NI 43-101 Technical Report on the Preliminary Economic Assessment

Georgia Lake Lithium Properties Beardmore, Ontario, Canada

for Rock Tech Lithium Inc., Canada

To:	Rock Tech Lithium Inc., 600 – 777 Hornby Street, Vancouver, British Columbia V6Z 1S4; Canada
Date of Report:	30th October 2018
Effective Date:	30th October 2018
Project No:	8115835263

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This Report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment Georgia Lake Lithium, Beardmore, Ontario, Canada", with an effective date of 30th October 2018 and a submission date of 30th October 2018, was prepared and signed by the following Qualified Persons.

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As a co-author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment Georgia Lake Lithium Properties Beardmore, Ontario, Canada" dated 30th October 2018. I, Keith McCandlish do hereby certify that:

1. I am employed by, and carried out this assignment for
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3. I have more than 40 years of experience as an exploration and resource geologist in the evaluation of mineral projects for all levels of feasibility and due diligence status, dealing with a variety of commodities, viz. metals (ferrous, base and precious metals), energy (shale oil, lignite and hard coal), pegmatite exploration and deposit evaluation in deferent project stages.

4. I have read the definitions of "qualified person" as set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as de fined in NI43-101) and past relevant work experience, I fulfil the requirements to be a qualified person for the purpose of NI43-101.

5. I visited the Georgia Lake project site on 11th and 12th September 2018.

6. I am responsible for the Peer Review of all sections of this report.

7. I am independent of the Rock Tech Lithium Inc., as described in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the mineral property that is subject of the Technical Report.

9. I have read the NI 43-101 and the portions of the technical report for which I am responsible have been prepared in compliance with the instrument.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Signing Date: 30th October 2018

Effective Date of report: 30th October 2018.

Keith McCandlish

STATEMENT OF AUTHORSHIP

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4. I have more than 30 years of experience as an exploration and resource geologist in the evaluation of mineral projects for all levels of feasibility and due diligence status, dealing with a variety of commodities, viz. metals (ferrous, base and precious metals), energy (shale oil, lignite and hard coal), pegmatite exploration and deposit evaluation in deferent project stages.
5. I have read the definitions of "qualified person" as set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfil the requirements to be a qualified person for the purpose of NI43-101.
6. I visited the Georgia Lake project site on October 24th, 25th and 26th 2017, July 12th until the 18th 2018, and September 10th until the 14th, 2018
7. I am responsible for all sections of this report.
8. I am independent of the Rock Tech Lithium Inc., as described in Section 1.5 of NI 43-101.
9. I have had no prior involvement with the mineral property that is subject of the Technical Report.
10. I have read the NI 43-101 and the portions of the technical report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

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IMPORTANT NOTICE

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GLOSSARY - UNITS OF MEASURE

UNIT	DESCRIPTION
%	Percentage
mm	Millimetre
m	Metre
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic metres per hour
t/h	Tonnes per hour
t/m^3	Tonne per cubic metres
Kg	Kilograms)
t	Metric tonnes
Mt	Million metric tons
Mtpa	Million metric tonnes per annum
mtpy	Million metric tonnes per year
m/min	Metres per minute
m/s	Metres per second
MW	Megawatt
ha	hectare
km^2	Square kilometres (metric)
l/s	Litres per second

GLOSSARY - ABBREVIATIONS AND ACRONYMS

TERM	ABBREVIATION
3D	Three dimensional
Al	Aluminium
asl	Above sea level
CAD	Canadian Dollar
CAPEX	Capital expenditure
CCIC	Caracle Creek International Consulting Inc.
conc	concentrate
CON	Conway pegmatite
Cs	Cesium
DMT	DMT GmbH & Co.KG
ESIA	Environmental & Social Impact Assessment
EIA	Environmental Impact Assessment
EUR	Euro
HAR	Harricana pegmatite
LCT-pegmatites	Lithium-Cesium-Tantalum-pegmatites
Li	Lithium
Li_2O	Lithium oxide
LIN	Line 60 Pegmatite
LoM	Life of Mine
MZN	Main Zone North pegmatite
MZSW	Main Zone South West pegmatite
NI 43-101	National Instrument 43-101 Standard of Disclosure for Mineral Projects, Canada
NSPA	Northern Spodumene Pegmatite Area
NQ drill holes	drill holes with diameter: outside 75,7 mm; inside 47,6 mm
O	Oxygen
MNDM	Ministry of Northern Development and Mines, Ontario
Rock Tech	Rock Tech Lithium Inc.
RoM	Run of Mine
SG	Specific gravity
SGS	SGS Group in former times *Société Générale de Surveillance*
SSPA	Southern Spodumene Pegmatite Area
Ta	Tantalum
USD	United States Dollars.

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SUMMARY

INTRODUCTION

DMT GmbH & Co. KG ("DMT") of Essen, North-Rhine Westphalia, Germany was contracted by Rock Tech Lithium Inc. ("Rock Tech") of Vancouver, British Columbia, Canada, to review the Georgia Lake Lithium Project (the "Property"), and prepare a Preliminary Economic Assessment (the "PEA"), compliant with National Instrument 43-101 ("NI43-101"). The report should be based on the available documentation inter alia DMT's Independent Technical Report on the updated the mineral resources based on latest data acquired.

The objective of this PEA is to demonstrate the economic potential for producing a lithium ion battery preproduct from the Rock Tech Lithium deposit.

The economic analysis contained in this Technical Report is based mainly on measured/indicated and a small portion of inferred mineral resources and is preliminary in nature. There is no certainty that the PEA will be realized.

The technical information and economic parameters used to prepare this Technical Report and PEA are current as of September/October 2018. This Technical Report was prepared under the supervision of qualified persons of DMT.

For this report all relevant claims and licenses, which belong to Rock Tech were used for the PEA. Professional Geologist Keith McCandlish and EurGeol K.-S. Peters, as qualified person, visited the project verifying results, operation procedures and the execution of the work in the field.

The report is addressed to Rock Tech Lithium Inc. (the issuer). DMT GmbH & Co. KG is an independent consulting firm based in Essen, Germany.

INTERPRETATIONS AND CONCLUSIONS

The deposit was evaluated previously in 1950ies as a potential source of the lithium mineral. While this market remains an opportunity, lithium ion battery technology has developed as the energy storage solution of choice for a variety of commercial applications and this has resulted in a significant increase in demand, and projected demand, for battery materials.

The parameters used in this Preliminary Economic Assessment include developing a 500,000 tpy open-pit mine, using diesel hydraulic equipment and operated by a contractor, development of an underground mine operation to reach a 1,000,000 tpy mine production and the construction of a processing plant at the mine site (crushing, grinding, flotation circuits) with a nominal capacity of 150t/h of ore at higher 90% availability

DMT examined the technical and economic aspects of the Project within the level of precision of a Preliminary Economic Assessment. As it stands, the Project contains an economic mineral resource based on mainly measured/indicated resources and a small portion of inferred resources ensuring a life of mine of 11 years.

Consequently, DMT concludes that the Project is technically feasible as well as economically viable. The values obtained for NPV, IRR and the payback period show that the Project is profitable. Table 1 presents the

main criteria of the Project. The authors of this Technical Report consider the Project to be sufficiently robust to warrant moving it to the Prefeasibility or even Feasibility level.

Georgia Lake Project	Unit	Value
LoM	Years	11
Production (diluted)	Mio t	9.6
Open pit	Mio t	2.7
Stripping Ratio	t:t	6:1
Underground	Mio t	6.9
Annual mine production	Mio t/a	0.87
Average Feed Grade (diluted)	%	0.87
Plant Recovery	%	78.00
Spodumene Concentrate Grade	%	6.20
Total Spodumene Concentrate	000 t	1,056
Annual Spodumene Concentrate	000 t/a	96
LoM OPEX		
Mining	Mio CAD	224.3
Processing	Mio CAD	140.4
G&A	Mio CAD	27.4
Other costs	Mio CAD	27.3
Total OPEX	Mio CAD	419.4
CAPEX		
Initial Capital	Mio CAD	65.3
Working Capital	Mio CAD	3.0
Total Pre-Production Costs	Mio CAD	68.3
Sustaining Capital	Mio CAD	62.0
Closure Costs	Mio CAD	3.0
Total CAPEX	Mio CAD	133.3
Royalty Rate (% of Revenue-OPEX)	%	1.50
Average LoM Spodumene Concentrate Price	USD/t	827

Table 1: Major Project Criteria of the Project

PROPERTY AND OWNERSHIP

PROPERTY

The Georgia Lake Property is located approximately 160 km northeast of Thunder Bay within the Thunder Bay Mining Division in NTS sheets 42E05NW and 52H08NE.

The Georgia Lake Property consists of 283 exploration claims and 81 dispositions. The total area of the claims is 5,693 ha (56.93 km²) and dispositions comprise a total area of 1,042 ha (10.42 km^2).

There are no mine workings, tailings ponds, and waste deposits on the Georgia Lake Lithium Property, except for a historic mine shaft on the MZN pegmatite on the Nama Creek disposition. The shaft was built in 1956 by Nama Creek Mines Ltd..

OWNERSHIP

All claims for the mining rights are owned 100% by Rock Tech. Rock Tech or its subsidiary JB. For the Nama Creek dispositions Rock Tech holds also the surface rights. The surface of all the other areas belong to the Crown. Rock Tech has legal access to all of its claims. The due dates for the claims range from May 2019 until Dec. 2021 and for dispositions until Jan. 2031 or Jan. 2033.

ACCESSEBILITY AND INFTRASTRUCTURE

ACCESSIBILITY

The Georgia Lake Lithium Property can be accessed by dirt roads off Highway 11 north of the town of Nipigon. The closest airport is located in Thunder Bay. The Nama Creek and Conway properties can be accessed by driving 60 km north of the town of Nipigon on Highway 11, then driving approximately 5 km east on a dirt road to reach the western boundary of the claims .

The McVittie and Jean Lake properties can be accessed by driving 40 km north of the town of Nipigon on Highway 11, then driving approximately 14 km northeast on a dirt road toward Postagoni Lake to reach the area and another 22 km to reach the northern area between Jean Lake and Foster-Lew.

The Aumacho area can be accessed by driving 40 km north of the town of Nipigon on Highway 11, then driving 7 km east on a dirt road to reach the area and another 6 km to reach then the Newkirk-Vegan property. Temporary bridges are needed to drive to the Newkirk-Vegan property.

The MNW property can be accessed by driving 31 km north of the town of Nipigon on Highway 11, then driving approximately 11 km east on a dirt road to reach the eastern boundary of the local claims, but temporary bridges are needed to drive to the property.

CLIMATE AND VEGETATION

The forest of the Georgia Lake area is mixed growth of spruce, balsam, jackpine, poplar, birch and cedar (Pye, 1965). Vegetation is typical of continental climate a mixture of coniferous (pine and black spruce) and deciduous (primarily birch and minor poplar

The climate is typical continental with cold and long winters (from November to late March) and significant snow accumulations. The temperature in the winter months (January and February) can reach -40° C but typically ranges between -10° and -25°C.

PHYSIOGRAPHY

Rock exposures in the area are abundant, and between the outcrops there is a thin mantle of glacial deposits. These glacial deposits consist mainly of stratified accumulations of unconsolidated sand and gravel. Some of them represent a ground moraine sorted by the action of glacial meltwaters; others form prominent terraces

along the shores of Lake Nipigon and in the valley occupied by Keemle and Wanogu Lakes, and are abandoned beach deposits.

Topography of the Georgia Lake Property is moderate. The minimum elevation is 250 m and the maximum elevation is 560 m asl. Thus, the range is 310 m. The low-lying areas are typically underlain by metasediments and the higher areas are underlain by Nipigon diabase.

INFRASTRUCTURE

The village of Beardmore is the closest community, located approximately 16 km north of the Georgia Lake Property. Nipigon is located 50 km south of the property. Several lakes, rivers and creeks are on the Georgia Lake Property.

There is a power line that runs along the TransCanada Highway #11 about 10 km from the property. There are three hydroelectric stations on the Nipigon River, all of which are controlled remotely by the headquarters in Thunder Bay: Alexander Station with 68 MW output (17 km north of the town of Nipigon), Cameron Falls with 87 MW output (17 km north of the town of Nipigon) and Pine Portage with 142 MW (39 km north of the town of Nipigon).

HISTORY

A comprehensive drilling programme was carried out then in 1955 and 1956 to the Spodumene pegmatites after these had been discovered during general prospection work as a potential source of the lithium mineral. Based on these results a first, NI43-101 non-compliant, resource estimate was prepared.

In consequence, additional drilling has been completed by Rock Tech in order to upgrade the historical resources to the mineral resource estimate from 2012 for MZN, MZSW, HAR, LIN, and CON. The data acquisition done from 2009 to 2012 is well documented including a comprehensive QA/QC management to validate the acquired data.

GEOLOGY AND MINERALIZATION

Lithium occurs as Spodumene in pegmatites in the Georgia Lake area. The pegmatites are hosted by metasediments. All of the pegmatites are Albite-Spodumene type. Spodumene is the dominant Li-bearing mineral in all of the pegmatites.

Overall, the pegmatite dyke internal zonation increases in complexity from north to south within the Georgia Lake pegmatite field:

- Nama Creek - MZN and MZSW have simple zonation: aplite or granitic border zone and a Spodumene zone with minor alternating aplite + pegmatite layers.
- Harricana, Line 60 and Conway have aplite or granitic border zone, a Spodumene zone and common alternating aplite + pegmatite layers.

The mineralization in the Georgia Lake Lithium Property consists of coarse-grained fresh pale green spodumene crystals, oriented perpendicular to the strike of the pegmatite dyke in homogeneous dykes, and randomly oriented within the inner Spodumene zone in simply zoned pegmatite dykes. The Spodumene may be

altered to muscovite or fine-grained muscovite near the contacts with the host rocks and near diabase dykes. The altered Spodumene has low Li contents and a high iron content.

EXPLORATION

In general, mapping, trenching and drilling has been carried out in 1955/1956 and again starting from 2009 in the area relevant for resource modelling and estimation. No geophysical surveys were applied during that exploration phase in these areas. In other parts of the property, geophysical ground surveys have been done, e.g. magnetic, electromagnetic. However, results were not significant.

From 2014 until 2018 additional mapping, drilling and channel sampling take place. The number of all bore-holes and channels of the whole area are listed in the Table 2. In all exploration claims and dispositions 48.3 km of drillholes and channels were drilled and cut during since the exploration began in early fifties of the last century.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
not recorded	33		2,340	
1955/56	205		28,677	
1957/58	26		1,787	
1987/89	3		199	
2009 to 2011	70	74	12,409	456
2016/2017	14	106	1,972	442
TOTAL	**351**	**180**	**47,384**	**898**

Table 2 Number of all boreholes and channels in the whole area of Rock Tech.

MINERAL PROCESSING AND METALLURGICAL TESTWORK

Main test results are based on the report „ The recovery of Spodumene from Georgia Lake Project, Project 122607-001, Oct./11/2011 by SGS Canada Inc.".

A total of 770 kg sample were taken from the Georgia Lake deposit. Three outcrop and one drill sample were mixed together to a head sample with a composite sample contained 19% spodumene, 32.4 % quartz, 34.4 % albite, 7.1 % muscovite and 7.1 % microline.

Because of the differences in specific gravity between Spodumene and tailings, gravity test with heavy liquid separation have been carried out in five single tests. Heavy liquid separation can achieve a concentrate with 6.0 % Li_2O and a recovery of Li_2O-approx. 70%.

Spodumene recovery from fine material; with grain size of approx. 300 µm; have been extensively tested in flotation testwork. First step of flotation was the recovery of mica and secondly of Spodumene. Different flotation schemes (rougher-cleaner-scavenger–combination) have been tested with 10 kg samples each.

As results of the combination of heavy liquid separation and flotation a total recovery of 81.5 % with 6.2 % Li_2O in concentrate have been achieved.

For a conservative approach, DMT used an overall recovery of 78% Spodumene recovery to achieve a 6.2 % Li_2O concentrate.

MINERAL RESOURCE ESTIMATE

Independent, NI 43-101 compliant resources at the Georgie lake properties of Rock Tech were estimated using validated and verified historical drill hole data, results from the 2010/11 drill and trenching programs conducted by CCIC on behalf of Rock Tech, and results of further drilling and trenching since 2012 conducted by Arriva Management Inc. Vancouver on behalf of Rock Tech. During the 2012 to the beginning of 2018 further exploration work was done. The resource model and estimate are dated April 2018.

For the five main pegmatites located in the NSPA a geological 3D model was built. The general concept, which underlies the wireframe interpretation is based on tabular mineralized bodies following dyke structures with a general orientation and extent of the main pegmatites.

Following international requirements, a Li_2O cut-off grade was applied to constrain the estimated mineral resources and to demonstrate reasonable prospects for eventual economic extraction. The reporting cut-off grade of 0.65% Li_2O was chosen based on the benchmarking of similar Lithium projects, but is not based on a financial model specific for this project, but based on comparable projects.

The NI 43-101 compliant resource estimate outlined a resource of 6.58 million t measured and indicated and 6.72 inferred resources as of August 2018 (Table 3).

Type of Resources	Tonnage [Mt]	Li_2O [%]	Cut off Li_2O [%]
Measured	1.89	1.04	0.65
Indicated	4.68	1.00	0.65
Inferred	6.72	1.16	0.65
TOTAL	**13.29**	**1.09**	**0.65**

Table 3: NI 43-101 compliant resources

MINING METHOD

For this study, three mining scenarios have been analysed for the pegmatite areas Main Zone/North, Main Zone/Southwest, Harricana, Line 60, and Conway: Open pit (OP) option, Underground (UG) option, and Hybrid option (OP-UG combination). All cost and operating parameters applied were based on preliminary estimates for developing the economic pit and the underground mine layout.

To evaluate the feasibility of an open pit only mining scenario, a standard pit optimisation analysis was conducted. The objective of the pit optimisation was to determine the ultimate pit limit based on the highest project cash flow (e.g. revenue, mine operational costs, ore processing and ore handling costs, etc.) in present value terms. In addition to these cost parameters, the mineral resource model, geotechnical slope parameters, processing recoveries and other project constraints (cut-off grade 0.65 % Li_2O) were included in the optimisation analysis.

The optimum pit shape for all pegmatite dykes in terms of pit value, required ROM grade, and stripping ratio was generated with RF 1.0. The ultimate pit, however, only contained around 57 % of the total resource.

Deeper sections of the pegmatite dykes were not included in pit envelope as mining them would be unfeasible due to a significant increase in stripping ratio and mining costs.

To evaluate the feasibility of an underground only option, the opportunity cost approach was applied. This modelling approach takes into account the opportunity value of an underground mine whilst optimising the pit outline. Based on the same optimisation constraints, a new pit shell was generated all pegmatite dykes using RF 1.0 and cut-off grade 0.65 % Li_2O.

The results indicate that a large portion of the resource can be economically extracted by underground mining. In addition, a certain amount of open pit mining will be required to harvest the uppermost sections of all pegmatite dykes. Therefore, the hybrid mining option was found to be the most suitable solution for all Nama Creek pegmatites. This mining strategy will allow fast extraction of outcropping and near-surface ore without the necessity of complex and cost intense developments generating revenue right from the start of the operation. All required infrastructure for underground mining can be developed in the meantime provided all pits contain enough ore tonnes to cover annual production requirements.

Optimisation of the hybrid option suggests that a sufficient portion of the ore (c. 35 %) can be extracted economically to a depth of around 300 m (final pit depth c. 70 m asl) covering the first year of production while a significant portion (c. 65 %) can be mined by underground methods.

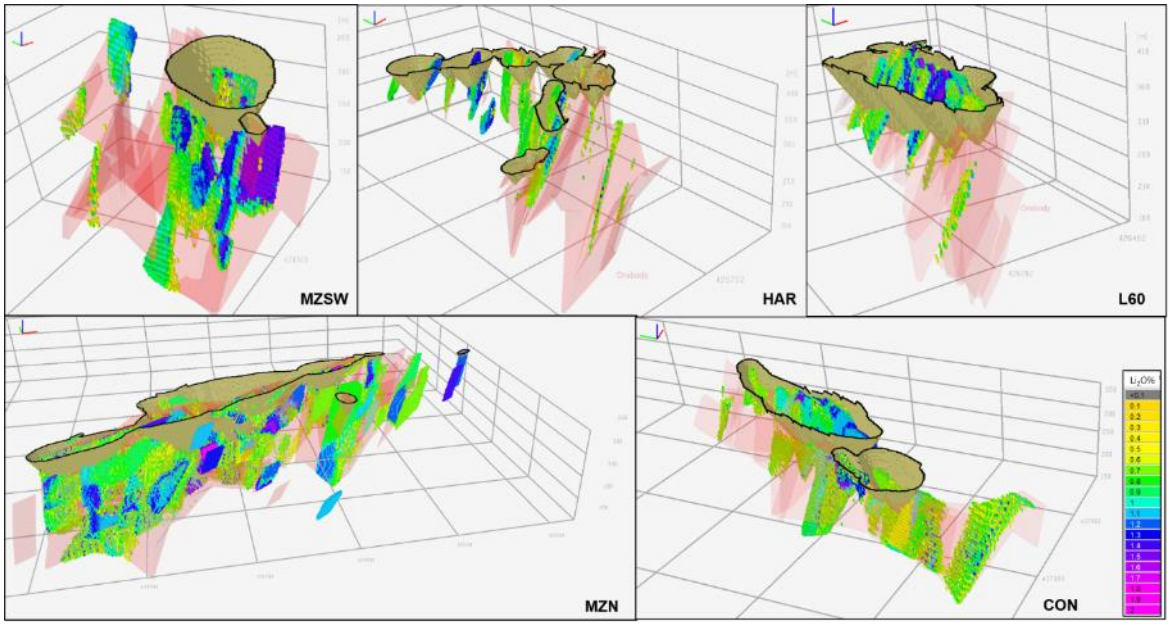


Figure 1: 3D views of pit shells of 5 mine areas, with original orebody, open pit and underground portions and blocks with Li_2O grades above cut-off.

For outcropping and near-surface pegmatite sections, standard open pit methodologies will be employed. The rock will be broken by drilling and blasting, and then trucked to the processing plant or the waste dump. Excavation progresses vertically and laterally from the surface via benches. Starting with a box cut, the pit will be developed successively maintaining an overall pit slope angle of 60°.

The remaining pegmatite sections will be mined by sublevel stoping with backfilling allowing the recovery of most of the resource at a reasonable cost level. Access to these sections will be gained through decline portals

sited within the existing open pit. With the exception of the MZN orebody, for which the ramp access will have to be located outside the pit as underground development and open pit production are scheduled to start simultaneously. A safety zone (crown pillar) of 11 m will have to be left in place to guarantee surface stability and operational safety of all underground workings. In addition, a ventilation raise will be required to ensure sufficient airflow throughout the mine. To prepare the orebody for extraction, many openings have to be driven in advance to facilitate roads, ventilation, power, air and water lines, shops and other infrastructure. In addition, ongoing openings have to be progressed during the operation. Orebody access will be gained through cross-cut drifts intersecting the pegmatites, which will be mined in 20 m slices. Production stope access will be provided by 2 sublevels. All mining stopes will be developed along strike of the orebody following the shape of the pegmatite dykes (maximum width 10 m). For both development and ore production conventional drill-blast-muck cycles will be applied. All stopes will be extracted according to a stoping sequence, which depends on the number of unmined or backfilled stopes to be left in place to conform to pillar size requirements. Additional infrastructure required for the operation includes sumps, refuge stations, passing bays, battery rooms, crusher chambers, explosives magazines, ventilation facilities, workshops and personnel facilities.

The preliminary mining sequence foresees ore production to commence in Main Zone/ North (Figure 2) starting with open pit mining followed by underground mining. The other pegmatite areas will then be mined accordingly in the following order: MZSW, HAR, L60, and CON.

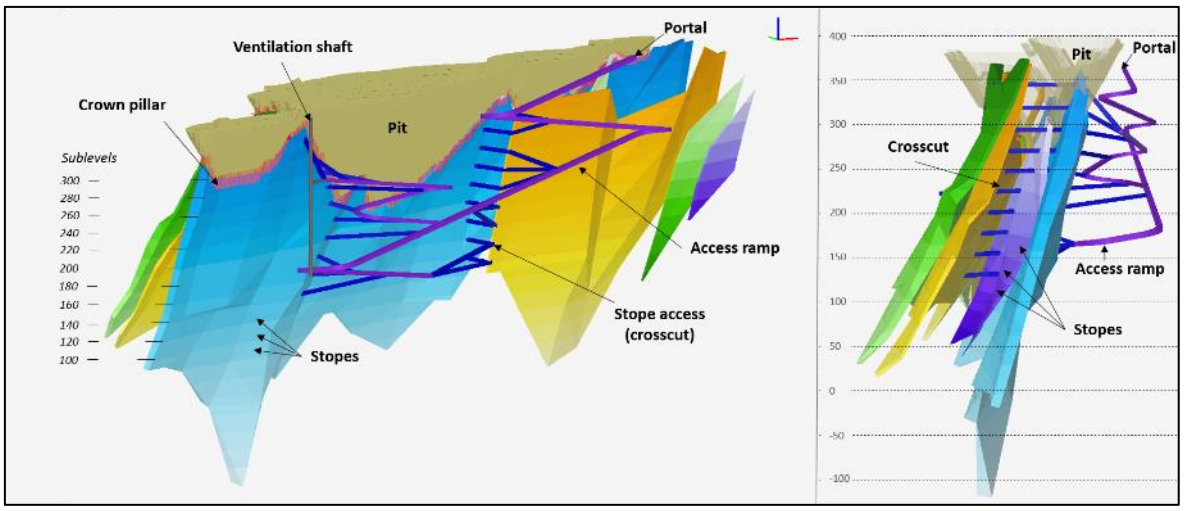


Figure 2: Conceptual underground mine layout for MZN (3D view).

RECOVERY METHODS

The metallurgical tests necessary for this report were carried out on a three outcrop and one drill core samples from the Georgia Lake deposit. According to the test results in, the suggested plant flow sheet is subdivided into the following sections:

- 3 step crushing
- Gravity separation
- Milling
- Flotation

- Dewatering section of different products

The plant is designed for a feed capacity of 150 t/h or 1,000,000 tpy. For financial calculation, it is recommended to use an adaptation of the SGS test results as follows:

- Li_2O content in feed of approximately 0.90 Li_2O (0.42 % Li)
- Li_2O content in concentrate 6.2 % Li_2O
- Li_2O overall recovery 78%

Figure 3 shows a simplified flowsheet of the processing plant.

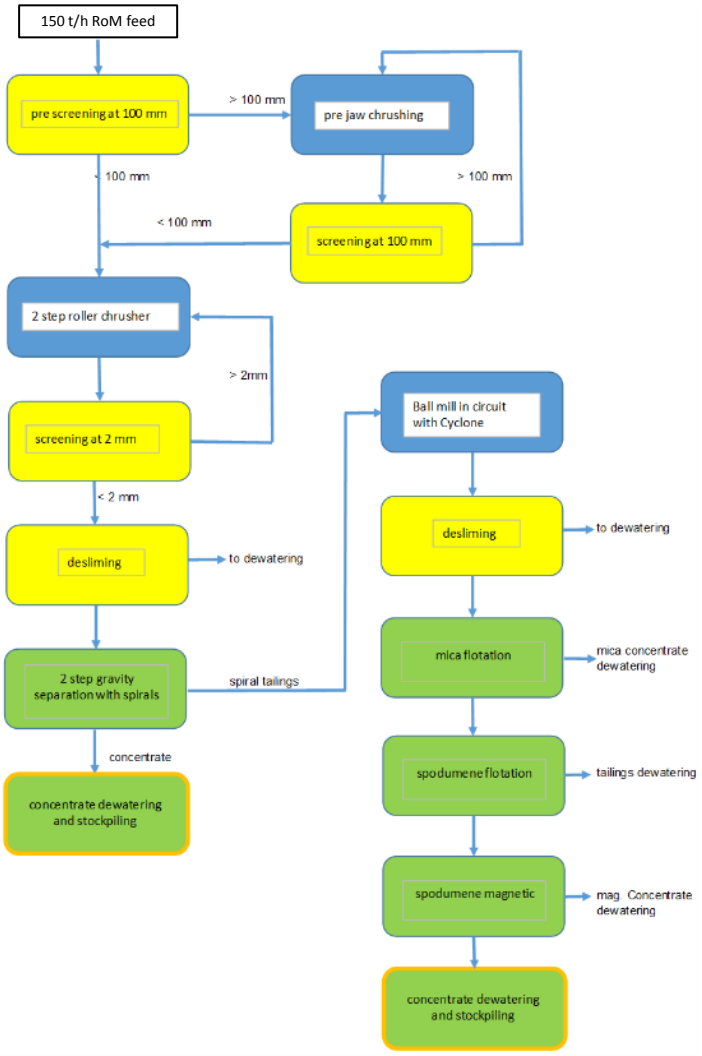


Figure 3: Recommended flow sheet for plant layout

PROJECT INFRASTRUCTURE

As a comprehensive greenfield project, the Project will require the development of supporting infrastructure. This would include the following items:

- A process plant or concentrator that will include crushing, grinding, flotation, regrinding, concentrate filtration, concentrate thickener, tailings thickener facilities and assay laboratory
- A warehouse, maintenance shop, administration offices, and supporting infrastructure
- A geomembrane-lined TDF, and structural earth dams, initial waste rock dumps
- A network of access and on-site roads
- A fresh water supply and distribution system
- Power supply and distribution, including a power transmission line, a substation at the plant site, and power distribution lines
- Other infrastructure including truck shop

The location of the single facilities is depending on the further exploration program and the inertization drilling in the referred areas. Furthermore, all facilities will be located close to the plant site with sufficient distance to the river and in a not swampy area.

Figure 4 shows the location of the Property in relation to principal supporting infrastructure. As there is no rail access to the mine/concentrator site, delivery of reagents to and shipment of concentrates from the site will be by truck. Concentrates will be shipped either to the clients or to a hydrometallurgical plant located in the vicinity of the operation.

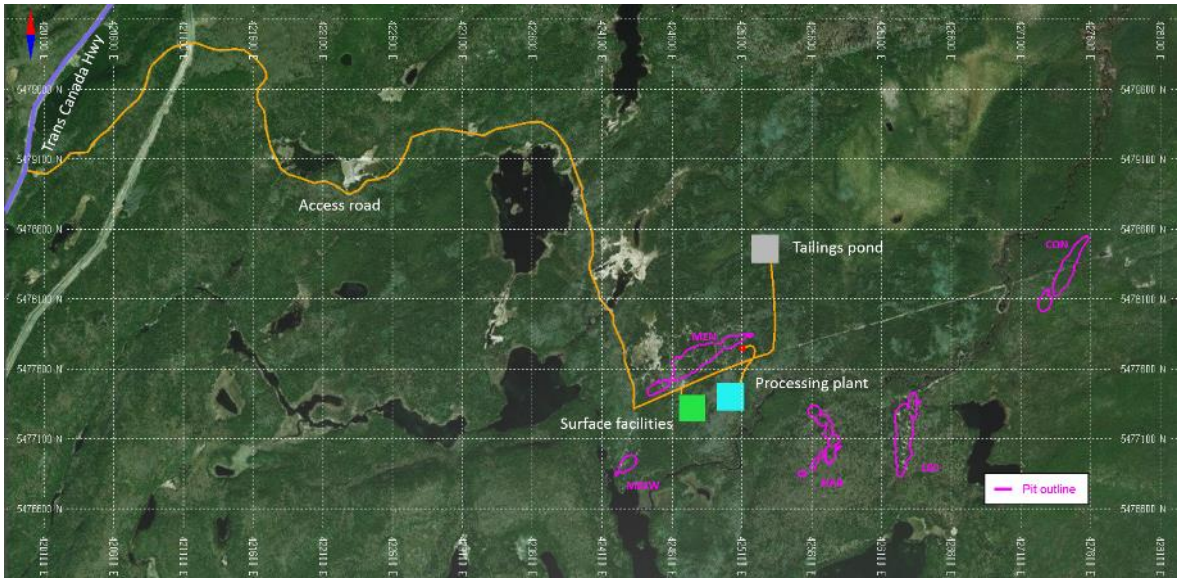


Figure 4: Preliminary location of the facilities together with pit outlines

MARKET STUDY AND CONTRACTS

LITHIUM MARKET

In 2018, USGS (United States Geological Survey) reported global lithium reserves to be 16 Mt Li (85 Mt LCE). The USGS also reported lithium resources at 53 Mt Li (282 Mt LCE), with continental brine resources accounting for 60% of the total resources. The three most commonly sold finished products are lithium carbonate, lithium hydroxide, and lithium Spodumene concentrate. Transactions are negotiated between the producer (or agent/trader) and the consumer (i.e. battery industry). Lithium is not traded on any exchange.

Based on research of Benchmark Minerals, there are presently 45 lithium ion battery megafactories under construction around the world equating to >1,000 GWh in the coming years. Based on these growth figures of lithium consumers, the demand for lithium products will likely surpass all current analysts' expectations. Consequently, the price for battery-grade lithium hydroxide is forecasted to rise further to USD20,000/t in 2030, from lithium contract prices of around USD 14,000/t in 2018.

Medium and long-term outlook for lithium consumption and lithium product prices remains very strong, with constantly increasing prices over the years. McKinsey projects an overall demand growth for lithium of 16-18% annually until 2030, creating pressure on supply. This would lead to a demand of nearly 2 Mt LCE in 2030.

LITHIUM SPODUMENE CONCENTRATE (LI_2O) PRODUCT PRICING

Rock Tech plans to fast track the project into production and assumes stable and strong lithium concentrate prices. In short, Spodumene concentrate prices have been derived from an analysis of forecasts of leading market players including Roskill, Benchmark Minerals, investment banks and industry peers, as well as a current market review of demand and supply trends. In line with analysts' forecasts, a constantly rising sales price during LoM with an assumed starting price of USD 800/t for a 6.2% Li_2O concentrate is expected.

ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Trow Associates Inc. (2011) conducted a baseline ecological study for the Georgia Lake Lithium Property in 2010. Environmental data were collected to document the baseline conditions before commencement of any mining activitiy.

In light of this, attempts were made to collect water samples at pre-selected locations situated upstream, within and downstream of proposed drilling (and potential future extraction) sites. Twenty-four sample stations were established and sampling was conducted at sites associated, to some degree, with all claim blocks. Field work which consisted of surface water sampling was conducted from November 22 to 26, 2010.

The project is small in scale without many of the risks frequently. Most of the tailings are supposed to be used as backfill, waste rock to be used as input dump, concentrate materials are expected to be inert and air and water quantities utilized and discharged are relatively small and can be managed to acceptable standards with conventional technologies.

Positive relationships have already been developed with Indigenous Peoples as well as political and community representatives.

Given the relatively small size and low environmental risk, no permitting problems are expected and all permits should be acquired in a timely manner that will not negatively influence the project schedule.

However, Environmental Impact Assessment (EIA) process is initiated later in the economic feasibility process. While working on the Preliminary Economic Assessment (PEA) Rock Tech decided to start as early as possible with the EIA process to move to the next stage.

CAPITAL COSTS

The mine site project covered in this study is based on the construction of a green field facility having a nominal daily processing capacity of average 1,500 tpd for open pit (Contractor's operation) and 2,000 tpd for underground mining (Owner's operation). The capital and operating cost estimates related to the mine, processing plant, site infrastructure have been developed by DMT.

The capital cost estimate for this project presented herein is considered to with an expected accuracy level of +30%/-30% and carrying a contingency of 20% on total initial estimated capital.

The capital cost estimate consists of the initial capital costs with CAD 66.5 million plus CAD 3.0 million as working capital. The sustaining capital sum up to CAD 68.2 million. The capital cost summary and its distribution by area is shown in Table 4.

CAPEX	Mio CAD
Initial Capital	
Mining	
Processing	46.1
G&A	1.0
Other Costs	18.3
Working Capital	3.0
Pre Production Capital	68.3
Sustaining Capital	
Mining	50.3
Processing	9.1
G&A	0.6
Other Costs	5.0
Closure Costs	3.0
Total Capital Expenditures	**136.3**

Table 4: LoM Capital Cost Summary

OPERATING COSTS

The average unit operating cost over the LoM was estimated at CAD 397/tonne concentrate. The unit operating costs include open pit and underground mining cost with CAD 212/tonne concentrate, mineral processing cost (CAD 133/tonne concentrate), general and administration (G&A) cost with CAD 26/tonne concentrate and other operating costs (CAD 26/tonne concentrate). Operating costs for the project are summarized in Table 5.

OPEX (Unit Costs)	Unit	Value
Mining	Mio CAD	224.3
	CAD/t conc	212
Processing	Mio CAD	140.4
	CAD/t conc	133
G&A	Mio CAD	27.4
	CAD/t conc	26
Other costs	Mio CAD	27.3
	CAD/t conc	26
Total OPEX	Mio CAD	419.4
	CAD/t conc	397

Table 5: LoM and Unit Operating Costs

Items listed in Table 5 include labor costs, which were estimated based on the manpower that will be necessary to operate the proposed mobile mining fleet and stationary equipment. Mining production rates and productivity as well as equipment mechanical availability and utilization factors were taken into account in the operating cost estimate. Annual salary projections were based on current mining industry standards.

ECONOMIC ANALYSIS

The economics of the project have been evaluated with an Excel-based real-basis financial model developed in 2018 CAD to present the cost structure and the economic evaluation of the project as a stand-alone entity.

The lithium pricing was developed by DMT in cooperation with Rock Tech based on the recent development of the market for a 6.2% Li_2O concentrate ranges from USD 800 in 2020 to USD 850 in 2026; this value is used from 2026 to the end of the Project in 2031.

Outcome I	Unit	Value
Initial CAPEX	Mio CAD	65.3
Average LoM Operating Costs	CAD/t conc	397
LoM Revenue	Mio CAD	1,136.2
LoM Operating Costs	Mio CAD	419.4
LoM EBITDA	Mio CAD	706.0
Annual EBITDA	Mio CAD	64.2
Pre Tax NPV_8	Mio CAD	312.2
After Tax NPV_8	Mio CAD	210.2
Pre Tax IRR	%	62%
After Tax IRR	%	48%
Pre Tax Payback	Years	3.1
After Tax Payback	Years	3.5

Table 6: Economic Indicators

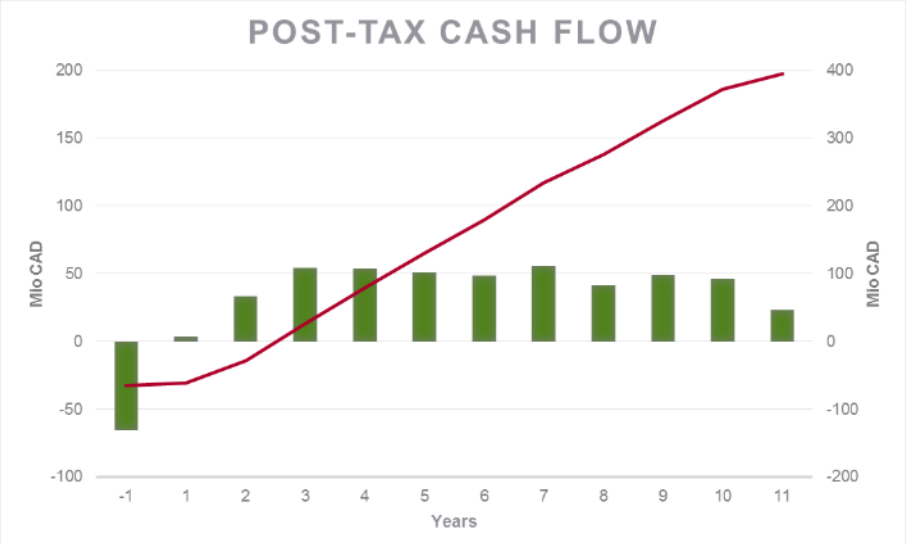


Figure 5: Post-Tax Cash Flow

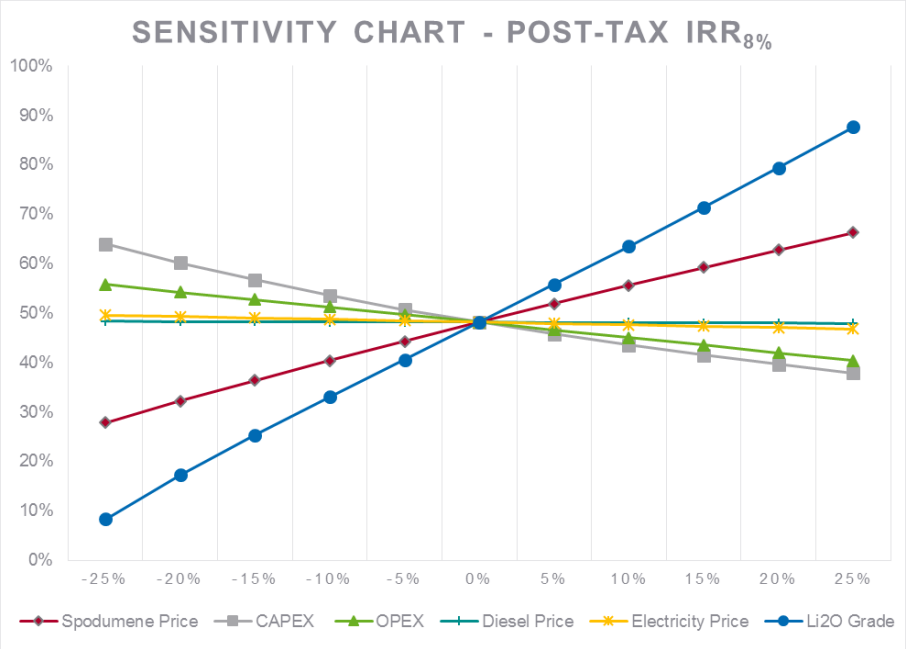


Figure 6: Post-Tax IRR Sensitivity Analysis

An exchange rate of 1.30 CAD to USD and 1.50 to EUR is used in the financial evaluation. The project cash flows were assessed to 2031. The financial model has been used to estimate future cash flows and evaluate the project on the basis of net present value (NPV), internal rate of return (IRR) and payback period. The results of the analysis are provided in the Table 6.

The total unit operating cost for Li_2O concentrate from Georgia Lake deposit is CAD 397 /t conc which is equivalent to USD 305 /t conc..

Post-tax free cash flow over the life of the project is summarised in Figure 5. The financial analysis completed examined the IRR sensitivity to the main factors affecting the Project, namely, Li_2O concentrate pricing and grade, capital, operating costs and some minor items such as electricity and diesel pricing. The results are shown in Figure 6. The project is most sensitive to changes in the Li_2O concentrate pricing and grade, less sensitive to changes in capital costs and operating costs and least sensitive to consumable price changes (diesel and electricity).

ADJACENT PROPERTIES

Exploration activities of adjacent properties have not been considered. North of the area the Beardmore-Geraldton area hosts several deposits types including vein-hosted gold and lithium pegmatites deposits.

OTHER RELEVANT DATA OR INFORMATION

In 2011 Rock Tech announced that it entered into a Memorandum of Understanding with Bingwi Neyaashi Anishinaabek, Biinjitiwaabik Zaaging Anishinaabek, and Animbiigoo Zaagi'igan Anishinaabek First Nations in regards to the development of the Georgia Lake Lithium project.

Since Rock Tech began exploration in December 2009, several First Nations members have been employed and equipment and material have been procured from the First Nations whenever feasible.

RECOMMENDATIONS

Recommendations for different areas of the project have been set out in section Recommendations. Mainly the recommendations refer to necessary testwork and assessments in order to minimize uncertainties. This includes necessary work in the fields of:

- Exploration with drilling investigation as well as more trenching, where appropriate for increasing the resources in the category measured and indicated and in general to get more detailed information about e.g. the rock mass and the hydrogeological situation,
- Geological mapping for best knowledge of the area and to explore for potential extensions of the lithium deposit to increase potentially recoverable lithium resources,
- Mineralogical studies to confirm identification of phosphate minerals at MZN and all the other pegmatites and in general in order to further refine mineralogical zonation patterns within the deposit,
- Geotechnical investigations to support the overall pit slopes and design of ramps and haulways, as well as the underground design. This implies as well the waste material for a potential dump design
- The selection and testing of the appropriate sensing systems for optical sorting,
- Additional processing testwork to confirm previous results, test different equipment such as spirals instead of heavy liquid separation, test the potential of by-products,
- Study on tailings concern a potential mixture for backfill, rheology for the fluid flow characteristics and defining the disposal options (TDF of filtered tailings),
- Geochemical analysis of tailings, concentrates, RoM ore and waste rock,
- Detailed evaluation on the access options to the Project area in terms of road upgrade, installation of permanent bridges and detailed site investigation for the location of the site facilities, processing plant and potential TDF,

- The conditions of a connection with the power supply should be studied in details as well as the potential extraction and discharge of water,
- The ESIA is the next step that should be conducted with the basis of the preparation of the complete historical environmental baseline validation, as well as complete a Project Description

INTRODUCTION

DMT GmbH & Co. KG ("DMT") of Essen, North-Rhine Westphalia, Germany was contracted by Rock Tech Lithium Inc. ("Rock Tech") of Vancouver, British Columbia, Canada, to review the Georgia Lake Lithium Project (the "Property"), and prepare a Preliminary Economic Assessment (the "PEA"), compliant with National Instrument 43-101 ("NI43-101"). The report should be based on the available documentation inter alia DMT's Independent Technical Report on the updated the mineral resources based on latest data acquired.

For this report all relevant claims and licenses, which belong to Rock Tech were used for PEA. Professional Geologist Keith McCandlish and EurGeol K.-S. Peters, as qualified person, visited the project verifying results, operation procedures and the execution of the work in the field.

Since the last report from "CCIC" (2012) exploration was still ongoing and more drill holes and channel samples were available for the updating of the resource estimate. A new 3D model were generated and all validation work was done as basis for the first pit optimization and an assessment for an open pit/underground economic operation.

TERMS OF REFERENCE

PRELIMINARY ECONOMIC ASSESSMENT

The objective of this PEA is to demonstrate the economic potential for producing a lithium ion battery material from the Rock Tech Lithium deposit. The deposit was evaluated previously in 1950ies as a potential source of the lithium mineral. While this market remains an opportunity, lithium ion battery technology has developed as the energy storage solution of choice for a variety of commercial applications and this has resulted in a significant increase in demand, and projected demand, for battery materials.

This PEA has been prepared by DMT under the terms of its agreement with Rock Tech. As discussed in the relevant sections of the report, DMT has prepared a mine plan and schedule and has prepared an economic analysis of the project. DMT has reviewed the metallurgical testwork carried out on the property and the mineral processing flowsheet, has reviewed infrastructure requirements, and has estimated capital and operating cost.

MINERAL RESOURCE ESTIMATE

The PEA is based on mineral resource estimates for lithium contained in the Nama Creek deposit, prepared by DMT, 18.04.2018

It should be noted that for the PEA the mineable mineral resources have been used do demonstrate the economic viability of the project. The economic evaluation on the basis of the mineral resources presented in this PEA are estimates based on available sampling and on assumptions and parameters available to the author. The comments in this Technical Report reflect DMT GmbH & Co. KG best judgement in light of the information available.

The reserve and/or resource estimates in this PEA have been prepared in accordance with the requirements of Canadian securities laws, which differ from the requirements of United States securities laws. Unless otherwise indicated, all reserve and resource estimates included in this PEA have been prepared in accordance with NI 43-101. NI 43-101 is a rule developed by the Canadian Securities Administrators, which establishes standards for all public disclosure an issuer makes of scientific and technical information concerning mineral projects.

RELATIONSHIP WITH ROCK TECH

DMT does not have, and has not previously had, any material interest in Rock Tech or any related entities. The relationship between DMT and Rocktech is solely a professional association between the client and the independent consultant. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

The conclusions and recommendations in this report reflect the authors' best independent judgment in light of the information available to them at the time of writing. The authors and DMT reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to them subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

This report is intended to be used by Rock Tech subject to the terms and conditions of its agreement with DMT. That agreement permits Rock Tech to file this report as a Technical Report with the Canadian Securities Administrators pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

The requirements of electronic document filing on SEDAR necessitate the submission of this report as an unlocked, editable pdf (portable document format) file. DMT accepts no responsibility for any changes made to the file after it leaves its control.

QUALIFIED PERSONS, SITE VISITS AND AREAS OF RESPONSIBILITIES

The primary authors of this report and Qualified Persons are: Keith McCandlish, P. Geol., P. Geo., FGC, FEC (Hon) and Karl Stephan Peters, Eur Geol, who have been visiting the site between 10th and 14th September 2018.

Keith McCandlish did a Peer Review of all sections and Karl Stephan Peters was involved in the whole process of the preparation of this report.

RELIANCE ON OTHER EXPERTS

This report has been prepared by DMT, for the client. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to DMT at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by the client and other third party sources, e.g. contracted consultant company Arriva Management Inc. Vancouver.
- Discussions with representatives from the client who are familiar with the Property and the area in general
- DMT has assumed that the reports and other data listed in the "References" section of this report are substantially accurate and complete.

For the purpose of this report, DMT has relied on ownership information provided by the client. The public source of information regarding land tenure is the MNDMF website (MNDMF website: http://www.mndm.gov.on.ca).

DMT has not researched property title or mineral rights for the project and expresses no opinion as to the legal ownership status of the property. DMT has checked the website of the Government of Ontario, which shows the legal status of the claims as here reported. In according with the governmental rules Rock Tech paid all necessary fees for the government to hold the licences. All paper work was done on time for all the exploration permits and the mining dispositions.

The dates, titles and authors of all reports that were used as a source of information for this Technical Report are listed in the "References" section of this report. The dates and authors of these reports also appear in the text of this Report where relevant, indicating the extent of the reliance on these reports.

The information contained in this report with respect to the Mineral Resources is based on information provided by Rock Tech and reviewed by DMT in 2017/18. Whilst exercising all reasonable diligence in checking, confirming and testing this information, DMT have assumed that the data presented by Rock Tech is reasonable in formulating its opinion.

This report includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, DMT do not consider them to be material.

The various agreements under which Rock Tech hold title to the mineral properties for this project have neither been investigated nor confirmed by DMT and DMT offer no opinion as to the validity of the mineral title claimed by Rock Tech. The description of the property, and ownership thereof, as set out in this report, is provided for general information purposes only.

LOCATION

The Georgia Lake Property is located approximately 160 km northeast of Thunder Bay within the Thunder Bay Mining Division in NTS sheets 42E05NW and 52H08NE. Rock Tech's current claims and dispositions are shown in Figure 7and in detail in Figure 8

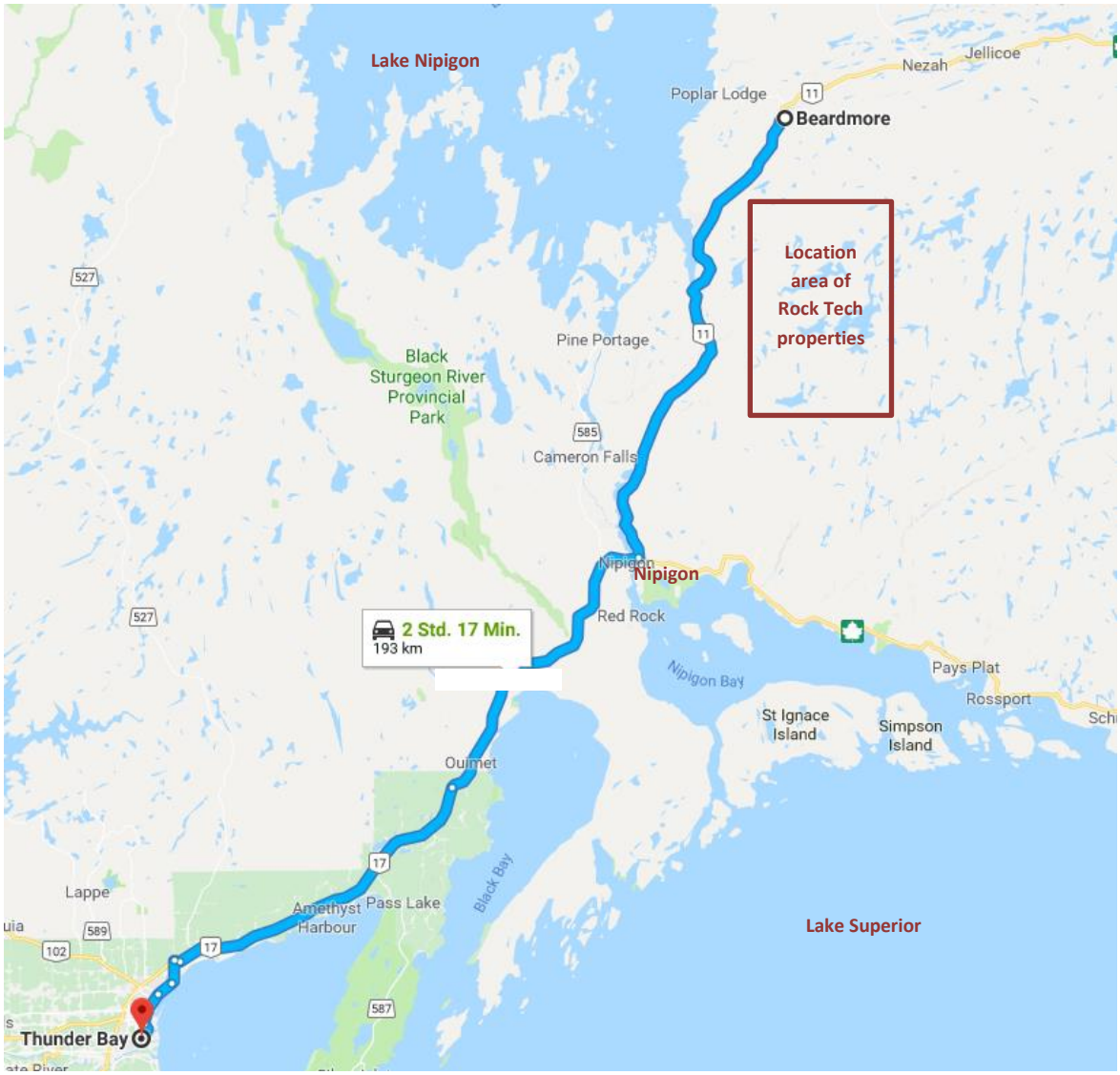


Figure 7. Location map of the Rock Tech properties and highway 11 as access road from the town of Thunder Bay (Source: Google Maps).

The blue line shows the way from Thunder Bay into the area along the Highway Nr.11. The town Nipigon on the norther side of the Lake Superior and Lake Nipigon North West of the investigation area (Figure 8). In Figure 8 the Highway 11 is running west of the claim in the east side of the Lake Nipigon to the north.

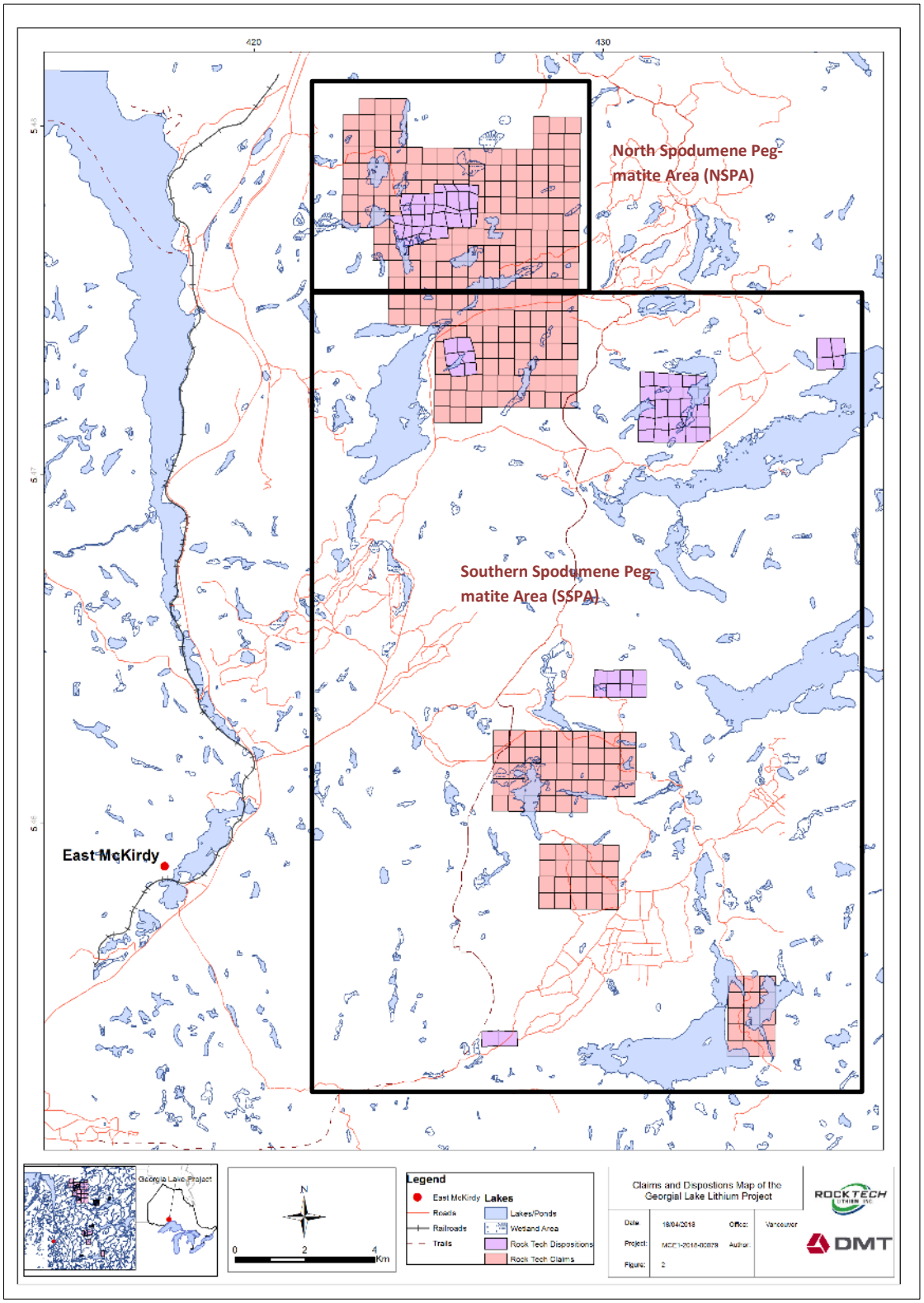


Figure 8. Location of exploration claims (red) and dispositions (magenta) currently held directly by Rock Tech Inc. and/or by its subsidiary James Bay Midarctic Developments Inc. (JB), and the location of the two areas NSPA and SSPA. The shown railway track parallel to the high way 11 as of today is out of order.

DESCRIPTION OF OWNERSHIP

The Georgia Lake Property consists of 283 exploration claims and 81 dispositions (Table 7 to Table 11). The boundaries of the claims and dispositions are shown in Figure 8. The total area of the claims is 5,693 ha (56.93 km²) and dispositions comprise a total area of 1,042 ha (10.42 km$^{2)}$.

All claims for the mining rights are owned 100% by Rock Tech. Rock Tech or its subsidiary JB. For the Nama Creek dispositions Rock Tech holds also the surface rights. The surface of all the other areas belong to the Crown. Rock Tech has legal access to all of its claims. From the web side of the MNDMF's Geoscience Assessment Office in Ontario, the data were checked and shows no gaps. The due dates for the claims range from May 2019 until Dec. 2021 and for dispositions until Jan. 2031 or Jan. 2033.

The dispositions (leases) consist of 6 blocks and comprise a total area of 1,042.43 ha (10.43 km²). All of the disposition blocks are contiguous with the mining claims, but are not contiguous with each other. The Nama Creek dispositions consist of 36 contiguous dispositions and a total area of 329.065 ha (3.29 km², Figure 9 and Table 7). The renewal of Nama Creek dispositions is finished now and run until 2031 and 2033 according to MNDM. The McVittie dispositions consist of 6 contiguous dispositions and a total area of 87.639 ha (0.87 km², Figure 9 and). The Parole Lake dispositions consist of 25 contiguous dispositions and a total area of 385.621 ha (3.85km², Figure 10 and Table 9). The Foster-Lew dispositions consist of 4 contiguous dispositions and a total area of 75.231ha (0.75 km Figure 10 and Table 10). The Newkirk-Vegan dispositions consist of 8 contiguous dispositions and a total area of 120.63 ha (1.20 km², Figure 11 and Table 12). The MNW dispositions consist of 2 contiguous dispositions and a total area of 44.248 ha (0.44 km², Figure 11 and Table 11).

Rock Tech is the owner of the mining and surface rights on the Nama Creek dispositions. James Bay Midarctic Developments Inc. is the owner of the mining rights only on all of the other dispositions (i.e., McVittie, Parole Lake, Foster-Lew, Newkirk-Vegan and MNW). James Bay Midarctic Developments Inc. is a subsidiary of Rock see report from 2012 CCIC (Selway, J., et. al. (2012b). The surface rights are owned by the Crown for all dispositions except for Nama Creek. The surface rights of the Nama Creek dispositions belong to Rock Tech. Rock Tech has legal access to all of its dispositions. All of Rock Tech's leases are for a 21 year term. In Ontario the Ministry of Northern Development and Mines is responsible for the claims and disposals (web site: https://www.mndm.gov.on.ca/en).

The following tables and figures show the list of leases according to the area and location.

Mining Right No	Legal Description	Status	Type	Expiry Date	Lease/Licence	Township/Area	Area (ha)
LEA-108972	TB67132	Active	Lease	2033-Jan-31	108972	Kilkenny	
LEA-108973	TB67133	Active	Lease	2033-Jan-31	108973	Kilkenny	
LEA-108974	TB67134	Active	Lease	2033-Jan-31	108974	Kilkenny	
LEA-108975	TB67135	Active	Lease	2033-Jan-31	108975	Kilkenny	
LEA-108976	TB67136	Active	Lease	2033-Jan-31	108976	Kilkenny	
LEA-108977	TB67137	Active	Lease	2033-Jan-31	108977	Kilkenny	
LEA-108978	TB67138	Active	Lease	2033-Jan-31	108978	Kilkenny	
LEA-108979	TB67139	Active	Lease	2033-Jan-31	108979	Kilkenny	
LEA-108980	TB67140	Active	Lease	2033-Jan-31	108980	Kilkenny	
LEA-108981	TB67157	Active	Lease	2033-Jan-31	108981	Kilkenny	
LEA-108982	TB67160	Active	Lease	2033-Jan-31	108982	Kilkenny	
LEA-108983	TB67162	Active	Lease	2033-Jan-31	108983	Kilkenny	
LEA-108984	TB67163	Active	Lease	2033-Jan-31	108984	Kilkenny	
LEA-108985	TB67164	Active	Lease	2033-Jan-31	108985	Kilkenny	
LEA-108986	TB67167	Active	Lease	2033-Jan-31	108986	Kilkenny	
LEA-108987	TB67177	Active	Lease	2033-Jan-31	108987	Kilkenny	
LEA-108988	TB67178	Active	Lease	2033-Jan-31	108988	Kilkenny	
LEA-108989	TB67180	Active	Lease	2033-Jan-31	108989	Kilkenny	
LEA-108990	TB67181	Active	Lease	2033-Jan-31	108990	Kilkenny	
LEA-108991	TB67145	Active	Lease	2033-Jan-31	108991	Kilkenny	
LEA-108992	TB67146	Active	Lease	2033-Jan-31	108992	Kilkenny	
LEA-108993	TB67147	Active	Lease	2033-Jan-31	108993	Kilkenny	
LEA-108994	TB67154	Active	Lease	2033-Jan-31	108994	Kilkenny	
LEA-108995	TB67165	Active	Lease	2033-Jan-31	108995	Kilkenny	
LEA-108996	TB67166	Active	Lease	2033-Jan-31	108996	Kilkenny	
LEA-108997	TB67158	Active	Lease	2033-Jan-31	108997	Kilkenny	
LEA-108998	TB67168	Active	Lease	2033-Jan-31	108998	Kilkenny	
LEA-108999	TB67169	Active	Lease	2033-Jan-31	108999	Kilkenny	
LEA-109000	TB67170	Active	Lease	2033-Jan-31	109000	Kilkenny	
LEA-109001	TB67171	Active	Lease	2033-Jan-31	109001	Kilkenny	
LEA-109002	TB67172	Active	Lease	2033-Jan-31	109002	Kilkenny	
LEA-109003	TB67173	Active	Lease	2033-Jan-31	109003	Kilkenny	
LEA-109004	TB67174	Active	Lease	2033-Jan-31	109004	Kilkenny	
LEA-109005	TB67175	Active	Lease	2033-Jan-31	109005	Kilkenny	
LEA-109006	TB67176	Active	Lease	2033-Jan-31	109006	Kilkenny	
LEA-109007	TB67185	Active	Lease	2033-Jan-31	109007	Kilkenny	
Total							**329.065**

Table 7. Dispositions (leases) for Rock Tech's Nama Creek Property

Mining Right No	Legal Description	Status	Type	Expiry Date	Lease/Licence	Township/Area	Area (ha)
LEA-108505	TB732171	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732172	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732173	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732174	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732175	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732176	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
Total							**87.639**

Table 8. Dispositions (leases) for Rock Tech's McVittie property

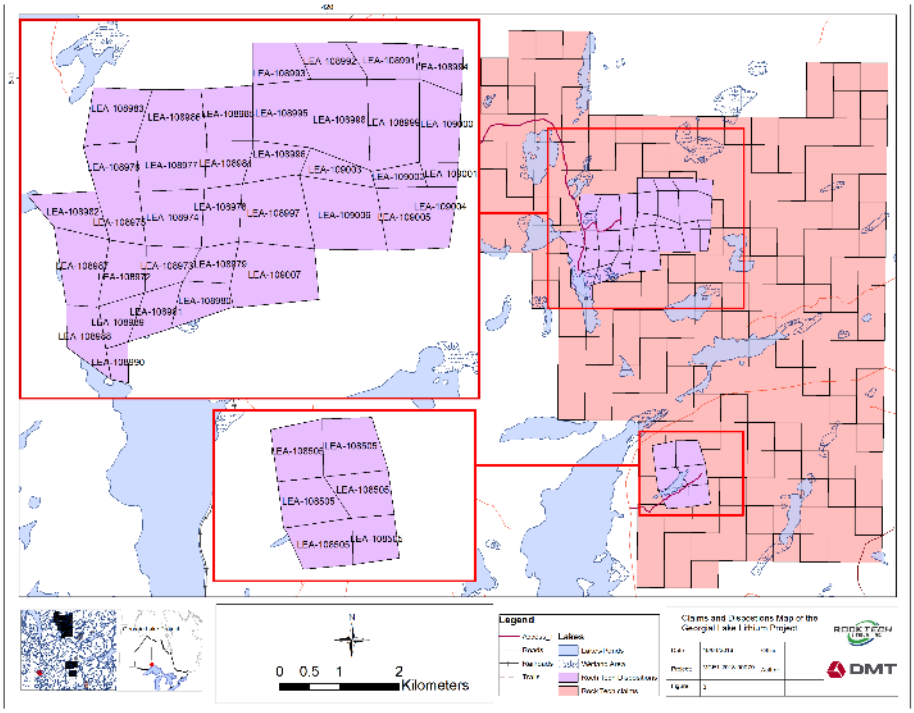


Figure 9. Location of claims and dispositions for Rock Tech's Nama Creek, Conway and McVittie properties.

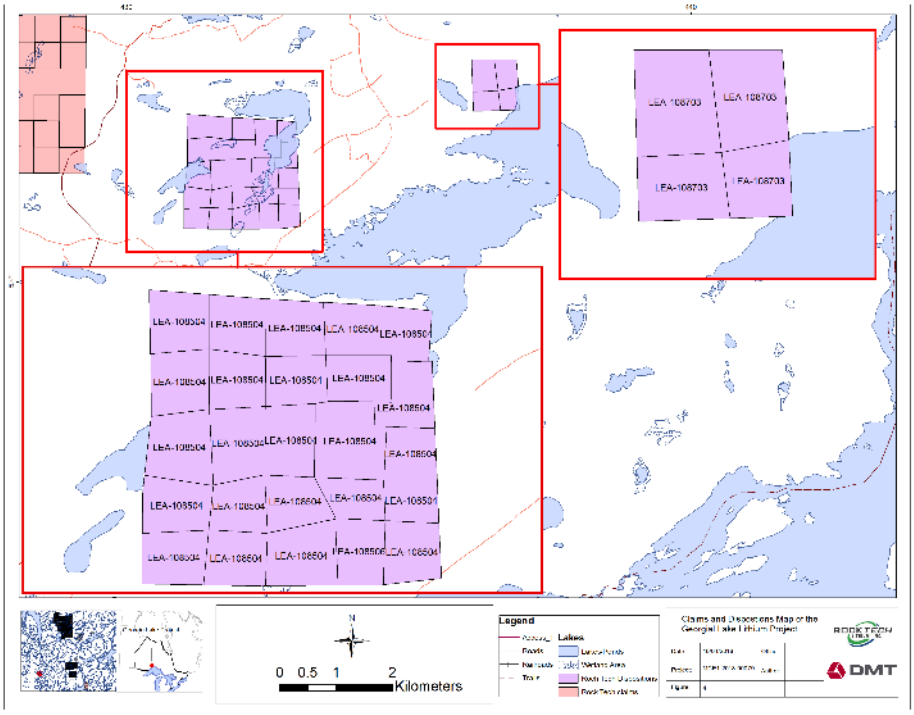


Figure 10. Location of claims and dispositions for Rock Tech's Parole Lake and Foster-Lew properties.

Mining Right Nur	short legal Discri	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108504	TB1009072	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB1009073	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB1020967	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756468	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756469	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756470	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756471	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756472	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756473	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756474	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756476	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756477	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756478	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756479	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756480	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756481	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756497	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756498	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756499	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108506	TB756500	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756501	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756536	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756537	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756538	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756539	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108703	TB1005886	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005887	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005888	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005889	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
Total							**385.621**

Table 9. Dispositions (leases) for Rock Tech's Parole Lake property

Mining Right Nur	short legal Discri	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108703	TB1005886	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005887	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005888	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005889	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
Total							**75.231**

Table 10. Dispositions (leases) for Rock Tech's Foster Lew property

Mining Right Nur	short legal Discri	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha
LEA-108704	TB863303	Active	Lease	2032-Jan-31	108704	Hanson Lake Area	
LEA-108704	TB863304	Active	Lease	2032-Jan-31	108704	Hanson Lake Area	
Total							**44.248**

Table 11. Dispositions (leases) for Rock Tech's MNW property

Mining Right Nur	short legal Discrip	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108506	TB824969	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824970	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824971	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824972	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824973	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824974	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB863418	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB863419	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
Total							**120.633**

Table 12. Dispositions (leases) for Rock Tech's Newkirk-Vegan property

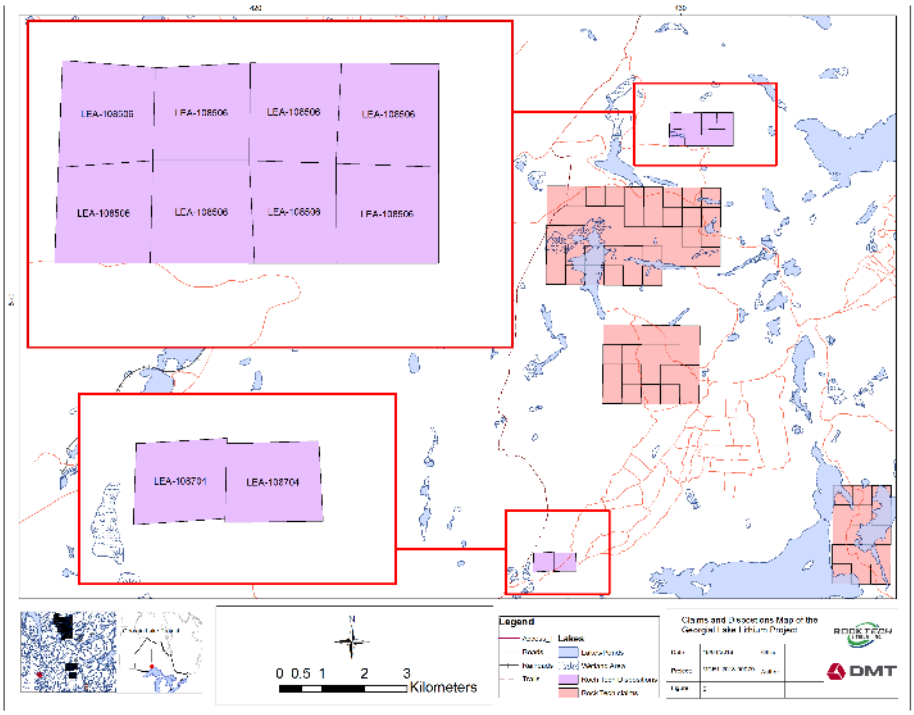


Figure 11. Location of claims and dispositions for Rock Tech's Aumacho, Newkirk-Vegan and MNW properties

OTHER STAKEHOLDERS

DMT did not verify in detail, if there are other rights / surface rights existing in the areas of interest. Known from other reports are:

In the Report from 2012 CCIC (Selway, J., et. al. (2012b)) reported that there are 6 alienations on the Georgia Lake Property type notice, class wind power: WP2005-12, WP2006-21, WP2006-21, WP2008-121, WP2008-154 and WP2008-359. These alienations are for surface rights only and the area is available for staking. There is one alienation withdraw order W-TB-139/11 which withdraws surface rights from staking while the construction site of a Transmission Line for waterpower development project is under review.

The location of all known mineralized zones is described in chapter Property Geology and in the CCIC report 2012 (Selway, J., et. al. (2012b). There are no mine workings, tailings ponds, and waste deposits on the Georgia Lake Lithium Property, except for a historic mine shaft on the MZN pegmatite on the Nama Creek disposition LEA – 108977 (old number TB67137). The shaft was built in 1956 by Nama Creek Mines Ltd.. To the best of DMT's knowledge the terms of the royalty require a payment of 1.5% NSR on the Georgia Lake Lithium Property other than the agreement that James Bay Midarctic Developments Inc. is a subsidiary of Rock Tech. To the best of DMT's knowledge, there are no environmental liabilities on the Georgia Lake Lithium Property.

To DMT's best knowledge there are no significant factors or risks that may affect access, title or the right or ability to perform work on the property, other than the requirement to obtain permits for temporary bridges to cross small creeks to drive to necessary properties.

In Figure 12 shows in detail the NSPA in the northern part of the whole property (see Figure 8). Location of exploration claims (red) and dispositions (magenta) currently held by Rock Tech and subsidiaries. The dots show the location of the drill holes beside the individual pegmatites whish ere modelled in 3D. The 5 pegmatites were the MZN (Nama Creek Main Zone North), MZSW (Nama Creek Main Zone South-West), HAR (Harricana), LIN (line 60), CON (Conway). They are all located in the NSPA. All other pegmatites mentioned in this report are all situated with in the SSPA. This SSPA is situated more east and southwards from the NSPA (see Figure 8).

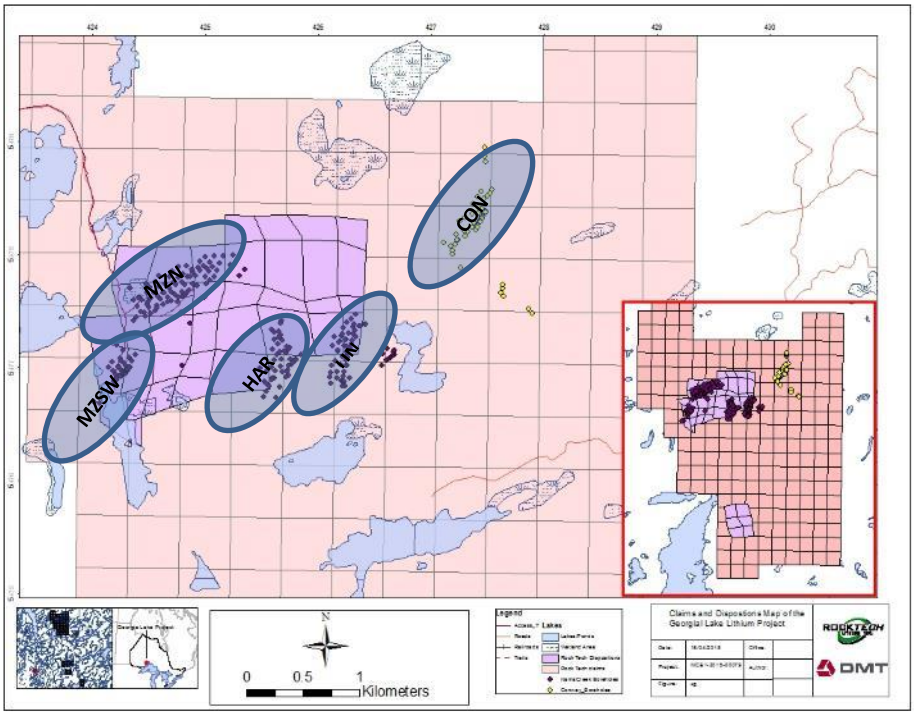


Figure 12. NSPA with the drillhole locations beside the 5 modelled pegmatites.

The altitude of the drilled areas of relevance for resource model and estimate ranges from 360 m up to 420 m asl.

ACCESSIBILITY

The Georgia Lake Lithium Property can be accessed by dirt roads off Highway 11 north of the town of Nipigon. The closest airport is located in Thunder Bay. The Nama Creek and Conway properties can be accessed by driving 60 km north of the town of Nipigon on Highway 11, then driving approximately 5 km east on a dirt road to reach the western boundary of the claims (see Figure 8).

The McVittie and Jean Lake properties can be accessed by driving 40 km north of the town of Nipigon on Highway 11, then driving approximately 14 km northeast on a dirt road toward Postagoni Lake to reach the area and another 22 km to reach the northern area between Jean Lake and Foster-Lew.

The Aumacho area can be accessed by driving 40 km north of the town of Nipigon on Highway 11, then driving 7 km east on a dirt road to reach the area and another 6 km to reach then the Newkirk-Vegan property. Temporary bridges are needed to drive to the Newkirk-Vegan property.

The MNW property can be accessed by driving 31 km north of the town of Nipigon on Highway 11, then driving approximately 11 km east on a dirt road to reach the eastern boundary of the local claims, but temporary bridges are needed to drive to the property.

CLIMATE AND VEGETATION

The forest of the Georgia Lake area is mixed growth of spruce, balsam, jackpine, poplar, birch and cedar (Pye, 1965). Vegetation is typical of continental climate a mixture of coniferous (pine and black spruce) and deciduous (primarily birch and minor poplar).

	January	February	March	April	May	June	July	August	September	October	November	December
Avg. Temperature (°C)	-16.4	-13.4	-7.2	1.5	8.7	13.4	17.1	16.1	10.8	5.2	-2.8	-11.7
Min. Temperature (°C)	-22.4	-19.8	-13.5	-4.5	2.1	7	10.7	10	5.5	0.7	-6.6	-16.5
Max. Temperature (°C)	-10.4	-7	-0.8	7.6	15.3	19.9	23.6	22.2	16.1	9.7	1	-6.8
Avg. Temperature (°F)	2.5	7.9	19.0	34.7	47.7	56.1	62.8	61.0	51.4	41.4	27.0	10.9
Min. Temperature (°F)	-8.3	-3.6	7.7	23.9	35.8	44.6	51.3	50.0	41.9	33.3	20.1	2.3
Max. Temperature (°F)	13.3	19.4	30.6	45.7	59.5	67.8	74.5	72.0	61.0	49.5	33.8	19.8
Precipitation / Rainfall (mm)	48	33	42	40	65	86	80	87	89	77	68	55

Figure 13. Climate data for Nipigon (Source: https://en.climate-data.org/location/767939/)

The climate is typical continental with cold and long winters (from November to late March) and significant snow accumulations. The temperature in the winter months (January and February) can reach -40° C but typically ranges between -10° and -25°C. The Canadian Climate normals for 1971-2000 from Environment Canada (/www.climate.weatheroffice.gc.ca/climate_normals/) for Geraldton (closest weather station to the property) indicate that the daily average temperature ranges from -19ºC in January to 17ºC in July. The highest average accumulation of rain for a month is 112 mm in July. The highest average accumulation of snow for a month is 49 cm in November. The highest average snow depth is 48 cm in February.

Drilling can be conducted year round except for spring thaw in mid-March and April. Geological mapping and outcrop sampling can be conducted May to November when there is no snow on the ground.

The climate may be challenging for open cast mining during the winter months.

PHYSIOGRAPHY

Pye (1965) summarized the topography of the Georgia Lake area: "The Georgia Lake area is one of topographic contrasts. The parts of the area in which metasediments are exposed are, for the most part, of low relief. In contrast, the parts underlain by granitic rocks are rugged, with rounded hills rising up to about 150 ft (=45.7 m) above the general level. Most conspicuous, however, are high, imposing vertical or near-vertical cliffs at the boundaries of large exposed sheet-like masses of diabase."

"Rock exposures in the area are abundant, and between the outcrops there is a thin mantle of glacial deposits. These glacial deposits consist mainly of stratified accumulations of unconsolidated sand and gravel. Some of them represent a ground moraine sorted by the action of glacial meltwaters; others form prominent terraces along the shores of Lake Nipigon and in the valley occupied by Keemle and Wanogu Lakes, and are abandoned beach deposits. Esker ridges also are present but are not high and do not extend for any great distances."

Topography of the Georgia Lake Property is moderate. The minimum elevation is 250 m and the maximum elevation is 560 m asl. Thus, the range is 310 m. The low-lying areas are, typically underlain by metasediments and the higher areas are underlain by Nipigon diabase.

