is link to the highway 109 between Rivière-Héva and Amos (Figure 4-1).

20.2 Physical Aspects

20.2.1 Physical Environment and Impact

20.2.1.1 Topography

The topography of Authier Property is relatively flat. The average elevation, 350 metres, varies from 320 to 380 metres. The relief is located to the northeast and consists in little hills due to an Esker. The overburden is thin (2 to 3 metres) and is characterized by tills and clay. To the south, the water is flowing to the south, in the rest of the Property the water is flowing to the west in little streams and ponds. The area is mostly covered by forest composed of black spruce trees, fir trees and poplars.

20.2.1.2 Hydrology and Hydrogeology

Two main watersheds divided the water flow in the Authier Property: Kapitagama Lake watershed and Croteau Lake watershed. The watershed limit is located at the northern limit of the study zone. This report reveals no bodies of water or streams in the study area, taking apart little streams and ponds. The hydrographic network is composed of the little Sigouin stream branches. Two lakes are located at the margin at southeast end of the study zone. Summary inventories did not revealed major features concerning fish's habitats and benthic community in water streams.

The Esker of St-Mathieu-Berry is located to the north end of the property. The groundwater contained under a layer of clay has exceptional water qualities and is the water source of nearby towns. The groundwater flows towards north, except in the study zone where it is heading east and to the Harricana River.

The Esker is located at 2 km from the project area and the study zone is on another watershed. Thus the impact of future works in the study zone was qualified as low.

However, Dessau and GFE recommends avoiding every construction to the north as: in so far as the Esker is a source of drinkable water for Amos, continue applying corrective measures for the protection of the Esker as stipulated in the document "Protection des aquifères granulaires (Esker)" of the September 22th 2011 by la "Tables locales de gestion intégrée des ressources et du territoire". These measures aim to prevent any impact on the Esker.

20.2.1.3 Water Quality

Surface water was sampled during August 2012. The visited water bodies included five ponds and one stream in Preissac watershed. Statements contain physico-chemical profils of samples. Results show that the water body's physic-chemical profils didn't match the recommendations' of CCME (Canadian Council of Ministers of the Environment).

The chemical results summary for surface water is showing that the following parameters exceed CCME and/or MDDEFP: alumina, calcium, iron, hydrocarbons C10-C50; nitrite & nitrates; total phosphor, pH, lead, zinc and turbidity.

20.2.2 Biological Environment

For the floristic, fauna and terrestrial habitats aspects, all species that might be affected by the mining project were documented from public databases and inventoried during the field campaign. Following inventories were made:

- Floristic Summary
- Vegetation Summary
- Fauna summary
- Amphibian and reptilian summary
- Bird summary.

20.2.2.1 Forest environments

Field visit was made in August 2012. In spite that all the area wasn't visited, this work permits to validate information from the forestry map study area, where dominates a mix of coniferous trees. One deciduous zone was identified on the forestry maps from MRNF.

Forest of interest corresponds to old forest or mature forest. Both categories were identified on the forestry maps from the MRNF. Old forest in the zone is only 90 years and mature forest is 70 years. At least, in the study zone, 52% (483 ha, 480 ha mature forest, 3 ha old forest) of the area is covered by forest of interest.

20.2.2.2 Wetlands

As part of the application of the LQE (« Loi sur la Qualité de l'Environnement du Gouvernement du Québec »), the inventory of wetlands was done to verify the necessity to make a request for authorization from the Ministry before any constructions or works in the wetlands.

Wetlands were identified with forestry maps from MRNF. Three types of wetlands were identified, unproductive forest lands, forest environment with bad drainage (potential wetlands) and forest environment with organic deposit (peat bog). According to the MRNF maps, the study area contained 202 hectares (22% of the area) of those three types with 185 hectares of peat bog.

Only fews bogs were found in the project area. Those few bogs did not revealed any major particularities during inventories. The only recommendation will be, if there is new drilling program to the north, to proceed during winter to avoid water flows towards the bogs.

20.2.2.3 Terrestrial Fauna

The description of the fauna in the study area is based on databases from the CDPNQ (Centre de Données sur le Patrimoine Naturel du Québec), AARQ (Association des Aménagistes Régionaux du Québec), AONQ (Atlas des Oiseaux Nicheurs du Québec) and ÉPOQ (Étude des populations d'oiseaux du Québec). Also data from inventories prepared by MDDEFP (Ministère du Développement Durable, de l'Environnement et de la Faune) were used.

Herpetofauna: the summary confirmed the presence of four amphibians and one reptilian in the area. Beaver ponds and wetlands contained the most part of those species.
Avifauna: observed birds were mainly associated to the ponds of the wetlands because they were observed during the characterization of the ponds and wetlands. Some other species were found in the wood.

Mammals: observations led during field work confirm the presence of four mammal species (squirrel, beaver, moose and mouse).

20.2.2.4 Fishes and Habitat

Visual characterizations were made for five ponds and one stream in Preissac watershed. Results show that each pond is populated by beaver and contains beaver dams. The substract of each pond is made of sand and organic material. The surface of ponds varies between 503 and 10,120 m2.

A total of thirteen sticklebacks were found in two of the five ponds. Result shows that the average weight is 0.86 kg in the first stream and 1.81 kg in the second one. The average lengths inside the same streams are 49.70 and 60.25 cm.

20.2.2.5 Benthic community

The benthic community of the different sampling stations is mostly constituted of, nematodes, annelids, insect larva's and mollusks. Results are showing between 4 and 34 different species with a variation of the number following the sampling stations.

Endangered Wildlife and "habitats fauniques désignés", this study was documented using databases from CDPNQ, MRN, and from COSEWIC (Committee on the Status of Endangered Wildlife in Canada, COSEPAC in Quebec). If a species is not registered in the database for a specific area, it doesn't justify the real absence of this species. In theory, the studied zone could include about 18 species with specific status. This information came from the analysis of the public database and from the species repartition area. Among those 18 species, 9 birds and 9 mammals could be present on the property.

20.2.2.6 Environmental requirements

The MDDEFP did also some site drilling and future mine site inspections during winter. GFE did also an inspection in September 2012 to verify the vegetal regeneration. The 3 drilling programs of Glen Eagles Resources modified the territory. This type of work created new paths and drill pads. Those workings were done within supervision to minimize impacts. Conclusions show that regeneration has already started on all visited drill sites and access paths.

20.2.2.7 Conclusion

According to public databases and from field inventories lead during this study by Dessau and GFE, no endangered species or habitats were found. However it is recommended to produce exhaustive inventories to validate or invalidate the presence of specific fauna, flora or habitat. At the end of the drilling program, the revegetation appears to be in a good state.

20.3 Social Impact

The Authier property is located about 26 kilometres from the Algonquin community of Pikogan and it is in Algonquin nation claimed territory. The "Conseil Tribal de la Nation Algonquine Anishinabeg" claimed in 2011 over 650 000 km2 of territory including Abitibi and Authier Property. Furthermore, municipalities of La Motte, Preissac, Rivière-Héva and Amos are located close to Authier Property.

In this context, a communication plan will be prepared and presented to open a dialogue concerning interests and preoccupations of municipalities, communities and organisms implied directly or indirectly with the mining project of Authier.

20.4 Environment permitting

Following permits and authorizations were already delivered to Glen Eagle for exploration and drilling campaign to create paths and drill pads.

Annual intervention permit 2010-2011, MRNF, Number 3008769, July 28, 2010.
Annual intervention permit 2010-2011, MRNF, Number 3009512, December 3, 2010.
Annual intervention permit 2011-2012, MRNF, Number 3010663, July 22, 2011.
Annual intervention permit 2011-2012, MRNF, Number 3011051, September 29, 2011.

Once the Authier Project will have undergone a complete Environmental Impact Assessment and be authorized by the Government pursuant to Section 31.5 of the Act, it would still be subject to Section 22 of the Environment Quality Act and must therefore obtain a general certificate of authorization. The issuance of that authorization, however, should only be a formality as the certificate issued pursuant to Section 31.5 of the Act binds the Minister as to where he exercises the powers provided in Section 22. In addition to the certificate of authorization required under Section 22 of the Environment Quality Act, the proponent must obtain various permits, authorizations, approvals, certificates and leases required from the appropriate authorities.

- A complete environmental baseline study shall be performed in order to define, in compliance with all applicable guidelines, policies, regulations and laws in Quebec and Canada, the current state of reference of the Authier Project receiving environment;
- Exhaustive inventories for the fauna and the flora to confirm that there is no identified species with a specific status;
- An acoustic study for obtaining permits and authorizations;
- An archaeological and cultural potential study for obtaining permits and authorizations;
- A mining infrastructure placement and a risk analysis study for obtaining permits and authorizations.

The authorization application and permitting process is expected to take many months. Applications may be filed concurrently with the construction work and should not therefore impact the project development should it proves to be positive. The aim of the permits and authorizations is to take account all environmental requirements to the provincial and federal level.

End of extracts from the study « Réalisation d'une étude environnementale préliminaire sur la propriété Authier » by DESSAU and GFE.

20.5 Conceptual Closure and Reclamation Plan

At this time, there is no detailed closure plan for the future mine operation in Authier Property. The following sections are presenting the required work to close the site according to the actual regulations and requirements of the Quebec Environmental Act.

20.5.1 Overview

In accordance with the law on mines (L.R.Q., chapter M – 13.1, section III, article 232, August 2012) a detailed closure plan must be submitted to the Ministère des Ressources Naturelles et de la Faune ("MRNF") prior to getting the Global Certificate of Authorization. The Guide and method of preparation for the plan as well as the general requirements for the restoration of Quebec mining sites (Guidelines for Preparing a Mining Site Rehabilitation Requirements, 1997, hereafter to as Guide), prepared by MRNF in collaboration with the Ministère du Développement durable, de l'Environnement et des Parcs ("MDDEP") must be followed with the closure plan for a mining site.

The main closure objectives of a mine are following:

- Dismantle and remove buildings and infrastructures used during the mining process;
- Secure the site (open pit, piles…);
- Characterize impacted soil, surface water and groundwater during mining operation;
- Treat the soil and water to the accepted level of contamination for the site;
- Establish adequate vegetation for long-term stabilisation of the land surface;
- Limit or eliminate long-term care and maintenance for the site.

20.5.2 Site Characterization and Reclamation

As required for the closure of a mine site, the characterization work based on the requirements of the Quebec Environmental Act must be carried out. If land contamination is proven, an environmental rehabilitation plan will must be produced and submitted to the MDDEP, in accordance to the Directive 019 on the mining industry. The land will have to be decontaminated in order to meet the applicable criteria.

20.5.3 Dismantling of Buildings and Infrastructure

All the buildings and infrastructures will be dismantled or demolished following the end of operations. Following the removal of the buildings, infrastructures and the decontamination of soils, the surface will be shaped and vegetation established. Sanitation facilities, petroleum products, waste and hazardous waste sites will be dismantled in accordance with all regulations of the current law.

20.5.4 Waste Rock and Tailings Pile

The waste rock and tailings pile will had to meet the minimum requirements as defined in the Guide and will had to meet the terms of stability. The waste rock and tailings will be planned to enable revegetation during the project. At the end of the operation, the revegetation will be completed. The pile will be conformed to the minimum stables slopes determined from stability analyses performed for the most critical part. Thus, no stabilization will be required at closure. A guarantee will have to be prepared for the Quebec Ministry of Natural Resources to cover a portion of the reclamation costs.

20.5.5 Open Pit

Following the mining operations, the open pit will be filled with groundwater and precipitation (rain and snow falls). Generally this consists to stopping the pumping system. All road access to the pit will be closed with waste rock from the piles.

20.5.6 Mining Effluent

Following the closure, the processing unit and water control structures will remain in operation until MRNF/MDDEP criteria are met. Then the ditches will be filled with excavation material used as protective berms during construction of the mine. The ditches will have to be revegetated. The dikes of the sedimentation ponds will be breached to allow free circulation surface run-off water.

20.5.7 Monitoring Program

An environmental monitoring will be necessary during the production period and the rehabilitation work. The future complete baseline report will presents precise estimated cost and procedures, as well as the five-year minimum monitoring program required.

20.5.8 Timing and Economic Considerations

The economic considerations and the timing will be covered in detail in the detailed closure plan. According to the Mining Act currently in force, the amount of financial guarantee required presently is equivalent to 70% of the anticipated restoration cost for the accumulation areas.

At this stage of the project, it is not possible to make a detailed estimation of the restoration; a preliminary estimate was prepared based on previous in house studies and in agreement with the regulations of the Mining Act, the following Table summarizes these costs.

Table 20-1: Complete Restoration Costs

	Description	Cost - CAD$
1	**Satationary Installations**	1,200,000
	Dismantling maintenance facilities	
	Dismantling plant facilities	
	Demolition of stationary equipment	
	Fill drainage ditches and basins, including berm protection	
	Overburden transport and revegetation	
2	**Open-Pit Restoration**	450,000
	Embankments protection with drainage ditches	
	Closure and securing the pit	
3	**Waste Rock and Tailings Pile**	1,125,000
	Overburden transport and installation	
	Fill drainage ditches and basins, including berm protection	
	Revegetation	
4	**Sedimentation (polishing) pond**	125,000
	Dams clearing	
	Overburden transport and revegetation	
	Revegetation	
5	**General**	350,000
	Characterisation of contaminated sites and decontamination of soils	
	Desinfectation of septic tanks	
6	**Five year Follow-up Respect of Environmental Regulations**	750,000
	Total	**4,000,000**

According to the Mining Act currently in force, the amount of financial guarantee required presently is equivalent to 70% of the anticipated restoration cost for the accumulation areas. It is important to remind that the Quebec Government has already stated that the amount of the financial guarantee will be increase to 100% during 2013. From the above Table the amount of the financial guarantee is the total of: 3-The Waste Rock and Tailings Pile, plus 4-The Sedimentation Pond, plus 6-the cost of the follow-up of the Respect of the Environment Regulations, on which total we apply a factor of 70%. These costs have to be paid to the Government on a three year basis: 50% on the first year, 25% for each of the two following years, the summary is in the following Table.

Table 20-2: Restoration Cost of Accumulation Areas

	Description	Cost - CAD$
3	**Waste Rock and Tailings Pile** Overburden transport and installation Fill drainage ditches and basins, including berm protection Revegetation	1,125,000
4	**Sedimentation (polishing) Pond** Dams clearing Overburden transport and revegetation Revegetation	125,000
6	**Five year Follow-up Respect of Environmental Regulations**	750,000
	Total	2,000,000
	Amount of the financial guarantee: 70%	**1,400,000**

These costs are part of the sustaining capital and are included in the Project Cash Flow.

21 Capital and Operating Costs

21.1 Capital Cost

The total capital expenditures cost (CAPEX) is estimated at an overall accuracy of ±30%, which is the standard for a preliminary economic assessment. The CAPEX were defined by SGS and Bumigeme using in-house database, the Mine & Mill Equipment Costs Estimator's Guide: Capital & Operating Costs (2012) and budget bid prices. The total required investment is estimated at 45.0 M$ and includes a contingency of 12-15 %. Refer to Table 21-1for the CAPEX breakdown.

Based on the fact that 29.5 % of the in-pit resource is located within a claim share between Glen Eagle Resource (70%) and a third party (30%), it was suppose that the third party will have to participate into the initial investment in order to get his 30 % profits royalty (NPI). Therefore, the amount to be paid by the third party is 8.9 % of the total CAPEX (29.5 % x 30 %), and is equivalent to 4.0 M$. The resulting CAPEX left to be paid by Glen Eagle is then 41.0 M$.

The capital costs do not include:

- Costs to obtain permits;
- Costs for pre-feasibility and feasibility studies;
- Any provision for changes in exchange rates;
- GST/QST, or any other Taxes;
- Project financing and interest charges;
- Price/cost escalation during construction;
- Import duties and custom fees;
- Pilot plant and other testwork;
- Sunk cost;
- Exploration activities.

Table 21-1: Capital Cost Summary

Item	Cost	% of total
Infrastructures		
Hauling road upgrade	$ 200,000	0.4%
Site preparation	$ 500,000	1.1%
Drinking water treatment plant	$ 50,000	0.1%
General maintenance workshop	$ 2,250,000	5.0%
Warehouse	$ 200,000	0.4%
Fuel storage tank & pumping station	$ 60,000	0.1%
Gas storage tank & pumping station	$ 25,000	0.1%
Main office building	$ 1,400,000	3.1%
Explosive magazines + Access	$ 80,000	0.2%
Telecommunication system	$ 125,000	0.3%
Electrical power line	$ 300,000	0.7%
Sub-station	$ 1,000,000	2.2%
Processing plant	$ 28,350,000	63.0%
Tailings and polishing pond (site prep., dike)	$ 500,000	1.1%
Concentrate loading facilities for railroad transport	$ 350,000	0.8%
Equipments not under leasing		
Mechanical Service Truck	$ 165,000	0.4%
Boom and flatbed truck	$ 150,000	0.3%
Tower lights	$ 60,000	0.1%
Loader CAT966 Concentrate	$ 400,000	0.9%
Water pumps + accessories	$ 100,000	0.2%
Pick-Up 4 x 4	$ 300,000	0.7%
Others		
Equipments spare parts - Starting inventory	$ 585,000	1.3%
Computers / Softwares / Printers / Network	$ 300,000	0.7%
EPCM - Mains Infrastructures (10%) *	$ 535,000	1.2%
Mill working capital	$ 3,275,000	7.3%
Mine working capital	$ 540,000	1.2%
Royalties by-back	$ 1,000,000	2.2%
Subtotal	$ 42,800,000	95.2%
Contingency (15%)	$ 2,167,500	4.8%
Total	$ 44,967,500	100.0%
Total rounded	**$ 45,000,000**	100.0%
Paid by the Third Party	**$ 4,000,000**	8.9%
Paid by Glen Eagle	**$ 41,000,000**	91.1%

* The EPCM and the contingency attributable to the Processing Plant are already accounted for into the 28.35 M$ cost.

21.2 Sustaining Capital Cost

The sustaining capital will last overall operating years (1 to 10) and will consist of:

- Site preparation;
- Tailing ponds dikes enhancement;
- Rehabilitation and decommissioning;
- Others (a provision for any unexpected costs).

The sustaining capital will be spread as follow:

Table 21-2: Sustaining Capital

Year		0	1	2	3	4	5
Site preparation	$	Included in CAPEX	250,000	250,000			
Equipment leasing	$		1,526,225	1,526,225	1,825,225	2,184,200	1,974,725
Tailing ponds	$			400,000		400,000	
Rehab. and decommissioning	$		700,000	350,000	350,000		
Total	$		**2,476,225**	**2,526,225**	**2,175,225**	**2,584,200**	**1,974,725**

Year		6	7	8	9	10	Total
Site preparation	$						**500,000**
Equipment leasing	$	657,975	657,975	149,500	-	-	**10,122,550**
Tailing ponds	$	400,000		400,000			**1,600,000**
Rehab. and decommissioning	$					2,600,000	**4,000,000**
Total	$	**1,057,975**	**657,975**	**549,500**	**-**	**2,600,000**	**16,222,550**

21.2.1 Site preparation

Site preparation will be spread over 3 years (Year 0, 1 and 2) and is defined as preparing the field for overburden removal, construction of gravel roads, excavation of trenches to deviate watercourses, etc.

21.2.2 Equipment leasing

In order to reduce the burden of a high Capex, the option of leasing was retained. The leasing procedure is an option offered by heavy equipments manufacturers, like Caterpillar, Komatsu, Hitachi and others, to future clients that are requiring new mining equipments. The arrangements are made under leasing contracts, in which the manufacturer agree to lease for a fixed rate and period of time, an equipment for which the ownership will be transferred to the client when all conditions are fulfilled. The leasing contracts are generally for a 4 to 5 years period of time. Here is one example to better illustrate this business agreement.

Table 21-3: Equipment Leasing Cost compared to Purchase

Description	Year-1	Year-2	Year-3	Year-4	Year-5	Total - $	%
Hydraulic Shovel - 6m^3 bucket	Budget Price Value					**1,650,000**	100
Leasing payments - 23% of budget price per year	379,500	379,500	379,500	379,500	379,500	**1,897,500**	115
Leasing cost compared to purchase						**+247,500**	

In this study the following equipments are assumed to be under leasing:

- Hydraulic shovel
- Off-road truck
- Wheel loader
- Bulldozer
- Grader
- Secondary shovel
- Production drill

For the duration of the project, the total estimated leasing cost is 10.1 M$, as shown in the CashFlow spreadsheet.

21.2.3 Tailing ponds

Rehabilitation and decommissioning

*Please refer to Section 20 for cost details and cost breakdown.

21.3 Operating Costs

The operating costs (OPEX) are estimated at an overall accuracy of ±30%, which is the standard for a preliminary economic assessment. These costs were defined by SGS and Bumigeme using in-house database, the Mine & Mill Equipment Costs Estimator's Guide: Capital & Operating Costs (2012) and formal bids.

- The operating costs do not include:
- Any provision for inflation;
- Any provision for changes in exchange rates;
- GST/QST;
- Corporate administration and head offices costs; and
- Exploration activities.

21.3.1 Mining Costs

The total mining cost related to open pit operations over the life of the mine is estimated at 180.71 M$. From this total:

- 100.4 M$ is attributable to hard rock mining by the operator;
- 67.9 M$ is attributable to hard rock mining by a specialized mining contractor;
- 12.4 M$ is attributable to overburden removal by a combination of the operator and a specialized contractor.

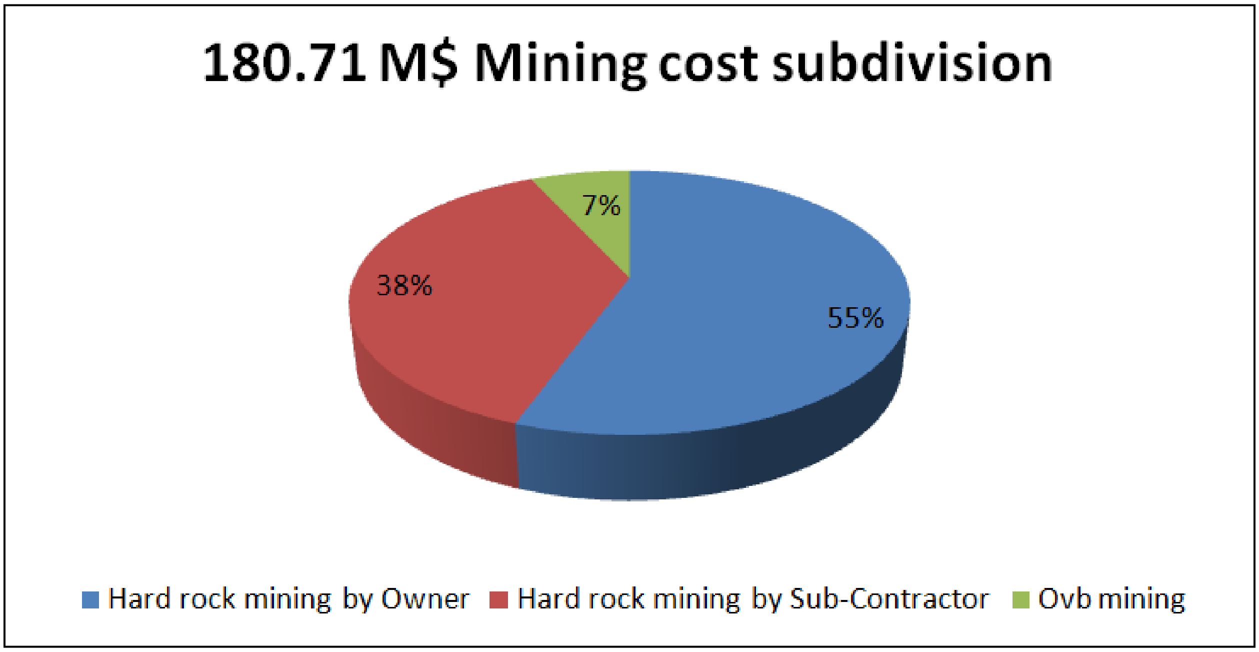

Figure 21-1: Mining Cost Subdivision

21.3.1.1 Hard rock mining cost (operator)

The hard rock mining costs have been estimated for the the whole duration of the project, therefore there is time escalation. The mining costs are including the leasing costs for the main mining equipments as described above in the item referred to as Equipment leasing. The global hard rock mining costs are summarized in the following Table.

Table 21-4: Mining Cost in Hard Rock

Hard Rock Mining Cost	$/t
Drill & Blast: including redrill, reblast and preshearing	0.96
Loading & Hauling: ore & waste	1.22
Mining Services: water pumping, road maintenance, etc.	1.02
Total mining cost	**3.20**

21.3.1.2 Hard rock mining cost (sub-contractor)

The estimated mining cost attributable to a specialized mining contractor was estimated using the general mining cost calculated for the operator plus a prime of 35 % to account of all sub-contractor charges and profit margin. The final estimated contracting operating cost is $4.32 /tonne mined (3.20 $/t mined + 35 %). According to the mine plan, the contractor should be limited to mine only waste material

21.3.1.3 Overburden mining cost

The overburden mining cost was estimated to 2.50 $/tonne mined. From discussion with local contractor working on the site since few years, the overburden removal will be free digging (no drilling and blasting will be necessary). This operating cost is based on SGS experience and from benchmarking studies of similar projects.

21.3.2 Processing Costs

The processing involve 7.77 M tonnes of resources for an amount of 107.5 M$ which equates to an average of $13.83 per tonne processed through the life of mine.

Refer to Section 17 for cost breakdown.

21.3.3 General and Administration (G&A) Costs

The total G&A cost is estimated at 39.0 M$ through the life of the mine, and average $5.02 per tonne milled, and is shown in the following Table.