INFRASTRUCTURE AND LOCAL RESOURCES

The village of Beardmore is the closest community, located approximately 16 km north of the Georgia Lake Property. Field programs are based out of the town of Beardmore where there is a restaurant, hotel and Rock Tech's core shack. Beardmore is part of Greenstone, an amalgamated town encompassing Nakina, Geraldton, Longlac, Beardmore, Caramat, Jellicoe, Macdiarmid and Orient Bay. The population of Greenstone is 4,906 people (Statistics Canada, www.statcan.gc.ca) and the population of Beardmore is approximately 200 people (http://www.highway11.ca/ThunderBay/06Beardmore). Beardmore has limited accommodation and restaurants.

The town of Nipigon has most of the basic supplies needed for exploration work in the Georgia Lake area. Nipigon has grocery stores, a hardware store, restaurants, hotels, a hospital and an OPP station.

Nipigon is located 50 km south of the property. The population for Nipigon Township is 1,752 people in 2006 (Statistics Canada, www.statcan.gc.ca). The city of Thunder Bay also has all of the required supplies for exploration work including grocery stores, hardware stores, exploration equipment supply stores, restaurants, hotels, a hospital, OPP stations and an international airport with daily flights to Toronto, Ontario and Winnipeg, Manitoba and the United States. The population of the city of Thunder Bay was 109,140 people in 2006 (Statistics Canada, www.statcan.gc.ca). Many junior exploration and mining companies are based in Thunder Bay, and thus the city is a source of skilled mining labour. Thunder Bay is a transportation hub for Canada, as the TransCanada highways 11 and 17 link eastern and western Canada. Thunder Bay is close to the Canada-U.S. border and highway 61 links Thunder Bay with Minnesota, United States. Thunder Bay is also the largest outbound port on the St. Lawrence Seaway system which ships dominantly grain and pulp and paper on Lake Superior.

There is a power line that runs along the TransCanada highway #11 about 10 km from the property. There are three hydroelectric stations on the Nipigon River, all of which are controlled remotely by the headquarters in Thunder Bay: Alexander Station with 68 MW output (17 km north of the town of Nipigon), Cameron Falls with 87 MW output (17 km north of the town of Nipigon) and Pine Portage with 142 MW (39 km north of the town of Nipigon) (http://www.opg.com/power/hydro/northwest_plant_group/).

There are several lakes, rivers and creeks on the Georgia Lake Property. The lakes on the Nama Creek, Conway and McVittie claim blocks include the Postagoni Lake, Pawky Lake, Dump Lake, Downey Lake, Palace Lake, Pain Lake, Piper Lake, Parsnip Lake, Dive Lake and Pennon Lake. The Little Postagoni River, Phantom Creek and Palace Creek cross the northwestern claim block. Water on the Jean Lake – Foster-Lew property is available from Lake Jean, Parland Lake, Peanut Lake, Parole Lake, Piece Lake, Pomace Lake, Woodpigeon Lake, Pound Creek and Pomace Creek. The source of water on the Aumacho and MNW claim blocks include the Cosgrave Lake, Blay Lake, Hansen Lake, Claus Lake, Pond Lake, Abner Lake, Jackfish River, Namewaminiken River, Hansen Creek and Dot Creek. There is an unnamed creek and a small unnamed lake on the Newkirk-Vegan dispositions.

Rock Tech's Georgia Lake project is in the exploration stage and had a NI 43-101 compliant reserve from 2012 (CCIC report 2012). In 2018 DMT conducted a renewal of a NI43-101 compliant resource report. This study will be the first economical study in form of a Preliminary Economic Assessment.

The potential Processing Facility is planned to be located close to the Main Zone North deposit with the potential tailings storage area and waste disposal area.

Rock Tech has surface rights for Nama Creek dispositions and the crown owns the surface rights for all other dispositions and claims.

A comprehensive drilling programme was carried out in 1955 and 1956 to the Spodumene pegmatites after these had been discovered during general prospection work. Based on these results a first resource estimate was prepared. The key assumptions, parameters and methods used to prepare the historical estimates are unknown. These historical resource estimates do not use the categories outlined in Sections 1.2. and 1.3. of NI 43-101: Standards of Disclosure for Mineral Projects. In other words, these historical resources are not NI43-101 compliant.

Table 13 and Table 14 summarizes the known historical resource estimates for the properties until 1965 (Pye, 1965).

Property	Zone	Owner	Estimated Reserves (ton)	Average Li_2O wt. %	Reference
Nama Creek	Main Zone/ North (MZN)	Nama Creek Mines Ltd.	2,784,000	1.11	Isaacs, R. J. (1955): Unpublished company report, Nama Creek Mines Ltd. (Pye, 1965)
Nama Creek	Main Zone/ Southwest (MZSW)	Nama Creek Mines Ltd.	1,508,332	0.96	Isaacs, R. J. (1955): Unpublished company report, Nama Creek Mines Ltd. (Pye, 1965)
Nama Creek sum	North and Southwest (MZN+MZSW)	Nama Creek Mines Ltd.	4,292,332	1.06	Isaacs, R. J. (1955): Unpublished company report, Nama Creek Mines Ltd. (Pye, 1965)
Conway	Conway (CON)	E.S.Conway/ Leitch Gold Mines Ltd.	1,830,000	0.96	Pye, E. G. (1965): Personal Communications with G. A. McKay, Manager, Leitch Gold Mines Ltd.

Table 13. Historical resource ("reserve") estimates for the Nama Creek and Conway Property (Pye, 1965)

In consequence, additional drilling has been completed by Rock Tech in order to upgrade the historical resources to the mineral resource estimate from 2012 for MZN, MZSW, HAR, LIN, and CON (Selway et al., 2012b, CCIC).

Property	Resource Class	Tonnage [Mt]	Grade Li_2O [%]
MZN	Indicated	2.47	1.11
Conway	Indicated	0.72	1.05
Total	Indicated	3.19	1.10
MZN	Inferred	2.50	0.98
Conway	Inferred	0.59	1.02
Line60	Inferred	1.30	0.93
MZSW	Inferred	0.97	1.09
Harricana	Inferred	0.95	1.03
Total	Inferred	6.31	1.00

Table 14. Mineral resource statement1 (CICC, Aug. 29th, 2012) reported at a cut-off grade of 0.6 Li_2O%

The data acquisition done from 2009 to 2012 is well documented including a comprehensive QA/QC management to validate the acquired data. Twinning could reproduce logged mineralization and assays of historical holes in acceptable ranges. However, there seems to be an offset in location, which might be explained by an error in survey of collar position or drill path. However, this is estimated not to have a significant effect on the tonnage and grade estimates based on the 2012 NI 43-101 resource model and estimate. A model vali-

dation chapter or cross sections demonstrating that the resource model (interpreted wireframes or interpolated block values) is representative to the primary data was not documented in this report. Table 15 shows the old block model tonnages and grades reported at various cut-off grades of Li_2O% of the five pegmatites shown in the report from CICC, Aug. 29th, 2012.

Property	Cut – Off Li_2O%	Resource Class	Tonnage [Mt]	Grade Li_2O [%]
MZN	0.4	Indicated	2.50	1.10
MZN	0.6	Indicated	2.47	1.11
MZN	0.8	Indicated	1.89	1.14
MZN	1.0	Indicated	1.69	1.22
Conway	0.4	Indicated	0.73	1.05
Conway	0.6	Indicated	0.72	1.05
Conway	0.8	Indicated	0.65	1.09
Conway	1.0	Indicated	0.36	1.24
MZN	0.4	Inferred	2.67	0.95
MZN	0.6	Inferred	2.50	0.98
MZN	0.8	Inferred	1.76	1.09
MZN	1.0	Inferred	1.23	1.20
Conway	0.4	Inferred	0.62	1.00
Conway	0.6	Inferred	0.59	1.02
Conway	0.8	Inferred	0.50	1.07
Conway	1.0	Inferred	0.31	1.16
Line	0.4	Inferred	1.93	0.79
Line	0.6	Inferred	1.30	0.93
Line	0.8	Inferred	0.83	1.06
Line	1.0	Inferred	0.49	1.17
MZSW	0.4	Inferred	0.97	1.09
MZSW	0.6	Inferred	0.97	1.09
MZSW	0.8	Inferred	0.88	1.13
MZSW	1.0	Inferred	0.57	1.23
Harricana	0.4	Inferred	0.95	1.03
Harricana	0.6	Inferred	0.95	1.03
Harricana	0.8	Inferred	0.84	1.07
Harricana	1.0	Inferred	0.54	1.15

Table 15. Block model tonnages and grades reported at various cut-off grades of Li_2O% per pegmatite (CICC, Aug. 29th, 2012)

REGIONAL GEOLOGY

The Georgia Lake area is located within the Quetico Subprovince of the Superior Province. The Quetico Subprovince is bounded by the granite-greenstone Wabigoon Subprovince to the north and Wawa Subprovince to the south (Williams, 1991). The Quetico Subprovince is composed of predominantly metasediments consisting of wacke, iron formation, conglomerate, ultramafic wacke and siltstone, which deposited between 2.70 and 2.69 Ga. The igneous rocks in the Quetico Subprovince include abundant felsic and intermediate intrusions, metamorphosed rare mafic and felsic extrusive rocks and an uncommon suite of gabbroic and ultramafic rocks. The earlier felsic intrusions occurred 5 to 10 million years after the accumulation of sediments and are interpreted to be I-type intrusions (White and Chapell, 1983). The later felsic intrusions occurred 20 million years after the sedimentation and are designated as S-type (White and Chapell, 1983).

The Quetico Subprovince was subjected to four deformational events between approximately 2700 and 2660 million years (Williams, 1991). The predominant stratigraphic-facing direction is north (Carter, 1984, 1987, 1988; Harris, 1970; Perdue, 1938; Williams, 1988). Regional schistosity is variably developed and oriented and is interpreted to be the result of regional shortening and dextral shearing.

Four major faults cut through the Quetico Subprovince: the easterly trending Quetico fault (Fumerton, 1982; Bau, 1979; Kennedy, 1984), the Rainy Lake-Seine River fault (Fumerton, 1982, Davis et al., 1989), the northeasterly trending Gravel River fault (Williams, 1989) and the Kapuskasing Structural Zone (Percival, 1989).

Metamorphism, migmatite formation and granite intrusion occurred between 2.67 and 2.65 Ga (Williams, 1991). The grade of metamorphism ranges from lower greenschist to amphibolite facies and tends to be lower in the marginal rocks of the subprovince and higher in the core regions (Percival, 1989).

Widespread economic mineralization within the Quetico Subprovince is generally lower than in the adjacent greenstone dominated terranes (Williams, 1991). Minor gold mineralization is associated with veining along the Quetico Fault (Poulsen, 1983). Molybdenite occurs in biotite leucogranites in the Dickinson Lake area (Carter, 1975, 1985).

The only potentially important ore deposit type consists of the late-stage pegmatites that contain the rare elements lithium, beryllium, tantalum, niobium and tin (Williams, 1991). The rare-element pegmatites have widespread distribution in the Quetico Subprovince covering at least a 540 km strike length from west to east and a large percentage of pegmatites occur in the centre of the subprovince (Breaks, Selway and Tindle, 2005):

- Spodumene-subtype pegmatites at Wisa Lake, Lac La Croix area
- Fertile granites and beryl-type pegmatites in Niobe-Nym lakes and Onion Lake areas
- Albite-Spodumene-type pegmatites of the Georgia Lake area
- Complex-type, lepidolite subtype Lowther Township pegmatite near Hearst (Breaks, Selway and Tindle, 2003a)

The pegmatites in the Quetico Subprovince are hosted by metasediments and/or by their parent granite (Pye, 1965; Breaks, Selway and Tindle, 2003a, 2003b).

LOCAL GEOLOGY

The geology of the Georgia Lake area is of Precambrian age and is discussed by Pye (1965). The local rocks were build up from the following types:

METASEDIMENTS

The oldest rocks are the Archean metasediments. The metasediments strike east-northeast and dip steeply, in general, to the north. The dominant metasedimentary rock is biotite-quartz-feldspar schist or gneiss. It is a grey, rather dark coloured rock, having a distinct banded appearance due to compositional variations reflecting an original sedimentary stratification, with individual layers less than an inch to several feet thick.

There is a distinct foliation due to parallel alignment of biotite crystals. Microscopic examination of the biotite-quartz-feldspar schist shows that it is made up of: 15-40 vol.% biotite , 20-35 vol.% quartz, 25- 45 vol.% plagioclase, 1-3 vol.% magnetite, trace amounts of zircon and rare hornblende. Secondary minerals include chlorite, sericite and epidote. The plagioclase shows myrmekite texture. The most abundant texture in the biotite-quartz-feldspar schist or gneiss is granoblastic, but porphyroblastic rocks are also present with porphyroblasts of garnet, staurolite and cordierite.

METAGABBRO

The metagabbro has intrusive relationships and have been metamorphosed and intruded by granitic rocks. East of Cosgrave Lake and south of Barbara Lake, the metasediments were intruded by metagabbro. The metagabbro bodies range in size from a few hundred feet across to 9,500 feet (=2.9 km) across. The metagabbro is dark-coloured (mesocratic), medium- to coarse-grained with a brownish weathered surface. For the most part, it is massive, but it is gneissic near its contacts with metasediments. The major minerals are: green hornblende and plagioclase (sodic andesine). The minor minerals include: microcline and biotite and trace amounts of magnetite and apatite. The alteration minerals are chlorite, epidote and sericite.

The porphyritic metagabbro differs from the metagabbro only in the presence of feldspar phenocrysts (usually microcline). The feldspar phenocrysts are pale-pink to red, stubby, rectangular, subhedral to euhedral and range in size from ¼ by 1/8 inch (=0.6 by 0.3 cm) to 2 by 1 inches (5 by 2.5 cm). The porphyritic metagabbro is best developed near the margins of the metagabbro bodies close to the granites. Metagabbro dykes and sills cross cut the metasediments near Dump and Pawky lakes and near Blay, Georgia and Conner lakes. All of the dykes and sills are small with ticknesses of 3 feet or less (=0.9 m). They are thought to be genetically related to the metagabbro, as they are similar in appearance and composition. They are cross cut by pegmatite and feldspar porphyry dykes.

GRANITE

The metasediments were also intruded by large masses of granitic rocks and by numerous sills and dykes of genetically-related porphyry, pegmatite and aplite. The granitic rocks are pale-grey or pale-pink in color and their essential components are: 45-65 vol.% feldspar (microcline and plagioclase), 40 vol.% quartz, and one or both of muscovite and biotite and rarely little hornblende. The plagioclase has a composition of albite. Minor components of the granites include magnetite, zircon, and garnet, and secondary minerals: chlorite, sericite and epidote. For the most part the granites are equigranular, but porphyritic phases with microcline

phenocrysts also occur. The contacts between the equigranular granitic rocks and the metasediments are generally abrupt.

PEGMATITE

There is an abundance of pegmatites close to and within the large masses of granitic rocks. A regional zoning is apparent and a genetic association of pegmatites and granite is indicated. The pegmatites occur in two geometries: as irregular-shaped bodies and as thin dykes, sills and attenuated lenses. The irregular bodies of pegmatite are intimately associated with the granite bodies often within a few hundred feet of the contact zone. They typically are medium- to coarse-grained, up to very coarse-grained and are made up of quartz, microcline, perthite and little muscovite. These would be classified as potassic pegmatites. Accessory minerals include biotite, tourmaline and garnet.

The pegmatite dykes, sills and lenses can be subdivided into rare-element pegmatites and granitic pegmatites. The rare-element pegmatites are of economic significance and they contain microcline or perthite, albite, quartz, muscovite and Spodumene and minor amounts of beryl, columbite-tantalite and cassiterite. The granitic pegmatites are similar to the irregular pegmatites described above except that they contain more abundant plagioclase. Some of the pegmatites are parallel to the foliation or bedding of the metasediments, whereas others occur in joints in either the metasediments or granite. Contacts are usually sharp and, except where dykes cut granitic rocks, often found to be marked by a thin border zone of aplite or granitoid composition. A few pegmatites are internally zoned with mica-rich or tourmaline-rich rock along or close to the walls and quartz cores.

SEDIMENTARY ROCKS

The Proterozoic is represented by sedimentary rocks (sandstone and shale). Since these are not present on the Georgia Lake Lithium property, they are not discussed here and the reader is referred to Pye (1965) for more information on them.

DIABASE

Intrusive into the Proterozoic sedimentary rocks and the older formations are bodies of diabase. The largest occur as flat sheets (Logan sills), up to about 650 ft (=198.1 m) in thickness, and as dykes of vertical or near-vertical attitude. Most of the dykes are related closely to the sheets and are Keweenawan age. The gently dipping diabase sheets are dark colored and massive. The diabase sheets are well-jointed and most of the joints are vertical or steeply dipping. In outcrop, the diabase shows poorly-formed columnar structure.

There are two types of diabase dykes: one is equigranular and the other is porphyritic. The equigranular dykes are more abundant. Some of the dykes along or close to the contact zone of the large granite mass strike easterly; most dykes in other localities strike north or within 20º of north. With few exceptions the dykes are vertical or dip steeply. The porphyritic diabase dykes are massive medium-grained, darkcolored rock characterized by many pale-greenish yellow phenocrysts of highly altered plagioclase. Porphyritic diabase dykes are found near the MZN pegmatite called from the geologists in the 1955 reports the “Jackpot”.

The following figures show an overview of the geology of the area and the location of the claims and dispositions from Rock Tech in detail. Rock Tech owns rights to 283 claims (56.93 km^2) and 81 dispositions (10.42 km^2).

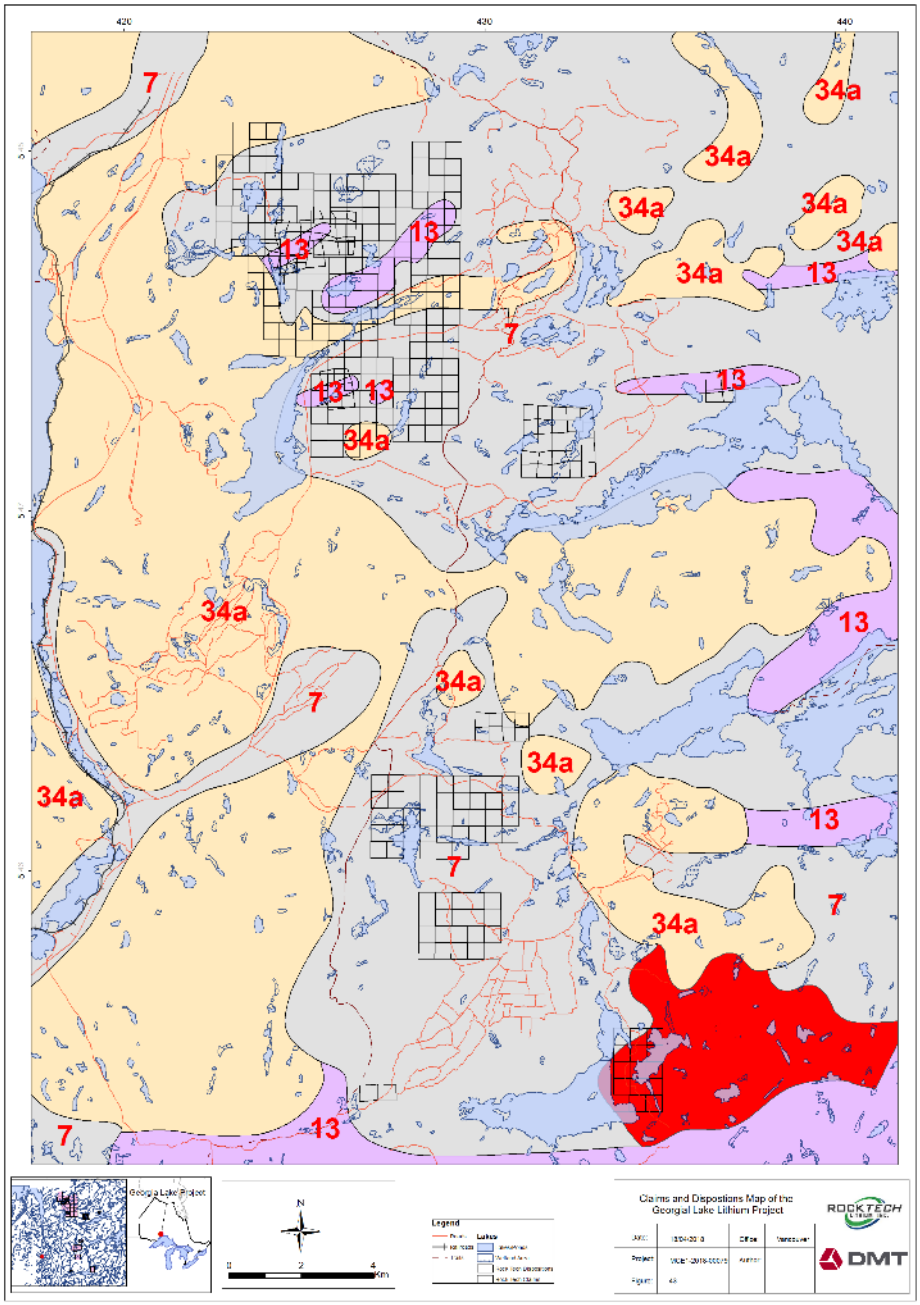


Figure 14. Overview of the local Geology. Red numbers show the rock type and are listed in the text below. Black rectangles show the claims and dispositions of Rock Tech. The shown railway track parallel to the high way 11 as of today is out of order.

On surface glacial sediments dominate. Only in some areas host rocks are exposed. In these areas with out-crop, lithium-bearing pegmatites were found during reconnaissance work, which started in places some 60 years back. These occurrences of Spodumene pegmatites were drill-tested in order to get information about thickness, grade and orientation of the pegmatite bodies.

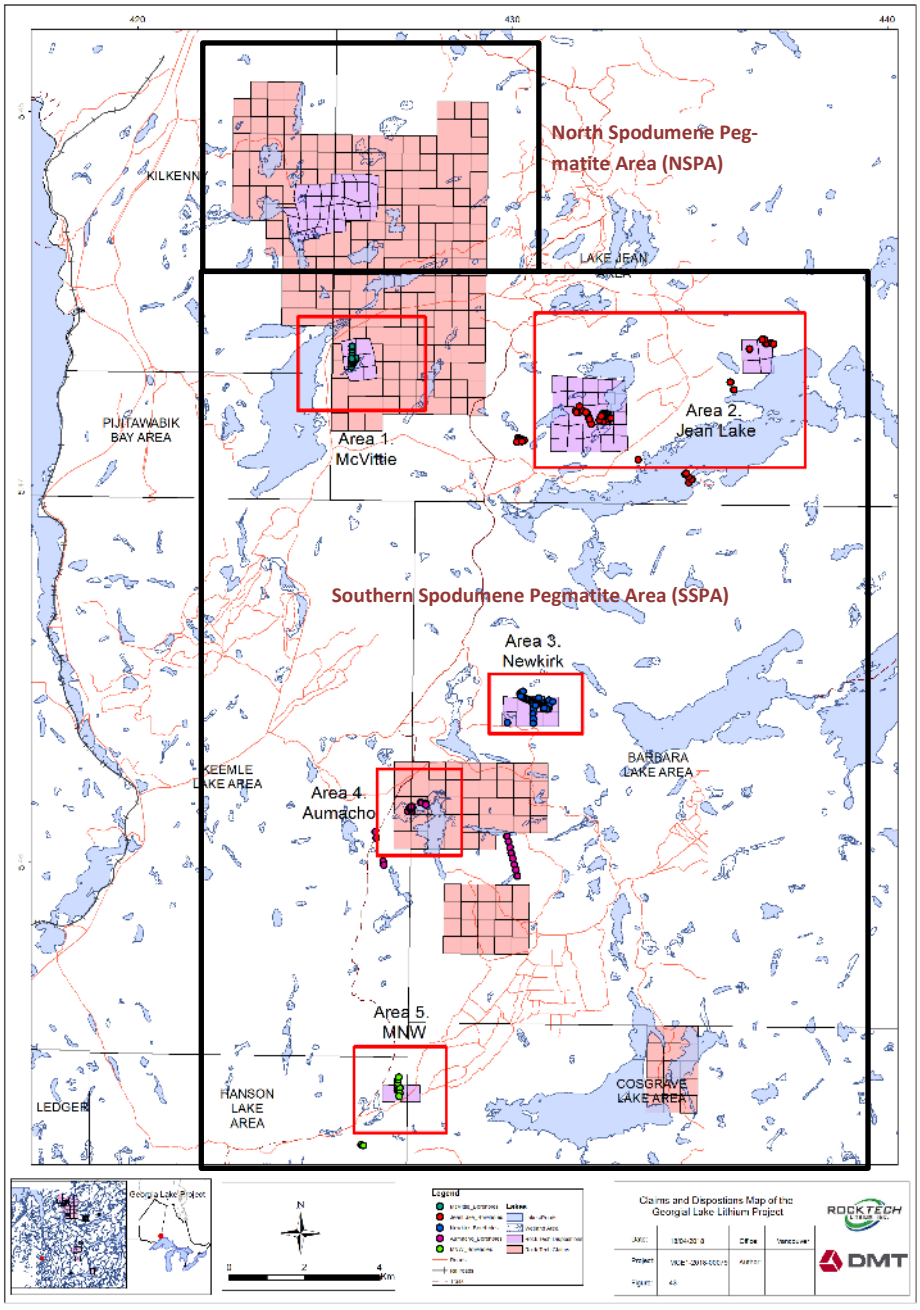


Figure 15. Overview of sub areas and drillhole locations in the SSPA. The shown railway track parallel to the high way 11 as of today is out of order.

Figure 14 shows the geology of the area and the claim boundaries in black. The lithological units, as shown on Figure 14 are explained in the following:

- 7 (grey): metasedimentary rocks
- 34a (light brown): Logan and Nipigon sills / diabase sills
- 13 (magenta) : Muscovite-bearing granitic rock
- 15 (red) : massive granodiorite to granite rock

The whole property are divided in a northern area NSPA and a southern area SSPA. In the SSPA 5 subareas are shown in Figure 9. In these subareas Spodumene pegmatites occur at the surface and sampling and drilling was done. Figure 15 shows an overview of the boreholes in the SSPA as well.

In the SSPA South are the five subareas with pegmatites located. These areas are: McVittie (area 1), Jean Lake east and west (area 2), Newkirk (area 3), Aumacho (area 4) and MNW (area 5) see Figure 15.

In some cases drilling and channel sampling has been executed during the latest exploration phase. The next figures show an overview of these activities in the different areas. All these areas belong to Rock Tech.

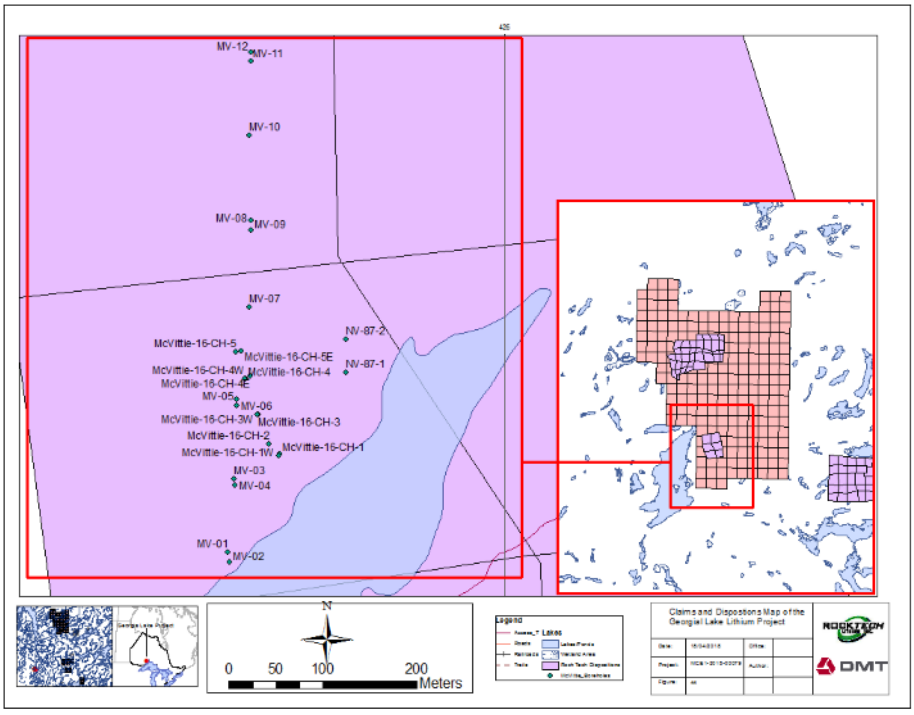


Figure 16. Drill hole and channel locations of the McVittie area

The shape of the McVittie pegmatite has been defined through field investigations and the chemical analyses. The north south strike length on the map is about 400 m. The deepest drillhole intersection is about 100 m below surface. The drillhole data shows in average true thickness of 4 m of intersected Spodumene pegmatite.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
1955/66	12			
1987	2			
2016		10		
Total	**14**	**10**	**1243.3**	**40.86**

Table 16. Numbers of drill holes and channel samples in the McVittie area during the different exploration phases

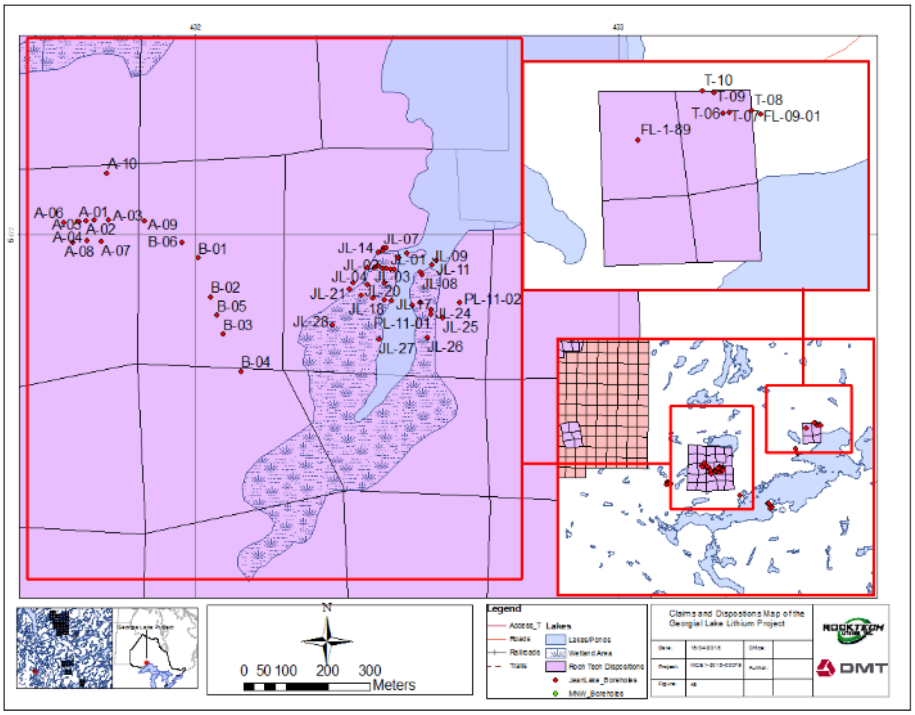


Figure 17. Drill hole and channel locations of the Jean Lake area.

The Jean Lake pegmatite area is divided in a western and eastern part. The shape of the western Jean Lake pegmatite has been defined through field investigations and chemical analyses. The east west strike length on the map is about 350 m. The deepest drillhole intersection is about 200 m below surface. The drillhole data shows in average true thickness of 6 m of intersected Spodumene pegmatite.

The east Jean Lake pegmatite was drilled in the past. Over a length of 600 m Spodumene Pegmatite are outcropping. The results were confirmed by 6 intersections within historic drillholes. No analyses were recorded. Therefore this pegmatite has been excluded from the resource statement. Further exploration work is recommended.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
1955/66/65	49			
1989	1			
2009/2011	2			
2016		2		
2017	7			
Total	**59**	**2**	**9275.63**	**7.23**

Table 17. Numbers of drill holes and channel samples of the area Jean Lake during the different exploration phases

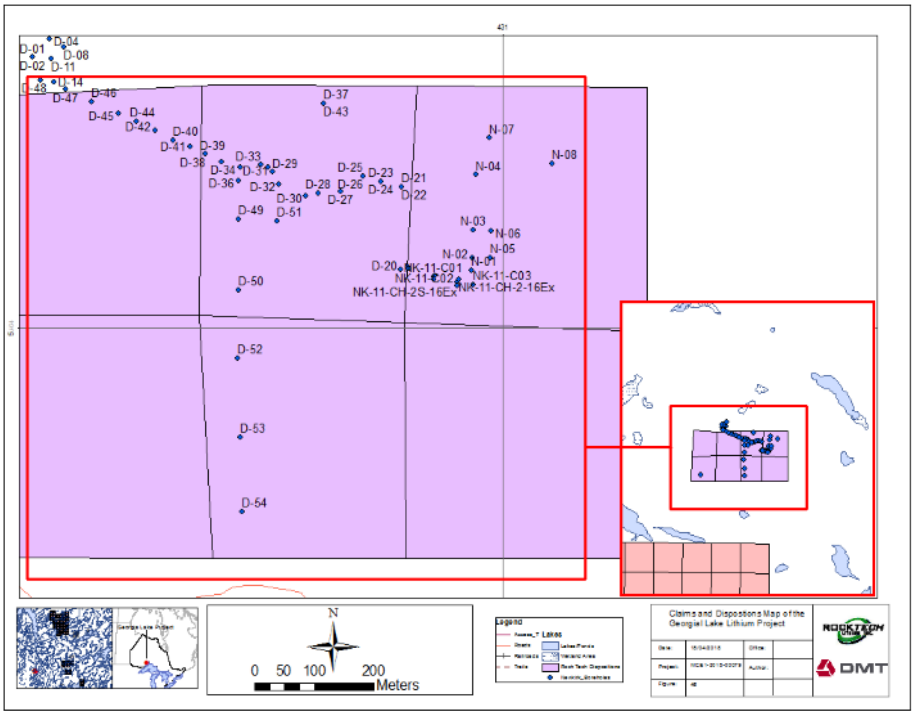


Figure 18. Drill hole and channel locations of the Newkirk area

The shape of the Newkirk pegmatite has been defined through field investigations and chemical analyses. The east west strike length on the map is about 900 m. The deepest drillhole intersection is about 150 m below surface. The drillhole data shows an average true thickness of 3 m of Spodumene pegmatite.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
1955/56	42			
2011		3		
2016		6		
Total	**42**	**9**	**2997.5**	**46.26**

Table 18. Numbers of drill holes and channel samples in the area Newkirk during the different exploration phases

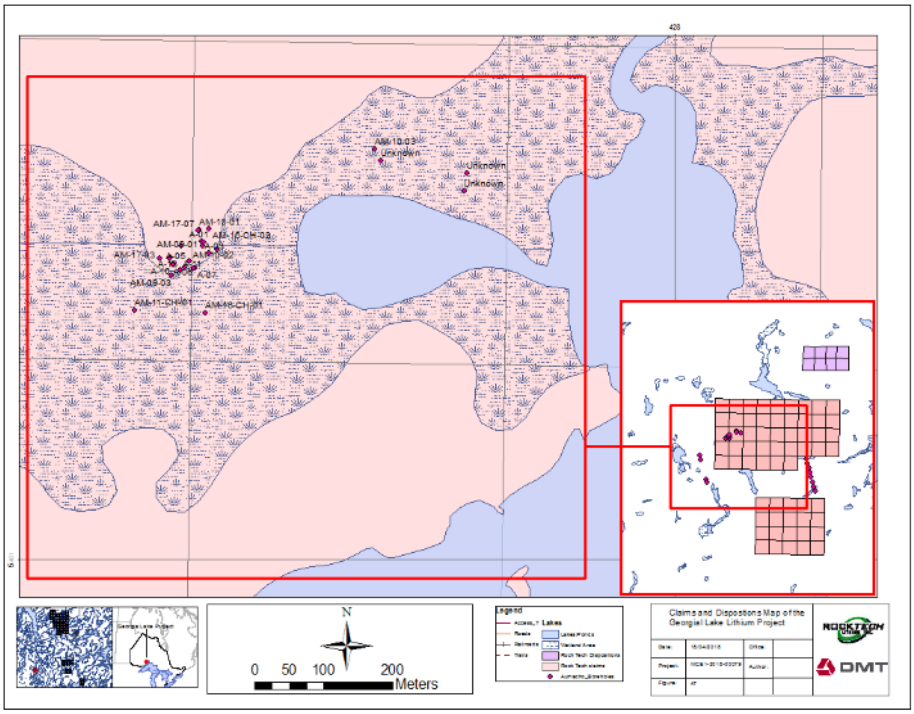


Figure 19. Drill hole and channel locations of the Aumacho area.

The shape of the western Aumacho pegmatite has been defined through field investigations and chemical analyses. The north south strike length on the map is about 250 m. The deepest drillhole intersection is about 90 m below surface. The drillhole data shows an average true thickness of 4 m of Spodumene pegmatite.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
1955	16			
2009 to 2011	6		1	
2016			3	
2017	7			
Total	**29**	**4**	**2107.9**	**24.07**

Table 19. Numbers of drill holes and channel samples in the area Aumacho during the different exploration phases.

The MNW pegmatite is situated in the southernmost part of the claims and disposition area of SSPA. Data from historic and recent drillholes are present. However, the historic drillhole results are insufficient to outline shape of the pegmatite. Therefore this pegmatite has been excluded from the resources estimate. Further exploration work is recommended.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
1956	11			
Total	**11**		**616.9**	

Table 20. Numbers of drill holes and channel samples in the area MNW during the different exploration phases

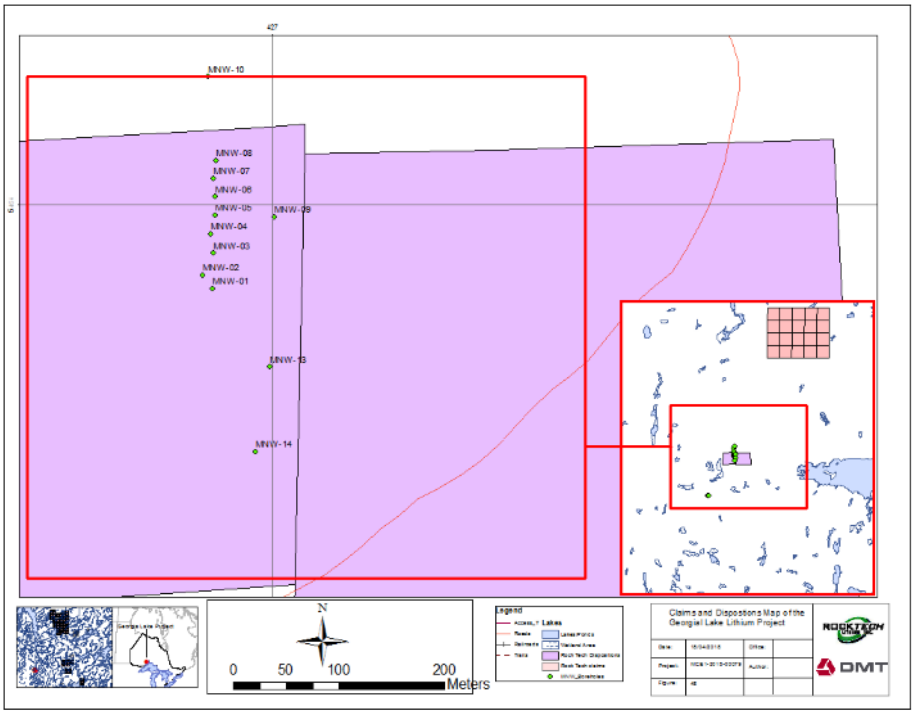


Figure 20. Drill hole and channel locations of the MNW area.

RARE-ELEMENT PEGMATITES OF SUPERIOR PROVINCE

Rare-element pegmatites may host several economic commodities, such as tantalum (Ta-oxide minerals), tin (cassiterite), lithium (ceramic-grade Spodumene and petalite), rubidium (lepidolite and K-feldspar), and cesium (pollucite) collectively known as rare elements, and ceramic-grade feldspar and quartz (Selway et al., 2005). Two families of rare-element pegmatites are common in the Superior Province, Canada: Li-Cs-Ta enriched ("LCT") and Nb-Y-F enriched ("NYF"). LCT pegmatites are associated with S-type, peraluminous (Al-rich), quartz-rich granites. S-type granites crystallize from a magma produced by partial melting of preexisting sedimentary source rock. They are characterized by the presence of biotite and muscovite, and the absence of hornblende. NYF pegmatites are enriched in rare earth elements ("REE"), U, and Th in addition to Nb, Y, F, and are associated with A-type, subaluminous to metaluminous (Al-poor), quartz-poor granites or syenites (Černý, 1991a).

Rare-element pegmatites derived from a fertile granite intrusion are typically distributed over a 10 to 20 km^2 area within 10 km of the fertile granite (Breaks and Tindle, 1997). A fertile granite is the parental granite to rare-element pegmatite dykes. The granitic melt first crystallizes several different granitic units (e.g., biotite granite to two mica granite to muscovite granite), due to an evolving melt composition, within a single parental fertile granite pluton. The residual melt enriched in incompatible elements (e.g., Rb, Cs, Nb, Ta, Sn) and volatiles (e.g., H_2O, Li, F, BO_3, and PO_4) from such a pluton can then migrate into the host rock and crystallize pegmatite dykes. Volatiles promote the crystallization of a few large crystals from a melt and increase the ability of the melt to travel greater distances. This results in pegmatite dykes with coarse-grained crystals occurring in country rocks considerable distances from their parent granite intrusions.

There are several geological features that are common in rare-element pegmatites of the Superior province of Ontario (Breaks and Tindle, 2001; Breaks et al., 2003) and Manitoba (Černý et al., 1981; Černý et al., 1998) (Selway et al., 2005):

1) Subprovincial boundaries: The pegmatites tend to occur along subprovincial boundaries.
2) Metasedimentary-Dominant Subprovince: Most pegmatites in the Superior province occur along subprovince boundaries, except for those that occur within the metasedimentary Quetico subprovince.
3) Greenschist to Amphibolite Metamorphic Grade: Pegmatites are absent in the granulite terranes.
4) Fertile Parent Granite: Most pegmatites in the Superior province are genetically derived from a fertile parent granite.
5) Host Rocks: Highly fractionated Spodumene- and petalite-subtype pegmatites are commonly hosted by mafic metavolcanic rocks (amphibolite) in contact with a fertile granite intrusion along subprovincial boundaries. Pegmatites within the Quetico subprovince are hosted by metasedimentary rocks or their fertile granitic parents.
6) Metasomatized Host Rocks: Biotite and tourmaline are common minerals, and holmquistite is a minor phase in metasomatic aureoles in mafic metavolcanic host rocks to Spodumene- and petalite-subtype pegmatites. Tourmaline, muscovite, and biotite are common, and holmquistite is rare in metasomatic aureoles in metasedimentary rocks.

7) Li Minerals: Most of the complex-type pegmatites of the Superior province contain Spodumene and/or petalite as the dominant Li mineral, except for a few pegmatites, which have lepidolite as the dominant Li mineral.
8) Cs Minerals: Cesium-rich minerals only occur in the most extremely fractionated pegmatites.
9) Ta-Sn Minerals: Most pegmatites in the Superior province contain ferrocolumbite and manganocolumbite as the dominant Nb-Ta-bearing minerals. Some pegmatites contain manganotantalite or wodginite as the dominant Ta-oxide mineral. Tantalum-bearing cassiterite is relatively rare in pegmatites of the Superior province.
10) Pegmatite Zone Hosting Ta Mineralization: Fine-grained Ta-oxides (e.g., manganotantalite, wodginite, and microlite) commonly occur in the aplite, albitized K-feldspar, mica-rich, and Spodumene core zones in pegmatites in the Superior province.

GEORGIA LAKE PEGMATITE FIELD

The majority of the pegmatites in the Postagoni Lake group and Georgia Lake group can be classified as albite-Spodumene type pegmatites. Albite-Spodumene type pegmatites are characterized by homogenous dykes with coarse-grained Spodumene + K-feldspar aligned perpendicular to the dyke walls, Spodumene is the dominant or only Li-bearing mineral and albite is more abundant than K-feldspar.

The Aumacho – Brink pegmatite is classified as a Spodumene-subtype pegmatite. Spodumene-subtype pegmatites are characterized by complex internal zonation, Spodumene is the dominant Li-bearing mineral and albite is more abundant than K-feldspar.

The MNW pegmatite in the south of the hole claim area is classified as a petalite-subtype pegmatite. Petalite-subtype pegmatites are characterized by complex internal zonation, petalite is the dominant Li-bearing mineral and K-feldspar is more abundant than albite. Often petalite is rare in the pegmatite, and SQUI (Spodumene-quartz intergrowth due to the breakdown of petalite) is common instead.

In general, mapping, trenching and drilling has been carried out in 1955/1956 and again starting from 2009 in the area relevant for resource modelling and estimation. No geophysical surveys were applied during that exploration phase in these areas. In other parts of the property, geophysical ground surveys have been done, e.g. magnetic, electromagnetic. However, results were not significant (CCIC, 2012).

From 2014 until 2018 additional mapping, drilling and channel sampling take place. (see the individual figures and tables in Chapter 7).

The number of all boreholes and channels of the whole area are listed in the Table 21. In all exploration claims and dispositions 47.4 km of drillholes and channels were drilled and cut during since the exploration began in early fifties of the last century.

Year	Boreholes	Channels	Sum Drilled (m)	Channel sampled (m)
not recorded	33		2,340	
1955/56	205		28,677	
1957/58	26		1,787	
1987/89	3		199	
2009 to 2011	70	74	12,409	456
2016/2017	14	106	1,972	442
TOTAL	**351**	**180**	**47,384**	**898**

Table 21 Number of all boreholes and channels in the whole area of Rock Tech.

Starting from November 2009, 62 NQ drill holes (11 588 m) have been drilled in the 5 properties MZN, MZSW, HAR, LIN and CON. Logging and sampling followed standard operating procedures (SOPs) implemented and supervised by CCIC during the main programs in 2010 and 2011. Both programs were laid out for resource defintion and historical database confirmation.

AREA	Number of Boreholes	Meters drilled
MZN	32	6,775
MZSW	4	728
HAR	6	966
LIN	5	808
CON	15	2,311
TOTAL	**62**	**11,588**

Table 22. Drilling done since 2009

Additionally, 138 channels (722 m) have been cut to investigate the outcropping Spodumene pegmatite.

AREA	Number of Trenches	Meters trenched
MZN	18	144
MZSW	28	77
HAR	31	134
LIN	39	295
CON	22	72
TOTAL	**138**	**722**

Table 23. Trenches done since 2009

These data were used to confirm historic data and extent the data base. In total, 113 historic drill holes (17,614 m) are available.

AREA	Number of Boreholes	Meters drilled
MZN	45	8,601
MZSW	13	2,216
HAR	18	2,547
LIN	22	3,305
CON	15	945
TOTAL	**113**	**17,614**

Table 24. Historic drilling done 1955/56

All in all, 175 drill holes totaling to more than 29 000 m were available for the resource estimate for the 5 3D modeled pegmatites from the NSPA.

AREA	Number of Boreholes	Meters drilled
MZN	77	15,376
MZSW	17	2,944
HAR	24	3,513
LIN	27	4,113
CON	30	3,256
TOTAL	**175**	**29,202**

Table 25. Historic and recent drilling

All drill holes were drilled in a sectional pattern with a spacing of some 60 m in between sections and 40 to 60 m along sections. All drill holes were drilled inclined in order to intersect the main pegmatites as perpendicular as possible.

The following Figure 21 give an overview about drill holes locations, spacing and distribution of the MZN pegmatite. The row of outcrops of the pegmatite is shown with the white line.

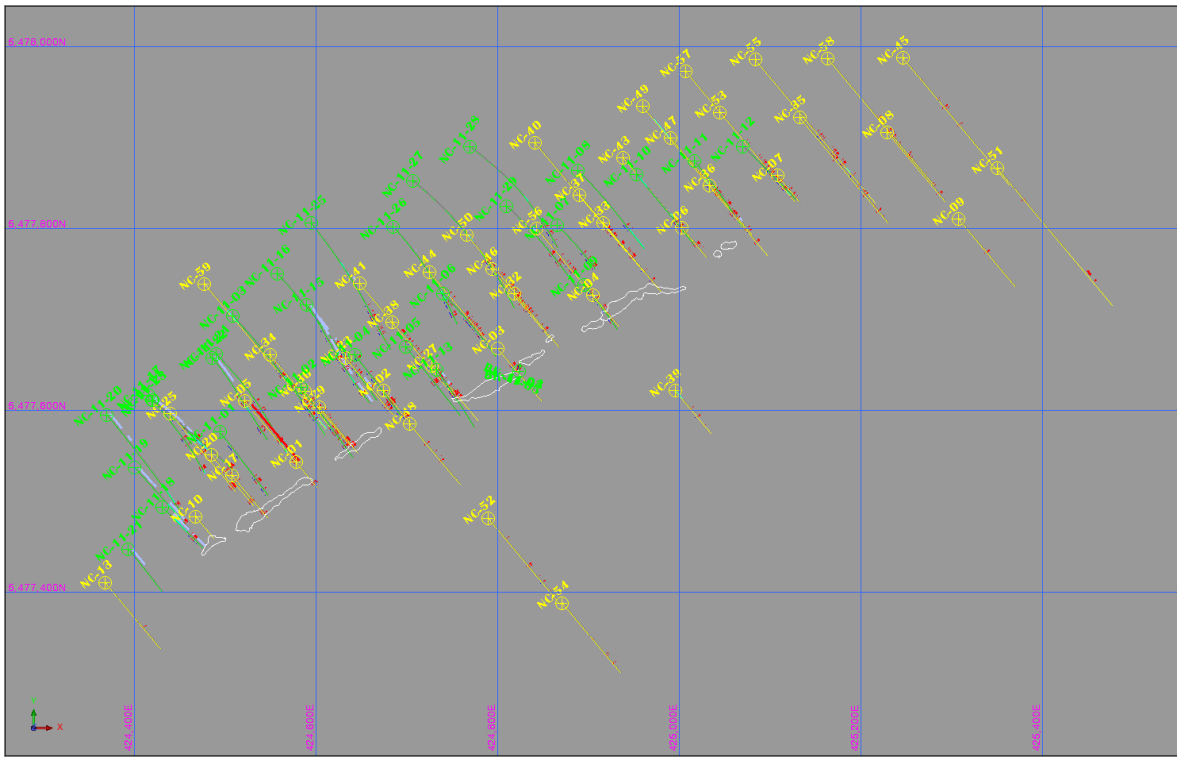


Figure 21. Drill holes in MZN; historic holes (yellow) and recent drill holes (green). Grid spacing 200 m. Outcrop of pegmatite white line.

SOPs were implemented and supervised by CCIC during the main programs of 2009 and 2011 in order to confirm and extent the historical database, for which no procedures or results of any QA/QC management are documented.

All work of data acquisition was done or re-done following these SOPs. For details about the procedures applied see document: 'Independent Technical Report and Updated Resource for Nama Creek Main Zone North Pegmatite Georgia Lake Lithium Property, Beardmore, Ontario, Canada (CCIC, Oct. 5, 2012).

All the drill core samples from the 2010-2011 winter drill program were submitted to SGS Toronto for analysis. Channel samples and drill core of later programs were prepared by Actlabs' preparation lab in Geraldton, Ontario and then shipped to Actlabs' analytical lab, Ancaster, Ontario for analysis. SGS and Actlabs Laboratories are ISO17025 certified.

Once the core samples were received by the laboratory a confirmation of receipt was e-mailed to CCIC. The samples were dried, crushed to 75 %, split and a 250 g aliquot was pulverized to 85 % at 75 µm (SGS sample preparation code PRP89). Samples were weighed (SGS code WGH79) and fused with a sodium peroxide fusion. Trace element analysis was completed with an ICP-AES (SGS code IC90A). Major elements were analyzed using XRF with a tetraborate fusion (SGS code XRF76C).

CCIC inserted standards and blanks into the sample stream in regular intervals: every tenth sample was either a low-grade standard (STDL), a high-grade standard (STDH) or a blank. One in 20 samples was a core duplicate.

SGS included internal blanks, standards: SY4 (certified for 37 ppm Li), NBS 183 (certified for 19,140 ppm Li) and NBS 97B (certified for 550 ppm Li), and pulp duplicates as part of their internal quality control.

Sample preparation for drill core by Actlabs was similar to the SGS' sample preparation. At Actlabs, the entire channel sample is crushed to a nominal minus 10 mesh (1.7 mm), mechanically split (riffled) to obtain a representative sample and then pulverized to at least 95% minus 150 mesh (106 microns). They automatically use cleaner sand between each sample.

The samples were analyzed by Actlabs' Code 8 – REE assay package, which grinds the samples to 95% -200 mesh to ensure complete fusion of resistate minerals. The samples were then digested using lithium metaborate/tetraborate fusion and analyzed the major elements by ICP and trace elements by ICP/MS. The lab used mass balance as a quality control technical and elemental totals of the oxides should be between 98-101 %.

The Li % was analyzed by Actlabs Code 8 – Lithium Ore analysis package which digests the samples by sodium peroxide fusion and analyses them using ICP/OES. The detection limit for Li % was 0.01 %. Selected samples were also analyzed for specific gravity. Actlabs used the following internal standards for the Li analysis: ZW-C (certified for 1.13 %Li), NCS DC86303 (certified for 0.21 % Li), NCS DC86304 (certified for 1.06 % Li), NCS DC8614 (certified for 1.81 % Li). Actlabs used their internal lab tolerance of 95 to 105 % for the pass/fail of the internal standards. Silica flour was used as a standard for the specific gravity measurement. Actlabs also analyzed pulp and preparation duplicates and method blank as part of their quality control. Actlabs uses distilled water as a "method blank" and "sand blank" from sample preparation is also used (CCIC report 2012).

All further exploration work followed these methodology and were checked by local consultants. During the site visit the procedures were discussed with the local experts and analyses from the certificated laboratory show results of the blanks, references and duplicate samples with acceptable deviations to the former results. A peer visual check took place during the site visit in October 2017.

DATA VERIFICATION

Data verification was done on several levels, which are described in detail below.

SITE VISIT

A site visit inspecting the ongoing work of trenching and mapping and existing drill core, logged and sampled since 2009 has been done in October 2017 in the course of the Resource confirmation by DMT's Karl-Stephan Peters, who is a qualified person in accordance to NI 43-101.

According to the requirements of the NI43-101 further visits have been executed in the course of the Preliminary Economic Assessment in September 2018 by Karl-Stephan Peters and Keith McCandlish from DMT Geoscience Canada, a Professional Geologist.

STANDARD OPERATING PROCEDURES (SOPS)

SOPs, set-up specifically for this project, have been implemented and supervised by CCIC in 2010. The SOPs included a comprehensive QA/QC management and thus enables DMT to verify the quality and representativeness of acquired data, e.g. core recovery was noted and QA/QC sample sets were included.

DMT assesses that the procedures, as outlined in this report, are suitable to have produced representative data appropriate for use in a resource estimate.

AVAILABILITY OF DATA

All digital project data were available for this report. The existing database were maintained and all hard-copies were scanned or available in *.pdf or *.doc format. A GIS system is also available. All drill core photos and channel sample photos are stored in a digital format.

For the main investigation area of the modelled 5 pegmatites 195 drill holes (29,428 m) and 118 trenches (496 m) have been drilled and cut in the main north area. The following tabled data sets are available for these holes and channels:

- Collar location and orientation
- Hole deviation
- Drill diameter
- Core recovery
- RQD: geotechnical rock quality data
- Geological logs distinguishing host rocks and lithium mineralization
- Assay data including QA/QC data.
- Assay certificates as PDF signed by SGS and Actlabs and corresponding Excel files including the specific gravity in [t/m3] Li and other chemical parameters:
- Coordinates and elevations of drill hole collars were provided by the client to DMT in map datum UTM NAD83 Zone 16 Northern Hemisphere.

The following GIS data were provided to DMT in map datum UTM NAD83 Zone 16 Northern Hemisphere and have been used for resource modelling

- License areas were provided as vector files. License certificates were provided to DMT as well.
- Mapped outcrops of Spodumene pegmatites and diabase as vector data
- A regional geological map scaled to 1:25 000
- A set of property scales geological maps
- A topographical map scaled to 1:25 000
- A digital terrain model with a grid spacing of around 20 m.

DATA PREPARATION AND MANAGEMENT

All data of drilling and trenching has been provided by the client in a Relational Database Management Software (RDBMS), Microsoft Access, and has been checked by DMT for consistency and completeness. Thereafter, data has been transferred to the modelling software Geovia Surpac to visualize the drill holes in a 3D environment. A digital terrain model (DTM) was also added to Surpac and visualized. Topographic and geological have been draped onto the DTM. Available collar locations were validated against the DTM, surface topographic features, geology mapped and license boundaries.

All these data are the underlying basis for the geological interpretation and wireframe modelling.

DRILLING LOCATION AND ORIENTATION

For all historic and recent drill holes data of collar position and down-hole orientation are available. Trenches were surveyed from start to end and treated as horizontal drill holes.

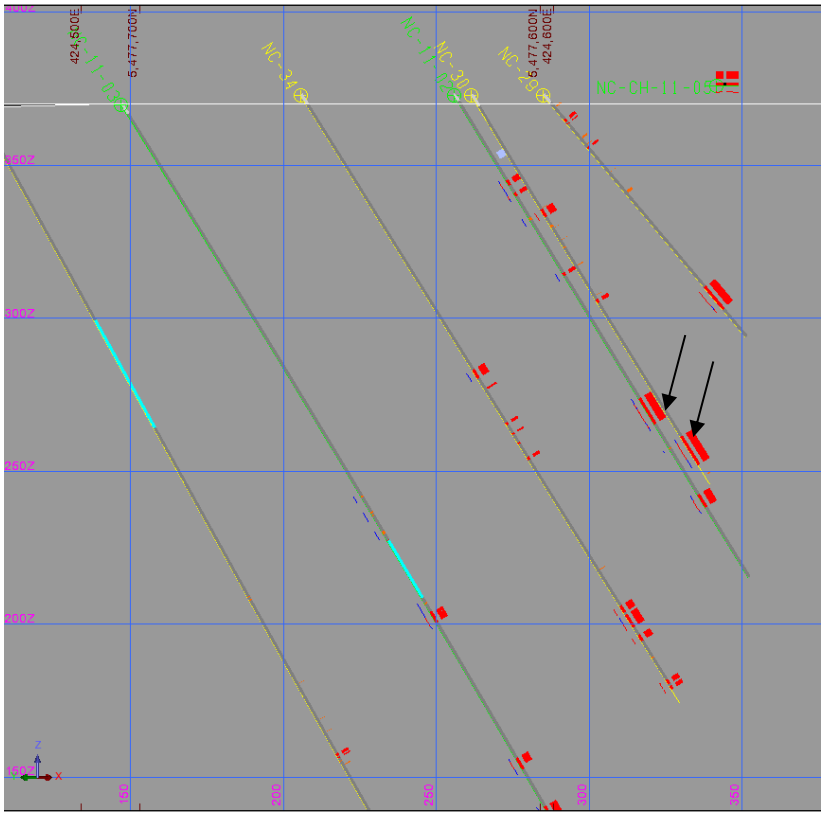


Figure 22. Deviations of intersected mineralization in twin holes (green) and historic holes (yellow). Grid spacing 50 m.

Collar positions of the recent holes were surveyed by Rock Tech using a Trimble Differential GPS. Historic casings were re-surveyed using this Trimble DGPS and surveyed data were entered into the database to ensure that historic collar positions are at a high level of accuracy. However, some twin drilling shows increased deviations of intersected mineralized Spodumene pegmatites. This might be also explained with inaccuracies of either the historic survey or the down-hole surveying.

DRILLING RECOVERY AND DIAMETER

Recent drilling was aimed at maximising sample recovery in order to ensure representative nature of the samples. The overall core recovery is almost 100 % in all drill core, in intervals logged as Spodumene pegmatite, in intervals assayed for Li_2O and in interval interpreted as Spodumene pegmatite bodies. All recent holes are drilled NQ. For historic holes no recovery data are available. In 4 drill holes into MZSW, the recovery was recorded summarily as 100 %. It is assessed that more detailed logging of core recoveries for these 4 holes will not change the general outcome significantly. In consequence, a sample bias caused by poor core recovery can be excluded.

AREA	All drill core within the model [%]	In intervals logged as spodumen pegmatite [%]	In intervals as-sayed for Li_2O [%]	In interval interpreted as mineralized domain [%]
MZN	97	97.3	97	98.4
MZSW	100	100	100	100
HAR	98.5	99	98.5	99.1
LIN	98.8	98.5	98.8	98.3
CON	98.8	98.8	98.8	98.9

Table 26. Core recovery of the recent drillings (2009-20017) which were used for the 3D modelling of the 5 pegmatites

GEOLOGICAL LOGGING

Drill core has been geologically and geotechnical logged to a level of detail, which is assessed as suitable to support geological resource modelling.

Table 27 lists the total lengths and percentages of the relevant rock types intersected and logged.

The following table lists the rock types within the wireframes. It is obvious that also other rocks than Spodumene pegmatite have been included in wireframes, when chemical data of Li_2O were available for these rock types. Rock types logged as pegmatite has been included in some wireframes in order to achieve a lateral continuity of the mineralized bodies considering thin-outs and splitting of the dyke structures. Intervals of rocks included into wireframes without assay data of Li_2O, have been attributed with 0 percent Li_2O, in order to avoid an overestimate in block modelling.

AREA	Rock Types	Logged Meters	Logged Percentages
MZN	DIABASE	852	5.6
MZN	OB	290	1.9
MZN	PARAGNEISS	13,008	85.4
MZN	PEG	102	0.7
MZN	SPD PEG	976	6.4
TOTAL		**15,232**	**.**
MZSW	DIABASE	190	6.5
MZSW	OB	52	1.8
MZSW	PARAGNEISS	2,482	84.4
MZSW	PEG	12	0.4
MZSW	SPD PEG	205	7.0
TOTAL		**2,941**	**.**
HAR	OB	70	2.0
HAR	PARAGNEISS	3,130	89.1
HAR	PEG	31	0.9
HAR	SPD PEG	281	8.0
TOTAL		**3,511**	
LIN	DIABASE	9	0.2
LIN	OB	93	2.2
LIN	PARAGNEISS	3,498	81.2
LIN	PEG	97	2.2
LIN	SPD PEG	584	13.6
TOTAL		**4,305**	
CON	DIABASE	11	0.3
CON	OB	110	3.4
CON	PARAGNEISS	2,809	85.8
CON	PEG	33	1.0
CON	SPD PEG	310	9.5
TOTAL		**3,274**	

Table 27. Lengths and percentages of the relevant rock types

The intersected rock types were: DIABASE are sills and dikes cutting the pegmatites in some areas; OB is overburden like sand and gravel with clay unconsolidated sediments from the ice ages; PARAGNEISS is one of the host rocks of the pegmatites bodies; PEG are quartz and feldspar pegmatites without Spodumene mineralisation; SPD PEG are pegmatites with the typical Spodumene mineralisation (the main mining target).

AREA	Rock Types	Logged Meters	Logged Percentages	Li_2O Grade [%]
MZN	DIABASE	21	1.4	0.02
MZN	OB	1	0.1	<0.02
MZN	PARAGNEISS	560	39.4	0.1
MZN	PEG	19	1.3	0.07
MZN	SPD PEG	821	57.7	0.84
MZN	UNK	1	0.1	<0.02
TOTAL		1,423	100.0	
MZSW	PARAGNEISS	42	18.3	0.1
MZSW	PEG	2	0.8	0.06
MZSW	SPD PEG	186	80.9	0.72
TOTAL		230	100.0	
HAR	PARAGNEISS	63	18.1	0.04
HAR	PEG	11	3.1	0.06
HAR	SPD PEG	273	78.8	0.52
TOTAL		346	100.0	
LIN	PARAGNEISS	143	20.1	0.05
LIN	PEG	46	6.5	0.14
LIN	SPD PEG	515	72.6	0.57
LIN	UNK	5	0.8	0.1
TOTAL		710	100.0	
CON	PARAGNEISS	75	19.3	0.06
CON	PEG	21	5.4	0.03
CON	SPD PEG	294	75.1	0.8
CON	UNK	1	0.2	<0.02
TOTAL		**391**	**100.0**	

Table 28. Lengths and percentages of the relevant rock types in interpreted wireframes and related average Li_2O grades

SAMPLING

Sampling was done on varying sample intervals. However, 2/3 of all samples are ranging between 0.9 and 1.1 m. Thus, the composite length is set to 1 m.

Mean		1.10
Minimum		0.00
Maximum		3.81
Percentiles	10	0.65
	20	0.90
	30	1.00
	40	1.00
	50	1.00
	60	1.00
	70	1.05
	80	1.50
	90	1.53

Table 29. Statistics of sample intervals.

The structure of the pegmatite bodies shows a complex system of thinning, thickening and split-offs. In order to model laterally continuous mineralized wireframes, a significant portion of low mineralized rock or non-mineralized pegmatites was included in these wireframes. Rock, which was not assayed, was attributed with 0 % Li_2O, in order not to overestimate the block model.

AREA	Wireframe Intersection [m]	Non-assayed Interval [m]	Percentage o non-assayed interval
MZN	1,428	401	28
MZSW	231	53	23
HAR	347	75	22
LIN	711	145	20
CON	392	46	12

Table 30. Overview about non-assayed rock (host rock or low mineralized rock) included in the wireframes (were used with 0 % Li_2O)

SAMPLE PREPARATION AND ANALYSIS

Due to the lack of commercially available lithium standards in the ore grade % Li_2O range, Rock Tech created their own lithium standards. The material for the customized standard came from the MZN Spodumene pegmatite. Grab samples were sent to CDN Resources Inc., Vancouver in early December 2010 to produce the standards. Two standards were produced: one high grade standard with the original composition of the Spodumene pegmatite (STDH) and one that was diluted with pure quartz by 50% to produce a low grade standard (STDL). The standards were pulverized in a large rod mill, screened through 270 mesh and homogenized in a large rotating mixer. A total of 10 g of powdered standard was put in each package.

Then 10 samples of each standard were sent to six different labs for a round robin for a total of 60 analyses of STDH and 60 analyses of STDL. The six labs analyzed the Li content in the standards using sodium peroxide fusion digestion with ICP finish to match the method of the primary lab. Once the results of the round robin were completed in early February 2011, Barry Smee, Ph.D., P.Geol of Smee & Associates Consulting Ltd, North Vancouver compiled the round robin results and calculated the certified mean and standard deviation.

All results of assayed STH and STL fall within the recommended range of two standard deviations, which means a confidence level of around 90 %. One sample was excluded from each the STH and STL plot because of erratic high or low Li_2O concentrations, which must be explained by a labelling mix-up. In consequence, the analytical method applied is assessed as suitable to have produced reliable chemical assay data of Li_2O.

As blank material silca was used, in some cases dolomite. Li_2O concentrations of all 151 blank samples was below 0.2 % Li_2O, except one was 0.4 % Li_2O and another one was erratically high, which is explained by a labelling mix-up. In consequence, sample preparation method is assessed to be free of contamination.

Quarter core duplicates reproduced Li_2O with a deviation of around 20 %. This increased short-range deviation of Li_2O might be caused by the Spodumene minerals, which are longish orientated perpendicular to the strike direction of the pegmatite dykes. In summary, the core cutting procedure is assessed to have produced representative results. However, down-hole variography done by CCIC in 2012 confirms a high nugget effect for Li_2O. Very coarse-grain size of Spodumene crystals will create a nugget effect, as one ¼ drill core sampling

may intersect the Spodumene crystal and the other ¼ drill core sample not. Hence, for NQ core half core sampling is the preferred size of sampling. More details see Resource report 2018 (Peters S., Lowicki F. 2018)

DENSITY DETERMINATION

Theoretically, Spodumene has a density of 3.1 t/m³ in maximum and thus is heavier than the hosting quartz-feldspar matrix assumed to have a theoretical density of 2.65 t/m³. In consequence, it was assumed that the bulk density of Spodumene pegmatites is positively correlated to Li_2O concentration.

In total, for 333 samples densities were available. A cross plot of all Li_2O concentrations above 0.65 % (178 samples) vs. density has shown only a poor correlation. It is assumed that an accuracy error in density determination or less-representatively chosen samples are overlying the relative slight difference in density of Spodumene and quartz-feldspar, hampering the quality of the correlation.

However, despite high scatter, applying the resulting regression equation, an averaged 1% Li_2O grade results in a bulk density of 2.71 t/m³, which is assessed as a reasonable approach. In consequence, the regression equation was applied to the Li_2O block values to attribute a bulk tonnage.

CONFIRMATION OF HISTORICAL DATA ACQUISITION

In total, 14 drill holes were twinned within a distance not exceeding 15 m in order to confirm the historical database. Historical drill core was not available anymore and thus re-sampling not possible.

Cross sections show a good spatial re-production of the logged mineralization. In some twin holes the mineralization could be confirmed but some meters up or downwards then expected, which might be explained by poor depth control or inaccurate surveying of historic holes.

In general, results of the historic holes were confirmed considering the short range variations and nugget effects described above. Hence, results of the historic holes are assessed as suitable for resource modelling and estimation. Figure 23 to Figure 24 display cross sections of the above given confirmation holes.

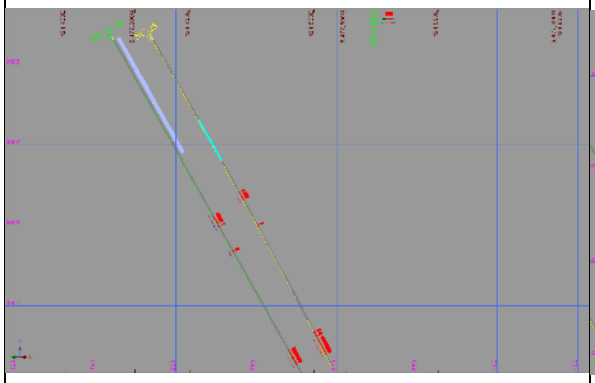


Figure 23. Twin drilling MZN: NC-11-14 twinned NC-25. Grid spacing 50 m.

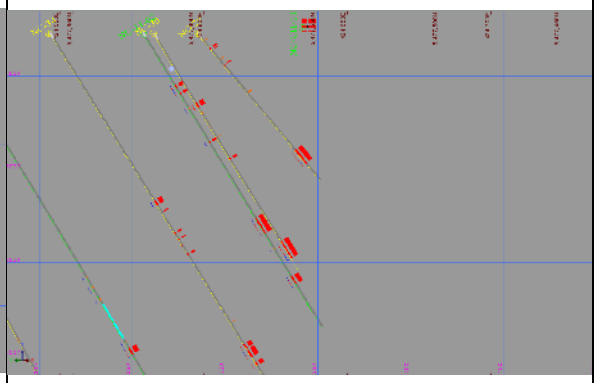


Figure 24. Twin drilling MZN: NC-11-03 twinned NC-30 Grid spacing 50 m.

Figure 23 shows an example of good reproduction of the mineralized dyke in both holes, while Figure 24 shows an example of good re-production, but discrepancy in depths (see (Peters S., Lowicki F. 2018).

VERIFICATION OF CHANNEL SAMPLES

The following table shows the mean grades of Li_2O for recent drilling and channels. Applying a Li_2O cut-off grade of 0.65 % the mean Li_2O grade of channels samples meet the results of drilling with an error below 5 %. In consequence, the assays of channels have been used for resource modelling as well.

TYPE	Number of Boreholes	Li_2O [%] all samples	Li_2O [%] >0.65 % Li_2O
Channel	138	0.75	1.30
Drillholes	62	0.52	1.24

Table 31. Mean grade of Li_2O of recent drill and channel samples

CONCESSION AREA

Rock Tech holds 283 claims (56.93 km^2) and 81 dispositions (10.42 km^2). Claims and dispositions were given to DMT as shape files and checked for spatial consistency with drill holes and trenches used for resource modelling and estimation. Also a general database was used to check all data.

DIGITAL TERRAIN MODEL

A topographic survey is available at 20 m x 20 m spacing is available and was checked for consistency with collar elevations. No major off-set was observed.

MINED OUT AREA

As mentioned above shaft sinking is reported, but production status was not reached. Hence, mining has not taken place to date. Old plans show a final depth of that shaft of 152.65 m below the surface. 3 levels were planned 48.93 and 137.00 m below surface.

DATA QUALITY SUMMARY

DMT assesses, that the quality and quantity of data available is sufficient to state a resource in compliance to NI 43-101.

TESTWORK BASIS

The statements below are based on the following document:

(1) The recovery of Spodumene from Georgia Lake Project, Project 122607-001, Oct./11/2011 by SGS Canada Inc.

One bulk sample was produced from three short drill holes on MZN (BK-11-03, 04, 05) Feb. 16-18, 2011. In addition, three blasted grab samples were collected from three sites (two from MZN and one from MZSW). These bulk samples were considered to be representative of the style and type of mineralization and the mineral deposit as a whole. All material (in total about 770 kg) were shipped to SGS Metallurgical Operations, Lakefield, Ontario in February 2011. For processing testwork the material was composited to produce a composite head sample representative of the Spodumene mineralization on the property.

The average Li_2O content was 1.49 % (0.69% Li). The main Li-mineral is Spodumene, a Lithium Al-Silicate ($LiAl(Si_2O_6)$ with a specific density of approx. 3.15 t/m³.

The composite head sample contained 19.0 wt% Spodumene, 32.4 wt% Quartz, 34.4wt%, Albite, 7.1 wt% Muscovite and 7.1 wt% Microcline (Table 34). . DMT notes that these samples yielded a head grade of 1.49 % Li_2O, which is higher than the average grade of the new resource estimate.

Because of the different specific gravity between Spodumene and gangue, gravity separation was tested. Flotation of fine fraction can increase Spodumene content in concentrate to 6.2% Li_2O and overall recovery to 81.5%. With gravity separation (in test heavy liquid separation) a concentrate with approx. 6.0% and a recover of approx. 70 % was possible.

Each sample was crushed down to ½". About 75 kg of each sample was riffled out and churched down to 6 mm. 15 kg of each sample was used for heavy liquid separation test work and bond work index testing. A small sample was taken out for chemical analysis (Table 32).

It was decided to mix all samples to a composite head sample.

	Assay %															
Sample ID	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	Sum
Spodumene Drill Cores	0.32	0.69	74.5	15.7	0.69	0.09	0.31	4.44	2.99	< 0.01	0.17	0.05	0.02	< 0.01	0.68	99.7
Spodumene Bulk #1	0.76	1.64	76.0	16.3	0.56	0.07	0.28	4.08	1.71	< 0.01	0.19	0.08	0.02	< 0.01	0.49	99.8
Spodumene Bulk #2	0.88	1.89	75.6	15.9	0.69	0.09	0.26	3.54	1.64	< 0.01	0.18	0.06	0.02	< 0.01	0.57	98.6
Spodumene Bulk #3	0.79	1.70	74.5	15.9	0.62	0.10	0.23	3.68	1.97	< 0.01	0.19	0.10	0.02	< 0.01	0.57	97.9

Table 32: Chemical analysis of samples (1)

	Assay %															
Sample ID	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	Sum
Head Comp	0.69	1.49	75.6	16.0	0.48	0.11	0.26	3.92	2.01	< 0.01	0.16	0.08	0.01	< 0.01	0.65	99.3

Table 33: Chemical analysis of composite head samples for investigation (1)

Mineral	Spod Head Composite (wt %)
Quartz	32.4
Albite	34.4
Spodumene (Monoclinic)	19.0
Microcline	7.10
Muscovite	7.20
TOTAL	100

Table 34: Mineralogical analysis of composite head samples for investigation (1)

Heavy Liquid Test No.	Products Sample ID	Heavy Liquid Density g/cm³	Product		Assay %								Distribution %						
			wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
1	Silicate Gangue (-3/8+ 0.5 mm Float 2.65 SG)	2.65	4,203	42.8	0.02	0.03	76.6	13.8	0.16	6.17	2.64	0.26	0.93	44.2	36.2	13.7	64.5	55.2	41.1
	Final HL Spodumene Concentrate	3.00	1,462	14.9	3.06	6.59	64.0	25.9	0.96	0.60	0.56	0.21	62.1	12.8	23.7	29.0	2.18	4.05	11.7
	Flotation Feed		4,164	42.4	0.64	1.38	75.1	15.4	0.67	3.21	1.96	0.30	36.9	42.9	40.2	57.3	33.3	40.7	47.2
	Feed (Calc.)		9,828	100	0.73	1.58	74.1	16.3	0.49	4.09	2.04	0.27	100	100	100	100	100	100	100
2	-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	4,332	43.6	0.02	0.04	77.4	13.3	0.18	5.79	2.54	0.26	1.10	45.4	35.8	12.8	63.9	53.9	40.8
	Final HL Spodumene Concentrate	3.00	1,883	19.0	2.88	6.21	64.9	24.5	1.10	0.79	0.65	0.24	75.5	16.6	28.7	34.1	3.81	6.02	16.4
	Flotation Feed		3,719	37.4	0.45	0.98	75.4	15.3	0.86	3.41	2.20	0.32	23.4	38.0	35.5	53.0	32.3	40.1	42.8
	Feed (Calc.)		9,934	100	0.72	1.56	74.3	16.2	0.61	3.95	2.05	0.28	100	100	100	100	100	100	100
3	-3/8+ 0.5 mm Float 2.70, Silicate Gangue, 1st Pass	2.70	4,342	44.0	0.08	0.17	76.7	14.0	0.26	5.68	2.46	0.29	4.37	45.6	36.7	23.5	61.5	55.7	43.2
	-3/8+ 0.5 mm Float 2.65, Silicate Gangue, 2nd Pass	2.70	818	8.28	0.07	0.15	82.8	10.7	0.24	4.92	0.79	0.22	0.71	9.27	5.28	4.14	10.0	3.36	6.26
	Final HL Spod Conc. 1st Pass	2.95	1,285	13.0	2.87	6.18	66.2	24.0	0.82	0.76	0.57	0.19	46.5	11.7	18.7	21.9	2.43	3.80	8.46
	Final HL Spod Conc. 2nd Pass	2.90	912	9.24	2.68	5.77	66.7	24.3	0.81	1.02	0.76	0.25	30.8	8.33	13.4	15.5	2.31	3.61	8.03
	Flotation Feed		2,518	25.5	0.56	1.20	72.9	17.1	0.67	3.79	2.55	0.39	17.7	25.1	26.0	35.0	23.8	33.5	34.1
	Feed (Calc.)		9,876	100	0.80	1.73	73.9	16.8	0.49	4.07	1.94	0.29	100	100	100	100	100	100	100
4	-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	6,175	68.2	0.02	0.05	76.4	13.5	0.24	5.67	2.55	0.26	4.55	70.2	59.0	35.5	80.7	68.4	64.0
	Final HL Spodumene Concentrate	3.00	657	7.25	2.92	6.29	64.8	25.3	1.12	0.73	0.84	0.24	60.8	6.33	11.8	17.7	1.10	2.40	6.21
	Flotation Feed		2,225	24.6	0.49	1.06	71.0	18.5	0.87	3.55	3.02	0.34	34.6	23.5	29.2	46.8	18.2	29.2	29.8
	Feed (Calc.)		9,057	100	0.35	0.75	74.2	15.6	0.46	4.79	2.54	0.28	100	100	100	100	100	100	100
5	-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.70	3,486	45.8	0.05	0.10	79.9	11.9	0.20	5.19	1.90	0.23	2.27	49.2	32.7	16.6	66.1	52.3	41.6
	Final HL Spodumene Concentrate	2.90	1,730	22.7	2.95	6.34	65.4	25.4	0.98	0.67	0.42	0.19	73.6	20.0	34.6	39.6	4.24	5.80	17.2
	Flotation Feed		2,390	31.4	0.70	1.50	73.0	17.4	0.78	3.40	2.22	0.33	24.1	30.8	32.7	43.8	29.6	41.9	41.3
	Feed (Calc.)		7,606	100	0.91	1.96	74.5	16.7	0.56	3.60	1.67	0.25	100	100	100	100	100	100	100

Table 35: Summary of heavy liquid separation and flotation test work, (1) page 18

HEAVY LIQUID SEPARATION

The bond index was determined with 12.5 kWh/t, which is in a normal range for this material. Some heavy liquid separation test was carried out with head sample and different specific gravity of heavy liquid. The material was crushed down to 3/8" (9.5 mm) and separated at 2.65 t/m³. The sink product crushed to 10 mesh (1.65 mm) and separated at 3.0, 2.9, and 2.65 t/m³.

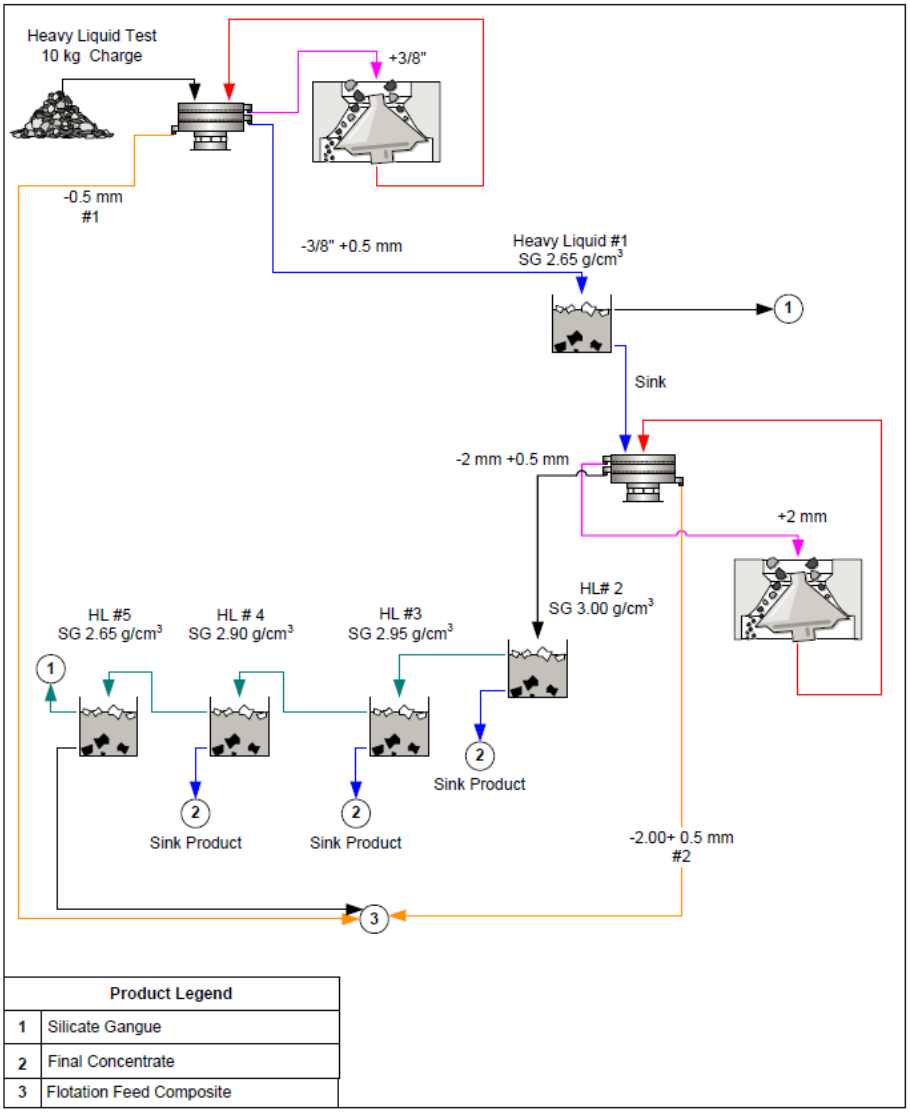


Figure 25: Scheme of heavy liquid test 1 separation test work, (1) page 3

The final product achieved a Li_2O content of 6.59% with a Li-recovery of 62.1% (according Table 4 in (1)). Magnetic separation can increase the Li-content and recovery but increase the iron content too. That is not

acceptable for use in ceramic industry. In total, five heavy liquid separation tests were carried out with almost similar results in Li-content and Li-recovery.

FLOTATION TEST WORK

The flotation test work was carried out in a 10 kg Denver D12 cell. The material for flotation was grinded to <0.3 mm and deslimed with a 50 mm Mozley cyclone. Approx. 3.5% of head sample mass was reported to cyclone overflow, which recovers 2.4 % of Li to waste.

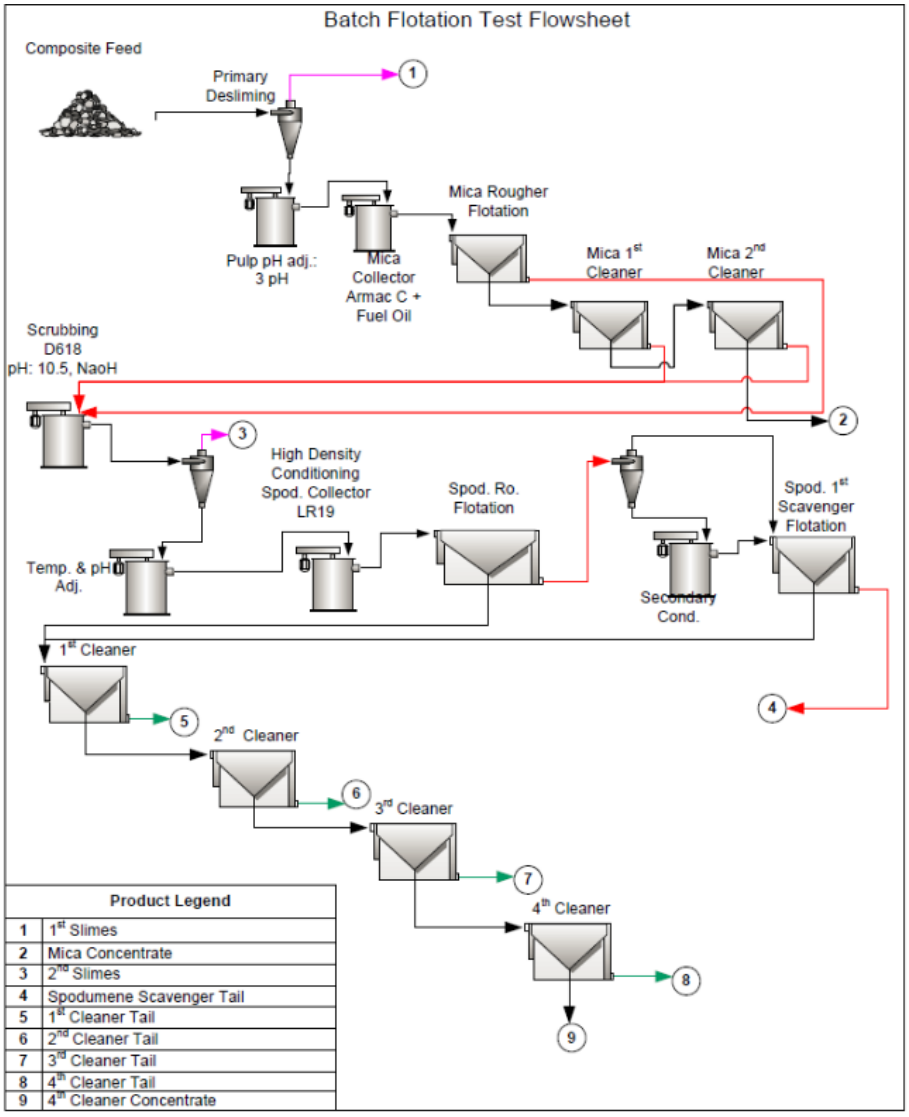


Figure 26: Flotation test work flowsheet (1), page 24

A concentrate above 6.5% Li_2O can be produced. Disadvantage is, that the iron contend with approx. 1% is fairly high. Li-recovery is above 80%.

Intensive flotation test work including mica flotation was carried out. In addition, middling of heavy liquid separation were tested with flotation. Figure 26and Figure 27 show typical multistep flotation test flow sheet

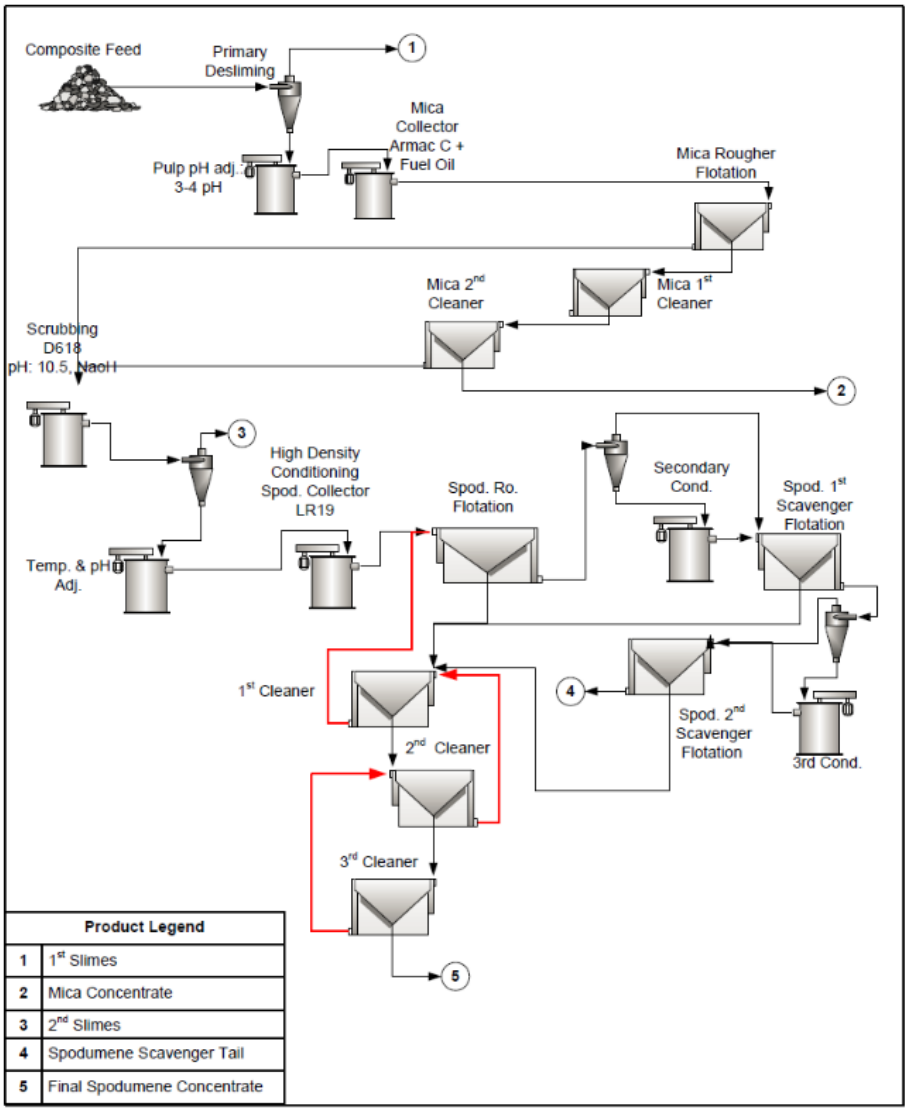


Figure 27: Flotation test work flowsheet (1), page 40

After the successful generation of an initial Li_2CO_3 product SGS, Lakefield site focused on processing concentrates through the standard lithium carbonate hydrometallurgical flow sheet. The program examined three concentrate ("conc") samples: one low Fe con, one high Fe con and one heavy media con.

The low Fe and high Fe cons were produced using floatation methods and the heavy media con was produced using heavy liquid separation. After two bicarbonate polishing tests, the resulting solids met all of the product specifications and had a Li_2CO_3 grade of 99.988% and 61 ppm Ca. This indicates that a high grade Li_2CO_3 product can be produced from Rock Tech's mineralized samples. For details of investigations and results the reader should refer to technical report of resource estimate prepared by CCIC in 2012 and the SGS Report 2011.

MINERAL RESOURCE ESTIMATES

INTRODUCTION

Independent, NI 43-101 compliant resources at the Georgie lake properties of Rock Tech were estimated using validated and verified historical drill hole data, results from the 2010/11 drill and trenching programs conducted by CCIC on behalf of Rock Tech, and results of further drilling and trenching since 2012 conducted by Arriva Management Inc. Vancouver on behalf of Rock Tech. During the 2012 to the beginning of 2018 further exploration work was done. The resource model and estimate are dated April 2018.

GEOLOGICAL MODEL

For the five main pegmatites located in the NSPA a geological 3D model was built. The general concept, which underlies the wireframe interpretation is based on tabular mineralized bodies following dyke structures with a general orientation and extent of the main pegmatites as follows:

- MZN: strikes N 55º E and dips 70º NW; strike length 1000 m
- MZSW: strikes N 45º E, dips 70º NW; strike length 300 m
- HAR: strikes N30ºE, dips 70º NW; strike length 450 m
- LIN: strikes N30ºE, dips 70º NW; strike length 450 m
- CON: strikes N 30º E, dips 70º NW; strike length 800 m

The pegmatite veins show varying concentrations of Spodumene and are showing thinning, thickening and split-offs. The pegmatites are partly exposed outcrops. The remainder is overlain by glacial deposits consisting mainly of stratified accumulations of unconsolidated sand and gravel. The depth of glacial erosion is still unknown. Further investigations will be required.

Diabase veins intersect the pegmatites of MZN vertically and horizontally.

STATISTICAL ANALYSIS

In total 38 wireframes were interpreted with a total volume of 8.74 Mm^3. It should be noted that waste rocks (pegmatites without spodumene or host rocks) were partially included in these wireframes in order to achieve continuous bodies even spodumene pegmatites are showing thin-outs and split-offs.

The following basic statistics tabulations show average Li_2O and average densities for several Li_2O cut-off grades applied to drilled material inside the wireframes. It is obvious that around 40 % of the wireframed material has Li_2O concentrations less than 0.2 % for MZN, MZSW and CON, and 60 % for HAR and LIN. This is material of weakly mineralized spodumene pegmatites, un-mineralized pegmatites or host rock, which were included in the wireframes to respect thinning, thickening and split-offs. This approach ensured continuity in the wire framing. In order to avoiding over-estimation un-sampled intervals were set to 0 % Li_2O. This has not been done in the resource estimate prepared by CCIC in 2012, which is acceptable as an inferred resource with higher uncertainties, but must be done for resource estimate at higher confidence levels than inferred.

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
MZN	0.0	1508	2.68	0.53	100.00%
MZN	0.2	866	2.70	0.90	57.43%
MZN	0.4	645	2.72	1.11	42.78%
MZN	0.6	574	2.72	1.19	38.02%
MZN	0.8	482	2.73	1.28	31.99%
MZN	1.0	380	2.73	1.38	25.16%
MZN	1.2	268	2.74	1.50	17.80%
MZN	1.4	162	2.75	1.64	10.73%
MZN	1.6	84	2.76	1.77	5.56%
MZN	1.8	24	2.77	1.96	1.62%
MZN	2.0	7	2.78	2.12	0.46%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
MZSW	0.0	231	2.69	0.60	100.00%
MZSW	0.2	144	2.71	0.94	62.42%
MZSW	0.4	129	2.71	1.01	55.81%
MZSW	0.6	109	2.72	1.11	47.36%
MZSW	0.8	82	2.72	1.25	35.29%
MZSW	1.0	65	2.73	1.33	27.92%
MZSW	1.2	42	2.74	1.45	18.31%
MZSW	1.4	26	2.74	1.56	11.26%
MZSW	1.6	8	2.76	1.77	3.53%
MZSW	1.8	4	2.76	1.91	1.56%
MZSW	2.0	1	2.77	2.05	0.43%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
HAR	0.0	364	2.68	0.43	100.00%
HAR	0.2	163	2.70	0.89	44.92%
HAR	0.4	137	2.71	1.00	37.65%
HAR	0.6	111	2.72	1.12	30.61%
HAR	0.8	86	2.72	1.25	23.69%
HAR	1.0	68	2.73	1.33	18.76%
HAR	1.2	42	2.74	1.48	11.66%
HAR	1.4	20	2.75	1.67	5.57%
HAR	1.6	8	2.77	1.97	2.18%
HAR	1.8	5	2.78	2.17	1.36%
HAR	2.0	3	2.79	2.40	0.82%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
LIN	0.0	731	2.68	0.44	100.00%
LIN	0.2	336	2.70	0.90	45.95%
LIN	0.4	277	2.71	1.04	37.84%
LIN	0.6	222	2.72	1.17	30.42%
LIN	0.8	174	2.73	1.31	23.82%
LIN	1.0	140	2.73	1.41	19.15%
LIN	1.2	95	2.74	1.56	13.03%
LIN	1.4	67	2.75	1.67	9.20%
LIN	1.6	35	2.76	1.82	4.76%
LIN	1.8	14	2.77	1.98	1.96%
LIN	2.0	6	2.78	2.11	0.80%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
CON	0.0	412	2.69	0.63	100.00%
CON	0.2	271	2.71	0.93	65.73%
CON	0.4	231	2.71	1.04	56.20%
CON	0.6	205	2.72	1.12	49.87%
CON	0.8	158	2.72	1.23	38.50%
CON	1.0	111	2.73	1.38	26.91%
CON	1.2	70	2.74	1.54	16.93%
CON	1.4	44	2.75	1.68	10.66%
CON	1.6	24	2.76	1.82	5.80%
CON	1.8	10	2.77	1.97	2.52%
CON	2.0	3	2.78	2.15	0.84%

Table 36. Average Li_2O and Densities at several Li_2O cut-off grades for wireframe intersections shown for the 5 main pegmatites

The following figures show frequency plots of Li_2O for all 3D modelled pegmatites. It is obvious that there are two sample populations included in the wireframes, mineralization of Spodumene pegmatites and weakly and non-mineralized rocks. However, both sample population are following broadly a normal distribution without a significant amount of outlier. Hence, grade capping has not been applied.

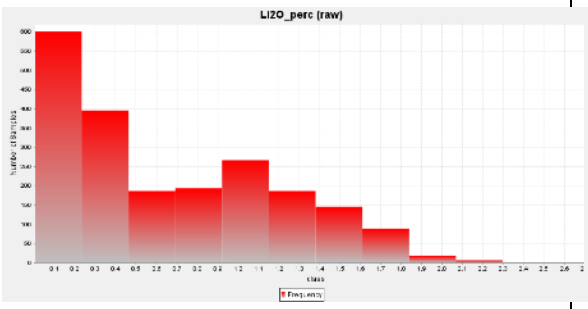


Figure 28. Frequency plots of Li_2O for MZN.

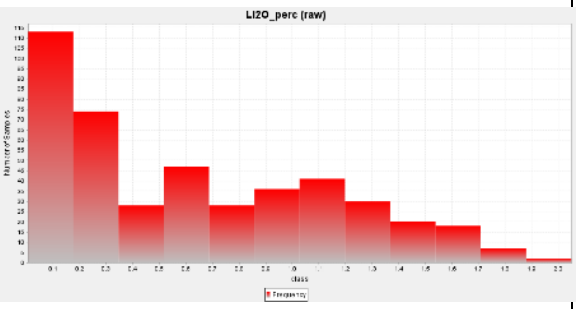


Figure 29. Frequency plots of Li_2O for MZSW

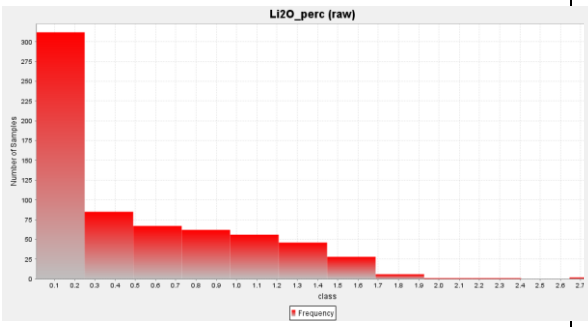


Figure 30. Frequency plots of Li_2O for HAR

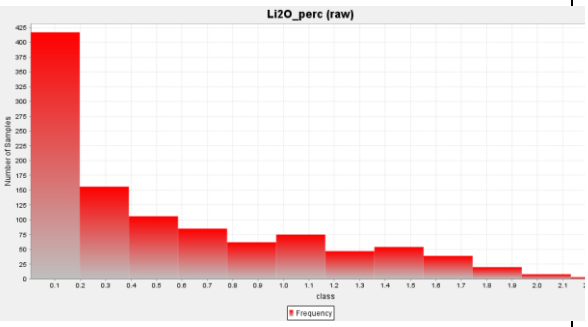


Figure 31. Frequency plots of Li_2O for LIN

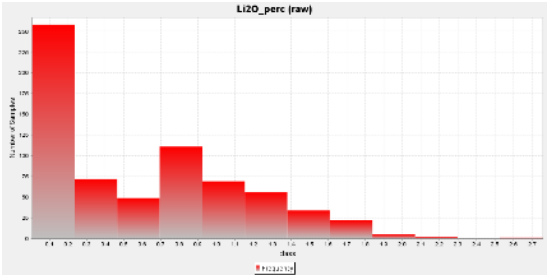


Figure 32. Frequency plots of Li_2O for CON

INTERPRETATION OF MINERALIZED ZONES (DOMAINS)

Wireframes were modelled following the concept of a dyke structures of the pegmatite bodies comprising spodumene. The interpretation was based on sections perpendicular to strike on a hole to hole interpretation following the general dip of the pegmatites from hole to hole. The wireframes were extended to undrilled areas with around 50 m in maximum based on the general drill spacing.

WIREFRAME MODEL

In total, 38 wireframes were interpreted with a total volume of 8.74 Mm³. The following Figures show 3D views of these wireframes Figure 35.

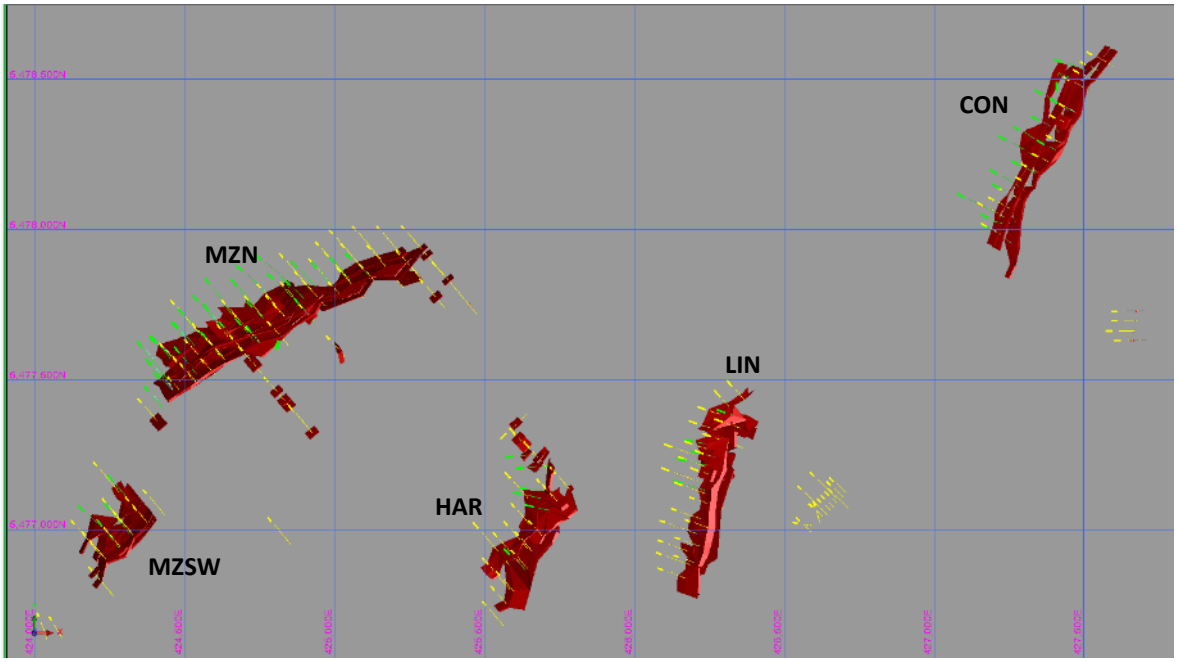


Figure 33. Plan view onto wireframes of Spodumene pegmatites (red) with in the NSPA. Grid spacing 500 m.

Table 29 shows the volume of the wireframes. These wireframes build the basis of the 3D model of the pegmatites.

AREA	Volume of Wireframes [Mm³]
MZN	4.00
MZSW	0.75
HAR	1.17
LIN	1.74
CON	1.08
TOTAL	**8.74**

Table 37: Volume of the wireframes [Mm³] of MZN, MZSW, HAR, LIN, CON

GRADE CAPPING / COMPOSITING / BLOCK MODEL DEFINITION

No grade capping is applied to composites, because no significant amount of outlier have been observed in the frequency plots.

Sampling was done on varying sample intervals. However, 2/3 of all samples is ranging in-between 0.9 and 1.1 m. Thus, the composite length is set to 1 m.

Parent block size has been set to X=2m, Y=2m, Z=5m. Considering the general drill spacing is around 50 m x 50 m this is only 4 %. However, the smaller block sizes were defined in order to consider the often thin and very steep and varying dipping pegmatite dykes. As long as the mining method is not chosen and related selectivity and minimum mining unit is unknown, this smaller blocks give the possibility to re-block to larger units. The older resource estimate done by CCIC in 2012 was based on 5 x 5 x 5 m blocks.

GEOSTATISTICS / INTERPOLATION METHOD / BLOCK MODEL

Geostatistical analysis has been done in order to analyse continuity of the mineralisation and to define the kriging parameters with the objective to minimize the interpolation error. The only high quality variogram was possible for MZN, because of the high drill hole density. The results were applied to other main pegmatite bodies assuming a very similar distribution of Li_2O. Based on the results the range is set to 150 m, the nugget to 0.04 and sill to 0.06.:

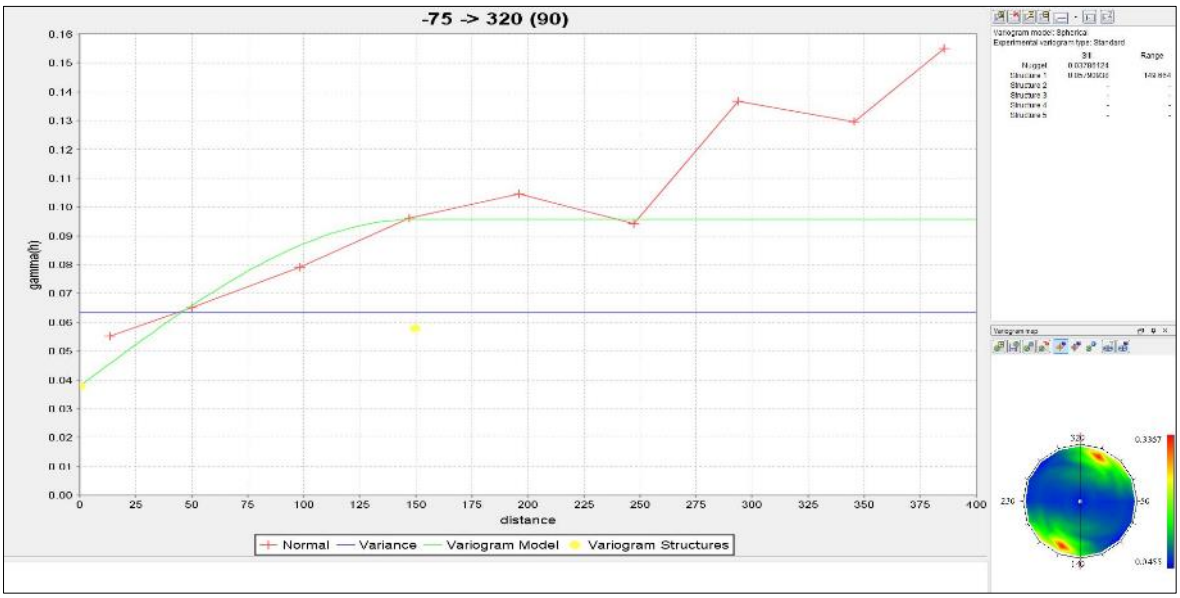


Figure 34. Experimental variogram (red) and variogram model (green) for MZN.

The ordinary kriging was done in 10 passes with increasing search radii from 15 m to 150 m in major and semi-major direction and from 2 m to 20 m in minor direction successively. This was done to ensure that blocks very near a sample are not effected by far away samples and thus minimize possible dilution. A minimum of 1 sample and a maximum of 15 samples were used in the estimation of individual blocks.

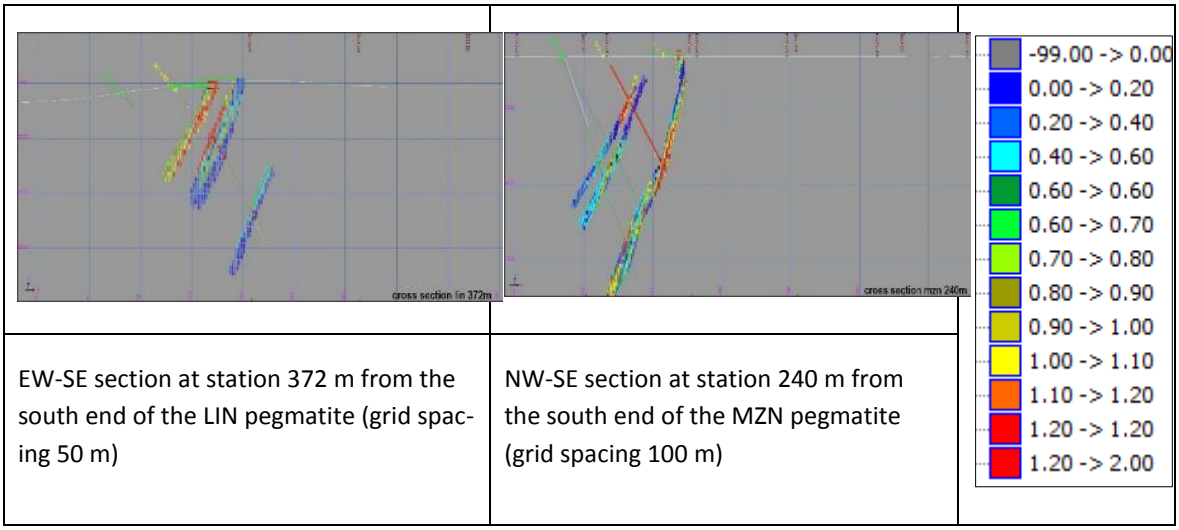
Area	Bearing [degree]	Plunge [degree]
MZN	320	-75
MZSW	320	-75
HAR	320	-75
LIN	290	-65
CON	300	-70

Table 38. Orientation of search ellipsoid for the 5 areas following the dip direction and dip of the main pegmatite bodies

The wireframed volume was assigned to the block model using the partial percentage attribute in Surpac. Each block was assigned a volume correction factor which is the proportion of block volume within the domained wireframe and below topography. Thus, a volume discrepancy based on stair step effects relative to block size could be avoided.

The **bulk tonnage** was calculated based on block volume (2m x 2m x 5m) * (bulk density) * (the proportion of the block within the solid and under the surface topography)

The **Li_2O average grade** related to this tonnage is the arithmetic average of Li_2O weighted by the bulk tonnage of each block.



EW-SE section at station 372 m from the south end of the LIN pegmatite (grid spacing 50 m)

NW-SE section at station 240 m from the south end of the MZN pegmatite (grid spacing 100 m)

Figure 35 sections of the block model colour shows the Li^2O percentage of the blocks

The sections of the block model of two pegmatites show the distribution of the Li^2O contend within the model.

RESOURCE CLASSIFICATION

The definitions for resource categories used in this report are consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves 2014.

Under the CIM classification system, a Mineral Resource is defined as:

…" a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

"The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

Resources are classified into Measured, Indicated and Inferred categories based upon geological knowledge and confidence (Figure 36). Mineral resources are not mineral reserves and do not have demonstrated economic viability.

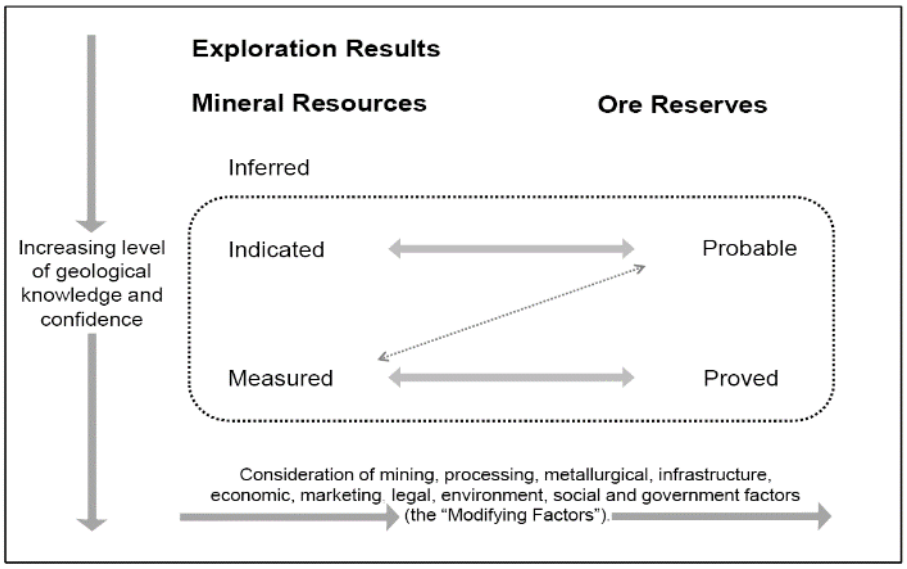


Figure 36. Relationship between Exploration Results, Mineral Resources & Ore Reserves

Resource classification within mineralisation envelopes is generally based on drillhole spacing, grade continuity, and overall geological continuity. The distance to the nearest composite and the number of drillholes are also considered in the classification. In classifying the resource estimate, the following key factors have been considered:

- Confidence in data quantity and specifically sample spacing of Li_2O data;
- Confidence in the geological interpretation and continuity (geological complexity); and
- Confidence in mineralisation / grade continuity (complexity of spatial grade distribution).
- Considering the above, the following criteria have been applied for classification into the various mineral resource categories for this estimate.

MEASURED RESOURCES

All blocks within the wireframed constraints and a distance to the nearest Li_2O sample being less than 20 m.

INDICATED RESOURCES

All blocks within the wireframed constraints and a distance to the nearest Li_2O sample being equal or above 20 m and less than 70 m.

All blocks with a distance to the nearest Li_2O sample, which is above 10 m below topography.

INFERRED RESOURCES

All blocks within the wireframed pegmatites which are not defined as Measured or Indicated but are included in the interpreted wireframes,

Outside the block model with in NSPA, derived by applying an average true thickness of spodumene pegmatite and extrapolating the pegmatite bodies in maximum 50 m below the deepest drillhole in the 3D model.

And all drilled material of isolated drill holes which were not used for modelling within the claims and dispositions.

PRELIMINARY CUT-OFF GRADE ASSUMPTIONS

Following international requirements, a Li_2O cut-off grade was applied to constrain the estimated mineral resources and to demonstrate reasonable prospects for eventual economic extraction.

The reporting cut-off grade of 0.65 Li_2O% was chosen based on the benchmarking of similar Lithium projects, but is not based on a financial model specific for this project, but based on comparable projects.

MODEL VALIDATION

In order to check that the interpolation of Li_2O has worked appropriately, the interpolated block model has been validated against the corresponding domained primary data from drilling using the following techniques:

- Visual inspection of block grades in plan and section and comparison with drill hole grades (Appendix 1); and
- Comparison of mean grades of block model data with primary data within mineralised domain at several Li_2O cut-off grades (Table 39 and Table 39. Model validation for MZN and MZSW

Table 39 compares primary data from drilling with estimated block values. It is obvious that drilled Li_2O concentrations and frequencies are representatively estimated in the block model. Slight discrepancy are caused by the drill pattern, which is producing 100 % regular intersection and by smoothing, which is typical in the interpolation process. In this case smoothing might be somewhat pronounced because waste rock was included within the domained wireframes. However, near the 0.65 % Li_2O cut-off the averaged block values are generally slightly lower than averaged values from drilling. In consequence, an overestimation caused by interpretation and interpolation can be excluded.

		Drilling Data				Block Data			
AREA	Cut-off Li2O [%]	Drilled interval [m]	Drilled interval [%]	Density [t/m³]	Li2O [%]	Tonnage [Mt]	Tonnage [%]	Density [t/m³]	Li2O [%]
MZN	0.0	1508	100.0%	2.68	0.53	10.43	100.0%	2.68	0.52
MZN	0.2	866	57.4%	2.70	0.90	7.47	71.6%	2.69	0.70
MZN	0.4	645	42.8%	2.72	1.11	5.65	54.1%	2.70	0.83
MZN	0.6	574	38.0%	2.72	1.19	3.94	37.8%	2.71	0.97
MZN	0.8	482	32.0%	2.73	1.28	2.68	25.7%	2.72	1.10
MZN	1.0	380	25.2%	2.73	1.38	1.62	15.5%	2.72	1.24
MZN	1.2	268	17.8%	2.74	1.50	0.84	8.0%	2.73	1.37
MZN	1.4	162	10.7%	2.75	1.64	0.29	2.8%	2.74	1.54
MZN	1.6	84	5.6%	2.76	1.77	0.07	0.7%	2.75	1.72
MZN	1.8	24	1.6%	2.77	1.96	0.01	0.1%	2.76	1.88
MZN	2.0	7	0.5%	2.78	2.12	0.00	0.0%	2.78	2.13
MZSW	0.0	231	100.0%	2.69	0.60	2.00	100.0%	2.68	0.53
MZSW	0.2	144	62.4%	2.71	0.94	1.39	69.4%	2.69	0.74
MZSW	0.4	129	55.8%	2.71	1.01	1.10	55.0%	2.70	0.85
MZSW	0.6	109	47.4%	2.72	1.11	0.80	40.3%	2.71	0.97
MZSW	0.8	82	35.3%	2.72	1.25	0.53	26.5%	2.72	1.12
MZSW	1.0	65	27.9%	2.73	1.33	0.32	16.1%	2.73	1.27
MZSW	1.2	42	18.3%	2.74	1.45	0.18	9.1%	2.73	1.41
MZSW	1.4	26	11.3%	2.74	1.56	0.09	4.7%	2.74	1.51
MZSW	1.6	8	3.5%	2.76	1.77	0.00	0.2%	2.75	1.66
MZSW	1.8	4	1.6%	2.76	1.91	0.00	0.0%	2.74	1.84
MZSW	2.0	1	0.4%	2.77	2.05				

Table 39. Model validation for MZN and MZSW

		Drilling Data				Block Data			
AREA	Cut-off Li2O [%]	Drilled interval [m]	Drilled interval [%]	Density [t/m³]	Li2O [%]	Tonnage [Mt]	Tonnage [%]	Density [t/m³]	Li2O [%]
HAR	0.0	364	100.0%	2.68	0.43	3.07	100.0%	2.67	0.32
HAR	0.2	163	44.9%	2.70	0.89	1.28	41.7%	2.69	0.66
HAR	0.4	137	37.6%	2.71	1.00	0.93	30.4%	2.70	0.80
HAR	0.6	111	30.6%	2.72	1.12	0.65	21.1%	2.71	0.94
HAR	0.8	86	23.7%	2.72	1.25	0.43	14.1%	2.71	1.05
HAR	1.0	68	18.8%	2.73	1.33	0.25	8.1%	2.72	1.17
HAR	1.2	42	11.7%	2.74	1.48	0.09	2.9%	2.73	1.28
HAR	1.4	20	5.6%	2.75	1.67	0.01	0.2%	2.74	1.48
HAR	1.6	8	2.2%	2.77	1.97	0.00	0.0%	2.76	1.75
HAR	1.8	5	1.4%	2.78	2.17	0.00	0.0%	2.77	1.84
HAR	2.0	3	0.8%	2.79	2.40				
LIN	0.0	731	100.0%	2.68	0.44	4.62	100.0%	2.67	0.33
LIN	0.2	336	46.0%	2.70	0.90	2.06	44.5%	2.69	0.69
LIN	0.4	277	37.8%	2.71	1.04	1.46	31.7%	2.70	0.85
LIN	0.6	222	30.4%	2.72	1.17	1.02	22.1%	2.71	0.99
LIN	0.8	174	23.8%	2.73	1.31	0.69	14.8%	2.72	1.14
LIN	1.0	140	19.1%	2.73	1.41	0.45	9.7%	2.73	1.27
LIN	1.2	95	13.0%	2.74	1.56	0.25	5.5%	2.73	1.41
LIN	1.4	67	9.2%	2.75	1.67	0.12	2.6%	2.74	1.55
LIN	1.6	35	4.8%	2.76	1.82	0.03	0.7%	2.75	1.70
LIN	1.8	14	2.0%	2.77	1.98	0.00	0.1%	2.76	1.86
LIN	2.0	6	0.8%	2.78	2.11	0.00	0.0%	2.82	2.05
CON	0.0	412	100.0%	2.69	0.63	2.89	100.0%	2.68	0.53
CON	0.2	271	65.7%	2.71	0.93	1.99	68.9%	2.69	0.74
CON	0.4	231	56.2%	2.71	1.04	1.58	54.5%	2.70	0.86
CON	0.6	205	49.9%	2.72	1.12	1.22	42.2%	2.71	0.97
CON	0.8	158	38.5%	2.72	1.23	0.84	29.0%	2.71	1.08
CON	1.0	111	26.9%	2.73	1.38	0.48	16.4%	2.72	1.23
CON	1.2	70	16.9%	2.74	1.54	0.20	7.0%	2.73	1.40
CON	1.4	44	10.7%	2.75	1.68	0.08	2.8%	2.74	1.56
CON	1.6	24	5.8%	2.76	1.82	0.02	0.8%	2.75	1.69
CON	1.8	10	2.5%	2.77	1.97	0.00	0.1%	2.76	1.91
CON	2.0	3	0.8%	2.78	2.15	0.00	0.0%	2.78	2.06

Table 40. Model validation for HAR, LIN and CON

The resource data are closely aligned with the historic resource regarding the average grades within a 10 % error range. Slightly lower Li_2O concentrations of the DMT resource estimate might be caused by attributing waste rocks or un-sampled material with 0 % Li_2O, which was not done by CCIC in 2012. However, a proper model validation was not available by CCIC in 2012 (see Table 14).

The comparison illustrates that no obvious bias has been introduced during the block modelling process. On the basis of its review and validation procedures, DMT is of the opinion that the block model is valid and acceptable for estimating Mineral Resources.

DMT prepared an independent, NI 43-101 compliant resource estimate for the Georgia Lake Lithium Property within all claims and dispositions which belongs to Rock Tech today. The total resource estimate described in this report is based on different areas within the claims and dispositions of Rock Tech. For this report DMT divided the claim and disposition area from Rock Tech into 2 Areas (north and south). The Northern Spodumene Pegmatite Area "NSPA" and the Southern Spodumene Pegmatite Area "SSPA" see Figure 8. All mentioned pegmatites in this report are situated with in the claims and dispositions of Rock Tech.

This resource update at the Georgie lake properties of Rock Tech were estimated using validated and verified historical drill hole data, results from the 2010/11 drill and trenching programs conducted by CCIC on behalf of Rock Tech, and results of further drilling and trenching since 2012 until end of 2017 conducted by Arriva Management Inc. Vancouver on behalf of Rock Tech. The resource model and estimate of this chapter are dated 18.04.2018.

DMT constructed 3D models using historical drill hole data as well as results acquired since November 2009 up to date for MZN, MZSW, Harricana, Line 60 and Conway spodumene pegmatites. 3D wireframes (solids) representing the mineralized areas within the spodumene pegmatites were constructed and used to constrain the tonnage and grade estimation. GEMCOM's Geovia software V.7.2 was used to generate the 3D model and perform the grade estimation. Grades for Li_2O were estimated using the anisotropic ordinary kriging method. A specific gravity ("SG") of averaging to 2.71 [t/m³] was applied using a regression equation based on densities measured by Rock Tech.

Five of the pegmatites, for which 3D models have been constructed, are located in the NSPA. From these models measured and indicated resources have been defined. Additional inferred resources have been estimated in the NSPA at depth, where drill core intersection density is too low.

Additional inferred resources within the properties of Rock Tech Inc. have been estimated for the pegmatites in the SSPA. In these exploration claims and dispositions spodumene pegmatites occur at the surface and also drilling and channel sampling has taken place in these areas. Ongoing exploration activities will help to update these resources into higher confidence levels in the future.

The main resources are located in the northern part, where up to date most exploration work has been conducted. In this area for five pegmatites 3D models were created. These models were necessary to define for these pegmatites the measured and indicated resources. The mineral resource estimates of the 3D model for the MZN, MZSW, HAR, LIN, CON spodumene pegmatites presented below.

Mineral resource estimates for MZN, MZSW, HAR, LIN and CON are presented below

Area	Type of Resources	Tonnage [Mt]	Li_2O [%]	Cut off Li_2O [%]
NSPA	Measured	1.89	1.04	0.65
NSPA	Indicated	4.68	1.00	0.65
TOTAL	**Measured and Indicated**	**6.58**	**1.01**	**0.65**

Table 41 Measured + Indicated Resource in the NPGA from all 5 pegmatites which were modelled in 3D

Following NI 43-101 requirements and considering potential economic viability a 0.65 % Li_2O cut-off grade has been assumed based on comparable projects, which yields a total Measured and Indicated Resource of 6.58 Mt at a grade of 1.01% Li_2O (see data of green line in Table 42 to Table 44)

It should be noted that mineral resources are not mineral reserves. Investigations on modifying factors are recommended to convert measured and indicated mineral resources to mineral reserves of certain degree of economic/technical feasibility.

Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
0.40	10.20	2.70	0.84
0.50	8.75	2.70	0.90
0.60	7.27	2.71	0.97
0.65	6.58	2.71	1.01
0.70	6.03	2.71	1.04
0.80	4.93	2.72	1.11
0.90	3.96	2.72	1.17

Table 42. Measured + Indicated Resource (green line) and grade sensitivities comprising all five areas MZN, MZSW, HAR, LIN and CON

The next tables show the distribution of the individual pegmatites in the main investigation area in the northern part of the whole area (NSPA).

Area	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
MZN	**0.40**	5.37	2.70	0.83
MZN	**0.50**	4.52	2.70	0.90
MZN	**0.60**	3.73	2.71	0.98
MZN	**0.65**	3.37	2.71	1.01
MZN	**0.70**	3.09	2.71	1.04
MZN	**0.70**	3.09	2.71	1.04
MZN	**0.80**	2.54	2.72	1.11

Area	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
MZSW	0.40	1.03	2.70	0.85
MZSW	0.50	0.94	2.70	0.89
MZSW	0.60	0.76	2.71	0.97
MZSW	0.65	0.66	2.71	1.02
MZSW	0.70	0.61	2.71	1.05
MZSW	0.70	0.61	2.71	1.05
MZSW	0.80	0.50	2.72	1.11

Area	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
HAR	0.40	0.82	2.70	0.81
HAR	0.50	0.71	2.70	0.87
HAR	0.60	0.59	2.71	0.94
HAR	0.65	0.54	2.71	0.97
HAR	0.70	0.49	2.71	1.00
HAR	0.70	0.49	2.71	1.00
HAR	0.80	0.39	2.71	1.06

Area	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
LIN	0.40	1.46	2.70	0.85
LIN	0.50	1.26	2.70	0.91
LIN	0.60	1.02	2.71	0.99
LIN	0.65	0.91	2.71	1.04
LIN	0.70	0.83	2.71	1.07
LIN	0.70	0.83	2.71	1.07
LIN	0.80	0.69	2.72	1.14

Area	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
CON	0.40	1.52	2.70	0.86
CON	0.50	1.32	2.71	0.92
CON	0.60	1.17	2.71	0.97
CON	0.65	1.09	2.71	0.99
CON	0.70	1.01	2.71	1.02
CON	0.70	1.01	2.71	1.02
CON	0.80	0.81	2.71	1.09

Table 43. Measured + Indicated Resource (green line) and grade sensitivities separated by MZN, MZSW, HAR, LIN and CON

Resource Class	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
measured	**0.40**	2.68	2.70	0.89
measured	**0.50**	2.36	2.71	0.95
measured	**0.60**	2.05	2.71	1.01
measured	**0.65**	1.89	2.71	1.04
measured	**0.70**	1.75	2.71	1.07
measured	**0.80**	1.48	2.72	1.13
measured	**0.90**	1.22	2.72	1.19

Resource Class	Cut-off Li2O [%]	Tonnage [Mt]	Density [t/m³]	Li2O [%]
indicated	0.40	7.52	2.70	0.82
indicated	0.50	6.39	2.70	0.88
indicated	0.60	5.22	2.71	0.96
indicated	0.65	4.68	2.71	1.00
indicated	0.70	4.28	2.71	1.03
indicated	0.80	3.45	2.72	1.09
indicated	0.90	2.73	2.72	1.16

Table 44. Measured and Indicated Resource (green line) and grade sensitivities comprising all five areas MZN, MZSW, HAR, LIN and CON

Area	Extrapolated surface area [m²]	Av. thickness pegmatite 0.65 % Li_2O cut-off grade [m]	Volume [Mm³]	Tonnage [Mt]
MZN	100 000	6.86	0.69	1.85
HAR	40 000	4.69	0.19	0.51
LIN	50 000	5.91	0.3	0.8
CON	50 000	5.16	0.26	0.7
TOTAL			**1.43**	**3.85**

Table 45. Additional inferred resource from the extrapolating of the 3 D modelled pegmatite bodies 50 m below the deepest drilled intersection.

In addition to these measured and indicated resources inferred resources were estimated from three different areas within the claims of Rock Tech. Within the block model **0.33 Mt** inferred resources are estimated. Outside the block model with in NSPA, there are additional inferred resources of **3.85 Mt** derived by applying an average true thickness of spodumene pegmatite and extrapolating the pegmatite bodies in maximum 50 m below the 3D model see Table 45. The Lithium content was estimated as the same as in the pegmatites up the surface was detected.

In the western part of the main pegmatite area between the MZSW and the MZN pegmatite three boreholes drilled at different depths into a gabbroidal sill. Doe to the presence of the sill the extrapolation to further depths of the pegmatite MZSW is not possible. The surface of this basic intrusion dips with 10 degrees to the west. The deepest drillholles located at the pegmatites towards the west do not intersect the sill. During the extrapolation of the pegmatites to the depth we take that issue into account.

Further additional inferred resources have been derived from all drilling and trenching on current claims and dispositions in the SSPA. All drilled material above 0.65 % Li_2O has been added, based on an influence area of 50 x 50 m and the results of chapter 7.3. The density is assumed with 2.7 t/m^3 similar to all the other results. This additional inferred resource from the SSPA is estimated with **2.54 Mt** (Table 46)

Name	Area	Drilled interval [m]	Tonnage [Mt]
McVittie	subarea 1 SSPA	94	0.67
Jean Lake (West)	subarea 2 SSPA	188	1.28
Newkirk	subarea 3 SSPA	20	0.13
Aumacho	subarea 4 SSPA	60	0.46
TOTAL		**362**	**2.54**

Table 46. Additional inferred resource from drilling and trenching on current claims and dispositions in SSPA; @ 0.65 % Li_2O cut-off

AREA	Type of Resources	Tonnage [Mt]	Li_2O [%]	Approach of estimation
NSPA	Inferred	0.33	1.01	From the 3D Block models pegmatites in the NSPA
NSPA	Inferred	3.85	1.01	From extrapolating below the existing 3D modelled pegmatites
SSPA	Inferred	2.54	1.41	From other pegmatites in the SSPA
TOTAL	**Inferred**	**6.72**	**1.16**	

Table 47 List of the total inferred resource within claims and dispositions of Rock Teck from different areas, @ cut off 0.65 % Li_2O

The total inferred recourses in the claims and dispositions of Rock Tech are **6.72 Mt.** For all of these estimated inferred resources the cut off of 0.65 % Li_2O was used (Table 47).

In all claims and dispositions of Rock Tech situated in the Georgia Lake area **6.58 Mt** measured and indicated resources are estimated. Additionally **6.72 Mt** of inferred resources were estimated within this area.

MINERAL RESERVE ESTIMATES

A mineral reserve is the economically mineable part of a Measured or Indicated Mineral Resource. A mineral reserve has not been estimated for the project as part of this PEA.

MINING METHODS

INTRODUCTION

In general, the first step in selecting a suitable mining method is the choice between surface and underground mining methods. For shallow, near-surface deposits, open pit mining is usually the best solution. As mining progresses deeper, stripping ratio (SR) and transportation costs increases considerably, which, in addition to geotechnical constraints, economically limits the depth of open pit mines. Deeper deposits are therefore exploited by underground methods.

Near-surface deposits with considerable vertical extend are often mined by a combination of open pit and underground methods. The optimum strategy is to harvest the uppermost portion of the deposit by open pit mining, while the remaining portion is mined by underground methods. In these cases there is usually an area containing material, which could be mined by either method, also referred to as transition zone.

For this study, three mining scenarios have been analysed for the pegmatite areas Main Zone/North, Main Zone/Southwest, Harricana, Line 60, and Conway:

- Open pit (OP) option,
- Underground (UG) option, and
- Hybrid option (OP-UG combination).

For developing a suitable mine design with in the project different geotechnical aspects has to be investigated. The following main points has to be determined based on available geotechnical parameters:

- Maximum slope angle for the open pits
- Maximum stope height and width for underground mining
- Minimum thickness of the crown pillar between the open pit and the underground workings

GEOTECHNICAL DESIGN

GEOTECHNICAL TEST WORK

At the beginning of the project no geotechnical parameters were available. The main parameters, which are required for this preliminary design, are on one-hand rock mechanical parameters like uniaxial compressive strength (UCS), on the other hand structural geological parameters like joint density, joint direction or joint conditions.

With these parameters, an assessment of the rock mass regarding the quality (rock mass classification) can be done and the design parameters for the mine from the view of geotechnics can be determined.

DMT staff were on side to explore the structural geological parameters for selected drill core. Additionally samples from the significant rock types were taken and USC tests were performed in the DMT labs.

TEST MATERIAL

Test material were 12 drilling cores with an overall length between 0.18 m and 0.30 m. Due to the wedge-shaped ends the true length for preparing specimen was often much shorter.

Another two hand-pieces (grab samples) were provided as well. Due to the irregular shape and some cracks, the extraction of drilling-cores was limited.

The test material has been documented and can be found in Annex A3, where the complete report is attached to the PEA.

PREPARATION OF SPECIMEN AND TESTING

Each drilling core was cut by wet-diamond-cutting after determining an individual cutting-scheme. This determination was done considering cracks and the aim of gaining long specimen. It was aimed to keep the length to diameter ratio as close to 2 as possible. Most of the drilling-cores had a diameter of 47.6 mm; two samples had a slightly larger diameter of 50.7 mm. Grab pieces were drilled in the laboratory with 51.7 core-diameter.

After cutting, the specimen were ground by wet-diamond-grinding in order to achieve smooth and parallel end-planes for vertical load-impact in the test frame. Before performing the UCS-test, each specimen was dried, weighed and measured.

As testing machine, a class 1 (highest possible accuracy) calibrated test frame of the type ToniTechnik 600 kN was used. Due to its two-column compact design, the test-frame represents a stiff construction.

Figure 37: Testing machine ToniTechnik 600 kN

DGGT (German Association of Geotechnique) provides with Recommendation 1 a suitable test procedure for testing UCS of soil and hard rock. Each test was run displacement- controlled at a deformation rate of 0.001 mm/s with a duration of more than 15 min (minimum recommended duration of testing.

TEST RESULTS

The results of the lab tests are documented in a separate report. A conclusion of the test results are listed in Table 48.

	Average UCS	Average standard deviation UCS	min UCS	max UCS	average density
	MPa	MPa	MPa	MPa	g/cm³
Paragneiss	133,7	42,5	101,7	244,8	2,7
Diabase	117,6	42,4	71,6	201,2	3,0
Pegmatite	101,5	32,5	65,1	150,3	2,6
ore body	84,7	5,6	80,7	88,6	2,7

Table 48: Conclusion of the test results from the UCS tests for the different rock types

It can be said that the average strength is high for all rock types. Some low values are influenced by occurring joints in the samples. It must be noted the e.g. for the paragneiss one UCS value (244.8 MPa) is considerably higher that the other 9 values. In the geotechnical analysis a conservative approach were selected. Therefore this high value was not considered in the later analysis.

GEOTECHNICAL LOGGING

Based on the structural geological logging a rock mass classification with the Q system from BARTON was done. This considers the join frequency, the joint roughness, the joint alteration, the RQD, the water occurrence and the stress situation. These data were logged or estimated and the Q values are calculated for each rock mass. The main rock types which are especially important for underground mining are the paragneiss and the ore body. Generally the ore body will be mainly in the roof of possible stopes. The sidewalls will be created with paragneiss. The analysis results in Q values of approx. 15 for the paragneiss and approx. 23 for the ore body. Based on the classification both rock types can be described as “good rock”.

GEOTECHNICAL INTERPRETATION

INTRODUCTION

For the determination of a slope angle but also for the design of the underground stopes a general dip of the main joint set were required. It was not possible to determine this values clearly because the drill cores were not orientated and geophysical measurements (e.g. optical or acoustical log) were not performed in the drill holes. But based on the drill cores 2 probable joint dips could be estimated. One probable joint set could be parallel to the ore body with a dip of approx. 70 degrees. Another joint set could be perpendicular to this with a dip of 20 degrees. Both possibilities were considered in the design.

DEFINITION OF THE SLOPE ANGLE

The slope angle of the open pits is highly influenced by the joint occurrence and their dipping as well as their filling and roughness. Experiences from the site (e.g. highway slopes) have shown that high slope angles are possible. But it must be noted that the open pit could have a large depth. In this case berms are between single slopes. The overall slope angle depends on the slope angle for the single slopes as well as the berm width. An idealized kinematic analysis has shown the maximum slope angle of 70 degrees is allowed due to the dip of the possible steep joint set with 70 degrees. For a joint set dip of 20 degrees the limitation is the friction angle on the joint. It is assumed based on logging (roughness of joins, undulation joints) that the friction angle is higher that 20 degrees. Therefore there is no general limitation in this case. It must be noted that these value must be confirmed in the next investigation steps. Due to the limitation of maximum 70 degrees for the single slope and the berms between the single slopes an overall slope angle for the open pit of 60 degrees is proposed.

EVALUATION OF THE OPEN ROOM

The stope dimensions were determined with the Stability graph from MATTEWS and POTVIN. This is common practice for the estimation of stope dimensions at early stage of the project. On one hand a stability factor for the significant rock mass which creating the stope walls (mainly side walls and roof) was determined based on the rock mass classification, the ratio between the occurring stress and the UCS for the rock and the dip of the main joint set. In this case both probable joint sets were considered in the analysis. By means of a diagram based on a multitude of case studies the limits for the stope walls were determined. An example for a result is presented in Figure 38.

In this case the side wall of a stope in the pegmatite is presented. The green areas are walls with stable conditions, the yellow areas are dimensions where significant roof fall cannot excluded and the red areas show dimension where caving occurs. For the actual project the green areas are mandatory. It can be seen in this case the also up to a stope length of 900 m (maximum length of the largest ore body) an unsupported stope height in the paragneiss of 20 m seems to be possible. A risk analysis with increased stress and reduced rock mas strength came to the result that in areas deeper than 300 m the unsupported stope height has to be reduced to 18 m for an open stope length of 900 m. But up to an open stope length of 250 m the 20 m are in the green area. An additional aspect in the deep areas is a possible stress relief due to the overlying open pit. Based on this a maximum stope height of 20 m is proposed.

Comparable analysis were performed for the stope width. It must be noted that the maximum width of the ore body is approx. 10 m. The analysis has shown that this stope width is possible in the occurring rock mass (ore body).

Sidewall		average					unsupported											
	Length in m																	
Heigth in m	50	100	150	200	250	300	350	400	450	500	550	600	650	700	750	800	850	900
15	5,77	6,52	6,82	6,98	7,08	7,14	7,19	7,23	7,26	7,28	7,30	7,32	7,33	7,34	7,35	7,36	7,37	7,38
16	6,06	6,90	7,23	7,41	7,52	7,59	7,65	7,69	7,73	7,75	7,77	7,79	7,81	7,82	7,83	7,84	7,85	7,86
17	6,34	7,26	7,63	7,83	7,96	8,04	8,11	8,15	8,19	8,22	8,25	8,27	8,28	8,30	8,31	8,32	8,33	8,34
18	6,62	7,63	8,04	8,26	8,40	8,49	8,56	8,61	8,65	8,69	8,71	8,74	8,76	8,77	8,79	8,80	8,81	8,82
19	6,88	7,98	8,43	8,68	8,83	8,93	9,01	9,07	9,12	9,15	9,18	9,21	9,23	9,25	9,27	9,28	9,29	9,30
20	7,14	8,33	8,82	9,09	9,26	9,38	9,46	9,52	9,57	9,62	9,65	9,68	9,70	9,72	9,74	9,76	9,77	9,78
21	7,39	8,68	9,21	9,50	9,69	9,81	9,91	9,98	10,03	10,08	10,11	10,14	10,17	10,19	10,21	10,23	10,25	10,26
22	7,64	9,02	9,59	9,91	10,11	10,25	10,35	10,43	10,49	10,54	10,58	10,61	10,64	10,66	10,69	10,71	10,72	10,74
23	7,88	9,35	9,97	10,31	10,53	10,68	10,79	10,87	10,94	10,99	11,04	11,08	11,11	11,13	11,16	11,18	11,20	11,21
24	8,11	9,68	10,34	10,71	10,95	11,11	11,23	11,32	11,39	11,45	11,50	11,54	11,57	11,60	11,63	11,65	11,67	11,69
25	8,33	10,00	10,71	11,11	11,36	11,54	11,67	11,76	11,84	11,90	11,96	12,00	12,04	12,07	12,10	12,12	12,14	12,16
26	8,55	10,32	11,08	11,50	11,78	11,96	12,10	12,21	12,29	12,36	12,41	12,46	12,50	12,53	12,56	12,59	12,61	12,63
27	8,77	10,63	11,44	11,89	12,18	12,39	12,53	12,65	12,74	12,81	12,87	12,92	12,96	13,00	13,03	13,06	13,08	13,11
28	8,97	10,94	11,80	12,28	12,59	12,80	12,96	13,08	13,18	13,26	13,32	13,38	13,42	13,46	13,50	13,53	13,55	13,58
29	9,18	11,24	12,15	12,66	12,99	13,22	13,39	13,52	13,62	13,71	13,77	13,83	13,88	13,92	13,96	13,99	14,02	14,05

Figure 38: Stability of the side wall for paragneis with a dip of the main joint set of 70 degrees

THE BORDER BETWEEN OPEN PIT AND UNDERGROUND

A further point which is important for the mine design is the crown pillar between the open pit and the underground stopes. For the determination of the crown pillar dimension, a simplified numerical model was created in conjunction with at a simplified open pit and ore body. All footwall and hanging wall rock was defined as paragneiss.

The model has a dimension of approx. 400 x 400 m. As shown in Figure 39 the stopes were backfilled. The backfill material has the parameter of a consolidated soil. The model is based on the general assumption that the stope was extracted from the bottom to the top with a height of a single stope not higher than the determined 20 m. After extraction of one slice, the stope will be backfilled on the overlying stope can be extracted by working on the backfill material.

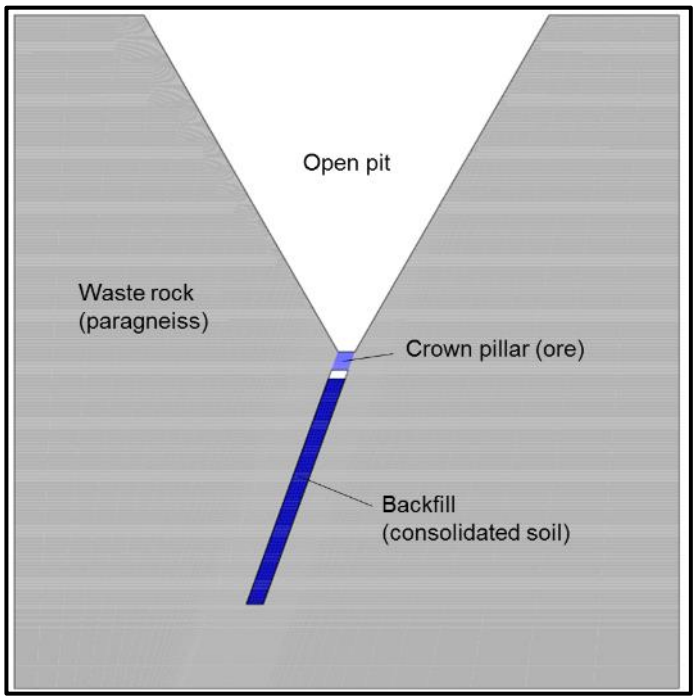


Figure 39: Geotechnical numerical model (software FLAC2D) for investigation of the crown pillar thickness

RESULTS OF THE GEOTECHNICAL EVALUATION

Several variations were performed considering the structural geological conditions (joint dip and conditions) as well. Based on these calculations a sufficient safety factor (SF > 2) was achieved with a minimum crown pillar thickness of 11 m.

The design parameters for the mine planning can be concluded as follows:

- Overall slope angle: 60 degrees
- Stope height: 20 m
- Stope width 10 m
- Crown pillar thickness: 11 m

- Backfilling with a consolidated material (sand, broken rock etc.)

Generally it must be noted that the analysis were performed for the average rock mass quality. Fault region were considered in the average value for the rock mass classification during risk analysis but not in a separate manner. Fault areas will influence the stability considerably and should be avoided, if possible, or has to be supported or designed separately. Therefore these areas must be explored and investigated more detailed in the next planning steps (geophysics, core logging). But these investigations should also be done for the areas without faults for verification of the used parameters and to confirm or improve the assumptions done in this analysis.

HYDROLOGICAL AND HYDROGEOLOGICAL ASPECTS

RECENT HYDROGEOLOGICAL FIELD INVESTIGATIONS

BACKGROUND

In June 2018 initial hydrogeological investigations were conducted within the exploration claims Nama Creek Main Zone North (MZN), Harricana (HAR), Line 60 (LIN) and Conway (CON) by DMT. Besides groundwater level measurements on existing boreholes, water permeability tests in boreholes using open systems were carried out.

In addition, the site visit served to get a first hydrogeological overview and to plan the next investigations. During the field trip between June 25th and 27th 2018, all in all the following investigations were carried out in the scope of a basic determination:

- hydraulic tests on existing drill holes (slug-tests)
- installation of a water level logger in the existing shaft within MZN
- overview with regard to estimation of groundwater recharge area
- core study and analysis with regard to hydrogeological properties
- measurement of water tables in boreholes
- check of potential discharge point/area for sump water

BASIC HYDROGEOLOGICAL INFORMATION AND PERMEABILITY

According to earlier investigations (e.g. Independent Technical Report by Caracle Creek International Consulting Inc., August 29th, 2012), “Two major aquifer systems are found in the area:

- A sandy overburden aquifer system; and
- A fractured bedrock aquifer system

Overburden aquifers are found at depths up to 20 m below existing ground surface. It is expected that the overburden sandy aquifer is mostly under water table conditions.

The primary water supply aquifer system in the area is the fractured bedrock aquifer system. Approximately 80% of water bearing fractures are been encountered at depths less than 50 m below ground surface (mbgs). Up to 20% water bearing fractures are encountered within 10 m of the bedrock surface.

The fractured aquifer system is expected to be under semi-confined to confined conditions. Bedrock fractures less than 10 m from the bedrock surface are expected to be hydraulic connected with the overburden aquifer system.

The overburden aquifer system appears to be connected to the surface water system. The regional bedrock aquifer system does not appear to contribute to the local groundwater flow system."

Therefore, the initial water permeability tests were performed on following drill holes/lithology:

- Drill holes, where bedrock/pegmatite is outcropping on the surface
- Drill holes, where also the sandy overburden aquifer (glacial, recent sediments) system occurs

Drill holes, in which loose rock overlay the bedrock, were usually no longer accessible. In contrast, for the bedrock hydraulic conductivities with values between 2·10-7 m/s and 1·10-6 m/s could be calculated. These values represent the permeability of the top approx. 10 m of the bedrock aquifer systems. In general, at this project stage the derived transmissivities and hydraulic conductivities show low permeabilities for the tested bedrock and pegmatite.

In this study phase, no values for the deeper bedrock or the overburden aquifer could be determined. Nevertheless, it can be assumed that the overburden aquifer has a slightly higher permeability based on porosity (depositional porosity). The deeper bedrock and pegmatite, however, will certainly have slightly lower values based on a low porosity with mainly post depositional porosity. In the presence of fractures and discontinuities, however, these can also be significantly higher. These parameters must be investigated in future studies.

GROUNDWATER LEVEL MEASUREMENTS

For the purpose of continuous groundwater level measurements, an automatic water level logger was installed in the existing shaft within the MZN exploration area by DMT. For this, the cover of the shaft was opened. Subsequently, the groundwater level was first measured manually by an electronic water level indicator. It was 2.55 m below the cover (this corresponds approximately to the ground level).

The temperature of the water in the shaft was 5.4° Celsius, which is quite cool considering the outside temperatures of the last days and weeks (consistently higher than 20° Celsius). Therefore, it can be assumed that the water in the shaft is groundwater, which cannot be considered as standing water. Possible water ingress and outlet zones as well as potential flow paths along the shaft wall can be determined in the course of geophysical and hydraulic investigations at the shaft.

BEDROCK GROUNDWATER DISCHARGE TO AN OPEN PIT

The amount of water resulting from the drainage of the open pit has a significant impact on the total cost of water management. For a rough estimate of the pit water pumping costs, the volume of water is calculated based on results of the current investigations like values for permeability. In addition, some assumptions must be made for the layout of the open pit.

In this project phase, the bedrock groundwater discharge is calculated using an equation for the yield of a deep well (well formula of Dupuit-Thiem, 1906). Strictly speaking, the equation applies to processes during water extraction from pore aquifers and some of the applicability requirements do not apply. However, for a

first approximate calculation of the quantities of water to be discharged, it can provide sufficient results, even if the rocks to be considered predominantly build joint aquifers.

For this equation, it is assumed that one well is located in the center of the open pit, which is fully penetrating the aquifer and produces the drawdown cone that drains the open pit. The formula assumes (quasi)stationary groundwater flow conditions. For an unconfined aquifer the equation is:

$$Q = \pi \times kf \times \frac{(H^2 - h^2)}{(\ln R - \ln r)}$$

where

Q	volume of water discharge [m³/s]
kf	hydraulic conductivity of bedrock [m/s]
H	height of the groundwater surface at the edge of the pit above an impermeable layer [m]
h	height of the lowered groundwater level in the well above the impermeable layers [m]
R	range of the drawdown cone, here the so-called equivalent radius (ARE) is used [m]
r	radius of the full penetrating well, here half the width of the bottom of the pit is used [m]

The equivalent radius includes the layout (geometry) of the open pit. The equation is

$$\text{ARE} = R = \sqrt{\frac{a \times b}{\pi}}$$

where

a	width of open pit at ground surface [m]
b	length of open pit at ground surface [m]

The ground plan of the open pit is assumed to be 140 m (= a) x 400 m (= b). Therefore, the equivalent radius is 133.5 m (= R). The impermeable layer is assumed to be 100 m (= H) deep, which at the same time represents the bottom of the open pit. The width of the bottom of the open pit is assumed to be 10 m (= 2 x r). The height of the lowered groundwater table above the impermeable layer is set to 2 m (= h), so the total drawdown is 98 m.

The greatest influence on the result is the value for the hydraulic conductivity (for 20° Celsius water temperature). Values between 1·10-8 m/s and 1·10-6 m/s are specified for this approach. With an extra amount of 30% on the discharge rate, the following volumes of water discharge result for the different hydraulic conductivities:

- kf = $1 \cdot 10^{-6}$ m/s: Q = 44.7 m³/h
- kf = $1 \cdot 10^{-7}$ m/s: Q = 4.5 m³/h
- kf = $1 \cdot 10^{-8}$ m/s: Q = 0.45 m³/h = 10.7 m³/d

The following cost estimates should be based on the most disadvantageous assumptions (kf = 1·106 m/s). Since some of the recent hydraulic tests have revealed hydraulic conductivities of up to 1·10-6 m/s, this is

also plausible. The extra amount of 30% also takes into account the water, which can still flow from the overburden aquifer system and from the underlying bedrock below the pit bottom into the open pit. In addition, the proportion of precipitation is taken into account.

OUTLOOK

For current and subsequent feasibility studies and environmental impact assessments (EIA) valuable data and preparations have already been identified. In the near future further investigations will be carried out, which will end in a complete hydrogeological study.

This study will determine baseline conditions such as ground water flow direction, the catchment areas and groundwater recharge rates, hydraulic conductivities of the hydrogeological formations and probable weakness zones in the bedrock with the potential to channel larger amounts of water. The outcome of the study will help evaluate the potential local and regional impacts of the mining activities on groundwater and surface water and propose an appropriate monitoring plan. The monitoring will be related to hydraulics (groundwater and surface water levels) and quality (groundwater and surface water chemistry).

Specifically, for example, it is planned to complete some of further exploration drill holes to groundwater measuring points or wells (observation wells). They will be constructed in such a way that all lithologies that have potentially to be dewatered can be identified and tested. At these wells, further hydraulic tests will be carried out. It is also planned to carry out a longer pumping test at the exiting shaft at MZN following geophysical investigations.

In this framework also the results of the hydrogeological work performed on the property in the late 1950's will be taken into account. In addition, a verification and evaluation of the hydrological results by Trow's Water Balance Study 2011 will be done. Therefore the Water Well database from Ministry of the Environment will be checked, the weather (meteorological) data will be updated and a digital terrain model will be provided. The surface water quality monitoring Jun 2011 – Nama Creek Claim Block by exp Services Inc. within the framework of a baseline ecological study on the Georgia Lake Property will also be included in the Environmental Impact Assessment (EIA).

MINE OPTIMIZATION

INTRODUCTION

A common approach to decide where to finish the open pit and start the underground mine (transition problem), is to determine the economic size of the open pit. Underground mining then focuses on the remaining portion of the deposit outside the pit shell. Other methods, such as the opportunity cost approach, focus on the optimisation of the transition from surface to underground mining considering both open pit and underground mining costs. Therefore, the optimum pit design changes if shallow parts of the orebody can also be economically exploited by underground mining. An important factor hereby is the planning cut-off grade, which affects the amount of waste to be removed during the mining operation.

Table 49 represents the parameters used for the mine optimisation analysis and the mine design. All cost and operating parameters are based on preliminary estimates for developing the economic pit and the underground mine layout.

Parameter		Open Pit	Underground
Costs	Ore Mining	3 USD/t	25 USD/t
	Waste Removal	3 USD/t	
	Processing	20 USD/t	
Mining	Dilution	10 %	15 %
	Losses	5 %	10 %
Mine Design	Pit Slope Angle	60 °	
Processing	Recovery (Li_2O)	80 %	80 %
	ROM grade	app. 1 %	app. 1 %
	Concentrate (Li_2O)	6 %	6 %
	Price concentrate	730 USD/t	730 USD/t

Table 49: Economic and operating parameters.

OPEN PIT OPTION

PIT OPTIMISATION PROCEDURE (OP ONLY)

To evaluate the feasibility of an open pit only mining scenario, a standard pit optimisation analysis was conducted. The objective of the pit optimisation is to determine the ultimate pit limit, which will maximise specific economic and technical criteria whilst satisfying practical operational requirements. The optimum pit shape is therefore based on the highest project cash flow (e.g. revenue, mine operational costs, ore processing and ore handling costs, general overhead costs) in present value terms. In addition to these cost parameters, the mineral resource model, geotechnical slope parameters, processing recoveries and other project constraints are also included in the optimisation analysis.

The MineSight Economic Planner (MSEP) software package was used to carry out the OP optimisation analysis for this study. Based on the Lerch-Grossmann Algorithm (LG Algorithm), MSEP calculates the value of all blocks of the resource block model and then optimises the pit outline subject to a range of side constraints. This procedure basically involves finding the subset of blocks in the block model that maximises total value whilst obeying pit slope constraints and taking into account the costs of uncovering a block. Applying a cut-off grade can significantly impact the optimisation result as material below the cut-off grade will be treated as waste. The standard MSEP output is a sequence of three-dimensional pit outlines (nested pit shells), which can be used as a guide to detailed and practical mine design. The input parameters used for the pit optimisation include:

- Overall pit slope angle;
- Mining and processing costs, incl. ore handling and transportation, royalties, etc.;
- Mining dilution and recovery;
- Processing recovery;
- Commodity/ product revenues;
- Cut-off grade.

An array of incremental pit shells was generated with MSEP for all pegmatite dykes by varying the revenue factor (RF).Tonnage and grades associated with these pit shells are summarised in Table 50, Figure 40 and Figure 41. Cross sections through all pegmatite areas with several of the pit shells are illustrated in Figure 42 to Figure 46. The cut-off grade used for the optimisation analysis was 0.65 % Li_2O.

Pit	Revenue Factor	Ore (Mt)	Waste (Mt)	Li_2O (%)	Value (MUSD)	Stripping Ratio
10	0.1	0.0	0.0	0.00	0.0	0.0
11	0.2	0.0	0.0	0.00	0.0	0.0
12	0.3	0.0	0.0	1.63	0.1	0.3
13	0.4	0.1	0.1	1.28	13.6	0.7
14	0.5	0.6	1.6	1.12	55.2	2.5
15	0.6	1.3	5.2	1.08	98.6	4.1
16	0.7	2.3	16.1	1.03	147.4	7.0
17	0.8	3.2	26.8	1.00	178.6	8.5
18	0.9	3.6	32.6	0.98	189.0	9.1
19	1.0	4.1	43.1	0.96	194.1	10.5
20	1.1	5.1	67.6	0.94	187.4	13.4
21	1.2	5.4	78.8	0.94	177.7	14.5
22	1.3	5.7	87.5	0.93	166.8	15.5
23	1.4	5.9	98.5	0.93	152.1	16.7
24	1.5	6.1	105.6	0.93	140.9	17.4
25	1.6	6.2	114.9	0.92	124.5	18.5
26	1.7	6.3	120.4	0.92	114.1	19.1
27	1.8	6.4	126.1	0.92	103.2	19.7
28	1.9	6.6	140.5	0.92	73.1	21.3
29	2.0	6.7	148.7	0.92	56.4	22.2

Table 50: Pit optimisation results

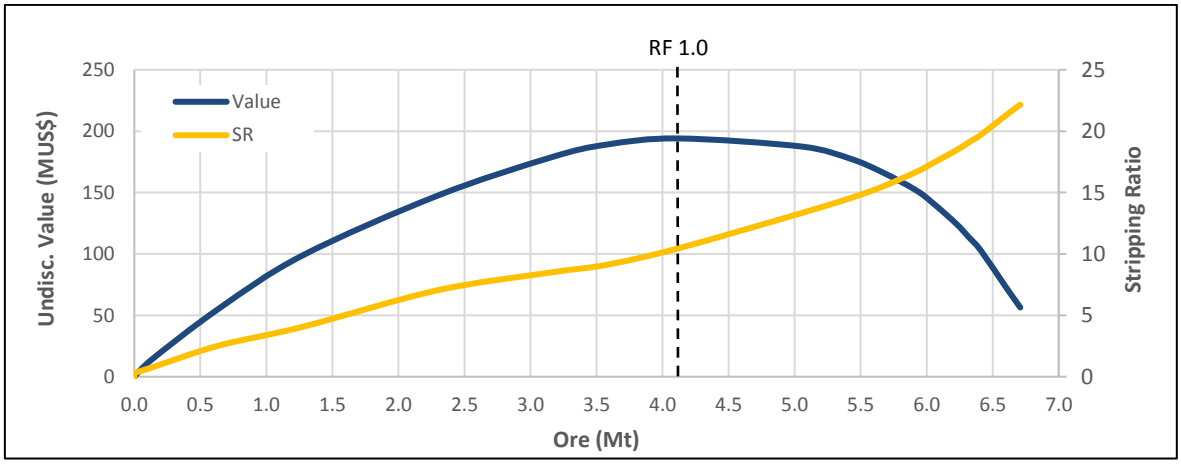


Figure 40: Pit optimisation result graph – SR &pit value (undiscounted) based on tonnes of ore mined at different revenue factors.

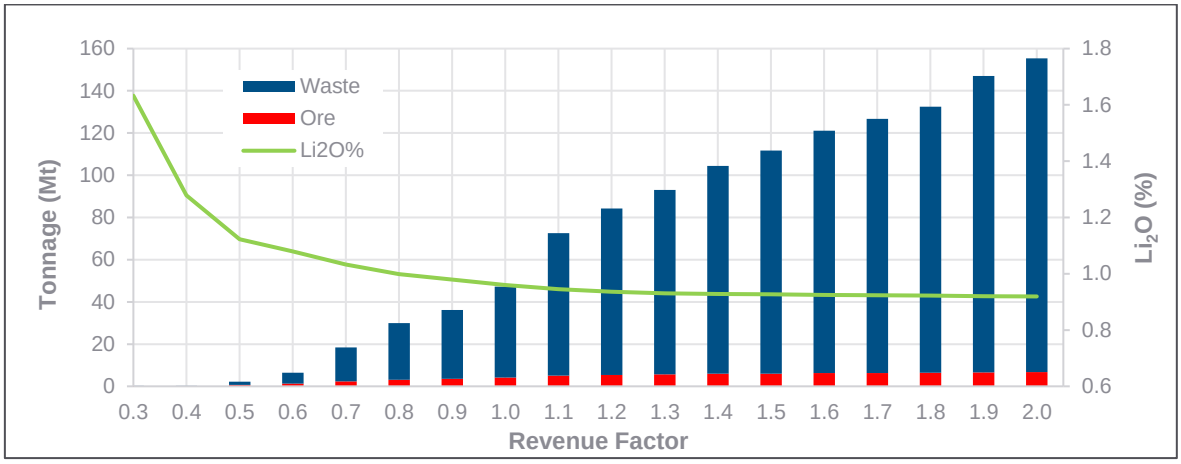


Figure 41: Pit optimisation result graph – ROM tonnes and grade at different revenue factors.

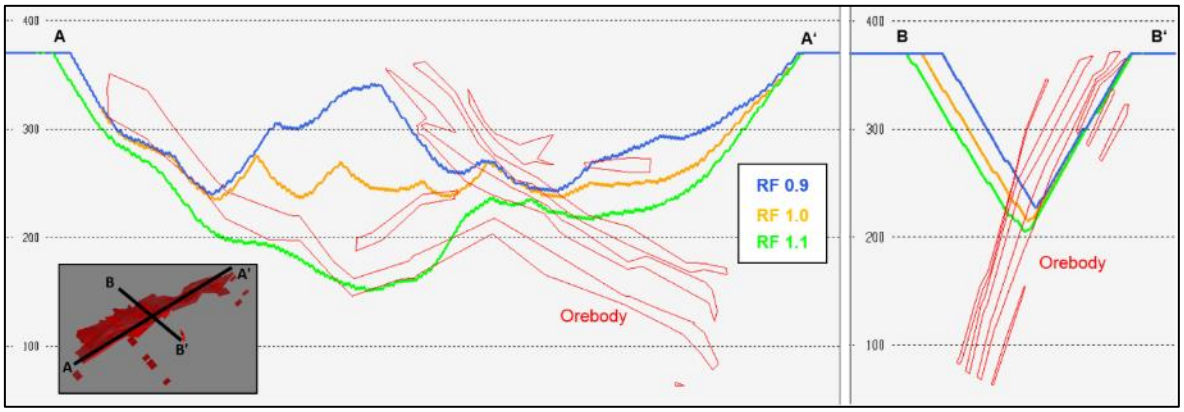


Figure 42: Pit optimisation shells for Main Zone/ North (OP only).