Table 21-5: General and Administration Costs

Staff Salaries	Qty	$/pers/year	30% benefits	Total $/year	$/t milled
Open-pit Operation					
Mine superintendant	1	$ 100,000	$ 30,000	$ 130,000	$ 0.17
Shiftboss	4	$ 85,000	$ 25,500	$ 442,000	$ 0.57
Clerk	1	$ 50,000	$ 15,000	$ 65,000	$ 0.08
Mine Maintenance					
Maintenance superintendant	1	$ 100,000	$ 30,000	$ 130,000	$ 0.17
Shiftboss/Planner	2	$ 80,000	$ 24,000	$ 208,000	$ 0.27
Mine Engineering					
Engineering superintendant	1	$ 110,000	$ 33,000	$ 143,000	$ 0.19
Mine engineer	1	$ 90,000	$ 27,000	$ 117,000	$ 0.15
Surveyor	1	$ 65,000	$ 19,500	$ 84,500	$ 0.11
Surveyor assistant	2	$ 50,000	$ 15,000	$ 130,000	$ 0.17
Environmental technician	1	$ 55,000	$ 16,500	$ 71,500	$ 0.09
Geology					
Geology superintendant	1	$ 110,000	$ 33,000	$ 143,000	$ 0.19
Grade control geologist	1	$ 90,000	$ 27,000	$ 117,000	$ 0.15
Geoloogist technician	1	$ 65,000	$ 19,500	$ 84,500	$ 0.11
Administration					
Mine manager	1	$ 125,000	$ 37,500	$ 162,500	$ 0.21
Managing secretary	1	$ 50,000	$ 15,000	$ 65,000	$ 0.08
HR agent	2	$ 55,000	$ 16,500	$ 143,000	$ 0.19
Accountant clerk	1	$ 60,000	$ 18,000	$ 78,000	$ 0.10
Warehouse responsible	1	$ 75,000	$ 22,500	$ 97,500	$ 0.13
Warehouse employees	2	$ 60,000	$ 18,000	$ 156,000	$ 0.20
Nurse	1	$ 70,000	$ 21,000	$ 91,000	$ 0.12
I.T.	1	$ 65,000	$ 19,500	$ 84,500	$ 0.11
General					
Janitor	1	$ 45,000	$ 13,500	$ 58,500	$ 0.08
Security guards	4	$ 50,000	$ 15,000	$ 260,000	$ 0.34
Pick-Up driver	1	$ 50,000	$ 15,000	$ 65,000	$ 0.08
Total Salaries	**34**	**$ 1,755,000**	**$ 526,500**	**$ 3,126,500**	**$ 4.06**
Office fees				$ 50,000	$ 0.06
Roads maintenance				$ 90,000	$ 0.12
On site general expenses				$ 500,000	$ 0.65
Operating capital				$ 100,000	$ 0.13
Total G & A				**$ 3,866,500**	**$ 5.02**

21.3.3.1 Salaries

All staff personnel salaries plus 30% fringe benefits: government obligations, CSST, etc.

21.3.3.2 Office fees

All generals expenses such as electricity, printers, computers upgrade, softwares fees, telephone, internet, maintenance consumables, etc.

21.3.3.3 Road maintenance

Road maintenance cost account for the maintenance of the 5-6 km gravel road between the mine site and road 109, please refer to Figure 18-1. This cost will include adding sporadically crushed material and snow removal to maintain the road condition optimal for concentrate transport between mine site and deliveries locations. The maintenance of this road will be done by the mining equipments from the operator of the Project.

21.3.3.4 On site general expenses

On site expenses includes a multitude of generals expenses associated to a mining operating, ex: Major building maintenance, fuel tanks maintenance, any unplanned expenses, etc.

21.3.3.5 Operating capital

Operating capital includes all future estimated studies: environmental, rock mechanic, hydrogeological, marketing, etc.

21.4 Concentrate Transport Costs

The base case scenario evaluated in this PEA supposes that 50% of the produced spodumene concentrate will be sold to local market (within Abitibi), while the remaining portion will be sold to the US markets.

21.4.1 Local market

To evaluate the potential of the Project, SGS made the assumption that fifty percent of the produced concentrate, i.e. 50,000 tonnes per year, will be sent by truck to a local processing plant that is currently producing spodumene concentrate and is aiming at producing lithium carbonate (Li_2CO_3), in the near future.

The purpose of this study (the PEA) is to assess the economical potential of the Project by doing various assumptions. Therefore, it is important to highlight that at this time, there is no signed agreement between Glen Eagle Resources and any potential customer to buy Authier Project's spodumene production.

The local transport cost is calculated as follow:

Table 21-6: Cost of Local Concentrate Transportation

Activities	Cost	Unit
Truck loading at mine site	1.92	$/t conc.
Trucking transport cost (78 km) Truck unloading at destination	7.80	$/t conc.
Total	**9.72**	**$/t conc.**

21.4.2 US markets

To evaluate the potential of the Project, SGS made the assumption that fifty percent of the produced concentrate, i.e. 50,000 tonnes per year, will be sent by truck to a local railway station and then shipped by train to US market.

The purpose of this study (the PEA) is to assess the economical potential of the Project by doing various assumptions. Therefore, it is important to highlight that at this time, there is no signed agreement between Glen Eagle Resources and any potential customer to buy Authier Project's spodumene production.

To assess this cost, SGS discuss with a local contractor, J&R Dumas Inc., who his actually responsible to coordinate all concentrate shipping activities for Agnico-Eagle Mines zinc concentrate to a consumer located in Valleyfield, Qc. This contractor owns two railway sidings and a concentrate storage facility; all located in the town of Cadillac, please refer to Figure 21.2. This contractor has demonstrated an interest to provide Glen Eagle Resources with the same service that he is offering to Agnico-Eagle Mines. The costs estimation is a combination of J&R Dumas Inc. and SGS, and the total transport cost is shown in the following Table 21-7.

Table 21-7: Cost of Concentrate Transportation to USA

Activities	Cost	Unit
Truck loading at mine site	1.92	$/t conc.
Trucking transport cost (20 km) Truck unloading at destination	2.00	$/t conc.
Concentrate loading into train wagons G&A costs of loading activities Operators salary	8.50	$/t conc.
Train transport to USA market	55.00	$/t conc.
Total	**67.42**	$/t conc.

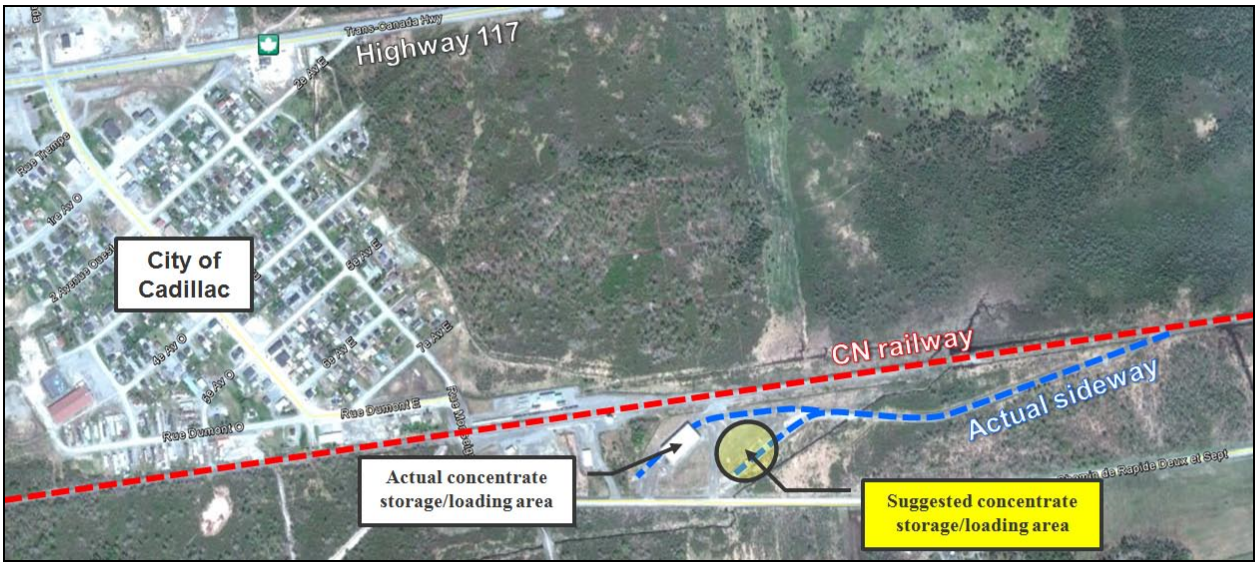

Figure 21-2: Proposed Local Railway Storage & Loading Concentrate Facilities

22 Economic Analysis

22.1 Principal assumptions

SGS made a numbers of assumptions in order to develop the Authier Project financial model:

- price of 6.0 % Li_2O spodumene concentre at $525 per tonne;
- processing rate of 2,200 tonnes per day (777,000 tonnes per year)
- constant exchange rate of $1.00 (US$:CDN$);
- discount rate of 6.00 %;
- economical analysis is presented as pre-finance and pre-tax;
- sunk costs and owner's costs are not included in the model;
- 1 year construction period (infrastructures and site preparation);
- 10 years of mining operation;
- initial capital cost will be totally spend during the first year of construction;
- initial capital costs are financed by equity;
- open-pit will be carried out by Glen Eagle and a specialized mining contractor;
- specialized mining contractor will provide his own mining equipments;
- mining and processing expenses will commence in the first year of operation.

22.2 Cash flow forecasts

A summary of the base case results is given in Table 22-1 while the detailed cash flow statement related to the base case scenario is presented by Table 22-2.

Table 22-1: Results of Base Case Cash Flow

Item	Value (CAD$ M)
Total Revenues	528.3
Total Operating Costs	364.9
Pre-production Capital Costs	42.1
Sustaining Capital Costs	16.2
Royalty paid	19.8
Salvage value + Working capital recovery	8.8
Undiscounted benefits	94.1
Discounted benefits (at 6.00 %)	53.3

Table 22-2: Base Case Cash Flow

Year		0	1	2	3	4	5	6	7	8	9	10	Total
Ore treated	tonnes		777,000	777,000	777,000	777,000	777,000	777,000	777,000	777,000	777,000	777,000	7,770,000
Input grade	%Li2O		0.99	0.90	0.85	0.84	0.93	0.84	0.81	1.01	0.99	0.99	0.91
Metal recovery	%		85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00
Concentrate	tonnes		108,446	98,847	93,322	92,386	102,084	92,540	89,557	111,616	108,919	108,600	1,006,317
Concentrate value	$/t. conc.		525	525	525	525	525	525	525	525	525	525	525
Revenues	$		56,934,092	51,894,859	48,993,832	48,502,622	53,593,866	48,583,528	47,017,436	58,598,426	57,182,587	57,014,997	528,316,245
Waste rock mined	tonnes	200,000	526,393	827,841	2,351,889	9,009,301	8,457,829	5,504,066	7,162,696	3,485,843	1,312,947	437,897	39,276,702
% of waste rock mined by the operator		100.0	100.0	100.0	100.0	40.0	42.6	65.5	50.3	100.0	100.0	100.0	60.0%
% of waste rock mined by a contractor	tonnes	-	-	-	-	60.0	57.4	34.5	49.7	-	-	-	40.0%
Ovb mined	tonnes	200,000	405,911	1,089,556	1,292,710	1,982,569	-	-	-	-	-	-	4,970,746
Total mined (ore + waste rock + ovb)	tonnes	400,000	1,709,304	2,694,397	4,421,599	11,768,870	9,234,829	6,281,066	7,939,696	4,262,843	2,089,947	1,214,897	52,017,448
Stripping ratio	t. waste / t. ore		1.20	2.47	4.69	14.15	10.89	7.08	9.22	4.49	1.69	0.56	5.69
Hard rock mining by Owner	(3.20$/t rock) $	640,800	4,176,071	5,141,911	10,024,960	14,035,828	14,033,633	14,040,451	14,032,995	13,658,149	6,696,190	3,892,530	100,373,518
Hard rock mining by Sub-Contractor	(4.32 $/t rock) $	-	-	-	-	23,352,108	20,972,709	8,203,260	15,378,595	-	-	-	67,906,672
Ovb mining	(2.50 $/t ovb) $	500,000	1,014,778	2,723,890	3,231,775	4,956,423	-	-	-	-	-	-	12,426,865
G&A	(5.02 $/t ore) $		3,900,540	3,900,540	3,900,540	3,900,540	3,900,540	3,900,540	3,900,540	3,900,540	3,900,540	3,900,540	39,005,400
Truck concencatre transport	(9.72 $/t conc.) $		527,047	480,398	453,543	448,996	496,126	449,745	435,247	542,454	529,347	527,796	4,890,699
Railway concentrate transport	(67.42 $/t conc.) $		3,655,711	3,332,144	3,145,871	3,114,330	3,441,237	3,119,525	3,018,967	3,762,577	3,671,667	3,660,906	33,922,935
Processing	(13.83 $/t ore) $		10,745,910	10,745,910	10,745,910	10,745,910	10,745,910	10,745,910	10,745,910	10,745,910	10,745,910	10,745,910	107,459,100
Royalties	$		4,666,556	3,862,282	2,302,926	- 834,398	482,078	1,006,564	471,131	2,058,817	2,479,984	3,346,960	19,842,900
Capital expenditures	$	41,000,000	-	-	-	-	-	-	-	-	-	-	41,000,000
Sustaining Capital	$		2,476,225	2,526,225	2,175,225	2,584,200	1,974,725	1,057,975	657,975	549,500	-	2,600,000	16,222,550
Salvage value	$	-	-	-	-	-	-	-	-	-	-	5,000,000	5,000,000
Working capital recovery	$	-	-	-	-	-	-	-	-	-	-	3,815,000	**3,815,000**
Pre-tax benefits	**$**	**- 42,140,800**	**25,771,255**	**19,181,558**	**13,013,083**	**- 13,801,314**	**- 2,453,091**	**6,059,557**	**- 1,623,925**	**23,380,479**	**29,158,948**	**37,155,355**	**94,080,606**
End-of-year cumulative pre-tax benefits	**$**	**- 42,140,800**	**- 16,369,545**	**2,812,013**	**15,825,096**	**2,023,781**	**- 429,310**	**5,630,248**	**4,006,323**	**27,386,802**	**56,545,751**	**93,701,106**	

22.3 Net present value, internal rate of return and payback period

The financial analysis results of the Authier Project for the base case scenario are calculated as:

- 53.3 M$ net present value (NPV) at 6.00% discount rate;
- 28.7 % internal rate of return (IRR);
- 1.9 years payback (from start of production) on 42.1 M$ initial capital.

All amounts are stated in Canadian dollars (unless otherwise noted).

22.4 Taxes, royalties and interests

Taxes

This preliminary economic assessment of the Authier Project was done without taking into consideration any governments taxes. However, Glen Eagle Resources will be subject to current and planned federal and Quebec tax rates and related tax rules. The applicable tax rates for 2013 are:

Federal corporate tax rate:	15.0 %
Provincial corporate tax rate:	11.9 %, However, Quebec levies mining taxes under the Mining Tax Act at a flat rate of 16.0 %.

A mining corporation in Quebec will be subject to mining taxes on the annual profit earned on its property that is reasonably attributable to the mine and that can be reasonable be attributable to the operations of the mine. For the purpose of the Mining Tax Act, annual profit is determined by subtracting from gross revenue the operating expenses and allowances directly related to the mine, including:

- exploration and development expenses;
- depreciation;
- a processing allowance;
- an additional allowance for a mine located in the North or mid North (not applicable).

Royalties

As is it required in a NI 43-101 PEA study, royalties has been considered into the economical analysis. SGS used the royalties and obligations defined into section 4.3. However, SGS made the assumption that Glen Eagle will purchase 1% NSR of claims CDC2183454, 2183455 and 2194819 for the amount of 1.0 M$. This assumption is based on a signed agreement between Glen Eagle and a third party owning a NSR royalties of those claims. The resulting royalties for the entire deposit are therefore presented in the next Figure.

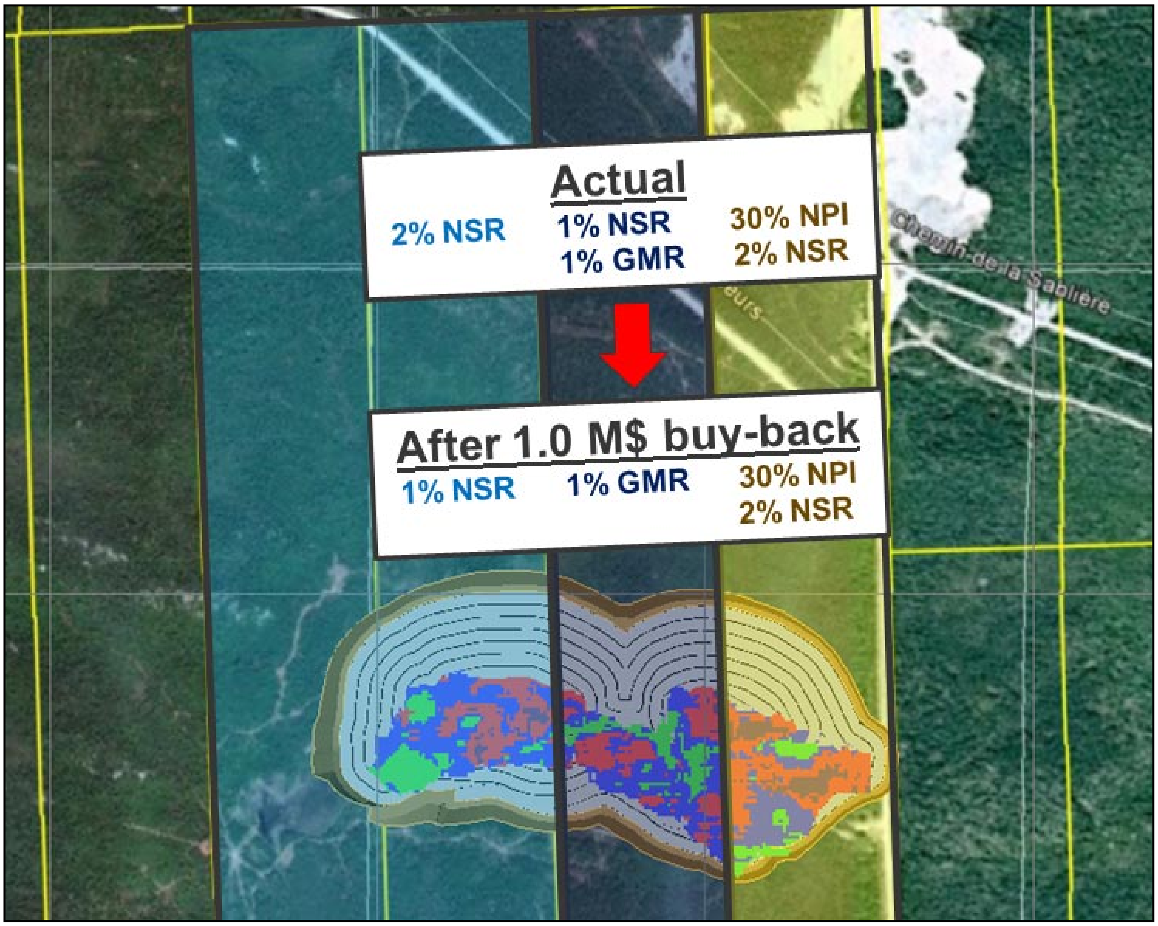

Figure 22-1: Royalties on the Authier Property

Interests

All the economical analysis presented in this study are calculated as pre-financed, so no interests attributable to capital financing was considered.

22.5 Sensitivity analysis

22.5.1 Sensitivity on base case scenario

The sensitivity of the pre-tax Net Present Value was evaluated for changes in key driven variables and parameters such as:

- Capital investment (CAPEX)
- Processing recovery
- Spodumene concentrate value
- Open pit mining cost (OPEX)
- Processing cost (OPEX)

Table 22-3: Sensitivity to Capex, Recovery, Concentrate Value and Opex

Variation		-30%	-20%	-10%	0%	10%	20%	30%
CAPEX	*$*	31,500,000	36,000,000	40,500,000	45,000,000	49,500,000	54,000,000	58,500,000
NPV	*$*	65,700,000	61,600,000	57,500,000	53,300,000	49,300,000	45,200,000	41,100,000
IRR	*%*	48.6	39.8	33.5	28.7	25.0	21.9	19.4
Processing recovery	*%*	59.5	68.0	76.5	85.0	93.5		
NPV	*$*	- 54,300,000	- 18,400,000	17,500,000	53,300,000	89,300,000		
IRR	*%*			13.1	28.7	44.7		
Spodumene value	*$/t conc.*	367.5	420.0	472.5	525.0	577.5	630.0	682.5
NPV	*$*	- 61,000,000	- 23,200,000	15,400,000	53,300,000	91,900,000	129,700,000	168,300,000
IRR	*%*			12.3	28.7	45.9	62.2	78.3
Mining cost	*$/t mined*	2.24	2.56	2.88	3.20	3.52	3.84	4.16
NPV	*$*	75,300,000	68,000,000	60,700,000	53,300,000	46,100,000	38,800,000	31,500,000
IRR	*%*	37.3	34.5	31.7	28.7	25.7	22.6	19.5
Processing cost	*$/t treated*	9.68	11.06	12.45	13.83	15.21	16.60	17.98
NPV	*$*	77,100,000	69,200,000	61,300,000	53,300,000	45,500,000	37,500,000	29,700,000
IRR	*%*	39.3	35.8	32.2	28.7	25.2	21.7	18.3

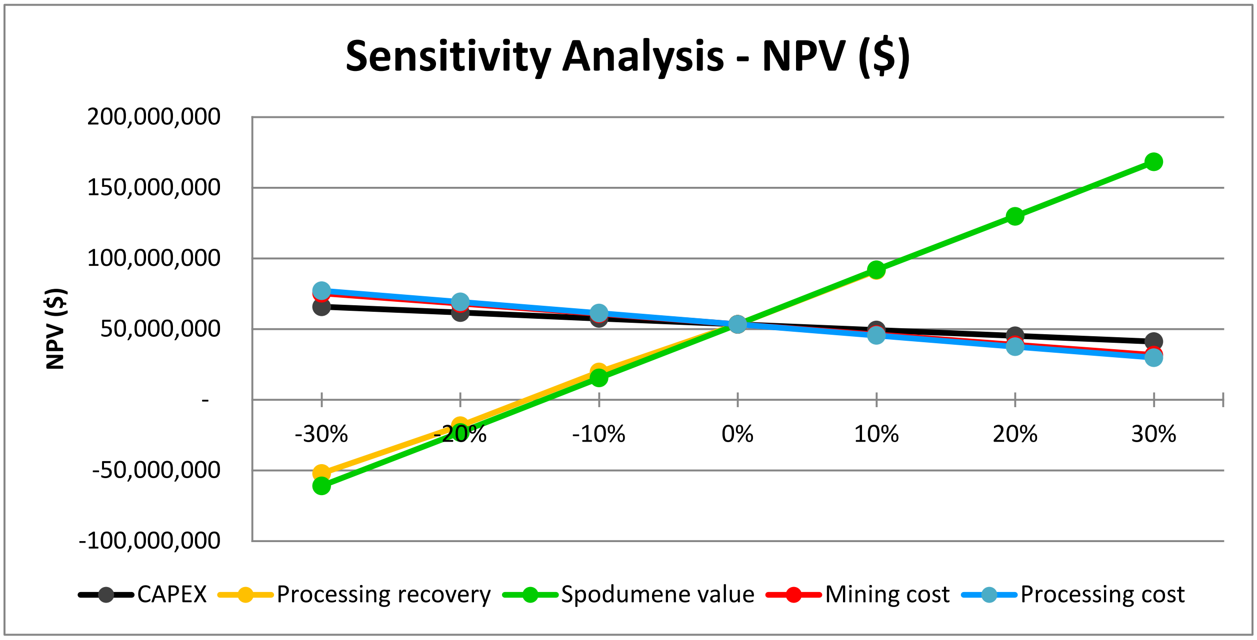

Figure 22-2: Sensitivity Graph of NPV's Results

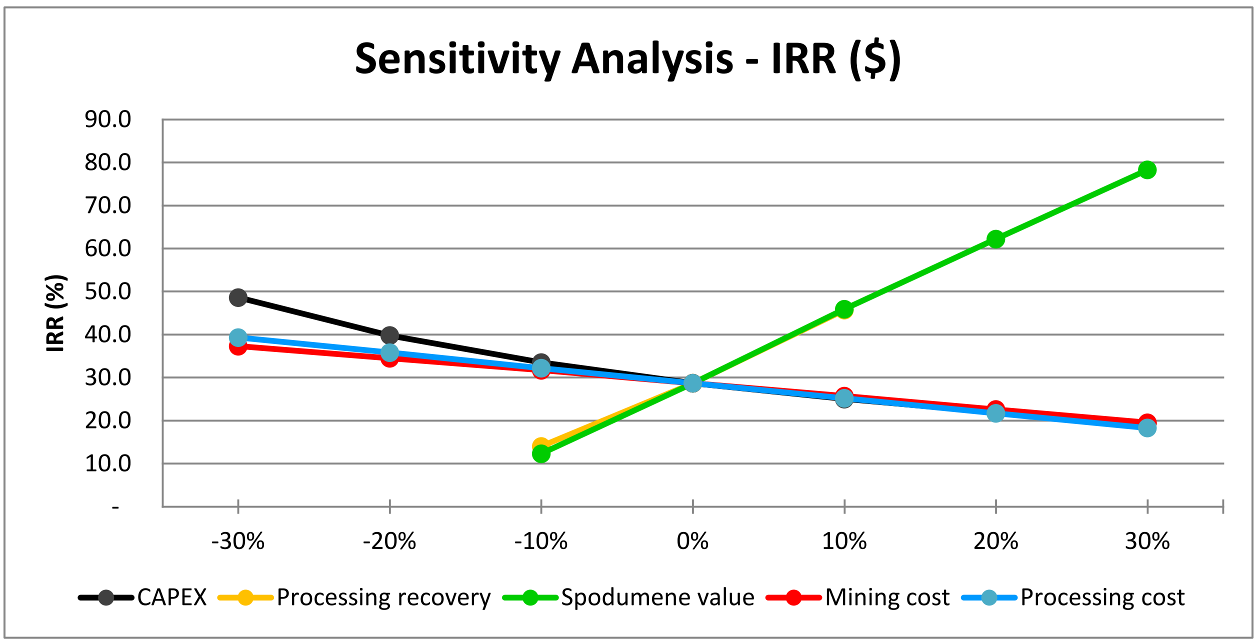

Figure 22-3: Sensitivity Graph of IRR's Results

The sensitivity analysis clearly demonstrates that the project is mainly sensitive to the processing recovery and the spodumene concentrate value. A drop of 14-15% of processing recovery or spodumene value makes the project economically marginal. On the other hand, a raise of these parameters strongly improve the Project economic. Others parameters, such as CAPEX, mining and processing cost do not affect Project economic as much as the ones discussed previously. For example, for an augmentation of 30% of CAPEX, or mining cost, or processing cost, the Project IRR stays over 18.3 %.

22.5.2 Project sensitivity vs. spodumene concentrate destination

The base case scenario supposed that the concentrate production will be sold to the local market (within Abitibi) and to the US market (50% - 50%). To assess the project sensitivity to the potential market, two scenarios were analyses were 100% of the production will be sold to each market respectively. The results are shown in the following Table.

Table 22-4: Concentrate Value vs Market Scenarios

Market	Production sold 100% locally	Production sold 100% to US
NPV (M$)	74.7	32.1
IRR (%)	38.2	19.4
Payback (Years)	1.6	2.3

23 Adjacent Properties

The area surrounding the Property, which is located between Val d'Or, Amos and Malartic, is well known for the mineral exploration activity, especially for gold, copper and zinc. Several exploration properties owned by different companies are surrounding the Authier property.