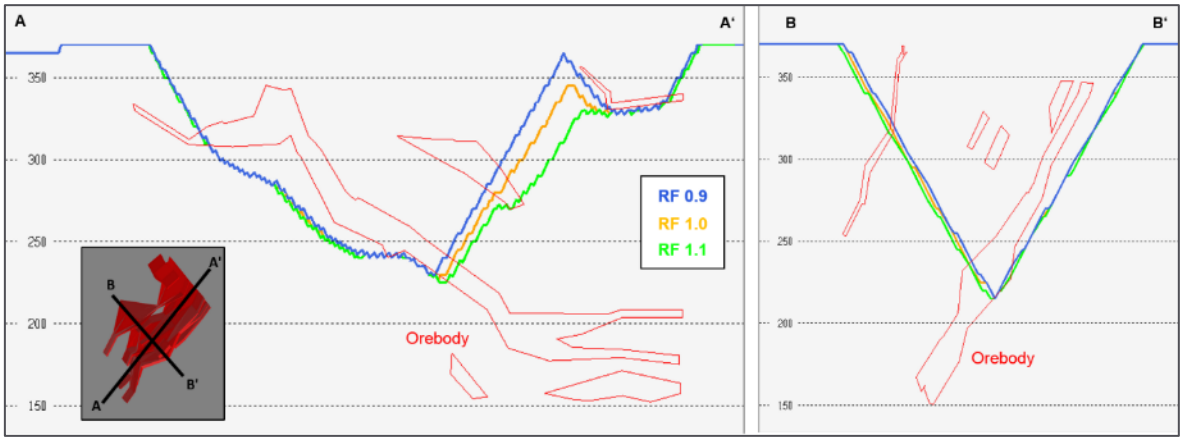


Figure 43: Pit optimisation shells for Main Zone/ Southwest (OP only).

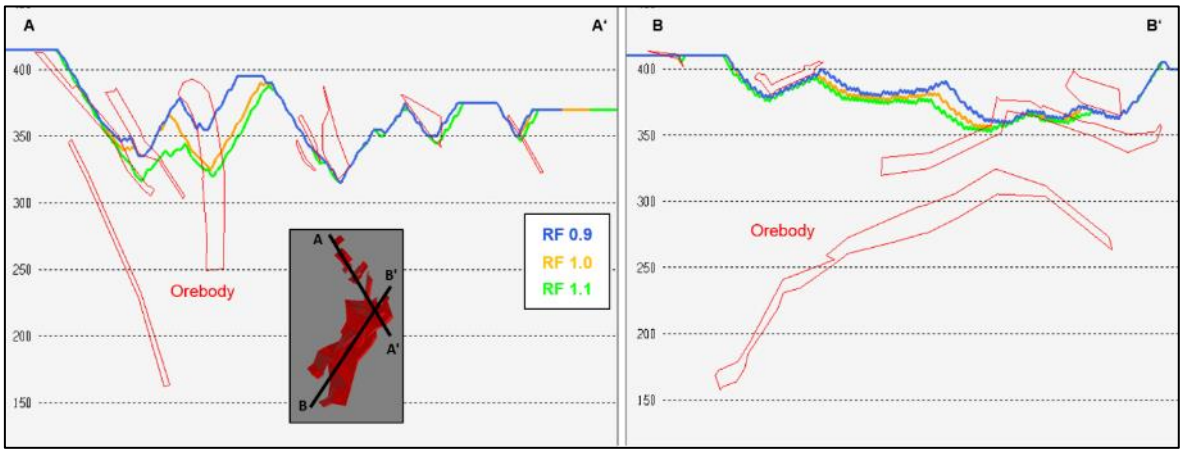


Figure 44: Pit optimisation shells for Harricana (OP only).

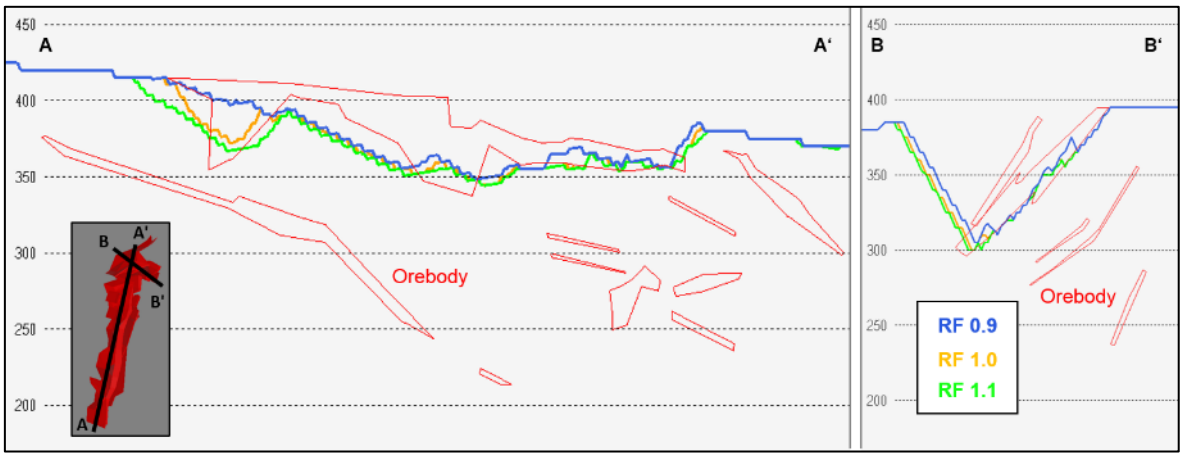


Figure 45: Pit optimisation shells for Line 60 (OP only).

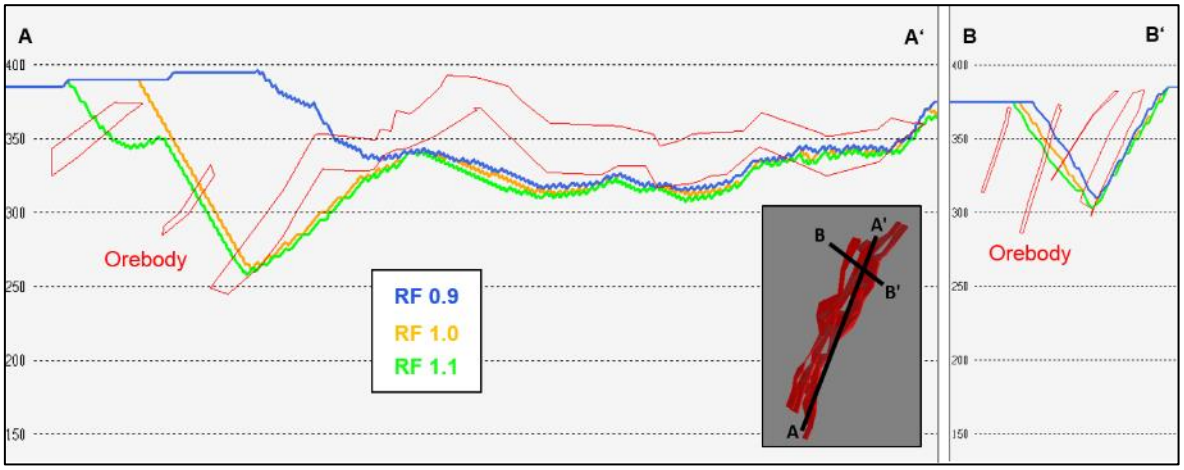


Figure 46: Pit optimisation shells for Conway (OP only).

The results show that the ultimate pit shell in terms of pit value[1] and required ROM grade was generated with RF 1.0 (Pit 19). Revenue factors below 1.0 produced smaller pits with higher grades and lower stripping ratio, while RF greater than 1.0 generated larger pits with decreased profitability, lower grades, and significantly higher stripping ratios. The open pit design should therefore be based on Pit 19 (Figure 47). However, in addition to the relatively high SR of 10.5, the deeper sections of the pegmatite dykes were not included in the ultimate pit envelope. Apart from the L60 pegmatite area, where over 90 % of the ore can be extracted by open pit mining, nearly half the total resource (43 %) is not contained within the UPL (ultimate pit limits) (Table 51).

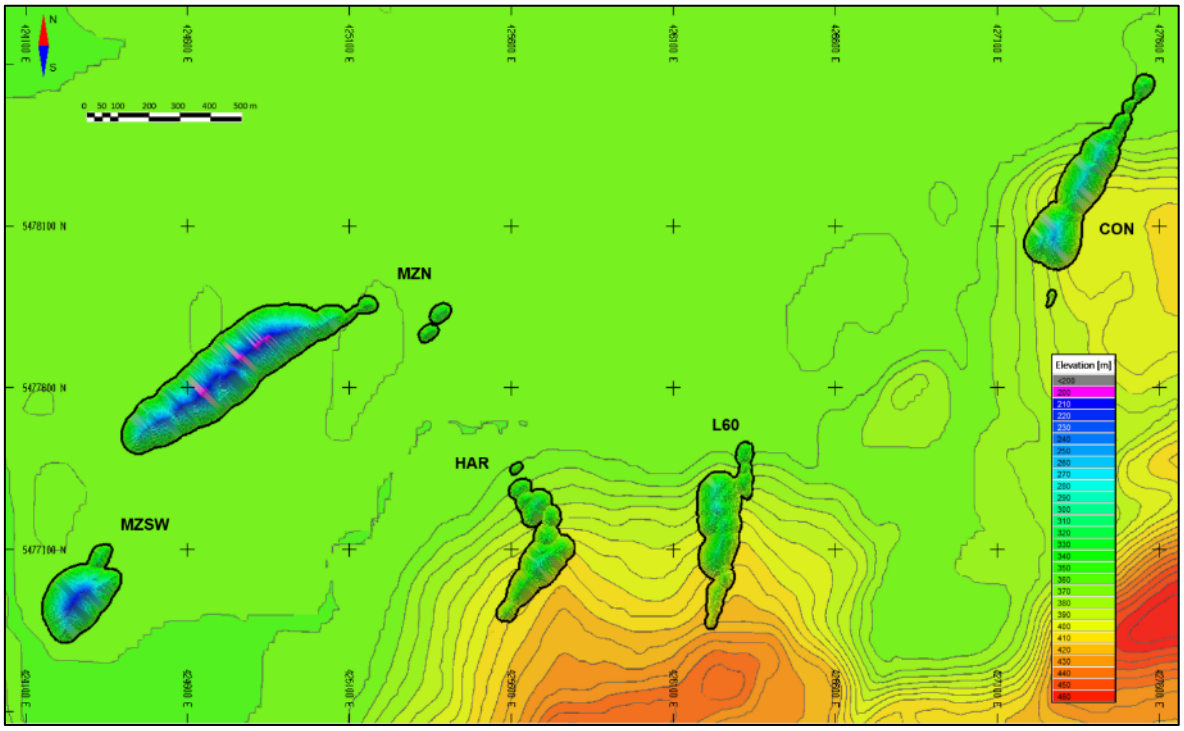


Figure 47: Plan view of pegmatite zones and ultimate pit shell (Pit 19) of the OP only option.

Zone	Within UPL			Outside UPL		
	Ore (Mt)	Li_2O%	%	Ore (Mt)	Li_2O%	%
MZN	1.59	0.96	42	2.17	0.88	58
MZSW	0.42	1.04	58	0.3	0.79	42
HAR	0.40	0.92	63	0.23	0.79	37
L60	0.88	0.95	92	0.07	0.74	8
CON	0.83	0.95	70	0.36	0.78	30
TOTAL	4.12	0.96	57	3.14	0.85	43

Table 51: Resources contained within and outside the ultimate pit limit (UPL).

[1] Pit values generated in the optimisation process only served as guide for pit shell selection as they do not take into account capital expenditures and other financial parameters apart from the above.

It should be noted that a decreased planning cut-off grade will result in a larger pit envelope allowing to mine a larger portion of the pegmatites. However, the deeper parts of the pegmatites will have to be left in the ground as mining these sections would be unfeasible due to a significant increase in stripping ratio and mining costs.

UNDERGROUND OPTION

UNDERGROUND OPTIMISATION PROCEDURE (OP VS. UG)

To evaluate the feasibility of an underground only option, the opportunity cost approach was applied. This modelling approach, which extends the use of the LG Algorithm, takes into account the opportunity value of an underground mine whilst optimising the pit outline. This concept is based on maximum allowable stripping ratios that are accounted for by way of charging the open pit an "opportunity cost" equal to the value lost by not mining a block by an underground method. Although no uncovering costs are required for blocks mined by underground methods, other extraction costs (costs for excavating drifts, shafts, declines etc.) have to be considered to for each block.

Mine optimisation was conducted using a combination of MSEP, MineSight Stope (MSStope), and MinSight's Model Calculation Tool (MCT). Input parameters included economic and operational parameters for both OP and UG mining.

UNDERGROUND OPTIMISATION RESULTS

Based on the same optimisation constraints, a new pit shell (Pit 152) was generated with MSEP for all pegmatite dykes using RF 1.0 and cut-off grade 0.65% Li2O. Figure 48 to Figure 53 illustrate the outline of this pit shell for all pegmatite areas. The pit limit marks the transition depth between surface and underground mining.

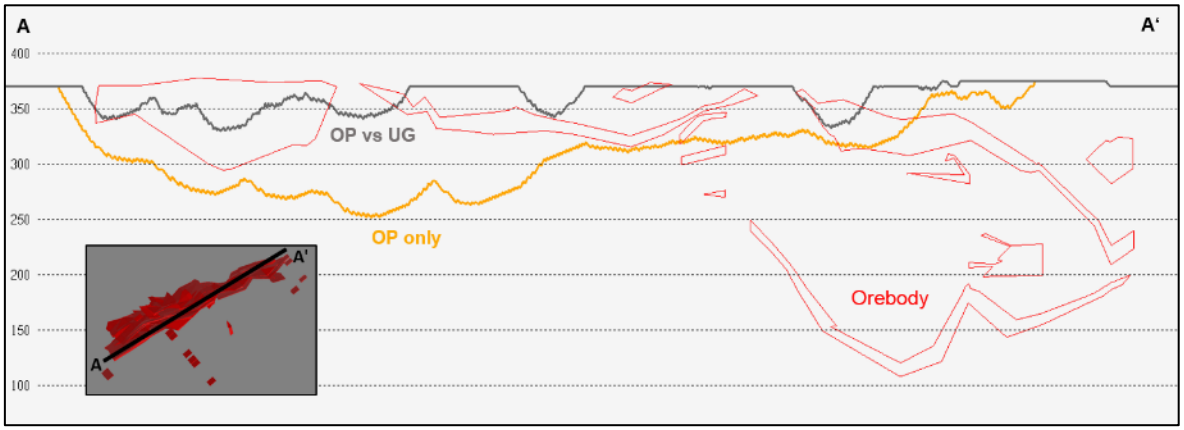


Figure 48: Pit optimisation shell for Main Zone/ North (UG option).

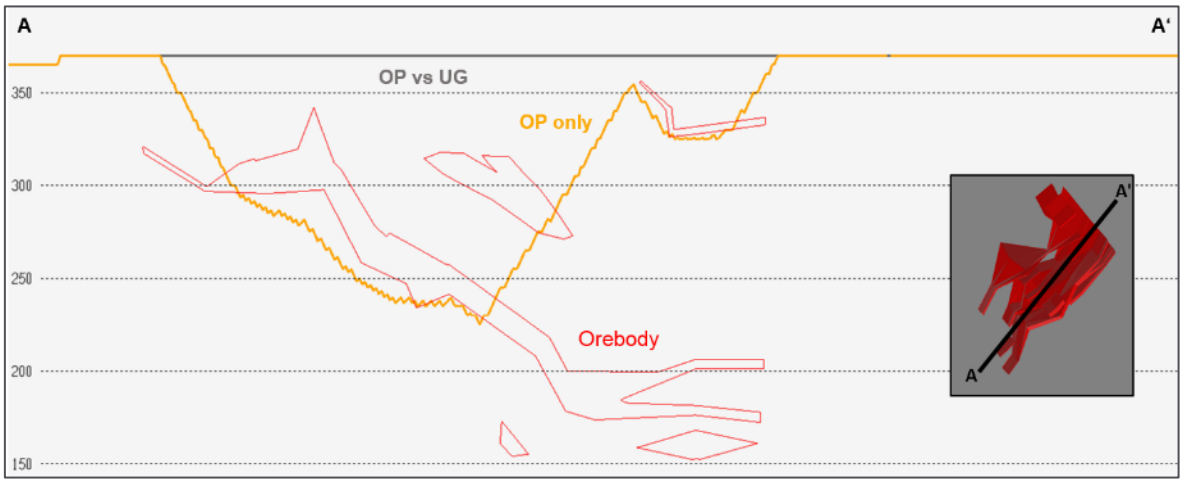


Figure 49: Pit optimisation shell for Main Zone/ Southwest (UG option).

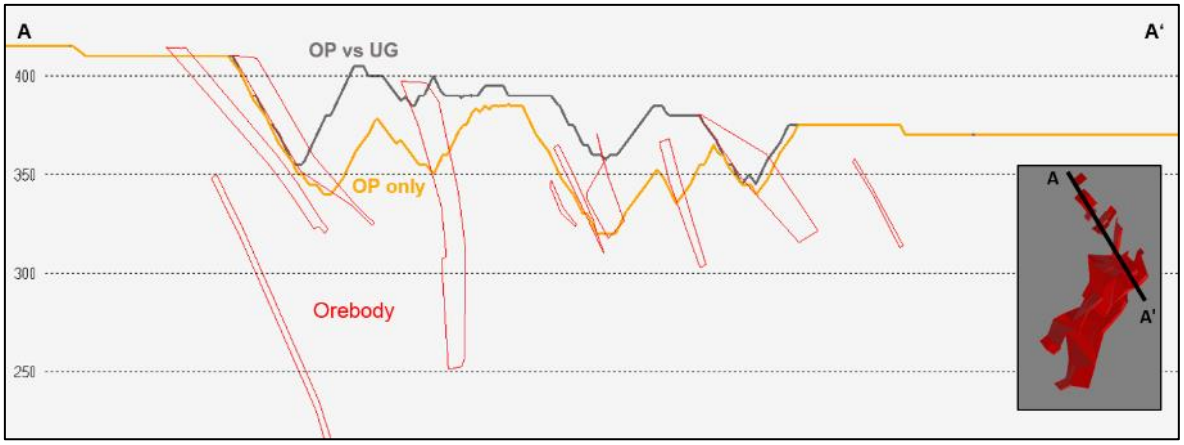


Figure 50: Pit optimisation shell for Harricana (UG option).

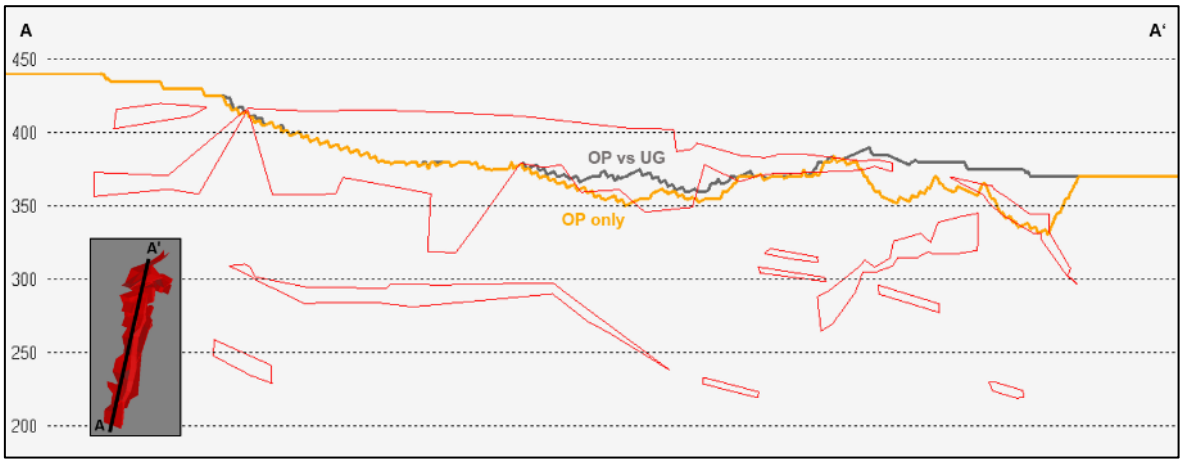


Figure 51: Pit optimisation shell for Line 60 (UG option).

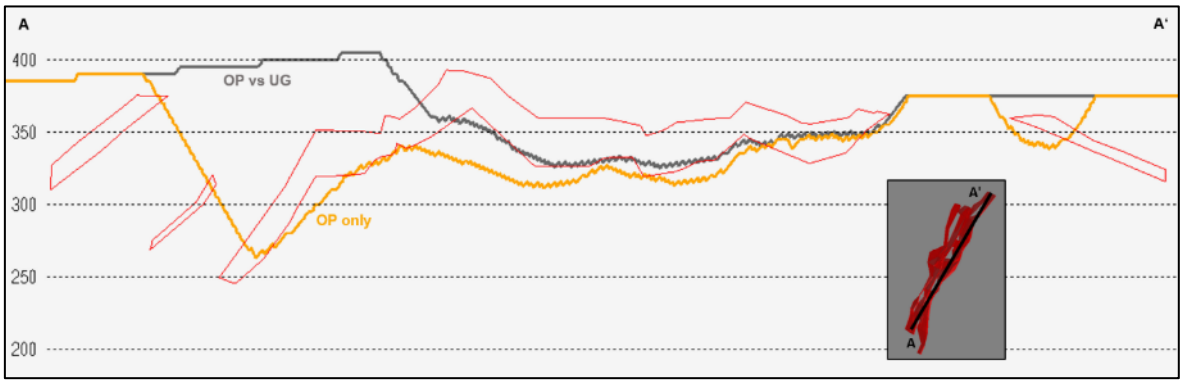


Figure 52: Pit optimisation shell for Conway (UG option).

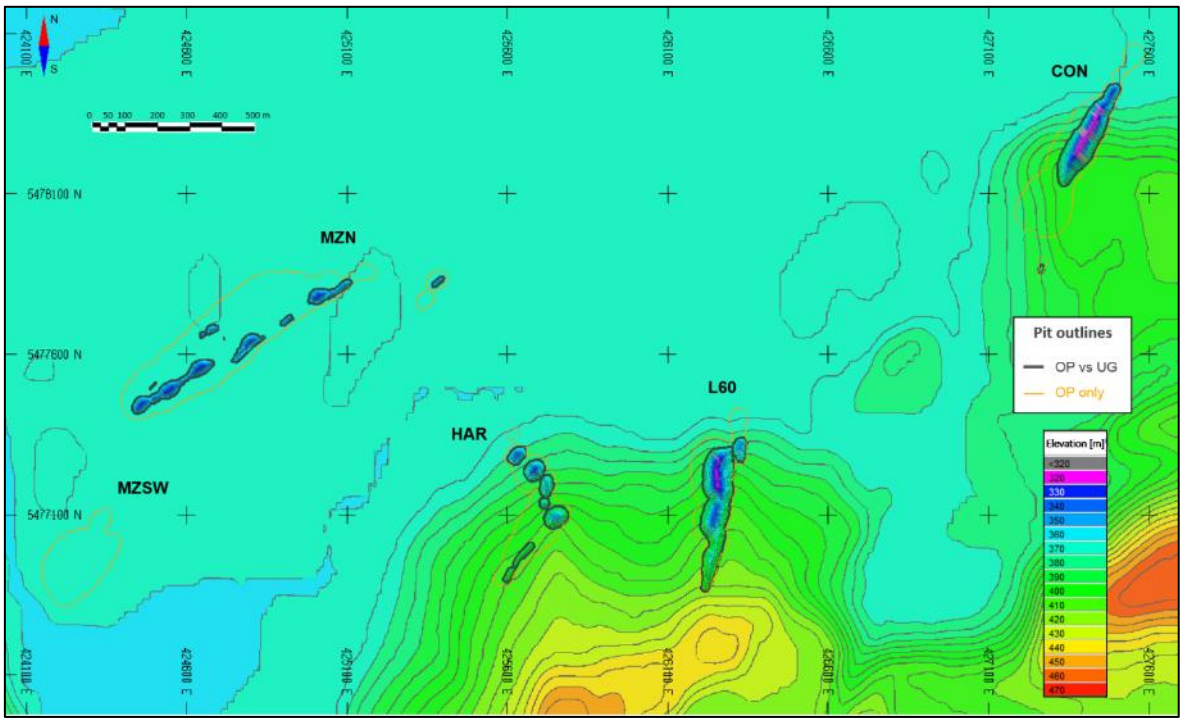


Figure 53: Plan view of pegmatite zones and ultimate pit shells of OP only and OP vs UG optimisation analysis.

The results indicate that a large portion of the resource (81 %) can be economically extracted by underground mining (Table 52). However, apart from the MZSW pegmatite area, which may be fully mined by UG methods, a certain amount of open pit mining is required to harvest the uppermost sections of all other pegmatite dykes.

Zone	Within UPL			Outside UPL		
	Ore (Mt)	Li_2O%	%	Ore (Mt)	Li_2O%	%
MZN	0.18	0.96	5	3.6	0.91	95
MZSW	-	-	-	0.7	0.94	100
HAR	0.15	0.95	18	0.5	0.84	75
L60	0.66	1.00	58	0.3	0.80	30
CON	0.40	1.01	34	0.8	0.84	66
TOTAL	1.40	0.99	19	5.9	0.89	81

Table 52: Resources contained within and outside Pit 152

HYBRID OPTION (OPEN PIT AND UNDERGROUND)

SELECTION AND OPTIMISATION PROCEDURE

Based on the results of the previous section, the hybrid mining option was found to be the most suitable solution for all Nama Creek pegmatites. While the upper portions of the pegmatite dykes will be mined by open pit, deeper sections will be exploited by underground mining. This will allow fast extraction of outcropping and near-surface ore without the necessity of complex and cost intense developments generating revenue right from the start of the operation.

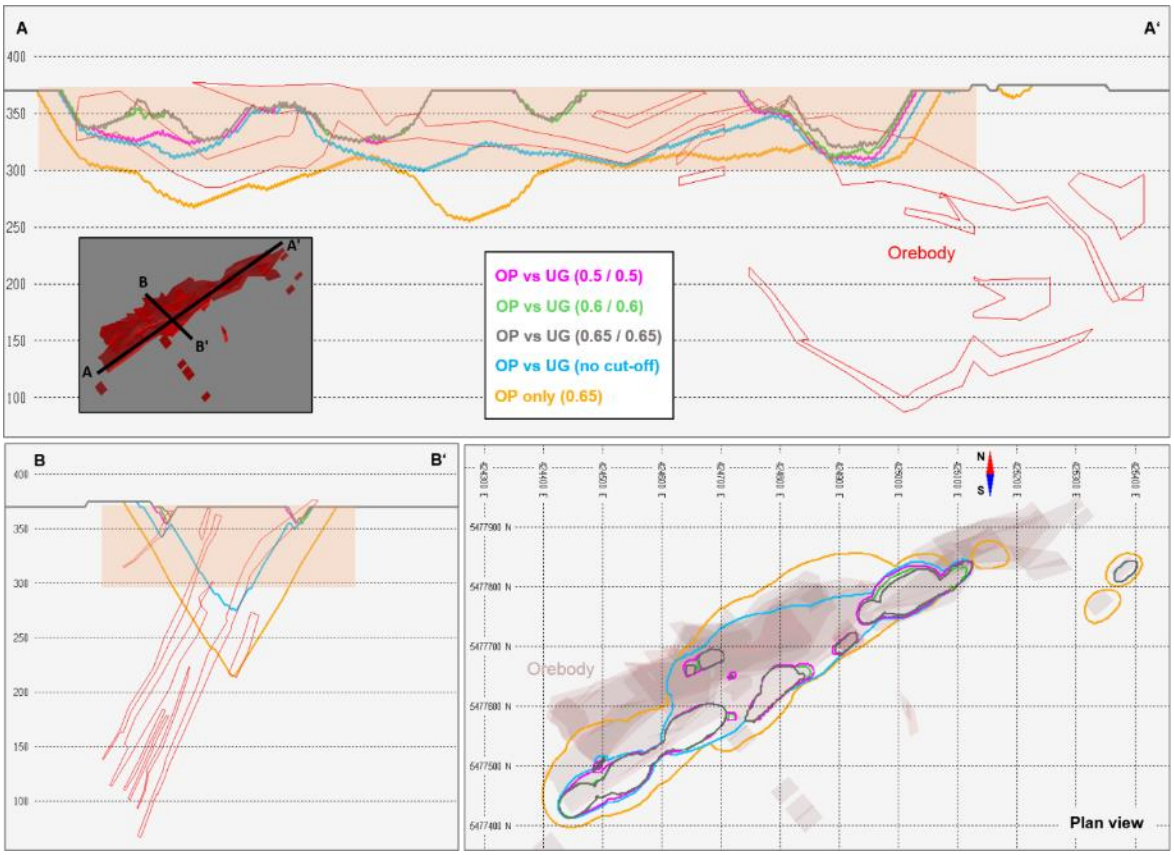


Figure 54: Selected pit optimisation shells (top, left) and pit outlines (right) based on different cut-off grades for MZN. Red highlighted zones indicate pegmatite sections to be extracted by OP mining.

All required infrastructure for underground mining can be developed in the meantime provided all pits contain enough ore tonnes to cover annual production requirements.

The optimum transition depth between open pit and underground mining was determined by comparing different pit shells generated between the limits of Pit 19 and Pit 152 by varying the cut-off grade. Modification of the OP and UG cut-off grade during the optimisation analysis ultimately resulted in a smaller or larger pit envelope.

Figure 54 demonstrates the selection process for the MZN pegmatite area. The result suggests that a sufficient portion of the ore can be extracted economically to a depth of around 300 m (final pit depth c. 70 m asl), which would cover the first year of production.

OP/ UG OPTIMISATION RESULTS

Solving the transition problem, a suitable pit envelope for all pegmatite areas was selected based on the following conditions:

- UPL within the economic limits of an open pit mining operation (OP only);
- Annual ore production for initial mining periods must be fully covered by OP mining;
- One large pit per pegmatite area as opposed to several small pits.

Resources contained within and outside the ultimate pit limit are summarised in Table 53. According to these results, around 35 % of the ore can be recovered by open pit mining while a significant portion (c. 65 %) can be mined by underground methods. The optimised pit shell is based on a cut-off grade of 0.55 % for open pit mining and 0.65 % for underground mining. Figure 55 to Figure 61 illustrate the outline and extend of the preliminary ultimate pit shell for all pegmatite areas.

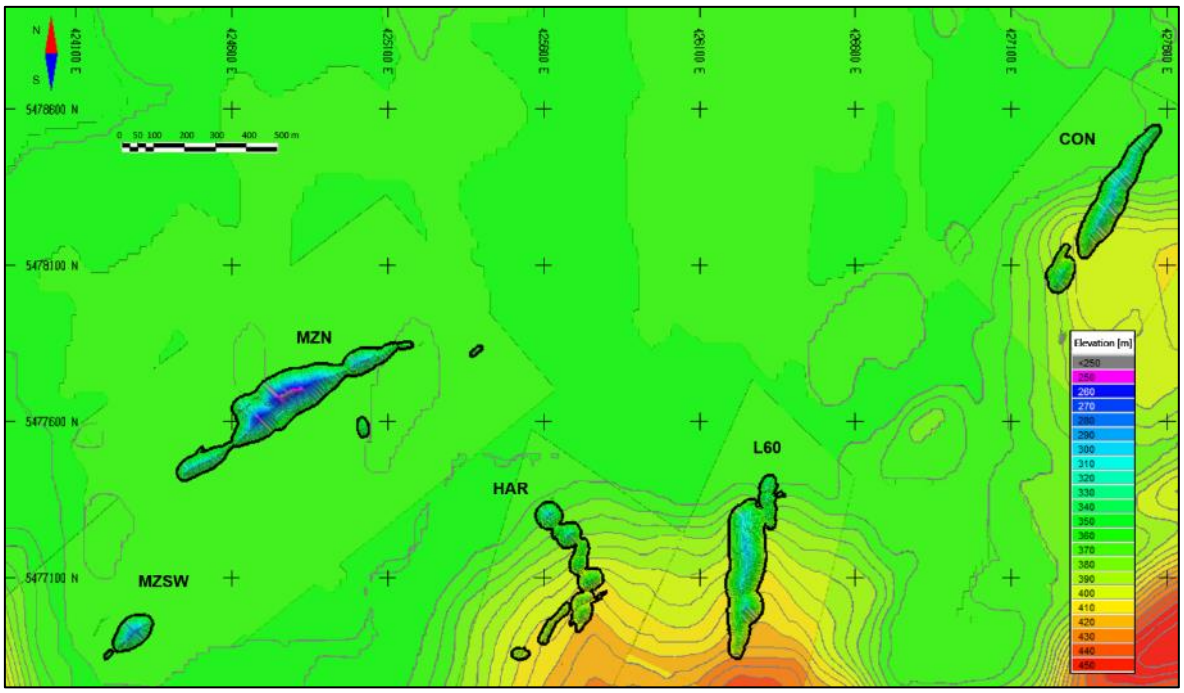


Figure 55: Plan view of pegmatite zones and outline of the ultimate pit (hybrid option).

Zone	OP Mine (0.55 % Li_2O cutoff)			UG Mine (0.65 % Li_2O cutoff)		
	Ore (Mt)	Li_2O%	%	Ore (Mt)	Li_2O%	%
MZN	0.77	0.96	21	2.96	1.00	79
MZSW	0.09	0.96	12	0.69	1.03	88
HAR	0.21	1.00	35	0.40	0.92	65
L60	0.97	0.97	90	0.10	0.89	10
CON	0.51	1.04	43	0.67	0.92	57
TOTAL	2.56	0.99	35	4.82	0.99	65

Table 53: Resources (measured and indicated) recovered by the hybrid option.

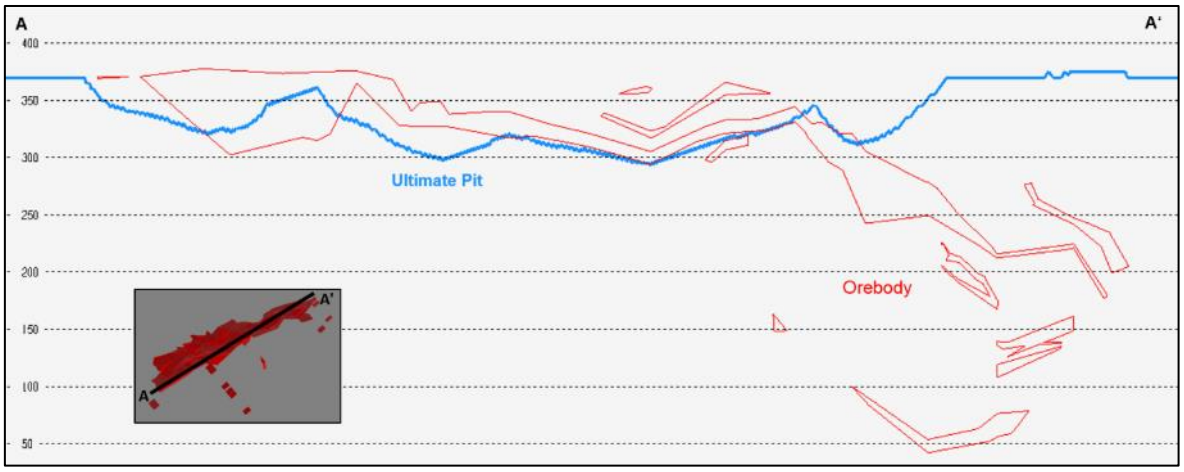


Figure 56: Ultimate shell for Main Zone/ North (hybrid option).

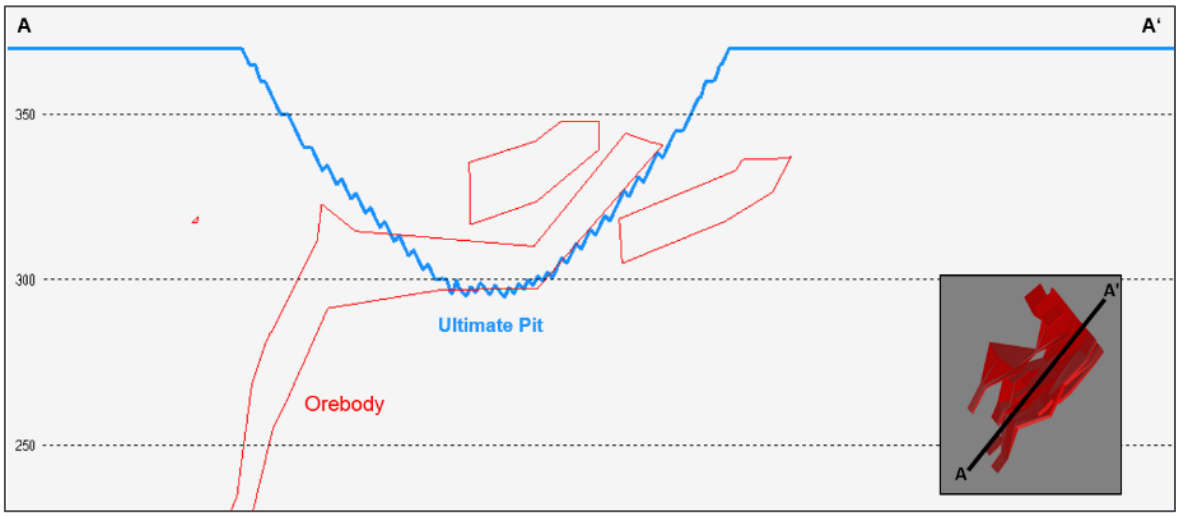


Figure 57: Ultimate pit shell for Main Zone/ Southwest (hybrid option).

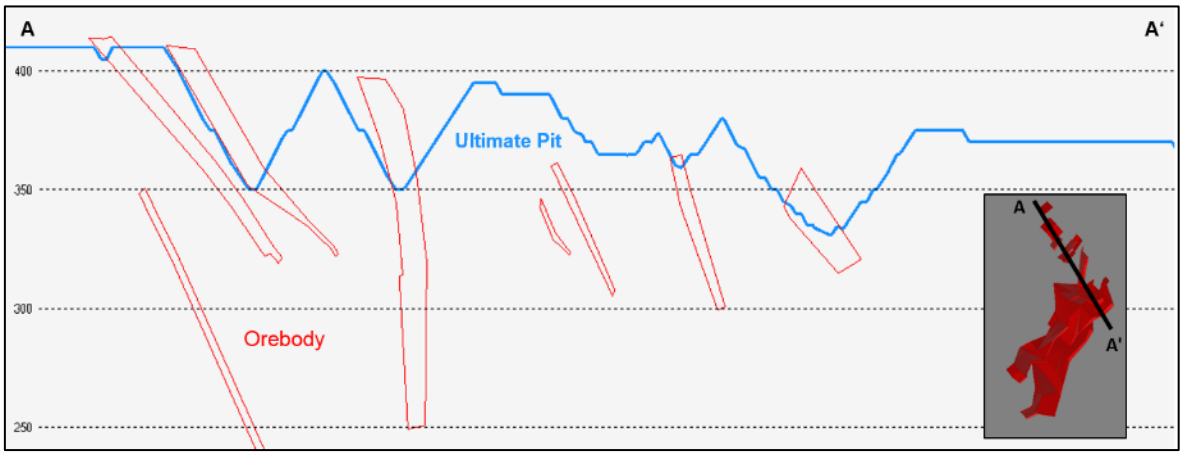


Figure 58: Ultimate pit shell for Harricana (hybrid option).

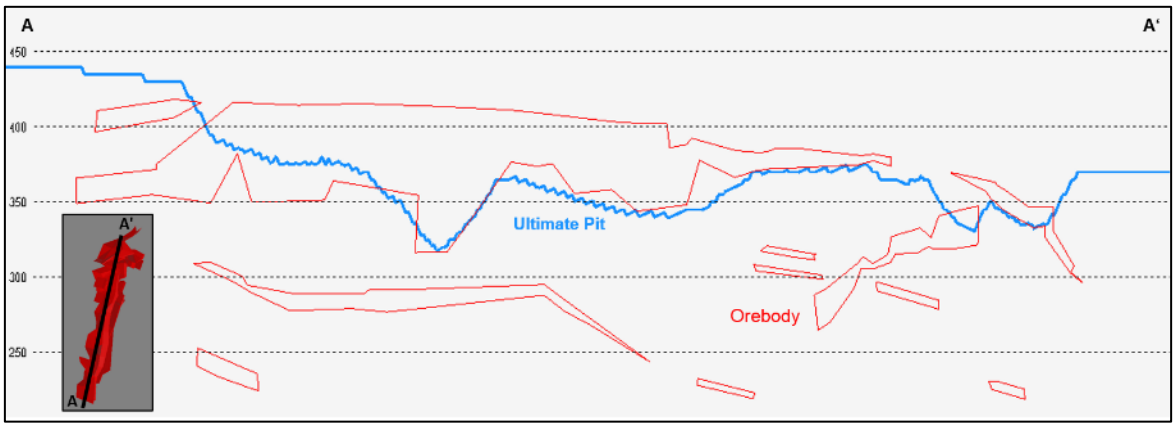


Figure 59: Ultimate pit shell for Line 60 (hybrid option).

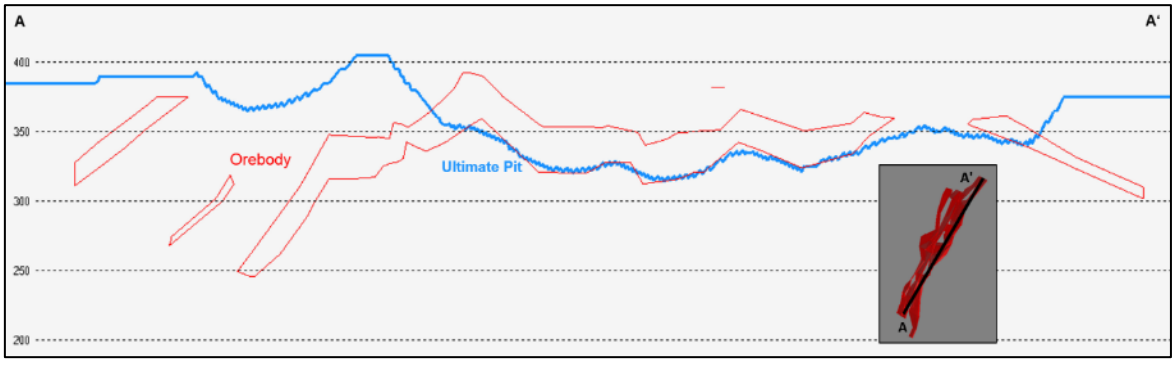


Figure 60: Pit optimisation shell for Conway (hybrid option).

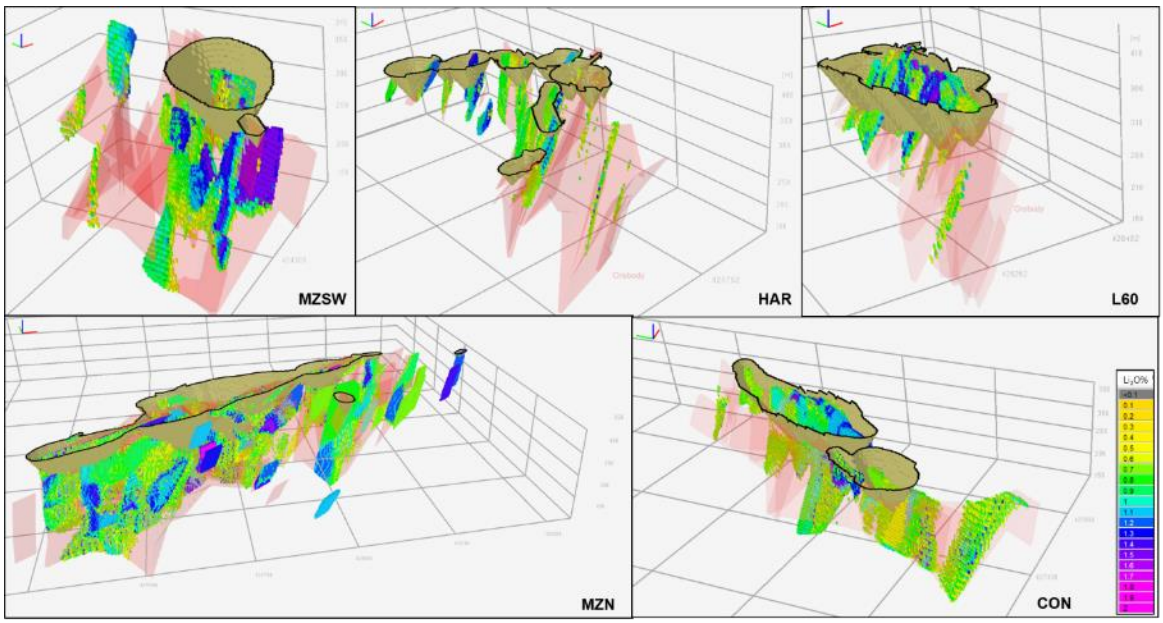


Figure 61: 3D views of ultimate pit shells of 5 pegmatite areas, with original orebody wireframes (red) and the block model showing blocks with Li_2O grades above cut-off.

MINE DESIGN

BASIS OF PRELIMINARY MINE DESIGN

STRATEGY FOR PRELIMINARY MINE DESIGN

Compared to surface operations, the selection of a suitable method as well as the preliminary design and planning of the mine layout are more complex in underground environments. The complexity and expense of underground mining operations require careful consideration and extreme care during mine design and development. The main objectives of mine layout design and sequencing are:

- Maximisation of ore extraction;
- Minimisation of costs per extracted ore tonne;
- Ensuring production at constant rates to meet annual production requirements and balance equipment utilisation;
- Minimisation of ground control issues;
- Minimisation of the up-front development capital;
- Minimisation of losses and dilution (grade control).

MINERAL POTENTIAL

Based on the optimisation as well as applying losses and dilution and additional inclusion of conservative 30% of the inferred resource the following mineral potential can be outlined (Table 54). The mineral potential was used for the development of the preliminary mine plan and design.

Zone	OP Mine (0.55 % Li_2O cutoff)			UG Mine (0.65 % Li_2O cutoff)		
	Ore (Mt)	Li_2O%	%	Ore (Mt)	Li_2O%	%
Based on Measured and Indicated Resources						
MZN	0.81	0.87	21	3.06	0.87	79
MZSW	0.10	0.87	12	0.72	0.90	88
HAR	0.22	0.91	35	0.42	0.80	65
L60	1.01	0.89	91	0.11	0.77	9
CON	0.53	0.95	44	0.69	0.80	56
Based on Inferred Resources						
MZN/HAR				2.00	0.89	100
TOTAL	2.67	0.90	28	6.99	0.87	72

Table 54: Mineral Potential

DESIGN PARAMETERS

Preliminary mine design parameters based on rock mechanical properties (see section 0) and assumptions are summarised in Table 55.

Parameter	Open Pit	Underground
Pit slope angle	60°	
Stope width		10 m
Stope height		20 m
Stope length		< 900 m
Crown pillar thickness		11 m
Declines (width x height)		5 m x 5 m
Access drifts (width x height)		4 m x 4 m

Table 55: Mine Design Parameters

MINING STRATEGY

INTRODUCTION

The most suitable mining strategy for all Nama Creek pegmatites is a combination of surface and underground methods (Figure 62).

In general, underground access can be developed from surface or from the pit bottom of the open pit with reduced incline length for ore body access, money and development time.

Both cases have been applied in the mining strategy, wherever possible access form the pit bottom have been chosen. Only the access to the MZN ore body has to be developed while the open pit is still in operation and therefore a separate access has been chosen.

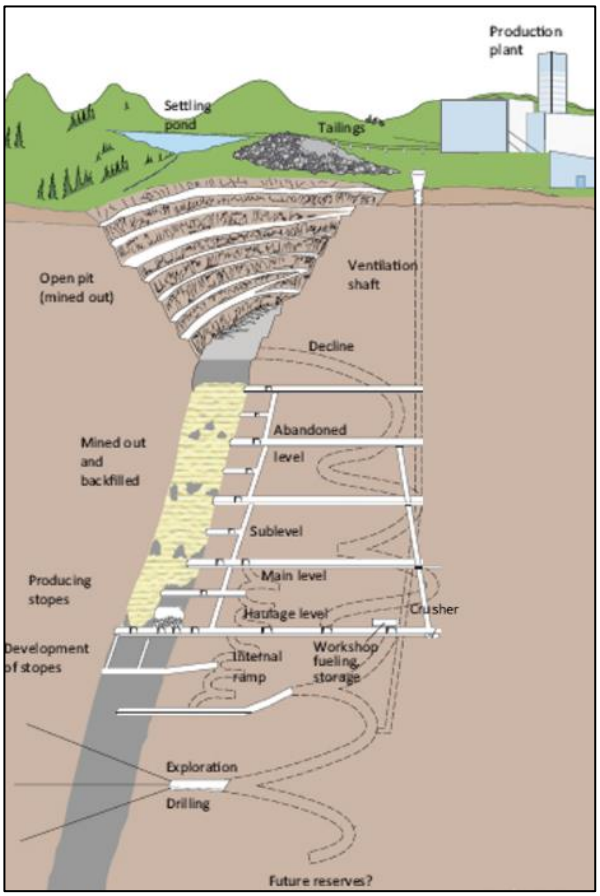


Figure 62: Mining strategy for Nama Creek (Unifiliar drawing adapted from Atlas Copco, 2000).

OPEN PIT STRATEGY

For outcropping and near-surface pegmatite sections, standard open pit methodologies will be employed. The rock will be destroyed by drilling and blasting, and then trucked to the processing plant (ore), the waste dump (wall rock, material below cut-off grade) or at some point as backfill underground. Excavation progresses vertically and laterally from the surface via benches.

The following heavy-duty mining equipment and machinery will be required for the open pit operation:

- Production drill rigs;
- Explosives charging trucks,
- Hydraulic shovels;
- Haul trucks;
- Other utility and supply vehicles.

UNDERGROUND MINING STRATEGY

The remaining pegmatite sections will be mined by sublevel stoping with backfilling, which is a combination of cut and fill mining and sublevel stoping, allowing the recovery of most of the resource at a reasonable cost

level. To prepare the orebody for extraction, portals have to be prepared as well as openings have to be driven in advance. Initial openings have to provide access to the mine and expose the orebody. Such access ways are usually single openings facilitating roads, ventilation, power, air and water lines, shops and other infrastructure. In addition, ongoing openings have to be progressed during the operation.

For both development and ore production conventional drill-blast-muck cycles will be applied. All stopes will be mined based on a stoping sequence, which depends on the number of unmined or backfilled stopes to be left in place to conform to pillar size requirements, and is subject to detailed production scheduling.

Production stope access will be provided by 2 sublevels, which will follow the shape of the pegmatite dykes. Drilling will be done from either level depending on the local conditions. Remote-controlled load-haul-dump loaders (LHDs) will be deployed for mucking and hauling of the blasted rock, which will avoid the exposure of mine workers to the risk of rockfall inside the stopes after the blasting process.

The broken material will then be hauled through the footwall drifts to ore passes. These conduits connect all extraction levels with the central decline serving as main material haulage route to the surface. To guarantee structural integrity of all underground developments, all stopes will have to be backfilled after extraction.

The following heavy-duty mining equipment and machinery will be required:

- Production drill rigs (Top-hammer longhole drills);
- Face drill rigs for development (tunnelling jumbos);
- Explosives charging trucks,
- Scaling rigs;
- Load-Haul-Dump units (LHDs);
- Underground trucks;
- Concrete spraying equipment;
- Rock bolting rigs,
- Other utility and supply vehicles (e.g. fuelling vehicles, water trucks).

UNDERGROUND PRE-PROCESSING

In order to reduce dilution to a minimum and to reduce ore transport to surface the installation of a crushing and sorting unit has been foreseen. Ore sorting in this case means separation of waste from ore or separation of low grade ore from high grade ore. The ore is sorted to remove waste and improve the efficiency and capacity of processing plant while increasing the grade.

The optical sorting technology has made enormous progress in the last decades. The capacities have been increase to 50 to 150 tph and machine depending on the grain size and task. Typical feed sizes are 20 to 100 mm.

The proposed underground installation will be a combination of crusher and depending on the results of the future testwork, two optical sorters in containerized version. Since the installation of these machines is getting more and more widely used in open pits and underground mines to optimize operating expenses, sorting technology is becoming standard for minerals and ores.

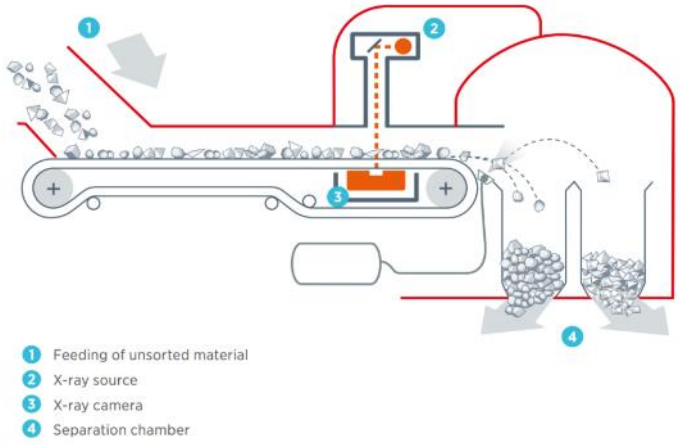
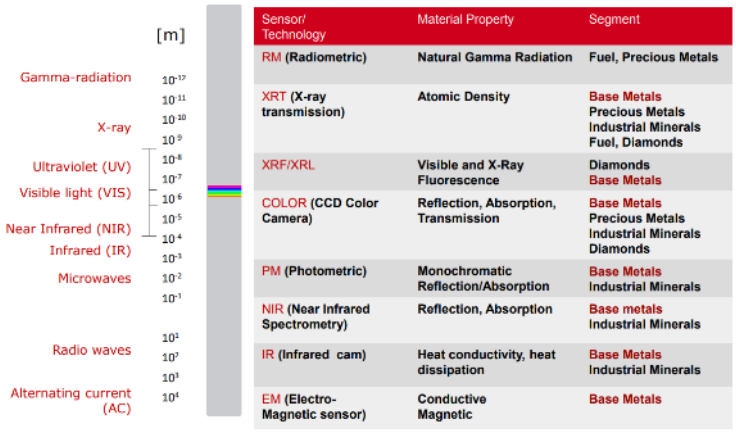


Figure 63: Drawing of sorting principles on a belt sorter (Source: TOMRA flyer May 2017)



Sensor/ Technology	Material Property	Segment
RM (Radiometric)	Natural Gamma Radiation	Fuel, Precious Metals
XRT (X-ray transmission)	Atomic Density	Base Metals Precious Metals Industrial Minerals Fuel, Diamonds
XRF/XRL	Visible and X-Ray Fluorescence	Diamonds Base Metals
COLOR (CCD Color Camera)	Reflection, Absorption, Transmission	Base Metals Precious Metals Industrial Minerals Diamonds
PM (Photometric)	Monochromatic Reflection/Absorption	Base Metals Industrial Minerals
NIR (Near Infrared Spectrometry)	Reflection, Absorption	Base metals Industrial Minerals
IR (Infrared cam)	Heat conductivity, heat dissipation	Base Metals Industrial Minerals
EM (Electro-Magnetic sensor)	Conductive Magnetic	Base Metals

Figure 64: Principle and available sensing for sorting (Source: TOMRA flyer May 2017)

Figure 65: Example of installed machine with capacity (Source: TOMRA flyer May 2017)

OPEN PIT MINE DESIGN

Based on the optimised ultimate pit envelope, a standard open pit mine will be constructed for all Nama Creek pegmatites Figure 66. Starting with a box cut, the pit will be developed successively in vertical and lateral direction (pushbacks) maintaining an overall pit slope angle of 60°.

Ore and waste material will be removed in successive layers (benches), which are accessed via ramps. Several benches may be in operation simultaneously at levels of the pit. Bench geometry and ramp design are usually based on rock mechanical properties of ore and waste rock, and the dimensions of the mining equipment.

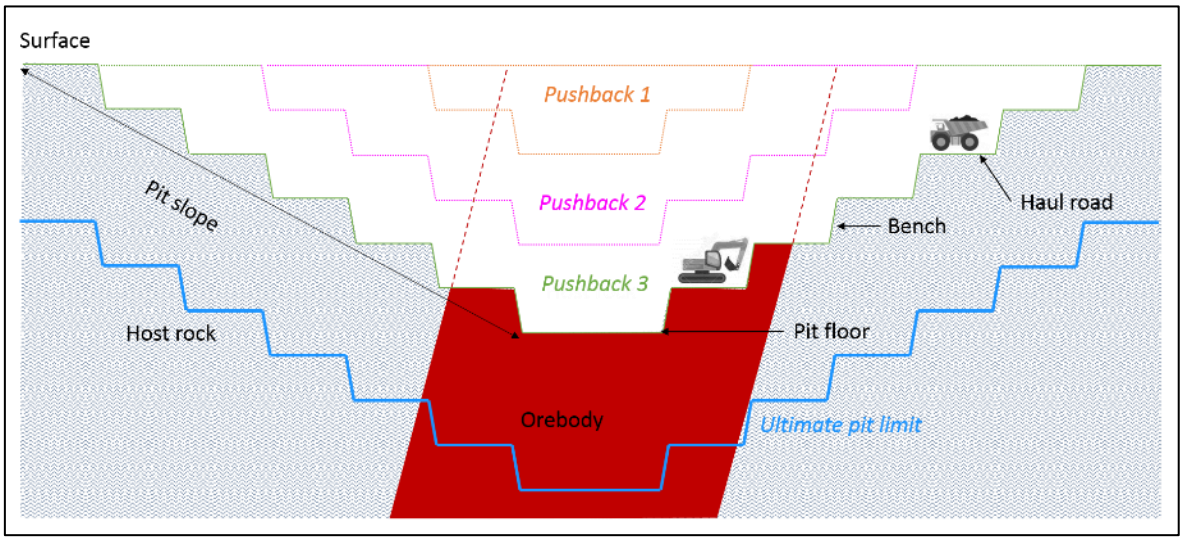


Figure 66: Conceptual open pit mine design.

The final pushback design is based on annual production requirements and serves as a basis for the production scheduling optimisation.

UNDERGROUND MINE DESIGN

Access to the lower pegmatite sections will be gained through decline portals sited within the existing open pit. With the exception of the MZN orebody, for which the ramp access will have to be located outside the pit as underground development and open pit production are scheduled to start simultaneously.

A safety zone (crown pillar) of 11 m will have to be left in place to guarantee surface stability and operational safety of all underground workings. In addition, a ventilation raise to ensure sufficient airflow throughout the mine will be required.

Orebody access will be gained through cross-cut drifts intersecting the pegmatites, which will be mined in 20 m slices. All mining stopes will be developed along strike of the orebody following the shape of the pegmatite dykes (maximum width 10 m). All stopes will be extracted according to a stoping sequence, which

depends on the number of unmined or backfilled stopes to be left in place to conform to pillar size requirements, and is subject to detailed planning and scheduling.

Final design and dimensions of all access routes will depend on the selected operating equipment and required installations for the mining process. Additional infrastructure required for the operation includes sumps, refuge stations, passing bays, battery rooms, crusher chambers, explosives magazines, ventilation facilities, workshops and personnel facilities.

All developments and infrastructure will be constructed at the footwall of the pegmatite bodies, which is a common procedure in underground mining to eliminate ground instabilities due to the mining activity.

The conceptual mine layout for the MZN pegmatite area is illustrated in Figure 67 and Figure 68.

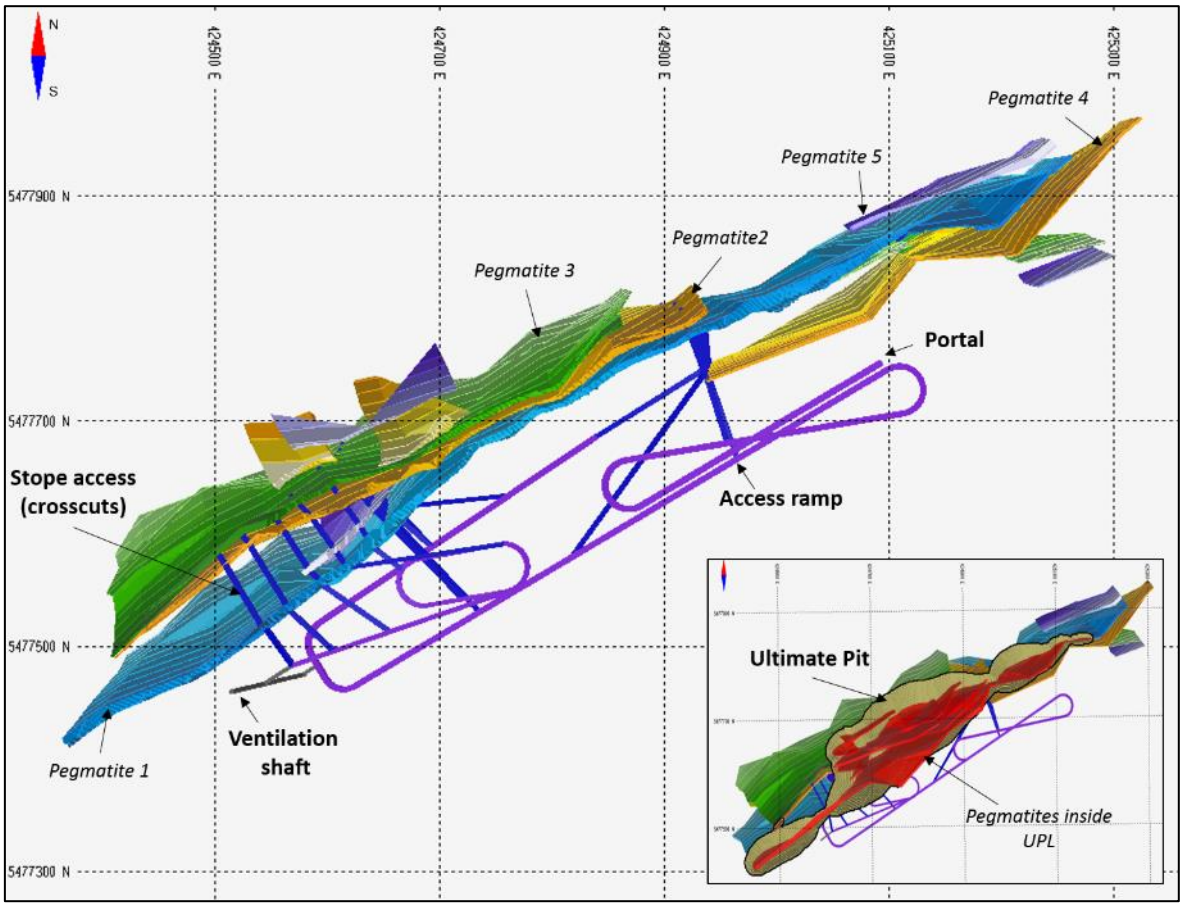


Figure 67: Conceptual underground mine layout for MZN (plan view).

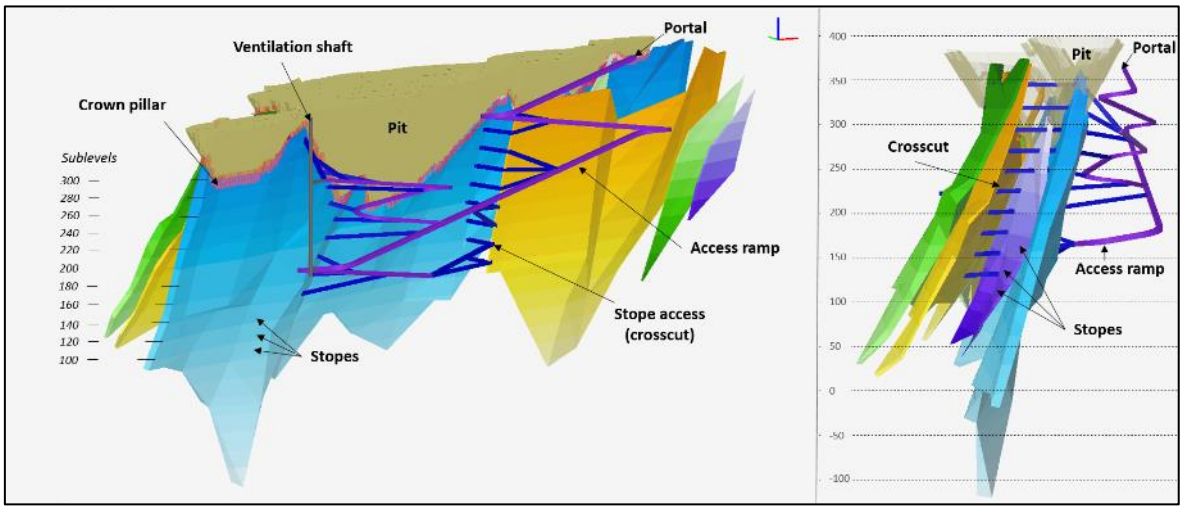


Figure 68: Conceptual underground mine layout for MZN (3D view).

EXPECTED PRODUCTION RATES AND LIFE OF MINE

MINE SCHEDULE

The preliminary mining sequence foresees ore production to commence in Main Zone/ North starting with open pit mining followed by underground mining. The other pegmatite areas will then be mined accordingly in the following order: Main zone/ South West (MZSW), Harricana (HAR), Line 60 (L60) and Conway (CON).

MINE SCHEDULE OPEN PIT MINING

Open pit mining operation will last for roughly 6 years. The overburden removal and ore extraction will be outsourced to a contractor. The main focus will be on the Main Zone/ North and the Line 60 with the highest mineral potential. Starting in the Main Zone just has the advantage of the biggest resources and the easiest access for the operation start.

Production Schedule Open Pit		Unit	Year	1	2	3	4	5	6	Total
Main Zone N	Waste Removal	Mio t		4.4	2.7					7.1
	Ore Extraction	Mio t		0.5	0.3					0.8
	Stripping Ratio	t:t		8.7	8.7					8.7
	Ore Grade	%		0.87	0.87					0.87
Main Zone SW	Waste Removal	Mio t			0.7					0.7
	Ore Extraction	Mio t			0.1					0.1
	Stripping Ratio	t:t			7.3					7.3
	Ore Grade	%			0.87					0.87
Harricana	Waste Removal	Mio t			1.4					1.4
	Ore Extraction	Mio t			0.2					0.2
	Stripping Ratio	t:t			6.1					6.1
	Ore Grade	%			0.91					0.91
Line 60	Waste Removal	Mio t				2.2	2.2	0.1		4.4
	Ore Extraction	Mio t				0.5	0.5	0.0		1.0
	Stripping Ratio	t:t				4.3	4.3	4.3		4.3
	Ore Grade	%				0.89	0.89	0.89		0.89
Conway	Waste Removal	Mio t						2.5	0.2	2.7
	Ore Extraction	Mio t						0.5	0.0	0.5
	Stripping Ratio	t:t						5.1	5.1	5.1
	Ore Grade	%						0.95	0.95	0.95
Total	Waste Removal	Mio t		4.4	4.7	2.2	2.2	2.5	0.2	16.2
	Ore Extraction	Mio t		0.5	0.6	0.5	0.5	0.5	0.0	2.7
	Stripping Ratio	t:t		8.7	7.6	4.3	4.3	5.1	5.1	6.1
	Ore Grade	%		0.87	0.89	0.89	0.89	0.95	0.95	0.90

Table 56: Open pit mine schedule based on Measured/Indicated Resources

MINE SCHEDULE UNDERGROUND MINING

Underground mining operation starts just one year after the start of the open pit operation. During that first year the access to the mine from the surface and the main levels will be prepared. This will be repeated in the upcoming years when accessing a new deposit with the difference of an access from the pit bottom of the mined out open pits.

Production Schedule Underground		Unit	Year	1	2	3	4	5	6	7	8	9	10	11	Total
Based on Measured and Indicated Resources															
Main Zone N	Ore Extraction	Mio t			0.2	0.5	0.5	0.5	1.0	0.4					3.1
	Ore Grade	%			0.87	0.87	0.87	0.87	0.87	0.87					0.87
Main Zone SW	Ore Extraction	Mio t								0.6	0.1				0.7
	Ore Grade	%								0.90	0.90				0.9
Harricana	Ore Extraction	Mio t									0.4				0.4
	Ore Grade	%									0.80				0.80
Line 60	Ore Extraction	Mio t									0.1				0.1
	Ore Grade	%									0.77				0.77
Conway	Ore Extraction	Mio t									0.3	0.4			0.7
	Ore Grade	%									0.80	0.80			0.80
Based on Inferred Resources															
Main Zone N/HAR	Ore Extraction	Mio t										0.6	0.8	0.5	1.9
	Ore Grade	%										0.89	0.89	0.89	0.89
Total	Ore Extraction	Mio t			0.2	0.5	0.5	0.5	1.0	1.0	1.0	1.0	0.8	0.5	6.9
	Ore Grade	%			0.87	0.87	0.87	0.87	0.87	0.89	0.81	0.86	0.89	0.89	0.87

Table 57: Underground mine schedule based on Measured/Indicated and Inferred Resources

MINE SCHEDULE OVER LOM

Table 58 shows the production schedule over the Life of Mine (LoM). Production starts in open pit operation with 0.5 Mtpy for about 5 years and then fading out in year 6. In parallel in year 2 underground operation starts with a ramp up period of roughly 4 years.

The total production was initially planned with 1 Mtpy in year 3 with a ramp up of 2 years and a total LoM of 11 years.

Production Schedule		Unit	Year	1	2	3	4	5	6	7	8	9	10	11	Total
Open Pit	Waste Removal	Mio t		4.4	4.7	2.2	2.2	2.5	0.2						16.2
	Ore Extraction	Mio t		0.5	0.6	0.5	0.5	0.5	0.0						2.7
	Stripping Ratio	t:t		8.7	7.6	4.3	4.3	5.1	5.1						6.06
	Ore Grade	%		0.87	0.89	0.89	0.89	0.95	0.95						0.90
Underground	Ore Extraction	Mio t			0.2	0.5	0.5	0.5	1.0	1.0	1.0	1.0	0.8	0.5	6.9
	Ore Grade	%			0.87	0.87	0.87	0.87	0.87	0.89	0.81	0.86	0.89	0.89	0.87
Total	Ore Extraction	Mio CAD		0.5	0.8	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.8	0.5	9.6
	Ore Grade	%		0.87	0.88	0.88	0.88	0.91	0.87	0.89	0.81	0.86	0.89	0.89	0.87

Table 58: LoM Schedule

OPERATING TIME MINE

The preliminary planning for the mine operation, open pit and underground was based on 2 shift per day and 335 days per year scenario (see Table 59).

Working Time Mine	Unit	Value
Number of shifts per day	sh/d	2
Duration of shifts	h/sh	12
Scheduled non-productive time	h/sh	1.5
Net operating time per day	h/d	21
Working days per year	d/y	335
Net operating time per year	h/y	7,035

Table 59: Mine operating time

MINING FLEET

OPEN PIT EQUIPMENT

Based on the LoM plan the following major equipment has been identified to execute the open pit operation. The operation will be outsourced to a contractor. Table 60 outlines the maximum yearly requirements for the 6 years surface extraction.

Equipment Open Pit	Size/Type	Quantity
Main Equipment		
Shovel	4 m³	2
Drill Rig (small)		1
Blasting Vehicle		1
Truck	40 t	4
Auxiliary Equipment		
Dozer	D8 or similar	1
Grader	140H or similar	1
Flood Light Plant		10
Water Truck	40m³	1
Field Service Truck		1
Pick-up (*)		2

Table 60: Open pit equipment requirements

UNDERGROUND EQUIPMENT

Table 61 shows the maximum requirements for underground equipment during LoM. It is assumed that the owner will execute the underground development and extraction. In this regards the study assumes the equipment will be required from year 1 on to develop the access and the main levels of the MZN underground operation.

Equipment Underground	Size/Type	Quantity
Development drill rig	Twin boom/3.8 m net	1
Bolting rig	1.2 m bolts	1
Stope drill	20 to 35 m net	1
Explosives charger		1
Concrete mixer and transporter		1
Concrete spraying equipment		1
Scaler		1
LHD	3 to 4 m^3	2
Truck	35 to 40 t	3
Service and fuel truck		1
Jeeps, Transporters		3

Table 61: Underground equipment requirements

RECOVERY METHODS

INTRODUCTION

Section Mineral Processing and Metallurgical Testing of this report provides a summary of the metallurgical test work to date. One locked-cycle test on the composite head sample, heavy liquid tests and several flotation tests were performed at the SGS Canada Inc. Testwork Facility in Ontario.

The results of which were used to derive the preliminary Process Flow Sheet of the facility. The Process design provides design parameters for the production and storage of a processing facility on site. Provision for the storage of all process inputs as well as concentrate output as necessitated to operate on a year-round basis.

Metallurgical testing has shown recovery to be fairly constant at around 80% with a technical concentrate containing a grade of 6.2% Li_2O. The analytical assay of the composite head sample are Li_2O 1.49 Li_2O (0.69 % Li).

General processing plant descriptions are provided in this section, which were based primarily testwork. This resulted in the development of a viable flowsheet for the production of technical Spodumene concentrate and served as the basis for both the mill design model and a first CAPEX estimate.

PRELIMINARY FLOWSHEET

The metallurgical tests necessary for this report were carried out on a three outcrop and one drill core samples from the Georgia Lake deposit. According to the test results in (1), the suggested plant flow sheet is subdivided into the following sections:

- 3 step crushing
- Gravity separation
- Milling
- Flotation
- Dewatering section of different products

The plant is designed for a feed capacity of 150 t/h or 1,000,000 tpy. For financial calculation, it is recommended to use an adaptation of the SGS test results as follows:

- Li_2O content in feed of approximately 0.90 Li_2O (0.42 % Li)
- Li_2O content in concentrate 6.2 % Li_2O
- Li_2O overall recovery 78%

Figure 69 shows a simplified flowsheet of processing plant.

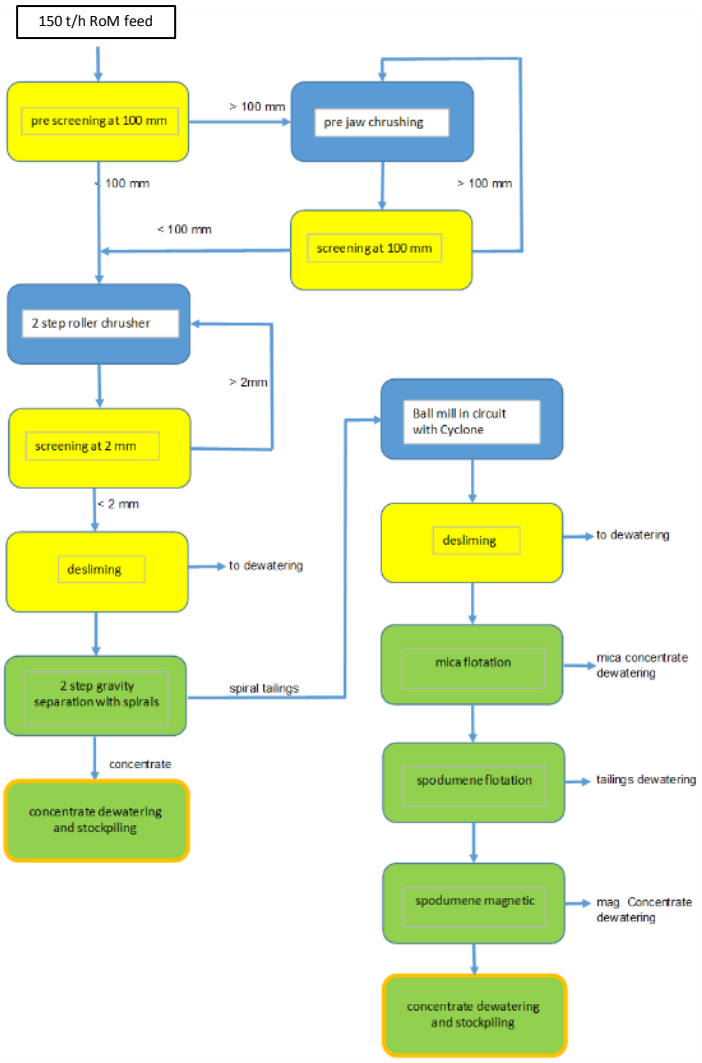


Figure 69: Recommended flow sheet for plant layout (1)

CRUSHING CIRCUIT

The ore from the feed hopper is discharged to the #100 mm screen. Pre-screened feed material will be crushed with a jaw crusher down to <100 mm. This crusher works in a circuit with a second #100 mm dry screen. The <100 mm material will be fed to the middle and fine crushing steps. There, the material will be crushed to < 2mm by roller crushers. The 2nd roller crushers is a HPGR which operated in closed circuit with the # 2mm wet screen.

GRAVITY SEPARATION SECTION

Instead of heavy liquid separation used in test work, DMT suggests gravity separation with spirals as mentioned above. The ore minus 2 mm will be fed to the pump sump box and transferred via a slurry to hydro-

cyclones for desliming. The overflow of the cyclones are directed to the thickener. Hydocyclones will deslime the feed to spirals approx. at 63 µm. Overflow should be free of fine Spodumene.

The underflow of the cyclones will be fed to the spirals for gravity separation. Middling of first step will be fed to second spiral step. Spiral section produce a final coarse Spodumene concentrate and spiral tailings. The final coarse Spodumene concentrate will be dewatered at dewatering screen and stockpiled afterwards.

MILLING SECTION

Tailings of spirals contain fine and not liberated Spodumene.

Therefore tailings from the gravity separation will be transferred to the grinding section and fed via a slurry pump to the cyclones in front of ball mill increases the solid concentration before milling. The overflow of the cyclones are directed to the thickener. Cyclone overflow is free of Spodumene.

The underflow of the cyclones will be directed to the ball mill. The discharges of the ball mill will be transferred via a slurry to a #0.3 mm screens. The ball mill operate in a closed circuit with this screens, the oversize goes back to the mill. The minus 0.3 mm fraction is the feed for flotation.

FLOTATION

MICA FLOTATION

The fraction minus 0.3 mm will be fed to the pump sump and pumped to the cyclones for desliming. The overflow of the cyclones are directed to the thickener.

The underflow of the cyclones are directed to a conditioner tank where the slurry is mixed with collectors.

The conditioned slurry flows to the mica rougher flotation. The rougher flotation follow the mica scavenger flotation. The concentrates of the rougher and scavenger flotation will be directed to a one-stages cleaner flotation. While the mica final concentrate is pumped to the mica concentrate thickener. The tailings of the scavenger of the mica flotation and the tailing from the mica cleaner flotation will be directed to the Spodumene flotation.

SPODUMENE FLOTATION

The first stage of the Spodumene flotation is the high density scrubbing. This scrubbing clean and activate the particles surface. First part of flotation reagents added on this point. High density scrubbing follow by desliming with hydrocyclones. The overflow of the desliming cyclones are directed to the thickener. This overflow is free of floatable Spodumene.

The conditioned slurry flows to the spodumene rougher flotation. The tailings of the Spodumene scavenger flotation directed to vacuum cell filter. The concentrates of rougher and scavenger Spodumene flotation will be directed to a three-stages cleaner flotation. Tailings of 1st, 2nd and 3rd cleaner are directed back to the high density scrubbing or vacuum cell filter. The concentrate of the 3rd cleaner are directed to a magnetic separator for the separation of mag. concentrate and the final Spodumene concentrate.

DEWATERING SECTION

All fine material coming from cyclone overflows are free of floatable Spodumene and fed to separate thickeners. Thickener will increase the solid concentration to approx. 350 … 400 g/l. This is enough to feed the followed filter press. Between thickener and filter press a slurry storage tank will be installed.

Overflow of the thickener will be used as process water in circuit. That minimized the water consumption of the plant. Only additional water to balance the water lost with final concentrates and tailings is necessary.

For Spodumene and mica concentrates vacuum cell filters foreseen. Filtrate of vacuum filters goes to process water tank

PROJECT INFRASTRUCTURE

OVERVIEW

As a comprehensive Greenfield project, the Project will require the development of supporting infrastructure.

These items include:

- A process plant or concentrator that will include crushing, grinding, flotation, regrinding, concentrate filtration, concentrate thickener, tailings thickener facilities and assay laboratory
- A warehouse, maintenance shop, administration offices, and supporting infrastructure
- A geomembrane-lined TDF, and structural earth dams, initial waste rock dumps
- A network of access and on-site roads
- A fresh water supply and distribution system
- Power supply and distribution, including a power transmission line, a substation at the plant site, and power distribution lines
- Other infrastructure including truck shop

The concentrator will be located at the mine site. Concentrate will be shipped either to the clients or to a hydrometallurgical plant located in the vicinity of the operation.

Figure 70 shows the location of the Property in relation to principal supporting infrastructure. As there is no rail access to the mine/concentrator site, delivery of reagents to and shipment of concentrates from the site will be by truck.