The most relevant mineral property located 27 km east the Authier project is the Quebec Lithium property owned by the company Canada Lithium Corporation ("Canada Lithium"). The Quebec Lithium property hosts a lithium deposit occurring in a series of spodumene-bearing pegmatite dykes which share strong similarities with the mineralised pegmatite intrusion observed at the Authier property. The pegmatite intrusions at the Quebec Lithium property are strongly dipping and oriented approximately N310° azimuth. The reported thicknesses of the dykes range from less than a metre to more than 45 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 are 33,239,000 tonnes 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 mineral reserve estimate, together with a mineral reserve estimate of a prior Feasibility Study, are amounting to 17,064,000 tonne at 1.17% Li_2O in the proven and probable categories and based on a cut-off grade of 0.6% Li_2O.

Construction on the mine and process plant began in August 2011. Construction is almost completed and commissioning is anticipated to begin late in 2012, while first production of lithium carbonate should occur late in the first quarter of 2013. The planned production rate is 20,000 tonnes of battery-grade lithium carbonate per annum.

The technical data provided in this section was taken from the Canada Lithium Web Site 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 Glen Eagle's Authier Lithium property that is the subject of this Technical Report.

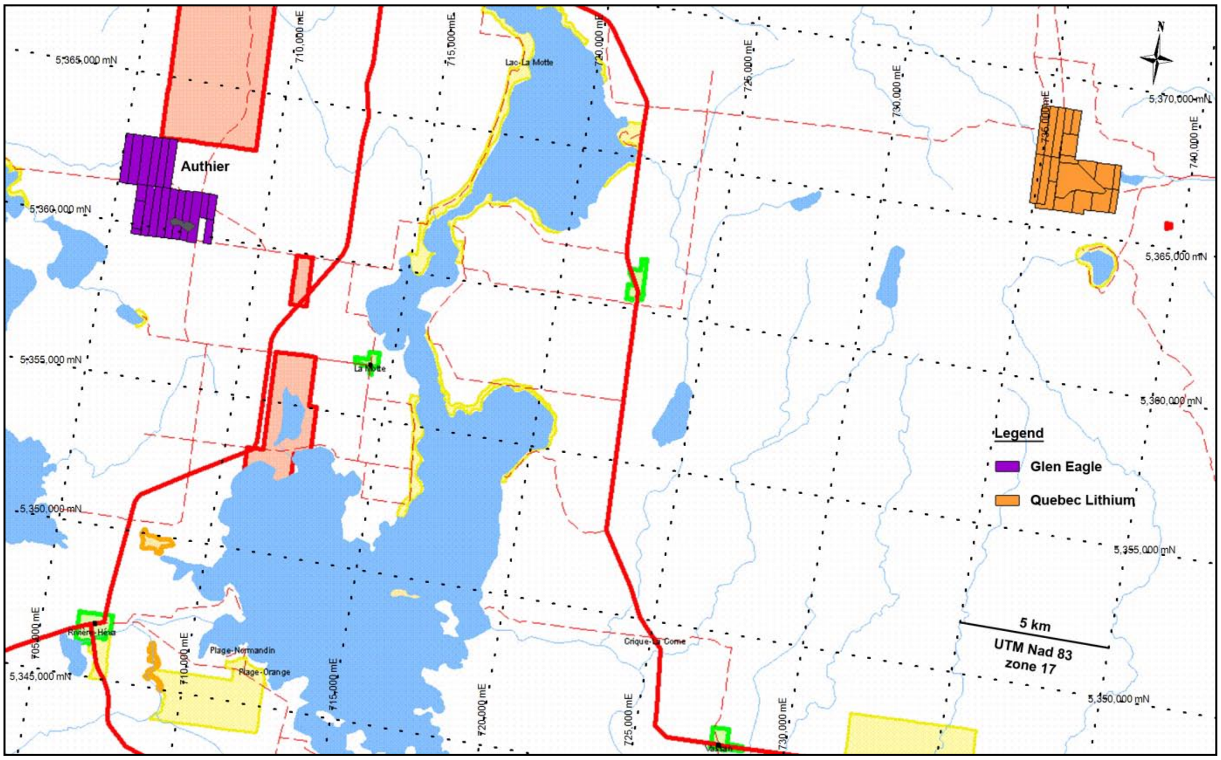

Figure 23-1: Adjacent Properties Map

24 Other Relevant Data and Information

To the author's knowledge, there is no other relevant data and information.

25 Interpretation and Conclusions

Geological

The MRE reported in this document is compliant with standards as outlined in the NI 43-101 regulations. The updated Authier in-pit MRE using a base case cut-off grade of 0.5% Li_2O totals 2,239,000 tonnes grading 0.95% Li_2O in the measured category, 5,148,000 tonnes grading 0.98 % Li_2O in the indicated category with an additional 572,000 tonnes grading 0.98% Li_2O in the inferred resources category.

The 2010 and 2012 independent sampling programs done by SGS Geostat revealed that the average grade of original assays for Li_2O may vary in the order of 10-15% from one lab to another. Additionally, a bias at a 95% confidence level is observed. SGS Geostat recommends continuing sending duplicate samples to two different laboratories in order to determine the percentage difference between labs. SGS Geostat recommends also continuing the QA/QC protocols.

The Authier Deposit contains enough resources to justify the launch of a preliminary economic assessment study. The company recently mandated SGS Geostat to conduct a PEA on behalf of Glen Eagle. The study is currently underway.

Upon a positive outcome of the PEA study, SGS Geostat would recommend additional drilling to increase the quality of the geological information at depth as well as the overall mineral resources and the geological model. With additional drilling and the revision of the geological model, an updated MRE would increase mostly the Measured and Indicated resources categories. The overall resources will most likely not augment drastically the MRE but rather upgrade the category after validation.

Economical

The parameters retained in this PEA are including the development of an open-pit mine using standard equipments, at a rate of 2,200 tonnes per day to be process by an on-site concentrator having an estimated recovery of 85%. The final product will be a spodumene concentrate at 6.0% Li_2O. The base case is assuming that 50% of the concentrate will be sold to local market and the rest transported by railway either to the USDA market or to oversea. No contract or purchasing agreements for the concentrate have been negotiated, or signed, at the date of this report. The cash flow analysis prepared from a concentrate price of $525/t, is showing that the Authier Project contains economic Mineral Resource.

26 Recommendations

Geological

SGS Geostat recommends the continuation of detailed mapping-sampling and prospecting of known and potential mineralized oxides occurrences throughout the property.

Following a positive outcome of the PEA study, The Authier Lithium property will need additional definition diamond drilling. This can be realized from the surface. The additional drilling will most likely not augment drastically the MRE but rather upgrade the category after validation.

SGS Geostat recommends carrying out down hole survey measurements (dip, direction of surveyed holes) on all 2010-2012 drill holes and the implementation of core orientation surveys on selected future drill targets.

A small analytical bias encountered on the 2012 SGS Geostat check sampling program, SGS Geostat recommends continuing sending core duplicates to a second certified laboratory in order to establish if there is a significant error from one laboratory to another. This suggestion is in line with the finding of the Data Verification section. SGS Geostat is of the opinion that certain reference materials (standards) have been mislabelled and that the Company must do adequate follow-up on these analyses accompanied with a detailed QAQC report. SGS Geostat has received confirmation that a follow-up is currently being done by Glen Eagle.

Additionally, SGS Geostat recommends sending its homemade reference materials to a certification laboratory in order to have a certified expected value and standard deviation.

SGS Geostat recommends additional infill drilling (Phase 1) to increase the quality of the geological information at depth as well as the overall mineral resources and the geological model. With additional drilling and the revision of the geological model, an updated MRE would increase mostly the Measured and Indicated resources categories. The overall resources will most likely not augment drastically the MRE but rather upgrade the category after validation. The infill drilling of the Authier Lithium deposit is estimated at a cost of $1,000,000.

The Company has available geophysical data, geochemical data and geological surface interpretations of the Authier Lithium property. SGS Geostat recommends (Phase 2) additional exploration, mapping and sampling of its prime exploration targets to increase the geological information and resources on the Property. The additional exploration cost is estimated at a cost of $100,000.

Following the positive outcome of the exploration campaigns, SGS Geostat recommends additional drilling (Phase 3) of its prime exploration targets to increase the quality of the geological information and the possibility for added resources on the Property. The exploration drilling of the Authier Lithium property is estimated at a cost of $500,000.

The following budget recommendation (Table 26-1) is based on a three (3) phase's program. It is purely conceptual and does not include accommodations, meals, transportation and equipment rental costs.

Table 26-1: Recommended Program and Budget for the Authier Lithium Property

Description	Units (m)	$/Unit	Price
Phase 1			
Systematic infill drilling (NQ) on drilled sections 50m radius of deeper and less defined (classified) areas	5000	200	1,000,000
Phase 2			
Detailed mapping and outcrop sampling of additional prime targets			100,000
Phase 3			
Drilling (NQ) of prime exploration targets, extentions drilling and infill drilling	2500	200	500,000
Tecnical services			
Core Logging and Field supervision			275,000
Advanced Reports			75,000
Assays(20%)			300,000
Contingencies (10%)			292,500
		Total	**2,542,500**

Mining

Complete a geotechnical study to optimize the open-pit slopes.
Assess the hydrogeological properties of the open-pit area.

Processing

The metallurgical spodumene concentrates recovery; expressed in lithium oxide Li_2O, obtained during the laboratory tests by SGS Minerals Services in Lakefield, Ontario, was good, more than 85%. But, the head grade of the sample was quite high, 1.3% Li_2O, which is above the estimated mill feed grade of the base case at 0.91% Li2O. Therefore, for future studies, we recommend doing some additional laboratory tests and pilot plant tests involving a more representative mill feed grade, and his effect on the overall recovery. These additional tests should include the followings:

- testing the floatability of the lower grade samples;
- making a flotation lock cycle test;
- testing the efficiencies of more reagents and water from plant location;
- rheological tests consisting of slurry thickening and filtration.

The estimate cost of these lab tests is ± $100,000.

In addition it will be worthy to make a pilot plant flotation campaign. The campaign will require at least 50 tonnes of resources. We estimate the cost of that pilot plant campaign at ± $500,000.

Economical

As mentioned in this study, there was no market study available to prepare this economic assessment. All commercial and marketing values and data used in this study were obtained from public information's and/or from ongoing projects results disclosures.

We therefore recommend a complete marketing study dedicated to the final product of the Authier Project.

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Certificates of Qualified Persons

Maxime Dupéré, P. Geo., SGS Geostat
Jonathan Gagné, Eng., SGS Geostat
Gaston Gagnon, Eng., SGS Geostat
Florent Baril, Eng., Bumigeme

CERTIFICATE OF QUALIFICATION

MAXIME DUPÉRÉ
maxime.dupere@sgs.com

I, Maxime Dupéré, P. Geo., Quebec, do hereby certify that:

a) I am a geologist with SGS Canada Inc, Geostat, with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville Quebec Canada, J7C 3V5.

b) This certificate applies to the technical report entitled NI 43-101 Preliminary Economic Assessment (PEA) on the Authier Project, Qc, for Glen Eagle Resources Inc., with the effective date of January 22, 2013 (the "Technical Report").

c) I am a graduate from the Université de Montréal, Quebec in 1999 with a B.Sc. in geology and I have practiced my profession continuously since 2001.I am a member in good standing of the Ordre des Géologues du Québec (#501), I have 11 years of experience in mining exploration in diamonds, gold, silver, base metals, and Iron Ore. I have prepared and made several mineral resource estimations for different exploration projects at different stages of exploration. I am aware of the different methods of estimation and the geostatistics applied to metallic, non-metallic and industrial mineral projects. I am a qualified person for the purposes of the National Instrument 43-101 (the "Instrument").

d) I am responsible for the Items 1 to 12, 14, 15 and 23 to 29, of the Technical Report.

e) I visited the property site on July 30, 2012.

f) I am independent of Glen Eagle Resources Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject or the Technical Report.

h) I have read the Instrument and the sections of the Technical Report that I am responsible for which have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report, or parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 1st day of March 2013 at Blainville, Quebec, Canada.

"Original document signed and dated by Maxime Dupéré, P. Geo."
Maxime Dupéré, P. Geo.
Geologist
SGS Canada Inc. – Geostat

CERTIFICATE OF QUALIFICATION

JONATHAN GAGNÉ
jonathan.gagne@sgs.com

I, Jonathan Gagné, Eng., do hereby certify that:

a) I am an Engineer with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Qc, Can, J7C 3V5.

b) This certificate applies to the technical report entitled NI 43-101 Preliminary Economic Assessment (PEA) on the Authier Project, Qc, for Glen Eagle Resources Inc., with the effective date of January 22, 2013 (the "Technical Report")

c) I am a graduate of the École Polytechnique de Montréal (B.Sc. Mining Engineer, in 2007). I am a member of good standing, No. 146075, of the l'Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes working as a mine planning engineer for a gold mining company and working as a consulting engineer to evaluate the potential of various mining projects. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I have visited the Property on December 21, 2012.

e) I am responsible for the preparation of Items 16, 18, 19, 21 and 22 of this Technical Report.

f) I am independent of Glen Eagle Resources Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument, and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report, or parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 1st day of March 2013, at Blainville, Quebec.

Original document signed and dated
by Jonathan Gagné, Eng"
Jonathan Gagné, Eng.
Mining Engineer
SGS Canada Inc. - Geostat

CERTIFICATE OF QUALIFICATION

GASTON GAGNON

gaston.gagnon@sgs.com

I, Gaston Gagnon, Eng., of Saint-Eustache, Quebec, do hereby certify:

a) I am Senior Mining Engineer with SGS Canada Inc. - Geostat with an office at 10 Boul. de la Seigneurie Est, Suite 203, Blainville, Quebec, Canada, J7C 3V5.

b) This certificate applies to the technical report entitled NI 43-101 Preliminary Economic Assessment (PEA) on the Authier Project, Qc, for Glen Eagle Resources Inc., with the effective date of January 22, 2013 (the "Technical Report")

c) I am a graduate of the University of Laval in Quebec City (B.Sc. Mining Engineering, 1964). I am a member of good standing (#15918) of the l'Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). My relevant experience includes over 40 years of experience in mining minerals in underground and surface producers, processing mainly gold, silver, copper, zinc, aggregates and niobium. Experience also includes 5 years of consulting for several mining projects under development. EPCM experience covers scoping (now PEA) studies and prefeasibility studies, detailed economic estimation and construction management in Canada, Africa, Mexico, South America and Saudi Arabia. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I didn't visit the property.

e) I am responsible for Items 1, 2, 25 to 27, I collaborated to Items 16, 18, 19, 20, 21, 22, and I am responsible for the coordination of the Technical Report.

f) I am independent of Glen Eagle Resources Inc., as defined by Section 1.5 of the Instrument.

g) I have no prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument, and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

i) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report, or parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 1st day of March 2013, at Blainville, Quebec.

"Original document signed and dated by Gaston Gagnon, Eng."
Gaston Gagnon, Eng.
Senior Mining Engineer
SGS Canada Inc. - Geostat

CERTIFICATE OF QUALIFICATION

FLORENT BARIL
fbaril@bumigeme.com

I, Florent Baril, of Montreal, Qc, do hereby certify that:

j) I am a Senior Metallurgical Engineer and the Owner and President of Bumigeme Inc., located at 615, René Lévesque West, Room 750, Montreal, Quebec, H3B 1P5.

k) This certificate applies to the technical report entitled NI 43-101 Preliminary Economic Assessment (PEA) on the Authier Project, Qc, for Glen Eagle Resources Inc., with the effective date of January 22, 2013 (the "Technical Report")

l) I am a graduate of the University of Laval in Quebec City (B.Sc., Metallurgy, 1954). I am a member in good standing (#6972) of the l'Ordre des Ingénieurs du Québec (Order of Engineers of Quebec). I have practiced my profession for over 50 years. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

m) I have visited the property in 1998.

n) I am responsible for Items 13 and 17, and I have collaborated to the Items 21 and 26 of the Technical Report.

o) I am independent of Glen Eagle Resources Inc., as defined by Section 1.5 of the Instrument.

p) I have had prior involvement with the property that is the subject of the Technical Report. In 1999, I was involved in the metallurgical tests performed by COREM in Quebec City for the same property which at that time was known as Lithium LaMotte and was a joint venture of Raymor Resources and Soquem; I was also involved in a pre-feasibility study completed in 2000 for Soquem.

q) I have read the Instrument, and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

r) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report, or parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 1st day of March 2013, at Montreal, Quebec.

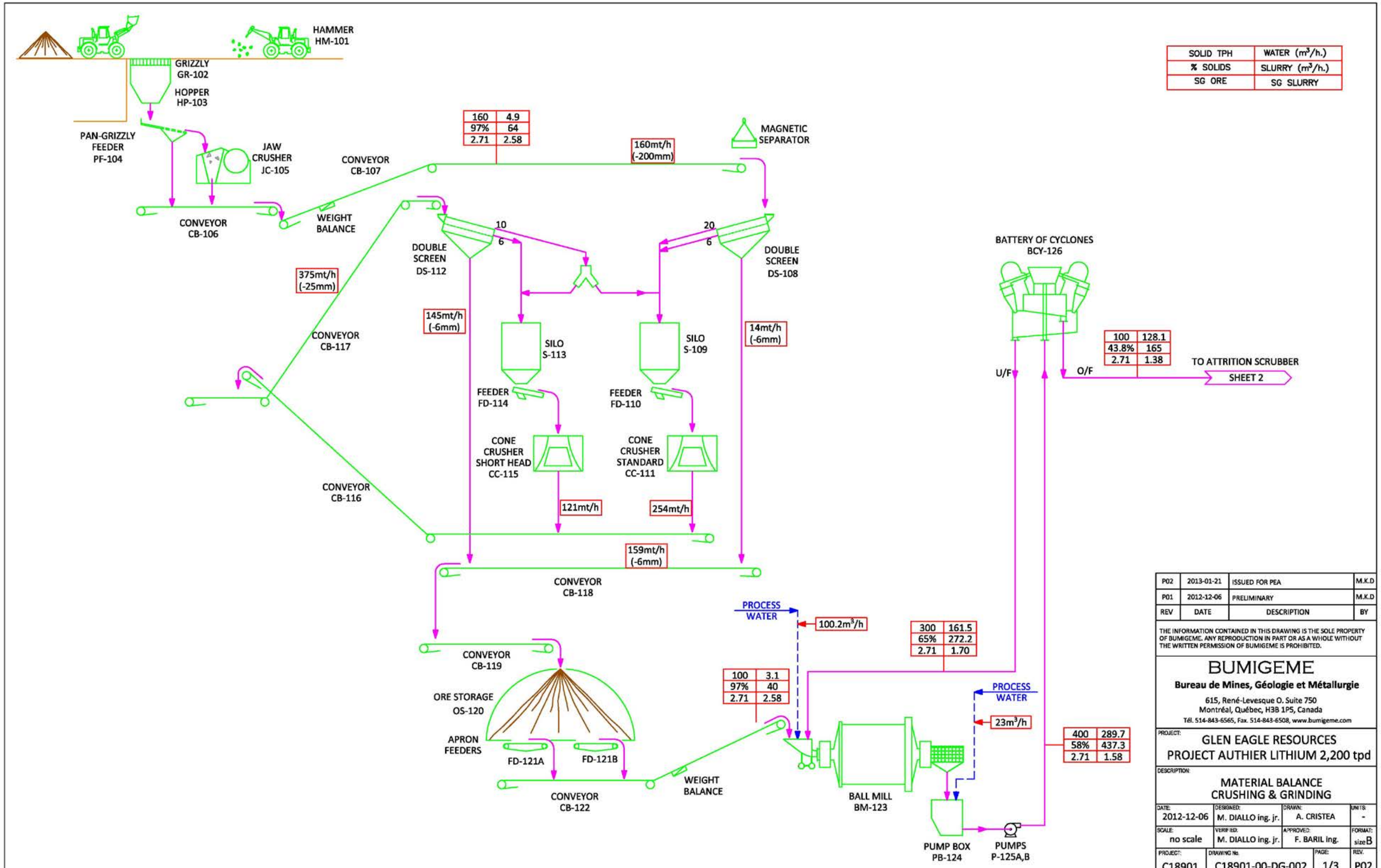
"Original document signed and dated by Florent Baril, Eng."
Florent Baril, Eng.
Senior Metallurgical Engineer
Bumigeme Inc., Montreal

Appendices

Appendix-1: Concentrator Material Balance, 3 pages

Appendix-2: Concentrator Main Equipment List, 2 pages

Appendix-3: SGS Metallurgical Tests Report, 57 pages


SOLID TPH | WATER (m³/h.)
% SOLIDS | SLURRY (m³/h.)
SG ORE | SG SLURRY
HAMMER HM-101
GRIZZLY GR-102
HOPPER HP-103
PAN-GRIZZLY FEEDER PF-104
JAW CRUSHER JC-105
CONVEYOR CB-106
CONVEYOR CB-107
WEIGHT BALANCE
160 4.9 97% 64 2.71 2.58
160mt/h (-200mm)
MAGNETIC SEPARATOR
DOUBLE SCREEN DS-112
DOUBLE SCREEN DS-108
10
6
20
6
375mt/h (-25mm)
145mt/h (-6mm)
14mt/h (-6mm)
SILO S-113
SILO S-109
FEEDER FD-114
FEEDER FD-110
CONVEYOR CB-117
CONE CRUSHER SHORT HEAD CC-115
CONE CRUSHER STANDARD CC-111
CONVEYOR CB-116
121mt/h
254mt/h
159mt/h (-6mm)
CONVEYOR CB-118
CONVEYOR CB-119
ORE STORAGE OS-120
APRON FEEDERS
FD-121A
FD-121B
CONVEYOR CB-122
WEIGHT BALANCE
PROCESS WATER
100.2m³/h
100 3.1 97% 40 2.71 2.58
BALL MILL BM-123
300 161.5 65% 272.2 2.71 1.70
PROCESS WATER
23m³/h
PUMP BOX PB-124
PUMPS P-125A,B
400 289.7 58% 437.3 2.71 1.58
BATTERY OF CYCLONES BCY-126
U/F
O/F
100 128.1 43.8% 165 2.71 1.38
TO ATTRITION SCRUBBER
SHEET 2
P02 2013-01-21 ISSUED FOR PEA M.K.D
P01 2012-12-06 PRELIMINARY M.K.D
REV DATE DESCRIPTION BY
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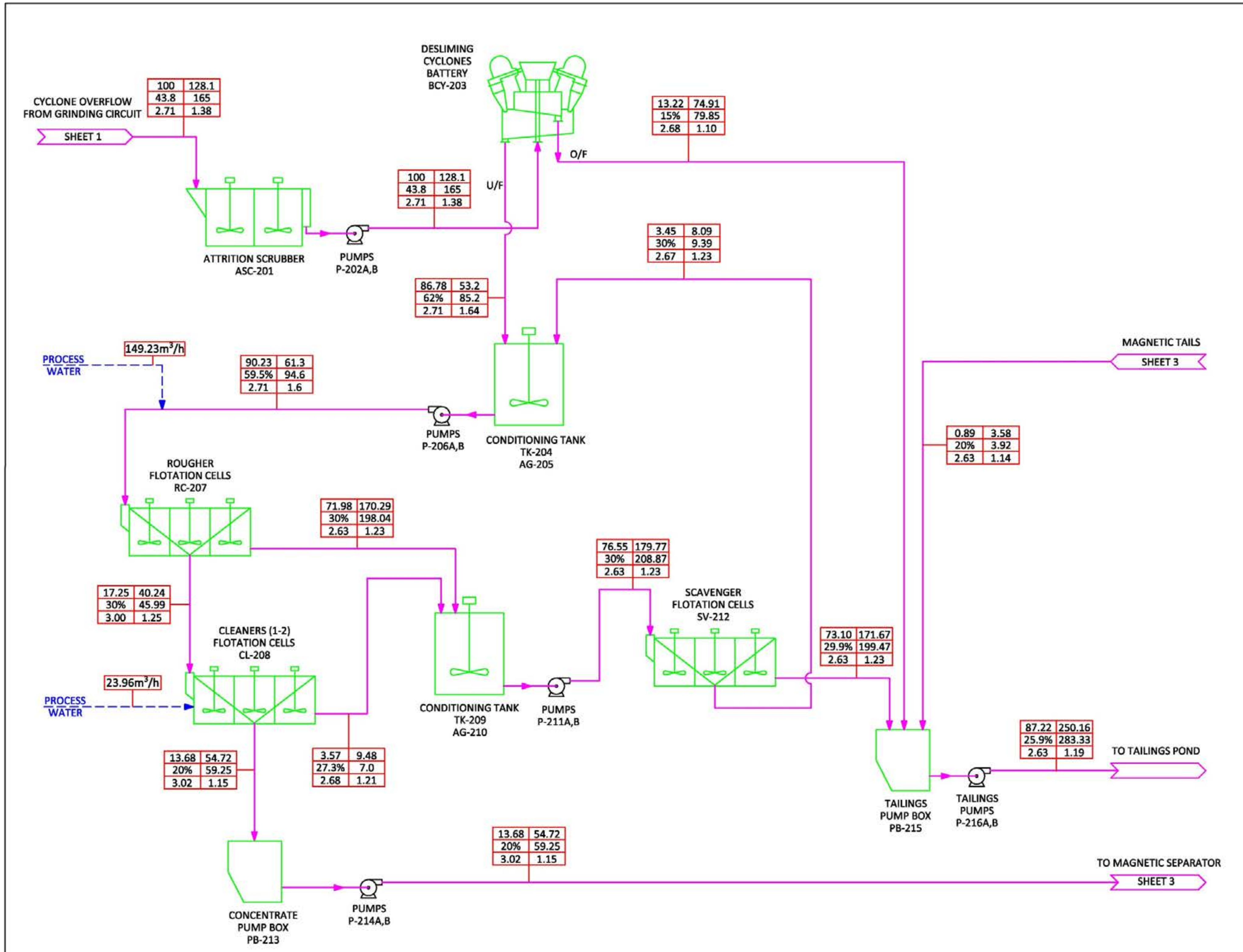
615, René-Levesque O. Suite 750
Montréal, Québec, H3B 1P5, Canada
Tél. 514-843-6565, Fax. 514-843-6508, www.bumigeme.com
PROJECT:
GLEN EAGLE RESOURCES
PROJECT AUTHIER LITHIUM 2,200 tpd
DESCRIPTION:
MATERIAL BALANCE
CRUSHING & GRINDING
DATE: 2012-12-06
DESIGNED: M. DIALLO ing. jr.
DRAWN: A. CRISTEA
UNITS: -
SCALE: no scale
VERIFIED: M. DIALLO ing. jr.
APPROVED: F. BARIL ing.
FORMAT: size B
PROJECT: C18901
DRAWING No. C18901-00-DG-002
PAGE: 1/3
REV. P02



REV	DATE	DESCRIPTION	BY
P02	2013-01-21	ISSUED FOR PEA	M.K.D
P01	2012-12-06	PRELIMINARY	M.K.D

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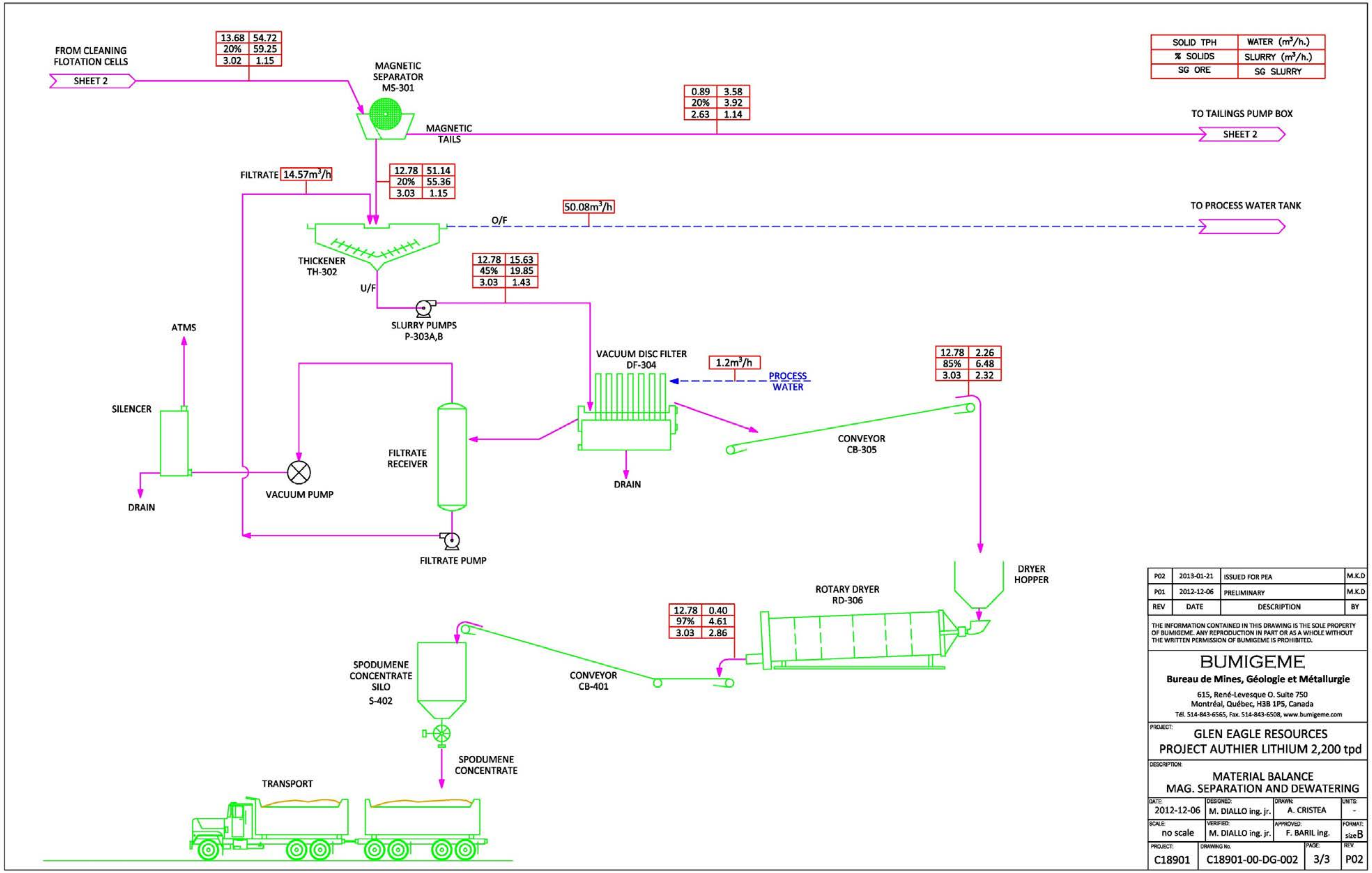
Montréal, Québec, H3B 1P5, Canada
Tél. 514-843-6565, Fax. 514-843-6508, www.bumigeme.com

PROJECT: GLEN EAGLE RESOURCES
PROJECT AUTHIER LITHIUM 2,200 tpd

DESCRIPTION: MATERIAL BALANCE
SCRUBBING AND FLOTATION CIRCUIT

DATE: 2012-12-06	DESIGNED: M. DIALLO ing. jr.	DRAWN: A. CRISTEA	UNITS: -
SCALE: no scale	VERIFIED: M. DIALLO ing. jr.	APPROVED: F. BARIL ing.	FORMAT: size B
PROJECT: C18901	DRAWING No. C18901-00-DG-002	PAGE: 2/3	REV: P02


FROM CLEANING FLOTATION CELLS
SHEET 2
13.68 | 54.72
20% | 59.25
3.02 | 1.15
MAGNETIC SEPARATOR MS-301
MAGNETIC TAILS
0.89 | 3.58
20% | 3.92
2.63 | 1.14
SOLID TPH | WATER (m³/h.)
% SOLIDS | SLURRY (m³/h.)
SG ORE | SG SLURRY
TO TAILINGS PUMP BOX
SHEET 2
FILTRATE 14.57m³/h
12.78 | 51.14
20% | 55.36
3.03 | 1.15
O/F
50.08m³/h
TO PROCESS WATER TANK
THICKENER TH-302
U/F
12.78 | 15.63
45% | 19.85
3.03 | 1.43
SLURRY PUMPS P-303A,B
ATMS
SILENCER
DRAIN
VACUUM PUMP
FILTRATE RECEIVER
FILTRATE PUMP
VACUUM DISC FILTER DF-304
1.2m³/h
PROCESS WATER
DRAIN
12.78 | 2.26
85% | 6.48
3.03 | 2.32
CONVEYOR CB-305
DRYER HOPPER
ROTARY DRYER RD-306
12.78 | 0.40
97% | 4.61
3.03 | 2.86
CONVEYOR CB-401
SPODUMENE CONCENTRATE SILO S-402
SPODUMENE CONCENTRATE
TRANSPORT
P02 | 2013-01-21 | ISSUED FOR PEA | M.K.D
P01 | 2012-12-06 | PRELIMINARY | M.K.D
REV | DATE | DESCRIPTION | BY
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615, René-Levesque O. Suite 750
Montréal, Québec, H3B 1P5, Canada
Tél. 514-843-6565, Fax. 514-843-6508, www.bumigeme.com
PROJECT:
GLEN EAGLE RESOURCES
PROJECT AUTHIER LITHIUM 2,200 tpd
DESCRIPTION:
MATERIAL BALANCE
MAG. SEPARATION AND DEWATERING
DATE: 2012-12-06
DESIGNED: M. DIALLO ing. jr.
DRAWN: A. CRISTEA
UNITS: -
SCALE: no scale
VERIFIED: M. DIALLO ing. jr.
APPROVED: F. BARIL ing.
FORMAT: size B
PROJECT: C18901
DRAWING No. C18901-00-DG-002
PAGE: 3/3
REV: P02

QTY	EQUIPMENT	PFD #	KW	DIMENSIONS
Reception, Crushing and Storage				
1	Hydraulic Hammer	**HM-101**		
1	Grizzly	**GR-102**		600mm x 600mm
1	Hopper	**HP-103**		50T
1	Pan Grizzly feeder	**PF-104**	18.5	
1	Jaw Crusher	**JC-105**	132	1100mm x 860mm
1	Belt conveyor #1	**CB-106**	11	
1	Belt conveyor #2	**CB-107**	11	
1	Double screen	**DS-108**	18.5	4500mm x 1800mm
1	Silo 8m³	**S-109**		8m³
1	Vibrating Feeder	**FD-110**	7.5	2000mm x 1000mm
1	Cone Crusher standard Auxiliary equipment	**CC-111**	200 50	
1	Double screen	**DS-112**	18.5	5000mm x 2000mm
1	Silo 8m³	**S-113**		8m³
1	Vibrating Feeder	**FD-114**	7.5	2000mm x 1000mm
1	Cone Crusher short head Auxiliary equipment	**CC-115**	200 50	
1	Belt conveyor #3	**CB-116**	11	
1	Belt conveyor #4	**CB-117**	9	
1	Belt conveyor #5	**CB-118**	9	
1	Belt conveyor #6	**CB-119**	9	
1	Storage Dome and tunnel	**OS-120**		10500 tm
2	Apron Feeder	**FD-121A,B**	22	
1	Crane 5t		5.5	

Grinding and Disliming				
1	Belt conveyor #7	**CB-122**	9	
1	Ball Mill with Trommel Auxiliary equipment	**BM-123**	1500 15	Ø3.8m x 7.5m
1	Pump Box	**PB-124**		40m³
2	Pumps	**P-125A or B**	110	
1	Battery of Cyclones	**BCY-126**		Battery of 3 cyclones-gMAX20
1	Attrition Scrubber (2)	**ASC-201**	2 x 150	
2	Pumps	**P-202A or B**	45	
1	Hydrocyclones-disliming	**BCY-203**		Battery of 3 cyclones-gMAX15

QTY	EQUIPMENT	PFD #	KW	DIMENSIONS
Spodumene Flotation and Magnetic Separation				
1	Conditioning Tank	**TK-204**	11	40m³
2	Pumps	**P-206A or B**	22	
1	Rougher Cells (5)	**RC-207**	5 x (11.2+0.75)	
1	Scavenger Flotation Cells (2)	**SV-212**	2 x (11.2+0.75)	
1	Cleaner Flotation Cells 1st & 2nd (3+3)	**CL-208**	(3+3) x (11.2+0.75)	
1	Conditioning Tank	**TK-209**	7.5	15m³
2	Pumps	**P-211A or B**	22	
1	Concentrate Pump Box	**PB-213**		10m³
2	Concentrate Pumps	**P-214A or B**	22	
1	Tailings Pump Box	**PB-215**		20m³
2	Tailings Pumps	**P-216A or B**	90	
1	Magnetic Separator	**MS-301**	11	

Decantation, Filtration, Drying and Storage				
1	Thickener	**TH-302**	0.75	
2	Slurry Pumps	**P-303A or B**	7.5	
1	Disc Filter Package	**DF-304**	10	
1	Conveyor belt	**CB-305**	7.5	
1	Rotary Dryer Package Auxiliary equipment	**RD-306**	22.4 1.5+7.5	Ø1.8m x 10.7m
1	Conveyor belt	**CB-401**	7.5	
1	Storage Silo Concentrate Spodumene	**S-402**		1000T
1	Blower		30	
2	Sump Pumps		15	

An Investigation into

THE RECOVERY OF SPODUMENE FROM THE AUTHIER LITHIUM DEPOSIT SAMPLE

prepared for

GLEN EAGLE RESOURCES INC.

Project 13650-001 Final Report
January 7, 2013

SGS Canada Inc. | P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0
Tel: (705) 652-2000 Fax: (705) 652-6365 www.met.sgs.com www.ca.sgs.com
Member of the SGS Group (SGS SA)

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Synthèse

Un échantillon du gisement Authier, situé environ 45km au nord de Val d'Or, Québec, a été soumis pour caractérisation minéralogique et métallurgique au laboratoire métallurgique de SGS Minerals Services de Lakefield. L'échantillon, reçu sous forme de demi-carottes, fut combiné au complet pour préparer un seul composite.

L'analyse minéralogique identifia le seul le spodumène comme le seul porteur de lithium, représentant 14.9% de l'échantillon. Les autres minéraux inclus dans l'échantillon sont, en ordre décroissant, l'albite (37.2%), le quartz (26.5%), le microcline (16.2%), la muscovite (4.8%), et la magnétite (0.3%).

Les indices de broyages de Bond sont qualifiés de mou pour le broyeur à barres (RWI) avec un indice de 12.3 kWh/t et moyen pour le broyeur à boulets (BWI) avec un indice de 15.6 kWh/t.

Des tests préliminaires de flottation, sans préflotation du mica (muscovite) et incluant deux étape de nettoyage par flottation et une étape de nettoyage par séparation magnétique à haute intensité (WHIMS), a produit un concentré de spodumène avec une teneur de 6.09% Li_2O, récupérant 88% du lithium.

Les points suivant découlent des tests de flottation complétés :

- L'acide gras saturé (FA-2) est un bon collecteur pour le spodumène. D'autres collecteurs qui peuvent aussi être considérés sont l'Aero 704 et l'Aero 845 de Cytec. Ceux-ci risquent de ne pas être aussi performants que le FA-2
- La cinétique de flottation est très rapide avec une récupération d'environ 60% de la masse dans la première minute de flottation.
- Un broyage par étape visant une granulométrie à 100% passant 150µm ou un K_{80} d'environ 120µm est nécessaire à l'obtention d'un concentré ayant la teneur désirée et une bonne récupération
- Le conditionement à haute intensité avec un lignosulphonate (D618) à pH 11 suivit de déschlamage est bénéfique au procédé
- L'utilisation de 2, ou d'un maximum de 3, étapes de nettoyage semble adéquat. La distribution de fer dans le concentré de lithium semble quand même élevé (1.25% Fe_2O_3) mais il est possible de soutirer tout fer non associé avec le spodumène à l'étape de séparation magnétique
- La préflotation du mica (muscovite) n'est pas nécessaire à l'obtention de la teneur désirée pour le concentré de spodumène.

Executive Summary

Drill core samples from the Authier property, located 45 km North West of Val d'Or, Quebec were submitted to SGS Minerals Services for metallurgical testwork by Glen Eagle Resources Inc. One composite sample was prepared from these drill cores. The mineralogical analysis of the composites samples is as follows:

Mineral	Authier Li Deposit Drill Core Sample (wt %)
Albite	37.2
Quartz	26.5
Microcline	16.2
Spodumene	14.9
Muscovite	4.8
Magnetite	0.3
TOTAL	99.9

In batch flotation tests and after passing the concentrate through WHIMS, a spodumene concentrate grading 6.09% Li_2O was generated at 88% (global) recovery after two cleaning stages (test F8) without mica pre-flotation. Flotation kinetics was fairly fast and within one minute a large portion of the spodumene mass, more than 60%, reported to the concentrate.

In batch cleaner testing, the following key points have been identified:

- FA-2 is a suitable spodumene collector. Alternatively, Aero 704 and 845 might be considered. The flotation performance with these collectors may not be as good as with FA-2 collector.
- Stage grinding to a closing size of 150 µm and a K_{80} around 120 µm was required to achieve Li concentrate with good grade and recovery.
- Scrubbing with lignin sulfonate at pH 11 followed by desliming was also beneficial.
- Implementing 2 or a maximum of 3 cleaning stages was a suitable flotation strategy. The iron content of the lithium concentrate was high (1.25% Fe_2O_3). It was only possible to remove a portion of the iron in the concentrate (not associated with spodumene as solid solution) using wet high intensity (15000 Gauss) magnetic separation.
- Mica pre-flotation was not required to achieve high grade concentrate

Introduction

A request was received by Mr. Stephane Baril of Bumigeme Inc. for a scoping test program on a sample from the Authier property. The objective was to develop a flowsheet to produce spodumene concentrate with a grade of 6.0 to 6.5% Li_2O, suitable for hydrometallurgy operation. In this report, details of the mineral beneficiation development testwork are discussed. In addition, recommendations for the development of a commercial process are provided.

During the development of the testwork, progress was discussed with Mr. Mamadou Diallo through emails and telephone calls.

Aghamirian

Massoud Aghamirian, Ph.D
Senior Metallurgist

Dan Imeson, MSc.
Manager, Mineral Processing

Experimental work by: Yanling_Sheng
Report preparation by: Massoud Aghamirian,
Reviewed by: S. McKenzie, D. Lascelles, D. Imeson

Testwork Summary

1. Sample Receipt, Description, Preparation and Characterization

Drill core samples were received in one shipment with a combined total weight of 270 kg. The receipt number assigned was 0043-AUG12. The inventory list of drill core samples is provided in Table 1. All of the drill cores were used to prepare the composite sample. The sample preparation procedure is shown in Figure 1. Two head samples were submitted for chemical analyses and the results are presented in Table 2. Figure 2 can be used to convert Li_2O grade to spodumene grade in samples. All development test work was conducted on this composite sample.

Table 1: Inventory of Drill Cores

Sample ID	Weight (KG)	Sample ID	Weight (KG)
J765656	3.38	J165446	3.68
J765664	3.45	J164127	3.64
J765666	3.36	J164126	3.2
J765658	3.36	J164125	3.53
J765659	3.36	J164129	3.13
J765660	3.36	J164128	3.29
J765661	3.05	J164118	3.58
J765662	3.36	J164119	3.15
J765665	3.36	J164120	3.15
J765663	3.33	J164123	3.26
J765667	2.22	J164121	3.57
J765657	3.16	J164122	3.57
J765655	3.17	J164124	3.27
J765654	3.27	J165449	3.86
J765652	3.31	J165451	4.14
J765649	2.92	J165452	3.53
J765653	3.06	J165450	4.07
J765651	3.26	J165453	3.71
J765650	3.4	J164130	3.39
J765646	3.28	J164132	3.19
J765648	3.58	J164133	3.29
J765645	3.39	J164131	3.14
J765644	3.26	J164134	3.52
J765647	3.13	J164114	3.1
J765642	3.45	J164115	3.18
J765643	3.11	J164116	3.54
J765641	3.33	J164117	3.52
J765637	3.68	J165460	3.52
J765636	3.42	J165459	3.8
J765634	3.22	J165461	3.7
J765635	3.5	J164111	3.39
J765638	3.45	J164112	2.75
J765639	3.45	J164113	3.47
J765640	3.45	J164110	3.32
J165442	3.56	J165456	3.78
J165443	3.86	J165454	3.76
J165444	3.86	J165455	3.61
J165447	3.74	J165458	3.49
J165448	3.73	J165457	3.75
J165445	3.68		

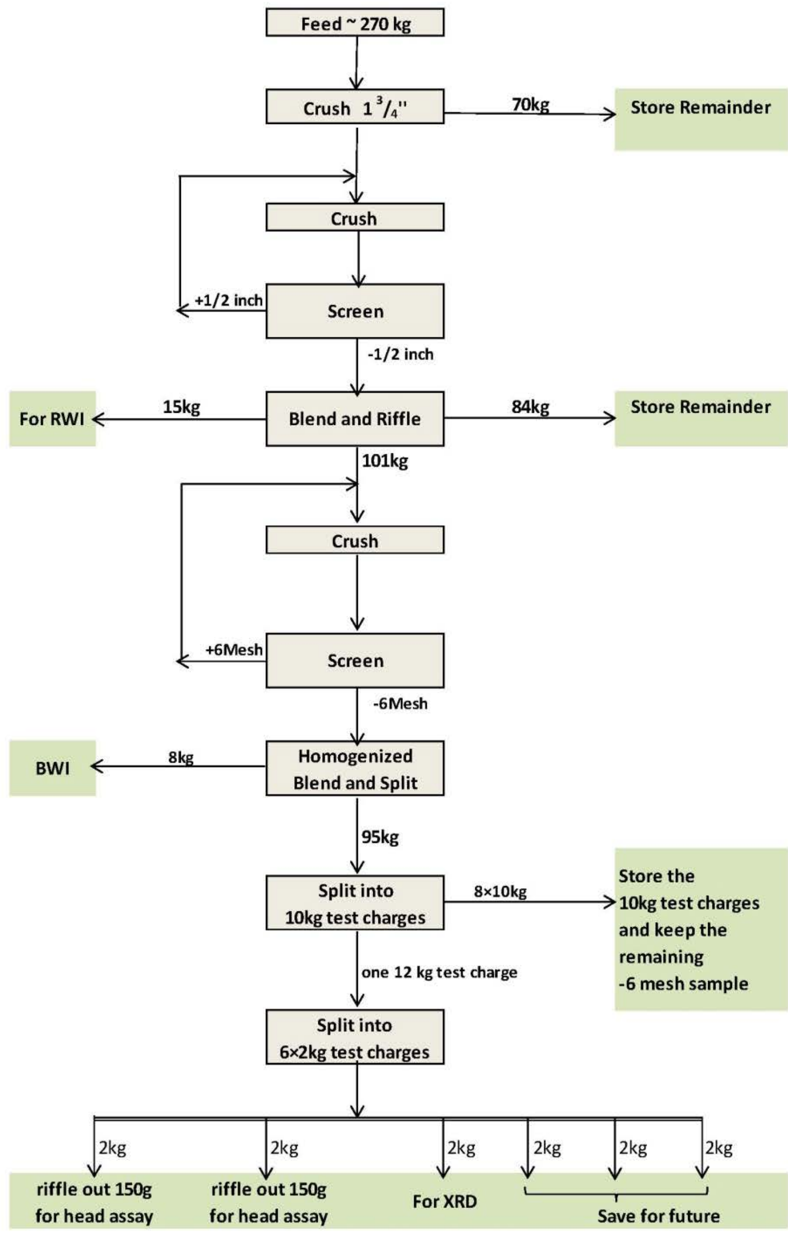

Figure 1: Sample Preparation Flowsheet

Table 2: Chemical Analysis of Representative Samples

Sample ID	Cut A	Cut B
Li	0.57	0.57
Li_2O	1.23	1.23
SiO_2	74.8	75
Al_2O_3	15.8	15.8
Fe_2O_3	0.58	0.59
MgO	0.07	0.07
CaO	0.17	0.18
Na_2O	4.27	4.16
K_2O	3.08	3.08
TiO_2	< 0.01	< 0.01
P_2O_5	0.02	0.02
MnO	0.1	0.1
Cr_2O_3	0.02	0.02
V_2O_5	< 0.01	< 0.01
LOI	0.43	0.36
Sum	99.3	99.5
S %	0.01	0.01

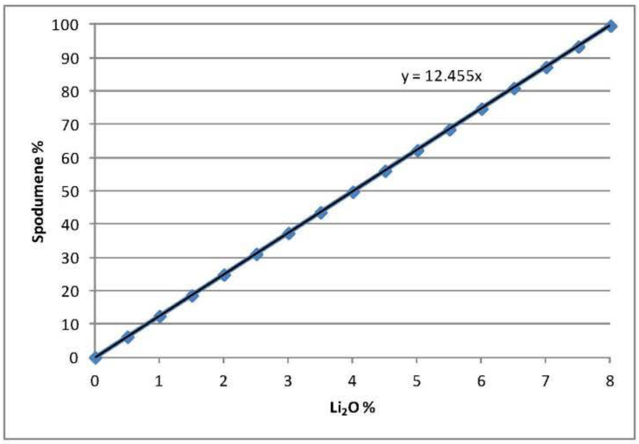

Figure 2: Conversion of Lithium Oxide Grade to Spodumene Grade

2. Mineralogical Analysis

A subsample from the master composite was submitted for XRD determination and the semi-quantitative head composition is shown in Table 3. Test details can be found in Appendix A.

Table 3: Head Sample Mineral Composition of the Head Composite

Mineral	Authier Li Deposit Drill Core Sample (wt %)
Albite	37.2
Quartz	26.5
Microcline	16.2
Spodumene	14.9
Muscovite	4.8
Magnetite	0.3
TOTAL	99.9

Roughly 14.9% of the head composite sample is composed of spodumene. The following conclusions can be made from XRD analysis:

- The sole Li mineral identified is spodumene. There was no petalite identified in the samples. If petalite is present, it would be in trace quantities.
- The sample consists of major amounts of albite (37.2%), quartz (26.5%), microcline (16.2%), spodumene (14.9%) and muscovite (4.8%).

3. Bond Mill Work Index

3.1. Bond Rod Mill Work Index

The master composite was submitted for Bond rod mill work index (RWI) determination as per the standard procedure using a product D_{100} size of -1180 µm. A Bond rod mill work index (RWI) of 12.3 kWh/t (metric) was obtained for the composite. Details of the test are included in Appendix B. The results are compared to the SGS database in Figure 3. The sample is characterized as soft in RWI terms with a percentile of 27%.

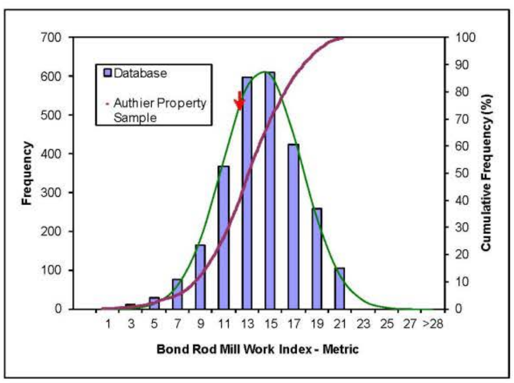

Figure 3: Head Composite Bond Rod Mill Work Index Comparison to SGS Database

(a) 2kg Grinding Mill

(b) Acrylic Cylinder for Scrubbing

(c) Denver Flotation Machine

Figure 5: Lab Equipment Used for Flotation Testwork

4.1. Stage Grinding

All stage grinding (100% passing 48 mesh or 65 mesh or 100 mesh) was performed on 2-kg charges in a 2kg rod mill as follows:

- The feed was first screened on the target screen with the undersize fraction set aside as feed for the next stage and the oversize fraction further ground;
- After each grinding stage, the product was screened on the target screen;
- The pulp density in each grinding stage was roughly adjusted to ~65% solids;
- With a closing size of 48 mesh, the grinding stages were performed for 15, 10 and 6 minutes. With a closing size of 65 mesh, the grinding stages were performed for 16, 11 and 6 minutes. With a closing size of 100 mesh, the grinding stages were performed for 18, 13 and 8 minutes.

4.2. Development of a Spodumene Flotation Scheme

The main variables investigated in the flotation testwork were spodumene collector type, flotation feed size, pre-mica flotation, and effect of cold temperature on conditioning and flotation. Desliming after scrubbing was performed in a 16L acrylic cylinder. For 2-kg charges, it was initially filled to 11L, allowed to settle for 5, 6 or 8 minutes, and decanted to 1" above the settled pulp line. The entire procedure, (filling the acrylic container with water, mixing for 30 seconds settling, and decanting to the desired level) was repeated twice.

A number of reagents were used in this testwork. The most important ones are as follows:

- Armac C was used as the collector for muscovite flotation;
- Fuel oil was used as the promoter for coarse mica flotation;
- FA-2, Aero 6493, Aero 704, and Aero 845 from Cytec were used as spodumene collectors.

The flotation test results are shown in Table 4 and Table 5. The flowsheet development results are discussed in the following sections.