SITE LOCATION AND SUPPLY

LOCATION

The property is readily accessible from Beardmore vollage by traveling 16 km. The village of Beardmore is the closest community of the Georgia Lake Property. Beardmore is part of Greenstone, an amalgamated town encompassing Nakina, Geraldton, Longlac, Beardmore, Caramat, Jellicoe, Macdiarmid and Orient Bay. The population of Greenstone is 4,906 people (Statistics Canada, www.statcan.gc.ca) and the population of Beardmore is approximately 200 people

Nipigon is located 50 km south of the property. The population for Nipigon Township is 1,752 people in 2006.

The blue line shows the way from Thunder Bay into the area along the Highway Nr.11. The town Nipigon on the norther side of the Lake Superior and Lake Nipigon North West of the investigation area. In Figure 70 and Figure 71 the Highway 11 is running west of the claim in the east side of the Lake Nipigon to the north.

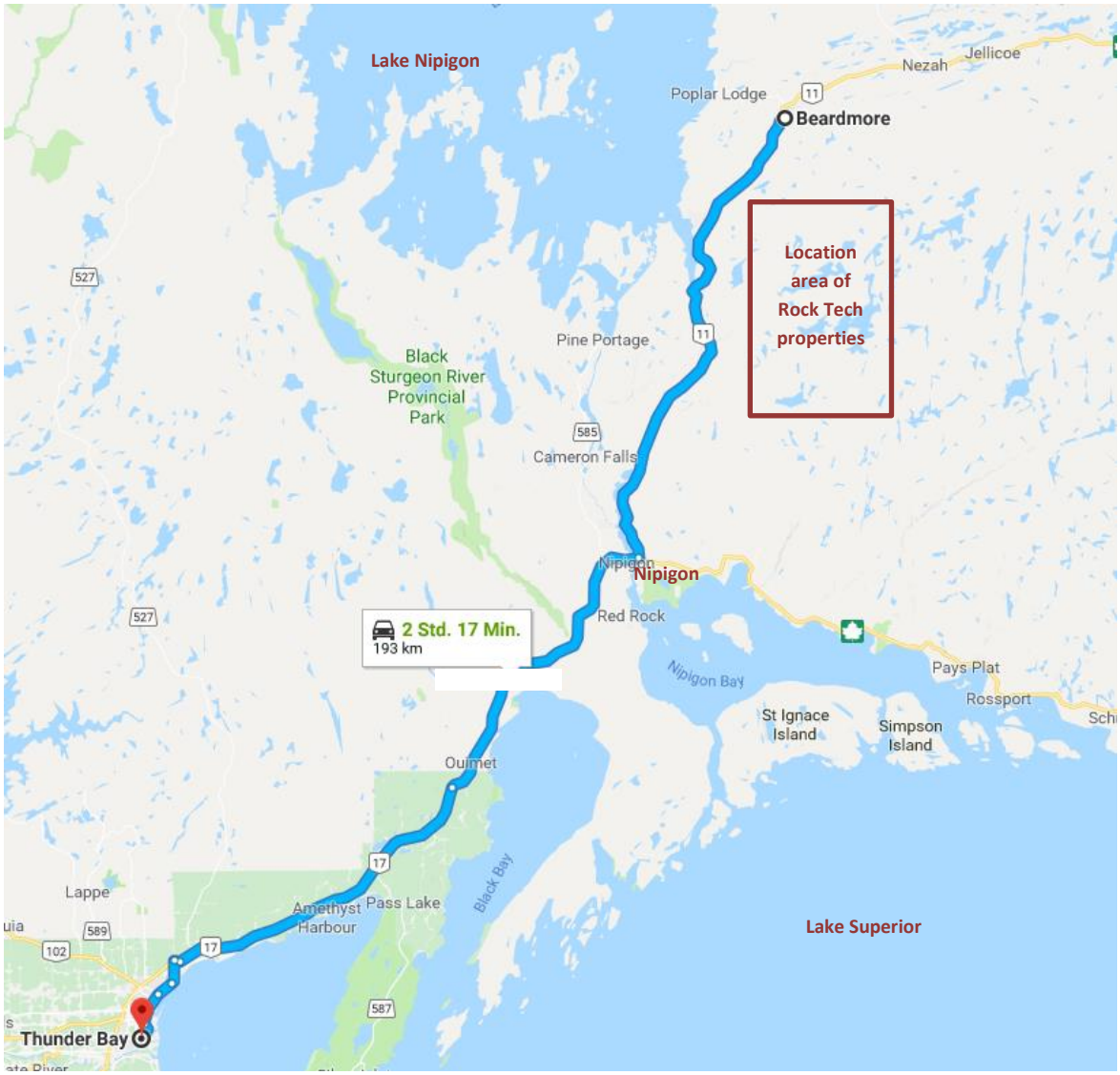


Figure 70. Location map of the Rock Tech properties and highway 11 as access road from the town of Thunder Bay (Source: Google Maps).

ROAD ACCESS

OVERVIEW

The area is connected by a gravel road from the Highway N11 to the Main Zone. The other deposits can be at the moment also reached by a forestry road from the highway.

A direct connection to the Southern deposits passing the little river doesn't not exist at the moment and has to be constructed.

Existing bridges either in the field or on Highway N11 are suitable for passing with trucks.

Figure 71: N 11 between Main Zone and Beardmore

Figure 72: Dam for bridge construction 1 km from Main Zone

Figure 73: Bridge close to the Main Zone with up to 60 t payload

Figure 74: River separating Northern and Southern deposits (about 8m width)

Figure 75: Forestry road for accessing Southern deposits

ACCESS ROAD

As mentioned above, there exists a road access to the Project area as gravel road. However, this road of about 8km needs to be upgraded.

The design of the road needs to be studied in the further development of the project, alternatives might be also available. A separate environmental impact assessment might be required for the road upgrade by the authorities before approval of the Project EIA.

However, capital costs for this road are included in the Financial Model.

INTERNAL ROADS

Building of new roads and upgrading of existing roads have been envisaged for the internal traffic between the premises and facilities of the Project.

The internal roads should have a width of 6 m to allow the transit of standard trucks and light vehicles. The roads will receive a layer of hard, compacted base coarse.

For the internal roads a total of 2.0 to 3.0 km of new roads have been considered.

POWER

There is a power line that runs along the TransCanada highway #11 about 10 km from the property. There are three hydroelectric stations on the Nipigon River, all of which are con-trolled remotely by the headquarters in Thunder Bay: Alexander Station with 68 MW output (17 km north of the town of Nipigon), Cameron Falls with 87 MW output (17 km north of the town of Nipigon) and Pine Portage with 142 MW (39 km north of the town of Nipigon)

Approximately 6 MW of power will be required for the mine and concentrator. Power will be supplied from the existing nearby 115 kV power line. A stepdown transformer will be installed at the connection point to the 115 kV line and approximately 10 km of transmission line will be installed to bring the power to the mine site. An additional stepdown transformer will be installed at the site to supply power to the local electrical distribution system. An emergency back-up generator will also be provided at the site fueled either by diesel or propane.

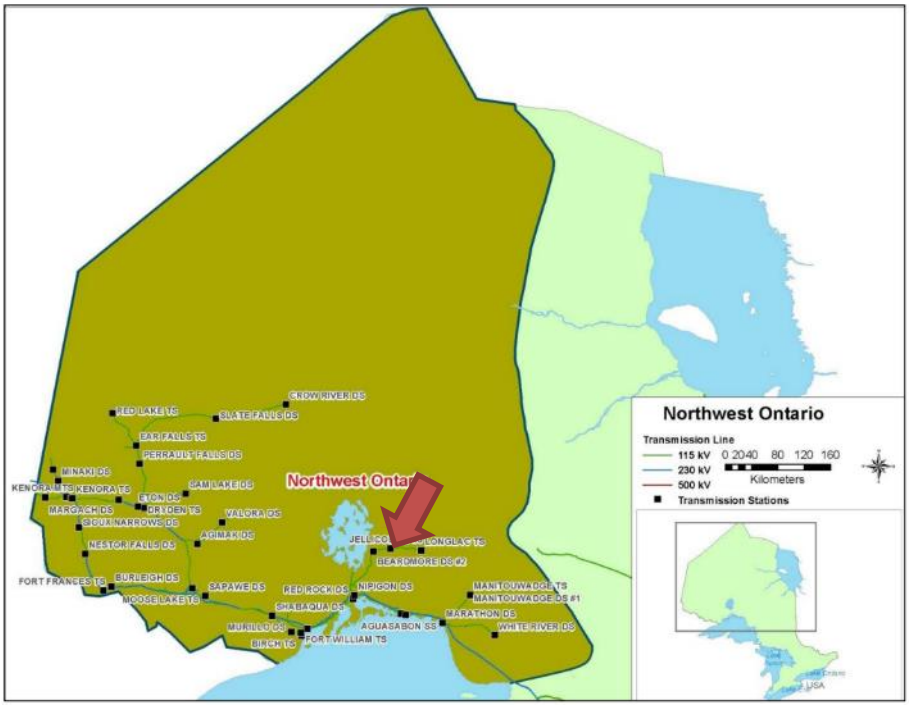


Figure 76: Map of Northwest Ontario Region, (Source: Northwest Ontario Regional Infrastructure Plan, 2017)

Figure 77: Map of Greenstone-Marathon Sub Region, (Source: IESO)

FRESH WATER

Fresh water and fire water for the site will be provided from the nearby river. An intake line will be installed to a sufficient depth in the river to be below the ice level. Water treatment facilities will be provided as required to supply potable water to the site.

SEWAGE

Sanitary waste water treatment will be provided at the site using appropriately sized parallel septic tanks and field bed. Waste water from the treatment unit will be discharged to the environment. Arrangements will be made with a local contractor for the periodic pumping of the septic tanks for removal and disposal of the sludge as required.

FUEL STORAGE

Diesel fuel storage facilities will be provided to supply the mine equipment and smaller site vehicles. Two double-wall diesel tanks will be provided on a concrete foundation.

A propane tank farm will also be installed to accommodate the site heating and back-up power generation.

COMMUNICATIONS

A telecommunications system will be installed at the site to provide telephone service and internet access, and to support the site security and fire detection systems. A mobile radio system will be installed to provide local communication to all parts of the mine and site facilities.

A microwave link will be installed to provide access to an internet service provider. A backup system will be provided using a cellular modem. Distribution will be provided by a fibre-optics system in the concentrator and related facilities and a wireless system for the mine site.

SURFACE FACILITIES

INTRODUCTION

The main compound encompasses the facilities located initially planned South of the MSN deposit. A general location map is provided in Figure 78. The location of the single facilities is depending on the further exploration program and the inertization drilling in the referred areas. Infrastructure already exists in terms of accessing gravel roads for the Project.

The facilities will be located close to the plant site with sufficient distance to the river and in a not swampy area. The planned activities are described in the following for the individual elements of the main compound.

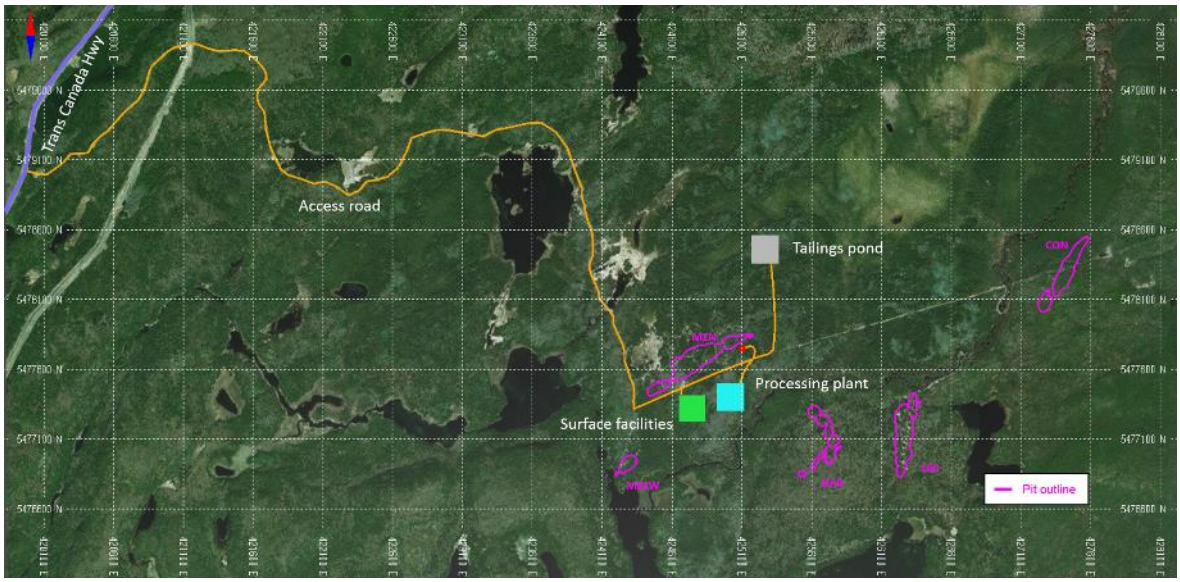


Figure 78: Preliminary location of the facilities together with pit outlines

OFFICE BULDING

There is a necessity to establish a small building with administration offices together with engineering offices for mine and processing plant as well as environmental, safety and infrastructure departments.

STAFF FACILITIES

Canteens and locker rooms with their respective washing facilities and toilets have to be considered in one building.

The building will have a dining room on the first floor and on the second floor a change room with lockers for each worker, as well as the required washing facilities and toilets.

CAMP

No camp facilities are envisioned for this project. It is anticipated that the work force will live in the surrounding area. Buses might be provided to transport workers between to the mine site.

LABORATORY

There is a need of a laboratory facility either in a separate building or as part of the office building or processing facility.

WORKSHOPS

The existing mechanical and electrical workshops will also be utilized for the new Project. For light and day to day maintenance the design includes one Maintenance Workshop adjacent to the Magnetic Plant. This building will have 3 different areas:

- Mechanical workshop: Steel structure equipped with a gantry crane and some bay areas for mechanical assembly, welding, lubricants, compression, lathe and drill. The clearance height will allow for truck access for delivery and receipt of components and materials.
- Electrical and instrumentation workshop: Steel structure with a gantry crane for handling of engines and boards.

Attached two story building with area for storage of tools and components on the first floor and on the second floor area offices for maintenance personnel including Superintend office, planning, technical library and meeting room.

WAREHOUSES, SOLIDS STORAGE

The warehouse and storage area should meet the requirements of storing spare parts and materials needed for the Project. The design should consider the following:

- Roofed area for spare parts and materials which require greater protection
- Fenced area for major components which do not require enclosed storage like plates, rods, profiles, etc.

To be discussed is the construction or the sharing of the warehouse for materials and spare parts for a contractor who is in charge of the mining operation.

EXPLOSIVES STORAGE

In general, ammonia nitrate is needed for explosives and will be stored in a magazine. The magazine will be located in the vicinity of the operation. The building shall have a capacity to store explosives for one month.

Blasting materials shall be stored in the blasting material warehouse.

TAILINGS DAM FACILITY (TDF)

INTRODUCTION

A conventional impoundment is a surface retaining structure designed to store both tailings and mine water, with the aim of reclaiming the water for use in the processing plant as required. There are principally two types of surface impoundments, a water retention type dam, and a raised embankment that has many configurations (Vick 1990). The main difference is that a water retention dam is constructed to its full height before any discharge to the impoundment (EPA 1994). Raised embankment dams are built higher as the requirement to store more tailings and process/storm water becomes apparent.

WATER RETENTION

Water retention dams for tailings differ only slightly from conventional water storage structures. The principal designs and construction techniques are the same, the main difference being that tailings storage embankments have a steeper upstream slope as they do not need to be engineered to cope with the rapid drawdown that is experienced in a conventional water retaining structure (Vick 1990).

Water retention tailings dams are mainly used for mineral operations that plan to store high volumes of water. Water storage may be required to keep a processing plant operational during the dry season or where surface water inundation can occur especially if the impoundment is in a catchment area.

RAISED EMBANKMENT DESIGNS

The raised embankment design is the most common construction technique used in tailings storage facilities. The three principal designs are

- downstream,
- upstream and
- centreline structures,

which designate the direction in which the embankment crest moves in relation to the starter dyke at the base of the embankment wall. Modified centreline is another method rarely used which is a combination between upstream and centreline construction.

The embankment is raised at certain time intervals to increase the available volume for the storage of tailings and water, thus they have a lower initial capital cost than retention dams because fill material and placement costs are phased over the life of the impoundment. The choices available for construction material are increased as smaller quantities are needed at any one time. For example, retention dams generally use natural soil whereas raised embankments can use natural soil, tailings, and waste rock in any combination. The most common materials used for embankment raises are waste mine rock, natural borrow soils, underground roadway development material, cycloned tailings (coarse fraction) and hydraulically deposited tailings. With the developments of high capacity earthmoving equipment in the 1940's raises can be compacted in a similar manner to that of water retention dam construction techniques. Raised embankment construction is almost always mechanised to gain the level of compaction required to increase the safety and lower the risk of instability of the storage facility.

The downstream design was developed to reduce the risks associated with the upstream design, particularly when subjected to dynamic loading as a result of earthquake shaking. The installation of impervious cores and drainage zones can also allow the impoundment to hold a substantial volume of water directly against the upstream face of the embankment without jeopardising stability.

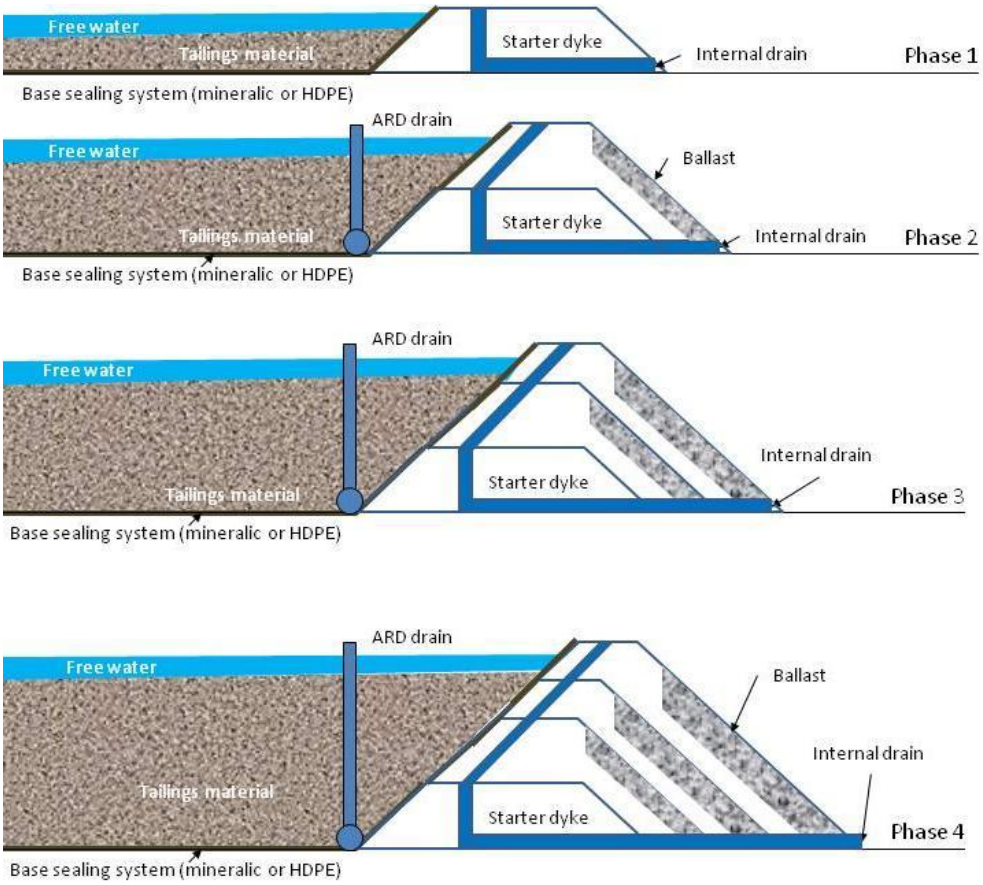


Figure 79: Schematic Drawing of a TDF

The downstream embankment design starts with an impervious starter dyke unlike the upstream design that has a pervious starter dyke. The tailings are at first deposited behind the dyke and as the embankment is raised the new wall is constructed and supported on top of the downstream slope of the previous section. This shifts the centreline of the top of the dam downstream as the embankment stages are progressively raised. An advantage to the downstream design is that the raised sections can be designed to be of variable porosity to tackle any problems with the phreatic surface of the embankment.

This can be particularly useful where a processing plant has made changes to increase efficiency and as a result alter the tailings characteristics. This may result in pumping more water to the tailings facility or alter the drainage characteristics of the newly deposited tailings.

Ballast (e.g. inert waste rock) can be applied to account for additional stability according to seismically induced loading of the dyke.

A sealing system may or not be applied, depending on the outcome of the environmental risk assessment of the TDF. It is, however, likely that storage of tailings derived from sulphidic ore will generate acidic rock drainage (ARD), which cannot be otherwise compounded.

Additionally an ARD system may be included in the design to monitor, collect and dispose the ARD in an orderly way.

TDF SPECIFICATION

The size and location of the TDF have been initially estimated. DMT prefers a location north of the MSN deposit in a flat not swampy area. However, before planning and designing the extension of one of the existing ore bodies or the existence of a further ore body has to be excluded.

Secondly further tests on tailings and the usage as backfill have to be executed. The usage of the tailings as backfill in the underground operation defines the size of the future tailings dam facility. Furthermore, the possibility of placement of tailings (in a clay layer) within the in-pit dumping should be approved.

The initial tailings dam facility has been estimated with 300m length and width and a height of 25m assuming that 80% of the open rooms underground can be backfilled.

Figure 80 shows the pit outline of the open pit operations and the location of the processing, surface facilities and TDF together with the areas declared as swamp (grey) and hard underground (green).

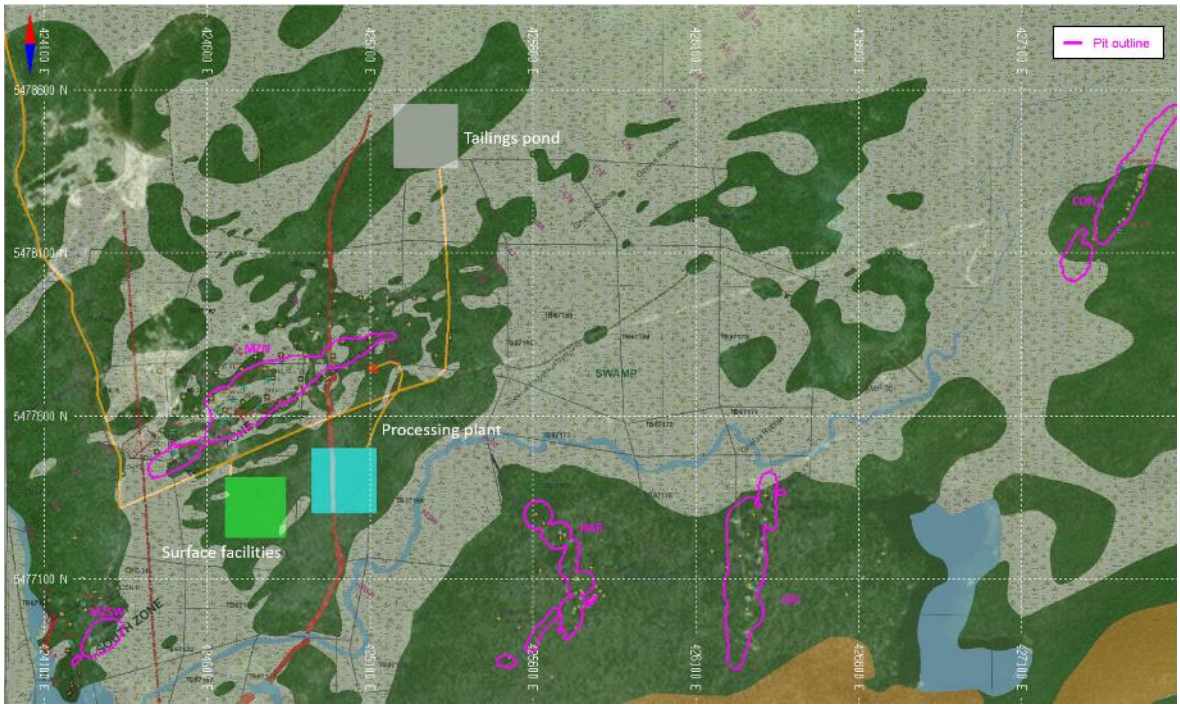


Figure 80: Preliminary location of the Facilities and the TDF in terms of swamp and non swamp areas

LITHIUM MARKET OVERVIEW

In 2018, USGS (United States Geological Survey) reported global lithium reserves to be 16 Mt Li (85 Mt LCE). The USGS also reported lithium resources at 53 Mt Li (282 Mt LCE), with continental brine resources accounting for 60% of the total resources. The three most commonly sold finished products are lithium carbonate, lithium hydroxide, and lithium Spodumene concentrate. Transactions are negotiated between the producer (or agent/trader) and the consumer (i.e. battery industry). Lithium is not traded on any exchange.

Growth in consumption has been led by increased use of lithium by the rechargeable battery industry (mostly driven by electric vehicles industry), growing at 19.3%py between 2000 and 2017. The rechargeable battery sector accounted for 45% of lithium consumption in 2017. Lithium carbonate is the most widely consumed product, finding – next to rechargeable batteries – application in ceramics, glass, metallurgical powders, aluminium and other uses.

China is the largest consumer of lithium, mostly due to its dominance in the battery cells production, accounting for around 40% of total consumption in 2017 - followed by Japan and South Korea at 19% and 14% of the global market for lithium respectively. Both Europe and North America are mature markets for lithium with growth stagnating since the early 2010s, which might change with the upcoming setup of battery cell and battery component factories.

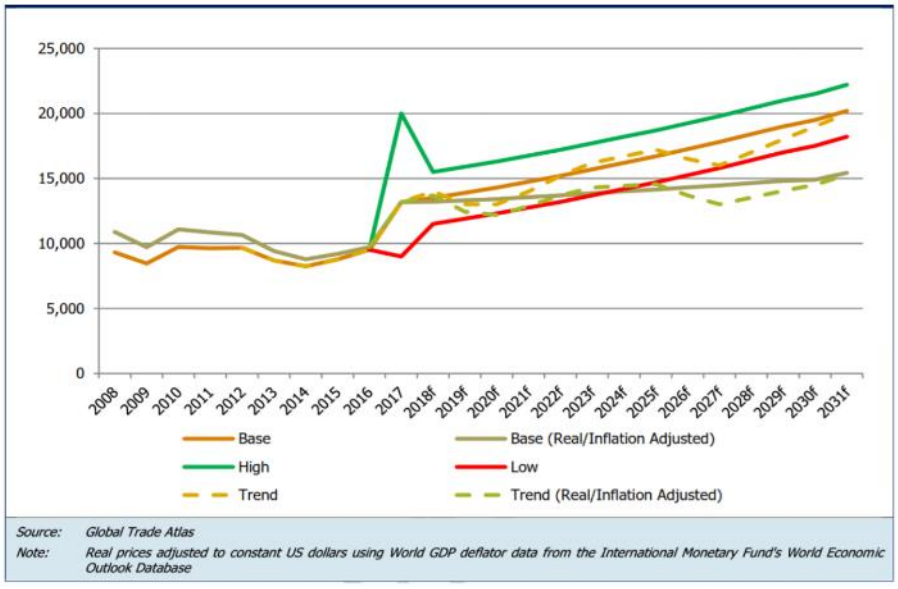


Figure 81: Price forecast for Lithium battery chemicals

There is a huge range of forecasts on supply, demand and pricing of battery chemicals products such as lithium carbonate and lithium hydroxide, by analysts such as Roskill, Benchmark Minerals, Bloomberg, Morgan Stanley, Goldman Sachs, Deutsche Bank, etc. These forecasts are relevant, as Spodumene mineral concentrates play a vital role in the supply of lithium chemicals to the wide variety of downstream industries. However, the consumption of lithium will continue to be driven by the rechargeable battery sector. In principle,

since 2000, growth in mine output has averaged 10%py. In 2017, the production of lithium totalled 360,256 t LCE. In the more optimistic case forecasts of analysts, this is forecasted to increase to 900 000 tpy by 2023 and > 1 Mtpy by 2025.

However, based on research of Benchmark Minerals, there are presently 45 lithium ion battery megafactories under construction around the world equating to >1,000 GWh in the coming years. Based on these growth figures of lithium consumers, the demand for lithium products will likely surpass all current analysts' expectations. Consequently, the price for battery-grade lithium hydroxide is forecasted to rise further to USD20,000/t in 2030, from lithium contract prices of around USD 14,000/t in 2018 - although there is expected to be a weakening in prices in the period 2019 - 2021. We don't expect this weakening to happen, due to the further growing and upcoming demand especially by Chinese battery producers.

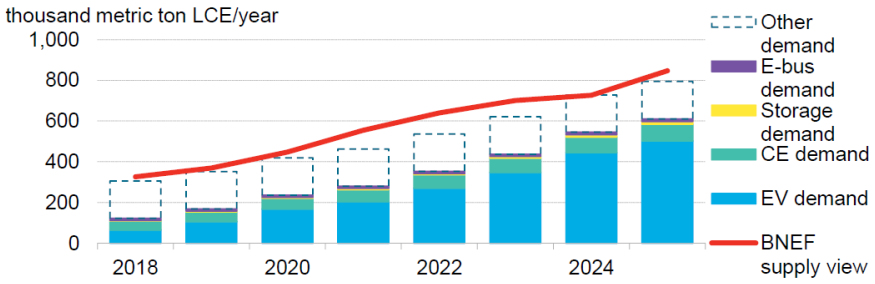
In summary, medium and long-term outlook for lithium consumption and lithium product prices remains very strong, with constantly increasing prices over the years. McKinsey projects an overall demand growth for lithium of 16-18% annually until 2030, creating pressure on supply. This would lead to a demand of nearly 2 Mt LCE in 2030.

RECENT MARKET DEVELOPMENTS

In the past months, fears of oversupply and of a lithium glut based on large nameplate capacity increases from miners dominated the market. Those fears are clearly overstated, despite miners announcing increases in their capacity. However, even if miners were to ramp up to their full stated capacity they would be likely to encounter delays beyond their control - processing challenges, legal problems, politics in "brine hosting" countries such as Chile and Argentina, and even weather permanently limit rapid expansions. Just recently, both Argentina and Chile have announced new export taxes or plans for new taxes and royalties that would increase production costs in those countries. Currently, Chile's congress is studying a proposal for an additional royalty payment for lithium miners operating in the country to bolster the development of the regions around their deposits, that suffer from water drain. In addition, the analysts of Bloomberg NEF argue that the established lithium miners are clear that they do not want supply to outstrip demand to the extent that it would risk lower lithium prices. Moreover, as shown above, significant volumes of additional capacity will be required by the mid-2020s to match demand growth later in the decade and into the 2030s.

We therefore expect that in the most optimistic supply case, the lithium market will only maintain a close balance between supply and demand out to 2025 (as shown in Figure 82), and likely an under-supply onwards.

However, Bloomberg expects that there will be a small surplus of lithium products up until 2020 as miners start to ramp up production. New supply shall come from junior miners in Australia opening or expanding Spodumene mines, and from Albemarle's and SQM's Salar de Atacama brine projects. This also includes supply from Argentina, Canada, China and Zimbabwe. Even if all those projects should get into production or increase production, Bloomberg calculates the expected surplus over this period to be around 20kmt LCE each year – so any delays to projects or missed production targets could tip the balance.



Source: Bloomberg NEF, Avicenne, Deutsche Bank. Note: Lithium demand from batteries will require lithium to be mined 12 months earlier. LCE refers to lithium carbonate equivalent

Figure 82: Mined Lithium supply and demand 2018 to 2025

LITHIUM SPODUMENE CONCENTRATE (LI_2O) PRODUCT PRICING

Independent product pricing forecasts have not been obtained as it is a Preliminary Economic Assessment level study. No sales contract or off-take agreements are established at this time.

Rock Tech plans to fast track the project into production and assumes stable and strong lithium concentrate prices. In short, Spodumene concentrate prices have been derived from an analysis of forecasts of leading market players including Roskill, Benchmark Minerals, investment banks and industry peers, as well as a current market review of demand and supply trends. In line with analysts' forecasts, we expect a constantly rising sales price during LOM with an assumed starting price of USD 800/t for a 6.2% Li_2O concentrate (our price assumptions is shown in Table 62).

The challenge with the analysis of the Spodumene concentrate market is that it is a new market and that there have only been few suppliers of Spodumene concentrate over the past years. Some recent examples include Galaxy Resources (and its Mt. Cattlin mine) and NeoMetals (and their Mt. Marion project), both of Australia, shipping lithium Spodumene mineral concentrates to China for processing to lithium chemicals. Before, Spodumene concentrates came from Talison's Greenbushes Mine, also in Australia. There is little historical information available regarding the selling price of their concentrates.

However, based on the analysts' demand expectations for lithium chemical products, we assume that also Spodumene concentrate prices are to remain robust and strong compared to current levels. We expect a overabundance of processing capacity available and coming up in China, and a growing abundance of processing capacity outside China (including in Canada (Nemaska's planned facilities in Quebec) as well as Australia (Tianqi Lithium's proposed new lithium hydroxide plant)). Likely, the profit margin is transferred from the converters in China, that process Spodumene concentrate into lithium chemicals, to the producers of the feedstock.

Analysts such as Stormcrow argue that if the lithium market behaves as other Chinese-dominated markets with excess processing capacity, the beneficiary of this overcapacity is the miner, as the processors compete to gain access to feedstock and market share. This is a pattern seen in basic commodities such as iron ore through to critical materials such as rare earths. The supply situation will be tightened with the current trend towards downstream integration by lithium producers (as outlined i.e. by Canaccord) that will lead to increased competition for Spodumene concentrate feedstock and therefore stable or higher prices. Due to the

recent developments in Chile and Argentina, the hard-rock mines in Canada and Australia are in the best position to deliver the needed lithium concentrate.

Production Year	USD per t 6.2% Li_2O conc
1	800
2	800
3	800
4	800
5	800
6	850
7	850
8	850
9	850
10	850
11	850

Table 62: Rock Tech Price assumptions over LoM

Looking at recent market prices for Spodumene concentrate, announcements of industry peers are used for evaluation. For example, Pilbara Minerals indicated in May 2018 that prices as high as USD 1,314 (delivered, VAT inclusive) were being paid for 6% Spodumene concentrate in China. Lithium producer and exporter Galaxy Resources recently commented on the market condition, saying it does not fear any supply surge that was suspected to come in the near future from two new Australian lithium mines, and said it received higher prices for its product over the past six months, despite the falls in some high-profile price indexes over the same period. Galaxy received USD 940 per dry metric tonne for its Spodumene concentrate product in the first half of 2018; that result was higher than the USD 868 per tonne received in the three months ended December 31 and the USD 783 per tonne received in the six months ended on December 31 last year.

In summary, the LoM price assumption with a starting price of USD 800/ton in year 1 for a 6.2 % Li_2O concentrate is far below current market price, in a market where demand is going to increase strongly.

BY-PRODUCTS

Rock Tech Lithium did not calculate to obtain any by-products.

ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

INTRODUCTION

Environmental Impact Assessment (EIA) process is initiated later in the economic feasibility process. While working on the Preliminary Economic Assessment (PEA) Rock Tech decided to start as early as possible with the EIA process to move to the next stage.

The environmental study that will be initiated will in essence support permitting requirements for a Lithium Mine and Mill in Ontario that include:

- Hydrology and water quality;
- Hydrogeology;
- Geochemistry;
- Terrestrial Biology;
- Aquatic Biology; and
- Archaeology.

The baseline studies will collect environmental information about the current site and areas surrounding the site that could potentially be affected by mining/milling activities. The baseline studies give an indication of conditions prior to any development. In addition, they can be used as a basis for the predictions of potential environmental impacts and appropriate mitigation can be incorporated into the project plan to minimize those impacts. Should the project plan or project footprint change, the baseline studies may have to be adjusted or expanded to incorporate the areas affected.

The work will be based on the historical baseline ecological and water balance study executed in 2010 and 2011 by Trow Associates Inc. Bramton, Ontario ("Trow") for Georgia Lake Property, Ontario.

HSTORIC WORK

Trow was retained by Rock Tech to conduct a Water Balance Study as part of a baseline study for the purpose of the permission process for future advanced exploration on the Georgia Lake Lithium Property at that time. The objective of the water balance study was to quantify existing groundwater and surface water flows and budgets for the Georgia Lake Lithium Property. An internal report was written by Trow to discuss the results of the study dated Feb. 18, 2011 (Trow, 2011a). This water balance study was based on available published data and no field work was carried out to collect any site specific information.

The Property is located in the Boreal eco-region of central Ontario. The area is mainly covered by wooded areas with deep rooted plants and trees. The closest Environment Canada weather station is in Geraldton, approximately 80 km northeast of the Property. Weather data from 2001 to 2011 indicates that the annual average precipitation is 769 mm, which includes 570 mm of rain and 241 mm of snow. The average annual temperature is 1.2 ºC with a daily maximum of 17.6 ºC in July and -17.6 ºC in January.

Each of Rock Tech's claim blocks contains bodies of water, wetlands and rivers. The soil on the Georgia Lake property is generally comprised of Podzol soils (mostly sandy soils) with rock outcrops, peat and grey wooded soils. Two major aquifer systems are found in the area:

- A sandy overburden aquifer system; and
- A fractured bedrock aquifer system

Overburden aquifers are found at depths up to 20 m below existing ground surface. It is expected that the overburden sandy aquifer is mostly under water table conditions.

The primary water supply aquifer system in the area is the fractured bedrock aquifer system. Approximately 80% of water bearing fractures are been encountered at depths less than 50 m below ground surface (mbgs). Up to 20% water bearing fractures are encountered within 10 m of the bedrock surface. The fractured aquifer system is expected to be under semi-confined to confined conditions.

Bedrock fractures less than 10 m from the bedrock surface are expected to be hydraulic connection with the overburden aquifer system. The overburden aquifer system appears to be connected to the surface water system. The regional bedrock aquifer system does not appear to contribute to the local groundwater flow system.

Of the total precipitation, approximately 55% to 60% is expected to be lost as evapo-transpiration. The evaporation from the surface of the Site's water bodies in the overall area is estimated to be less than 5% of the total precipitation. After evapo-transpiration, approximately 40% to 45% of surplus water is available for surface run-off and infiltration into local groundwater flow system.

An infiltration rate between 23% and 37% of the total precipitation is estimated for the area. The high infiltration rates are related to the sand and sandy surficial soil types present in the area. Any significant dewatering related to the development of any of the claim blocks may have an effect on the local water balance of each claim block and water balance of the water bodies and wetlands.

Trow was retained by Rock Tech to conduct a baseline ecological study for the Georgia Lake Lithium Property. A before-after-control-impact (BACI) study is used to predict and manage environmental impacts. Environmental data are collected both before and after mining activities have started to place the mine site activities in the context of baseline conditions. In light of this, attempts were made to collect water samples at preselected locations situated upstream, within and downstream of proposed drilling (and potential future extraction) sites. An internal report was written by Trow to discuss the results of the study and is dated March, 2011 (Trow, 2011b).

Rock Tech retained exp Services Inc. to assist with sampling of shaft water from Nama Creek Shaft 11. Collection and analysis of shaft water is a requirement for Ministry of the Environment permitting to dewater mine shafts for further exploration and development. If the shaft water is to be pumped to the surface near a watercourse, then shaft water chemistry data will help determine if the aquatic flora and fauna in the receiver watercourse will be potentially affected by the shaft water, based on the Provincial Water Quality Objectives (PWQO). An internal report was written by exp to discuss the results of the study and is dated Oct. 3, 2011 (exp, 2011b).

ENVIRONMENTAL BASELINE STUDY

The next step towards the implementation of the project is the initiation of the Environmental Baseline Study to support requirements for future project permitting for Advanced Exploration and Production (mining/processing) purposes.

The environmental baseline studies that will be initiated to support above requirements for Lithium mine and processing facility at Nama Creek include:

- Hydrology and water quality;
- Hydrogeology;
- Geochemistry;
- Terrestrial Biology;
- Aquatic Biology; and
- Archaeology.

The baseline studies will collect environmental information about the current site and areas surrounding the site that could potentially be affected by mining/processing activities. The baseline studies give an indication of conditions prior to any development. In addition, they can be used as a basis for the predictions of potential environmental impacts and appropriate mitigation can be incorporated into the project plan to minimize those impacts. Should the project plan or project footprint change, the baseline studies may have to be adjusted or expanded to incorporate the areas affected.

MINE CLOSURE

With the further feasibility process towards production, a Closure Plan will be required. A closure plan must be developed and acknowledged by the ministry before production can begin, and, after “approval”, if any material change is made on site.

A closure plan outlines how the affected land will be rehabilitated to approximate pre-development conditions, meet the requirements of the “Mine Rehabilitation Code” and the costs associated with doing so. Costs affiliated with post closure monitoring are required to be included.

To ensure that the rehabilitation work outlined in a closure plan is successfully performed, a financial guarantee (“financial assurance”) equal to the estimated cost of the rehabilitation work must be held in trust by the ministry. Financial assurance must be included with the submission of a closure plan.

CAPITAL AND OPERATING COSTS

BASIS

SOURCES

The sources of information used in this cost estimation can be summarized as:

- Data provided by the Rock Tech
- In-house database
- Suppliers
- Calculation from first principles
- Assumptions or estimates were used when none of the previous sources were available.

ESTIMATE BASE DATE AND VALIDITY PERIOD EXCHANGE RATE

DMT prepared this preliminary assessment estimate with a base date of Q2 2018. No escalation is applied to the estimate. All operating and capital costs were prepared in Canadian dollars ("CAD"). Any conversion from other currencies has been done with the following exchange rates:

- EUR @ 1.50 CAD
- USD @ 1.30 CAD

CONTINGENCY

Each of the estimates will have a contingency added which is 20 % for capital costs and 10% for operating expenditures.

WORKING CAPITAL

A working capital of CAD 3.0 million has been included.

MAJOR ASSUMPTIONS

Estimation is based on the project obtaining all relevant permits in a timely manner to meet the project schedule:

- All material mined as waste rock is suitable for backfill for surface and underground mining
- The major part of the tailings will be used as backfill, only a small part will be stored in the TDF
- Skilled staff, supervisors, and contractors are readily available.
- Site preparation, mass earthworks and haulage road construction is to be performed by a contractor;
- The geotechnical nature of the site is assumed to be sound, uniform, and able to support the intended structures and activities. Adverse or unusual geotechnical conditions requiring piles or soil densification have not been allowed for in this estimate
- Availability of construction materials for the TDF

MAJOR EXCLUSIONS

The following items were not included in this capital cost estimate:

- Provision for inflation, escalation, currency fluctuations and interests incurred during construction;
- Project financing costs;
- Changes to design criteria;
- Schedule delays
- Costs of income benefit agreements with First Nations communities;

CAPITAL COSTS

SUMMARY

The mine site project covered in this study is based on the construction of a green field facility having a nominal daily processing capacity of average 1,500 tpd for open pit (Contractor's operation) and 2,000 tpd for underground mining (Owner's operation). The capital and operating cost estimates related to the mine, processing plant, site infrastructure have been developed by DMT.

The capital cost estimate for this project presented herein is considered to with an expected accuracy level of +30%/-30% and carrying a contingency of 20% on total initial estimated capital.

The capital cost estimate consists of the initial capital costs with CAD 66.5 million plus CAD 3.0 million as working capital. The sustaining capital sum up to CAD 68.2 million. The capital cost summary and its distribution by area is shown in Table 63.

CAPEX	**Mio CAD**
Initial Capital	
Mining	
Processing	46.1
G&A	1.0
Other Costs	18.3
Working Capital	3.0
Pre Production Capital	68.3
Sustaining Capital	
Mining	50.3
Processing	9.1
G&A	0.6
Other Costs	5.0
Closure Costs	3.0
Total Capital Expenditures	**136.3**

Table 63: LoM Cost Summary

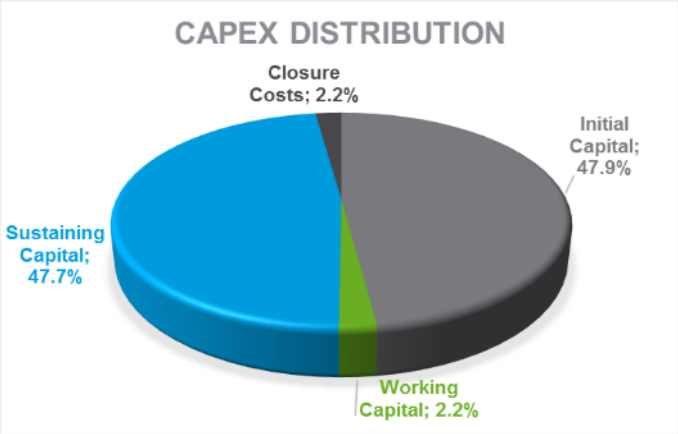


Figure 83: Distribution of Capital Costs

MINING CAPITAL COSTS

INTRODUCTION

The mine capital costs have been estimated by DMT from first principles according to the fleet requirement over the course of the mine life for both open pit and underground operation.

OPEN PIT MINING CAPITAL COSTS

The main pegmatites are outcropping on the surface. The Main Zone North was selected due to its accessibility and resource size to start with the mining operation.

In order to have sufficient capacities for the development of the underground mine while running the surface operation and in order to save initial capital, the employment of a mining contractor was envisaged from Year 1.

CAPEX Open Pit Mine	Mio CAD
Shovel	1.4
Drill Rig (small)	0.4
Blasting Vehicle	0.2
Trucks	1.5
Auxiliary Equipment	1.4
SUBTOTAL CAPEX	4.9
CONTINGENCIES	1.0
Total CAPEX Open Pit	**5.9**

Table 64: Open Pit Mine Capital Costs

A budget cost of CAD 2.72/t mined or CAD 19.10/t ore was calculated based on a lifetime of the open pit operation of roughly 5 years. An allowance of 20% has been included as profit margin for the mining contractor. The specific equipment requirements and capital costs are outlined in Table 64. Due to the short mine life no equipment replacement is foreseen.

UNDERGROUND MINING CAPITAL COSTS

The underground operation is planned as Owner's operation. Start of development of the underground access starts in Year 1. The start of the underground production in Year 2 and will last over the LoM. Peak production will be from Year 6 on reaching 1.0 Mtpy.

Mining capital costs consist of accessing the deposits, purchasing the mining, semi-mobile equipment (crusher and optical sorter) and the surface equipment required to start production. It also includes all requirements for dewatering and ventilating the operation as well as the electrical supply underground. Table 65 shows the underground mining capital costs over the LoM.

CAPEX Underground Mine	Mio CAD
Mine Access	14.0
Mobile equipment	22.8
Semimobile Equipment	1.3
Dewatering	0.4
Ventilation	1.7
Electricity supply, compressors	1.1
Workshop and other underground Instal	0.5
Surface requirements	0.2
SUBTOTAL CAPEX	41.9
CONTINGENCIES	8.4
Total CAPEX Underground Mine	**50.3**

Table 65: Underground Mine Capital Costs

PROCESSING PLANT CAPITAL COSTS

The processing capital cost estimate is a factored estimate based on the process mechanical equipment supply costs. The equipment was sized and selected on the basis of the production requirements and test work executed.

Factors for each area of the processing facility were applied to estimate the associated costs for civil and earthworks, concrete, structural steelwork, piping and instrumentation and electrical. The estimated process equipment selection, sizing and supply costs were based on the process design criteria and the process flowsheet discussed in section Recovery Methods of this report. The factors used for the estimate are based on DMT's experience and in-house database.

The Spodumene processing plant was designed for a daily feed capacity of 150t/h. The capital costs for the project mineral processing plants are estimated at CAD 46.1 million, divided between CAD 33.2 million as direct and CAD 5.2 million indirect costs plus additional CAD 7.7 million contingencies. These costs will all occur during pre-production Year-1.

CAPEX Processing Plant	Mio CAD
DIRECT COSTS	
Earthworks	3.8
Crushing Section	2.5
Gravity Separartion Section	1.7
Mill Section	1.8
Flotation Section	2.8
Dewatering Section	7.8
Tailings	1.5
Concentrate store	0.6
Electric	2.3
Instrumentation	1.8
Steel construction	3.8
Building	3.0
SUBTOTAL DIRECT COSTS	33.2
INDIRECT COSTS	
EPC	4.1
Commissioning	0.5
Transport	0.6
SUBTOTAL INDIRECT COSTS	5.2
CONTINGENCIES	7.7
Total CAPEX Processing Plant	**46.1**

Table 66: Processing Plant Capital Costs

OTHER CAPITAL COSTS

Under Other Costs mainly infrastructural costs such as site development costs (deforesting, installation of the pit dewatering system, road upgrade and grid connection), surface facility installations (admin building, operators building, workshops, warehouse, explosive store and fuel station) and costs for additional studies and exploration are included.

CAPEX Others		Mio CAD
Studywork	Exploration to FS level	1.5
	FS	1.5
	ESIA	0.5
Site development Costs		6.7
Surface Facilities		5.1
SUBTOTAL CAPEX		15.2
CONTINGENCIES		3.0
Total CAPEX Others		**18.3**

Table 67: Other Capital Costs

The total amount for the initial capital costs for Other Costs is CAD 15.2 million plus additional CAD 3.0 million contingencies.

CLOSURE COSTS

Mine closure costs for the project have been estimated at CAD 3.0 million. The mine closure costs were included as ongoing capital costs in Year 11, not as a preproduction capital cost item. Over the LoM yearly costs have been assumed for rehabilitation as well, but as operating expenditures.

At this stage no details on the mine closure and polishing the plant site exists. This will be done during the prefeasibility study or feasibility study stage.

OPERATING COSTS

SUMMARY

The average unit operating cost over the LoM was estimated at CAD 397/tonne concentrate. The unit operating costs include open pit and underground mining cost with CAD 212/tonne concentrate, mineral processing cost (CAD 133/tonne concentrate), general and administration (G&A) cost with CAD 26/tonne concentrate and other operating costs (CAD 26/tonne concentrate). Operating costs for the project are summarized in Table 68 and Figure 84.

OPEX (Unit Costs)	Unit	Value
Mining	Mio CAD	224.3
	CAD/t conc	212
Processing	Mio CAD	140.4
	CAD/t conc	133
G&A	Mio CAD	27.4
	CAD/t conc	26
Other costs	Mio CAD	27.3
	CAD/t conc	26
Total OPEX	Mio CAD	419.4
	CAD/t conc	397

Table 68: LoM and Unit Operating Costs

Items listed in Table 68 include labor costs, which were estimated based on the manpower that will be necessary to operate the proposed mobile mining fleet and stationary equipment. Mining production rates and productivity as well as equipment mechanical availability and utilization factors were taken into account in the operating cost estimate. Annual salary projections were based on current mining industry standards.

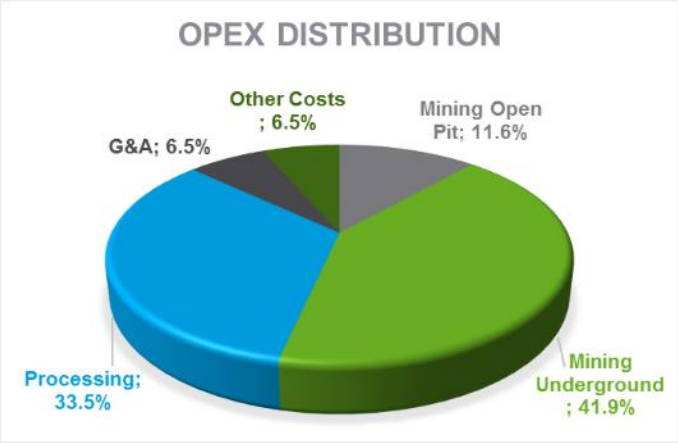


Figure 84: Distribution of Operating Costs

WORKING TIMES

In the following tables the basis for the manpower calculation based on working days, shift system, and scheduled non-productive times are summarized for mine and processing plant.

Working Time Mine	Unit	Value
Number of shifts per day	sh/d	2
Duration of shifts	h/sh	12
Scheduled non-productive time	h/sh	1.5
Net operating time per day	h/d	21
Working days per year	d/y	335
Net operating time per year	h/y	7,035

Table 69: Working Time Mining Operation

Working Time Plant		
Number of shifts per day	sh/d	3
Duration of shifts	h/sh	8
Scheduled non-productive time	h/sh	
Net operating time per day	h/d	24
Working days per year	d/y	300
Net operating time per year	h/y	7,200

Table 70: Working Time Processing Plant

MANPOWER

OPEN PIT

A contractor will execute the open pit extraction. The manpower requirements for the contractor have been separated into Management and Supervision, Mine Production, and Maintenance. The personnel will work

335 days per year on two 12 h shifts per day coverage. Table 71 is a list of the workforce and shows the typical open pit mine workforce requirement during peak production.

Open Pit Production (Contractor)	Quantity
Management and Supervision	3
Operator Main Equipment	
Shovel	6
Drill Rig (small)	2
Blasting Vehicle	1
Trucks	20
Operator Auxiliary Equipment	5
Maintenance	7
Total	44

Table 71: Contractor's Open pit workforce estimate

UNDERGROUND MINE

The peak production workforce for the underground operation will be approximately 46 people.

Underground Production	Quantity
Driller Development	3
Bolter	2
Driller Production	3
Charging	3
Transport Concrete	2
Support Installation	2
Scaler	2
LHD Operator	6
Truck Driver	9
Service Truck Operator	1
Mechanics	5
Electricians	2
Mine Services	3
Other Laborer	3
Total	46

Table 72: Underground Workforce

G&A

For the management and the supervision about 18 people as staff has been estimated as peak. In the first year of operation not all of the people have been employed.

Management & Supervision	Quantity
General Manager	1
Mine	
Manager Mine	1
Engineer	2
Geologist	3
Acounting/Controlling	2
Administrative Support	3
Processing Plant	
Manager Processing Plant	1
Engineer	2
Administrative Support	2
Infrastructure/Environmental	
Manager Infrastructure/Environmental	1
Total	18

Table 73: G&A Staff

TOTAL MANPOWER REQUIREMENTS OVER LOM

The manpower will vary along the mine life. It is estimated that up to 150 people will be working on the site during the peak production years. Table 73 shows the estimated manpower requirements based on the proposed fleet of mining equipment and two work shifts per day respectively 3 shifts per day for the processing plant. The professional staff (engineers, technicians and management) will work on a 5-day schedule per week.

Staffing	Year	1	2	3	4	5	6	7	8	9	10	11
Mining	Open Pit (Contractor)	36	44	28	32	34	17					
	Underground	24	28	28	28	33	39	36	39	39	33	33
Processing		23	23	23	23	23	23	23	23	23	23	23
Maintenance		8	10	13	13	13	15	15	15	15	13	10
Supervision		8	10	10	10	10	10	10	10	10	10	8
G&A		16	18	18	18	18	18	18	18	18	18	18
Infrastructure and Environmental		5	6	8	8	8	8	8	8	8	8	5
Safety and Security		5	8	8	8	8	8	8	8	8	8	5
Total		125	147	136	140	147	138	118	121	121	113	102

Table 74: Staffing requirements over LoM

OPEN PIT OPERATING COSTS

The LoM operating costs will be CAD 48.6 million, average costs varying between CAD 5 and 11 million depending on the yearly production height. The summary and cost distribution is shown in Table 75 and Figure 85.

Open Pit OPEX	Mio CAD
Diesel and Lube Cost	3.1
Spare Parts/Maintenance ar	6.5
Others	0.1
Labor and Supervision	21.7
Depreciation	6.0
Profit Margin	7.5
Contingency	3.7
TOTAL	**48.6**

Table 75: LoM Open Pit Operating Costs

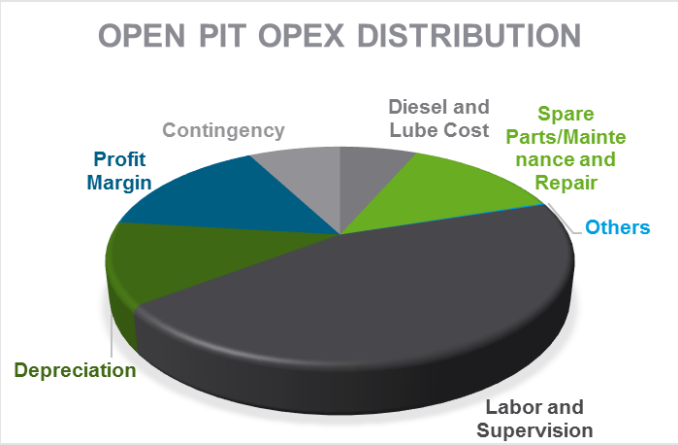


Figure 85: Distribution of LoM Open Pit Operating Costs (Contractor's Operation)

UNDERGROUND MINE OPERATING COSTS

The underground development starts in Year 1 and the first ore to be extracted is expected in Year 2. The underground mining will last until the final year of operation. Maximum production is 1 million tonnes in Year 6 to Year 9.

Underground OPEX	Mio CAD	% of Total
Development	7.4	4%
Stoping	50.9	29%
UG Sorting	1.0	1%
Other extraction costs	3.4	2%
Transport	5.3	3%
Backfill	7.6	4%
Power	32.4	18%
Labor	51.7	29%
Contingency	16.0	9%
TOTAL	**175.7**	**100%**

Table 76: LoM Underground Operating Costs

During that time in total 7.8 km in waste are estimated to be developed as well as 15.8 km in ore. Underground mining costs include besides the machine operation as well an estimate on the truck transport costs to the at site processing facility, mine dewatering and ventilation including heating during winter months.

Table 76 and Figure 86 are showing the LoM underground costs and the distribution of the single cost items.

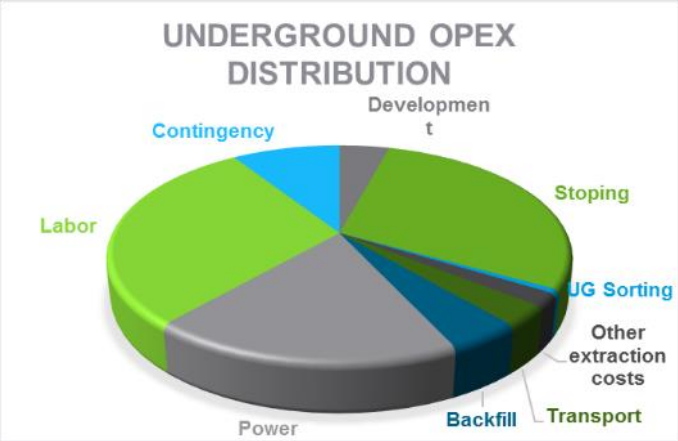


Figure 86: Distribution of LoM Underground Operating Costs (Owner's Operation)

PROCESSING PLANT OPERATING COSTS

Yearly operating costs during peak production have been estimated with CAD 13.9 million, whereof the major portion belongs to labor, power requirements, spare parts and maintenance. The distribution of the yearly costs are shown in the Figure 87.

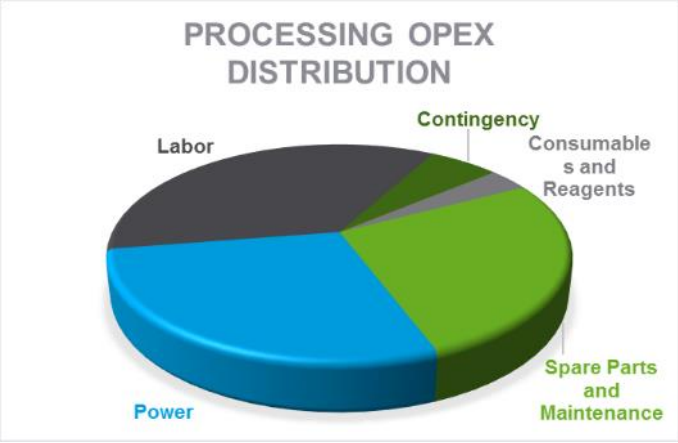


Figure 87: Distribution of Yearly Processing Operating Costs

Processing OPEX	Mio CAD
Consumables/Reagents	0.5
Spare Parts/Maintenance	3.8
Power	3.9
Labor	4.9
Contingency	0.8
TOTAL	**13.9**

Table 77: Yearly Processing Operating Costs

G&A OPERATING COSTS

G&A costs include the management and supervision costs for the contractor, the underground mine, processing plant and other costs. Yearly operating costs were estimated with CAD 2.5 million.

OTHER OPERATING COSTS

Other operating costs contain the allowances for the infrastructural and environmental necessities of the operation. Yearly costs have been estimated by DMT with CAD 2.5 million, which includes the rehabilitation costs.

Other OPEX	Mio CAD
Consumables/Reagents	0.5
Spare Parts/Maintenance	0.1
Power	0.6
Labor	0.9
Rehabilitation	0.3
Contingency	0.1
TOTAL	**2.5**

Table 78: Yearly Other Operating Costs

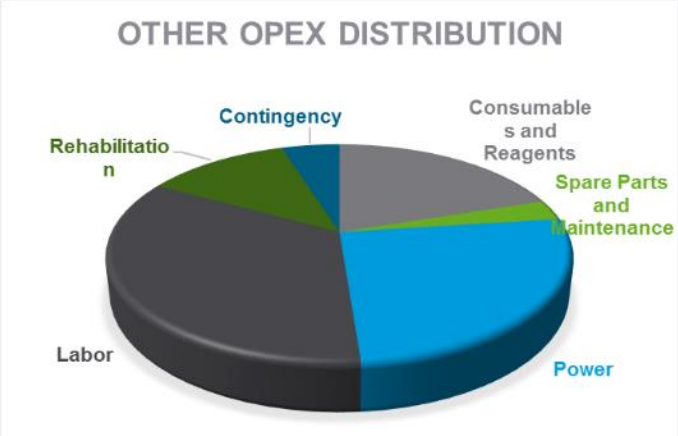


Figure 88: Distribution of Yearly Other Operating Costs (Infrastructure and Environmental)

PRODUCTION PARAMETERS IN THE FINANCIAL MODEL

The life-of-project input respectively average material tonnages, grades and concentrate production are shown in Table 79.

Georgia Lake Project	Unit	Value
LoM	Years	11
Production (diluted)	Mio t	9.6
Open pit	Mio t	2.7
Stripping Ratio	t:t	6:1
Underground	Mio t	6.9
Annual mine production	Mio t/a	0.87
Average Feed Grade (diluted)	%	0.87
Plant Recovery	%	78.00
Spodumene Concentrate Grade	%	6.20
Total Spodumene Concentrate	000 t	1,056
Annual Spodumene Concentrate	000 t/a	96
Royalty Rate (% of Revenue-OPEX)	%	1.50
Average LoM Spodumene Concentrate Price	USD/t	827

Table 79: Main Input and Production Parameters

DMT evaluated the base case using long-term forecast Spodumene concentrate prices of USD 800/t increasing up to USD 850/t. The financial models were established on a 100% equity basis, excluding debt financing, and loan interest charges.

BASIS OF FINANCIAL EVALUATION

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered production from the relationships of tonnage processed, head grades, and recoveries.

Spodumene concentrate (payable values) were calculated based on base case prices. Net invoice value was calculated each year as ex works prices no sales, transportation and insurance costs have been estimated.

Unit operating costs for mining, processing, power, fuel, and G&A were applied to annual mined/processed tonnages to determine the overall operating cost which was deducted from the revenues to derive annual operating cash flow.

Initial capital costs as well as working capital have been incorporated on a year-by-year basis over the mine life. Mine reclamation respectively rehabilitation costs are applied to the operating, closure costs to capital expenditure in the last production year. Capital expenditures are then deducted from the operating cash flow to determine the net cash flow before taxes and mining royalty.

Initial capital expenditures include costs accumulated prior to first production of concentrate; sustaining capital includes expenditures for mining and processing additions, replacement of equipment and others such as tailings handling.

Based on the mining schedule, first production will occur approximately one year following project approval and project start.

Working capital is assumed to be a portion of the annual operating cost and fluctuates from year to year based on the annual cost. The working capital is recovered at the end of the mine life.

PRE-TAX FINANCIAL ANALYSIS

SUMMARY OF FINANCIAL RESULTS

DMT prepared an economic evaluation of the Project based on a pre-tax financial model. The pre-tax financial results are:

- 62.2% IRR
- 3.1-year payback from the start of processing operations
- CAD 312 million NPV at an 8% discount rate.

The undiscounted annual net cash flow and cumulative net cash flow are illustrated in Figure 89.

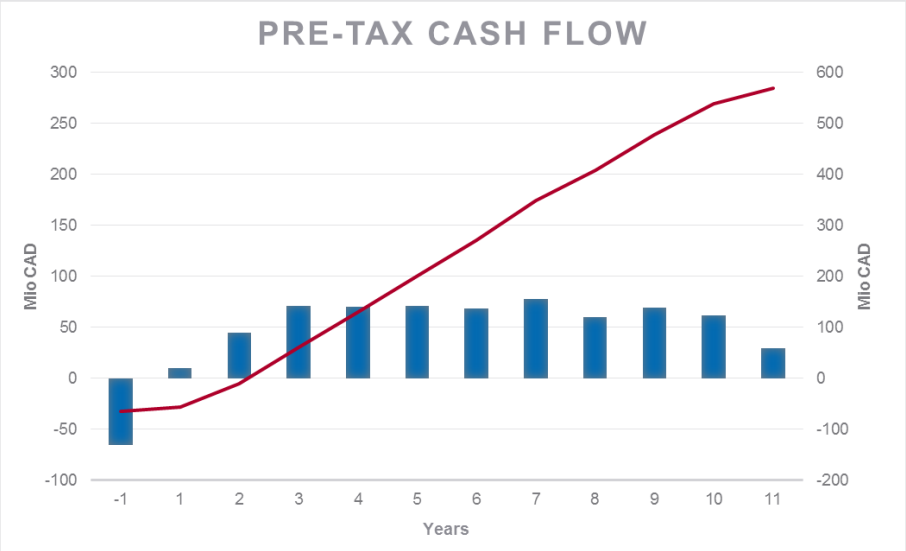


Figure 89: Pre-Tax Cash Flow

SENSITIVITY

Sensitivity of the project's pre-tax NPV, IRR to the Project key variables was investigated. Using the base case as a reference, each of the key variables was changed between ±25% at 5% intervals while holding the other variables constant. The following are the key variables investigated:

- Spodumene price
- Capital costs
- Operating costs
- Li_2O grades
- Diesel price
- Electricity price

As shown in Figure 90, the Project NPV, calculated at an 8% discount, is most sensitive to the changes in the Li_2O grade and Spodumene price and, in decreasing order, operating costs, capital costs and electricity price.

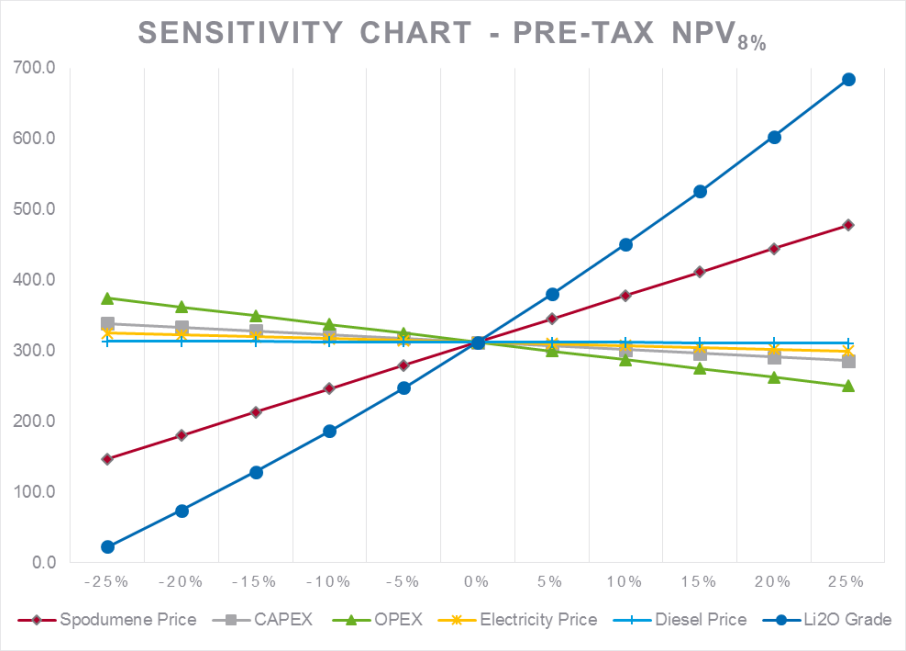


Figure 90: Pre-Tax NPV Sensitivity Analysis

As shown in Figure 22.4, the Project IRR is most sensitive to the Li_2O grade and Spodumene price followed by the capital costs, operating costs.

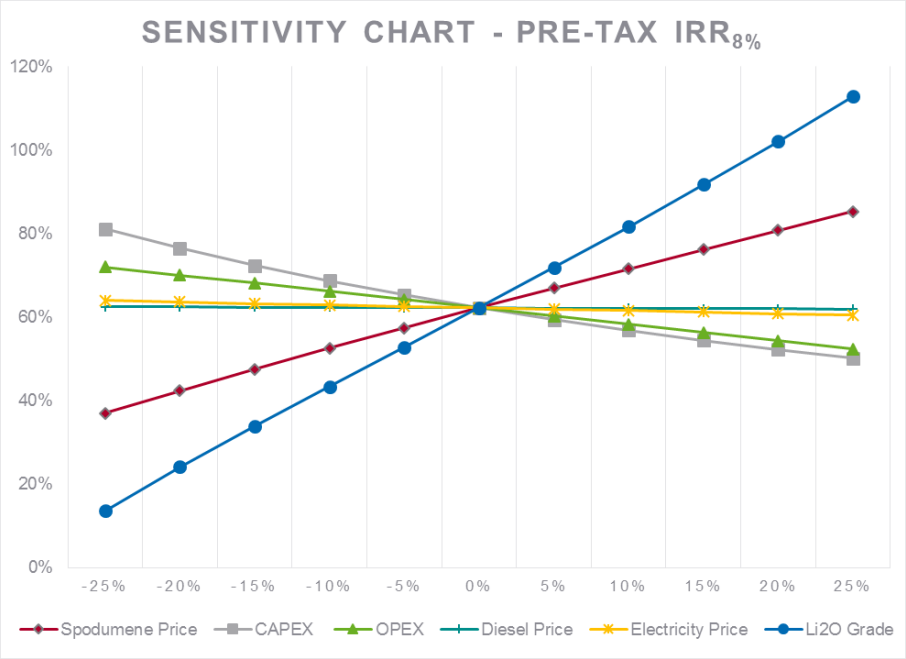


Figure 91: Pre-Tax IRR Sensitivity Analysis

POST-TAX FINANCIAL ANALYSIS

SUMMARY OF FINANCIAL RESULTS

DMT prepared a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable taxes and the royalty.

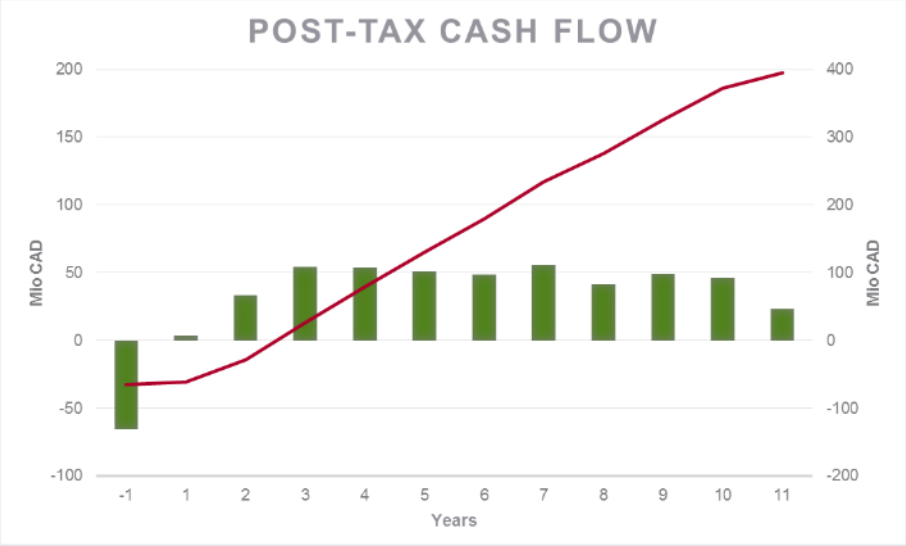


Figure 92: Post-Tax Cash Flow

The following post-tax financial parameters were calculated:

- 48.1% IRR
- 3.5-year payback from the start of processing operations
- CAD 210 million NPV at an 8% discount rate.

TAXATION

The following are the taxes applied to the Rock Tech Lithium Project:

Canadian Corporate Income Tax Rates Applicable to Mining (CCH, for December 31, 2013 year-end). Net federal tax rate on resource income is 15%, while the provincial income tax applied is 10%. The mining tax exemptions for new mines was estimated with no tax for the first 5 years and then with 5 percent tax rate.

ROYALTY

The only royalty on the project is a 1.5% NSR minus operating costs. This Royalty has been considered in the economic calculation of the project.

TRANSPORTATION

Transportation costs have not been considered in the post-tax cashlow model. Any costs have to be added.

INSURANCE

Insurance costs have not been considered in the post-tax cashlow model. Any costs have to be added.

SENSITIVITY

The sensitivity of project returns to changes in all revenue factors including grades, prices as well as capital and operating costs was tested over a range of ±25% above and below base case values.

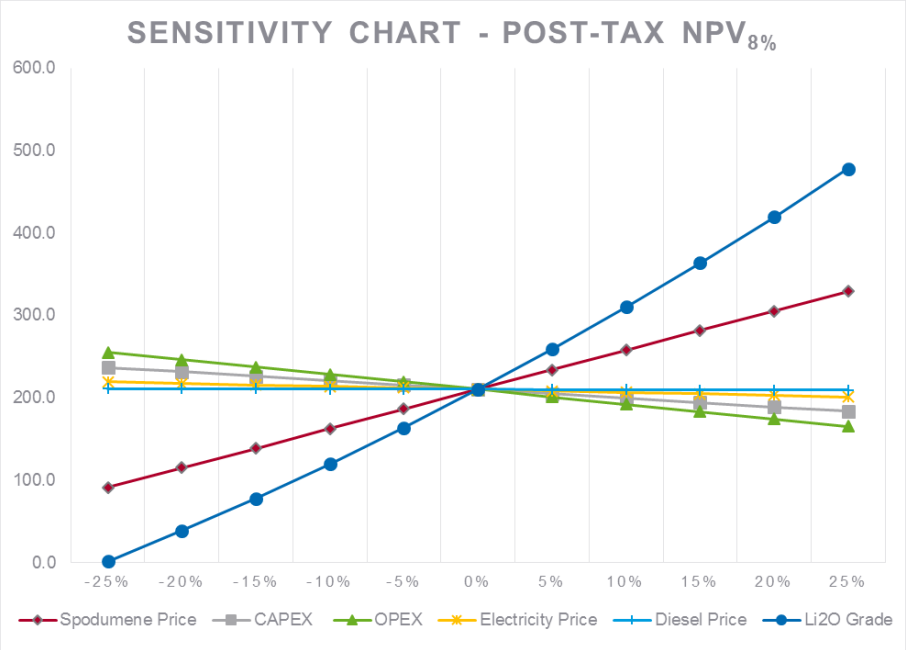


Figure 93: Post-Tax NPV Sensitivity Analysis

The chart suggests that the project is most sensitive to revenue drivers and is moderately sensitive to changes in operating costs and capital cost. While the latter remain positive across the range of the sensitivity analysis, NPV falls to zero for Li_2O grades of less than 75% of base case assumptions (see Figure 93).

The same applies to the IRR sensitivity analysis where at of less than 75% of base case assumptions the IRR would be as high as the discount factor (see Figure 94).

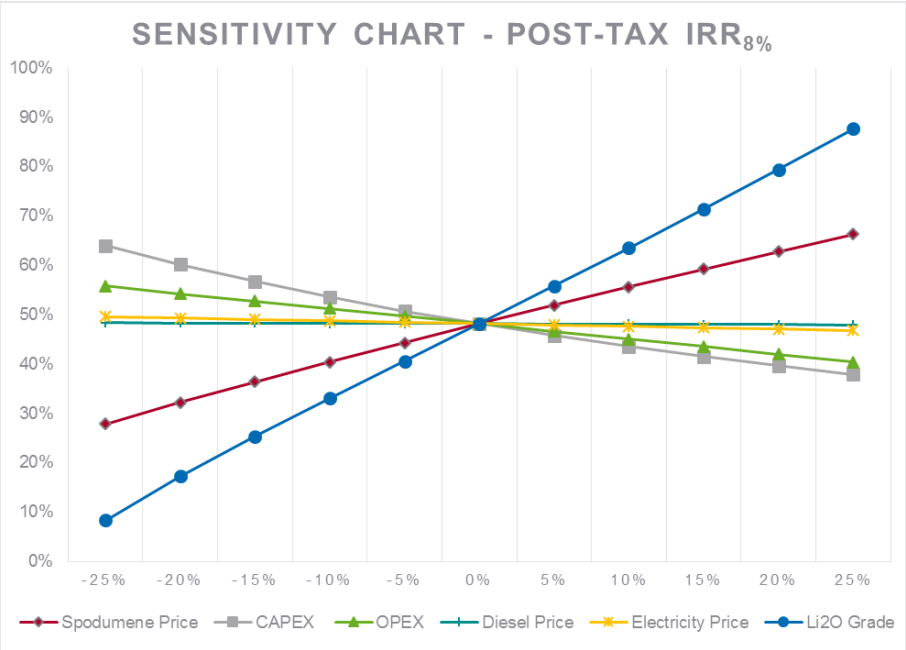


Figure 94: Post-Tax IRR Sensitivity Analysis

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. All costs in this section are expressed in Canadian Dollars (CAD). All results are summarized in Table 80 and Table 81.

Outcome I	Unit	Value
Initial CAPEX	Mio CAD	65.3
Average LoM Operating Costs	CAD/t conc	397
LoM Revenue	Mio CAD	1,136.2
LoM Operating Costs	Mio CAD	419.4
LoM EBITDA	Mio CAD	706.0
Annual EBITDA	Mio CAD	64.2
Pre Tax NPV_8	Mio CAD	312.2
After Tax NPV_8	Mio CAD	210.2
Pre Tax IRR	%	62%
After Tax IRR	%	48%
Pre Tax Payback	Years	3.1
After Tax Payback	Years	3.5

Table 80: Economic Indicators

Outcome II	Unit	Value
CAPEX		
Initial Capital	Mio CAD	65.3
Working Capital	Mio CAD	3.0
Total Pre-Production Costs	Mio CAD	68.3
Sustaining Capital	Mio CAD	62.0
Closure Costs	Mio CAD	3.0
Total CAPEX	Mio CAD	133.3
LoM OPEX		
Mining	Mio CAD	224.3
	CAD/t conc	212
Processing	Mio CAD	140.4
	CAD/t conc	133
G&A	Mio CAD	27.4
	CAD/t conc	26
Other costs	Mio CAD	27.3
	CAD/t conc	26
Total OPEX	Mio CAD	419.4
	CAD/t conc	397
LoM Royalties	Mio CAD	10.8

Table 81: Operating and Capital Costs

Analyses were conducted to assess the sensitivity of the pre-tax Project merit measures (NPV, IRR and payback periods) to the main inputs.

Yearly Summary			Unit	Year	-1	1	2	3	4	5	6	7	8	9	10	11	Total
Mining	Open Pit	Waste Removal	Mio t			4.4	4.7	2.2	2.2	2.5	0.2						16.2
		Ore Extraction	Mio t			0.5	0.6	0.5	0.5	0.5	0.0						2.7
		Stripping Ratio	t:t			8.7	7.6	4.3	4.3	5.1	5.1						6.1
	Underground	Ore Extraction	Mio t				0.2	0.5	0.5	0.5	1.0	1.0	1.0	1.0	0.8	0.5	6.9
Processing	Plant Feed		Mio t			0.5	0.8	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.8	0.5	9.6
	Plant Feed Grade		%			0.87	0.88	0.88	0.88	0.91	0.87	0.89	0.81	0.86	0.89	0.89	0.87
	Produced Concentrate @ 6.2%		000 t conc			55	89	111	111	114	110	111	102	108	90	56	1,056
Revenue			Mio CAD			57.2	92.4	114.9	114.9	119.0	121.6	123.1	113.0	119.3	99.0	61.9	1,136
Royalty Payments			Mio CAD			0.4	0.8	1.1	1.1	1.2	1.2	1.2	1.1	1.2	1.0	0.5	11
OPEX	Mining		Mio CAD			14.0	21.2	20.7	21.6	22.8	25.7	21.3	22.3	22.3	17.9	14.3	224
	Processing		Mio CAD			9.4	12.1	13.9	13.9	13.9	13.9	13.9	13.9	13.9	12.1	9.4	140
	G&A		Mio CAD			2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	27
	Other Costs		Mio CAD			2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	27
	Total OPEX		Mio CAD			28.4	38.3	39.6	40.5	41.7	44.6	40.2	41.2	41.2	35.0	28.7	419
EBITDA			Mio CAD			28.4	53.3	74.2	73.3	76.1	75.8	81.6	70.7	77.0	63.0	32.6	706
CAPEX	Mining		Mio CAD			13.4	4.2	0.4	1.7	3.3	5.8	2.5	9.9	5.8	1.6	1.6	50
	Processing		Mio CAD		46.1	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	55
	G&A		Mio CAD		1.0	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	2
	Other Costs		Mio CAD		18.3	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	3.5	26
	Working Capital		Mio CAD		3.0												3
	Total CAPEX		Mio CAD		68.3	14.7	5.5	1.7	3.1	4.6	7.2	3.8	11.2	7.2	3.0	6.0	136
Pre-Tax CF	Pre-Tax Cashflow		Mio CAD		(65.3)	9.7	45.1	70.7	70.3	71.2	68.5	77.5	60.3	69.3	61.5	29.5	568
	Pre-Tax Cumulative Cashflow		Mio CAD		(65.3)	(55.6)	(10.5)	60.2	130.5	201.7	270.2	347.8	408.1	477.4	538.9	568.4	
Taxes	Federal Corporate Income Tax		Mio CAD			3.6	7.0	10.0	9.9	10.2	10.0	11.0	9.3	9.9	7.8	3.2	92
	Provincial Corporate Income Tax		Mio CAD			2.4	4.6	6.7	6.6	6.8	6.7	7.3	6.2	6.6	5.2	2.2	61
	Mining Tax		Mio CAD							3.4	3.3	3.7	3.1	3.3	2.6	1.1	20
	Total Taxes		Mio CAD			6.0	11.6	16.7	16.4	20.4	20.1	22.0	18.6	19.9	15.5	6.5	174
Post-Tax CF	Post-Tax Cashflow		Mio CAD		(65.3)	3.7	33.5	54.0	53.9	50.8	48.4	55.5	41.8	49.4	46.0	23.0	395
	Post-Tax Cumulative Cashflow		Mio CAD		(65.3)	(61.6)	(28.1)	25.9	79.8	130.6	179.0	234.6	276.3	325.7	371.8	394.8	

Table 82: Base Case Life-of-Mine Annual Cash Flow

ADJACENT PROPERTIES

Exploration activities of adjacent properties have not been considered. North of the area the Beardmore-Geraldton area hosts several deposits types including vein-hosted gold and lithium pegmatites deposits (see CCIC Report 2012).

OTHER RELEVANT DATA AND INFORMATION

On July 14, 2011 Rock Tech announced that it entered into a Memorandum of Understanding ("MOU") with Bingwi Neyaashi Anishinaabek ("BNA"), Biinjitiwaabik Zaaging Anishinaabek ("BZA"), and Animbiigoo Zaagi'igan Anishinaabek ("AZA") First Nations (collectively referred to as "First Nations") in regards to the development of the Georgia Lake Lithium project. These First Nations communities are in close geographical proximity to the Georgia Lake Lithium project. While 100% of the project lies within First Nations' traditional territories, a 2 km stretch of the road accessing the Nama Creek mineral lease is on BNA's reserve land. Since Rock Tech began exploration in December 2009, several First Nations members have been employed and equipment and material have been procured from the First Nations whenever feasible.

No other relevant information is available.

DMT of Essen, Germany was contracted by Rock Tech of Vancouver, Canada, to review the Georgia Lake Lithium Project, and prepare a Preliminary Economic Assessment, compliant with National Instrument 43-101. The report should be based on the available documentation inter alia DMT's Independent Technical Report on the updated the mineral resources based on latest data acquired.

The objective of this PEA is to demonstrate the economic potential for producing a lithium ion battery pre-product from the Rock Tech Lithium deposit.

The deposit was evaluated previously in 1950ies as a potential source of the lithium mineral. While this market remains an opportunity, lithium ion battery technology has developed as the energy storage solution of choice for a variety of commercial applications and this has resulted in a significant increase in demand, and projected demand, for battery materials.

The economics of the project have been evaluated with an Excel-based real-basis financial model developed in 2018 CAD to present the cost structure and the economic evaluation of the project as a stand-alone entity.

DMT examined the technical and economic aspects of the Project within the level of precision of a Preliminary Economic Assessment. As it stands, the economic analysis is preliminary in nature, however, the Project contains an economic mineral resource based on mainly measured/indicated resources and a small portion of inferred resources ensuring a life of mine of 11 years.

The parameters used in this Preliminary Economic Assessment include developing a 500,000 tpy open-pit mine, using diesel hydraulic equipment and operated by a contractor, development of an underground mine operation to reach a 1,000,000 tpy total mine production and the construction of a processing plant at the mine site (crushing, grinding, flotation circuits, dewatering) with a nominal capacity of 150t/h of ore feed at higher 90% availability

Outcome I	Unit	Value
Initial CAPEX	Mio CAD	65.3
Average LoM Operating Costs	CAD/t conc	397
LoM Revenue	Mio CAD	1,136.2
LoM Operating Costs	Mio CAD	419.4
LoM EBITDA	Mio CAD	706.0
Annual EBITDA	Mio CAD	64.2
Pre Tax NPV_8	Mio CAD	312.2
After Tax NPV_8	Mio CAD	210.2
Pre Tax IRR	%	62%
After Tax IRR	%	48%
Pre Tax Payback	Years	3.1
After Tax Payback	Years	3.5

Table 83: Economic Indicators

The lithium pricing was developed by DMT in cooperation with Rock Tech based on the recent development of the market for a 6.2% Li_2O concentrate ranges from USD 800 in 2020 to USD 850 in 2026; this value is used from 2026 to the end of the Project in 2031.

An exchange rate of 1.30 CAD to USD and 1.50 to EUR is used in the financial evaluation. The project cash flows were assessed to 2031. The financial model has been used to estimate future cash flows and evaluate the project on the basis of net present value (NPV), internal rate of return (IRR) and payback period. The results of the analysis are provided in the Table 83.

The total unit operating cost for Li_2O concentrate from Georgia Lake deposit is CAD 397 /t conc which is equivalent to USD 305 /t conc..

Consequently, DMT concludes that the Project is technically feasible as well as economically viable. The values obtained for NPV, IRR and the payback period show that the Project is profitable. Figure 95 presents the post-tax cashflow of the Project. The authors of this Technical Report consider the Project to be sufficiently robust to warrant moving it to the Prefeasibility or even Feasibility level.

Post-tax free cash flow over the life of the project is summarised in Figure 95. The financial analysis completed examined the IRR sensitivity to the main factors affecting the Project. The project is most sensitive to changes in the Li_2O concentrate pricing and grade, less sensitive to changes in capital costs and operating costs and least sensitive to consumable price changes (diesel and electricity).

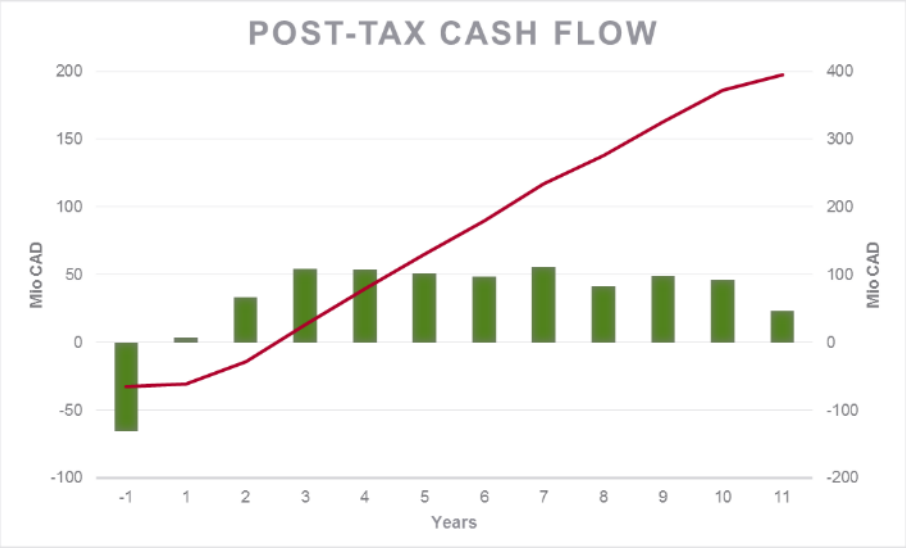


Figure 95: Post-Tax Cash Flow

RECOMMENDATIONS

INTRODUCTION

Recommendations for different areas of the project are set out below. Mainly the recommendations refer to necessary testwork and assessments in order to minimize uncertainties.

FURTHER EXPLORATION

In general, the focus on exploration activities is not only to upgrade the resources, but also to get more detailed information about e.g. the rock mass and the hydrogeological situation for more detailed further studies. However continued QA/QC checks and SG determinations should be routinely made on mineralized intersections.

DRILLINGAND OTHER INVESTIGATION

The purpose of further exploration work such as drilling is to validate more historic holes in order to increase the classification of the resource from inferred to indicated status and to carry on further exploration to check the extension of the dykes. The amount of further drilling work has to be defined in the near future with regular spaced definition drilling and channel sampling to upgrade Inferred resources to Indicated classification; e.g. 35m to 50m spaced centres. Drill spacing needs to target both along strike and down dip positions. General recommendations based on previous drill programs:

- Close monitoring of deviation in the azimuth downhole during drill programs
- Log the orientation of the rock structures for geotechnical reasons
- Looking for open holes to use them for hydraulic test and optional re-logging with a wireline service to get the true orientation of the rock mass structure.

TRENCHING

The purpose of the channel sampling is to get chemical analyses from more pegmatites and to get or upgrade the inferred resources or to get indication for further drilling targets.

The same as for drilling applies to trenching, regular spaced channel sampling to increase the inferred into the indicated or higher resource category and at the same time, upgrade exploration targets near surface to inferred status

MAPPING

Detailed mapping should be undertaken to explore for potential extensions of the lithium deposit to increase potentially recoverable lithium resources and explore for new zones. Further investigations into other potential sources of lithium minerals in the region, which could potentially provide additional feed material.

MINING

GEOTECH

The existing testwork were performed for the average rock mass quality. Fault areas will influence the stability considerably and should be avoided, if possible, or has to be supported or designed separately. Therefore these areas must be explored and investigated more detailed in the next planning steps (geophysics, core logging). These investigations should also be done for the areas without faults for verification of the used parameters and to confirm or improve the assumptions done in this analysis.

It is recommended to undertake a detailed geotechnical study with the potential drilling program to support the overall pit slopes and design of ramps and haulways, as well as the underground design.

WASTE MATERIAL

Geochemical assessment of the waste material to determine the potential for Acid Mine Drainage and assessment of the waste material in general to determine the dump angles.

OPTICAL SORTING

The idea of optical sorting is the pre-separation of waste rock and ore to save costs and transport capacities. The behaviour of the material in terms of the available sensing systems for sorting has to be tested.

SHAFT INSPECTION

Get access to the shaft and made a camera logging in the shaft to get information about the support system of the shaft to get information about the rock properties in this area after 60 years open entries. This improve the knowledge for further mine planning studies.

PROCESSING

INTRODUCTION

The main recommendations that already have been discussed with and initiated by Rock Tech is the requirement for additional testwork to further increase the knowledge. This would include the following steps:

MINERALOGY

Mineralogical studies to confirm identification of phosphate minerals at MZN and all the other pegmatites and in general in order to further refine mineralogical zonation patterns within the deposit.

BY PRODUCTS

The potentials of by-products, their recovery and marketing is recommended.

CONFIRMATION TESTS

DMT is of the opinion that the results of the SGS testwork should be confirmed with a wider range of samples in terms of different grades, especially lower grade samples should be tested to optimize the processing circuit. Tests on improving recoveries and reducing reagent consumptions are as well reasonable.

SPIRAL TESTS

DMT recommendation for a plant flowsheet is to use gravity separation with spirals instead of heavy liquid separation for material approx. 2mm – 0.3 mm. Tailings of spirals milled down to approx. 0.3 mm for flotation afterwards.

PILOT PLANT TEST

For a potential Feasibility Study one of the main requirements is the execution of a pilot plant test to confirm the results from the laboratory testwork. The sample is in the range of several tonnes and should reflect the composition of the deposit, means should be representative.

TAILINGS TEST

Tailings are of interest in different ways. First of all the mixture for the backfill is relevant since the amount of concrete to be added can increase or reduce underground mining costs distinctly. Secondly, tailings rheology should be tested to determine the fluid flow characteristics of the tailings. This is required to more accurately determine the pumping and pipeline designs. And as alternative due to the weather conditions a trade-off study to determine if filtered tailings is the better disposal and storage method.

Evident is a geochemical assessment of the tailings during the early stages of a project to determine the potential for Acid Mine Drainage.

ACCESSABILITY AND INFRASTRUCTURE

GENERAL ACCESS

Install permanent bridges for access to Conway and Line 60 instead of using temporary bridges.

ROAD ACCESS

An investigation on the best road access or the detailed evaluation of the effort for gravel road upgrading should be evaluated.

SITE INVESTIGATION

The location of the site facilities, TDF and processing plant is preliminary. A detailed study should be executed to find the optimum locations. This should be conducted in conjunction with intertization/exploration drilling.

POWER SUPPLY

Power lines and suppliers are available near the Project. The conditions of a connection should be studied in details.

WATER SUPPLY AND WATER TREATMENT

A hydrological study is recommended which would include the verification / evaluation of groundwater recharge rates based on Trow's Water Balance Study and any other available data. The study should as well investigate in more detail potential mine water inflow as well as legal requirements on water extraction and discharge.

ENVIRONMENTAL AND SOCIAL

The following should be undertaken as project development proceeds:

FIRST NATIONS

Rock Tech has already and MoU with the First Nations in the region. However, they should continue to engage with the local Indigenous Peoples, community, regulators and government to maximize local development opportunities and minimize undesirable environmental impacts.

ESIA

As part of the ESIA is the complete impact of the parts of the operation to the environment. As the basis Rock Tech should complete historical environmental baseline validation and fill in identified gaps, as well as complete a Project Description and ESIA. This would include inter alia the following:

- Initiate/update a groundwater study and assess the geotechnical design parameters for the pit, mine rock aggregate, concentrate and tailing management facilities. Assess the potential for river water to enter the open pit and make appropriate amendments as required (See Water Supply and Water Treatment, above).
- Complete biological toxicity testing of effluents and water treatment studies as required on pilot or demonstration plant water and tailing when available.
- Geotechnical and hydrogeological investigations for the TDF, initial waste dumps and stockpile locations, including identification and characterization of potential local construction materials (i.e. sand and gravel).
- Detailed topographic mapping should be obtained for the full project site. (See Mapping, above).
- Additional laboratory testing of the tailings and concentrates to better understand their physical properties as delivered to the TDF or concentrate storage facility (i.e., filterability, workability, placed density, strength, etc.) (See Tailings Test, above).

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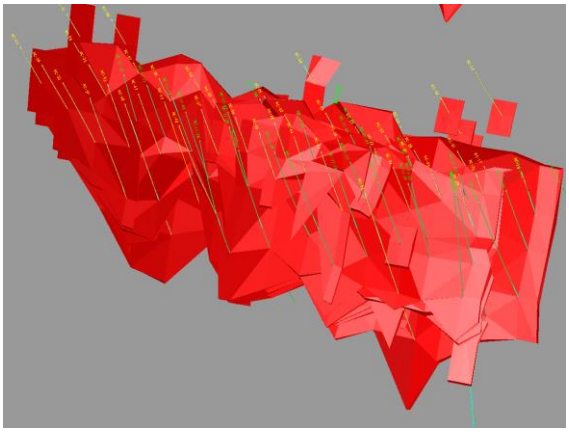
ANNEX A1: NI 43-10 TECHNICAL REPORT RESOURCE ESTIMATE – GEORGIA LAKE LITHIUM PROPERTIES BEARDMORE, ONTARIO, CANADA

NI 43-101 TECHNICAL REPORT

MINERAL RESOURCE ESTIMATION

UPGRADE 2018

GEORGIA LAKE LITHIUM PROPERTIES BEARDMORE, ONTARIO, CANADA



Submittet to : ROCK TECH LITHIUM INC.

Effective Date: 18th April 2018

Issue Date: 27th June 2018

Reference: MCE1-2018-00079

Title Page

Project Name:	Georgia Lake Lithium Properties Beardmore, Ontario, Canada
Title:	NI 43-101 Technical Report Mineral Resource Estimation Upgrade 2018 (The Technical Report)
For the company:	Rock Tech Lithium Inc.
Location:	Georgia Lake Lithium Properties Beardmore, Ontario, Canada
Effective Date:	18th April 2018
Issue Date:	27th June 2018

Qualified Persons:

Karl-Stephan Peters, EurGeol 787 Senior Geologist, was responsible for the all sections of this Technical Report, Chapter 12 and 14 as co- author.

For DMT GmbH & Co. KG Essen

Florian Lowicki, *SACNASP 400425/13* Project Manager Resource Definition was responsible for the Chapter 12 and 14 of the Technical Report

For DMT GmbH & Co. KG Essen

Statement of Authorship

This Report titled "independent Technical Report and Updated Estimated Resources Resource Estimate - Georgia Lake Lithium Property, Beardmore, Ontario, Canada", with an effective date of 18. April 2018 and a submission date of 27 June 2018, was prepared and signed by the following Qualified Persons.

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Signature of Competent Persons:		Date:
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I, Karl Stephan Peters, do hereby certify with respect to the Technical Report dated effective 18th April 2018 entitled "NI 43-101 technical report mineral resource estimation Upgrade 2018 Georgia Lake Lithium Properties Beardmore, Ontario, Canada" that:

1. I am a Senior Resource Geologist for DMT GmbH & Co. KG, Department Mining Consulting, Am Technologiepark 1, 45307 Essen, Germany.
2. I am a graduate from Johann Wolfgang Goethe University of Frankfurt am Main with a Dipl. in Geology and Palaeontology, (1986).
3. I am registered as EurGeol 787 as member of the European Federation of Geologists and which is a 'Recognised Professional Organisation' (RPO) included in a list promulgated by TSX from time to time.
4. I have more than 30 years of experience as an exploration and resource geologist in the evaluation of mineral projects for all levels of feasibility and due diligence status, dealing with a variety of commodities, viz. metals (ferrous, base and precious metals), energy (shale oil, lignite and hard coal), pegmatite exploration and deposit evaluation in deferent project stages.
5. I have read the definitions of "qualified person" as set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfil the requirements to be a qualified person for the purpose of NI43-101.
6. I visited the Georgia Lake project site on October 24th, 25th and 26th, 2017. I am responsible for all Sections of this report.
7. I have not had any involvement with the property that is the subject of the Mineral Resource Report prior to our engagement as geological consultants on technical matters, the results of which form part of the Mineral Resource Report. We hold no direct investment interest in Rock Tech Lithium Inc..
8. I am independent of the issuer as described in Section 1.5 of NI 43-101.
9. I have read the NI 43-101, Form 43-101F1, and CIM Standards on Mineral Resources and Reserves, and this technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the date of this Certificate, to the best of my knowledge, information and belief, this report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 27th day of June 2018.

Karl-Stephan Peters

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I, Florian Lowicki, do hereby certify, with respect to the Technical Report dated effective 18th April 2018 entitled "NI 43-101 technical report mineral resource estimation Upgrade 2018 Georgia Lake Lithium Properties Beardmore, Ontario, Canada" that:

1. I am a Resource Geologist for DMT GmbH & Co. KG, Department Mining Consulting, Am Technologiepark 1, 45307 Essen, Germany.

2. I am a graduate from Technische Universität of Berlin with a Dipl.-Ing. in Economic Geology and Applied Geochemistry.

3. I am a Member of the South African Council of Natural Scientific Professionals (SACNASP) which is a 'Recognised Professional Organisation' (RPO) included in a list promulgated by TSX from time to time.

4. I have 13 years of experience as a resource geologist in the evaluation of mineral projects for all levels of feasibility and due diligence status, dealing with a variety of commodities, viz. metals (ferrous, base and precious metals), energy (shale oil, tight gas, lignite and coal), heavy mineral sands and bulk minerals (phosphates).

5. I have read the definitions of "qualified person" as set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfil the requirements to be a qualified person for the purpose of NI43-101.

6. I have not visited the project site. I am responsible for Chapter 12 and 14 of the Technical Report, which I have prepared and to which this Consent Statement applies.

7. I have not had any involvement with the property that is the subject of the Mineral Resource Report prior to our engagement as geological consultants on technical matters, the results of which form part of the Mineral Resource Report. We hold no direct investment interest in Rock Tech Lithium.

8. I am independent of the issuer as described in Section 1.5 of NI 43-101.

9. I have read the NI 43-101, Form 43-101F1, and CIM Standards on Mineral Resources and Reserves, and this technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. As of the date of this Certificate, to the best of my knowledge, information and belief, this report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 27th day of June 2018.

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Glossary of Terms

Abbreviation	Meaning
3D	Three dimensional
%	Percentage
Al	Aluminium
asl	Above sea level
CCIC	Caracle Creek International Consulting Inc.
con	concentrate (“con”)
CON	Conway pegmatite
Cs	Cesium
DMT	DMT GmbH & Co.KG
ha	Hectare 1 ha = 2,47 acre
HAR	Harricana pegmatite
Kg	Kilo (1000 grams)
km²	Square kilometres (metric)
LCT-pegmatites	Lithium-Cesium-Tantalum-pegmatites
Li	Lithium
LIN	Line 60 Pegmatite
m	Meter
Mt	Million tons (metric)
MZN	Main Zone North pegmatite
MZSW	Main Zone South West pegmatite
NI 43-101	National Instrument 43-101 Standard of Disclosure for Mineral Projects, Canada
NSPA	Northern Spodumene Pegmatite Area
NQ drill holes	drill holes with diameter: outside 75,7 mm; inside 47,6 mm
O	Oxygen
mm	Millimetre
MNDM	Ministry of Northern Development and Mines, Ontario
Rock Tech	Rock Tech Lithium Inc.
SG	Specific gravity
SGS	SGS Group in former times *Société Générale de Surveillance*
SSPA	Southern Spodumene Pegmatite Area
t/m³	tons (1000 kg) per cubic meters
Ta	Tantalum

1 SUMMARY

DMT GmbH & Co. KG ("DMT") of Essen, North-Rhine Westphalia, Germany was contracted by Rock Tech Lithium Inc. ("Rock Tech") of Vancouver, British Columbia, Canada, to review the Georgia Lake Lithium Project (the "Property"), and prepare an Independent Technical Report (the "Report"), compliant with National Instrument 43-101 ("NI43-101"), companion policy NI43-101CP and Form 43-101F1.

The purpose of this Technical Report is to update the resource estimate since the last NI43-101 Report (2012) from CARACLE CREEK INTERNATIONAL CONSULTING INC. (“CCIC”) in Aug. 2012. New drilling and channel sampling exploration activities on the property were executed. In general, there are several areas of lithium pegmatites in the claims visible at the surface with outcrops. In the northern parts there are 5 pegmatites with thicknesses up to 10 m and a strike length up to 1,5 km. For these 5 pegmatites 3D models were built. A block model after data validation was created, which was also validated. Measured, indicated and inferred resources were defined. From areas around the modelled pegmatites additional inferred resources were estimated. In the areas more south of these main pegmatites several other spodumene pegmatites occur within licences belonging to Rock Tech. In these areas also drilling and channel sampling has taken place and additional inferred resources were estimated.

The five main pegmatites, for which a 3D block model was created are called:

- Nama Creek Main Zone North (“MZN”)
- Nama Creek Main Zone South-West (“MZSW”)
- Harricana (“HAR”)
- Line 60 (“LIN”)
- Conway (“CON”)

All the exploration claims and dispositions are located approximately 160 km northeast of Thunder Bay within the Thunder Bay Mining Division located in the NTS sheets 42E05NW and 52H08NE (see Table 2).

On the 10th of April 2018 the Ministry of Northern Development and Mines changed the general framework on the layout of exploration claims (OMNDM 2018). On that date, claim staking using physical stakes was stopped and mining claims are defined by their positions as shown on the provincial grid. Therefore, all numbering of claims changed. In Appendix 3 a table shows old and new claim numbers. All the figures in this report show the new claim and disposition numbers and shapes. The number of the exploration claims changed but the area of land is largely similar to the former layout. All exploration work and targets are still within the existing exploration claims of Rock Tech Inc. A list of all exploration claims and dispositions is shown in chapter 4.

In total 283 exploration claims with a combined surface area of 5,693 ha (56.93 km²) are licenced to Rock Tech Lithium. The dispositions consist of 6 blocks and comprise a total area of 1,042 ha (10.42 km^2) see Figure 2 and Table 3.

The Nama Creek dispositions consist of 36 contiguous dispositions and a total area of 336.410 ha. The McVittie dispositions consist of 6 contiguous dispositions and a total area of 87.639 ha. The Parole Lake dispositions consist of 25 contiguous dispositions and a total area of 385.621 ha. The Foster-Lew dispositions consist of 4 contiguous dispositions and a total area of 75.231 ha. The Newkirk-Vegan dispositions consist of 8 contiguous dispositions and a total area of 120.633 ha. The MNW dispositions consist of 2 contiguous dispositions and a total area of 44.248 ha. Rock Tech is the owner of the mining and surface rights of the Nama Creek dispositions. James Bay Midarctic Developments Inc. is the owner of the mining rights of all the other dispositions (i.e., McVittie, Parole Lake, Foster-Lew, Newkirk-Vegan and MNW). James Bay Midarctic Developments Inc. is a subsidiary of Rock Tech (100%). The surface rights are owned by the Crown for all dispositions except for Nama Creek.

Lithium was first discovered in the Georgia Lake pegmatite in 1955. The Georgia Lake area is located within the Quetico Subprovince of the Superior Province. Lithium occurs as spodumene in pegmatites in the Georgia Lake area. The pegmatites are hosted by metasediments. All of the pegmatites are albite-spodumene type. Spodumene is the dominant Li-bearing mineral in all of the pegmatites.

Overall, the pegmatite dyke internal zonation increases in complexity from north to south within the Georgia Lake pegmatite field:

- Nama Creek - MZN and MZSW have simple zonation: aplite or granitic border zone and a spodumene zone with minor alternating aplite + pegmatite layers.
- Harricana, Line 60 and Conway have aplite or granitic border zone, a spodumene zone and common alternating aplite + pegmatite layers.

The mineralization in the Georgia Lake Lithium Property consists of coarse-grained fresh pale green spodumene crystals, oriented perpendicular to the strike of the pegmatite dyke in homogeneous dykes, and randomly oriented within the inner spodumene zone in simply zoned pegmatite dykes. The spodumene may be altered to muscovite or fine-grained muscovite near the contacts with the host rocks and near diabase dykes. The altered spodumene has low Li contents and a high iron content.

The deposit model is well-described in the literature and ascribes the spodumene mineralization with Li-Cs-Ta ("LCT") rare-element pegmatite dykes. LCT pegmatites are associated with S-type, peraluminous (Al-rich), quartz-rich granites. S-type granites crystallize from a magma produced by partial melting of pre-existing sedimentary source rock. They are characterized by the presence of biotite and muscovite, and absence of hornblende. Rare-element pegmatites derived from a fertile granite intrusion are typically distributed over a 10 to 20 km^2 area within 10 km of the fertile granite. A fertile granite is the parental granite to rare-element pegmatite dykes.

Historic bulk samples for processing tests were composited from drill core and drill blast samples from the MZN and MZSW. These were assessed as representative for the style and type of mineralization and the mineral deposit as a whole. SGS Metallurgical Opera-

tions, Lakefield, Ontario received in February 2011 and subsequently processed the samples. Resulting concentrates were processed through the standard lithium carbonate hydrometallurgical flow sheet showing that a high grade Li_2CO3 product could be produced from Rock Tech's mineralized samples. DMT notes that these samples yielded a head grade of 1.49 % Li_2O, which is higher than the average grade of the new resource estimate.

It is documented that in 1956 Nama Creek Mines Ltd. sank a vertical shaft to initiate mining on the Nama Creek Property (Pye, 1965). However, production is not documented.

Trow Associates Inc. (2011) conducted a baseline ecological study for the Georgia Lake Lithium Property in 2010. Environmental data were collected to document the baseline conditions before commencement of any mining activitiy. In light of this, attempts were made to collect water samples at pre-selected locations situated upstream, within and downstream of proposed drilling (and potential future extraction) sites. Twenty-four sample stations were established and sampling was conducted at sites associated, to some degree, with all claim blocks. Field work which consisted of surface water sampling was conducted from November 22 to 26, 2010.

Previous independent, NI 43-101 compliant resources of the MZN, NMZSW, Harricana, Line 60 and Conway were prepared by CCIC in 2012 (CCIC, Selway et all (2012).

DMT prepared an independent, NI 43-101 compliant resource estimate for the Georgia Lake Lithium Property within all claims and dispositions which belongs to Rock Tech today. The total resource estimate described in this report is based on different areas within the claims and dispositions of Rock Tech. For this report DMT divided the claim and disposition area from Rock Tech in 2 Areas (north and south). The Northern Spodumene Pegmatite Area "NSPA" and the Southern Spodumene Pegmatite Area "SSPA" see Figure 2. All mentioned pegmatites in this report are situated with in the claims and dispositions of Rock Tech. The mineral resource update from DMT has an effective date of the 18th of April, 2018

DMT constructed 3D models for MZN, MZSW, Harricana, Line 60 and Conway spodumene pegmatites using historical drill hole data as well as results acquired since November 2009 up to date. 3D wireframes (solids) representing the mineralized areas within the spodumene pegmatite were constructed and used to constrain the tonnage and grade estimation. GEMCOM's Geovia software V.7.2 was used to generate the 3D model and perform the grade estimation. Grades for Li_2O were estimated using the anisotropic ordinary kriging method. An average specific gravity ("SG") of 2.71 [t/m³] was applied using a regression equation based on densities measured by Rock Tech.

Five of the pegmatites, for which 3D models have been constructed, are located in the NSPA. From these models measured and indicated resources have been defined. Additional inferred resources have been estimated in the NSPA at depth, where drill core intersection density is too low.

Additional inferred resources within the properties of Rock Tech Inc. have been estimated for the pegmatites in the SSPA. In these exploration claims and dispositions spodumene pegmatites occur at the surface and also drilling and channel sampling has taken place in

these areas. Ongoing exploration activities will help to update these resources into higher confidence levels in the future.

The main resources are located in the northern part, where up to date most exploration work has been conducted. In this area for five pegmatites 3D models were created. These models were necessary to define for these pegmatites the measured and indicated resources. The mineral resource estimates of the 3D model for the MZN, MZSW, HAR, LIN, CON spodumene pegmatites presented below.

Following NI 43-101 requirements and considering potential economic viability a 0.65 % Li_2O cut-off grade has been assumed based on comparable projects, which yields a total Measured and Indicated Resource of 6.58 Mt at a grade of 1.01 % Li_2O (see data in Table 1 and Table 37 to Table 39)

Table 1. Measured + Indicated Resource in the NSPA from all 5 pegmatites which were modelled in 3D (see Figure 32)

Table 1	Type of Resources	Tonnage [Mt]	Li²O [%]	Cut off Li²O [%]
NSPA	Measured	1.89	1.04	0.65
NSPA	Indicated	4.68	1.00	0.65
TOTAL	**Measured and Indicated**	**6.58**	**1.01**	**0.65**

It should be noted that mineral resources are not mineral reserves. Investigations on modifying factors are recommended to convert measured and indicated mineral resources to mineral reserves of certain degree of economic/technical feasibility.

In addition to these measured and indicated resources inferred resources were estimated from different areas within the claims of Rock Tech. Within the block model are **0.33 Mt** inferred resources are estimated. Outside the block model, there are additional inferred resources of **3.85 Mt** derived by applying an average true thickness of spodumene pegmatite and extrapolating the pegmatite bodies maximal 50 m below the 3D model. Further additional inferred resources have been derived from all drilling and trenching on current claims and dispositions in the SSPA. This additional inferred resource from the SSPA are estimated with **2.54 Mt.** The total inferred recourses in the claims and dispositions of Rock Tech are **6.72 Mt.** For all of these estimated inferred resources the cut off of 0.65 % Li_2O was used.

Table 2. List of the total inferred resource within the claims and dispositions of Rock Teck from different areas with the effective date 18.April.2018

AREA	Type of Resources	Tonnage [Mt]	Li^2O [%]	Approach of resource estimation
NSPA	Inferred	0.33	1.01	From the 3D Block models pegmatites in the NSPA
NSPA	Inferred	3.85	1.01	From extrapolating below the existing 3D modelled pegmatites
SSPA	Inferred	2.54	1.41	From pegmatites of different areas in the SSPA
TOTAL	Inferred	**6.72**	**1.16**	

In all claims and dispositions of Rock Tech situated in the Georgia Lake area 6.58 Mt measured and indicated resources are estimated. Additionally 6.72 Mt of inferred resources were estimated within this area.

2 INTRODUCTION

DMT GmbH & Co. KG ("DMT") of Essen, North-Rhine Westphalia, Germany was contracted by Rock Tech Lithium Inc. ("Rock Tech") of Vancouver, British Columbia, Canada, to review the Georgia Lake Lithium Project (the "Property"), and prepare an Independent Technical Report (the "Report"), compliant with National Instrument 43-101 ("NI43-101"), companion policy NI43-101CP and Form 43-101F1 to update the mineral resources based on latest data acquired after CCIC had prepared two technical reports on resource estimates in 2012.

For this report all actual claims and licenses, which belong to Rock Tech were used for the resource statement. K.-S. Peters, as qualified person, visited the project verifying results, operation procedures and the execution of the work in the field. Since the last report from “CCIC” (2012) exploration was still ongoing and more drill holes and channel samples were available for the updating of the resource estimate. A new 3D model were generated and all validation work was done.

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 DMT GmbH & Co. KG best judgement in light of the information available.

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by DMT, for the client.

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

- Information available to DMT at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by the client and other third party sources, e.g. contracted consultant company Arriva Management Inc. Vancouver.
- Discussions with representatives from the client who are familiar with the Property and the area in general
- DMT has assumed that the reports and other data listed in the "References" section and the Appendix 2 of this report are substantially accurate and complete.

For the purpose of this report, DMT has relied on ownership information provided by the client. The public source of information regarding land tenure is the MNDMF website (MNDMF website: http://www.mndm.gov.on.ca). DMT has not researched property title or mineral rights for the project and expresses no opinion as to the legal ownership status of the property. DMT has checked the website of the Government of Ontario, which shows the legal status of the claims as here reported. In according with the governmental rules Rock-Tech paid all necessary fees for the government to hold the licences. All paper work was done on time for all the exploration permits and the mining dispositions.

The dates, titles and authors of all reports that were used as a source of information for this Technical Report are listed in the "References" section of this report and Appendix 2. The dates and authors of these reports also appear in the text of this Report where relevant, indicating the extent of the reliance on these reports.

The information contained in this report with respect to the Mineral Resources is based on information provided by RockTech and reviewed by DMT in 2017/18. Whilst exercising all reasonable diligence in checking, confirming and testing this information, DMT have assumed that the data presented by RockTech is reasonable in formulating its opinion.

This report includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, DMT do not consider them to be material.

The various agreements under which RockTech hold title to the mineral properties for this project have neither been investigated nor confirmed by DMT and DMT offer no opinion as to the validity of the mineral title claimed by Rock Tech. The description of the property, and ownership thereof, as set out in this report, is provided for general information purposes only.

4 PROPERTY DESCRIPTION AND LOCATION

The Georgia Lake Property is located approximately 160 km northeast of Thunder Bay within the Thunder Bay Mining Division in NTS sheets 42E05NW and 52H08NE. Rock Tech's current claims and dispositions are shown in Figure 2

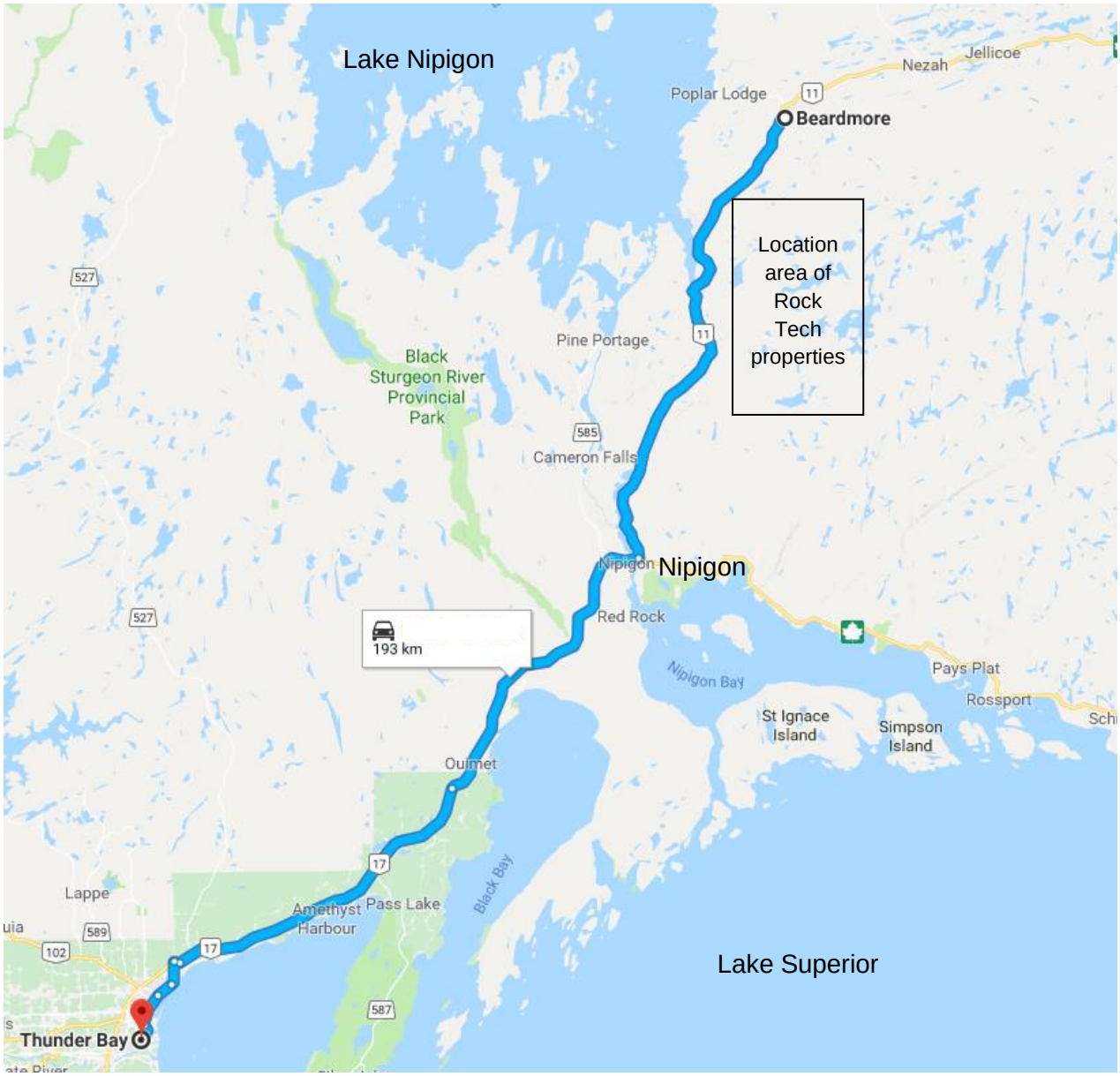


Figure 1. Location map of the Rock Tech properties and highway 11 as access road from the town of Thunder Bay (Source: Google Maps).

The blue line shows the way from Thunder Bay into the area along the Highway Nr.11. The town Nipigon on the norther side of the Lake Superior and Lake Nipigon North West of the investigation area Figure 2. In Figure 2 the Highway 11 is running west of the claim in the east side of the Lake Nipigon to the north.

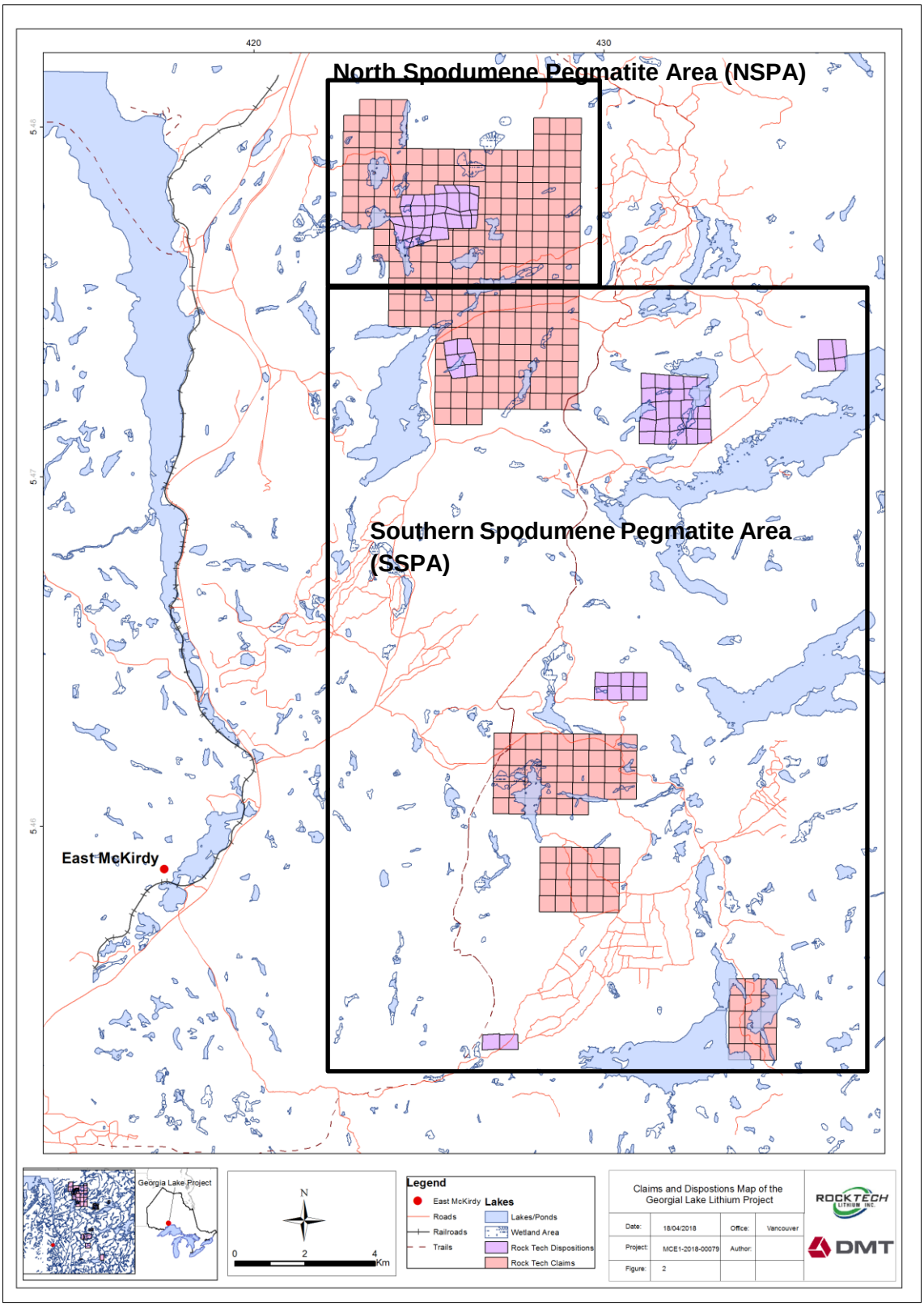


Figure 2. Location of exploration claims (red) and dispositions (magenta) currently held directly by Rock Tech Inc. and/or by its subsidiary James Bay Midarctic Developments Inc. (JB), and the location of the two areas NSPA and SSPA.

4.1 DESCRIPTION OF OWNERSHIP

The Georgia Lake Property consists of 283 exploration claims (Table 3) and 81 dispositions (Table 4 to Table 9). The boundaries of the claims and dispositions are shown in Figure 2. The total area of the claims is 5,693 ha (56.93 km²) and dispositions comprise a total area of 1,042 ha (10.42 km$^{2)}$.

All claims for the mining rights are owned 100% by Rock Tech. Rock Tech or its subsidiary JB. For the Nama Creek dispositions Rock Tech holds also the surface rights. The surface of all the other areas belong to the Crown. Rock Tech has legal access to all of its claims. From the web side of the MNDMF's Geoscience Assessment Office in Ontario, the data for the Table 2 to Table 9 were checked and collected. The claim due date is listed in the tables and shows no gaps. The due dates for the claims range from May 2019 until Dec. 2021 and for dispositions until Jan. 2031 or Jan. 2033.

Table 3. Rock Tech's Georgia Lake Lithium Property exploration claims (see Figure 2)

New Claim Number	Township/Area	Recorded Holder	Claim Due Date	Cell claim Type
344757	Kilkenny	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
334733	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
331861	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
327329	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
314571	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
314570	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
308917	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
305332	Kilkenny	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
296585	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
293192	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
293191	Kilkenny	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
288479	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
288478	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
284578	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
278007	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
277987	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
260008	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
260007	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
258045	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
258032	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
257144	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
254946	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
244210	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
239854	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
239853	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
231904	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
230728	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
224064	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
222058	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
218621	Kilkenny	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
218620	Kilkenny	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
212013	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
210049	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
210027	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
207141	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
207140	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell

New Claim Number	Township/Area	Recorded Holder	Claim Due Date	Cell claim Type
203998	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
194468	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
187622	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
184915	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
184914	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
184064	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
180171	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
172048	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
169789	Kilkenny	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
165287	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
157397	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
157396	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
146069	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
145282	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
141079	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
139846	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
132071	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
132070	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
128670	Kilkenny	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
128669	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
125308	Kilkenny	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
344793	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
344758	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
342241	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
342240	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
340997	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
340996	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
339163	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
339162	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
335846	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
335845	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
335844	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
334731	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
333104	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
332147	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
331609	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
328730	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
328711	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
327111	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
326591	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
323418	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
323417	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
323416	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
320172	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
313831	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
313830	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
312610	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
310367	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
309239	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
307702	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
305793	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
305792	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
305369	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
304416	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
303480	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
303459	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
303458	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
300299	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
293232	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
293190	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
291323	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
285135	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell

New Claim Number	Township/Area	Recorded Holder	Claim Due Date	Cell claim Type
275993	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
273068	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
266435	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
263715	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
258772	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Boundary Cell
258771	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Boundary Cell
258678	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Boundary Cell
257142	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
257141	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
255987	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
255953	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
255225	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
254950	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
253549	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
253224	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
248431	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
248430	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
247210	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
246696	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
244181	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
240377	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
239838	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
239837	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
239836	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
236143	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
236125	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
235294	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
234029	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
226614	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
225448	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
225352	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
225351	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
222709	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Boundary Cell
222708	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
218658	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
218657	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
218619	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
216677	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
215448	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
215447	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
211252	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
208593	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
207150	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
205988	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
204435	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
202635	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Boundary Cell
200580	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
199694	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
197101	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
197074	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
189326	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
189325	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
189324	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
189290	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
187449	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
187448	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
186743	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
183301	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
183300	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
183299	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
180689	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
180688	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell

New Claim Number	Township/Area	Recorded Holder	Claim Due Date	Cell claim Type
179970	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
177563	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
177537	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
177536	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
177535	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
169790	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
165473	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
158092	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
158011	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Boundary Cell
157382	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
155959	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
155958	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
153229	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
153196	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
153195	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
150200	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
148976	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
148959	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
148958	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
145105	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
141545	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
141085	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
138507	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
137835	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
137312	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
137311	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
135509	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
135508	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
135487	Lake Jean Area	ROCK TECH LITHIUM INC.	2022-AUG-23	Single Cell
135004	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
134754	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
123503	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
122741	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
122214	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
120898	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
120897	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
112545	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
112402	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-AUG-23	Single Cell
111996	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-AUG-23	Single Cell
111427	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
110212	Lake Jean Area	ROCK TECH LITHIUM INC.	2020-DEC-09	Single Cell
109132	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
106179	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
106178	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
106177	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-DEC-09	Single Cell
105386	Lake Jean Area	ROCK TECH LITHIUM INC.	2021-May-18	Single Cell
340907	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
336485	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
330035	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
330034	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
323525	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
318849	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
318123	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
318122	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
318121	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
318120	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
302202	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
296171	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
281965	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
281964	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
281963	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
252231	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell

New Claim Number	Township/Area	Recorded Holder	Claim Due Date	Cell claim Type
245372	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
233958	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
233225	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
233224	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
232827	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
227673	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-09	Single Cell
225207	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
211644	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
208806	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
200775	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
194433	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
192885	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
186148	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
185423	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
179398	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
178633	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
178632	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
178631	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
174909	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
171517	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-09	Single Cell
166643	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
166642	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
166641	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
162012	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
150096	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
148869	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
134146	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
134145	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
134144	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
134143	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
133440	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
133439	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
133438	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
121403	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
121402	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
115825	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-09	Single Cell
114362	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
106704	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
106094	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Boundary Cell
106093	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
106092	Barbara Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
330036	Keemile Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
232828	Keemile Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
216184	Keemile Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
196118	Keemile Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
196117	Keemile Lake Area	ROCK TECH LITHIUM INC.	2019-AUG-23	Single Cell
336306	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
308691	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
296565	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
296564	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
288460	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
248844	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
248843	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
248842	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
229857	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
229856	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
192621	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
192620	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
162596	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
140097	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell
140096	Cosgrave Lake Area	ROCK TECH LITHIUM INC.	2019-MAY-18	Single Cell

The dispositions (leases) consist of 6 blocks and comprise a total area of 1,042.43 ha (10.43 km²). All of the disposition blocks are contiguous with the mining claims, but are not contiguous with each other. The Nama Creek dispositions consist of 36 contiguous dispositions and a total area of 329.065 ha (3.29 km², **Figure 3** and **Table 4**). The renewal of Nama Creek dispositions is finished now and run until 2031 and 2033 according to MNDM. The McVittie dispositions consist of 6 contiguous dispositions and a total area of 87.639 ha (0.87 km², **Figure 3** and **Table 5**). The Parole Lake dispositions consist of 25 contiguous dispositions and a total area of 385.621 ha (3.85km², **Figure 4** and **Table 6**). The Foster-Lew dispositions consist of 4 contiguous dispositions and a total area of 75.231ha (0.75 km **Figure 4** and **Table 7**). The Newkirk-Vegan dispositions consist of 8 contiguous dispositions and a total area of 120.63 ha (1.20 km², Figure 5 and **Table 8**). The MNW dispositions consist of 2 contiguous dispositions and a total area of 44.248 ha (0.44 km², Figure 5 and **Table 9**).

Rock Tech is the owner of the mining and surface rights on the Nama Creek dispositions. James Bay Midarctic Developments Inc. is the owner of the mining rights only on all of the other dispositions (i.e., McVittie, Parole Lake, Foster-Lew, Newkirk-Vegan and MNW). James Bay Midarctic Developments Inc. is a subsidiary of Rock see report from 2012 CCIC (Selway, J., et. al. (2012b). The surface rights are owned by the Crown for all dispositions except for Nama Creek. The surface rights of the Nama Creek dispositions belong to Rock Tech. Rock Tech has legal access to all of its dispositions. All of Rock Tech's leases are for a 21 year term. In Ontario the Ministry of Northern Development and Mines is responsible for the claims and disposals (web site: https://www.mndm.gov.on.ca/en).

The following tables and Figures show the list of leases according to the area were they are situated.

Table 4. Dispositions (leases) for Rock Tech's Nama Creek Property

Mining Right Numbe Newr	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108972	TB67132	Active	Lease	2033-Jan-31	108972	Kilkenny	
LEA-108973	TB67133	Active	Lease	2033-Jan-31	108973	Kilkenny	
LEA-108974	TB67134	Active	Lease	2033-Jan-31	108974	Kilkenny	
LEA-108975	TB67135	Active	Lease	2033-Jan-31	108975	Kilkenny	
LEA-108976	TB67136	Active	Lease	2033-Jan-31	108976	Kilkenny	
LEA-108977	TB67137	Active	Lease	2033-Jan-31	108977	Kilkenny	
LEA-108978	TB67138	Active	Lease	2033-Jan-31	108978	Kilkenny	
LEA-108979	TB67139	Active	Lease	2033-Jan-31	108979	Kilkenny	
LEA-108980	TB67140	Active	Lease	2033-Jan-31	108980	Kilkenny	
LEA-108981	TB67157	Active	Lease	2033-Jan-31	108981	Kilkenny	
LEA-108982	TB67160	Active	Lease	2033-Jan-31	108982	Kilkenny	
LEA-108983	TB67162	Active	Lease	2033-Jan-31	108983	Kilkenny	
LEA-108984	TB67163	Active	Lease	2033-Jan-31	108984	Kilkenny	
LEA-108985	TB67164	Active	Lease	2033-Jan-31	108985	Kilkenny	
LEA-108986	TB67167	Active	Lease	2033-Jan-31	108986	Kilkenny	
LEA-108987	TB67177	Active	Lease	2033-Jan-31	108987	Kilkenny	
LEA-108988	TB67178	Active	Lease	2033-Jan-31	108988	Kilkenny	
LEA-108989	TB67180	Active	Lease	2033-Jan-31	108989	Kilkenny	
LEA-108990	TB67181	Active	Lease	2033-Jan-31	108990	Kilkenny	
LEA-108991	TB67145	Active	Lease	2033-Jan-31	108991	Kilkenny	
LEA-108992	TB67146	Active	Lease	2033-Jan-31	108992	Kilkenny	
LEA-108993	TB67147	Active	Lease	2033-Jan-31	108993	Kilkenny	
LEA-108994	TB67154	Active	Lease	2033-Jan-31	108994	Kilkenny	
LEA-108995	TB67165	Active	Lease	2033-Jan-31	108995	Kilkenny	
LEA-108996	TB67166	Active	Lease	2033-Jan-31	108996	Kilkenny	

Mining Right Numbe Newr	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108997	TB67158	Active	Lease	2033-Jan-31	108997	Kilkenny	
LEA-108998	TB67168	Active	Lease	2033-Jan-31	108998	Kilkenny	
LEA-108999	TB67169	Active	Lease	2033-Jan-31	108999	Kilkenny	
LEA-109000	TB67170	Active	Lease	2033-Jan-31	109000	Kilkenny	
LEA-109001	TB67171	Active	Lease	2033-Jan-31	109001	Kilkenny	
LEA-109002	TB67172	Active	Lease	2033-Jan-31	109002	Kilkenny	
LEA-109003	TB67173	Active	Lease	2033-Jan-31	109003	Kilkenny	
LEA-109004	TB67174	Active	Lease	2033-Jan-31	109004	Kilkenny	
LEA-109005	TB67175	Active	Lease	2033-Jan-31	109005	Kilkenny	
LEA-109006	TB67176	Active	Lease	2033-Jan-31	109006	Kilkenny	
LEA-109007	TB67185	Active	Lease	2033-Jan-31	109007	Kilkenny	
						total	329.065

Table 5. Dispositions (leases) for Rock Tech's McVittie property

Mining Right Number	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108505	TB732171	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732172	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732173	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732174	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732175	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
LEA-108505	TB732176	Active	Lease	2031-May-31	108505	Pijitawabik Bay Area	
						total	87.639

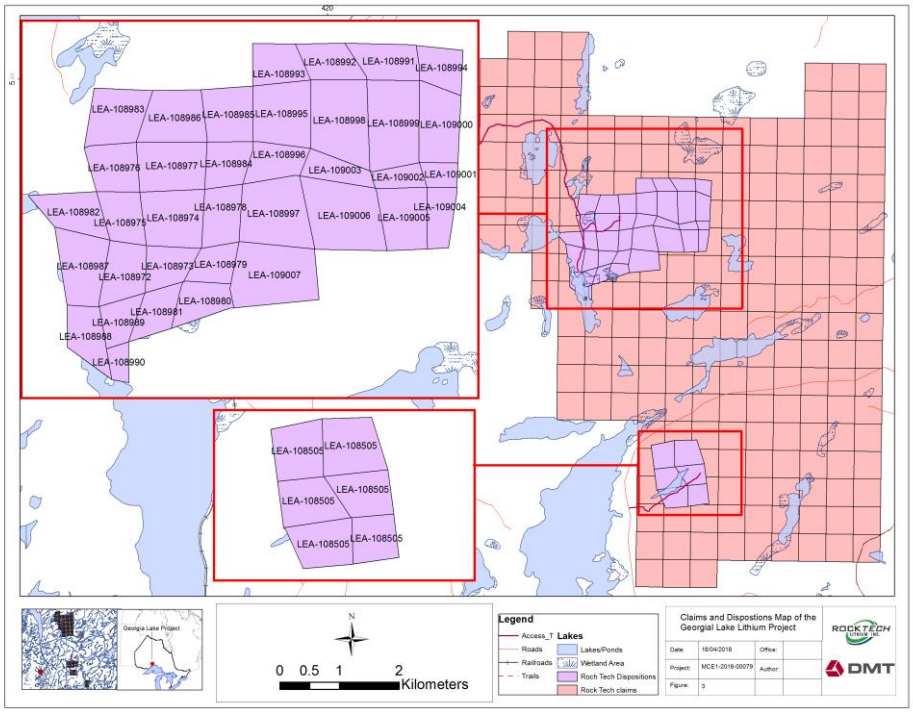


Figure 3. Location of claims and dispositions for Rock Tech's Nama Creek, Conway and McVittie properties.

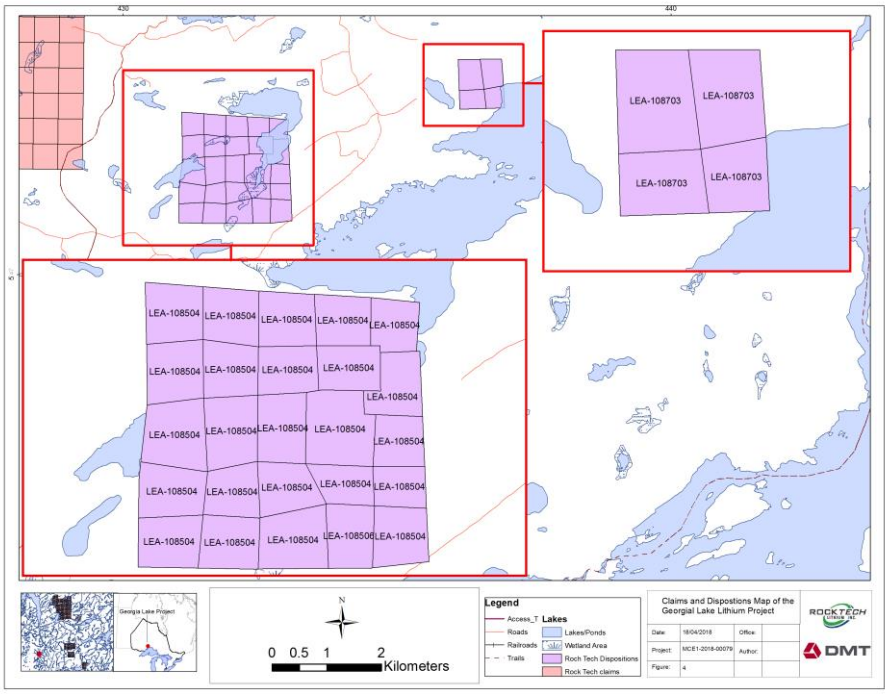


Figure 4. Location of claims and dispositions for Rock Tech's Parole Lake and Foster-Lew properties.

Table 6. Dispositions (leases) for Rock Tech's Parole Lake property

Mining Right Number	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108504	TB1009072	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB1009073	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB1020967	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756468	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756469	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756470	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756471	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756472	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756473	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756474	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756476	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756477	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756478	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756479	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756480	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756481	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756497	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756498	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756499	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108506	TB756500	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756501	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756536	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756537	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756538	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108504	TB756539	Active	Lease	2031-May-31	108504	Lake Jean Area	
LEA-108703	TB1005886	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005887	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005888	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005889	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
						total	385.621

Table 7. Dispositions (leases) for Rock Tech's Foster Lew property

Mining Right Number	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108703	TB1005886	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005887	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005888	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
LEA-108703	TB1005889	Active	Lease	2032-Jan-31	108703	Lake Jean Area	
						total	75.231

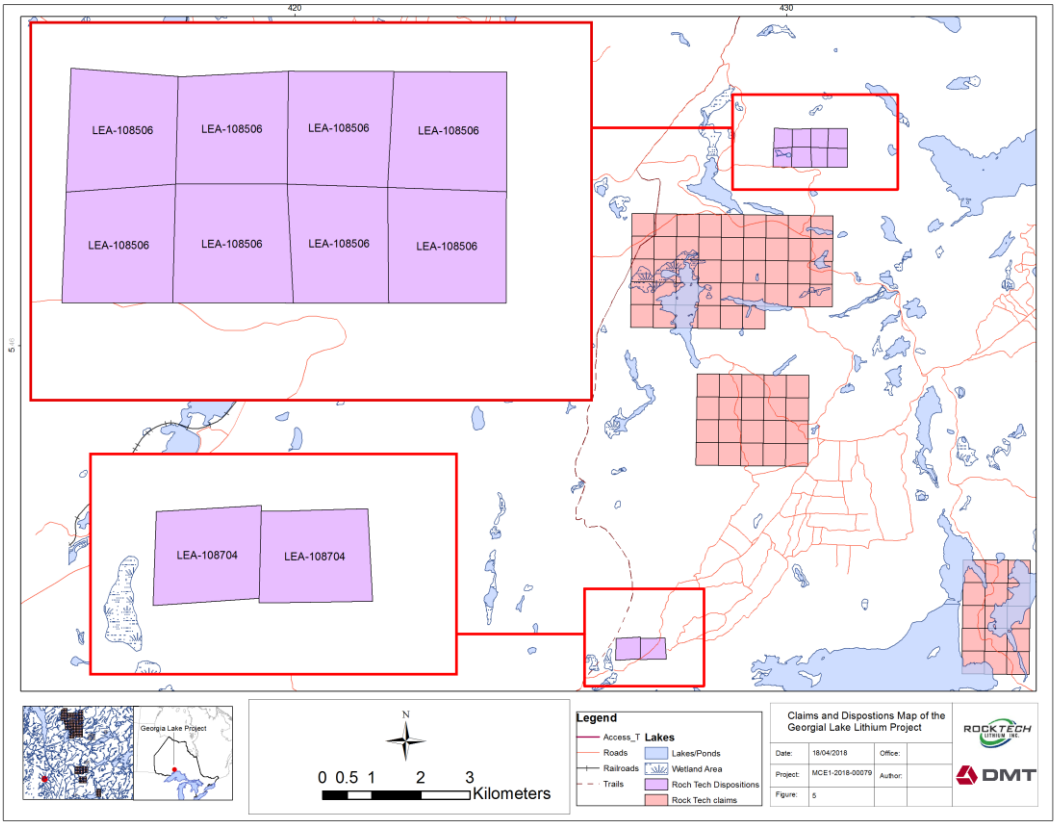


Figure 5. Location of claims and dispositions for Rock Tech's Aumacho, Newkirk-Vegan and MNW properties

Table 8. Dispositions (leases) for Rock Tech's Newkirk-Vegan property

Mining Right Number	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha)
LEA-108506	TB824969	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824970	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824971	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824972	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824973	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB824974	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB863418	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
LEA-108506	TB863419	Active	Lease	2031-Apr-30	108506	Barbara Lake Area	
						total	120.633

Table 9. Dispositions (leases) for Rock Tech's MNW property

Mining Right Number	short legal Discription	Status	Type	Expiry Date	Lease/Lic	Township/Area	Area (ha
LEA-108704	TB863303	Active	Lease	2032-Jan-31	108704	Hanson Lake Area	
LEA-108704	TB863304	Active	Lease	2032-Jan-31	108704	Hanson Lake Area	
						total	44.248

Other stakeholders:

DMT did not verify in detail, if there are other rights / surface rights existing in the areas of interest. Known from other reports is:

In the Report from 2012 CCIC (Selway, J., et. al. (2012b)) reported that there are 6 alienations on the Georgia Lake Property type notice, class wind power: WP2005-12, WP2006-21, WP2006-21, WP2008-121, WP2008-154 and WP2008-359. These alienations are for surface rights only and the area is available for staking. There is one alienation withdraw order W-TB-139/11 which withdraws surface rights from staking while the construction site of a Transmission Line for waterpower development project is under review.

The location of all known mineralized zones is described in chapter 7.3. Property Geology and in the CCIC report 2012 (Selway, J., et. al. (2012b). There are no mine workings, tailings ponds, and waste deposits on the Georgia Lake Lithium Property, except for a historic mine shaft on the MZN pegmatite on the Nama Creek disposition LEA – 108977 (old number TB67137). The shaft was built in 1956 by Nama Creek Mines Ltd.. To the best of DMT's knowledge, there are no known royalties, back-in rights, payments and other agreements and encumbrances on the Georgia Lake Lithium Property other than the agreement that James Bay Midarctic Developments Inc. is a subsidiary of Rock Tech. To the best of DMT's knowledge, there are no environmental liabilities on the Georgia Lake Lithium Property.

To the best of DMT's knowledge there are no significant factors or risks that may affect access, title or the right or ability to perform work on the property, other than the requirement to obtain permits for temporary bridges to cross small creeks to drive to necessary properties.

In Figure 6 shows in detail the NSPA in the northern part of the whole property (see Figure 2). Location of exploration claims (red) and dispositions (magenta) currently held by Rock Tech and subsidiaries. The dots show the location of the drill holes beside the individual pegmatites whish ere modelled in 3D. The 5 pegmatites were the MZN (Nama Creek Main Zone North), MZSW (Nama Creek Main Zone South-West), HAR (Harricana), LIN (line 60), CON (Conway). They are all located in the NSPA. All other pegmatites mentioned in this report are all situated with in the SSPA. This SSPA is situated more east and southwards from the NSPA (see Figure 2).

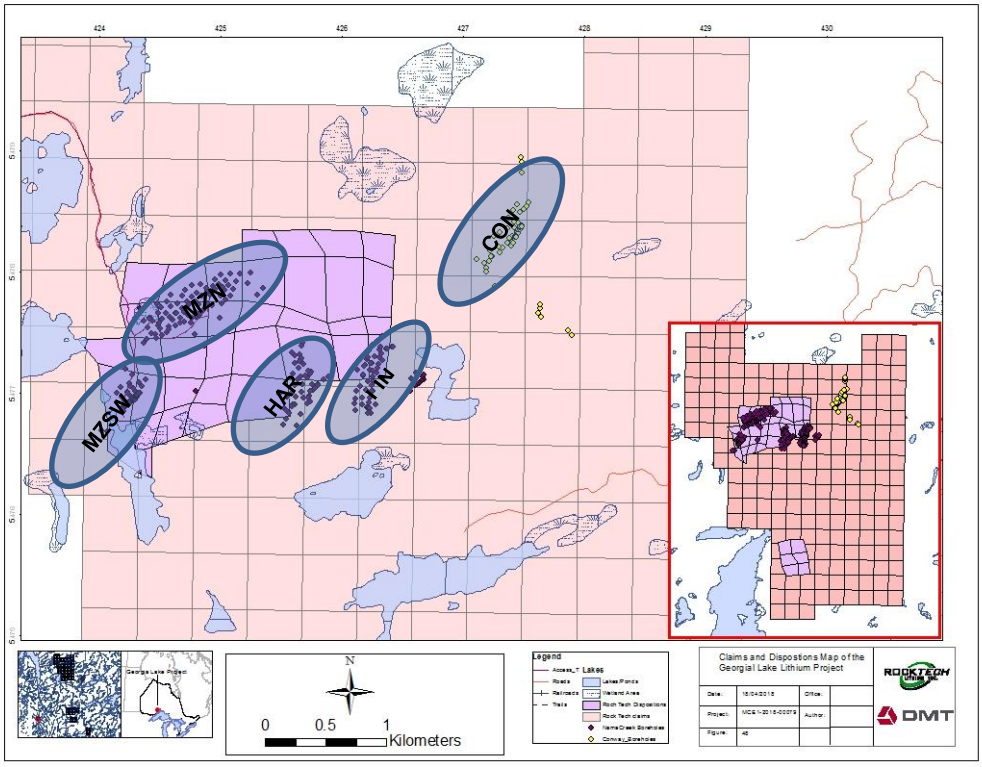


Figure 6. NSPA with the drillhole locations beside the 5 modelled pegmatites. The area and the short cuts of the pegmatites are shown.

The altitude of the drilled areas of relevance for resource model and estimate ranges from 360 m up to 420 m asl.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS

The Georgia Lake Lithium Property can be accessed by dirt roads off Highway 11 north of the town of Nipigon. The closest airport is located in Thunder Bay. The Nama Creek and Conway properties can be accessed by driving 60 km north of the town of Nipigon on Highway 11, then driving approximately 5 km east on a dirt road to reach the western boundary of the claims (see Figure 2).

The McVittie and Jean Lake properties can be accessed by driving 40 km north of the town of Nipigon on Highway 11, then driving approximately 14 km northeast on a dirt road toward Postagoni Lake to reach the area and another 22 km to reach the northern area between Jean Lake and Foster-Lew.

The Aumacho area can be accessed by driving 40 km north of the town of Nipigon on Highway 11, then driving 7 km east on a dirt road to reach the area and another 6 km to reach then the Newkirk-Vegan property. Temporary bridges are needed to drive to the Newkirk-Vegan property.

The MNW property can be accessed by driving 31 km north of the town of Nipigon on Highway 11, then driving approximately 11 km east on a dirt road to reach the eastern boundary of the local claims, but temporary bridges are needed to drive to the property.

5.2 CLIMATE AND VEGETATION

The forest of the Georgia Lake area is mixed growth of spruce, balsam, jackpine, poplar, birch and cedar (Pye, 1965). Vegetation is typical of continental climate a mixture of coniferous (pine and black spruce) and deciduous (primarily birch and minor poplar).

The climate is typical continental with cold and long winters (from November to late March) and significant snow accumulations. The temperature in the winter months (January and February) can reach -40° C but typically ranges between -10° and -25°C. The Canadian Climate normals for 1971-2000 from Environment Canada (/www.climate.weatheroffice.gc.ca/climate_normals/) for Geraldton (closest weather station to the property) indicate that the daily average temperature ranges from -19ºC in January to 17ºC in July. The highest average accumulation of rain for a month is 112 mm in July. The highest average accumulation of snow for a month is 49 cm in November. The highest average snow depth is 48 cm in February.

Drilling can be conducted year round except for spring thaw in mid-March and April. Geological mapping and outcrop sampling can be conducted May to November when there is no snow on the ground.

	January	February	March	April	May	June	July	August	September	October	November	December
Avg. Temperature (°C)	-16.4	-13.4	-7.2	1.5	8.7	13.4	17.1	16.1	10.8	5.2	-2.8	-11.7
Min. Temperature (°C)	-22.4	-19.8	-13.5	-4.5	2.1	7	10.7	10	5.5	0.7	-6.6	-16.5
Max. Temperature (°C)	-10.4	-7	-0.8	7.6	15.3	19.9	23.6	22.2	16.1	9.7	1	-6.8
Avg. Temperature (°F)	2.5	7.9	19.0	34.7	47.7	56.1	62.8	61.0	51.4	41.4	27.0	10.9
Min. Temperature (°F)	-8.3	-3.6	7.7	23.9	35.8	44.6	51.3	50.0	41.9	33.3	20.1	2.3
Max. Temperature (°F)	13.3	19.4	30.6	45.7	59.5	67.8	74.5	72.0	61.0	49.5	33.8	19.8
Precipitation / Rainfall (mm)	48	33	42	40	65	86	80	87	89	77	68	55

Figure 7. Climate data for Nipigon (Source: https://en.climate-data.org/location/767939/).

The climate may be challenge for open cast mining during the winter months.

5.3 PHYSIOGRAPHY

Pye (1965) summarized the topography of the Georgia Lake area:

“The Georgia Lake area is one of topographic contrasts. The parts of the area in which metasediments are exposed are, for the most part, of low relief. In contrast, the parts underlain by granitic rocks are rugged, with rounded hills rising up to about 150 ft (=45.7 m) above the general level. Most conspicuous, however, are high, imposing vertical or near-vertical cliffs at the boundaries of large exposed sheet-like masses of diabase.”

“Rock exposures in the area are abundant, and between the outcrops there is a thin mantle of glacial deposits. These glacial deposits consist mainly of stratified accumulations of unconsolidated sand and gravel. Some of them represent a ground moraine sorted by the action of glacial meltwaters; others form prominent terraces along the shores of Lake Nipigon and in the valley occupied by Keemle and Wanogu Lakes, and are abandoned beach deposits. Esker ridges also are present but are not high and do not extend for any great distances.”

Topography of the Georgia Lake Property is moderate. The minimum elevation is 250 m and the maximum elevation is 560 m asl. Thus the range is 310 m. The low-lying areas are, typically, underlain by metasediments and the higher areas are underlain by Nipigon diabase.

5.4 INFRASTRUCTURE AND LOCAL RESOURCES

The village of Beardmore is the closest community, located approximately 16 km north of the Georgia Lake Property. Field programs are based out of the town of Beardmore where there is a restaurant, hotel and Rock Tech’s core shack. Beardmore is part of Greenstone, an amalgamated town encompassing Nakina, Geraldton, Longlac, Beardmore, Caramat, Jellicoe, Macdiarmid and Orient Bay. The population of Greenstone is 4,906 people (Statistics Canada, www.statcan.gc.ca) and the population of Beardmore is approximately 200 people (http://www.highway11.ca/ThunderBay/06Beardmore). Beardmore has limited accommodation and restaurants.

The town of Nipigon has most of the basic supplies needed for exploration work in the Georgia Lake area. Nipigon has grocery stores, a hardware store, restaurants, hotels, a hospital and an OPP station.

Nipigon is located 50 km south of the property. The population for Nipigon Township is 1,752 people in 2006 (Statistics Canada, www.statcan.gc.ca). The city of Thunder Bay also has all of the required supplies for exploration work including grocery stores, hardware stores, exploration equipment supply stores, restaurants, hotels, a hospital, OPP stations and an international airport with daily flights to Toronto, Ontario and Winnipeg, Manitoba and the United States. The population of the city of Thunder Bay was 109,140 people in 2006 (Statistics Canada, www.statcan.gc.ca). Many junior exploration and mining compa-

nies are based in Thunder Bay, and thus the city is a source of skilled mining labour. Thunder Bay is a transportation hub for Canada, as the TransCanada highways 11 and 17 link eastern and western Canada. Thunder Bay is close to the Canada- U.S. border and highway 61 links Thunder Bay with Minnesota, United States. Thunder Bay is also the largest outbound port on the St. Lawrence Seaway system which ships dominantly grain and pulp and paper on Lake Superior.

There is a power line that runs along the TransCanada highway #11 about 10 km from the property. There are three hydroelectric stations on the Nipigon River, all of which are controlled remotely by the headquarters in Thunder Bay: Alexander Station with 68 MW output (17 km north of the town of Nipigon), Cameron Falls with 87 MW output (17 km north of the town of Nipigon) and Pine Portage with 142 MW (39 km north of the town of Nipigon)

(http://www.opg.com/power/hydro/northwest_plant_group/).

There are several lakes, rivers and creeks on the Georgia Lake Property. The lakes on the Nama Creek, Conway and McVittie claim blocks include the Postagoni Lake, Pawky Lake, Dump Lake, Downey Lake, Palace Lake, Pain Lake, Piper Lake, Parsnip Lake, Dive Lake and Pennon Lake. The Little Postagoni River, Phantom Creek and Palace Creek cross the northwestern claim block. Water on the Jean Lake – Foster-Lew property is available from Lake Jean, Parland Lake, Peanut Lake, Parole Lake, Piece Lake, Pomace Lake, Woodpigeon Lake, Pound Creek and Pomace Creek. The source of water on the Aumacho and MNW claim blocks include the Cosgrave Lake, Blay Lake, Hansen Lake, Claus Lake, Pond Lake, Abner Lake, Jackfish River, Namewaminiken River, Hansen Creek and Dot Creek. There is an unnamed creek and a small unnamed lake on the Newkirk-Vegan dispositions.

Rock Tech's Georgia Lake project is in the exploration stage and does only have a NI 43-101 compliant reserve or a prefeasibility study from 2012 (CCIC report 2012); no further study or report for mine plans has been done therefore, discussion on potential tailings storage areas, potential waste disposal areas, heap leach pad areas and potential processing tailings storage area for mining operations is not yet relevant. Rock Tech has surface rights for Nama Creek dispositions and the crown owns the surface rights for all other dispositions and claims.

6 HISTORY

A comprehensive drilling programme was carried out in 1955 and 1956 to the spodumene pegmatites after these had been discovered during general prospection work. Based on these results a first resource estimate was prepared. The key assumptions, parameters and methods used to prepare the historical estimates are unknown. These historical resource estimates do not use the categories outlined in Sections 1.2. and 1.3. of NI 43-101: Standards of Disclosure for Mineral Projects. In other words these historical resources are not 43-101 compliant.

Table 10 and Table 11 summarizes the known historical resource estimates for the properties until 1965 (Pye, 1965).

Table 10. Historical resource ("reserve") estimates for the Nama Creek and Conway Property (Pye, 1965)

Property	Zone	Owner	Estimated Reserves (ton)	Average Li_2O wt. %	Reference
Nama Creek	Main Zone/ North (MZN)	Nama Creek Mines Ltd.	2,784,000	1.11	Isaacs, R. J. (1955): Unpublished company report, Nama Creek Mines Ltd. (Pye, 1965)
Nama Creek	Main Zone/ Southwest (MZSW)	Nama Creek Mines Ltd.	1,508,332	0.96	Isaacs, R. J. (1955): Unpublished company report, Nama Creek Mines Ltd. (Pye, 1965)
Nama Creek sum	**North and Southwest (MZN+MZSW)**	**Nama Creek Mines Ltd.**	**4,292,332**	**1.06**	**Isaacs, R. J. (1955): Unpublished company report, Nama Creek Mines Ltd. (Pye, 1965)**
Conway	Conway (CON)	E.S.Conway/ Leitch Gold Mines Ltd.	1,830,000	0.96	Pye, E. G. (1965): Personal Communications with G. A. McKay, Manager, Leitch Gold Mines Ltd.

In consequence, additional drilling has been completed by Rock Tech in order to upgrade the historical resources to the mineral resource estimate from 2012 for MZN, MZSW, HAR, LIN, and CON (Selway et al., 2012b, CCIC).

The data acquisition done from 2009 to 2012 is well documented including a comprehensive QA/QC management to validate the acquired data. Twinning could reproduce logged mineralization and assays of historical holes in acceptable ranges. However, there seems to be an offset in location, which might be explained by an error in survey of collar position or drill path. However, this is estimated not to have a significant effect on the tonnage and grade estimates based on the 2012 NI 43-101 resource model and estimate. A model validation chapter or cross sections demonstrating that the resource model (interpreted wireframes or interpolated block values) is representative to the primary data was not documented in this report. Table 12 shows the old block model tonnages and grades reported at various cut-off grades of Li_2O% of the five pegmatites shown in the report from CICC, Aug. 29th, 2012.

Table 11. Mineral resource statement1 (CICC, Aug. 29th, 2012) reported at a cut-off grade of 0.6 Li_2O%

Property	Resource Class	Tonnage [Mt]	Grade Li_2O [%]
MZN	Indicated	2.47	1.11
Conway	Indicated	0.72	1.05
Total	Indicated	3.19	1.10
MZN	Inferred	2.50	0.98
Conway	Inferred	0.59	1.02
Line60	Inferred	1.30	0.93
MZSW	Inferred	0.97	1.09
Harricana	Inferred	0.95	1.03
Total	Inferred	6.31	1.00

Table 12. Block model tonnages and grades reported at various cut-off grades of Li_2O% per pegmatite (CICC, Aug. 29th, 2012)

Property	Cut – Off Li_2O%	Resource Class	Tonnage [Mt]	Grade Li_2O [%]
MZN	0.4	Indicated	2.50	1.10
MZN	0.6	Indicated	2.47	1.11
MZN	0.8	Indicated	1.89	1.14
MZN	1.0	Indicated	1.69	1.22
Conway	0.4	Indicated	0.73	1.05
Conway2	0.6	Indicated	0.72	1.05
Conway	0.8	Indicated	0.65	1.09
Conway	1.0	Indicated	0.36	1.24
MZN	0.4	Inferred	2.67	0.95
MZN2	0.6	Inferred	2.50	0.98
MZN	0.8	Inferred	1.76	1.09
MZN	1.0	Inferred	1.23	1.20
Conway	0.4	Inferred	0.62	1.00
Conway2	0.6	Inferred	0.59	1.02
Conway	0.8	Inferred	0.50	1.07
Conway	1.0	Inferred	0.31	1.16
Line	0.4	Inferred	1.93	0.79
Line	0.6	Inferred	1.30	0.93
Line	0.8	Inferred	0.83	1.06
Line	1.0	Inferred	0.49	1.17
MZSW	0.4	Inferred	0.97	1.09
MZSW2	0.6	Inferred	0.97	1.09
MZSW	0.8	Inferred	0.88	1.13
MZSW	1.0	Inferred	0.57	1.23
Harricana	0.4	Inferred	0.95	1.03
Harricana2	0.6	Inferred	0.95	1.03
Harricana	0.8	Inferred	0.84	1.07
Harricana	1.0	Inferred	0.54	1.15

7 GEOLOGICAL SETTING AND MINERALIZATION

This section is sub-divided into three sub-sections describing the regional to property scale geology and mineralization.

7.1 REGIONAL

The Georgia Lake area is located within the Quetico Subprovince of the Superior Province. The Quetico Subprovince is bounded by the granite-greenstone Wabigoon Subprovince to the north and Wawa Subprovince to the south (Williams, 1991). The Quetico Subprovince is composed of predominantly metasediments consisting of wacke, iron formation, conglomerate, ultramafic wacke and siltstone, which deposited between 2.70 and 2.69 Ga. The igneous rocks in the Quetico Subprovince include abundant felsic and intermediate intrusions, metamorphosed rare mafic and felsic extrusive rocks and an uncommon suite of gabbroic and ultramafic rocks. The earlier felsic intrusions occurred 5 to 10 million years after the accumulation of sediments and are interpreted to be I-type intrusions (White and Chapell, 1983). The later felsic intrusions occurred 20 million years after the sedimentation and are designated as S-type (White and Chapell, 1983).