Table 4: Summary of the Spodumene Flotation Test Results of the Head Composite

Objective	Product	Weight		Assays %							Distribution %					
		g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F1	Li Non Mag Final Conc. 15A	284	14.4	2.89	6.22	64.9	24.7	0.91	0.98	1.03	74.4	12.6	22.6	4.3	3.4	19.6
Applying Developed SGS Flowsheet on the Glen Eagle Composite Sample	Li Non Mag Final Conc. 5A	289	14.7	2.87	6.19	64.8	24.7	0.92	0.98	1.13	75.4	12.8	23.0	4.4	3.4	22.0
	Li 3rd Cl Conc	307	15.6	2.81	6.06	64.1	24.5	0.91	0.97	1.66	78.4	13.4	24.3	4.6	3.6	34.2
	Li 2nd Cl. Conc.	317	16.1	2.77	5.96	64.3	24.4	0.96	1.02	1.66	79.7	13.9	25.0	5.0	3.9	35.2
	Li 1st Cl Conc	343	17.4	2.66	5.73	64.8	24.0	1.07	1.18	1.61	82.9	15.2	26.6	6.1	4.9	37.0
	Li Ro Conc.	402	20.4	2.40	5.16	66.0	23.0	1.34	1.59	1.47	87.4	18.1	29.9	8.9	7.8	39.6
	Li Ro & Scav. Conc.	418	21.2	2.36	5.08	66.0	22.9	1.40	1.62	1.47	89.5	18.8	30.9	9.6	8.2	41.2
	F1 Li Ro Scav Tail	1,239	62.8	0.03	0.06	78.2	13.0	3.36	5.01	0.26	3.1	66.1	52.0	68.7	75.1	21.6
	F1 Mica Ro Con	224	11.3	0.21	0.45	70.7	16.8	4.32	4.27	1.38	4.3	10.8	12.1	15.9	11.6	20.7
	F1 Slime 2	21.6	1.1	0.59	1.27	65.0	17.7	3.27	4.13	4.53	1.2	1.0	1.2	1.2	1.1	6.6
	F1 Slime 1	71.3	3.6	0.30	0.65	69.4	16.1	3.88	4.68	2.07	1.9	3.4	3.7	4.6	4.0	9.9
	Head (calc.)	1,973	100	0.56	1.20	74.3	15.7	3.07	4.19	0.76	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,902	96.4	0.57	1.22	74.5	15.7	3.04	4.17	0.71	98.1	96.6	96.3	95.4	96.0	90.1
	Spod Flot Feed	1,656	84.0	0.62	1.33	75.1	15.5	2.87	4.16	0.57	92.6	84.9	82.9	78.3	83.3	62.8
F2	Li Non Mag Final Conc. 15A	122	6.2	3.26	7.02	64.2	25.8	0.41	0.48	1.25	36.6	5.4	10.3	0.9	0.7	10.7
Similar to Test F1 but Replacing FA-2 with Aero 6493	Li Non Mag Final Conc. 5A	125	6.3	3.24	6.97	64.0	25.7	0.41	0.48	1.41	37.1	5.5	10.5	0.9	0.7	12.3
	Li 3rd Cl Conc	134	6.8	3.10	6.67	62.8	25.5	0.40	0.47	2.45	38.0	5.8	11.1	0.9	0.8	23.0
	Li 2nd Cl. Conc.	151	7.7	3.09	6.65	62.9	25.4	0.43	0.50	2.41	42.8	6.6	12.5	1.1	0.9	25.5
	Li 1st Cl Conc	175	8.9	3.07	6.61	63.2	25.3	0.49	0.57	2.33	49.3	7.6	14.5	1.5	1.2	28.6
	Li Ro Conc.	218	11.1	2.90	6.24	64.0	24.7	0.70	0.84	2.13	58.2	9.7	17.6	2.6	2.2	32.5
	Li Ro & Scav. Conc.	231	11.7	2.84	6.11	64.2	24.5	0.77	0.95	2.10	60.3	10.3	18.5	3.0	2.7	34.0
	F2 Li Ro Scav Tail	1,496	75.9	0.24	0.52	75.7	13.8	3.15	4.66	0.34	33.0	78.3	67.5	80.0	84.7	35.6
	F2 Mica Con	147	7.4	0.23	0.50	69.0	17.5	4.51	4.25	1.32	3.1	7.0	8.4	11.2	7.6	13.6
	F2 Slime 2	38.6	2.0	0.31	0.67	70.1	16.8	3.82	4.70	1.78	1.1	1.9	2.1	2.5	2.2	4.8
	F2 Slime 1	57.3	2.9	0.48	1.03	64.6	18.6	3.37	4.16	3.00	2.5	2.6	3.5	3.3	2.9	12.0
	Head (calc.)	1,969	100	0.55	1.19	73.4	15.5	2.99	4.18	0.72	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,912	97.1	0.57	1.23	75.2	15.7	3.07	4.22	0.53	97.5	97.4	96.5	96.7	97.1	88.0
	Spod Flot Feed	1,727	87.7	0.61	1.31	75.7	15.5	15.00	4.20	0.46	93.3	88.6	86.0	83.0	87.3	69.6
F3	Li Non Mag Final Conc. 20A	269	13.7	2.91	6.26	64.4	25.4	0.96	0.80	1.10	70.6	12.0	22.3	4.4	2.6	23.0
Repeat Test F1 but Eleminating Mica Flotation	Li Non Mag Final Conc. 10A	273	14.0	2.89	6.22	64.2	25.4	0.97	0.80	1.24	71.3	12.1	22.6	4.5	2.7	26.3
	Li 3rd Cl Conc	285	14.6	2.83	6.08	63.5	25.2	0.96	0.80	1.77	72.8	12.5	23.5	4.7	2.8	39.3
	Li 2nd Cl. Conc.	307	15.6	2.75	5.92	63.6	25.0	1.11	0.88	1.76	76.0	13.5	25.1	5.8	3.3	41.9
	Li 1st Cl Conc	345	17.6	2.60	5.59	63.9	24.6	1.33	1.06	1.71	80.9	15.2	27.7	7.9	4.5	45.7
	Li Ro Conc.	461	23.5	2.15	4.64	65.4	23.1	1.84	1.68	1.48	89.7	20.9	34.7	14.6	9.5	53.0
	Li Ro & Scav. Conc.	481	24.6	2.10	4.53	65.5	22.9	1.91	1.75	1.46	91.3	21.8	36.0	15.8	10.3	54.8
	F3 Ro Scav Tail	1,312	67.0	0.03	0.07	77.5	12.8	3.22	5.00	0.22	4.0	70.3	54.9	72.7	80.2	22.4
	F3 Slime	165	8.4	0.31	0.67	69.0	16.9	4.03	4.67	1.77	4.6	7.9	9.1	11.5	9.4	22.8
	Head (calc.)	1,959	100	0.57	1.22	73.8	15.6	2.97	4.17	0.66	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,794	91.6	0.59	1.27	74.3	15.5	2.87	4.13	0.55	95.4	92.1	90.9	88.5	90.6	77.2
	Spod Flot Feed	1,794	91.6	0.59	1.27	74.3	15.5	2.87	4.13	0.55	95.4	92.1	90.9	88.5	90.6	77.2
F4	Li Non Mag Final Conc. 15A	379	19.3	2.44	5.25	65.9	23.7	1.33	1.34	0.97	84.8	17.2	28.8	8.5	6.1	28.1
Similar to Test 1, with the Objective to Improve Li Recovery by combining Li Ro and Scav Concentrates & Mica Cl Tail	Li 3rd Cl Conc	390	19.9	2.39	5.15	65.2	23.6	1.32	1.32	1.49	85.6	17.5	29.6	8.7	6.2	44.5
	Li 2nd Cl. Conc.	415	21.2	2.26	4.87	65.9	23.1	1.45	1.50	1.44	86.3	18.8	30.8	10.1	7.6	45.7
	Li 1st Cl Conc	456	23.3	2.09	4.50	66.8	22.4	1.62	1.79	1.35	87.6	21.0	32.9	12.4	9.9	47.0
	Li Ro and Scav. Conc.	597	30.5	1.66	3.57	68.7	20.6	2.02	2.52	1.10	90.9	28.2	39.5	20.3	18.3	50.4
	F4, Li Ro Scav Tail	1,117	57.1	0.02	0.04	78.3	13.0	3.32	5.10	0.12	1.9	60.2	46.7	62.3	69.0	10.3
	F4, Slime 2	55.7	2.8	0.54	1.16	68.8	17.2	3.33	4.38	3.05	2.8	2.6	3.1	3.1	3.0	13.0
	F4, Mica Con	89.8	4.6	0.15	0.32	67.4	18.7	5.24	4.00	1.65	1.2	4.2	5.4	7.9	4.4	11.4
	F4, Slime 1	97.4	5.0	0.35	0.75	70.7	17.0	3.89	4.58	2.01	3.1	4.7	5.3	6.4	5.4	15.0
	Head (calc.)	1,958	100	0.56	1.20	74.2	15.9	3.04	4.22	0.67	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,860	95.0	0.58	1.25	75.1	15.7	3.03	4.20	0.52	96.9	95.3	94.7	93.6	94.6	85.0
	Spod Flot Feed	1,715	87.6	0.60	1.30	75.6	15.5	2.90	4.20	0.40	92.9	88.5	86.2	82.6	87.3	60.6
F5	Li Non Mag Final Conc. 15A	339	17.4	2.75	5.92	65.6	24.6	1.19	1.16	0.98	82.5	15.2	27.1	6.8	4.8	25.4
Similar to Test 4 but with Finer Grinding Size, Stage Grinding to -65 M	Li 3rd Cl Conc	351	18.0	2.69	5.79	65.0	24.5	1.19	1.14	1.47	83.6	15.6	27.9	7.1	4.9	39.4
	Li 2nd Cl. Conc.	371	19.0	2.59	5.57	65.4	24.1	1.29	1.27	1.44	84.9	16.5	29.1	8.1	5.7	40.8
	Li 1st Cl Conc	395	20.3	2.45	5.28	66.0	23.5	1.40	1.46	1.38	85.7	17.8	30.2	9.3	7.0	41.7
	Li Ro and Scav. Conc.	475	24.4	2.07	4.46	68.0	21.9	1.69	2.01	1.19	87.0	22.1	33.8	13.6	11.6	43.4
	F5, Li Ro Scav Tail	1,139	58.5	0.01	0.02	79.7	12.8	3.23	5.13	0.09	1.0	61.9	47.4	62.3	71.1	7.9
	F5, Slime 2	35.9	1.8	0.63	1.36	67.5	17.9	2.88	4.01	4.17	2.0	1.7	2.1	1.8	1.8	11.5
	F5, Mica Con	159	8.2	0.40	0.86	70.3	17.8	4.96	4.06	1.22	5.6	7.6	9.2	13.4	7.8	14.8
	F5, Slime 1	140	7.2	0.35	0.75	70.8	16.6	3.79	4.54	2.10	4.3	6.7	7.5	9.0	7.7	22.5
	Head (calc.)	1,949	100	0.58	1.25	75.2	15.8	3.03	4.22	0.67	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,809	92.8	0.59	1.26	75.2	15.7	3.02	4.19	0.49	95.7	93.3	92.5	91.0	92.3	77.5
	Spod Flot Feed	1,615	82.8	0.61	1.30	75.9	15.5	2.82	4.21	0.36	88.1	84.0	81.2	75.9	82.7	51.2

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Objective	Product	Weight		Assays %							Distribution %					
		g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F6	Li Non Mag Final Conc. 15A	438	22.7	1.91	4.11	71.2	19.5	1.49	1.71	0.81	78.8	21.8	28.3	11.0	9.2	25.7
	Li 3rd Cl Conc	464	24.1	1.93	4.15	70.5	19.7	1.45	1.65	1.16	84.4	22.8	30.3	11.3	9.4	39.1
	Li 2nd Cl. Conc.	533	27.7	1.71	3.68	71.7	18.8	1.62	1.96	1.05	85.8	26.7	33.1	14.6	12.9	40.5
	Li 1st Cl Conc	698	36.2	1.32	2.85	74.0	17.1	1.91	2.50	0.84	87.0	36.0	39.5	22.4	21.4	42.3
	Li Ro. & Scav. Conc.	1,013	52.5	0.93	2.01	75.4	15.9	2.33	3.32	0.60	88.9	53.3	53.4	39.8	41.2	44.1
Similar to Test F5 but with Finer Grinding Size, Stage Grinding to -100 M	F6 Li Ro. Scav. Tail	548	28.4	0.01	0.01	74.9	14.4	3.80	5.85	0.09	0.3	28.6	26.2	35.1	39.4	3.6
	F6 SLime 2	64.9	3.4	0.59	1.27	69.0	16.9	3.07	4.22	3.50	3.6	3.1	3.6	3.4	3.4	16.4
	F6 Mica Con	153	8.0	0.15	0.32	70.6	17.3	4.80	4.14	1.13	2.2	7.5	8.8	12.4	7.8	12.5
	F6 Slime 1	148	7.7	0.36	0.77	71.6	16.3	3.76	4.53	2.18	5.0	7.4	8.0	9.4	8.3	23.4
	Head (calc.)	1,928	100	0.55	1.19	74.4	15.7	3.08	4.23	0.72	100	100	100	100	100	100
	Head (Dir.)	0.00	0.00	0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,779	92.3	0.59	1.26	75.1	15.7	3.02	4.19	0.49	95.0	92.6	92.0	90.6	91.7	76.6
	Spod Flot Feed	1,561	81.0	0.63	1.35	75.8	15.5	2.85	4.19	0.34	89.2	81.9	79.6	74.8	80.6	47.6
F7	Li Non Mag Final Conc. 15A	291	14.9	3.07	6.61	64.9	25.3	0.72	0.81	1.46	82.0	12.9	24.0	3.5	2.9	24.0
	Li 3rd Cl Conc	307	15.8	3.00	6.46	64.2	25.2	0.71	0.79	2.01	84.6	13.5	25.2	3.7	3.0	34.9
	Li 2nd Cl. Conc.	316	16.2	2.96	6.37	64.3	25.0	0.77	0.85	2.00	85.9	13.9	25.8	4.1	3.3	35.8
	Li 1st Cl Conc	330	16.9	2.87	6.17	64.5	24.7	0.88	0.98	1.97	87.0	14.6	26.6	4.9	3.9	36.7
	Li Ro Conc.	388	19.9	2.48	5.33	66.0	23.3	1.28	1.50	1.78	88.4	17.6	29.4	8.3	7.1	39.1
	Li Ro Conc. & Escav. Conc.	457	23.5	2.12	4.57	67.1	22.2	1.75	1.78	1.70	89.1	21.1	33.1	13.4	9.9	43.9
Repeat Test F6 but with Lower Collector Dossage	F7 Ro Scav. Tail	1,201	61.7	0.01	0.02	78.9	13.0	3.36	5.14	0.22	0.9	65.1	50.9	67.7	74.8	14.9
	F7 slime 2	103.8	5.3	0.47	1.01	70.7	16.3	3.35	4.43	2.75	4.5	5.0	5.5	5.8	5.6	16.1
	F7 mica con	55	2.8	0.21	0.45	65.2	19.7	5.34	3.77	2.33	1.1	2.5	3.6	5.0	2.5	7.3
	F7 slime 1	130	6.7	0.37	0.80	71.1	16.3	3.70	4.54	2.41	4.4	6.4	6.9	8.1	7.2	17.7
	Head (calc.)	1,948	100	0.56	1.20	74.8	15.8	3.06	4.23	0.91	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,818	93.3	0.58	1.26	75.2	15.8	3.03	4.19	0.52	95.6	93.6	93.1	91.9	92.8	82.3
	Spod Flot Feed	1,658	85.1	0.60	1.30	75.8	15.6	2.93	4.19	0.40	90.0	86.1	84.0	81.1	84.7	58.8
F8	Li Non Mag Final Conc. 15A	305	16.1	2.99	6.44	64.8	25.1	0.76	0.82	1.06	85.0	14.2	25.8	4.1	3.1	23.8
	Li 4th Cl Conc	316	16.7	2.92	6.29	64.1	25.0	0.76	0.81	1.58	86.1	14.5	26.6	4.2	3.2	36.8
	Li 3rd Cl Conc	323	17.1	2.89	6.21	64.3	24.9	0.81	0.86	1.58	87.0	14.9	27.1	4.6	3.5	37.6
	Li 2nd Cl. Conc.	333	17.6	2.83	6.09	64.4	24.7	0.88	0.94	1.57	88.0	15.4	27.7	5.1	3.9	38.6
	Li 1st Cl Conc	348	18.4	2.74	5.89	64.7	24.4	1.00	1.06	1.57	88.8	16.1	28.6	6.1	4.6	40.3
	Li Ro Conc.	353	18.7	2.70	5.81	64.8	24.3	1.05	1.11	1.56	88.9	16.4	28.9	6.5	4.9	40.5
Repeat Test F7 but without Mica Flotation	Li Ro Conc. & Scav. Conc.	411	21.8	2.35	5.07	65.1	23.7	1.65	1.42	1.52	90.2	19.2	32.9	11.9	7.3	46.3
	F8 Li Ro Scav Tail	1229.7	65.0	0.01	0.02	77.0	12.8	3.33	5.07	0.14	1.1	68.0	53.1	71.8	78.3	12.7
	F8 Slime 1	250	13.2	0.37	0.80	71.3	16.7	3.74	4.57	2.23	8.6	12.8	14.0	16.4	14.3	41.0
	Head (calc.)	1,891	100	0.57	1.22	73.7	15.7	3.02	4.21	0.72	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,641	86.8	0.60	1.29	75.3	15.6	2.97	4.16	0.40	91.4	87.2	86.0	83.6	85.7	59.0
	Spod Flot Feed	1,641	86.8	0.60	1.29	75.3	15.6	2.97	4.16	0.40	91.4	87.2	86.0	83.6	85.7	59.0
F9	Li Non Mag Final Conc. 15A	294	15.0	2.93	6.31	64.7	25.0	0.72	0.82	1.18	81.8	13.2	24.2	3.6	2.8	22.8
	Li 3rd Cl Conc	296	15.1	2.92	6.29	64.6	25.0	0.74	0.83	1.19	82.2	13.3	24.4	3.7	2.9	23.3
	Li 2nd Cl. Conc.	307	15.7	2.84	6.12	63.8	24.8	0.74	0.82	1.85	83.0	13.6	25.1	3.8	2.9	37.3
	Li 1st Cl Conc	320	16.4	2.79	6.00	64.0	24.7	0.83	0.90	1.84	84.9	14.2	26.1	4.5	3.4	38.7
Repeat Test F8 but Replacing FA2 with Aero 704 and Aero 845 (Cytec) Collector	Li Ro Conc.	340	17.4	2.68	5.76	64.3	24.3	1.00	1.05	1.80	86.6	15.2	27.3	5.7	4.2	40.3
	Li Ro Conc. & Scav. Conc.	398	20.4	2.33	5.01	65.4	23.2	1.43	1.50	1.64	88.1	18.1	30.4	9.6	7.0	42.9
	F9 Li Ro Scav Tail	1,256	64.2	0.02	0.03	77.3	12.8	3.34	5.27	0.18	1.8	67.4	53.0	70.6	77.4	14.9
	F9 Slime	237.4	12.1	0.34	0.73	69.9	16.2	3.73	4.70	2.41	7.7	11.5	12.7	14.9	13.0	37.7
	Head (calc.)	1,956	100	0.54	1.16	73.7	15.5	3.04	4.38	0.78	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
	Flotation Feed	1,718	87.9	0.60	1.29	75.4	15.7	2.98	4.17	0.42	92.3	88.5	87.3	85.1	87.0	62.3
	Spod Flot Feed	1,718	87.9	0.60	1.29	75.4	15.7	2.98	4.17	0.42	92.3	88.5	87.3	85.1	87.0	62.3
F10	Li Non Mag Final Conc. 15A	327	16.7	2.91	6.26	65.30	24.30	0.90	1.02	1.10	84.5	14.7	26.1	5.1	4.1	19.0
	Li 3rd Cl Conc	346	17.6	2.82	6.07	64.06	23.99	0.89	0.99	2.58	86.7	15.2	27.2	5.3	4.2	47.3
	Li 2nd Cl. Conc.	354	18.0	2.77	5.97	64.26	23.81	0.94	1.06	2.55	87.1	15.6	27.6	5.7	4.6	47.7
	Li 1st Cl Conc	369	18.8	2.67	5.74	64.67	23.46	1.05	1.20	2.47	87.5	16.4	28.4	6.6	5.4	48.3
	Li Ro Conc.	415	21.2	2.39	5.14	65.72	22.45	1.32	1.60	2.25	88.0	18.8	30.6	9.4	8.1	49.4
Repeat Test F8 but with Cold Water	Li Ro Conc. & Scav. Conc.	478	24.4	2.10	4.52	66.38	21.78	1.68	1.99	2.05	89.3	21.8	34.2	13.8	11.6	51.9
	F10 Li Ro Scav Tail	1227.8	62.6	0.02	0.04	77.70	13.00	3.34	4.94	0.29	1.9	65.7	52.4	70.6	74.1	18.9
	F10 Slime	255	13.0	0.39	0.84	71.20	16.00	3.57	4.56	2.16	8.8	12.5	13.4	15.7	14.2	29.2
	Head (calc.)	1,961	100	0.57	1.23	74.09	15.53	2.96	4.17	0.96	100	100	100	100	100	100
	Head (Dir.)			0.57	1.23	74.90	15.80	3.08	4.22	0.59						
	Flotation Feed	1,706	87.0	0.60	1.29	75.34	15.73	2.99	4.16	0.48	91.2	87.5	86.6	84.3	85.8	70.8
	Spod Flot Feed	1,706	87.0	0.60	1.29	75.34	15.73	2.99	4.16	0.48	91.2	87.5	86.6	84.3	85.8	70.8

Table 5: Flotation Reagent Overview

Test No.	Objective	Sample	Grinding	Reagents Added, g/t										
				Armac C	Armac T	Fuel Oil	NaOH	Na_2CO_3	D618	FA2	Aero 6493	Aero 704	Aero 845	Na2F6 Si
Test 1	Applying Developed SGS Flowsheet on the Glen Eagle Composite	2kg Composite Head Sample	Stage ground to -300 µm,	55	100	50	175	50	450	950				
Test 2	Similar to Test F1 but Replacing FA-2 with Aero 6493	2kg Composite Head Sample	Stage ground to -300 µm,	50	0	50	200	70	450		950			
Test 3	Repeat Test F1 but Eleminating Mica Flotation	2kg Composite Head Sample	Stage ground to -300 µm,	0	0	0	400	220	450	900				275
Test 4	Similar to Test 1, with the Objective to Improve Li Recovery by combining Li Ro and Scav Concentrates & Mica Cl Tail	2kg Composite Head Sample	Stage ground to -300 µm,	50	0	50	275	50	450	975				
Test 5	Similar to Test 4 but with Finer Grinding Size, Stage Grinding to -65 M	2kg Composite Head Sample	Stage ground to -210 µm,	0	50	50	260	50	450	975				
Test 6	Similar to Test F5 but with Finer Grinding Size, Stage Grinding to -100 M	2kg Composite Head Sample	Stage ground to -150 µm,	50	0	50	500	50	450	925				
Test 7	Repeat Test F6 but with Lower Collector Dossage	2kg Composite Head Sample	Stage ground to -150 µm,	50	0	50	270	50	450	675				
Test 8	Repeat Test F7 but without Mica Flotation	2kg Composite Head Sample	Stage ground to -210 µm,	0	0	0	275	50	450	675				
Test 9	Repeat Test F8 but Replacing FA2 with Aero 704 and Aero 845 (Cytec) Collector	2kg Composite Head Sample	Stage ground to -150 µm,	0	0	0	275	50	450	0		450	200	
Test 10	Repeat Test F8 but with Cold Water	2kg Composite Head Sample	Stage ground to -150 µm,	0	0	0	175	50	450	675				

4.3. Spodumene Flotation Tests

The initial flotation test, F1, was performed based on a flotation scheme previously developed at SGS on a similar ore system. Stage grinding was completed at a closing size of 48 mesh screen. An initial desliming stage was performed prior to mica flotation. Pre-mica flotation was conducted using Armac T. Since the froth was not strong, Armac C was then used with fuel oil to float mica. Mica flotation was conducted at natural pH. The mica flotation tailings were then conditioned with NaOH and Marasperse D618, followed by desliming before spodumene flotation. Sodium carbonate was used to adjust the pulp pH to about 8.5. Spodumene collector, FA-2, at 700 g/t was then added to the cell. Conditioning was performed at a high pulp density (>50%) for 5 minutes followed by flotation. Spodumene scavenger flotation was also performed on the rougher tailings by adding another 200 g/t of the collector to the rougher tailings. The concentrate from the spodumene rougher flotation was cleaned 3 times. The flotation kinetics was quick as a large portion of the concentrate mass was floated within 1 minute. The final 3rd cleaner concentrate was further upgraded using wet high intensity magnetic separation at 5 and 15 Amps (about 14000 Gauss). The results are illustrated in Figure 6. In this test, the final concentrate grade reached 6.22% Li_2O at a recovery of 74.4%. After two cleaners, the concentrate grade reached 5.96% Li_2O at a recovery of 79.4%. The scavenger concentrate grade was 3.24% Li_2O with a recovery of 2.14%.

Test F2 was similar to test F1 but the spodumene collector was replaced with Aero 6493. In comparison with test F1, the final concentrate grade in test F2 was higher at 7.02% Li_2O with a recovery of 36.6%. Even the rougher concentrate had a high grade, 6.24% Li_2O, but the recovery was 58.2%. Thus, it was concluded that Aero 6493 is more selective but is also a weaker collector. Additional tests are required to achieve higher recovery with this collector.

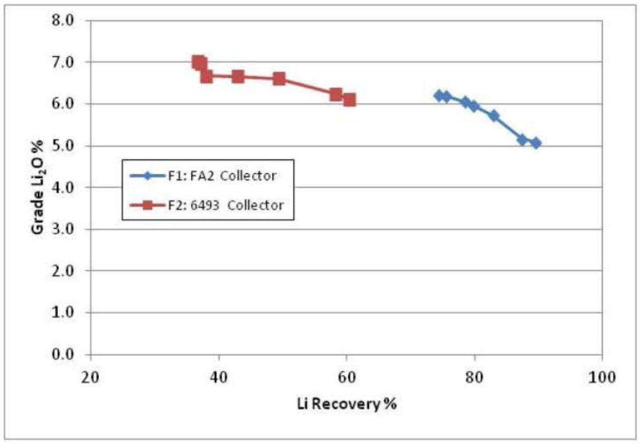


Figure 6: Spodumene Grade-Recovery Curves – Initial Tests on Master Composite

Test F3 was conducted without mica pre-flotation. The flotation feed was conditioned with NaOH and Marasperse D618, followed by desliming before spodumene flotation. FA-2 was used as the main spodumene collector. Conditioning was performed at a high pulp density for 5 minutes followed by rougher flotation. The Li rougher concentrate was then cleaned 3 times. Sodium fluorosilicate (Na_2F_6Si) was added to the cleaner to depress mica. The results are provided in Figure 7. In this test, the final concentrate grade reached 6.26% Li_2O with a lithium recovery of 70.6%. After two cleaners, the concentrate grade reached 5.92% Li_2O at 76.0% Li recovery. The lower Li concentrate grade is likely due to the recovery of mica as mica pre-flotation was not performed.