The Quetico Subprovince was subjected to four deformational events between approximately 2700 and 2660 million years (Williams, 1991). The predominant stratigraphic-facing direction is north (Carter, 1984, 1987, 1988; Harris, 1970; Perdue, 1938; Williams, 1988). Regional schistosity is variably developed and oriented and is interpreted to be the result of regional shortening and dextral shearing.

Four major faults cut through the Quetico Subprovince: the easterly trending Quetico fault (Fumerton, 1982; Bau, 1979; Kennedy, 1984), the Rainy Lake-Seine River fault (Fumerton, 1982, Davis et al., 1989), the northeasterly trending Gravel River fault (Williams, 1989) and the Kapuskasing Structural Zone (Percival, 1989).

Metamorphism, migmatite formation and granite intrusion occurred between 2.67 and 2.65 Ga (Williams, 1991). The grade of metamorphism ranges from lower greenschist to amphibolite facies and tends to be lower in the marginal rocks of the subprovince and higher in the core regions (Percival, 1989).

Widespread economic mineralization within the Quetico Subprovince is generally lower than in the adjacent greenstone dominated terranes (Williams, 1991). Minor gold mineralization is associated with veining along the Quetico Fault (Poulsen, 1983). Molybdenite occurs in biotite leucogranites in the Dickinson Lake area (Carter, 1975, 1985). The only potentially important ore deposit type consists of the late-stage pegmatites that contain the rare elements lithium, beryllium, tantalum, niobium and tin (Williams, 1991). The rare-element pegmatites have widespread distribution in the Quetico Subprovince covering at least a 540 km strike length from west to east and a large percentage of pegmatites occur in the centre of the subprovince (Breaks, Selway and Tindle, 2005):

- Spodumene-subtype pegmatites at Wisa Lake, Lac La Croix area

- Fertile granites and beryl-type pegmatites in Niobe-Nym lakes and Onion Lake areas
- Albite-spodumene-type pegmatites of the Georgia Lake area
- Complex-type, lepidolite subtype Lowther Township pegmatite near Hearst (Breaks, Selway and Tindle, 2003a)

The pegmatites in the Quetico Subprovince are hosted by metasediments and/or by their parent granite (Pye, 1965; Breaks, Selway and Tindle, 2003a, 2003b).

7.2 LOCAL

The geology of the Georgia Lake area is of Precambrian age and is discussed by Pye (1965). The local rocks were build up from the following types:

Metasediments:

The oldest rocks are the Archean metasediments. The metasediments strike east-northeast and dip steeply, in general, to the north. The dominant metasedimentary rock is biotite-quartz-feldspar schist or gneiss. It is a grey, rather dark coloured rock, having a distinct banded appearance due to compositional variations reflecting an original sedimentary stratification, with individual layers less than an inch to several feet thick. There is a distinct foliation due to parallel alignment of biotite crystals. Microscopic examination of the biotite-quartz-feldspar schist shows that it is made up of: 15-40 vol.% biotite , 20-35 vol.% quartz, 25- 45 vol.% plagioclase, 1-3 vol.% magnetite, trace amounts of zircon and rare hornblende. Secondary minerals include chlorite, sericite and epidote. The plagioclase shows myrmekite texture. The most abundant texture in the biotite-quartz-feldspar schist or gneiss is granoblastic, but porphyroblastic rocks are also present with porphyroblasts of garnet, staurolite and cordierite.

Metagabbro:

The metagabbro has intrusive relationships and have been metamorphosed and intruded by granitic rocks. East of Cosgrave Lake and south of Barbara Lake, the metasediments were intruded by metagabbro. The metagabbro bodies range in size from a few hundred feet across to 9,500 feet (=2.9 km) across. The metagabbro is dark-coloured (mesocratic), medium- to coarse-grained with a brownish weathered surface. For the most part, it is massive, but it is gneissic near its contacts with metasediments. The major minerals are: green hornblende and plagioclase (sodic andesine). The minor minerals include: microcline and biotite and trace amounts of magnetite and apatite. The alteration minerals are chlorite, epidote and sericite. The porphyritic metagabbro differs from the metagabbro only in the presence of feldspar phenocrysts (usually microcline). The feldspar phenocrysts are pale-pink to red, stubby, rectangular, subhedral to euhedral and range in size from ¼ by 1/8 inch (=0.6 by 0.3 cm) to 2 by 1 inches (5 by 2.5 cm). The porphyritic metagabbro is best developed near the margins of the metagabbro bodies close to the granites. Metagabbro dykes and sills cross cut the metasediments near Dump and Pawky lakes and near Blay, Georgia

and Conner lakes. All of the dykes and sills are small with ticknesses of 3 feet or less (=0.9 m). They are thought to be genetically related to the metagabbro, as they are similar in appearance and composition. They are cross cut by pegmatite and feldspar porphyry dykes.

Granite:

The metasediments were also intruded by large masses of granitic rocks and by numerous sills and dykes of genetically-related porphyry, pegmatite and aplite. The granitic rocks are pale-grey or pale-pink in colour and their essential components are: 45-65 vol.% feldspar (microcline and plagioclase), 40 vol.% quartz, and one or both of muscovite and biotite and rarely little hornblende. The plagioclase has a composition of albite. Minor components of the granites include magnetite, zircon, and garnet, and secondary minerals: chlorite, sericite and epidote. For the most part the granites are equigranular, but porphyritic phases with microcline phenocrysts also occur. The contacts between the equigranular granitic rocks and the metasediments are generally abrupt.

Pegmatite:

There is an abundance of pegmatites close to and within the large masses of granitic rocks. A regional zoning is apparent and a genetic association of pegmatites and granite is indicated. The pegmatites occur in two geometries: as irregular-shaped bodies and as thin dykes, sills and attenuated lenses. The irregular bodies of pegmatite are intimately associated with the granite bodies often within a few hundred feet of the contact zone. They typically are medium- to coarse-grained, up to very coarse-grained and are made up of quartz, microcline, perthite and little muscovite. These would be classified as potassic pegmatites. Accessory minerals include biotite, tourmaline and garnet. The pegmatite dykes, sills and lenses can be subdivided into rare-element pegmatites and granitic pegmatites. The rare-element pegmatites are of economic significance and they contain microcline or perthite, albite, quartz, muscovite and spodumene and minor amounts of beryl, columbite-tantalite and cassiterite. The granitic pegmatites are similar to the irregular pegmatites described above except that they contain more abundant plagioclase. Some of the pegmatites are parallel to the foliation or bedding of the metasediments, whereas others occur in joints in either the metasediments or granite. Contacts are usually sharp and, except where dykes cut granitic rocks, often found to be marked by a thin border zone of aplite or granitoid composition. A few pegmatites are internally zoned with mica-rich or tourmaline-rich rock along or close to the walls and quartz cores.

Sedimentary rocks:

The Proterozoic is represented by sedimentary rocks (sandstone and shale). Since these are not present on the Georgia Lake Lithium property, they are not discussed here and the reader is referred to Pye (1965) for more information on them.

Diabase:

Intrusive into the Proterozoic sedimentary rocks and the older formations are bodies of diabase. The largest occur as flat sheets (Logan sills), up to about 650 ft (=198.1 m) in thickness, and as dykes of vertical or near-vertical attitude. Most of the dykes are related closely to the sheets and are Keweenawan age. The gently dipping diabase sheets are dark coloured and massive. The diabase sheets are well-jointed and most of the joints are vertical or steeply dipping. In outcrop, the diabase shows poorly-formed columnar structure.

There are two types of diabase dykes: one is equigranular and the other is porphyritic. The equigranular dykes are more abundant. Some of the dykes along or close to the contact zone of the large granite mass strike easterly; most dykes in other localities strike north or within 20º of north. With few exceptions the dykes are vertical or dip steeply. The porphyritic diabase dykes are massive medium-grained, darkcoloured rock characterized by many pale-greenish yellow phenocrysts of highly altered plagioclase. Porphyritic diabase dykes are found near the MZN pegmatite called from the geologists in the 1955 reports the “Jackpot”.

7.3 PROPERTY

The following figures show an overview of the geology of the area and the location of the claims and dispositions from Rock Tech in detail. Rock Tech owns rights to 283 claims (56.93 km^2) and 81 dispositions (10.42 km^2).

On surface glacial sediments dominate. Only in some areas host rocks are exposed. In these areas with outcrop, lithium-bearing pegmatites were found during reconnaissance work, which started in places some 60 years back. These occurrences of spodumene pegmatites were drill-tested in order to get information about thickness, grade and orientation of the pegmatite bodies.

The whole property are divided in a northern area NSPA and a southern area SSPA. In the SSPA 5 subareas are shown in Figure 9. In these subareas spodumene pegmatites occur at the surface and sampling and drilling was done.

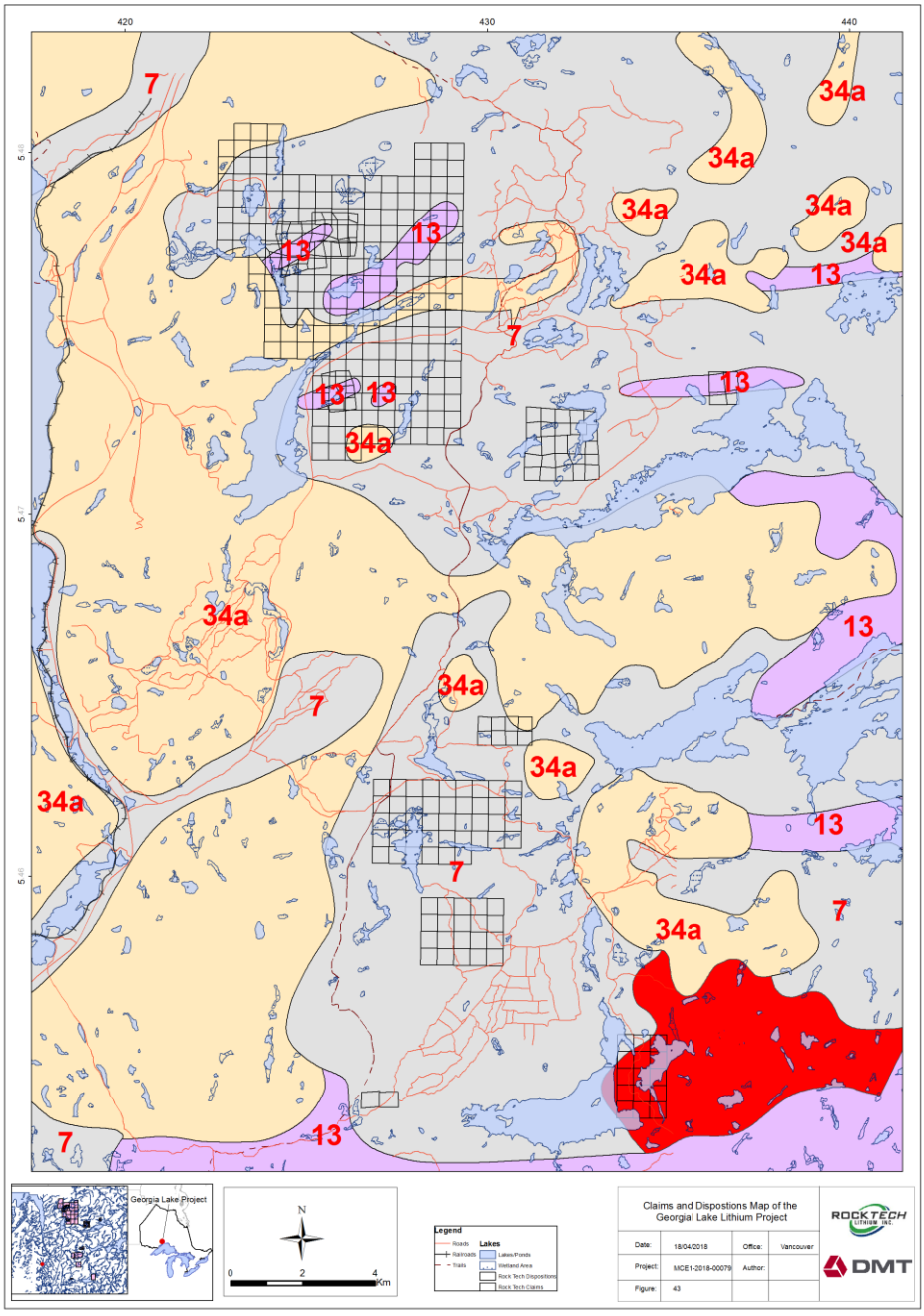


Figure 8. Overview of the local Geology. Red numbers show the rock type and are listed in the text below. Black rectangles show the claims and dispositions of Rock Tech.

Figure 8 shows the geology of the area and the claim boundaries in black. The lithological units, as shown on Figure 8 are explained in the following:

- 7 (grey): metasedimentary rocks
- 34a (light brown): Logan and Nipigon sills / diabase sills
- 13 (magenta) : Muscovite-bearing granitic rock
- 15 (red) : massive granodiorite to granite rock

Figure 9 shows an overview of the boreholes in the SSPA.

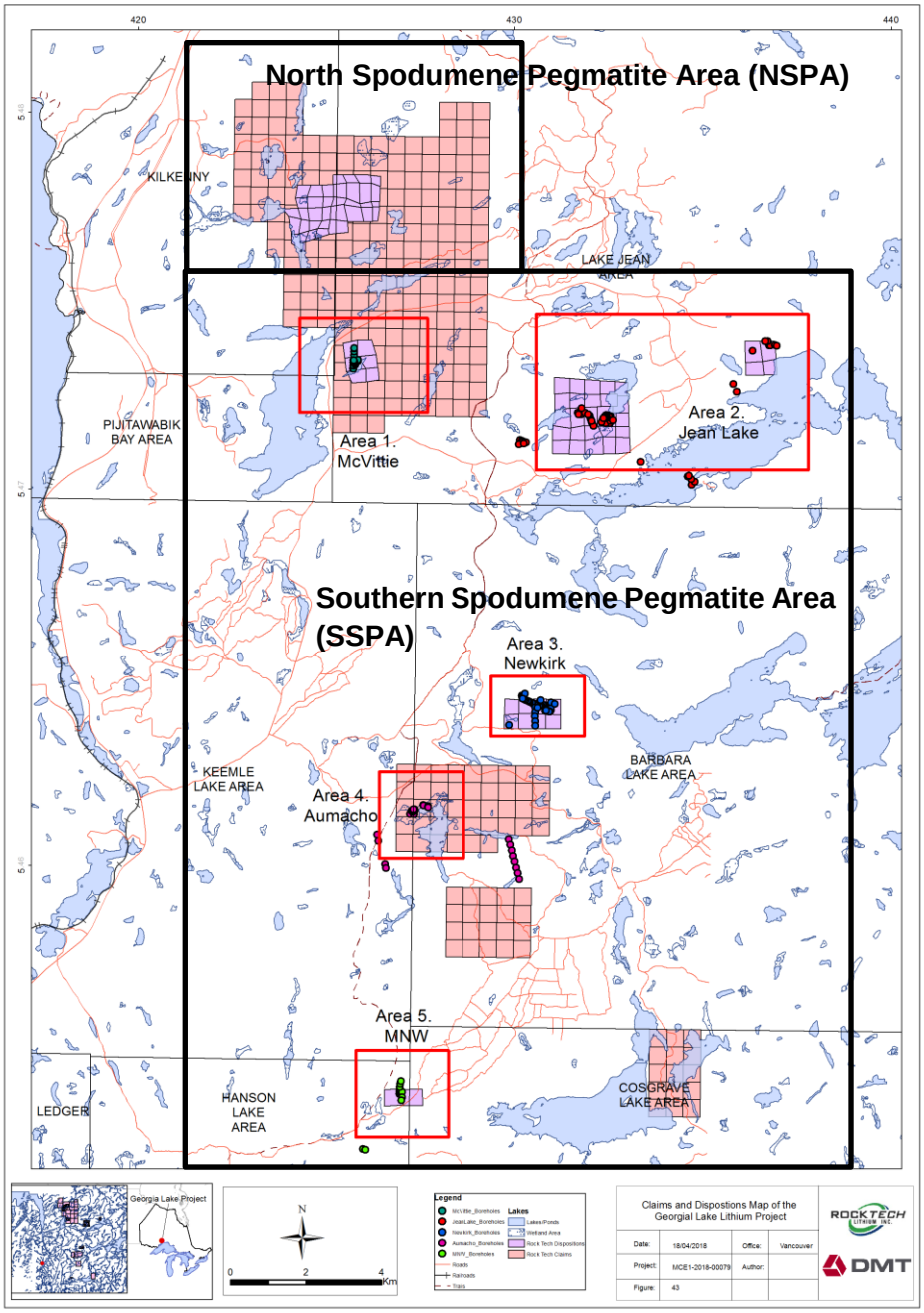


Figure 9. Overview of sub areas and drillhole locations in the SSPA.

In the SSPA South are the five subareas with pegmatites located. These areas are: McVittie (area 1), Jean Lake east and west (area 2), Newkirk (area 3), Aumacho (area 4) and MNW (area 5) see Figure 9.

In some cases drilling and channel sampling has been executed during the latest exploration phase. The next figures show an overview of these activities in the different areas. All these areas belong to Rock Tech.

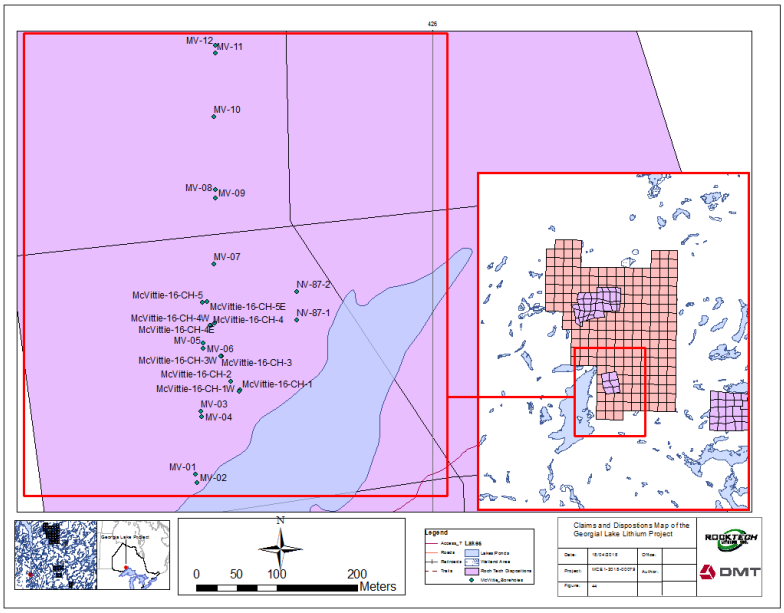


Figure 10. Drill hole and channel locations of the McVittie area

The shape of the McVittie pegmatite has been defined through field investigations and the chemical analyses. The north south strike length on the map is about 400 m. The deepest drillhole intersection is about 100 m below surface. The drillhole data shows in average true thickness of 4 m of intersected spodumene pegmatite.

Table 13. Numbers of drill holes and channel samples in the McVittie area during the different exploration phases

McVittie Boreholes Overview				
Year	**Boreholes**	**Channels**	**Sum of metres drilled**	**Metres channel sampled**
1955/66	12	0		
1987	2	0		
2016	0	10		
total	14	10	1243.3	40.86

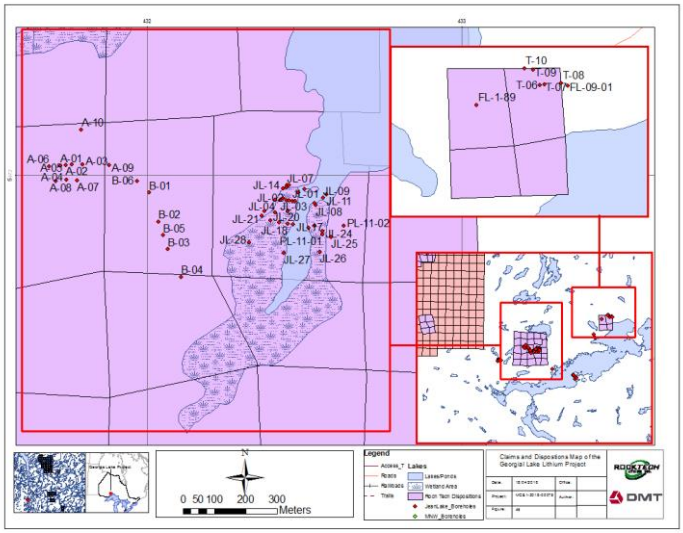


Figure 11. Drill hole and channel locations of the Jean Lake area.

The Jean Lake pegmatite area is divided in a western and eastern part.

The shape of the western Jean Lake pegmatite has been defined through field investigations and chemical analyses. The east west strike length on the map is about 350 m. The deepest drillhole intersection is about 200 m below surface. The drillhole data shows in average true thickness of 6 m of intersected spodumene pegmatite.

The east Jean Lake pegmatite was drilled in the past. The results indicate that the pegmatite has an 800 m long strike length, confirmed by 6 intersections within historic drillholes. No analyses were recorded. Therefore this pegmatite has been excluded from the resource statement. Further exploration work is recommended.

Table 14. Numbers of drill holes and channel samples of the area Jean Lake during the different exploration phases

Jean lake Boreholes				
year	boreholes	channels	sum of bore meters	meters of channel samples
1955/66/65	49	0		
1989	1	0		
2009/2011	2	0		
2016	0	2		
2017	7	0		
total	59	2	9275.63	7.23

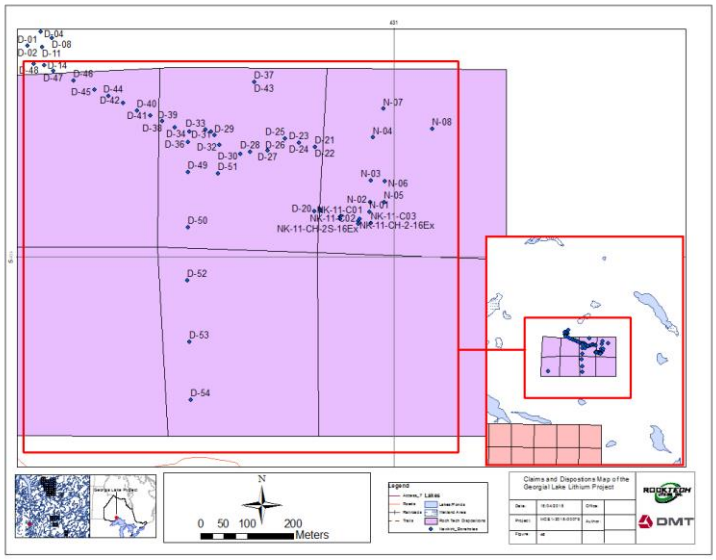


Figure 12. Drill hole and channel locations of the Newkirk area

The shape of the Newkirk pegmatite has been defined through field investigations and chemical analyses. The east west strike length on the map is about 900 m. The deepest drillhole intersection is about 150 m below surface. The drillhole data shows an average true thickness of 3 m of spodumene pegmatite.

Table 15. Numbers of drill holes and channel samples in the area Newkirk during the different exploration phases

Newkirk boreholes				
year	boreholes	channels	sum of bore meters	meters of channel samples
1955/56	42	0		
2011	0	3		
2016	0	6		
total	42	9	2997.5	46.26

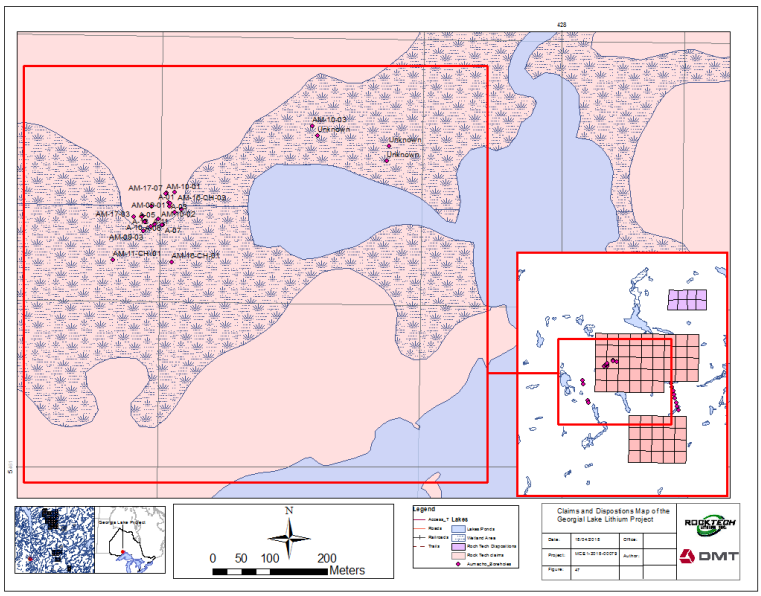


Figure 13. Drill hole and channel locations of the Aumacho area.

The shape of the western Aumacho pegmatite has been defined through field investigations and chemical analyses. The north south strike length on the map is about 250 m. The deepest drillhole intersection is about 90 m below surface. The drillhole data shows an average true thickness of 4 m of spodumene pegmatite.

Table 16. Numbers of drill holes and channel samples in the area Aumacho during the different exploration phases.

Aumacho drillholes				
year	boreholes	channels	sum of bore meters	meters of channel samples
1955	16	0		
2009/2010/2011	6	1		
2016	0	3		
2017	7	0		
total	29	4	2107.9	24.07

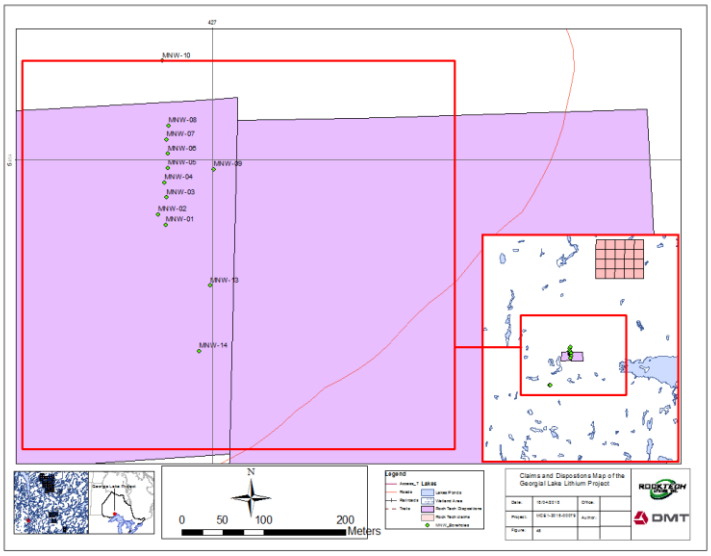


Figure 14. Drill hole and channel locations of the MNW area.

The MNW pegmatite is situated in the southernmost part of the claims and disposition area of SSPA. Data from historic and recent drillholes are present. However, both the historic and recent drillhole results are insufficient to outline shape of the pegmatite. Therefore this pegmatite has been excluded from the resources estimate. Further exploration work is recommended.

Table 17. Numbers of drill holes and channel samples in the area MNW during the different exploration phases

MNW drillholes				
year	**boreholes**	**channels**	**sum of bore meters**	**meters of channel samples**
1956	11	0		
total	11	0	616.9	0

8 DEPOSIT TYPES

8.1 RARE-ELEMENT PEGMATITES OF SUPERIOR PROVINCE

Rare-element pegmatites may host several economic commodities, such as tantalum (Ta-oxide minerals), tin (cassiterite), lithium (ceramic-grade spodumene and petalite), rubidium (lepidolite and K-feldspar), and cesium (pollucite) collectively known as rare elements, and ceramic-grade feldspar and quartz (Selway et al., 2005). Two families of rare-element pegmatites are common in the Superior Province, Canada: Li-Cs-Ta enriched ("LCT") and Nb-Y-F enriched ("NYF"). LCT pegmatites are associated with S-type, peraluminous (Al-rich), quartz-rich granites. S-type granites crystallize from a magma produced by partial melting of preexisting sedimentary source rock. They are characterized by the presence of biotite and muscovite, and the absence of hornblende. NYF pegmatites are enriched in rare earth elements ("REE"), U, and Th in addition to Nb, Y, F, and are associated with A-type, subaluminous to metaluminous (Al-poor), quartz-poor granites or syenites (Černý, 1991a).

Rare-element pegmatites derived from a fertile granite intrusion are typically distributed over a 10 to 20 km^2 area within 10 km of the fertile granite (Breaks and Tindle, 1997). A fertile granite is the parental granite to rare-element pegmatite dykes. The granitic melt first crystallizes several different granitic units (e.g., biotite granite to two mica granite to muscovite granite), due to an evolving melt composition, within a single parental fertile granite pluton. The residual melt enriched in incompatible elements (e.g., Rb, Cs, Nb, Ta, Sn) and volatiles (e.g., H_2O, Li, F, BO_3, and PO_4) from such a pluton can then migrate into the host rock and crystallize pegmatite dykes. Volatiles promote the crystallization of a few large crystals from a melt and increase the ability of the melt to travel greater distances. This results in pegmatite dykes with coarse-grained crystals occurring in country rocks considerable distances from their parent granite intrusions.

There are several geological features that are common in rare-element pegmatites of the Superior province of Ontario (Breaks and Tindle, 2001; Breaks et al., 2003) and Manitoba (Černý et al., 1981; Černý et al., 1998) (Selway et al., 2005):

1. Subprovincial boundaries: The pegmatites tend to occur along subprovincial boundaries.

2. Metasedimentary-Dominant Subprovince: Most pegmatites in the Superior province occur along subprovince boundaries, except for those that occur within the metasedimentary Quetico subprovince.

3. Greenschist to Amphibolite Metamorphic Grade: Pegmatites are absent in the granulite terranes.

4. Fertile Parent Granite: Most pegmatites in the Superior province are genetically derived from a fertile parent granite.

5. Host Rocks: Highly fractionated spodumene- and petalite-subtype pegmatites are commonly hosted by mafic metavolcanic rocks (amphibolite) in contact with a fertile granite intrusion along subprovincial boundaries. Pegmatites within the Quetico subprovince are hosted by metasedimentary rocks or their fertile granitic parents.

6. Metasomatized Host Rocks: Biotite and tourmaline are common minerals, and holmquistite is a minor phase in metasomatic aureoles in mafic metavolcanic host rocks to spodumene- and petalite-subtype pegmatites. Tourmaline, muscovite, and biotite are common, and holmquistite is rare in metasomatic aureoles in metasedimentary rocks.

7. Li Minerals: Most of the complex-type pegmatites of the Superior province contain spodumene and/or petalite as the dominant Li mineral, except for a few pegmatites, which have lepidolite as the dominant Li mineral.

8. Cs Minerals: Cesium-rich minerals only occur in the most extremely fractionated pegmatites.

9. Ta-Sn Minerals: Most pegmatites in the Superior province contain ferrocolumbite and manganocolumbite as the dominant Nb-Ta-bearing minerals. Some pegmatites contain manganotantalite or wodginite as the dominant Ta-oxide mineral. Tantalum-bearing cassiterite is relatively rare in pegmatites of the Superior province.

10. Pegmatite Zone Hosting Ta Mineralization: Fine-grained Ta-oxides (e.g., manganotantalite, wodginite, and microlite) commonly occur in the aplite, albitized K-feldspar, mica-rich, and spodumene core zones in pegmatites in the Superior province.

8.2 GEORGIA LAKE PEGMATITE FIELD

The majority of the pegmatites in the Postagoni Lake group and Georgia Lake group can be classified as albite-spodumene type pegmatites. Albite-spodumene type pegmatites are characterized by homogenous dykes with coarse-grained spodumene + K-feldspar aligned perpendicular to the dyke walls, spodumene is the dominant or only Li-bearing mineral and albite is more abundant than K-feldspar.

The Aumacho – Brink pegmatite is classified as a spodumene-subtype pegmatite. Spodumene-subtype pegmatites are characterized by complex internal zonation, spodumene is the dominant Li-bearing mineral and albite is more abundant than K-feldspar.

The MNW pegmatite in the south of the hole claim area is classified as a petalite-subtype pegmatite. Petalite-subtype pegmatites are characterized by complex internal zonation, petalite is the dominant Li-bearing mineral and K-feldspar is more abundant than albite. Often petalite is rare in the pegmatite, and SQUI (spodumene-quartz intergrowth due to the breakdown of petalite) is common instead.

9 EXPLORATION

In general, mapping, trenching and drilling has been carried out in 1955/1956 and again starting from 2009 in the area relevant for resource modelling and estimation. No geophysical surveys were applied during that exploration phase in these areas. In other parts of the property, geophysical ground surveys have been done, e.g. magnetic, electromagnetic. However, results were not significant (CCIC, 2012). From 2014 until 2018 additional mapping, drilling and channel sampling take place. (see the individual figures and tables in Chapter 7).

The number of all boreholes and channels of the whole area are listed in the Table 18. In all exploration claims and dispositions 48.3 km of drillholes and channels were drilled and cut during since the exploration began in early fifties of the last century.

Table 18 Number of all boreholes and channels in the whole area of Rock Tech.

Year of execution	Number of Boreholes	Number of Channels	Meters drilled	Meters of channels
not recorded	33		2,340	
1955/56	205		28,677	
1957/58	26		1,787	
1987/89	3		199	
2009/2010/2011	70	74	12,409	456
2016/2017	14	106	1,972	442
TOTAL	**351**	**180**	**47,384**	**898**

10 TRENCHING AND DRILLING

Starting from November 2009, 62 NQ drill holes (11 588 m) have been drilled in the 5 properties MZN, MZSW, HAR, LIN and CON. Logging and sampling followed standard operating procedures (SOPs) implemented and supervised by CCIC during the main programmes in 2010 and 2011. Both programmes were laid out for resoruce defintion and historical database confirmation.

Table 19. Drilling done since 2009

AREA	Number of holes	Meters drilled
MZN	32	6,775
MZSW	4	728
HAR	6	966
LIN	5	808
CON	15	2,311
TOTAL	62	11,588

Additionally, 138 channels (722 m) have been cut to investigate the outcropping spodumene pegmatite.

Table 20. Trenches done since 2009

AREA	Number of trenches	Meters trenched
MZN	18	144
MZSW	28	77
HAR	31	134
LIN	39	295
CON	22	72
TOTAL	138	722

These data were used to confirm historic data and extent the data base . In total, 113 historic drill holes (17614 m) are available.

Table 21. Historic drilling done 1955/56

AREA	Number of holes	Meters drilled
MZN	45	8,601
MZSW	13	2,216
HAR	18	2,547
LIN	22	3,305
CON	15	945
TOTAL	113	17,614

All in all, 175 drill holes totalling to more than 29 000 m were available for the resource estimate for the 5 3D moddeld pegmatites from the NSPA.

Table 22. Historic and recent drilling

AREA	Number of holes	Meters drilled
MZN	77	15,376
MZSW	17	2,944
HAR	24	3,513

AREA	Number of holes	Meters drilled
LIN	27	4,113
CON	30	3,256
TOTAL	175	29,202

All drill holes were drilled in a sectional pattern with a spacing of some 60 m in between sections and 40 to 60 m along sections. All drill holes were drilled inclined in order to intersect the main pegmatites as perpendicular as possible.

The following Figure 15 give an overview about drill holes locations, spacing and distribution of the MZN pegmatite. The row of outcrops of the pegmatite is shown with the white line.

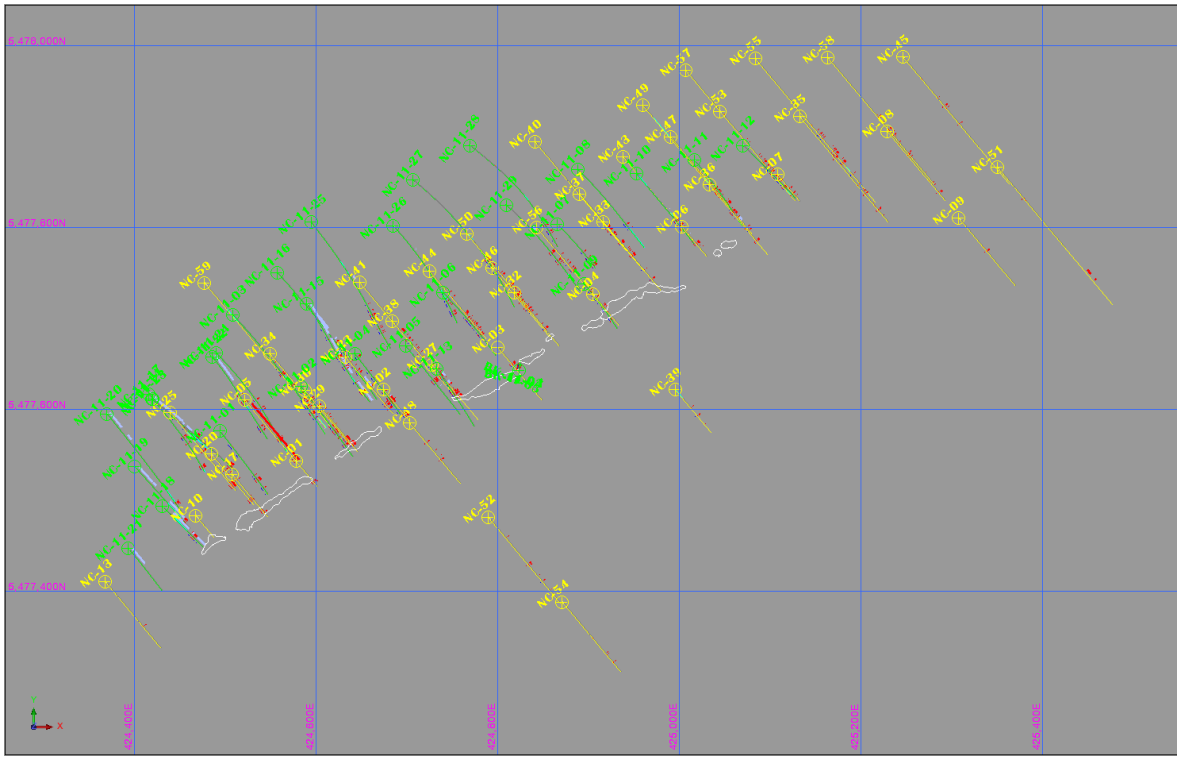


Figure 15. Drill holes in MZN; historic holes (yellow) and recent drill holes (green). Grid spacing 200 m. Outcrop of pegmatite white line.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

SOPs were implemented and supervised by CCIC during the main programmes of 2009 and 2011 in order to confirm and extent the historical database, for which no procedures or results of any QA/QC management are documented.

All work of data acquisition was done or re-done following these SOPs. For details about the procedures applied see document: 'Independent Technical Report and Updated Resource for Nama Creek Main Zone North Pegmatite Georgia Lake Lithium Property, Beardmore, Ontario, Canada (CCIC, Oct. 5, 2012).

All the drill core samples from the 2010-2011 winter drill program were submitted to SGS Toronto for analysis. Channel samples and drill core of later programs were prepared by Actlabs' preparation lab in Geraldton, Ontario and then shipped to Actlabs' analytical lab, Ancaster, Ontario for analysis. SGS and Actlabs Laboratories are ISO17025 certified.

Once the core samples were received by the laboratory a confirmation of receipt was e-mailed to CCIC. The samples were dried, crushed to 75 %, split and a 250 g aliquot was pulverized to 85 % at 75 µm (SGS sample preparation code PRP89). Samples were weighed (SGS code WGH79) and fused with a sodium peroxide fusion. Trace element analysis was completed with an ICP-AES (SGS code IC90A). Major elements were analyzed using XRF with a tetraborate fusion (SGS code XRF76C).

CCIC inserted standards and blanks into the sample stream in regular intervals: every tenth sample was either a low-grade standard (STDL), a high-grade standard (STDH) or a blank. One in 20 samples was a core duplicate.

SGS included internal blanks, standards: SY4 (certified for 37 ppm Li), NBS 183 (certified for 19,140 ppm Li) and NBS 97B (certified for 550 ppm Li), and pulp duplicates as part of their internal quality control.

Sample preparation for drill core by Actlabs was similar to the SGS' sample preparation. At Actlabs, the entire channel sample is crushed to a nominal minus 10 mesh (1.7 mm), mechanically split (riffled) to obtain a representative sample and then pulverized to at least 95% minus 150 mesh (106 microns). They automatically use cleaner sand between each sample.

The samples were analyzed by Actlabs' Code 8 – REE assay package, which grinds the samples to 95% -200 mesh to ensure complete fusion of resistate minerals. The samples were then digested using lithium metaborate/tetraborate fusion and analyzed the major elements by ICP and trace elements by ICP/MS. The lab used mass balance as a quality control technical and elemental totals of the oxides should be between 98-101 %. The Li % was analyzed by Actlabs Code 8 – Lithium Ore analysis package which digests the samples by sodium peroxide fusion and analyses them using ICP/OES. The detection limit for Li % was 0.01 %. Selected samples were also analyzed for specific gravity. Actlabs used the following internal standards for the Li analysis: ZW-C (certified for 1.13 %Li), NCS DC86303 (certified for 0.21 % Li), NCS DC86304 (certified for 1.06 % Li), NCS DC8614 (certified for

1.81 % Li). Actlabs used their internal lab tolerance of 95 to 105 % for the pass/fail of the internal standards. Silica flour was used as a standard for the specific gravity measurement. Actlabs also analyzed pulp and preparation duplicates and method blank as part of their quality control. Actlabs uses distilled water as a “method blank” and “sand blank” from sample preparation is also used (CCIC report 2012).

All further exploration work followed these methodology and were checked by local consultants. During the site visit the procedures were discussed with the local experts and analyses from the certificated laboratory show results of the blanks, references and duplicate samples with acceptable deviations to the former results. A peer visual check took place during the site visit in October 2017.

12 DATA VERIFICATION

Data verification was done on several levels, which are described in detail below.

12.1 SITE VISIT

A site visit inspecting the ongoing work of trenching and mapping and existing drill core, logged and sampled since 2009 has been done in October 2017 by Stephan Peters, who is a qualified person in accordance to NI 43-101.

12.2 STANDARD OPERATING PROCEDURES (SOPS)

SOPs, set-up specifically for this project, have been implemented and supervised by CCIC in 2010. The SOPs included a comprehensive QA/QC management and thus enables DMT to verify the quality and representativeness of acquired data, e.g. core recovery was noted and QA/QC sample sets were included.

DMT assesses that the procedures, as outlined in this report, are suitable to have produced representative data appropriate for use in a resource estimate.

12.3 AVAILABILITY OF DATA

All digital project data were avalible for this report. The existing database were maintaind and all hardcopies were scanned or avalible in *.pdf or *.doc format. A GIS system is also avalible. All drill core photos and channelsample photos are stored in a digital format.

For the main investigation area of the modelled 5 pegmatites 195 drill holes (29428 m) and 118 trenches (496 m) have been drilled and cut in the main north area. The following tabled data sets are available for these holes and channels:

- Collar location and orientation
- Hole deviation
- Drill diameter
- Core recovery
- RQD: geotechnical rock quality data
- Geological logs distinguishing host rocks and lithium mineralization
- Assay data including QA/QC data.
- Assay certificates as PDF signed by SGS and Actlabs and corresponding Excel files including the specific gravity in [t/m^3] Li and other chemical parameters:
- Coordinates and elevations of drill hole collars were provided by the client to DMT in map datum UTM NAD83 Zone 16 Northern Hemisphere.

The following GIS data were provided to DMT in map datum UTM NAD83 Zone 16 Northern Hemisphere and have been used for resource modelling

- License areas were provided as vector files. License certificates were provided to DMT as well.
- Mapped outcrops of spodumene pegmatites and diabase as vector data
- A regional geological map scaled to 1:25 000
- A set of property scales geological maps
- A topographical map scaled to 1:25 000
- A digital terrain model with a grid spacing of around 20 m.

12.4 DATA PREPARATION AND MANAGEMENT

All data of drilling and trenching has been provided by the client in a Relational Database Management Software (RDBMS), Microsoft Access, and has been checked by DMT for consistency and completeness. Thereafter, data has been transferred to the modelling software Geovia Surpac to visualize the drill holes in a 3D environment. A digital terrain model (DTM) was also added to Surpac and visualized. Topographic and geological have been draped onto the DTM. Available collar locations were validated against the DTM, surface topographic features, geology mapped and license boundaries.

All these data are the underlying basis for the geological interpretation and wireframe modelling.

12.5 DRILLING LOCATION AND ORIENTATION

For all historic and recent drill holes data of collar position and down-hole orientation are available. Trenches were surveyed from start to end and treated as horizontal drill holes.

Collar positions of the recent holes were surveyed by Rock Tech using a Trimble Differential GPS. Historic casings were re-surveyed using this Trimble DGPS and surveyed data were entered into the database to ensure that historic collar positions are at a high level of accuracy. However, some twin drilling shows increased deviations of intersected mineralized spodumene pegmatites. This might be also explained with inaccuracies of either the historic survey or the down-hole surveying.

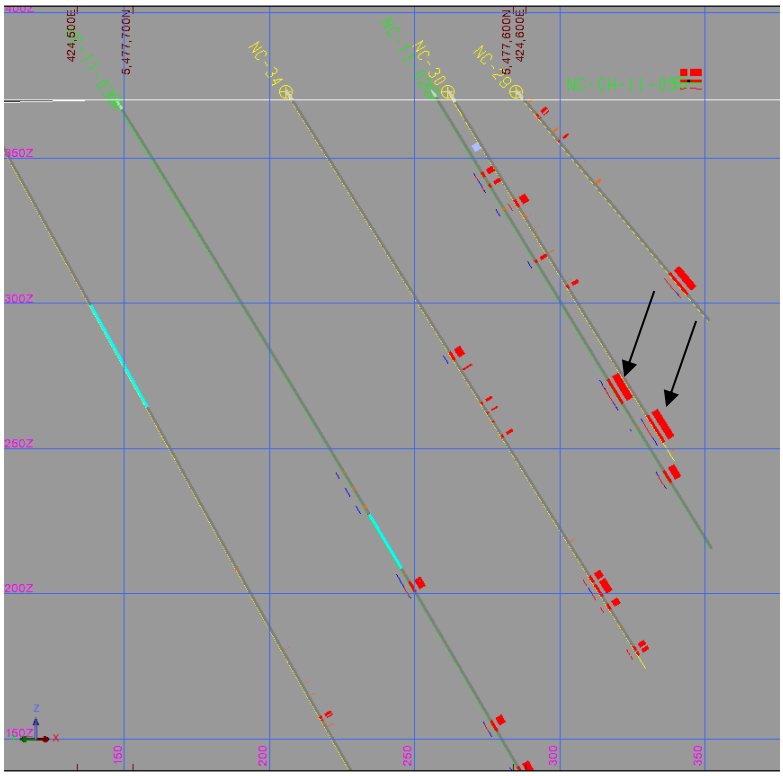


Figure 16. Deviations of intersected mineralization in twin holes (green) and historic holes (yellow). Grid spacing 50 m.

12.6 DRILLING RECOVERY AND DIAMETER

Recent drilling was aimed at maximising sample recovery in order to ensure representative nature of the samples. The overall core recovery is almost 100 % in all drill core, in intervals logged as spodumene pegmatite, in intervals assayed for Li_2O and in interval interpreted as spodumene pegmatite bodies. All recent holes are drilled NQ. For historic holes no recovery data are available. In 4 drill holes into MZSW, the recovery was recorded summarily as 100 %. It is assessed that more detailed logging of core recoveries for these 4 holes will

not change the general outcome significantly. In consequence, a sample bias caused by poor core recovery can be excluded.

Table 23. Core recovery of the recent drillings (2009-20017) which were used for the 3D modelling of the 5 pegmatites

AREA	All drill core [%]	In intervals logged as spodumen pegmatite [%]	In intervals assayed for Li_2O [%]	In interval interpreted as mineralized domain [%]
MZN	97.0	97.3	97.0	98.4
MZSW	100.0	100.0	100.0	100.0
HAR	98.5	99.0	98.5	99.1
LIN	98.8	98.5	98.8	98.3
CON	98.8	98.8	98.8	98.9

12.7 GEOLOGICAL LOGGING

Drill core has been geologically and geotechnical logged to a level of detail, which is assessed as suitable to support geological resource modelling.

Table 24 lists the total lengths and percentages of the relevant rock types intersected and logged.

The intersected rock types were:

DIABASE are sills and dikes cutting the pegmatites in some areas; OB is overburden like sand and gravel with clay unconsolidated sediments from the ice ages; PARAGNEISS is one of the host rocks of the pegmatites bodies; PEG are quartz and feldspar pegmatites without spodumene mineralisation; SPD PEG are pegmatites with the typical spodumene mineralisation (the main mining target).

Table 24. Lengths and percentages of the relevant rock types

AREA	Rock Types	Logged Meters	Logged Percentages
MZN	DIABASE	852	5.6
MZN	OB	290	1.9
MZN	PARAGNEISS	13,008	85.4
MZN	PEG	102	0.7
MZN	SPD PEG	976	6.4
TOTAL		15,232	.
MZSW	DIABASE	190	6.5
MZSW	OB	52	1.8
MZSW	PARAGNEISS	2,482	84.4
MZSW	PEG	12	0.4

AREA	Rock Types	Logged Meters	Logged Percentages
MZSW	SPD PEG	205	7.0
TOTAL		2,941	.
HAR	OB	70	2.0
HAR	PARAGNEISS	3,130	89.1
HAR	PEG	31	0.9
HAR	SPD PEG	281	8.0
TOTAL		3,511	
LIN	DIABASE	9	0.2
LIN	OB	93	2.2
LIN	PARAGNEISS	3,498	81.2
LIN	PEG	97	2.2
LIN	SPD PEG	584	13.6
TOTAL		4,305	
CON	DIABASE	11	0.3
CON	OB	110	3.4
CON	PARAGNEISS	2,809	85.8
CON	PEG	33	1.0
CON	SPD PEG	310	9.5
TOTAL		3,274	

The following table lists the rock types within the wireframes. It is obvious that also other rocks than spodumene pegmatite have been included in wireframes, when chemical data of Li_2O were available for these rock types. Rock types logged as pegmatite has been included in some wireframes in order to achieve a lateral continuity of the mineralized bodies considering thin-outs and splitting of the dyke structures. Intervals of rocks included into wireframes without assay data of Li_2O, have been attributed with 0 percent Li_2O, in order to avoid an overestimate in block modelling.

Table 25. Lengths and percentages of the relevant rock types in interpreted wireframes and related average Li_2O grades

AREA	Rock Types	Logged Meters	Logged Percentages	Li_2O Grade [%]
MZN	DIABASE	21	1.4	0.02
MZN	OB	1	0.1	<0.02
MZN	PARAGNEISS	560	39.4	0.10
MZN	PEG	19	1.3	0.07
MZN	SPD PEG	821	57.7	0.84
MZN	UNK	1	0.1	<0.02
TOTAL		1,423	100.0	
MZSW	PARAGNEISS	42	18.3	0.10
MZSW	PEG	2	0.8	0.06
MZSW	SPD PEG	186	80.9	0.72

AREA	Rock Types	Logged Meters	Logged Percentages	Li_2O Grade [%]
TOTAL		230	100.0	
HAR	PARAGNEISS	63	18.1	0.04
HAR	PEG	11	3.1	0.06
HAR	SPD PEG	273	78.8	0.52
TOTAL		346	100.0	
LIN	PARAGNEISS	143	20.1	0.05
LIN	PEG	46	6.5	0.14
LIN	SPD PEG	515	72.6	0.57
LIN	UNK	5	0.8	0.10
TOTAL		710	100.0	
CON	PARAGNEISS	75	19.3	0.06
CON	PEG	21	5.4	0.03
CON	SPD PEG	294	75.1	0.80
CON	UNK	1	0.2	<0.02
TOTAL		391	100.0	

12.8 SAMPLING

Sampling was done on varying sample intervals. However, 2/3 of all samples are ranging between 0.9 and 1.1 m. Thus, the composite length is set to 1 m.

Table 26. Statistics of sample intervals.

Mean		**1.10**
Minimum		0.00
Maximum		3.81
Percentiles	10	0.65
	20	0.90
	30	1.00
	40	1.00
	50	1.00
	60	1.00
	70	1.05
	80	1.50
	90	1.53

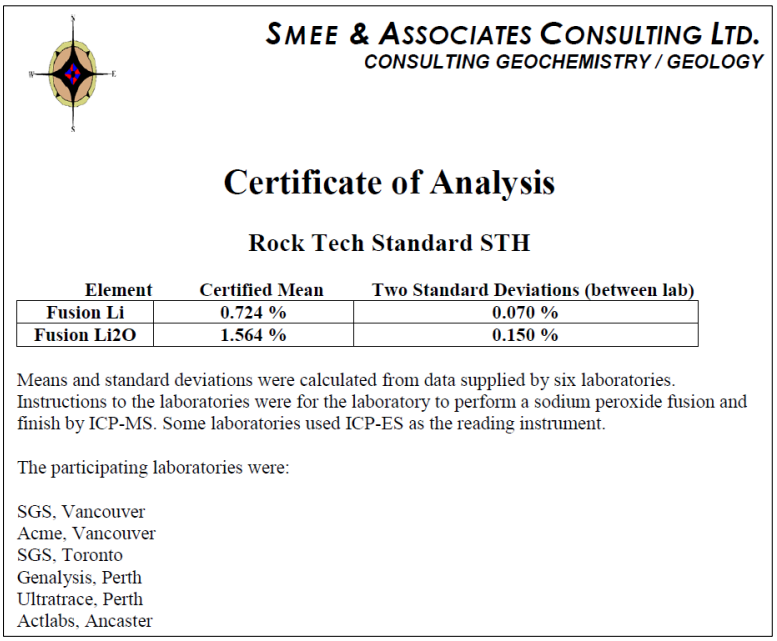
The structure of the pegmatite bodies shows a complex system of thinning, thickening and split-offs. In order to model laterally continuous mineralized wireframes, a significant portion of low mineralized rock or non-mineralized pegmatites was included in these wireframes. Rock, which was not assayed, was attributed with 0 % Li_2O, in order not to overestimate the block model.

Table 27. Overview about non-assayed rock (host rock or low mineralized rock) included in the wireframes (were used with 0 % Li_2O)

AREA	Wireframe Intersection [m]	Non-assayed Interval [m]	Percentage of non-assayed interval
MZN	1428	401	28
MZSW	231	53	23
HAR	347	75	22
LIN	711	145	20
CON	392	46	12

12.9 SAMPLE PREPARATION AND ANALYSIS

Due to the lack of commercially available lithium standards in the ore grade % Li_2O range, Rock Tech created their own lithium standards. The material for the customized standard came from the MZN spodumene pegmatite. Grab samples were sent to CDN Resources Inc., Vancouver in early December 2010 to produce the standards. Two standards were produced: one high grade standard with the original composition of the spodumene pegmatite (STDH) and one that was diluted with pure quartz by 50% to produce a low grade standard (STDL). The standards were pulverized in a large rod mill, screened through 270 mesh and homogenized in a large rotating mixer. A total of 10 g of powdered standard was put in each package. Then 10 samples of each standard were sent to six different labs for a round robin for a total of 60 analyses of STDH and 60 analyses of STDL. The six labs analyzed the Li content in the standards using sodium peroxide fusion digestion with ICP finish to match the method of the primary lab. Once the results of the round robin were completed in early February 2011, Barry Smee, Ph.D., P.Geo. of Smee & Associates Consulting Ltd, North Vancouver compiled the round robin results and calculated the certified mean and standard deviation.



SMEE & ASSOCIATES CONSULTING LTD.
CONSULTING GEOCHEMISTRY / GEOLOGY

Certificate of Analysis

Rock Tech Standard STH

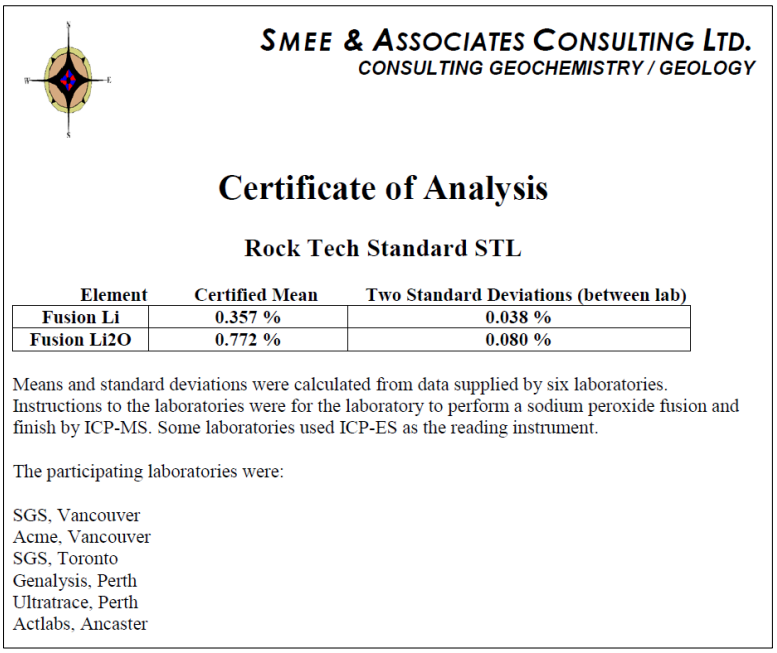
Element	Certified Mean	Two Standard Deviations (between lab)
Fusion Li	0.724 %	0.070 %
Fusion Li2O	1.564 %	0.150 %

Means and standard deviations were calculated from data supplied by six laboratories. Instructions to the laboratories were for the laboratory to perform a sodium peroxide fusion and finish by ICP-MS. Some laboratories used ICP-ES as the reading instrument.

The participating laboratories were:

SGS, Vancouver
Acme, Vancouver
SGS, Toronto
Genalysis, Perth
Ultratrace, Perth
Actlabs, Ancaster

Figure 17. Extract from certificate High Standard.



SMEE & ASSOCIATES CONSULTING LTD.
CONSULTING GEOCHEMISTRY / GEOLOGY

Certificate of Analysis

Rock Tech Standard STL

Element	Certified Mean	Two Standard Deviations (between lab)
Fusion Li	0.357 %	0.038 %
Fusion Li2O	0.772 %	0.080 %

Means and standard deviations were calculated from data supplied by six laboratories. Instructions to the laboratories were for the laboratory to perform a sodium peroxide fusion and finish by ICP-MS. Some laboratories used ICP-ES as the reading instrument.

The participating laboratories were:

SGS, Vancouver
Acme, Vancouver
SGS, Toronto
Genalysis, Perth
Ultratrace, Perth
Actlabs, Ancaster

Figure 18. Extract from certificate Low Standard.

All results of assayed STH and STL fall within the recommended range of two standard deviations, which means a confidence level of around 90 %. One sample was excluded from each the STH and STL plot because of erratic high or low Li_2O concentrations, which must be explained by a labelling mix-up. In consequence, the analytical method applied is assessed as suitable to have produced reliable chemical assay data of Li_2O.

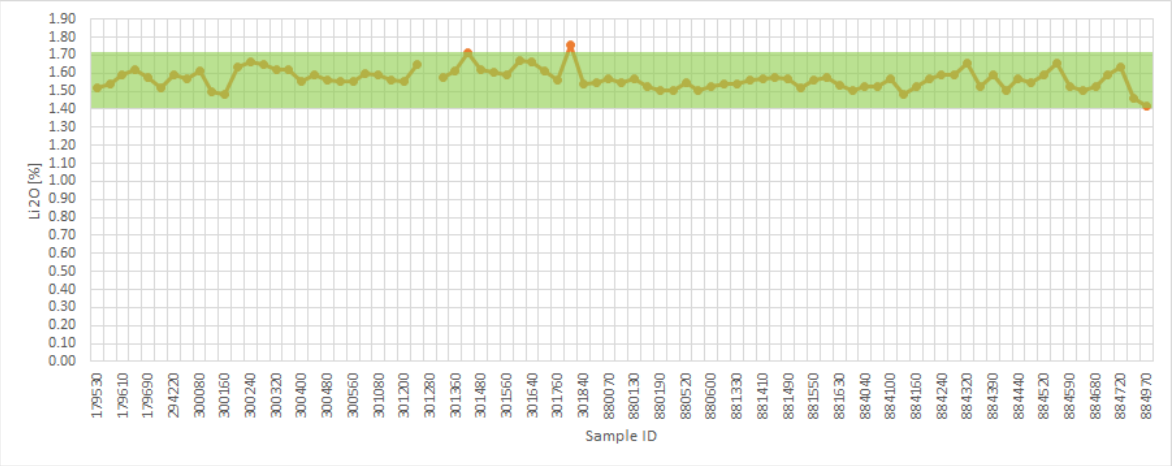


Figure 19. Assay results for CRMs STH for Li_2O.

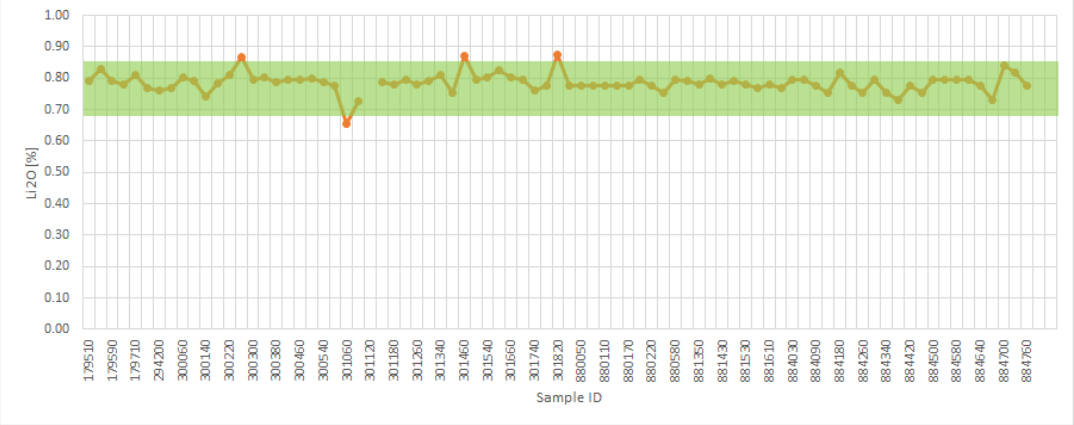


Figure 20. Assay results for CRMs STL for Li_2O.

As blank material dolomite was used. Li_2O concentrations of all 151 blank samples was below 0.2 % Li_2O, except one was 0.4 % Li_2O and another one was erraticly high, which is explained by a labelling mix-up. In consequence, sample preparation method is assessed to be free of contamination.

Quarter core duplicates reproduced Li_2O with a deviation of around 20 %. This increased short-range deviation of Li_2O might be caused by the spodumene minerals, which are long-ish orientated perpendicular to the strike direction of the pegmatite dykes. In summary, the core cutting procedure is assessed to have produced representative results. However, down-hole variography done by CCIC in 2012 confirms a high nugget effect for Li_2O. Very coarse-grain size of spodumene crystals will create a nugget effect, as one ¼ drill core sampling may intersect the spodumene crystal and the other ¼ drill core sample not. Hence, for NQ core half core sampling is the preferred size of sampling.

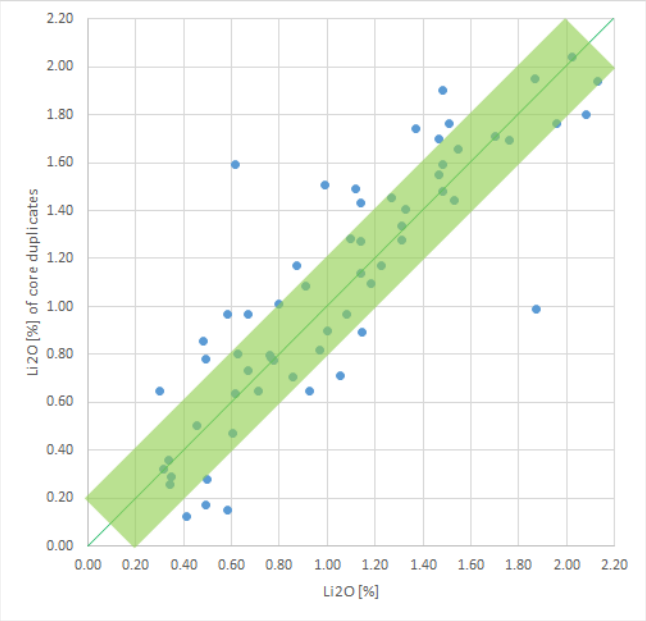


Figure 21. Result of Li_2O of core duplicates.

12.10 DENSITY DETERMINATION

Theoretically spodumene has a density of 3.1 t/m³ in maximum and thus is heavier than the hosting quartz-feldspar matrix assumed to have a theoretical density of 2.65 t/m³. In consequence, it was assumed that the bulk density of spodumene pegmatites is positively correlated to Li_2O concentration.

In total, for 333 samples densities were available. A cross plot of all Li_2O concentrations above 0.65 % (178 samples) vs. density has shown only a poor correlation. It is assumed that an accuracy error in density determination or less-representatively chosen samples are overlying the relative slight difference in density of spodumene and quartz-feldspar, hampering the quality of the correlation.

However, despite high scatter, applying the resulting regression equation, an averaged 1% Li_2O grade results in a bulk density of 2.71 t/m³, which is assessed as a reasonable approach. In consequence, the regression equation was applied to the Li_2O block values to attribute a bulk tonnage.

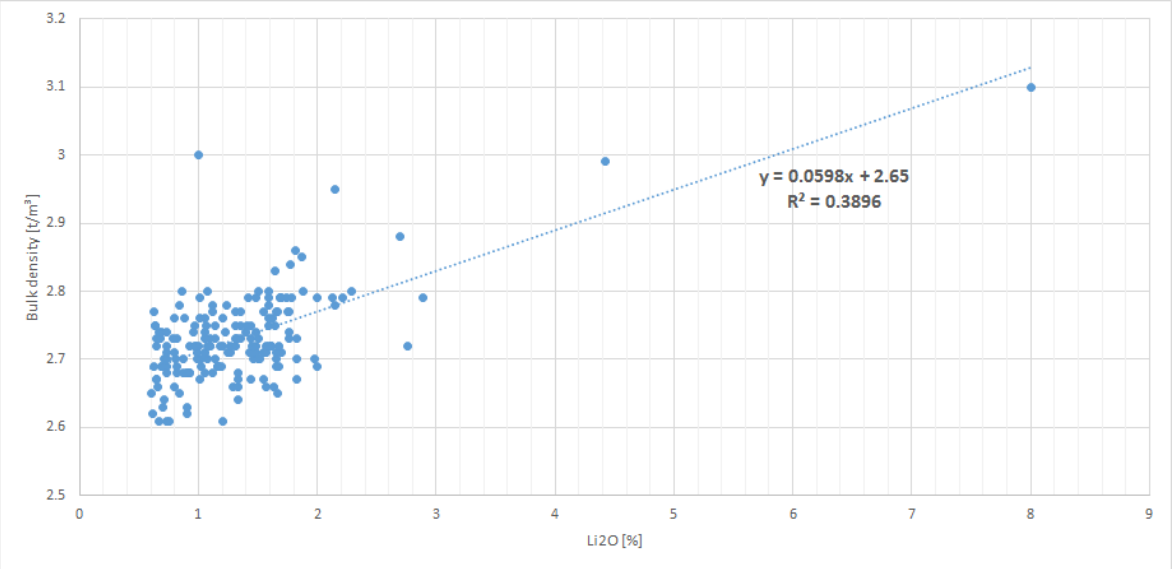


Figure 22. Cross plot bulk density [t/m^3] vs. Li_2O [%] showing a high scatter.

12.11 CONFIRMATION OF HISTORICAL DATA ACQUISITION

In total, 14 drill holes were twinned within a distance not exceeding 15 m in order to confirm the historical database. Historical drill core was not available anymore and thus re-sampling not possible.

The following cross sections show a good spatial re-production of the logged mineralization. In some twin holes the mineralization could be confirmed but some meters up or downwards then expected, which might be explained by poor depth control or inaccurate surveying of historic holes. Table 28 summarizes the logged meterage of spodumene pegmatite, pegmatite and assayed interval and shows differences. In general, logged spodumene pegmatite varies up to several meters, but is balanced in average. In confirmation holes, much more assay data were acquired.

Table 28. Summary of logged meters of spodumene pegmatite, pegmatite and assayed interval and differences in-between

DHID hist	SPDPEG [m]	PEG [m]	Li2O [m]	DHID conf	SPDPEG [m]	PEG [m]	Li2O [m]	SPDPEG DIFF[m]	PEG DIFF [m]	Li2O DIFF [m]
NC-25	21.1		15.3	NC-11-14	19.1		26.4	-1.9	0.0	11.1
NC-30	15.1	1.3	13.6	NC-11-02	19.2	1.2	33.2	4.1	-0.1	19.7
NC-23	17.8	0.6	15.5	NC-11-04	25.6	2.6	45.1	7.8	2.0	29.7
NC-27	10.5		10.4	NC-11-13	9.1	0.4	21.2	-1.3	0.4	10.8
NC-44	18.9	0.5	15.2	NC-11-06	13.1	1.8	40.7	-5.7	1.3	25.4
NC-04	5.9	0.2	1.8	NC-11-09	12.6	1.9	32.4	6.7	1.7	30.6
NC-37	9.7	4.1	10.4	NC-11-08	4.0	0.5	12.7	-5.7	-3.7	2.2
NC-18	9.6	2.6	9.1	NS-11-03	11.1		22.6	1.5	-2.6	13.4
HM-01	11.3			HW-11-03	12.3	0.4	18.7	1.0	0.4	18.7
NC-HM-21	23.0		20.1	L60-11-01	14.4	2.1	28.0	-8.6	2.1	7.9
NC-HM-28	27.9		27.9	L60-11-02	21.1		23.1	-6.8	0.0	-4.8
NC-HM-38	19.8	1.2	19.8	L60-11-03	14.8	21.5	42.6	-5.0	20.3	22.8
CON-15	12.6		12.6	CW-11-05	10.6		15.2	-2.0	0.0	2.6
CON-12	11.0	1.9	10.8	CW-11-02	21.0		28.8	10.0	-1.9	18.0

Table 29 summarizes the Li_2O grade of the assayed intervals applying a cut-off grade of 0.65 % Li_2O. Cumulative intervals with values of at least 0.65 % Li_2O or above are only in 4 confirmation holes higher than in historic holes, but lower for the remaining drill holes.

Table 29. Summary of the Li_2O average grade and corresponding assayed intervals applying a cut-off grade of 0.6 % Li_2O

DHID hist	Interval [m]	Li2O [%]	DHID conf	Interval [m]	Li2O [%]	Interval DIFF [m]	Li2O DIFF [%]
NC-25	15.2	1.27	NC-11-14	13.6	1.22	-1.64	-0.05
NC-30	8.9	0.79	NC-11-02	13.9	1.10	4.92	0.31
NC-23	13.3	1.08	NC-11-04	21.2	1.23	7.92	0.15
NC-27	5.0	1.53	NC-11-13	4.4	1.11	-0.59	-0.43
NC-44	10.3	1.04	NC-11-06	8.7	1.36	-1.63	0.32
NC-04	1.8	0.99	NC-11-09	4.5	1.05	2.72	0.06
NC-18	7.6	0.85	NS-11-03	6.5	1.09	-1.04	0.24
NC-HM-21	13.1	0.90	L60-11-01	10.2	1.43	-2.89	0.53
NC-HM-28	4.5	1.14	L60-11-02	1.0	1.06	-3.55	-0.09
NC-HM-38	10.6	0.96	L60-11-03	11.2	1.07	0.62	0.11
CON-15	6.9	1.23	CW-11-05	4.6	1.19	-2.31	-0.04
CON-12	8.2	0.96	CW-11-02	5.5	0.91	-2.70	-0.05

In general, results of the historic holes were confirmed considering the short range variations and nugget effects described above. Hence, results of the historic holes are assessed as suitable for resource modelling and estimation. Figure 23 to **Figure 26** display cross sections of the above given confirmation holes.

Figure 23 shows an example of good reproduction of the mineralized dyke in both holes, while Figure 24 shows an example of good re-production, but discrepancy in depths. Figure 25 shows an example where the confirmation hole intersected pegmatites while in the historic hole no pegmatites were logged. Figure 26 shows an example where the confirmation hole had no pegmatites logged, while the historic logs shows pegmatites. In such cases interpretation was supported by the results in the surrounding drill holes.

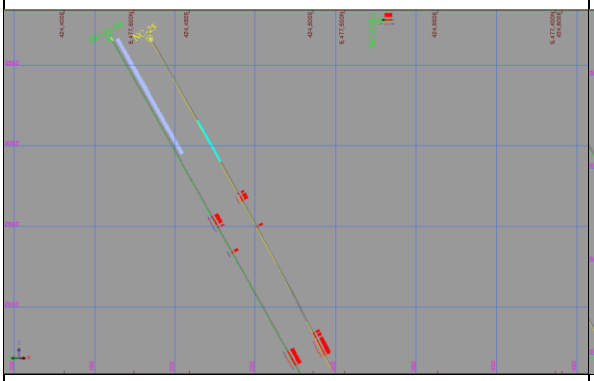


Figure 23. Twin drilling MZN: NC-11-14 twinned NC-25. Grid spacing 50 m.

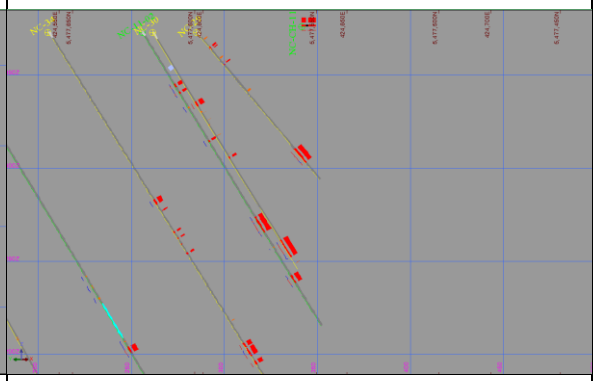


Figure 24. Twin drilling MZN: NC-11-03 twinned NC-30 Grid spacing 50 m.

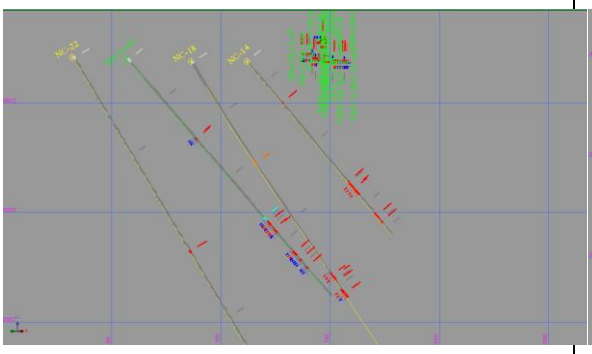


Figure 25. Twin drilling MZSW: NC-11-03 twinned NC-18 Grid spacing 50 m.

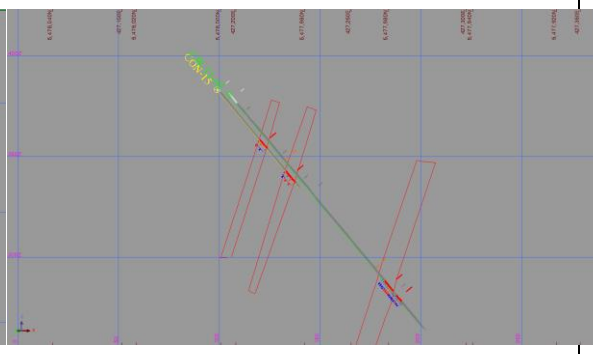


Figure 26. Twin drilling CON: CW-11-05 twinned CON-15 Grid spacing 50 m.

12.12 VERIFICATION OF CHANNEL SAMPLES

The following table shows the mean grades of Li_2O for recent drilling and channels. Applying a Li_2O cut-off grade of 0.65 % the mean Li_2O grade of channels samples meet the results of drilling with an error below 5 %. In consequence, the assays of channels have been used for resource modelling as well.

Table 30. Mean grade of Li2O of recent drill and channel samples

TYPE	Number of holes	Li2O [%] all samples	Li2O [%] samples >0.65 % Li2O
Channel	138	0.75	1.30
Drillholes	62	0.52	1.24

12.13 CONCESSION AREA

Rock Tech holds 283 claims (56.93 km^2) and 81 dispositions (10.42 km^2). Claims and dispositions were given to DMT as shape files and checked for spatial consistency with drill holes and trenches used for resource modelling and estimation. Also a general database was used to check all data.

A list off all exploration claims and all dispositions are listed and shown in Chapter 4.1. Description of Ownership. A list of the renumbering of the claims and disposition is added in Appendix 3

12.14 DIGITAL TERRAIN MODEL

A topographic survey is available at 20 m x 20 m spacing is available and was checked for consistency with collar elevations. No major off-set was observed.

12.15 MINED OUT AREA

As mentioned above shaft sinking is reported, but production status was not reached. Hence, mining has not taken place to date. Old plans show a final depth of that shaft of 152,65 m below the surface. 3 levels ware planned 48, 93 and 137 m below surface.

12.16 DATA QUALITY SUMMARY

DMT assesses, that the quality and quantity of data available is sufficient to state a resource in compliance to NI 43-101.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

One bulk sample was produced from three short drill holes on MZN (BK-11-03, 04, 05) Feb. 16-18, 2011. In addition, three blasted grab samples were collected from three sites (two from MZN and one from MZSW). These bulk samples were considered to be representative of the style and type of mineralization and the mineral deposit as a whole. All material (in total about 770 kg) were shipped to SGS Metallurgical Operations, Lakefield, Ontario in February 2011. For processing tests the material was composited to produce a composite head sample representative of the spodumene mineralization on the property. The composite head sample contained 1.49 %Li_2O and consisted of quartz, albite, spodumene and minor microcline and muscovite. DMT notes that these samples yielded a head grade of 1.49 % Li_2O, which is higher than the average grade of the new resource estimate.

After the successful generation of an initial Li_2CO3 product SGS, Lakefield site focused on processing concentrates through the standard lithium carbonate hydrometallurgical flow sheet. The program examined three concentrate ("con") samples: one low Fe con, one high Fe con and one heavy media con.

The low Fe and high Fe cons were produced using floatation methods and the heavy media con was produced using heavy liquid separation. After two bicarbonate polishing tests, the resulting solids met all of the product specifications and had a Li_2CO3 grade of 99.988% and 61 ppm Ca. This indicates that a high grade Li_2CO3 product can be produced from Rock Tech's mineralized samples. For details of investigations and results the reader should refer to technical report of resource estimate prepared by CCIC in 2012 and the SGS Report 2011.

14 MINERAL RESOURCE ESTIMATES

Independent, NI 43-101 compliant resources at the Georgie lake properties of Rock Tech were estimated using validated and verified historical drill hole data, results from the 2010/11 drill and trenching programs conducted by CCIC on behalf of Rock Tech, and results of further drilling and trenching since 2012 conducted by Arriva Management Inc. Vancouver on behalf of Rock Tech.

The resource model and estimate of this chapter are dated 18.04.2018.

14.1 GEOLOGICAL MODEL

For the five main pegmatites located in the NSPA a geological 3D model was built. The general concept, which underlies the wireframe interpretation is based on tabular mineralized bodies following dyke structures with a general orientation and extent of the main pegmatites as follows:

MZN: strikes N 55º E and dips 70º NW; strike length 1000 m

MZSW: strikes N 45º E, dips 70º NW; strike length 300 m

HAR: strikes N30ºE, dips 70º NW; strike length 450 m

LIN: strikes N30ºE, dips 70º NW; strike length 450 m

CON: strikes N 30º E, dips 70º NW; strike length 800 m

The pegmatite veins show varying concentrations of spodumene and are showing thinning, thickening and split-offs. The pegmatites are partly exposed outcrops. The remainder is overlain by glacial deposits consisting mainly of stratified accumulations of unconsolidated

sand and gravel. The depth of glacial erosion is still unknown. Further investigations will be required.

Diabase veins intersect the pegmatites of MZN vertically and horizontally.

14.2 STATISTICAL ANALYSIS

In total 38 wireframes were interpreted with a total volume of 8.74 Mm³. It should be noted that waste rocks (pegmatites without spodumene or host rocks) were partially included in these wireframes in order to achieve continuous bodies even spodumene pegmatites are showing thin-outs and split-offs.

The following basic statistics tabulations show average Li_2O and average densities for several Li_2O cut-off grades applied to drilled material inside the wireframes. It is obvious that around 40 % of the wireframed material has Li_2O concentrations less than 0.2 % for MZN, MZSW and CON, and 60 % for HAR and LIN. This is material of weakly mineralized spodumene pegmatites, un-mineralized pegmatites or host rock, which were included in the wireframes to respect thinning, thickening and split-offs. This approach ensured continuity in the wire framing. In order to avoiding over-estimation un-sampled intervals were set to 0 % Li_2O. This has not been done in the resource estimate prepared by CCIC in 2012, which is acceptable as an inferred resource with higher uncertainties, but must be done for resource estimate at higher confidence levels than inferred.