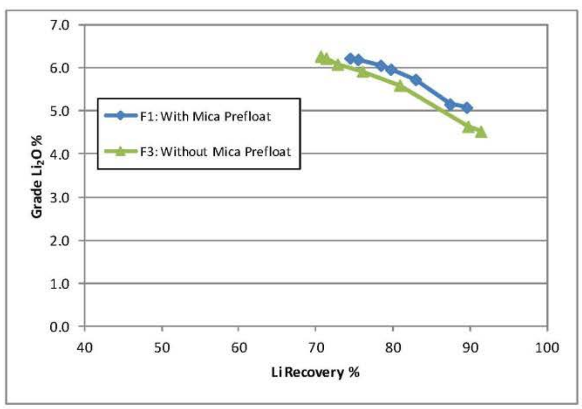


Figure 7: Effect of Mica Pre-Float on Spodumene Flotation

Test F4 was conducted with the objective of increasing Li recovery through a number of changes to test F1. Lithium losses to the mica rougher concentrate were reduced by cleaning the mica rougher concentrate. The mica cleaner tails was combined with the mica rougher tails for spodumene flotation. Furthermore, the spodumene scavenger concentrate was combined with the Li rougher concentrate for cleaning. The combined rougher and scavenger concentrate was cleaned three times. The results were not positive. The combined rougher and scavenger concentrate grade only reached 3.57% Li_2O but at a recovery of 90.9%. After 3 cleaners and magnetic separation, the final Li concentrate grade was 5.25% Li_2O with a recovery of 84.8%. Thus, the recovery has been improved but at the expense of grade. Figure 8 compares the grade-recovery of F4 against F1.

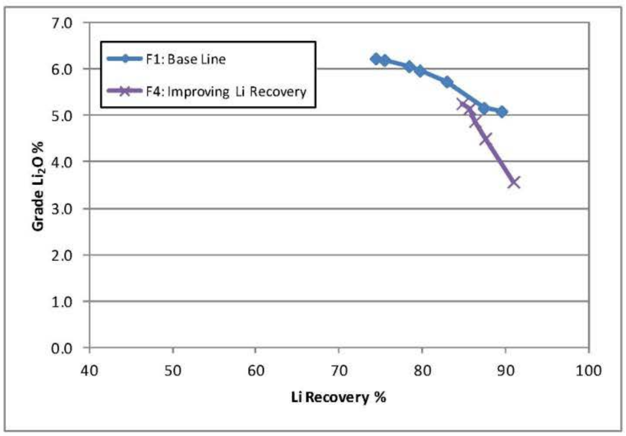


Figure 8: Spodumene Flotation Plots from Test F1 and F4

Test F5 was conducted with the objective to evaluate finer flotation feed. In this test, the flotation feed was stage ground to 65 mesh, 212 µm, for a K_{80} of 149 µm. This test was conducted similar to test F4. As illustrated in Figure 9, the results were better. The final lithium concentrate grade reached 5.92% Li_2O with a recovery of 82.5%. In test F6, the head sample was stage ground to 100 mesh, 150 µm, for a K_{80} of 112 µm. The flotation results were not positive. The combined rougher and scavenger concentrate grade was only about 2.01% Li_2O at 88.9% recovery. This grade is just slightly higher than the feed grade. Thus, even after three cleaners and magnetic separation, the final spodumene concentrate reached to 4.11% Li_2O with a recovery of 78.8%. It is believed that the spodumene collector dosage was excessive for the test at a K_{80} of 112 µm, resulting in low concentre grade. Consequently, in test F7, it was decided to reduce spodumene collector, FA-2, dosage from 700 g/t in rougher flotation to 450 g/t. Collector dosage in scavenger flotation was kept at 200 g/t but it was decided not to combine the rougher and scavenger flotation concentrate for cleaning. It is clear that by reducing collector dosage, the final concentrate grade can be improved significantly, reaching 6.61% Li_2O at 82% recovery. Even after a single cleaning stage, the concentrate grade reached 6.17% Li_2O with a recovery of 87.0%. This test clearly indicated that finer grinding with lower collector dosage is beneficial for spodumene flotation. The assays of Na_2O and K_2O in this concentrate were 0.81% and 0.72%, respectively. The iron assay in the concentrate was fairly high, even after magnetic separation, and reached 1.46% Fe_2O_3. It is believed that the majority of the iron is in the spodumene crystal structure and it cannot be removed by a magnet separator. Microprobe analysis might be conducted on handpicked spodumene crystals to determine the iron content. The majority of Li losses in this test were to the slimes (4.4% to the initial slime and 4.5% to the slime after scrubbing). Minor amounts of Li, 1.1%, reported to the mica concentrate.

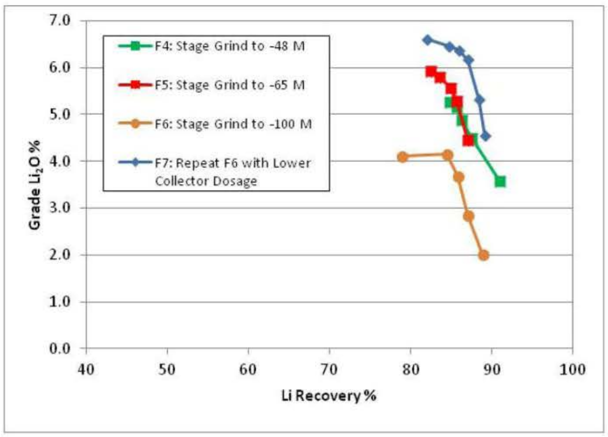


Figure 9: Effect of Grinding Fineness on the Spodumene Flotation Performance

Due to the success in test F7, test F8 was conducted similar to test F7, but without mica pre-flotation. The results are depicted in Figure 10. After two cleaners, the concentrate grade reached 6.09% Li_2O with a recovery of 88.0%. The assays of Na_2O, K_2O, and Fe_2O_3 in the 2^{nd} cleaner concentrate were 0.94%, 0.88%, and 1.57%, respectively. After the third cleaner and passing the final concentrate through WHIMS, the concentrate grade reached 6.44% Li_2O at a recovery of 85%. The assays of Na_2O, K_2O, and Fe_2O_3 in the final concentrate were reduced to 0.82%, 0.76%, and 1.06%, respectively. In conclusion, elimination of mica pre-flotation did not have any negative effect on the final concentrate grade or recovery. Like the previous test, the Li scavenger concentrate was collected but not combined with the rougher concentrate. The Li scavenger concentrate mass pull was 3.1% and the concentrate grade was 0.54% Li_2O with a recovery of 1.3%. It is obvious that this is a very low grade concentrate and will have a bigger impact in diluting the rougher concentrate for cleaning than the benefits of additional Li recovery. The rougher tailing Li assay was about 0.04% Li_2O with Li losses of 2.5%. The major Li losses in this test was in the slimes after scrubbing which reached 8.6% with a mass pull of 13.2%. The slimes graded 0.37% Li_2O. The Li loss to this slime is similar to the Li loss of the combined slimes generated in test F7.

Test F9 was conducted with the objective to evaluate CYTEC fatty acid collector, Aero 704, and the proposed promoter, Aero 845. The results are illustrated in Figure 10. The final concentrate grade reached 6.31% Li_2O with a recovery of 81.8%. After the 1st cleaning stage, the concentrate grade reached 6.00% Li_2O at 84.9% recovery. The assays of Na_2O, K_2O, and Fe_2O_3 in the 1st cleaner concentrate were 0.90%, 0.83%, and 1.84%, respectively. The scavenger concentrate grade was low at 0.86% Li_2O with a recovery of 2.4% and a mass pull of 3.3%. The Li assay of the rougher tail was 0.08% Li_2O with a total lithium loss of 4.2%.

The final test, F10, was conducted with the objective to evaluate if cold water can have a negative effect on the flotation performance. In test F10, scrubbing, desliming, conditioning, cleaner flotation were performed in cold water (8-10°C). The results are provided Figure 10. After two cleaners, the concentrate grade reached 5.97% Li_2O with a recovery of 87.1%. The assays of Na_2O, K_2O, Fe_2O_3 in this concentrate were 1.06%, 0.94%, and 2.55%, respectively. These assays, especially iron oxide, were higher than those measured in test F8. The final lithium concentrate grade reached 6.26% with a recovery of 84.5%. Thus, it does not appear that cold water has a significant negative effect on spodumene flotation.

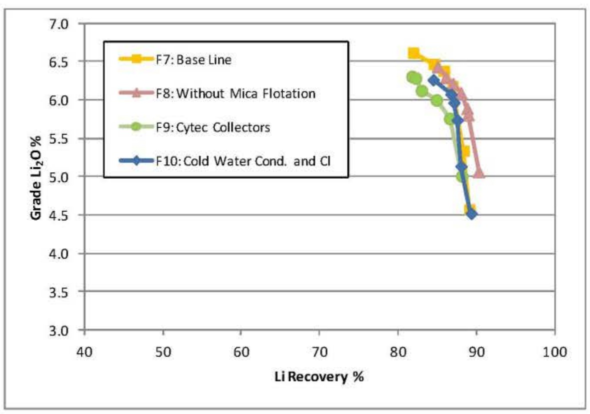
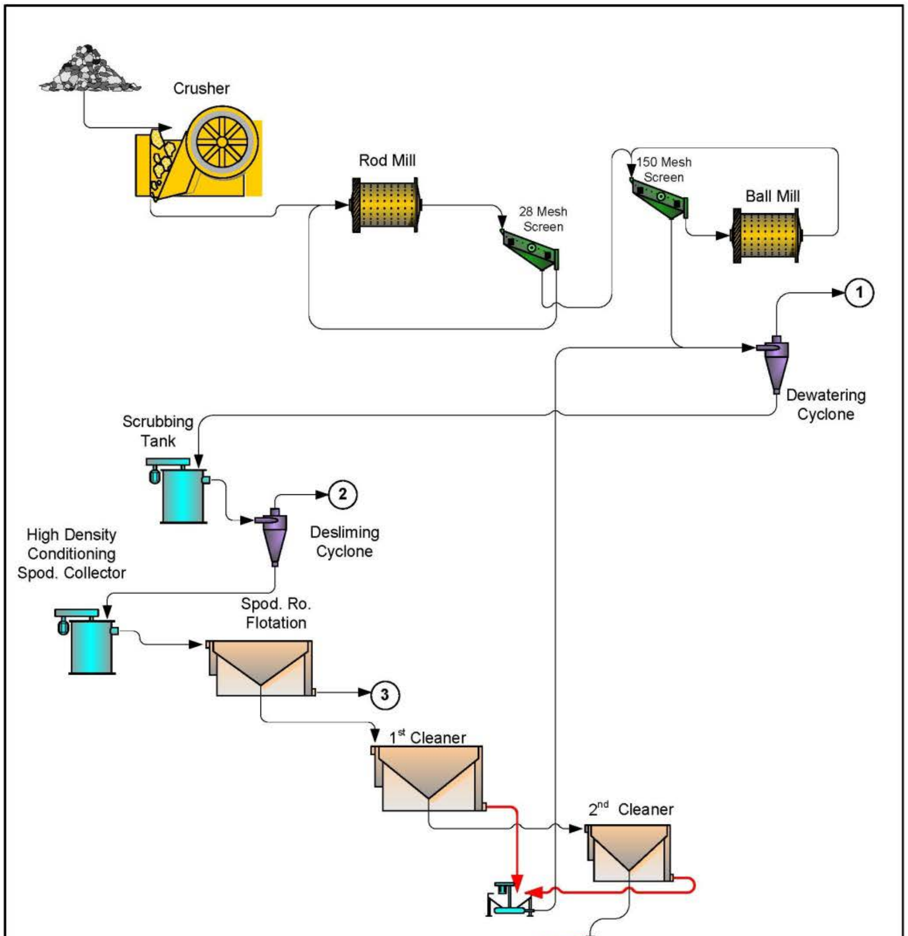


Figure 10: Final Development Flotation Test Results

4.4. Preliminary Spodumene Flotation Flowsheet

Based on the results presented in the previous section, a preliminary flotation flowsheet has been developed and presented in Figure 11. In this flowsheet, the flotation feed is ground to 150 mesh for a K_{80} of 110 µm. The ground material is dewatered before entering to the scrubber. Pulp density in the scrubber should be ideally 50 to 60%. The flotation feed is scrubbed for 10 minutes at pH 11 adjusted with NaOH and in the presence of D618 at 250 g/t. After desliming, the flotation feed is conditioned at 60% pulp density with FA-2 collector for 5 minutes. Collector dosage is 450 g/t. Pulp pH (8.5) in the conditioner is to be adjusted with soda ash. Rougher flotation is fairly fast and it is expected to be completed within 3 minutes. Two cleaners are expected to be sufficient in order to generate the target concentrate grade (~ 6.0 Li_2O) with approximately 85% recovery. Both cleaner tails might be rejected to the tailings dam (if the grade is low and similar to test F8) or combined and return to the dewatering cyclone. The final spodumene concentrate is washed with sulfuric acid to break the froth and improve magnetic separation efficiency. A magnetic intensity of about 15000 Gauss should be sufficient.



Product Legend	
1	Recycled Water
2	Slimes
3	Spodumene Rougher Tailings
4	Spodumene Concentrate
5	Magnetic Tail

Figure 11: Proposed Flotation Flowsheet

Conclusions and Recommendations

Metallurgical testwork was conducted on a master composite prepared from drill core samples taken from the Authier property. Conclusions of the testwork and recommendations are outlined below.

Conclusions

The following conclusions are drawn from this project:

- The sole Li mineral identified is spodumene. The head sample consists of major amounts of albite (37.2%), quartz (26.5%), microcline (16.2%), spodumene (14.9%) and muscovite (4.8%).
- Stage-grinding with a closing size of 100 mesh (150 µm) was necessary to optimize the flotation performance.
- Mica pre-flotation was not necessary.
- Removal of slimes was completed after a scrubbing stage. Lignin sulfonate and NaOH additions in the scrubbing stage were beneficial in terms of slime dispersion and separation. Lignin sulfonate (D618) was added to the scrubber at 250 g/t and pH was adjusted to 11.0 with NaOH.
- Soda ash was used as the pH regulator in spodumene flotation. Pulp pH was kept at around 8 during the rougher and cleaner flotation stages.
- FA-2 was the suitable collector for spodumene flotation at a dosage of 450 g/t in the rougher flotation. A hydroxamate collector, Aero 6493, which was also tested, provided better selectively but lower recovery. This test however was done at a coarser grind (48 Mesh) and it might provide better results with finer grinding (100 Mesh). Collectors Aero 704 and Aero 845 were also tested but did not provide better results.
- Flotation kinetics was fast and a large portion of the lithium was floated within 1 minute. Rougher flotation was normally completed in less than 3 minutes.
- A flotation concentrate grading 6.09% Li_2O at a lithium recovery of 88.0% was achieved on the master composite after two cleaners. About 8.6% of the lithium reported to the slimes and 2.5% to the rougher tailings. The lithium loss to the magnetic product was 1.1%. The grinding fineness (K_{80}) of the rougher tailings was 134 µm while the slimes fraction was 80% passing 18 µm.

Recommendations:

- Performing further flotation tests to identify the effect of D618 in the scrubber, lowering dosage of the FA-2 collector, testing other industrial collectors (such as SNF fatty acid reagents), and additional testing of hydroxamate collector, Aero 6493;
- Reducing the lithium losses to the slimes by targeting a finer cut size on the slime rejection;
- Performing HLS to reject silicate gangue before entering the flotation circuit and reduce the load on the grinding mill;
- Study the effect of in-situ mine water on flotation performance;

- Performing a locked cycle test to evaluate the effect of recycling streams;
- Performing grinding variability program on samples with different head grades and from different locations in the ore body to identify the effect of lithium head grade and sample mineralogy on overall grindability and flotation performance;
- Evaluating roasting, acid roasting and water leaching to extract lithium from spodumene concentrate;
- Evaluating lithium carbonate/hydroxide or other desired lithium compound formation from lithium leach solution.

Appendix A – XRD Details

SGS

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Metallurgical Operations*
Project Number/ LIMS No.	*13650-001/MI4518-AUG12*
Reporting Date:	*September 20, 2012*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations :	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit :	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Elena Valeyeva
Technologist, XRD

Huyun Zhou, Ph.D., P.Geo.
Senior Mineralogist

SGS Minerals
a division of SGS Canada Inc.
P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0
Tel: (705) 652-2000 Fax: (705) 652-6365 www.sgs.com www.sgs.com/met
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Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of a test sample material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) and released on software as a database of Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

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Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Authier Li Deposit Drill Core Sample	plagioclase	quartz, potassium-feldspar, spodumene *(monoclinic)*	mica	*magnetite

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Magnetite	Fe_3O_4
Mica	$K(Mg,Fe)Al_2Si_3AlO_{10}(OH)_2$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium-Feldspar	$KAlSi_3O_8$
Quartz	SiO_2
Spodumene	$LiAlSi_2O_6$

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.

Metallurgical Operations
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20-Sep-12

Semi-Quantitative X-ray Diffraction Results

Mineral	Authier Li Deposit Drill Core Sample (wt %)
Albite	37.2
Quartz	26.5
Microcline	16.2
Spodumene	14.9
Muscovite	4.8
Magnetite	0.3
TOTAL	99.9

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

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Chemical Balance

Authier Li Deposit Drill Core Sample

Name	Assay[1]	SQD[2]	Delta	Status
SiO2	74.8	74.4	0.37	Both
Al2O3	15.8	16.1	-0.34	Both
Na2O	4.27	4.40	-0.13	Both
K2O	3.08	3.31	-0.23	Both
Li2O	1.23	1.20	0.03	Both
Fe2O3	0.58	0.32	0.26	Both
CaO	0.17	-	0.17	XRF
MnO	0.10	-	0.10	XRF
MgO	0.07	-	0.07	XRF
Cr2O3	0.02	-	0.02	XRF
H2O	-	0.22	0.22	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

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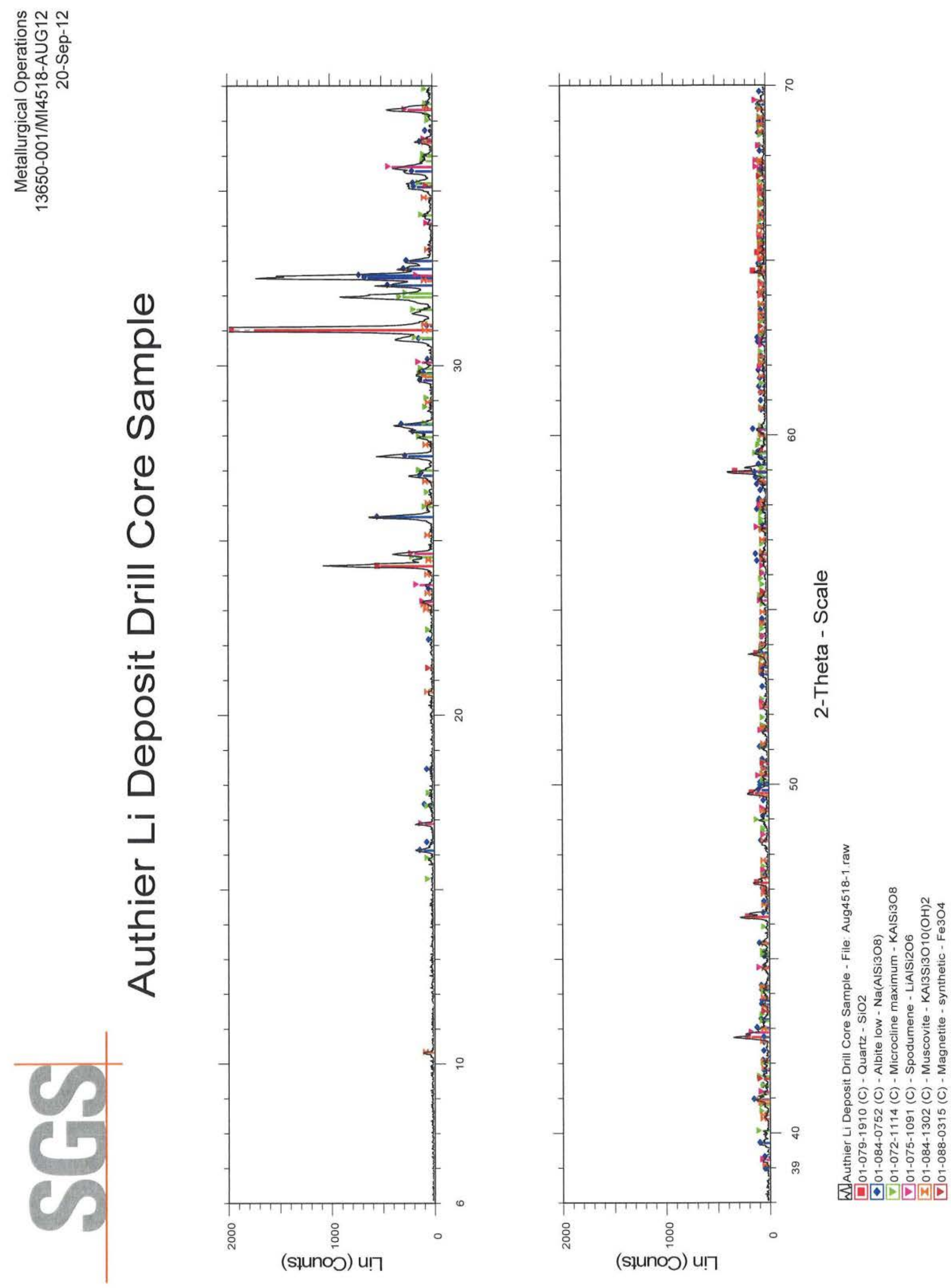

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0. Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Appendix B – Grindability Test Details

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SGS Minerals Services

Standard Bond Ball Mill Grindability Test

Project No.: 13650-001 Date: 14-Nov-12
Sample: Drill Cores

Purpose: To determine the ball mill grindability of the sample in terms of Bond work index number.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test Conditions:

Feed 100% Passing	6	mesh
Mesh of grind:	100	mesh
Test feed weight (700 mL):	1,307	grams
Equivalent to 1,867	kg/m³	at Minus 6 mesh
Weight % of the undersize material in the ball mill feed:	13.0%	
Weight of undersize product for 250% circulating load:	373 grams	

Results: Gram per Rev Average for the Last Three Stages = **1.59 g**
Circulation load = **255%**

CALCULATION OF A BOND WORK INDEX

$$BWI = \frac{44.5}{P1^{0.23} \times Grp^{0.82} \times \left\{\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right\}}$$

P1 = 100% passing size of the product	150	microns
Grp = Grams per revolution	1.59	grams
P_{80} = 80% passing size of product	127	microns
F_{80} = 80% passing size of the feed	2,408	microns

BWI = **14.1** kWh/t (imperial)

BWI = **15.6** kWh/t (metric)

Comments:

Stage No.	# of Revs	New Feed (grams)	Product in Feed (grams)	Material to Be Ground (grams)	Material Passing 100 mesh in Product (grams)	Net Ground Material (grams)	Material Ground Per Mill Rev (grams)
1	100	1,307	170	203	308	138	1.38
2	242	308	40	333	384	344	1.42
3	228	384	50	323	408	358	1.57
4	204	408	53	320	391	338	1.66
5	195	391	51	322	366	315	1.62
6	202	366	48	326	365	317	1.57
7	207	365	48	326	374	326	1.58
		Average for Last Three Stages =			368 g		1.59 g

BWI-Drill Cores.xls Results
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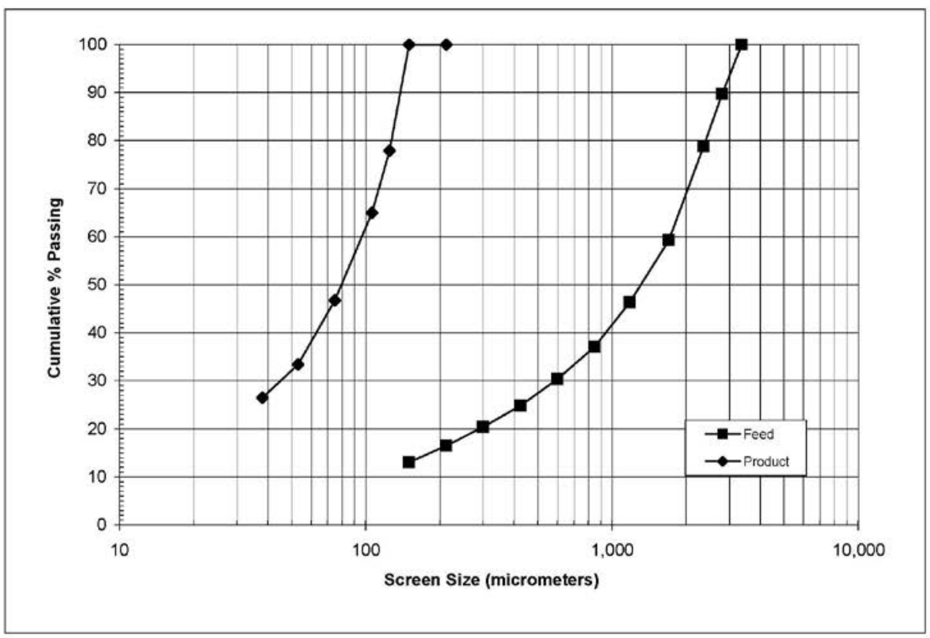
SGS Minerals Services

Standard Bond Ball Mill Grindability Test

Project No.: 13650-001 Date: 14-Nov-12
Sample: Drill Cores

Size		Feed Particle Size Analysis: Weight	% Retained		% Passing	Product Particle Size Analysis: Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative	grams	Individual	Cumulative	Cumulative
6	3,360	0.00	0.00	0.00	100.0				
7	2,800	68.9	10.3	10.3	89.7				
8	2,360	72.9	10.9	21.2	78.8				
10	1,700	130.0	19.5	40.7	59.3				
14	1,180	86.9	13.0	53.7	46.3				
20	850	61.6	9.22	62.9	37.1				
28	600	45.1	6.75	69.7	30.3				
35	425	36.5	5.46	75.1	24.9				
48	300	30.4	4.55	79.7	20.3				
65	212	25.6	3.83	83.5	16.5	0.00	0.00	0.00	100.0
100	150	23.1	3.46	87.0	13.0	0.00	0.00	0.00	100.0
115	125					34.5	22.1	22.1	77.9
150	106					20.1	12.9	35.0	65.0
200	75					28.5	18.3	53.3	46.7
270	53					20.8	13.3	66.6	33.4
400	38					10.7	6.86	73.5	26.5
Pan	-	87.1	13.0	100.0	-	41.3	26.5	100.0	-
Total	**-**	**668.1**	**100.0**	F_{80}:	**2,408**	**155.9**	**100.0**	P_{80}:	**127**



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SGS Minerals Services

Standard Bond Rod Mill Grindability Test

Project No.:	13650-001	Date:	30-Aug-12
Sample.:	Drill Cores		

Purpose: To determine the rod mill grindability of the sample in terms of a Bond work index number.

Procedure: The equipment and procedure duplicate the Bond method for determining rod mill work indices.

Test Conditions:

Feed 100% Passing	0.5	inch
Mesh of grind:	14	mesh
Test feed weight (1250 mL):	2,189	grams
Equivalent to : 1,751	kg/m³	at Minus 1/2"
Weight % of the undersize material in the rod mill feed:	12.3%	
Weight of undersize product for 100% circulating load:	1,095 grams	

Results: Gram per Rev Average for the Last Three Stages = **11.07 g**
Circulation load = **99%**

CALCULATION OF A BOND WORK INDEX

$$\boxed{RWI = \frac{62}{P1^{0.23} \times Grp^{0.625} \times \left\{\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right\}}}$$

P1 = 100% passing size of the product	1,180	microns
Grp = Grams per revolution	11.07	grams
P_{80} = 80% passing size of product	876	microns
F_{80} = 80% passing size of the feed	11,320	microns

RWI = **11.1** kWh/t (imperial)

RWI = **12.3** kWh/t (metric)

Comments:

Stage No.	# of Revs	New Feed (grams)	Product in Feed (grams)	Material to Be Ground (grams)	Material Passing 14 mesh in Product (grams)	Net Ground Material (grams)	Material Ground Per Mill Rev (grams)
1	50	2,189	269	826	671	402	8.05
2	126	671	82	1,012	1,347	1,265	10.04
3	93	1,347	165	929	1,172	1,007	10.82
4	88	1,172	144	951	1,111	967	10.99
5	87	1,111	136	958	1,091	955	10.97
6	88	1,091	134	961	1,112	978	11.11
7	86	1,112	137	958	1,092	955	11.11
		Average for Last Three Stages =			1,098 g		11.07 g

RWI-Drill Cores.xls Results
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SGS Minerals Services

Standard Bond Rod Mill Grindability Test

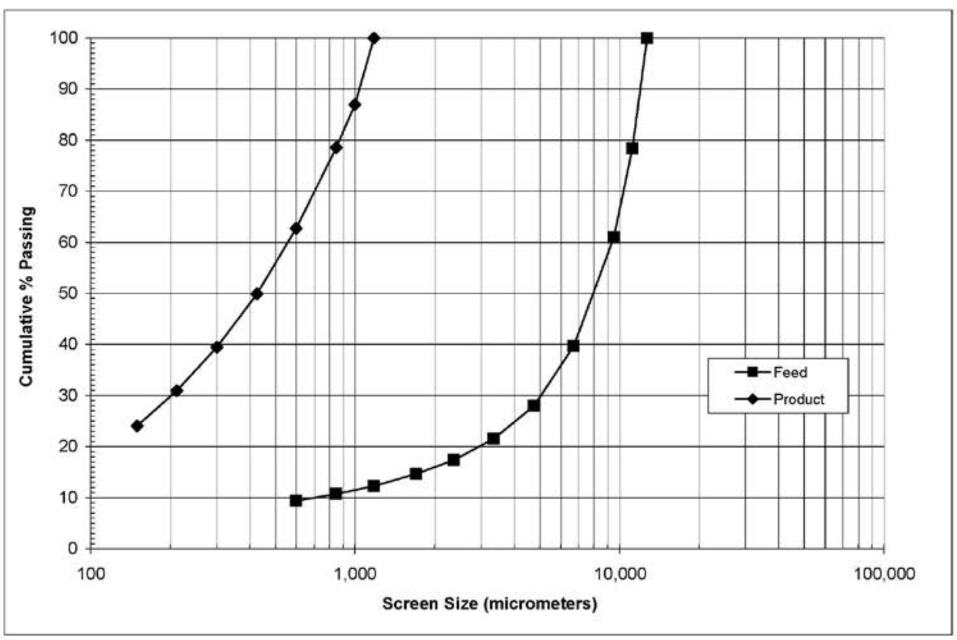
Project No.: 13650-001 Date: 30-Aug-12
Sample.: Drill Cores

Feed Particle Size Analysis

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
1/2"	12,700	0.00	0.00	0.00	100.0
7/16"	11,200	269.9	21.6	21.6	78.4
3/8"	9,500	217.6	17.4	39.1	60.9
3	6,700	265.3	21.3	60.4	39.6
4	4,750	145.2	11.6	72.0	28.0
6	3,350	80.9	6.49	78.5	21.5
8	2,360	52.0	4.17	82.7	17.3
10	1,700	33.4	2.68	85.3	14.7
14	1,180	29.7	2.38	87.7	12.3
18	1,000	-	-	-	-
20	850	19.6	1.57	89.3	10.7
28	600	17.0	1.36	90.7	9.34
35	425				
48	300				
65	212				
100	150				
Pan		116.5	9.34	100.0	-
Total	**-**	**1247.1**	**100.0**	**F_{80}:**	**11,320**

Product Particle Size Analysis

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
14	1,180	0.00	0.00	0.00	100.0
18	1,000	45.9	13.0	13.0	87.0
20	850	29.8	8.46	21.5	78.5
28	600	55.4	15.7	37.2	62.8
35	425	45.5	12.9	50.1	49.9
48	300	36.6	10.4	60.5	39.5
65	212	30.1	8.55	69.1	30.9
100	150	24.3	6.90	76.0	24.0
Pan		84.6	24.0	100.0	-
Total	**-**	**352.2**	**100.0**	**P_{80}:**	**876**



RWI-Drill Cores.xls Results
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Appendix C – Flotation Test Details

Test: F1 **Project No.:** 13650-001 **Operator**: Yangling **Date**: 08/22/2012

Purpose: **Applying Developed SGS Flowsheet on the Glen Eagle Composite Sample**

Procedure: As outlined below.

Feed: 2kg Composite Head Sample

Grind: Stage ground to -300 µm, 15+10min

Regrind: None

Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac T**	**Fuel Oil**	**Armac C**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO3)2**	**D618**	**FA-2**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**5%**	**10%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime														7.8	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
High density Condition (Natural pH)		100	50	40								3		8.0	24
Mica Ro (2kg cell)												1	3	7.7	24
Mica Cleaner Use appropriate Cell (maybe 0.5kg is good)															
Mica Clnr #1 (0.5kg cell) 1200 RPM				15								1	3	7.8	22
High Density Scrubbing					175				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.5	24
High Intensity Condition 2 (2kg cell) 1500 RPM										700		5		8.5	
Li Rougher 1 (2kg cell)													2.5	8.0	
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	7.8	24
Submit Scav Conc for Assay															
Only Clean Rougher Conc.						50									
Li 1st Cleaner (1kg cell)										50		1	2	8.7	24
Li 2nd Cleaner (1kg cell)												1	2	8.0	
Li 3rd Cleaner (1kg cell)												1	1	7.8	
Total	0	100	50	55	175	50			450	950	0	26	15	.	

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Test 1 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F1 15A WHIM non-mag	284	14.4	2.89	6.22	64.9	24.7	0.91	0.98	1.03	74.4	12.6	22.6	4.26	3.36	19.6
F1 15A WHIM mag	5.50	0.28	2.09	4.50	58.7	23.2	1.32	1.12	6.44	1.04	0.22	0.41	0.12	0.07	2.38
F1 5A WHIM mag	17.9	0.91	1.82	3.92	53.1	22.5	0.71	0.68	10.2	2.96	0.65	1.30	0.21	0.15	12.3
F1 Li 3rd Cl Tail	10.0	0.51	1.42	3.06	69.9	19.7	2.49	2.74	1.49	1.29	0.48	0.64	0.41	0.33	1.00
F1 Li 2nd Cl Tail	26.3	1.33	1.35	2.91	71.2	19.3	2.46	3.09	1.03	3.22	1.28	1.64	1.07	0.98	1.82
F1 Li 1st Cl Tail	58.5	2.97	0.84	1.81	73.2	17.2	2.94	4.03	0.64	4.46	2.92	3.25	2.84	2.85	2.51
F1 Li Ro Scav Con	15.8	0.80	1.49	3.21	65.4	21.0	2.85	2.34	1.56	2.14	0.70	1.07	0.74	0.45	1.65
F1 Li Ro Scav Tail	1,239	62.8	0.03	0.06	78.2	13.0	3.36	5.01	0.26	3.15	66.1	52.0	68.7	75.1	21.6
F1 Slime 2	21.6	1.09	0.59	1.27	65.0	17.7	3.27	4.13	4.53	1.16	0.96	1.23	1.17	1.08	6.57
F1 Mica Ro Con	224	11.34	0.21	0.45	70.7	16.8	4.32	4.27	1.38	4.26	10.8	12.1	15.9	11.6	20.7
F1 Slime 1	71.3	3.6	0.30	0.65	69.4	16.1	3.88	4.68	2.07	1.94	3.38	3.71	4.56	4.04	9.90
Head (calc.)	1,973	100	0.56	1.20	74.3	15.7	3.07	4.19	0.76	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.90	15.80	3.08	4.22	0.59						
Flotation Feed	1,902	96.4	0.57	1.22	74.5	15.7	3.04	4.17	0.71	98.1	96.6	96.3	95.4	96.0	90.1
Spod Flot Feed	1,656	84.0	0.62	1.33	75.1	15.5	2.87	4.16	0.57	92.6	84.9	82.9	78.3	83.3	62.8

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	284	14.4	2.89	6.22	64.9	24.7	0.91	0.98	1.03	74.4	12.6	22.6	4.26	3.36	19.6
Li Non Mag Final Conc. 5A	289	14.7	2.87	6.19	64.8	24.7	0.92	0.98	1.13	75.4	12.8	23.0	4.38	3.44	22.0
Li 3rd Cl Conc	307	15.6	2.81	6.06	64.1	24.5	0.91	0.97	1.66	78.4	13.4	24.3	4.59	3.59	34.2
Li 2nd Cl. Conc.	317	16.1	2.77	5.96	64.3	24.4	0.96	1.02	1.66	79.7	13.9	25.0	5.00	3.92	35.2
Li 1st Cl Conc	343	17.4	2.66	5.73	64.8	24.0	1.07	1.18	1.61	82.9	15.2	26.6	6.07	4.90	37.0
Li Ro Conc.	402	20.4	2.40	5.16	66.0	23.0	1.34	1.59	1.47	87.4	18.1	29.9	8.90	7.75	39.6
Li Ro & Scav. Conc.	418	21.2	2.36	5.08	66.0	22.9	1.40	1.62	1.47	89.5	18.8	30.9	9.65	8.20	41.2

Glen Eagle Flot Tests MA.xls F1
updated 01/08/2013

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Test: F2 **Project No.:** 13650-001 **Operator**: Yangling **Date:** 08/22/2012

Purpose: Similar [illegible]

Procedure: As outlined below.

Feed: 2kg Composite Head Sample

Grind: Stage ground to -300 µm, 15+10min

Regrind: None

Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15 A WHIMS

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac C	Fuel Oil	$Fe_2(SO4)_3$	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	6493	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
High density Condition (Natural pH)		50	50									3		7.7	24
Mica Ro (2kg cell)												1	3	7.6	24
High Density Scrubbing					200				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C						60								8.5	24
High Intensity Condition 2 (2kg cell) 1500 RPM										700		5		8.5	
Li Rougher 1 (2kg cell)						10							2.5	8.5	
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	7.8	24
Submit Scav Conc for Assay															
Only Clean Rougher Conc															
Li 1st Cleaner (1kg cell)										50		1	2	7.8	24
Li 2nd Cleaner (1kg cell)												1	2	7.6	
Li 3rd Cleaner (1kg cell)												1	1	7.5	
Total	0	50	50		200	70			450	950	0	25	12		

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Test 2 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F2 15A Non-Mag	122	6.2	3.26	7.02	64.2	25.8	0.41	0.48	1.25	36.6	5.4	10.3	0.9	0.7	10.7
F2 15A Mag	2.80	0.14	2.18	4.69	55.0	23.5	0.36	0.51	8.20	0.6	0.1	0.2	0.0	0.0	1.6
F2 5A Mag	8.6	0.44	1.10	2.37	44.7	21.3	0.23	0.34	17.7	0.9	0.3	0.6	0.0	0.0	10.7
F2 Li 3rd Cl Tail	17.3	0.88	3.04	6.54	64.1	25.1	0.71	0.75	2.10	4.8	0.8	1.4	0.2	0.2	2.5
F2 Li 2nd Cl Tail	23.8	1.21	2.94	6.33	65.1	24.8	0.88	0.96	1.83	6.4	1.1	1.9	0.4	0.3	3.1
F2 Li 1st Cl Tail	43.7	2.22	2.22	4.78	67.0	22.1	1.54	1.95	1.29	8.9	2.0	3.2	1.1	1.0	3.9
F2 Li Ro Scav Con	12.9	0.66	1.82	3.92	68.1	20.6	1.97	2.68	1.60	2.2	0.6	0.9	0.4	0.4	1.4
F2 Li Ro Scav Tail	1,496	75.9	0.24	0.52	75.7	13.8	3.15	4.66	0.34	33.0	78.3	67.5	80.0	84.7	35.6
F2 Slime 2	38.6	1.96	0.31	0.67	70.1	16.8	3.82	4.70	1.78	1.1	1.9	2.1	2.5	2.2	4.8
F2 Mica Con	147	7.45	0.23	0.50	69.0	17.5	4.51	4.25	1.32	3.1	7.0	8.4	11.2	7.6	13.6
F2 Slime 1	57.3	2.9	0.48	1.03	64.6	18.6	3.37	4.16	3.00	2.5	2.6	3.5	3.3	2.9	12.0
Head (calc.)	1,969	100	0.55	1.19	73.4	15.5	2.99	4.18	0.72	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.90	15.80	3.08	4.22	0.59						
Flotation Feed	1,912	97.1	0.57	1.23	75.2	15.7	3.07	4.22	0.53	97.5	97.4	96.5	96.7	97.1	88.0
Spod Flot Feed	1,727	87.7	0.61	1.31	75.66	15.50	15.00	4.20	0.46	93.3	88.6	86.0	83.0	87.3	69.6

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	122	6.2	3.26	7.02	64.2	25.8	0.41	0.48	1.25	36.6	5.4	10.3	0.9	0.7	10.7
Li Non Mag Final Conc. 5A	125	6.3	3.24	6.97	64.0	25.7	0.41	0.48	1.41	37.1	5.5	10.5	0.9	0.7	12.3
Li 3rd Cl Conc	134	6.8	3.10	6.67	62.8	25.5	0.40	0.47	2.45	38.0	5.8	11.1	0.9	0.8	23.0
Li 2nd Cl. Conc.	151	7.7	3.09	6.65	62.9	25.4	0.43	0.50	2.41	42.8	6.6	12.5	1.1	0.9	25.5
Li 1st Cl Conc	175	8.9	3.07	6.61	63.2	25.3	0.49	0.57	2.33	49.3	7.6	14.5	1.5	1.2	28.6
Li Ro Conc.	218	11.1	2.90	6.24	64.0	24.7	0.70	0.84	2.13	58.2	9.7	17.6	2.6	2.2	32.5
Li Ro & Scav. Conc.	231	11.7	2.84	6.11	64.2	24.5	0.77	0.95	2.10	60.3	10.3	18.5	3.0	2.7	34.0

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Test: F3 **Project No.:** 13650-001 **Operator**: Yangling **Date**: 08/30/2012

Purpose: **Repeat Test F1 but Eleminating Mica Flotation**

Procedure: As outlined below.

Feed: 2kg Composite Head Sample

Grind: Stage ground to -300 µm, 15+10min

Regrind: None

Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS

Size analysis on Slime and Combined Products

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac T+C	Fuel Oil	$Fe_2(SO_4)_3$	NaOH	Na_2CO_3	Na Silicate	Na2F6Si	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
High Density Scrubbing					400				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell)	1500 RPM Raise pulp Temp to at least 25'C													8.5	24
High Intensity Condition 2 (2kg cell)	1500 RPM									700		5		8.5	
Li Rougher 1 (2kg cell)													2.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						100		100	200	200		3	1	8.5	24
Submit Scav Conc for Assay															
Only Clean Rougher Conc						120									
Li 1st Cleaner (1kg cell)								100				1	2	8.5	24
Li 2nd Cleaner (1kg cell)								50				1	2	8.5	
Li 3rd Cleaner (1kg cell)								25				1	1	8.5	
Total	0	0	0		400	220		275	450	900	0	21	9	.	

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Test 3 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F3 15A Non-Mag	269	13.7	2.91	6.26	64.4	25.4	0.96	0.80	1.10	70.6	12.0	22.3	4.44	2.63	23.0
F3 15A Mag	4.50	0.23	1.71	3.68	53.1	22.6	1.43	0.91	9.37	0.69	0.17	0.33	0.11	0.05	3.28
F3 5A Mag	12.00	0.61	1.36	2.93	48.0	21.6	0.81	0.64	14.0	1.47	0.40	0.85	0.17	0.09	13.1
F3 Li 3rd Cl Tail	21.20	1.08	1.70	3.66	64.3	23.0	3.08	1.96	1.58	3.25	0.94	1.59	1.12	0.51	2.60
F3 Li 2nd Cl Tail	38.20	1.95	1.40	3.01	66.6	21.2	3.14	2.58	1.27	4.83	1.76	2.64	2.06	1.21	3.77
F3 Li 1st Cl Tail	116.4	5.94	0.84	1.81	69.9	18.5	3.36	3.52	0.81	8.83	5.62	7.03	6.73	5.01	7.33
F3 Ro Scav Con	20.3	1.04	0.91	1.96	67.7	19.1	3.46	3.34	1.12	1.67	0.95	1.27	1.21	0.83	1.77
F3 Ro Scav Tail	1,312	67.0	0.03	0.07	77.5	12.8	3.22	5.00	0.22	4.03	70.3	54.9	72.7	80.2	22.44
F3 Slime	165	8.44	0.31	0.67	69.0	16.9	4.03	4.67	1.77	4.63	7.89	9.13	11.47	9.44	22.76
Head (calc.)	1,959	100	0.57	1.22	73.8	15.6	2.97	4.17	0.66	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,794	91.6	0.59	1.27	74.3	15.5	2.87	4.13	0.55	95.4	92.1	90.9	88.5	90.6	77.2
Spod Flot Feed	1,794	91.6	0.59	1.27	74.3	15.5	2.87	4.13	0.55	95.4	92.1	90.9	88.5	90.6	77.2

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 20A	269	13.7	2.91	6.26	64.4	25.4	0.96	0.80	1.10	70.6	12.0	22.3	4.44	2.63	23.0
Li Non Mag Final Conc. 10A	273	14.0	2.89	6.22	64.2	25.4	0.97	0.80	1.24	71.3	12.1	22.6	4.55	2.68	26.3
Li 3rd Cl Conc	285	14.6	2.83	6.08	63.5	25.2	0.96	0.80	1.77	72.8	12.5	23.5	4.72	2.77	39.3
Li 2nd Cl. Conc.	307	15.6	2.75	5.92	63.6	25.0	1.11	0.88	1.76	76.0	13.5	25.1	5.84	3.28	41.9
Li 1st Cl Conc	345	17.6	2.60	5.59	63.9	24.6	1.33	1.06	1.71	80.9	15.2	27.7	7.90	4.49	45.7
Li Ro Conc.	461	23.5	2.15	4.64	65.4	23.1	1.84	1.68	1.48	89.7	20.9	34.7	14.63	9.50	53.0
Li Ro & Scav. Conc.	481	24.6	2.10	4.53	65.5	22.9	1.91	1.75	1.46	91.3	21.8	36.0	15.84	10.33	54.8

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Test: F4 **Project No.:** 13650-001 **Operator**: Yangling **Date**: 09/17/2012

Purpose: **Similar to Test 1, with the Objective to Improve Li Recovery by combining Li Ro and Scav Concentrates & Mica Cl Tail**

Procedure: As outlined below.

Feed: 2kg Composite Head Sample
Grind: Stage ground to -300 µm, 15+10+6min
Regrind: None
Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac T**	**Fuel Oil**	**Armac C**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO3)2**	**D618**	**FA-2**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**5%**	**10%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime														7.8	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
High density Condition (Natural pH)		0	50	50								3		8.0	24
Mica Ro (2kg cell)												1	3	8.0	24
Mica Cleaner Use appropriate Cell (maybe 0.5kg is good)															
Mica Clnr #1 (0.5kg cell) 1200 RPM				0								1	3	7.8	25
Combine Mica Cl Tail with the Mica Rougher Tail															
High Density Scrubbing					275				250			10		11.0	24
Deslime	***Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 2 times***														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25°C														8.0	26
High Intensity Condition 2 (2kg cell) 1500 RPM										700		5		7.8	25
Li Rougher 1 (2kg cell)													2.5	8.0	
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	7.8	24
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc						50									
Li 1st Cleaner (1kg cell)										50		1	2	8.7	25
Li 2nd Cleaner (1kg cell)										25		1	2	7.5	
Li 3rd Cleaner (1kg cell)												1	1	7.4	25
Total	0	0	50	50	275	50			450	975	0	26	15		

Test 4 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F4, 15A Non Mag	379	19.3	2.44	5.25	65.9	23.7	1.33	1.34	0.97	84.8	17.2	28.8	8.5	6.1	28.1
F4, 15A Mag	11.4	0.6	0.80	1.72	43.5	20.4	1.06	0.68	18.8	0.8	0.3	0.7	0.2	0.1	16.4
F4, Li 3rd Cl Tail	25.3	1.3	0.30	0.65	75.8	15.5	3.47	4.31	0.61	0.7	1.3	1.3	1.5	1.3	1.2
F4, Li 2nd Cl Tail	41.0	2.1	0.33	0.71	75.8	15.4	3.33	4.70	0.41	1.2	2.1	2.0	2.3	2.3	1.3
F4, Li 1st Cl Tail	141	7.2	0.26	0.56	74.9	14.7	3.32	4.90	0.31	3.4	7.3	6.7	7.9	8.4	3.4
F4, Li Ro Scav Tail	1,117	57.1	0.02	0.04	78.3	13.0	3.32	5.10	0.12	1.9	60.2	46.7	62.3	69.0	10.3
F4, Slime 2	55.7	2.8	0.54	1.16	68.8	17.2	3.33	4.38	3.05	2.8	2.6	3.1	3.1	3.0	13.0
F4, Mica Con	89.8	4.6	0.15	0.32	67.4	18.7	5.24	4.00	1.65	1.2	4.2	5.4	7.9	4.4	11.4
F4, Slime 1	97.4	5.0	0.35	0.75	70.7	17.0	3.89	4.58	2.01	3.1	4.7	5.3	6.4	5.4	15.0
Head (calc.)	1,958	100	0.56	1.20	74.2	15.9	3.04	4.22	0.67	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,860	95.0	0.58	1.25	75.1	15.7	3.03	4.20	0.52	96.9	95.3	94.7	93.6	94.6	85.0
Spod Flot Feed	1,715	87.6	0.60	1.30	75.6	15.5	2.90	4.20	0.40	92.9	88.5	86.2	82.6	87.3	60.6

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	379	19.3	2.44	5.25	65.9	23.7	1.33	1.34	0.97	84.8	17.2	28.8	8.5	6.1	28.1
Li 3rd Cl Conc	390	19.9	2.39	5.15	65.2	23.6	1.32	1.32	1.49	85.6	17.5	29.6	8.7	6.2	44.5
Li 2nd Cl. Conc.	415	21.2	2.26	4.87	65.9	23.1	1.45	1.50	1.44	86.3	18.8	30.8	10.1	7.6	45.7
Li 1st Cl Conc	456	23.3	2.09	4.50	66.8	22.4	1.62	1.79	1.35	87.6	21.0	32.9	12.4	9.9	47.0
Li Ro and Scav. Conc.	597	30.5	1.66	3.57	68.7	20.6	2.02	2.52	1.10	90.9	28.2	39.5	20.3	18.3	50.4

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Test: F5 **Project No.:** 13650-001 **Operator**: Yangling **Date**: 9,17,2012

Purpose: **Similar to Test 4 but with Finer Grinding Size, Stage Grinding to -65 M**

Procedure: As outlined below.

Feed: 2kg Composite Head Sample

Grind: Stage ground to -210 µm (65 Mesh), 16+11+6

Regrind: None

Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac T	Fuel Oil	Armac C	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime														7.8	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
High density Condition (Natural pH)		0	50	50								3		7.8	24
Mica Ro (2kg cell)												1	3	7.7	24
Mica Cleaner Use appropriate Cell (maybe 0.5kg is good)															
Mica Clnr #1 (0.5kg cell) 1200 RPM				0								1	3	7.8	22
Combine Mica Cl Tail with the Mica Rougher Tail															
High Density Scrubbing					260				250			10		11.0	24
Deslime	***Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 2 times***														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	24
High Intensity Condition 2 (2kg cell) 1500 RPM										700		5		7.0	25
Li Rougher 1 (2kg cell)													2.5	8.0	28
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	7.8	24
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc.						50									
Li 1st Cleaner (1kg cell)										50		1	2	8.3	24
Li 2nd Cleaner (1kg cell)										25		1	2	8.0	25
Li 3rd Cleaner (1kg cell)												1	1	7.8	25
Total	0	0	50	50	260	50			450	975	0	26	15		

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Test 5 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F5, 15A Non Mag	339	17.4	2.75	5.92	65.6	24.6	1.19	1.16	0.98	82.5	15.2	27.1	6.8	4.8	25.4
F5, 15A Mag	11.9	0.6	1.01	2.17	46.5	21.5	1.08	0.61	15.3	1.1	0.4	0.8	0.2	0.1	13.9
F5, Li 3rd Cl Tail	19.9	1.0	0.76	1.64	72.5	17.1	3.08	3.57	0.90	1.3	1.0	1.1	1.0	0.9	1.4
F5, Li 2nd Cl Tail	23.7	1.2	0.38	0.82	76.1	14.9	3.13	4.34	0.53	0.8	1.2	1.1	1.3	1.3	1.0
F5, Li 1st Cl Tail	81	4.1	0.19	0.41	78.0	13.7	3.11	4.70	0.27	1.4	4.3	3.6	4.2	4.6	1.7
F5, Li Ro Scav Tail	1,139	58.5	0.01	0.02	79.7	12.8	3.23	5.13	0.09	1.0	61.9	47.4	62.3	71.1	7.9
F5, Slime 2	35.9	1.8	0.63	1.36	67.5	17.9	2.88	4.01	4.17	2.0	1.7	2.1	1.8	1.8	11.5
F5, Mica Con	158.9	8.2	0.40	0.86	70.3	17.8	4.96	4.06	1.22	5.6	7.6	9.2	13.4	7.8	14.8
F5, Slime 1	139.7	7.2	0.35	0.75	70.8	16.6	3.79	4.54	2.10	4.3	6.7	7.5	9.0	7.7	22.5
Head (calc.)	1,949	100	0.58	1.25	75.2	15.8	3.03	4.22	0.67	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,809	92.8	0.59	1.26	75.2	15.7	3.02	4.19	0.49	95.7	93.3	92.5	91.0	92.3	77.5
Spod Flot Feed	1,615	82.8	0.61	1.30	75.9	15.5	2.82	4.21	0.36	88.1	84.0	81.2	75.9	82.7	51.2

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	339	17.4	2.75	5.92	65.6	24.6	1.19	1.16	0.98	82.5	15.2	27.1	6.8	4.8	25.4
Li 3rd Cl Conc	351	18.0	2.69	5.79	65.0	24.5	1.19	1.14	1.47	83.6	15.6	27.9	7.1	4.9	39.4
Li 2nd Cl. Conc.	371	19.0	2.59	5.57	65.4	24.1	1.29	1.27	1.44	84.9	16.5	29.1	8.1	5.7	40.8
Li 1st Cl Conc	395	20.3	2.45	5.28	66.0	23.5	1.40	1.46	1.38	85.7	17.8	30.2	9.3	7.0	41.7
Li Ro and Scav. Conc.	475	24.4	2.07	4.46	68.0	21.9	1.69	2.01	1.19	87.0	22.1	33.8	13.6	11.6	43.4

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Test: F6 **Project No.:** 13650-001 **Operator**: Yangling **Date**: oct5,2012

Purpose: **Similar to Test F5 but with Finer Grinding Size, Stage Grinding to -100 M**

Procedure: As outlined below.

Feed: 2kg Composite Head Sample

Grind: Stage ground to -150 µm (100 Mesh), 18+13+8min

Regrind: None

Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A

Stage	Reagents added, g/t										Time, minutes				
	H2SO4	Armac T	Fuel Oil	Armac C	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime														7.8	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
High density Condition (Natural pH)		0	50	50								3		7.3	24
Mica Ro (2kg cell)												1	3	7.3	24
Mica Cleaner Use appropriate Cell (maybe 0.5kg is good)															
Mica Clnr #1 (0.5kg cell) 1200 RPM				0								1	3	7.8	22
Combine Mica Cl Tail with the Mica Rougher Tail															
High Density Scrubbing					500				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C															24
High Intensity Condition 2 (2kg cell) 1500 RPM										700		5		6.8	25
Li Rougher 1 (2kg cell)													2.5	7.6	25
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	7.8	26
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc						50									
Li 1st Cleaner (1kg cell)										25		1	2	8.0	24
Li 2nd Cleaner (1kg cell)												1	2	7.8	25
Li 3rd Cleaner (1kg cell)												1	1	7.2	25
Total	0	0	50	50	500	50			450	925	0	26	15		

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Test 6 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F6 15A Non-Mag	438	22.7	1.91	4.11	71.2	19.5	1.49	1.71	0.81	78.8	21.8	28.3	11.0	9.20	25.7
F6 15A Mag	26.0	1.35	2.27	4.89	59.0	23.2	0.74	0.72	7.12	5.56	1.07	2.00	0.32	0.23	13.4
F6 Li 3rd Cl Tail	68.9	3.57	0.22	0.47	80.0	12.3	2.80	4.05	0.28	1.43	3.84	2.81	3.25	3.43	1.40
F6 Li 2nd Cl Tail	165	8.54	0.08	0.16	81.3	11.7	2.82	4.24	0.15	1.18	9.33	6.38	7.82	8.57	1.79
F6 Li 1st Cl Tail	315	16.3	0.06	0.14	78.6	13.3	3.27	5.12	0.08	1.90	17.3	13.9	17.4	19.8	1.82
F6 Li Ro. Scav. Tail	548	28.4	0.01	0.01	74.9	14.4	3.80	5.85	0.09	0.31	28.6	26.2	35.1	39.4	3.57
F6 SLime 2	64.9	3.37	0.59	1.27	69.0	16.9	3.07	4.22	3.50	3.60	3.12	3.63	3.36	3.36	16.4
F6 Mica Con	153	7.95	0.15	0.32	70.6	17.3	4.80	4.14	1.13	2.16	7.55	8.79	12.4	7.79	12.5
F6 Slime 1	148	7.70	0.36	0.77	71.6	16.3	3.76	4.53	2.18	5.03	7.41	8.02	9.40	8.25	23.4
Head (calc.)	1,928	100	0.55	1.19	74.4	15.7	3.08	4.23	0.72	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,779	92.3	0.59	1.26	75.1	15.7	3.02	4.19	0.49	95.0	92.6	92.0	90.6	91.7	76.6
Spod Flot Feed	1,561	81.0	0.63	1.35	75.8	15.5	2.85	4.19	0.34	89.2	81.9	79.6	74.8	80.6	47.6

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	438	22.7	1.91	4.11	71.2	19.5	1.49	1.71	0.81	78.8	21.8	28.3	11.0	9.20	25.7
Li 3rd Cl Conc	464	24.1	1.93	4.15	70.5	19.7	1.45	1.65	1.16	84.4	22.8	30.3	11.3	9.43	39.1
Li 2nd Cl. Conc.	533	27.7	1.71	3.68	71.7	18.8	1.62	1.96	1.05	85.8	26.7	33.1	14.6	12.9	40.5
Li 1st Cl Conc	698	36.2	1.32	2.85	74.0	17.1	1.91	2.50	0.84	87.0	36.0	39.5	22.4	21.4	42.3
Li Ro. & Scav. Conc.	1,013	52.5	0.93	2.01	75.4	15.9	2.33	3.32	0.60	88.9	53.3	53.4	39.8	41.2	44.1

Test: F7 **Project No.:** 13650-001 **Operator**: Yangling **Date**: otc18, 2012

Purpose: Repeat Test F6 but with Lower Collector Dossage

Procedure: As outlined below.

Feed: 2kg Composite Head Sample

Grind: Stage ground to -150 µm (100 Mesh), 18+13+8min

Regrind: None

Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac T	Fuel Oil	Armac C	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime														7.8	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
High density Condition (Natural pH)		0	50	50								3		7.8	24
Mica Ro (2kg cell)												1	3	7.6	24
Mica Cleaner Use appropriate Cell (maybe 0.5kg is good)															
Mica Clnr #1 (0.5kg cell) 1200 RPM				0								1	3	7.5	22
Combine Mica Cl Tail with the Mica Rougher Tail															
High Density Scrubbing					270				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 5 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.4	24
High Intensity Condition 2 (2kg cell) 1500 RPM										450		5		6.9	24
Li Rougher 1 (2kg cell)													2.5	7.1	
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	6.5	24
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc						50									
Li 1st Cleaner (1kg cell)										25		1	2	8.4	24
Li 2nd Cleaner (1kg cell)												1	2	7.5	24
Li 3rd Cleaner (1kg cell)												1	1	7.3	25
Total	0	0	50	50	270	50			450	675	0	26	15	.	

Stage	Condition	Mica Rougher	Li Rougher	1st -3rd Clnr	
Flotation Cell	2kg cell- High Dens.	2kg cell	2kg cell	1kg cell	
Speed: rpm	1500	1800	1800	1500	

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Test 7 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F7 15A mag	291	14.9	3.07	6.61	64.9	25.3	0.72	0.81	1.46	82.0	12.9	24.0	3.51	2.85	24.0
F7 15A non-mag	16.2	0.83	1.78	3.83	51.0	22.8	0.61	0.50	11.9	2.65	0.57	1.20	0.17	0.10	10.9
F7 Li 3rd cl Tail	9.00	0.46	1.52	3.27	67.5	20.6	2.65	2.93	1.72	1.26	0.42	0.60	0.40	0.32	0.88
F7 Li 2nd cl Tail	14.2	0.73	0.84	1.81	69.6	17.9	3.38	3.71	1.20	1.10	0.68	0.83	0.81	0.64	0.96
F7 Li 1st cl Tail	58.4	3.00	0.26	0.56	74.8	14.9	3.51	4.47	0.72	1.40	3.00	2.84	3.44	3.17	2.38
F7 Li Ro Scav Con	69.0	3.54	0.12	0.26	73.0	16.4	4.40	3.36	1.22	0.76	3.46	3.69	5.09	2.81	4.76
F7 Ro Scav. Tail	1,201	61.7	0.01	0.02	78.9	13.0	3.36	5.14	0.22	0.88	65.1	50.9	67.7	74.8	14.9
F7 slime 2	104	5.33	0.47	1.01	70.7	16.3	3.35	4.43	2.75	4.48	5.04	5.51	5.83	5.58	16.1
F7 mica con	55.4	2.84	0.21	0.45	65.2	19.7	5.34	3.77	2.33	1.07	2.48	3.56	4.96	2.53	7.30
F7 slime 1	130	6.68	0.37	0.80	71.1	16.3	3.70	4.54	2.41	4.42	6.35	6.91	8.08	7.16	17.7
Head (calc.)	1,948	100	0.56	1.20	74.8	15.8	3.06	4.23	0.91	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,818	93.3	0.58	1.26	75.2	15.8	3.03	4.19	0.52	95.6	93.6	93.1	91.9	92.8	82.3
Spod Flot Feed	1,658	85.1	0.60	1.30	75.8	15.6	2.93	4.19	0.40	90.0	86.1	84.0	81.1	84.7	58.8

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	291	14.9	3.07	6.61	64.9	25.3	0.72	0.81	1.46	82.0	12.9	24.0	3.51	2.85	24.0
Li 3rd Cl Conc	307	15.8	3.00	6.46	64.2	25.2	0.71	0.79	2.01	84.6	13.5	25.2	3.68	2.95	34.9
Li 2nd Cl. Conc.	316	16.2	2.96	6.37	64.3	25.0	0.77	0.85	2.00	85.9	13.9	25.8	4.08	3.27	35.8
Li 1st Cl Conc	330	16.9	2.87	6.17	64.5	24.7	0.88	0.98	1.97	87.0	14.6	26.6	4.88	3.91	36.7
Li Ro Conc.	388	19.9	2.48	5.33	66.0	23.3	1.28	1.50	1.78	88.4	17.6	29.4	8.32	7.08	39.1
Li Ro Conc. & Escav. Conc.	457	23.5	2.12	4.57	67.1	22.2	1.75	1.78	1.70	89.1	21.1	33.1	13.4	9.89	43.9

Glen Eagle Flot Tests MA.xls F7
updated 01/08/2013

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F **Test: F8** **Project No.:** 13650-001 **Operator**: Yangling **Date**: Nov5,2012

Purpose: Repeat Test F7 but without Mica Flotation

Procedure: As outlined below.

Feed: 2kg Composite Head Sample
Grind: Stage ground to -150 µm (100 Mesh), 18+13+8min
Regrind: None
Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac T	Fuel Oil	Armac C	NaOH	Na_2CO_3	Na Silicate	F6SiNa2	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
High Density Scrubbing					275				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 5 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell)	1500 RPM Raise pulp Temp to at least 25'C													8.1	24
High Intensity Condition 2 (2kg cell)	1500 RPM									450		5		8.0	24
Li Rougher 1 (2kg cell)													2.5	7.3	25
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	6.7	24
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc						50									
Li 1st Cleaner (1kg cell)										25		1	2	7.8	24
Li 2nd Cleaner (1kg cell)												1	2	7.2	24
Li 3rd Cleaner (1kg cell)												1	1	7.8	24
Li 4th Cleaner (1kg cell)								50				1	1	6.5	25
Total	0	0	0	0	275	50			450	675	0	22	10	.	

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Test 8 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F8 15A Non Mag	305	16.1	2.99	6.44	64.8	25.1	0.76	0.82	1.06	85.0	14.2	25.8	4.1	3.1	23.8
F8 15A Mag	10.8	0.57	1.05	2.26	45.6	21.3	0.75	0.57	16.3	1.06	0.4	0.8	0.1	0.1	13.0
F8 Li 4th Cl Tail	7.40	0.39	1.31	2.82	69.1	19.9	2.81	3.11	1.48	0.90	0.4	0.5	0.4	0.3	0.8
F8 Li 3rd Cl Tail	10.1	0.53	1.06	2.28	69.1	19.3	3.26	3.29	1.31	1.00	0.5	0.7	0.6	0.4	1.0
F8 Li 2nd Cl Tail	14.5	0.77	0.59	1.27	70.6	18.1	3.81	3.80	1.55	0.80	0.7	0.9	1.0	0.7	1.7
F8 Li 1st Cl Tail	5.52	0.29	0.23	0.50	73.6	16.3	4.00	4.38	0.63	0.12	0.3	0.3	0.4	0.3	0.3
F8 Li Ro Scav Con	58.0	3.07	0.25	0.54	67.1	20.2	5.30	3.32	1.34	1.35	2.8	4.0	5.4	2.4	5.7
F8 Li Ro Scav Tail	1,230	65.0	0.01	0.02	77.0	12.8	3.33	5.07	0.14	1.15	68.0	53.1	71.8	78.3	12.7
F8 Slime 1	250	13.2	0.37	0.80	71.3	16.7	3.74	4.57	2.23	8.61	12.8	14.0	16.4	14.3	41.0
Head (calc.)	1,891	100	0.57	1.22	73.7	15.7	3.02	4.21	0.72	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,641	86.8	0.60	1.29	75.3	15.6	2.97	4.16	0.40	91.4	87.2	86.0	83.6	85.7	59.0
Spod Flot Feed	1,641	86.8	0.60	1.29	75.3	15.6	2.97	4.16	0.40	91.4	87.2	86.0	83.6	85.7	59.0

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	305	16.1	2.99	6.44	64.8	25.1	0.76	0.82	1.06	85.0	14.2	25.8	4.1	3.14	23.8
Li 4th Cl Conc	316	16.7	2.92	6.29	64.1	25.0	0.76	0.81	1.58	86.1	14.5	26.6	4.2	3.22	36.8
Li 3rd Cl Conc	323	17.1	2.89	6.21	64.3	24.9	0.81	0.86	1.58	87.0	14.9	27.1	4.6	3.51	37.6
Li 2nd Cl. Conc.	333	17.6	2.83	6.09	64.4	24.7	0.88	0.94	1.57	88.0	15.4	27.7	5.1	3.9	38.6
Li 1st Cl Conc	348	18.4	2.74	5.89	64.7	24.4	1.00	1.06	1.57	88.8	16.1	28.6	6.1	4.6	40.3
Li Ro Conc.	353	18.7	2.70	5.81	64.8	24.3	1.05	1.11	1.56	88.9	16.4	28.9	6.5	4.9	40.5
Li Ro Conc. & Scav. Conc.	411	21.8	2.35	5.07	65.1	23.7	1.65	1.42	1.52	90.2	19.2	32.9	11.9	7.3	46.3
Li Ro Tail.	1,288	68.1	0.02	0.04	76.6	13.1	3.42	4.99	0.19	2.50	70.8	57.0	77.1	80.8	18.4

Test: F9 **Project No.:** 13650-001 **Operator**: Yangling **Date**:

Purpose: Repeat Test F8 but Replacing FA2 with Aero 704 and Aero 845 (Cytec)

Procedure: As outlined below.

Feed: 2kg Composite Head Sample
Grind: Stage ground to -150 µm (100 Mesh), 18+13+8min
Regrind: None
Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A
Pleas Ask for combined Size Aanalysis
Please Sumbit Slime for Cyclosizer

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Aero 704	Aero 845	Armac C	NaOH	Na_2CO_3	Na Silicate	F6SiNa2	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
High Density Scrubbing					275				250			10		11.0	24
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 5 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.1	24
High Intensity Condition 2 (2kg cell) 1500 RPM		300	150									5		8.0	24
Li Rougher 1 (2kg cell)													2.5	6.8	25
High Density Conditioning															
Li Rougher Scav 1		150	50						200			3	1	7.3	24
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc.						50									
Li 1st Cleaner (1kg cell)												1	2	6.8	24
Li 2nd Cleaner (1kg cell)												1	2	7.1	24
Li 3rd Cleaner (1kg cell)												1	1	7.2	24
Reverse floation				20								2	5	2.0	
Total	0	450	200	0	275	50			450	0	0	23	14	.	

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Test 9 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F9 Spodumen Conc.	293.6	15.0	2.93	6.31	64.7	25.0	0.72	0.82	1.18	81.8	13.2	24.2	3.6	2.8	22.8
F9 Siliate Conc.	2.6	0.1	1.80	3.87	49.9	23.9	3.24	1.77	2.86	0.4	0.1	0.2	0.1	0.1	0.5
F9 15A Mag	10.6	0.5	0.74	1.59	42.9	20.6	0.78	0.64	20.1	0.7	0.3	0.7	0.1	0.1	14.0
F9 Li 3rd Cl Tail	13.5	0.7	1.48	3.19	66.9	20.6	2.80	2.72	1.60	1.9	0.6	0.9	0.6	0.4	1.4
F9 Li 2nd Cl Tail	20.1	1.0	0.89	1.92	69.3	19.2	3.66	3.37	1.18	1.7	1.0	1.3	1.2	0.8	1.6
F9 Li 1st Cl Tail	57.8	3.0	0.28	0.60	72.2	16.2	3.99	4.19	0.69	1.5	2.9	3.1	3.9	2.8	2.6
F9 Li Ro Scav Con	64	3.3	0.40	0.86	69.0	18.3	4.60	3.48	1.08	2.4	3.1	3.9	4.9	2.6	4.5
F9 Li Ro Scav Tail	1,256	64.2	0.02	0.03	77.3	12.8	3.34	5.27	0.18	1.8	67.4	53.0	70.6	77.4	14.9
F9 Slime	237.4	12.1	0.34	0.73	69.9	16.2	3.73	4.70	2.41	7.7	11.5	12.7	14.9	13.0	37.7
Head (calc.)	1,956	100	0.54	1.16	73.7	15.5	3.04	4.38	0.78	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,718	87.9	0.60	1.29	75.4	15.7	2.98	4.17	0.42	92.3	88.5	87.3	85.1	87.0	62.3
Spod Flot Feed	1,718	87.9	0.60	1.29	75.4	15.7	2.98	4.17	0.42	92.3	88.5	87.3	85.1	87.0	62.3

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	294	15.0	2.93	6.31	64.7	25.0	0.72	0.82	1.18	81.8	13.2	24.2	3.6	2.81	22.8
Li 3rd Cl Conc	296	15.1	2.92	6.29	64.6	25.0	0.74	0.83	1.19	82.2	13.3	24.4	3.7	2.87	23.3
Li 2nd Cl. Conc.	307	15.7	2.84	6.12	63.8	24.8	0.74	0.82	1.85	83.0	13.6	25.1	3.8	2.9	37.3
Li 1st Cl Conc	320	16.4	2.79	6.00	64.0	24.7	0.83	0.90	1.84	84.9	14.2	26.1	4.5	3.4	38.7
Li Ro Conc.	340	17.4	2.68	5.76	64.3	24.3	1.00	1.05	1.80	86.6	15.2	27.3	5.7	4.2	40.3
Li Ro Conc. & Scav. Conc.	398	20.4	2.33	5.01	65.4	23.2	1.43	1.50	1.64	88.1	18.1	30.4	9.6	7.0	42.9
Li Ro Tail.	1,320	67.5	0.04	0.08	78.1	13.3	3.45	4.99	0.17	4.22	70.4	56.9	75.5	80.0	19.4

Test: F10 **Project No.:** 13650-001 **Operator**: Yangling **Date**: Nov13,2012

Purpose: **Repeat Test F8 but with Cold Water**

Procedure: As outlined below.
Feed: 2kg Composite Head Sample
Grind: Stage ground to -150 µm (100 Mesh), 18+13+8min
Regrind: None
Conditions: Final concentrate to be mixed with H2SO4 and then pass through 5 A WHIMS and repass the Non-mag at 15A WHIMS
Combine all the mag concentrates and repass through WHIMS at 15A

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac T	Fuel Oil	Armac C	NaOH	Na_2CO_3	Na Silicate	F6SiNa2	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	5%	10%		1%	5%	100%	Stage Grinding				
High Density Scrubbing					175				250			10		11.0	6-10
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 5 minutes decant by syphon 2 times*														
High Intensity Condition 1 (2kg cell)	1500 RPM keep temperature 6-10°C													8.5	8-9
High Intensity Condition 2 (2kg cell)	1500 RPM									450		5		8.5	8-10
Li Rougher 1 (2kg cell)													2.5	6.5	8-10
High Density Conditioning															
Li Rougher Scav 1									200	200		3	1	6.5	6-10
Combine Li Scav. Conc. and Li Rougher concentrate															
Only Clean Rougher Conc.						50									
Li 1st Cleaner (1kg cell)										25		1	2	6.9	8-10
Li 2nd Cleaner (1kg cell)												1	2	7.1	8-10
Li 3rd Cleaner (1kg cell)												1	1	7.8	8-10
Total	0	0	0	0	175	50			450	675	0	21	9		

Test 10 Metallurgical Balance

Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F10 15A non-mag	327	16.7	2.91	6.26	65.3	24.3	0.90	1.02	1.10	84.5	14.7	26.1	5.06	4.07	19.0
F10 15A mag	19.3	0.98	1.33	2.86	43.0	18.7	0.74	0.57	27.6	2.28	0.57	1.19	0.25	0.13	28.2
F10 Li 3rd Cl Tail	7.80	0.40	0.53	1.14	73.5	16.1	3.22	4.09	1.15	0.37	0.39	0.41	0.43	0.39	0.48
F10 Li 2nd Cl Tail	15.6	0.80	0.30	0.65	73.9	15.4	3.39	4.42	0.74	0.42	0.79	0.79	0.91	0.84	0.61
F10 Li 1st Cl Tail	45.7	2.33	0.12	0.26	74.2	14.3	3.52	4.77	0.42	0.49	2.33	2.15	2.77	2.66	1.02
F10 Li Ro Scav Con	63.2	3.22	0.23	0.50	70.7	17.4	4.02	4.57	0.77	1.29	3.08	3.61	4.37	3.53	2.58
F10 Li Ro Scav Tail	1,228	62.6	0.02	0.04	77.7	13.0	3.34	4.94	0.29	1.86	65.7	52.4	70.6	74.1	18.9
F10 Slime	255	13.0	0.39	0.84	71.2	16.0	3.57	4.56	2.16	8.84	12.5	13.4	15.7	14.2	29.2
Head (calc.)	1,961	100	0.57	1.23	74.1	15.5	2.96	4.17	0.96	100	100	100	100	100	100
Head (Dir.)			0.57	1.23	74.9	15.8	3.08	4.22	0.59						
Flotation Feed	1,706	87.0	0.60	1.29	75.3	15.7	2.99	4.16	0.48	91.2	87.5	86.6	84.3	85.8	70.8
Spod Flot Feed	1,706	87.0	0.60	1.29	75.3	15.7	2.99	4.16	0.48	91.2	87.5	86.6	84.3	85.8	70.8

Combined	Weight		Assays %							Distribution %					
Product	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Non Mag Final Conc. 15A	327	16.7	2.91	6.26	65.3	24.3	0.90	1.02	1.10	84.5	14.7	26.1	5.06	4.07	19.0
Li 3rd Cl Conc	346	17.6	2.82	6.07	64.1	24.0	0.89	0.99	2.58	86.7	15.2	27.2	5.30	4.21	47.3
Li 2nd Cl. Conc.	354	18.0	2.77	5.97	64.3	23.8	0.94	1.06	2.55	87.1	15.6	27.6	5.73	4.60	47.7
Li 1st Cl Conc	369	18.8	2.67	5.74	64.7	23.5	1.05	1.20	2.47	87.5	16.4	28.4	6.64	5.44	48.3
Li Ro Conc.	415	21.2	2.39	5.14	65.7	22.4	1.32	1.60	2.25	88.0	18.8	30.6	9.41	8.10	49.4
Li Ro Conc. & Scav. Conc.	478	24.4	2.10	4.52	66.4	21.8	1.68	1.99	2.05	89.3	21.8	34.2	13.8	11.6	51.9

3.2. Bond Ball Mill Work Index

The master composite was submitted for Bond ball mill work index (BWI) determination as per the standard procedure using a product D_{100} size of -150 µm. A ball mill work index (BWI) of 15.6 kWh/t (metric) was obtained for the head composite. Details of the test are included in Appendix B. The results are compared to the SGS database in Figure 4. The sample was found to be of medium hardness with a percentile of 64%.

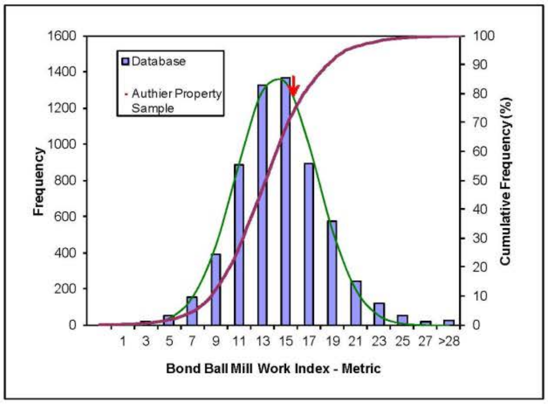

Figure 4: Head Composite Bond Ball Mill Work Index Comparison to SGS Database

4. Flotation Test Results

All of the flotation tests were carried out using a Denver D1/D2 flotation cell and D12 flotation machine. All stage grinding was performed in a 2 kg mill. Stainless steel rods were used as the grinding media in all tests, and the closing size for each test is shown in the flotation test details. Scrubbing was performed in a Denver flotation cell at pH of around 11 adjusted with NaOH and in the presence of lignin sulfonate (D618) for 10 minutes. Slime separation after scrubbing was done by settling and decanting in an acrylic container. The above described equipment used is illustrated Figure 5.

Flotation tests have shown that the Authier ore sample is amenable to flotation beneficiation to produce concentrate for the hydromet operation. Furthermore, high grade concentrate can be produced but the iron content of spodumene, most probably in the solid solution, was elevated making the concentrate unattractive for the ceramic industry. Flotation tests were mainly designed to generate spodumene concentrate higher than 6.0% Li_2O and above 80% recovery. Detailed descriptions of all the flotation tests are provided in Appendix C.