Table 31. Average Li_2O and Densities at several Li_2O cut-off grades for wireframe intersections shown for each of the 5 main pegmatites (MZN, MZSW, HAR, LIN and CON)

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
MZN	0.0	1508	2.68	0.53	100.00%
MZN	0.2	866	2.70	0.90	57.43%
MZN	0.4	645	2.72	1.11	42.78%
MZN	0.6	574	2.72	1.19	38.02%
MZN	0.8	482	2.73	1.28	31.99%
MZN	1.0	380	2.73	1.38	25.16%
MZN	1.2	268	2.74	1.50	17.80%
MZN	1.4	162	2.75	1.64	10.73%
MZN	1.6	84	2.76	1.77	5.56%
MZN	1.8	24	2.77	1.96	1.62%
MZN	2.0	7	2.78	2.12	0.46%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
MZSW	0.0	231	2.69	0.60	100.00%
MZSW	0.2	144	2.71	0.94	62.42%
MZSW	0.4	129	2.71	1.01	55.81%
MZSW	0.6	109	2.72	1.11	47.36%
MZSW	0.8	82	2.72	1.25	35.29%
MZSW	1.0	65	2.73	1.33	27.92%
MZSW	1.2	42	2.74	1.45	18.31%
MZSW	1.4	26	2.74	1.56	11.26%
MZSW	1.6	8	2.76	1.77	3.53%
MZSW	1.8	4	2.76	1.91	1.56%
MZSW	2.0	1	2.77	2.05	0.43%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
HAR	0.0	364	2.68	0.43	100.00%
HAR	0.2	163	2.70	0.89	44.92%
HAR	0.4	137	2.71	1.00	37.65%
HAR	0.6	111	2.72	1.12	30.61%
HAR	0.8	86	2.72	1.25	23.69%
HAR	1.0	68	2.73	1.33	18.76%
HAR	1.2	42	2.74	1.48	11.66%
HAR	1.4	20	2.75	1.67	5.57%
HAR	1.6	8	2.77	1.97	2.18%
HAR	1.8	5	2.78	2.17	1.36%
HAR	2.0	3	2.79	2.40	0.82%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
LIN	0.0	731	2.68	0.44	100.00%
LIN	0.2	336	2.70	0.90	45.95%
LIN	0.4	277	2.71	1.04	37.84%
LIN	0.6	222	2.72	1.17	30.42%
LIN	0.8	174	2.73	1.31	23.82%
LIN	1.0	140	2.73	1.41	19.15%
LIN	1.2	95	2.74	1.56	13.03%
LIN	1.4	67	2.75	1.67	9.20%
LIN	1.6	35	2.76	1.82	4.76%
LIN	1.8	14	2.77	1.98	1.96%
LIN	2.0	6	2.78	2.11	0.80%

AREA	Cut-off Li2O [%]	Drilled interval [m]	Density [t/m³]	Li2O [%]	Drilled interval [%]
CON	0.0	412	2.69	0.63	100.00%
CON	0.2	271	2.71	0.93	65.73%
CON	0.4	231	2.71	1.04	56.20%
CON	0.6	205	2.72	1.12	49.87%
CON	0.8	158	2.72	1.23	38.50%
CON	1.0	111	2.73	1.38	26.91%
CON	1.2	70	2.74	1.54	16.93%
CON	1.4	44	2.75	1.68	10.66%
CON	1.6	24	2.76	1.82	5.80%
CON	1.8	10	2.77	1.97	2.52%
CON	2.0	3	2.78	2.15	0.84%

The following figures show frequency plots of Li_2O for all 3D modelled pegmatites. It is obvious that there are two sample populations included in the wireframes, mineralization of spodumene pegmatites and weakly and non-mineralized rocks. However, both sample population are following broadly a normal distribution without a significant amount of outlier. Hence, grade capping has not been applied.

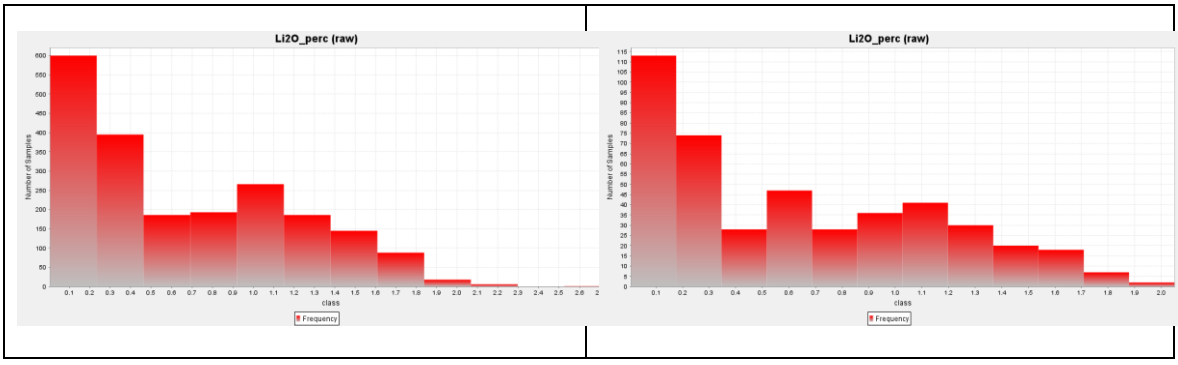


Figure 27. Frequency plots of Li_2O for MZN.	Figure 28. Frequency plots of Li_2O for MZSW

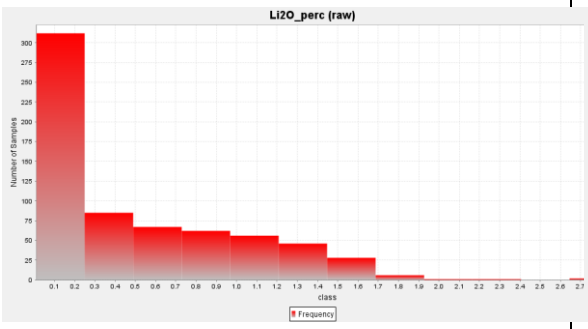
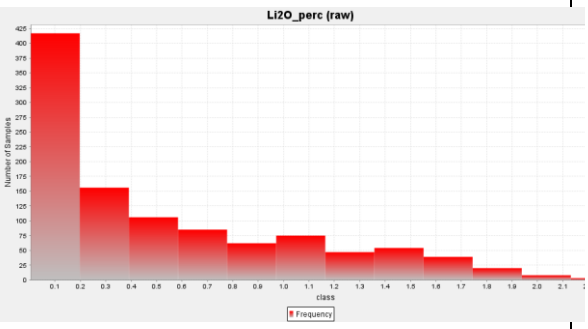

	

Figure 29. Frequency plots of Li_2O for HAR	**Figure 30. Frequency plots of Li_2O for LIN**

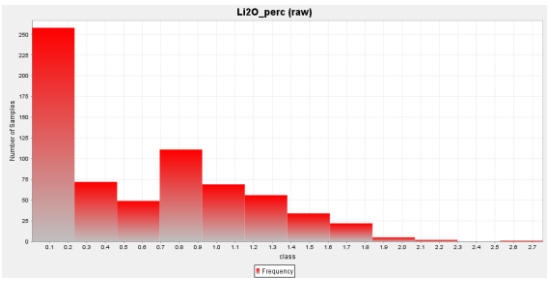



Figure 31. Frequency plots of Li_2O for CON	

14.3 INTERPRETATION OF MINERALIZED ZONES (DOMAINS)

Wireframes were modelled following the concept of a dyke structures of the pegmatite bodies comprising spodumene. The interpretation was based on sections perpendicular to strike on a hole to hole interpretation following the general dip of the pegmatites from hole to hole. The wireframes were extended to undrilled areas with around 50 m in maximum based on the general drill spacing.

14.4 WIREFRAME MODEL

In total, 38 wireframes were interpreted with a total volume of 8.74 Mm³. The following Figures show 3D views of these wireframes Figure 44.

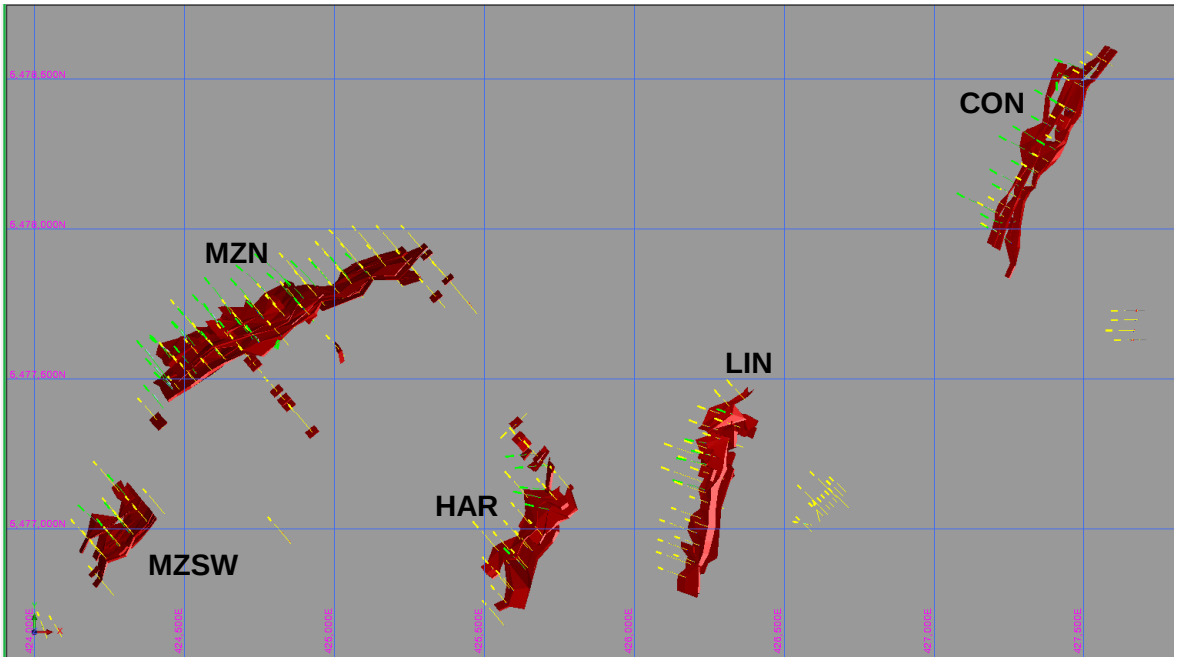


Figure 32. Plan view onto wireframes of spodumene pegmatites (red) with in the NSPA. Grid spacing 500 m.

Table 29 shows the volume of the wireframes. These wireframes build the basis of the 3D model of the pegmatites.

Table 32: Volume of the wireframes [Mm³] of MZN, MZSW, HAR, LIN, CON

AREA	Volume of Wireframes [Mm³]
MZN	4.00
MZSW	0.75
HAR	1.17
LIN	1.74
CON	1.08
TOTAL	8.74

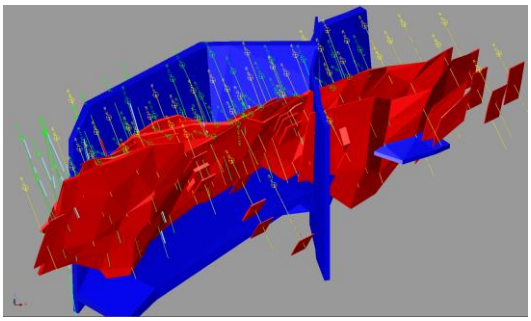


Figure 33. 3D view of MZN interpreted wireframe of pegmatite (red) and diabase (blue); looking NW. Estimated dimension (2D) pegmatite 1000 x 250 m.

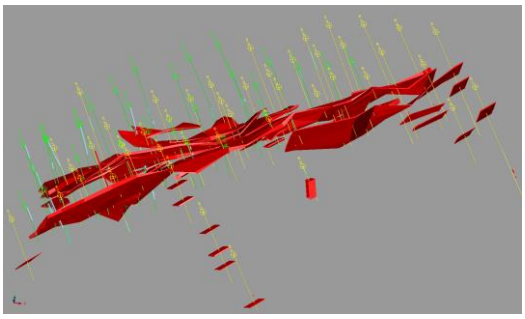


Figure 34. 3D view of MZN interpreted wireframe of pegmatite (red); looking from above along dip. Estimated dimension (2D) pegmatite 1050 x 250 m.

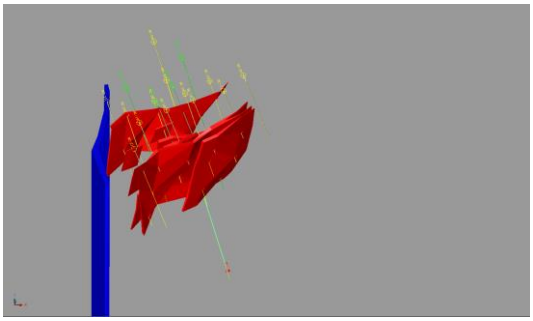


Figure 35. 3D view of MZSW interpreted wireframe of pegmatite (red) and diabase (blue); looking NW. Estimated dimension pegmatite (2D) 300 x 250 m.

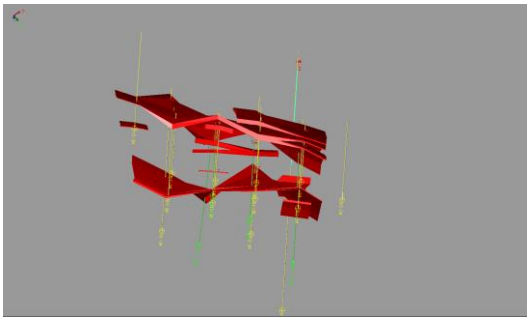


Figure 36. 3D view of MZSW interpreted wireframe of pegmatite (red); looking from above along dip. Estimated dimension pegmatite (2D) 300 x 250 m.

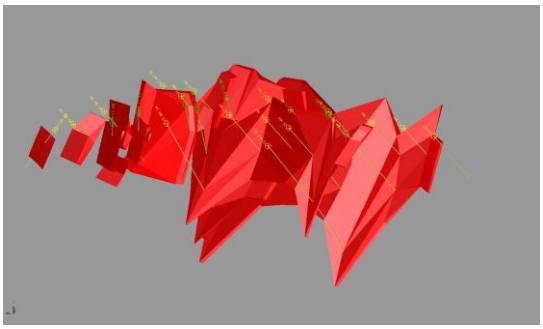


Figure 37. 3D view of HAR interpreted wireframe of pegmatite (red); looking NE. Estimated dimension pegmatite (2D) 500 x 250 m.

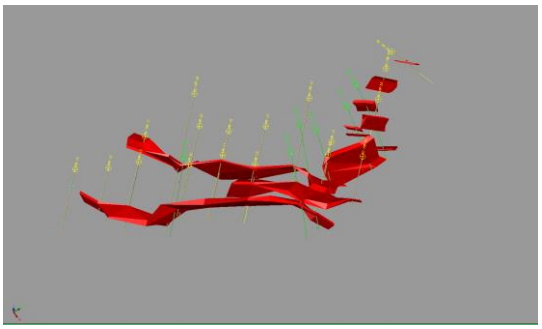


Figure 38. 3D view of HAR interpreted wireframe of pegmatite (red); looking from above along dip. Estimated dimension pegmatite (2D) 500 x 250 m.

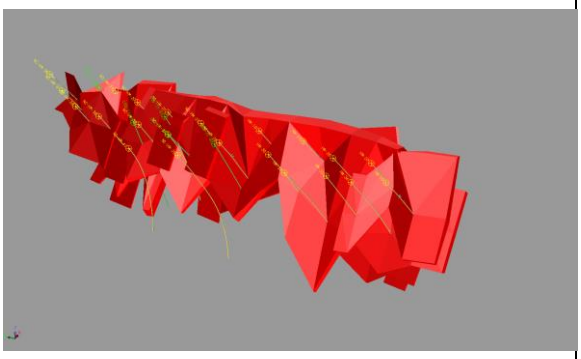


Figure 39. 3D view of LIN interpreted wireframe of pegmatite (red); looking NE. Estimated dimension pegmatite (2D) 650 x 200 m.

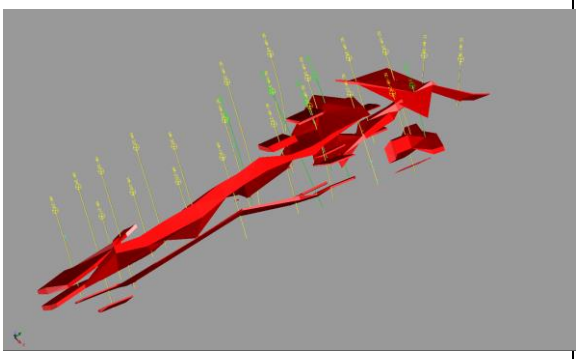


Figure 40. 3D view of LIN interpreted wireframe of pegmatite (red); looking from above along dip. Estimated dimension pegmatite (2D) 650 x 200 m.

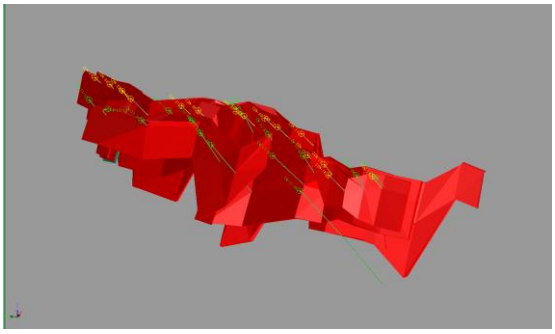


Figure 41. 3D view of CON interpreted wireframe of pegmatite (red); looking NE. Estimated dimension pegmatite (2D) 750 x 200 m.

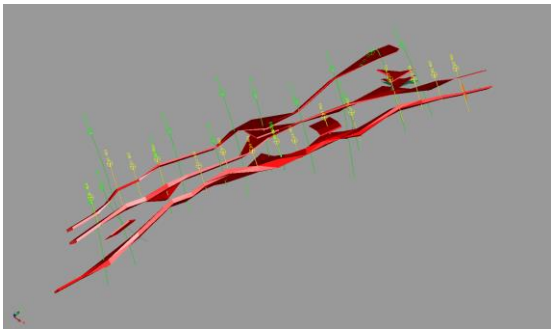


Figure 42. 3D view of CON interpreted wireframe of pegmatite (red); looking from above along dip. Estimated dimension pegmatite (2D) 750 x 200 m.

14.5 GRADE CAPPING / COMPOSITING / BLOCK MODEL DEFINITION

No grade capping is applied to composites, because no significant amount of outlier have been observed in the frequency plots.

Sampling was done on varying sample intervals. However, 2/3 of all samples is ranging in-between 0.9 and 1.1 m. Thus, the composite length is set to 1 m.

Parent block size has been set to X=2m, Y=2m, Z=5m. Considering the general drill spacing is around 50 m x 50 m this is only 4 %. However, the smaller block sizes were defined in order to consider the often thin and very steep and varying dipping pegmatite dykes. As long as the mining method is not chosen and related selectivity and minimum mining unit is unknown, this smaller blocks give the possibility to re-block to larger units. The older resource estimate done by CCIC in 2012 was based on 5 x 5 x 5 m blocks.

14.6 GEOSTATISTICS / INTERPOLATION METHOD / BLOCK MODEL

Geostatistical analysis has been done in order to analyse continuity of the mineralisation and to define the kriging parameters with the objective to minimize the interpolation error. The only high quality variogram was possible for MZN, because of the high drill hole density. The results were applied to other main pegmatite bodies assuming a very similar distribution of Li_2O. Based on the results the range is set to 150 m, the nugget to 0.04 and sill to 0.06.:

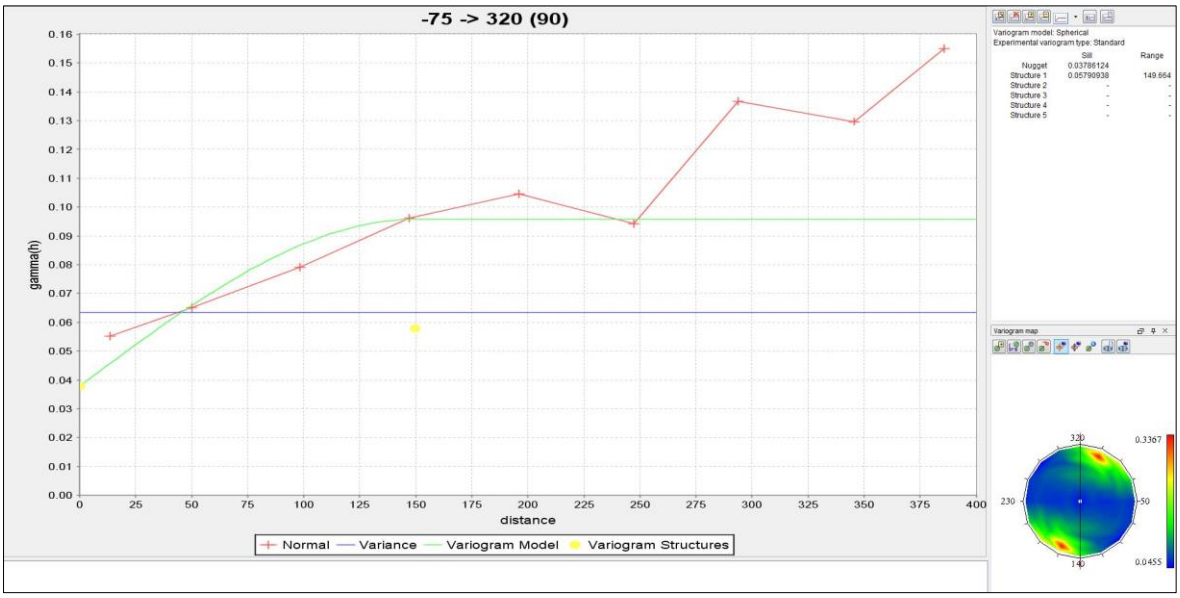


Figure 43. Experimental variogram (red) and variogram model (green) for MZN.

The ordinary kriging was done in 10 passes with increasing search radii from 15 m to 150 m in major and semi-major direction and from 2 m to 20 m in minor direction successively.

This was done to ensure that blocks very near a sample are not effected by far away samples and thus minimize possible dilution. A minimum of 1 sample and a maximum of 15 samples were used in the estimation of individual blocks.

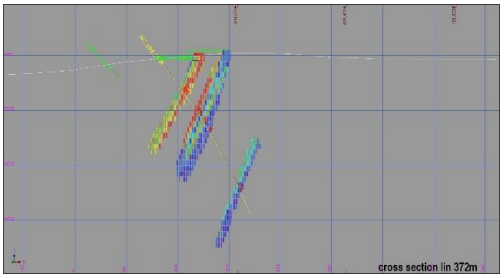
Table 33. Orientation of search ellipsoid for the 5 areas following the dip direction and dip of the main pegmatite bodies

Area	Bearing [degree]	Plunge [degree]
MZN	320	-75
MZSW	320	-75
HAR	320	-75
LIN	290	-65
CON	300	-70

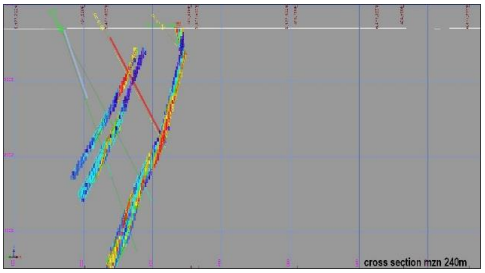
The wireframed volume was assigned to the block model using the partial percentage attribute in Surpac. Each block was assigned a volume correction factor which is the proportion of block volume within the domained wireframe and below topography. Thus, a volume discrepancy based on stair step effects relative to block size could be avoided.

The **bulk tonnage** was calculated based on block volume (2m x 2m x 5m) * (bulk density) * (the proportion of the block within the solid and under the surface topography)

The **Li_2O average grade** related to this tonnage is the arithmetic average of Li_2O weighted by the bulk tonnage of each block.



EW-SE section at station 372 m from the south end of the LIN pegmatite (grid spacing 50 m)



NW-SE section at station 240 m from the south end of the MZN pegmatite (grid spacing 100 m)

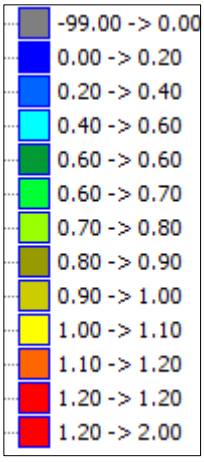


Figure 44 sections of the block model colour shows the Li²O percentage of the blocks

The sections of the block model of two pegmatites show the distribution of the Li²O contend within the model.

14.7 RESOURCE CLASSIFICATION

The definitions for resource categories used in this report are consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves 2014.

Under the CIM classification system, a Mineral Resource is defined as:

> *…" a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.*
>
> *"The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."*

Resources are classified into Measured, Indicated and Inferred categories based upon geological knowledge and confidence (Figure 45). Mineral resources are not mineral reserves and do not have demonstrated economic viability.

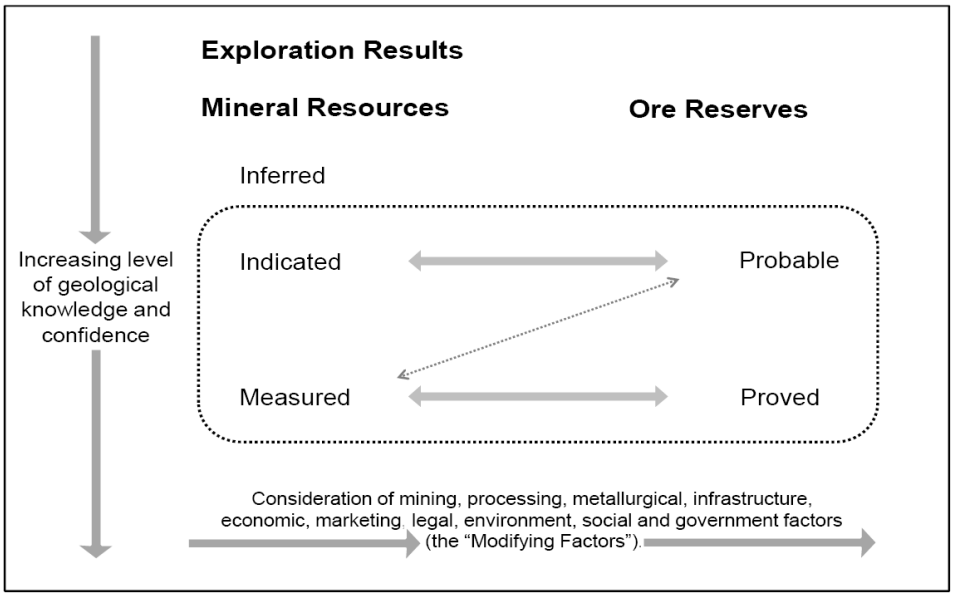


Figure 45. Relationship between Exploration Results, Mineral Resources & Ore Reserves

Resource classification within mineralisation envelopes is generally based on drillhole spacing, grade continuity, and overall geological continuity. The distance to the nearest composite and the number of drillholes are also considered in the classification.

In classifying the resource estimate, the following key factors have been considered:

- Confidence in data quantity and specifically sample spacing of Li_2O data;
- Confidence in the geological interpretation and continuity (geological complexity); and
- Confidence in mineralisation / grade continuity (complexity of spatial grade distribution).

Considering the above, the following criteria have been applied for classification into the various mineral resource categories for this estimate.

Measured resources

- All blocks within the wireframed constraints and a distance to the nearest Li_2O sample being less than 20 m

Indicated resources

- All blocks within the wireframed constraints and a distance to the nearest Li_2O sample being equal or above 20 m and less than 70 m.
- All blocks with a distance to the nearest Li_2O sample, which is above 10 m below topography.

Inferred resources

- All blocks within the wireframed pegmatites which are not defined as Measured or Indicated but are included in the interpreted wireframes,
- Outside the block model with in NSPA, derived by applying an average true thickness of spodumene pegmatite and extrapolating the pegmatite bodies in maximum 50 m below the deepest drillhole in the 3D model.
- And all drilled material of isolated drill holes which were not used for modelling within the claims and dispositions.

14.8 PRELIMINARY CUT-OFF GRADE ASSUMPTIONS

Following international requirements a Li_2O cut-off grade was applied to constrain the estimated mineral resources and to demonstrate reasonable prospects for eventual economic extraction.

The reporting cut-off grade of 0.65 Li_2O% was chosen based on the benchmarking of similar Lithium projects, but is not based on a financial model specific for this project, but based on comparable projects.

14.9 MODEL VALIDATION

In order to check that the interpolation of Li_2O has worked appropriately, the interpolated block model has been validated against the corresponding domained primary data from drilling using the following techniques:

- Visual inspection of block grades in plan and section and comparison with drill hole grades (Appendix 1); and
- Comparison of mean grades of block model data with primary data within mineralised domain at several Li_2O cut-off grades (Table 34 and Table 35).

Table 34 compares primary data from drilling with estimated block values. It is obvious that drilled Li_2O concentrations and frequencies are representatively estimated in the block model. Slight discrepancy are caused by the drill pattern, which is producing 100 % regular intersection and by smoothing, which is typical in the interpolation process. In this case smoothing might be somewhat pronounced because waste rock was included within the domained wireframes. However, near the 0.65 % Li_2O cut-off the averaged block values are generally slightly lower than averaged values from drilling. In consequence, an overestimation caused by interpretation and interpolation can be excluded.

Table 34. Model validation for MZN and MZSW

		Drilling Data				Block Data			
AREA	Cut-off Li2O [%]	Drilled interval [m]	Drilled interval [%]	Density [t/m³]	Li2O [%]	Tonnage [Mt]	Tonnage [%]	Density [t/m³]	Li2O [%]
MZN	0.0	1508	100.0%	2.68	0.53	10.43	100.0%	2.68	0.52
MZN	0.2	866	57.4%	2.70	0.90	7.47	71.6%	2.69	0.70
MZN	0.4	645	42.8%	2.72	1.11	5.65	54.1%	2.70	0.83
MZN	0.6	574	38.0%	2.72	1.19	3.94	37.8%	2.71	0.97
MZN	0.8	482	32.0%	2.73	1.28	2.68	25.7%	2.72	1.10
MZN	1.0	380	25.2%	2.73	1.38	1.62	15.5%	2.72	1.24
MZN	1.2	268	17.8%	2.74	1.50	0.84	8.0%	2.73	1.37
MZN	1.4	162	10.7%	2.75	1.64	0.29	2.8%	2.74	1.54
MZN	1.6	84	5.6%	2.76	1.77	0.07	0.7%	2.75	1.72
MZN	1.8	24	1.6%	2.77	1.96	0.01	0.1%	2.76	1.88
MZN	2.0	7	0.5%	2.78	2.12	0.00	0.0%	2.78	2.13
MZSW	0.0	231	100.0%	2.69	0.60	2.00	100.0%	2.68	0.53
MZSW	0.2	144	62.4%	2.71	0.94	1.39	69.4%	2.69	0.74
MZSW	0.4	129	55.8%	2.71	1.01	1.10	55.0%	2.70	0.85
MZSW	0.6	109	47.4%	2.72	1.11	0.80	40.3%	2.71	0.97
MZSW	0.8	82	35.3%	2.72	1.25	0.53	26.5%	2.72	1.12
MZSW	1.0	65	27.9%	2.73	1.33	0.32	16.1%	2.73	1.27
MZSW	1.2	42	18.3%	2.74	1.45	0.18	9.1%	2.73	1.41
MZSW	1.4	26	11.3%	2.74	1.56	0.09	4.7%	2.74	1.51
MZSW	1.6	8	3.5%	2.76	1.77	0.00	0.2%	2.75	1.66
MZSW	1.8	4	1.6%	2.76	1.91	0.00	0.0%	2.74	1.84
MZSW	2.0	1	0.4%	2.77	2.05				

Table 35. Model validation for HAR, LIN and CON

		Drilling Data				Block Data			
AREA	Cut-off Li2O [%]	Drilled interval [m]	Drilled interval [%]	Density [t/m³]	Li2O [%]	Tonnage [Mt]	Tonnage [%]	Density [t/m³]	Li2O [%]
HAR	0.0	364	100.0%	2.68	0.43	3.07	100.0%	2.67	0.32
HAR	0.2	163	44.9%	2.70	0.89	1.28	41.7%	2.69	0.66
HAR	0.4	137	37.6%	2.71	1.00	0.93	30.4%	2.70	0.80
HAR	0.6	111	30.6%	2.72	1.12	0.65	21.1%	2.71	0.94
HAR	0.8	86	23.7%	2.72	1.25	0.43	14.1%	2.71	1.05
HAR	1.0	68	18.8%	2.73	1.33	0.25	8.1%	2.72	1.17
HAR	1.2	42	11.7%	2.74	1.48	0.09	2.9%	2.73	1.28
HAR	1.4	20	5.6%	2.75	1.67	0.01	0.2%	2.74	1.48
HAR	1.6	8	2.2%	2.77	1.97	0.00	0.0%	2.76	1.75
HAR	1.8	5	1.4%	2.78	2.17	0.00	0.0%	2.77	1.84
HAR	2.0	3	0.8%	2.79	2.40				
LIN	0.0	731	100.0%	2.68	0.44	4.62	100.0%	2.67	0.33
LIN	0.2	336	46.0%	2.70	0.90	2.06	44.5%	2.69	0.69
LIN	0.4	277	37.8%	2.71	1.04	1.46	31.7%	2.70	0.85
LIN	0.6	222	30.4%	2.72	1.17	1.02	22.1%	2.71	0.99
LIN	0.8	174	23.8%	2.73	1.31	0.69	14.8%	2.72	1.14
LIN	1.0	140	19.1%	2.73	1.41	0.45	9.7%	2.73	1.27
LIN	1.2	95	13.0%	2.74	1.56	0.25	5.5%	2.73	1.41
LIN	1.4	67	9.2%	2.75	1.67	0.12	2.6%	2.74	1.55
LIN	1.6	35	4.8%	2.76	1.82	0.03	0.7%	2.75	1.70
LIN	1.8	14	2.0%	2.77	1.98	0.00	0.1%	2.76	1.86
LIN	2.0	6	0.8%	2.78	2.11	0.00	0.0%	2.82	2.05
CON	0.0	412	100.0%	2.69	0.63	2.89	100.0%	2.68	0.53
CON	0.2	271	65.7%	2.71	0.93	1.99	68.9%	2.69	0.74
CON	0.4	231	56.2%	2.71	1.04	1.58	54.5%	2.70	0.86
CON	0.6	205	49.9%	2.72	1.12	1.22	42.2%	2.71	0.97
CON	0.8	158	38.5%	2.72	1.23	0.84	29.0%	2.71	1.08
CON	1.0	111	26.9%	2.73	1.38	0.48	16.4%	2.72	1.23
CON	1.2	70	16.9%	2.74	1.54	0.20	7.0%	2.73	1.40
CON	1.4	44	10.7%	2.75	1.68	0.08	2.8%	2.74	1.56
CON	1.6	24	5.8%	2.76	1.82	0.02	0.8%	2.75	1.69
CON	1.8	10	2.5%	2.77	1.97	0.00	0.1%	2.76	1.91
CON	2.0	3	0.8%	2.78	2.15	0.00	0.0%	2.78	2.06

The resource data are closely aligned with the historic resource regarding the average grades within a 10 % error range. Slightly lower Li_2O concentrations of the DMT resource estimate might be caused by attributing waste rocks or un-sampled material with 0 % Li_2O, which was not done by CCIC in 2012. However, a proper model validation was not available by CCIC in 2012 (see Table 11).

The comparison illustrates that no obvious bias has been introduced during the block modelling process. On the basis of its review and validation procedures, DMT is of the opinion that the block model is valid and acceptable for estimating Mineral Resources.

14.10 ESTIMATE OF MINERAL RESOURCES

DMT prepared an independent, NI 43-101 compliant resource estimate for the Georgia Lake Lithium Property within all claims and dispositions which belongs to Rock Tech today. The total resource estimate described in this report is based on different areas within the claims and dispositions of Rock Tech. For this report DMT divided the claim and disposition area from Rock Tech into 2 Areas (north and south). The Northern Spodumene Pegmatite Area "NSPA" and the Southern Spodumene Pegmatite Area "SSPA" see Figure 2. All mentioned pegmatites in this report are situated with in the claims and dispositions of Rock Tech.

This resource update at the Georgie lake properties of Rock Tech were estimated using validated and verified historical drill hole data, results from the 2010/11 drill and trenching programs conducted by CCIC on behalf of Rock Tech, and results of further drilling and trenching since 2012 until end of 2017 conducted by Arriva Management Inc. Vancouver on behalf of Rock Tech. The resource model and estimate of this chapter are dated 18.04.2018.

DMT constructed 3D models using historical drill hole data as well as results acquired since November 2009 up to date for MZN, MZSW, Harricana, Line 60 and Conway spodumene pegmatites. 3D wireframes (solids) representing the mineralized areas within the spodumene pegmatites were constructed and used to constrain the tonnage and grade estimation. GEMCOM's Geovia software V.7.2 was used to generate the 3D model and perform the grade estimation. Grades for Li_2O were estimated using the anisotropic ordinary kriging method. A specific gravity ("SG") of averaging to 2.71 [t/m³] was applied using a regression equation based on densities measured by Rock Tech.

Five of the pegmatites, for which 3D models have been constructed, are located in the NSPA. From these models measured and indicated resources have been defined. Additional inferred resources have been estimated in the NSPA at depth, where drill core intersection density is too low.

Additional inferred resources within the properties of Rock Tech Inc. have been estimated for the pegmatites in the SSPA. In these exploration claims and dispositions spodumene pegmatites occur at the surface and also drilling and channel sampling has taken place in these areas. Ongoing exploration activities will help to update these resources into higher confidence levels in the future.

The main resources are located in the northern part, where up to date most exploration work has been conducted. In this area for five pegmatites 3D models were created. These models were necessary to define for these pegmatites the measured and indicated resources. The mineral resource estimates of the 3D model for the MZN, MZSW, HAR, LIN, CON spodumene pegmatites presented below.

Mineral resource estimates for MZN, MZSW, HAR, LIN and CON are presented below

Following NI 43-101 requirements and considering potential economic viability a 0.65 % Li_2O cut-off grade has been assumed based on comparable projects, which yields a total Measured and Indicated Resource of 6.58 Mt at a grade of 1.01% Li_2O (see data of green line in Table 37 to Table 39)

Table 36 Measured + Indicated Resource in the NPGA from all 5 pegmatites which were modelled in 3D

Area	Type of Resources	Tonnage [Mt]	Li²O [%]	Cut off Li²O [%]
NSPA	Measured	1.89	1.04	0.65
NSPA	Indicated	4.68	1.00	0.65
TOTAL	**Measured and Indicated**	**6.58**	**1.01**	**0.65**

It should be noted that mineral resources are not mineral reserves. Investigations on modifying factors are recommended to convert measured and indicated mineral resources to mineral reserves of certain degree of economic/technical feasibility.

Table 37. Measured + Indicated Resource (green line) and grade sensitivities comprising all five areas MZN, MZSW, HAR, LIN and CON

Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
0.40	10.20	2.70	0.84
0.50	8.75	2.70	0.90
0.60	7.27	2.71	0.97
0.65	6.58	2.71	1.01
0.70	6.03	2.71	1.04
0.80	4.93	2.72	1.11
0.90	3.96	2.72	1.17

The next tables show the distribution of the individual pegmatites in the main investigation area in the northern part of the whole area (NSPA).

Table 38. Measured + Indicated Resource (green line) and grade sensitivities separated by MZN, MZSW, HAR, LIN and CON

Area	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
MZN	**0.40**	5.37	2.70	0.83
MZN	**0.50**	4.52	2.70	0.90
MZN	**0.60**	3.73	2.71	0.98
MZN	**0.65**	3.37	2.71	1.01
MZN	**0.70**	3.09	2.71	1.04
MZN	**0.70**	3.09	2.71	1.04
MZN	**0.80**	2.54	2.72	1.11

Area	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
MZSW	0.40	1.03	2.70	0.85
MZSW	0.50	0.94	2.70	0.89
MZSW	0.60	0.76	2.71	0.97
MZSW	0.65	0.66	2.71	1.02
MZSW	0.70	0.61	2.71	1.05
MZSW	0.70	0.61	2.71	1.05
MZSW	0.80	0.50	2.72	1.11

Area	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
HAR	0.40	0.82	2.70	0.81
HAR	0.50	0.71	2.70	0.87
HAR	0.60	0.59	2.71	0.94
HAR	0.65	0.54	2.71	0.97
HAR	0.70	0.49	2.71	1.00
HAR	0.70	0.49	2.71	1.00
HAR	0.80	0.39	2.71	1.06

Area	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
LIN	0.40	1.46	2.70	0.85
LIN	0.50	1.26	2.70	0.91
LIN	0.60	1.02	2.71	0.99
LIN	0.65	0.91	2.71	1.04
LIN	0.70	0.83	2.71	1.07
LIN	0.70	0.83	2.71	1.07
LIN	0.80	0.69	2.72	1.14

Area	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
CON	0.40	1.52	2.70	0.86
CON	0.50	1.32	2.71	0.92
CON	0.60	1.17	2.71	0.97
CON	0.65	1.09	2.71	0.99
CON	0.70	1.01	2.71	1.02
CON	0.70	1.01	2.71	1.02
CON	0.80	0.81	2.71	1.09

Table 39. Measured and Indicated Resource (green line) and grade sensitivities comprising all five areas MZN, MZSW, HAR, LIN and CON

Resource Class	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
measured	**0.40**	2.68	2.70	0.89
measured	**0.50**	2.36	2.71	0.95
measured	**0.60**	2.05	2.71	1.01
measured	**0.65**	1.89	2.71	1.04
measured	**0.70**	1.75	2.71	1.07
measured	**0.80**	1.48	2.72	1.13
measured	**0.90**	1.22	2.72	1.19

Resource Class	Cut-off Li_2O [%]	Tonnage [Mt]	Density [t/m³]	Li_2O [%]
indicated	0.40	7.52	2.70	0.82
indicated	0.50	6.39	2.70	0.88
indicated	0.60	5.22	2.71	0.96
indicated	0.65	4.68	2.71	1.00
indicated	0.70	4.28	2.71	1.03
indicated	0.80	3.45	2.72	1.09
indicated	0.90	2.73	2.72	1.16

In addition to these measured and indicated resources inferred resources were estimated from three different areas within the claims of Rock Tech. Within the block model **0.33 Mt** inferred resources are estimated. Outside the block model with in NSPA, there are additional inferred resources of **3.85 Mt** derived by applying an average true thickness of spodumene pegmatite and extrapolating the pegmatite bodies in maximum 50 m below the 3D model see Table 40. The Lithium content was estimated as the same as in the pegmatites up the surface was detected.

Table 40. Additional inferred resource from the extrapolating of the 3 D modelled pegmatite bodies 50 m below the deepest drilled intersection.

Area	Extrapolated surface area [m²]	Average true thickness of drilled spodumen pegmatite with 0.65 % Li_2O cut-off grade	Volume [Mm³]	Tonnage [Mt]
MZN	100 000	6.86	0.69	1.85
HAR	40 000	4.69	0.19	0.51
LIN	50 000	5.91	0.30	0.80
CON	50 000	5.16	0.26	0.70
TOTAL			**1.43**	**3.85**

In the western part of the main pegmatite area between the MZSW and the MZN pegmatite three boreholes drilled at different depths into a gabbroidal sill. Doe to the presence of the sill the extrapolation to further depths of the pegmatite MZSW is not possible. The surface of this basic intrusion dips with 10 degrees to the west. The deepest drillholles located at the pegmatites towards the west do not intersect the sill. During the extrapolation of the pegmatites to the depth we take that issue into account.

Further additional inferred resources have been derived from all drilling and trenching on current claims and dispositions in the SSPA. All drilled material above 0.65 % Li_2O has been added, based on an influence area of 50 x 50 m and the results of chapter 7.3. The density is assumed with 2.7 t/m^3 similar to all the other results. This additional inferred resource from the SSPA is estimated with **2.54 Mt** (Table 41)

Table 41. Additional inferred resource from all drilling and trenching on current claims and dispositions in SSPA; a 0.65 % Li_2O cut-off grade was applied

Name	Area	Drilled interval [m]	Tonnage [Mt]
McVittie	subarea 1 SSPA	94	0.67
Jean Lake (West)	subarea 2 SSPA	188	1.28
Newkirk	subarea 3 SSPA	20	0.13
Aumacho	subarea 4 SSPA	60	0.46
TOTAL		362	**2.54**

The total inferred recourses in the claims and dispositions of Rock Tech are **6.72 Mt.** For all of these estimated inferred resources the cut off of 0.65 % Li_2O was used (Table 42).

Table 42 List of the total inferred resource within the claims and dispositions of Rock Teck from different areas, cut of is 0.65 % Li_2O

AREA	Type of Re-sources	Tonnage [Mt]	Li_2O [%]	Approach of estimation
NSPA	Inferred	0.33	1.01	From the 3D Block models pegmatites in the NSPA
NSPA	Inferred	3.85	1.01	From extrapolating below the existing 3D modelled pegma-tites
SSPA	Inferred	2.54	1.41	From other pegmatites in the SSPA
TOTAL	Inferred	**6.72**	**1.16**	

In all claims and dispositions of Rock Tech situated in the Georgia Lake area **6.58 Mt** measured and indicated resources are estimated. Additionally **6.72 Mt** of inferred resources were estimated within this area.

15 MINERAL RESERVE ESTIMATES

This is an early stage project and no mineral reserve estimate has been done.

16 MINING METHODS

This is an early stage project and no study on mining methods has been done.

17 RECOVERY METHODS

This is an early stage project and no study on recovery methods has been done.

18 PROJECT INFRASTRUCTURE

This is an early stage project and no study on the project infrastructure has been done.

19 MARKET STUDIES

This is an early stage project and no market study has been done.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

In the year 2010 and 2011 a baseline ecological study and a water balance study were made from Trow Associates Inc. Bramton, Ontario for Rock Tech Lithium Inc, Georgia Lake Property, Ontario.

Trow Associates Inc. (“Trow”) was retained by Rock Tech to conduct a Water Balance Study as part of a baseline study for the purpose of the permission process for future advanced exploration on the Georgia Lake Lithium Property. The objective of the water balance study is to quantify existing groundwater and surface water flows and budgets for the Georgia Lake Lithium Property. An internal report was written by Trow to discuss the results of the study and is dated Feb. 18, 2011 (Trow, 2011a). This water balance study was based on available published data and no field work was carried out to collect any site specific information.

The Property is located in the Boreal eco-region of central Ontario. The area is mainly covered by wooded areas with deep rooted plants and trees. The closest Environment Canada weather station is in Geraldton, approximately 80 km northeast of the Property. Weather data from 2001 to 2011 indicates that the annual average precipitation is 769 mm which includes 570 mm of rain and 241 mm of snow. The average annual temperature is 1.2 ºC with a daily maximum of 17.6 ºC in July and -17.6 ºC in January.

Each of Rock Tech's claim blocks contains bodies of water, wetlands and rivers. The soil on the Georgia Lake property is generally comprised of Podzol soils (mostly sandy soils) with rock outcrops, peat and grey wooded soils. Two major aquifer systems are found in the area:

A sandy overburden aquifer system; and
A fractured bedrock aquifer system

Overburden aquifers are found at depths up to 20 m below existing ground surface. It is expected that the overburden sandy aquifer is mostly under water table conditions.

The primary water supply aquifer system in the area is the fractured bedrock aquifer system. Approximately 80% of water bearing fractures are been encountered at depths less than 50 m below ground surface (mbgs). Up to 20% water bearing fractures are encountered within 10 m of the bedrock surface. The fractured aquifer system is expected to be under semi-confined to confined conditions. Bedrock fractures less than 10 m from the bedrock surface are expected to be hydraulic connection with the overburden aquifer system. The overburden aquifer system appears to be connected to the surface water system. The regional bedrock aquifer system does not appear to contribute to the local groundwater flow system.

Of the total precipitation, approximately 55% to 60% is expected to be lost as evapo-transpiration. The evaporation from the surface of the Site's water bodies in the overall area is estimated to be less than 5% of the total precipitation. After evapo-transpiration, approximately 40% to 45% of surplus water is available for surface run-off and infiltration into local groundwater flow system. An infiltration rate between 23% and 37% of the total precipitation is estimated for the area. The high infiltration rates are related to the sand and sandy surficial soil types present in the area. Any significant dewatering related to the development of any of the claim blocks may have an effect on the local water balance of each claim block and water balance of the water bodies and wetlands.

Trow was retained by Rock Tech to conduct a baseline ecological study for the Georgia Lake Lithium Property. A before-after-control-impact (BACI) study is used to predict and manage environmental impacts. Environmental data are collected both before and after mining activities have started to place the mine site activities in the context of baseline conditions. In light of this, attempts were made to collect water samples at pre-selected locations situated upstream, within and downstream of proposed drilling (and potential future extraction) sites. An internal report was written by Trow to discuss the results of the study and is dated March, 2011 (Trow, 2011b).

Rock Tech retained exp Services Inc. to assist with sampling of shaft water from Nama Creek Shaft 11. Collection and analysis of shaft water is a requirement for Ministry of the Environment permitting to dewater mine shafts for further exploration and development. If the shaft water is to be pumped to the surface near a watercourse, then shaft water chemistry data will help determine if the aquatic flora and fauna in the receiver watercourse will be potentially affected by the shaft water, based on the Provincial Water Quality Objectives (PWQO). An internal report was written by exp to discuss the results of the study and is dated Oct. 3, 2011 (exp, 2011b).

21 CAPITAL AND OPERATING COSTS

This is an early stage project and no studies on capital and operating costs have been done.

22 ECONOMIC ANALYSIS

This is an early stage project and no economic studies have been done.

23 ADJACENT PROPERTIES

Exploration activities of adjacent properties have not been considered. North of the area the Beardmore-Geraldton area hosts several deposits types including vein-hosted gold and lithium pegmatites deposits (see CCIC Report 2012).

24 OTHER RELEVANT DATA AND INFORMATION

On July 14, 2011 Rock Tech announced that it entered into a Memorandum of Understanding (“MOU”) with BINGWI NEYAASHI ANISHINAABEK (“BNA”), BIINJITIWAABIK ZAAGING ANISHINAABEK (“BZA”), AND ANIMBIIGOO ZAAGI’IGAN ANISHINAABEK (“AZA”) FIRST NATIONS (collectively referred to as “First Nations”) in regards to the development of the Georgia Lake Lithium project. These First Nations communities are in close geographical proximity to the Georgia Lake Lithium project. While 100% of the project lies within First Nations’ traditional territories, a 2 km stretch of the road accessing the Nama Creek mineral lease is on BNA’s reserve land. Since Rock Tech began exploration in December 2009, several First Nations members have been employed and equipment and material have been procured from the First Nations whenever feasible.

No other relevant information is available.

25 INTERPRETATION AND CONCLUSIONS

Rock Tech's Georgia Lake Lithium Property covers several spodumene pegmatite dykes: Nama Creek MZN, Nama Creek MZSW, Conway, Line 60, Harricana, McVittie, Parole Lake, Foster-Lew, Aumacho, Newkirk and MNW. Abundant fresh spodumene occurs in these pegmatite dykes and spodumene is an ore mineral for lithium. Other rare-elements that are also present with the dykes include: Rb, Cs, Be, Nb and Ta. The majority of the pegmatites are hosted by metasediments, except for McVittie, Foster, Aumacho and MNW pegmatites which are hosted by biotite granite. All of the pegmatites are albite-spodumene type, except for Aumacho – Brink pegmatite which is spodumene-subtype and MNW which is petalite-subtype. Spodumene is the dominant Li-bearing mineral in all of the pegmatites, except for MNW in which SQUI is dominant.

Historical drilling and geological mapping was conducted during the lithium boom in 1955 and 1956. This historical data is readily available in Ontario MNDMF assessment files and in Pye's 1965 report. The Nama Creek MZN, MZSW, Line 60 and Harricana dykes have a total of 99 historic holes (1955-1956); Conway has a total of 26 historic holes (1957-1958); Parole Lake and Foster-Lew has a total of 63 historic holes (1955-1965); and Aumacho has a total of 38 historic holes (1955-1957). In addition, James Bay Midarctic drilled a total of 11 holes on the Georgia Lake area in 2009. While almost all of the collars for the historic holes have disappeared in the field, the collars were georeferenced from historic plan maps and correlated with existing exposed outcrop of pegmatite dykes to correctly position the collars. Some of the historical casings from MZN and Line 60 were located in the field and surveyed to confirm the geo-referencing. The historic drill hole collars, survey, lithology and assays were entered into a database and a 3D model was built. The 3D model was used to propose holes for the 2010-2011 drill program. The Phase 1 and 2 drill program by Rock Tech successfully validated the data from the historic holes and thus the resource estimate could be calculated using both the historic and current drill holes. Phase 3 diabase relogging and mapping program successfully located two diabase dykes on surface and at depth to improve the 3D model for MZN.

One bulk sample was collected from three short drill holes on Nama Creek MZN (BK-11-03, 04, 05) Feb. 16-18, 2011. In addition to the drill core bulk, three blasted grab sample bulks were collected from three sites (two from MZN and one from MZSW). These bulk samples are considered to be representative of the style and type of mineralization and the mineral deposit as a whole. The three grab sample bulks and one drill core bulk (in total about 770 kg) were shipped to SGS Metallurgical Operations, Lakefield, Ontario in February 2011. The four samples were mixed to produce a composite head sample representative of the spodumene mineralization on the property. The composite head sample contained 1.49 %Li_2O and consisted of quartz, albite, spodumene and minor microcline and muscovite.

After the successful generation of an initial Li_2CO_3 product SGS, Lakefield site focused on processing the three concentrates through the standard lithium carbonate hydrometallurgical flow sheet. The program examined three concentrate ("con") samples: one low Fe con, one high Fe con and one heavy media con. The low Fe and high Fe cons were produced

using floatation methods and the heavy media con was produced using heavy liquid separation. After two bicarbonate polishing tests, the resulting solids met all of the product specifications and had a Li_2CO_3 grade of 99.988% and 61 ppm Ca. This indicates that a high grade Li_2CO_3 product can be produced from Rock Tech's mineralized samples.

Trow (2010) conducted a Water Balance Study as part of a baseline study. The water balance study quantified existing groundwater and surface water flows and budgets for the Georgia Lake Lithium Property. The study examined the fractured bedrock aquifer system, surface water system, total precipitation, infiltration and evapo-transition. Trow (2010) conducted a baseline ecological study for the Georgia Lake Lithium Property. Twenty-four sample stations were established and water sampling was conducted at sites associated, to some degree, with all claim blocks. Field work which consisted of collections of surface water was conducted from November 22 to 26, 2010.

Exp Services Inc. ("exp") was retained by Rock Tech to conduct baseline ecological studies. In June 2011, surface water samples were collected for chemistry analysis from locations in proximity to the Nama Creek Claim Block where Rock Tech has concentrated its current exploration efforts. On June 22nd, ten sites were visited and eight samples were collected. Key findings include: the concentrations of parameters such as total aluminum and total iron frequently exceed government guidelines in November 2010 and in June 2011; total copper sometimes exceed government guidelines in 2010 and in 2011; and most other parameters occur at low concentrations or below government guidelines. Parameters such as conductivity, pH, and dissolved oxygen were observed at concentrations typical of surface waters with a range of flow rates, from stagnant to free flowing for the various stations.

Rock Tech retained exp to assist with sampling of shaft water from Nama Creek Shaft 11. Collection and analysis of shaft water is a requirement for Ministry of the Environment permitting to dewater mine shafts for further exploration and development. If the shaft water is to be pumped to the surface near a watercourse, then shaft water chemistry data will help determine if the aquatic flora and fauna in the receiver watercourse will be potentially affected by the shaft water, based on the Provincial Water Quality Objectives (PWQO). Water samples were collected from four levels within the shaft; at surface, 10m, 30m, and 60m depth. The sample at the surface was collected on August 16 and the other levels were sampled on August 11, 2011. Key findings were similar to the surface sampling: aluminum concentrations in the upper 30 m of the shaft water were 1.3 to 3.3 times above the PWQO; copper concentrations of the surface shaft water sample exceeded the PWQO by 1.6 times; all but the surface sample had zirconium concentrations 1.5 to 2 times above the PWQO limit of 4µg/L; and iron concentrations in all shaft water samples exceeded the PWQO level of 300µg/L by 24 to 243 times. Exp suggested that some treatment may be required before release to natural surface water features on the Georgia Lake property.

The only significant risk and uncertainty that may affect the reliability or confidence in the exploration information is georeferencing error for historic drill hole collar locations for properties that currently don't have a 3D model, but as Rock Tech does verification holes and channel sampling on each property these errors are being fixed.

There are no significant risks to the Mineral Resource estimation as the block model tonnages and grades were verified.

The only risk to the projected economic outcome is diabase dykes in close proximity to the spodumene pegmatite dykes especially at MZN. Pye's report (1965) mentions that the spodumene in the spodumene pegmatite dykes is altered to dark green and have low Li_2O contents when it is in close proximity to diabase dykes. The impact of the diabase dykes may limit the ability to obtain Li_2O-rich, Fe-poor spodumene near them, but the recommended drilling will likely indicate that unaltered spodumene occurs surrounding the diabase dykes, just not in close proximity to them.

The Qualified Persons for this Report conclude that the ongoing exploration work in the Georgia Lake deposit from Rock Tech Inc. with drilling and channel sampling upgrade the resources of the deposit. The Qualified Persons are confident that the recommended exploration work like drilling and channel sampling programs will intersect additional spodumene pegmatite to increase the size of the Mineral Resources on spodumene dykes within the Georgia Lake Lithium Property.

The deposit is so far explored that a resource statement is possible. The deposit is open to the depth and lateral open to new findings in the area under the glacier sediments and unexposed in the host rock.

The dolerite and the sill structure of gabbro structure had to be explored. Beneath these sills the similar host rock type explored. That means further pegmatites could be situated in these areas.

The next step for this property should be a preliminary economic assessment study (PEA), to have a first information about the economic position of this project.

26 RECOMMENDATIONS

These recommendation will focus on exploration activities not only to upgrade the resources but also to get more detailed information about e.g. the rock mass and the hydrogeological situation for more detailed further studies.

General recommendations based on previous drill programs:

1. Install permanent bridges for access to Conway and Line 60 instead of using temporary bridges
2. Close monitoring of deviation in the azimuth downhole during drill programs
3. Log the orientation of the rock structures for geotechnical reasons
4. Looking for open holes to use them for hydraulic test and optional re-logging with a wireline service to get the true orientation of the rock mass structure.
5. Get access to the shaft and made a camera logging in the shaft to get information about the support system of the shaft to get information about the rock properties in this area after 60 years open entries. This improve the knowledge for further mine planning studies.
6. Mineralogical studies to confirm identification of phosphate minerals at MZN and all the other pegmatites.

General recommendations based on the resource 3D model:

7. Regular spaced channel sampling to upgrade exploration targets near surface to Inferred status in the southern area.
8. Regular spaced definition drilling and channel sampling to upgrade Inferred resources to Indicated classification; e.g. 35m to 50m spaced centres. Drill spacing needs to target both along strike and down dip positions.
9. Continued QA/QC checks.
10. SG determinations should be routinely made on mineralized intersections.

The purpose of further exploration work like drilling is to validate more historic holes in order to increase the classification of the resource from inferred to indicated status and to carry on further exploration to check the extension of the dykes. Also geophysical measurement in the drill holes should document the true orientation of the rock structure. The purpose of the channel sampling is to get chemical analyses from more pegmatites and to get or upgrade the inferred resources or to get indication for further drilling targets.

DMT proposes that the Phase 3 of exploration program consist of approximately 1,000 m (about 3 holes) of validation and exploration drilling. This phase will also include a total of 30 channels on different properties in the southern area. The purpose of the drilling in the main zone is to validate more historic holes in order to increase the classification of the resource from inferred to indicated status and to carry on further exploration to check the

extension of the main dyke to the east. The purpose of the channel sampling in the southern area is to obtain assays from the surface outcrops and check if these pegmatites show the same mineralogy as the other already investigated pegmatites. These information should give an indication for ore processing options. The purpose of the channels on Foster is to gain information on the location of the pegmatite dykes on surface to aid in drill targeting in the future.

The drill and channel information will be used to update the resource. This will be followed by another 43-101 Independent Technical Report and an Assessment Report. The total cost for this Phase recommendations is $722,119

Table 43 Phase 3 recommended exploration program

Item	Unit	No. of Units	$/Unit	Total Estimate	Subtotal	Project codes
Channel sampling for the southern properties (total 30 channels)						
geologist (log channel samples)	hour	300	$95	$28,500		
prospector/ channel cutter	day	35	$450	$15,750		
excavator	day	30	$150	$4,500		
assays	samples	300	$75	$22,500		
					$71,250	
***Drilling based on 1000 m for the main zone**						
geologist (log drill core)	hour	100	$95	$9,500		
drilling	meters	1000	$250	$250,000		
assays	samples	250	$75	$18,750		
					$278,250	
Travel for geologist						
travel days @ 75%						
flights	flight	3	$1000	$3000		estimate
meals	day	50	$75	$3,750		
accommodation (if needed)				cost + 15%		
truck rental	day	50	$100	$5,000		
					$11,750	
Database management	hour	40	$115	$4,600		
					$4,600	
Project Phase 3 subtotal (channels + drilling)						**$365,850**

Project Phase 3 geophysical investigations						
Drillhole logging (deviation optical measurement, gamma, and full wave sonic)					**$23,200**	
Seismic measurement on surface and in the drillholes for the fault detection.					**$80,500**	
Geophysical measurement with neutron tools					**$31,300**	
Soil sampling and mapping and trenching					**$120,000**	
Project Phase 3 sub total (geophysical investigations, sampling and mapping)						**$255,000**
Project management (logistics) @ 10% time	hour	98	$150	$14,700	**$14,700**	
Resource						
wireframing, block modelling, classification	hour	150	$190	$28,500		
software fee	hour	150	$15	$2,250		
43-101 report writing/validation	hour	50	$190	$9,500		
					$40,250	
43-101 report (4 properties)						
QA/QC of assays	hour	40	$150	$6,000		
report writing	hour	64	$150	$9,600		
GIS drafting of figures	hour	32	$75	$2,400		
senior review	hour	8	$150	$1,200		
					$19,200	
Assessment Filing						
assessment filing - drill report, work report	hours	96	$135	$12,960	**$12,960**	
Project Phase 3 subtotal (43-101 report)						**$87,110**
Subtotal					**$707,960**	
PLI (2%)					$14,159	
Total					**$722,119**	

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APPENDIX 1: DRILL HOLE SUMMARY

In this appendix two tables of drillholes are listed the location of these drillholes in the tables are different. All holes are drilled in several exploration phases since 1955 up to date and are situated within the claims and dispositions belonging to Rock Tech Inc..

In the Table 44 all drilllholes were drilled in the areas were the 3D models of the 5 pegmatites were build (northern main area). The table shows the information about collar location and orientation and availability of logged mineralization, assayed Li_2O and determined density.

The Table 45 shows the collar table of all holes in the more southern areas. These drillholes are situated within the SSPA.

Table 44. Table summarizing collar location and orientation of historical (HIST) and recent (NEW) drill holes (DH) and channels (CH) plus the meters of logged spodumene pegmatite (SPDPEG), pegmatite (PEG), assayed Li_2O and determined density (DENS) from the holes which were used for the 3D model of the pegmatites in the NSPA.

AREA	PROG	TYPE	DHID	EASTING	NORTHING	ELEV	EODH	AZIM	INCL	SPD PEG [m]	PEG [m]	Li_2O [m]	DENS [m]
MZN	HIST	DH	NC-01	424577.24	5477543.63	372.00	58.83	140	-50	5.0	1.2	6.2	
MZN	HIST	DH	NC-02	424673.89	5477621.73	375.00	69.80	140	-50	8.9		8.2	
MZN	HIST	DH	NC-03	424799.34	5477668.59	375.00	119.79	140	-50	4.9	0.2	4.9	
MZN	HIST	DH	NC-04	424903.79	5477727.16	375.00	71.02	140	-50	5.9	0.2	1.8	
MZN	HIST	DH	NC-05	424521.11	5477610.50	371.00	185.32	140	-60	17.5		12.2	
MZN	HIST	DH	NC-06	425002.39	5477800.87	378.00	67.06	140	-50	8.2		6.2	
MZN	HIST	DH	NC-07	425107.34	5477858.47	377.00	60.96	140	-50	4.0			
MZN	HIST	DH	NC-08	425228.39	5477905.81	374.00	152.40	140	-50	9.5	0.9	3.7	
MZN	HIST	DH	NC-09	425306.98	5477810.63	371.00	153.01	140	-50	2.4	0.5	2.1	
MZN	HIST	DH	NC-10	424466.93	5477483.59	368.00	65.53	140	-60				
MZN	HIST	DH	NC-13	424367.35	5477410.86	367.00	184.10	140	-59	2.1			
MZN	HIST	DH	NC-17	424506.95	5477528.99	370.00	96.32	140	-50	12.3	0.7	9.9	
MZN	HIST	DH	NC-20	424483.90	5477551.44	369.00	148.13	140	-60	15.0		12.2	
MZN	HIST	DH	NC-23	424631.91	5477657.85	374.00	159.41	140	-60	17.8	0.6	15.5	
MZN	HIST	DH	NC-25	424439.11	5477596.83	367.00	235.32	140	-60	21.1		15.3	
MZN	HIST	DH	NC-27	424727.10	5477649.55	375.00	123.44	140	-50	10.5		10.4	
MZN	HIST	DH	NC-29	424603.11	5477604.16	373.00	103.34	140	-50	11.9		9.1	
MZN	HIST	DH	NC-30	424587.01	5477621.24	373.00	149.35	140	-59	15.1	1.3	13.6	
MZN	HIST	DH	NC-32	424817.89	5477728.63	374.00	124.98	140	-50	11.9	0.5	8.8	
MZN	HIST	DH	NC-33	424916.00	5477806.24	376.00	158.50	140	-51	11.8	0.9	8.3	
MZN	HIST	DH	NC-34	424548.44	5477661.75	373.00	234.39	140	-58	19.4	0.5	16.1	
MZN	HIST	DH	NC-35	425131.75	5477922.41	375.00	236.84	140	-50	12.0	0.5	8.1	
MZN	HIST	DH	NC-36	425032.17	5477847.24	376.00	169.77	140	-52	10.7	0.5	10.1	
MZN	HIST	DH	NC-37	424889.64	5477836.99	374.00	215.49	140	-60	9.7	4.1	10.4	
MZN	HIST	DH	NC-38	424683.65	5477696.90	376.00	200.25	140	-60	18.1	0.8	16.4	
MZN	HIST	DH	NC-39	424995.07	5477621.73	377.00	94.18	140	-50	2.9		1.8	
MZN	HIST	DH	NC-40	424840.34	5477894.59	373.00	281.33	140	-60	12.5	1.5	11.8	
MZN	HIST	DH	NC-41	424647.53	5477739.85	375.00	241.40	140	-60	14.5	0.3	11.5	
MZN	HIST	DH	NC-43	424937.47	5477877.99	375.00	243.23	140	-60	0.3			
MZN	HIST	DH	NC-44	424724.17	5477752.54	375.00	207.26	140	-60	18.9	0.5	15.2	
MZN	HIST	DH	NC-45	425245.48	5477987.82	374.00	285.29	140	-50	7.3	2.3	4.4	
MZN	HIST	DH	NC-46	424792.99	5477755.96	374.00	203.30	140	-60	15.1	0.5	10.2	
MZN	HIST	DH	NC-47	424990.19	5477899.47	374.00	272.49	140	-60	12.9	1.8	12.4	
MZN	HIST	DH	NC-48	424702.69	5477585.12	375.00	174.65	140	-60	6.8	0.3	6.0	
MZN	HIST	DH	NC-49	424958.95	5477934.61	373.00	321.87	140	-60	12.4	1.1	10.1	
MZN	HIST	DH	NC-50	424765.66	5477793.06	373.00	210.92	140	-60	18.1	1.6	12.6	
MZN	HIST	DH	NC-51	425349.45	5477866.76	373.00	298.09	140	-50	3.6	0.9	2.7	
MZN	HIST	DH	NC-52	424789.09	5477481.64	376.00	199.95	140	-50	7.0	0.7	6.6	
MZN	HIST	DH	NC-53	425044.37	5477927.78	374.00	260.92	140	-60	14.6	1.4	11.2	
MZN	HIST	DH	NC-54	424870.60	5477388.41	374.00	152.40	140	-50	3.0	0.8	1.0	
MZN	HIST	DH	NC-55	425082.93	5477985.87	372.00	335.58	140	-50	20.5	1.4	17.9	
MZN	HIST	DH	NC-56	424842.29	5477800.38	372.00	238.96	140	-60	16.4	2.0	13.0	
MZN	HIST	DH	NC-57	425006.30	5477973.17	372.00	335.28	140	-60	21.6	2.3	16.3	
MZN	HIST	DH	NC-58	425162.99	5477986.84	373.00	277.37	140	-50	13.3	0.5		
MZN	HIST	DH	NC-59	424476.69	5477739.37	368.00	423.38	140	-61	17.1	1.3	15.7	
MZN	NEW	CH	NC-CH-11-01	424474.55	5477449.23	370.55	8.50	170	0	5.5		7.5	1.0
MZN	NEW	CH	NC-CH-11-02	424521.79	5477478.31	377.72	9.50	170	0	5.7		6.7	1.0
MZN	NEW	CH	NC-CH-11-03	424555.04	5477503.47	373.06	8.10	130	0	5.7		6.7	1.0
MZN	NEW	CH	NC-CH-11-04	424577.46	5477521.13	373.65	10.90	160	0	6.3		7.1	1.0
MZN	NEW	CH	NC-CH-11-05	424629.32	5477551.75	376.10	7.40	140	0	6.2		6.2	1.0
MZN	NEW	CH	NC-CH-11-06	424668.35	5477577.80	375.89	6.65	140	0	5.7		5.7	1.0

AREA	PROG	TYPE	DHID	EASTING	NORTHING	ELEV	EODH	AZIM	INCL	SPD PEG [m]	PEG [m]	Li_2O [m]	DENS [m]
MZN	NEW	CH	NC-CH-11-07	424757.78	5477575.59	375.62	6.80	120	0	3.3	0.3	6.8	1.0
MZN	NEW	CH	NC-CH-11-08	424766.15	5477616.77	377.03	4.30	170	0	1.9	0.2	4.3	1.1
MZN	NEW	CH	NC-CH-11-09	424794.31	5477628.62	376.15	16.50	160	0	5.3		10.3	1.0
MZN	NEW	CH	NC-CH-11-09A	424809.46	5477626.14	373.28	6.70	360	0	3.3		6.7	1.0
MZN	NEW	CH	NC-CH-11-10	424823.97	5477644.78	373.30	6.55	150	0	4.4		6.6	1.0
MZN	NEW	CH	NC-CH-11-10A	424825.00	5477642.00	377.00	7.20	160	0	5.2		7.2	1.0
MZN	NEW	CH	NC-CH-11-11	424837.79	5477657.05	374.95	7.49	160	0	6.7		7.5	1.0
MZN	NEW	CH	NC-CH-11-12	424857.81	5477681.08	375.02	3.70	150	0		2.0	3.7	
MZN	NEW	CH	NC-CH-11-14	424932.97	5477722.67	375.28	21.90	142	0	2.6	2.6	10.8	1.0
MZN	NEW	CH	NC-CH-11-16	425008.05	5477741.26	379.61	4.58	141	0	1.5		4.6	1.5
MZN	NEW	CH	NC-CH-11-17	425042.00	5477773.00	386.00	2.50	141	0	2.5		2.5	2.0
MZN	NEW	CH	NC-CH-11-18	425081.94	5477798.21	374.89	4.91	154	0	2.9		4.9	1.9
MZN	NEW	DH	BK-11-03	424823.00	5477643.00	374.00	46.75	10	-80	38.8	0.4	44.1	2.1
MZN	NEW	DH	BK-11-04	424823.00	5477643.00	374.00	32.00	10	-70	8.1		10.0	
MZN	NEW	DH	BK-11-05	424823.00	5477643.00	374.00	40.80	20	-80	36.2	0.4	40.6	
MZN	NEW	DH	NC-11-01	424493.50	5477576.50	369.00	177.10	140	-60	14.3	9.4	47.6	3.3
MZN	NEW	DH	NC-11-02	424582.90	5477625.20	373.00	185.00	140	-60	19.2	1.2	33.2	4.9
MZN	NEW	DH	NC-11-03	424507.70	5477704.30	370.00	326.00	140	-60	16.2	2.9	35.8	4.1
MZN	NEW	DH	NC-11-04	424641.90	5477661.60	374.00	164.40	140	-60	25.6	2.6	45.1	5.8
MZN	NEW	DH	NC-11-05	424698.00	5477670.20	375.00	164.00	140	-55	10.6	1.8	21.3	2.5
MZN	NEW	DH	NC-11-06	424739.00	5477728.70	376.00	185.00	140	-60	13.1	1.8	40.7	
MZN	NEW	DH	NC-11-07	424865.20	5477803.30	373.00	126.00	140	-60	11.9		20.3	
MZN	NEW	DH	NC-11-08	424887.80	5477863.80	374.00	220.90	140	-60	4.0	0.5	12.7	3.2
MZN	NEW	DH	NC-11-09	424894.20	5477738.50	375.00	98.20	140	-50	12.6	1.9	32.4	2.8
MZN	NEW	DH	NC-11-10	424952.50	5477859.10	375.00	196.90	140	-60	10.5		20.6	3.0
MZN	NEW	DH	NC-11-11	425015.80	5477874.30	375.00	191.30	140	-60	10.5	0.4	19.2	2.0
MZN	NEW	DH	NC-11-12	425069.30	5477890.40	376.00	170.10	140	-60	7.8	0.5	17.3	2.0
MZN	NEW	DH	NC-11-13	424731.80	5477645.00	375.00	119.00	140	-50	9.1	0.4	21.2	2.0
MZN	NEW	DH	NC-11-14	424418.60	5477611.30	367.00	245.00	128	-60	19.1		26.4	3.8
MZN	NEW	DH	NC-11-15	424589.31	5477716.63	375.60	260.00	145	-60	21.8	1.4	43.9	5.9
MZN	NEW	DH	NC-11-16	424556.56	5477750.63	374.62	359.00	140	-60	18.9	5.2	35.1	8.0
MZN	NEW	DH	NC-11-17	424415.48	5477616.90	370.45	440.00	140	-79	10.6	2.2	16.4	3.9
MZN	NEW	DH	NC-11-18	424430.44	5477493.55	367.53	119.00	140	-60	9.9	0.6	16.1	2.0
MZN	NEW	DH	NC-11-19	424399.76	5477537.48	369.39	173.00	140	-60	8.4	1.1	10.5	1.0
MZN	NEW	DH	NC-11-20	424369.15	5477595.50	368.65	260.00	140	-60	6.6	5.0	10.9	1.0
MZN	NEW	DH	NC-11-21	424392.38	5477447.84	366.76	101.00	140	-52				
MZN	NEW	DH	NC-11-22	424485.33	5477657.68	368.67	314.06	140	-70	24.8	2.0	42.6	4.0
MZN	NEW	DH	NC-11-23	424419.16	5477611.79	371.36	278.11	140	-70	22.4	0.9	35.6	4.3
MZN	NEW	DH	NC-11-24	424489.47	5477662.34	368.02	266.00	140	-60	16.8	4.8	27.8	5.0
MZN	NEW	DH	NC-11-25	424594.07	5477805.97	372.32	329.00	140	-60	19.0	3.9	34.9	10.1
MZN	NEW	DH	NC-11-26	424684.40	5477802.16	371.91	251.00	140	-62	12.2	2.9	23.1	5.7
MZN	NEW	DH	NC-11-27	424706.34	5477852.24	373.49	359.00	140	-60	12.3	8.1	20.2	4.2
MZN	NEW	DH	NC-11-28	424768.77	5477890.49	373.06	323.00	140	-62	12.2	1.8	23.0	
MZN	NEW	DH	NC-11-29	424808.72	5477824.91	372.42	254.00	140	-60	12.6	5.2	19.2	3.0
MZSW	HIST	DH	NC-11	424351.73	5477052.58	367.00	91.44	140	-50	6.1		3.7	
MZSW	HIST	DH	NC-14	424299.02	5477016.95	369.00	103.34	140	-50	13.9		6.1	
MZSW	HIST	DH	NC-15	424325.37	5477083.33	368.00	154.08	140	-59	12.6	0.5	8.8	
MZSW	HIST	DH	NC-16	424253.13	5476981.32	369.00	106.07	140	-50	14.1		10.9	
MZSW	HIST	DH	NC-18	424283.40	5477036.96	369.00	156.36	140	-59	9.6	2.6	9.1	
MZSW	HIST	DH	NC-19	424207.25	5476937.87	368.00	95.11	140	-50	7.8		7.8	
MZSW	HIST	DH	NC-21	424202.37	5476851.48	367.00	153.16	140	-50	2.4	1.9		
MZSW	HIST	DH	NC-22	424248.25	5477077.96	371.00	238.35	140	-59	21.0	0.1	16.3	
MZSW	HIST	DH	NC-24	424383.95	5477108.72	366.00	121.92	140	-50		0.5		
MZSW	HIST	DH	NC-26	424230.19	5477007.67	370.00	155.46	140	-59	15.6		10.7	
MZSW	HIST	DH	NC-28	424182.35	5476965.70	368.00	153.92	140	-58	18.3	0.7	14.3	
MZSW	HIST	DH	NC-31	424145.75	5477003.77	368.00	236.52	140	-59	16.9		16.9	
MZSW	HIST	DH	NC-60	424218.93	5477200.00	374.00	450.20	140	-60	12.0	3.7	9.7	
MZSW	NEW	CH	NamaCreekSW-11-CH-1A-16Ex	424218.00	5476891.00	383.00	1.45	277	0	1.5		1.5	
MZSW	NEW	CH	NamaCreekSW-11-CH-1B-16Ex	424218.20	5476893.50	383.00	1.03	277	0	1.0		1.0	
MZSW	NEW	CH	NamaCreekSW-11-CH-2-16Ex	424256.42	5476934.00	372.92	2.65	332	0		0.2	2.7	
MZSW	NEW	CH	NamaCreekSW-11-CH-2A-16Ex	424256.42	5476923.24	372.92	3.89	292	0	2.8		3.9	1.1
MZSW	NEW	CH	NamaCreekSW-11-CH-2B-16Ex	424253.29	5476920.40	372.92	0.97	329	0			1.0	
MZSW	NEW	CH	NamaCreekSW-11-CH-4-16Ex	424295.12	5476949.86	371.00	1.13	328	0			1.1	
MZSW	NEW	CH	NamaCreekSW-11-CH-4S-16Ex	424295.12	5476947.66	371.00	1.13	329	0			1.1	
MZSW	NEW	CH	NamaCreekSW-11-CH-5-16Ex	424312.50	5476968.85	368.77	1.00	326	0			1.0	
MZSW	NEW	CH	NamaCreekSW-11-CH-5N-16Ex	424307.80	5476971.93	369.00	1.01	328	0			1.0	
MZSW	NEW	CH	NamaCreekSW-11-CH-6-16Ex	424305.14	5476979.80	369.86	1.15	324	0			1.2	
MZSW	NEW	CH	NamaCreekSW-11-CH-6A-16Ex	424310.95	5476979.65	378.00	0.61	302	0	0.6		0.6	
MZSW	NEW	CH	NamaCreekSW-11-CH-6AN-16Ex	424309.00	5476981.00	378.00	2.91	310	0	2.1		2.9	0.8
MZSW	NEW	CH	NamaCreekSW-11-CH-7-16Ex	424327.11	5476999.86	371.80	1.06	312	0			1.1	
MZSW	NEW	CH	NamaCreekSW-16-CH-1	424192.56	5476876.24	367.52	3.10	331	0	1.1		3.1	
MZSW	NEW	CH	NamaCreekSW-16-CH-1N	424193.99	5476880.81	368.12	2.92	334	0	0.8		2.9	
MZSW	NEW	CH	NamaCreekSW-16-CH-2	424192.99	5476876.81	368.12	2.02	325	0	1.0		2.0	1.0
MZSW	NEW	CH	NamaCreekSW-16-CH-2S	424193.53	5476874.81	368.12	1.00	322	0			1.0	
MZSW	NEW	CH	NamaCreekSW-16-CH-3	424336.93	5476998.62	369.37	2.97	306	0	3.0		3.0	
MZSW	NEW	CH	NamaCreekSW-16-CH-3N	424333.75	5477001.23	369.98	1.00	330	0			1.0	
MZSW	NEW	CH	NS-CH-11-01	424216.08	5476900.25	372.00	5.96	124	0	5.5		6.0	1.4
MZSW	NEW	CH	NS-CH-11-01A	424218.00	5476891.00	383.00	2.74	80	0	1.8		2.7	0.9
MZSW	NEW	CH	NS-CH-11-02	424256.42	5476934.55	372.92	9.83	118	0	5.3		7.1	2.2
MZSW	NEW	CH	NS-CH-11-03	424282.56	5476961.73	371.13	3.92	128	0	2.9		3.9	1.1
MZSW	NEW	CH	NS-CH-11-04	424294.79	5476952.97	371.77	4.09	156	0	2.7		3.6	1.8
MZSW	NEW	CH	NS-CH-11-05	424307.80	5476971.93	368.77	7.00	132	0	3.1	0.3	6.3	2.1
MZSW	NEW	CH	NS-CH-11-06	424303.44	5476981.25	369.86	2.50	146	0	1.9		2.5	0.9
MZSW	NEW	CH	NS-CH-11-06A	424309.00	5476981.00	378.00	2.90	140	0	2.9		2.9	1.0
MZSW	NEW	CH	NS-CH-11-07	424323.36	5477003.86	371.82	4.90	124	0	2.8		4.9	1.8

AREA	PROG	TYPE	DHID	EASTING	NORTHING	ELEV	EODH	AZIM	INCL	SPD PEG [m]	PEG [m]	Li_2O [m]	DENS [m]
MZSW	NEW	DH	NS-11-01	424177.50	5477050.10	369.00	221.00	140	-60	16.7	0.4	27.3	3.1
MZSW	NEW	DH	NS-11-02	424236.30	5476998.30	370.00	119.00	140	-50	10.9	1.2	19.4	4.0
MZSW	NEW	DH	NS-11-03	424260.90	5477056.20	370.00	143.11	140	-50	11.1		22.6	3.0
MZSW	NEW	DH	NS-11-04	424272.30	5477148.90	372.00	244.85	140	-60	17.1	0.4	34.0	4.0
HAR	HIST	DH	HM-01	425612.54	5476907.12	409.00	97.55	140	-50	11.3			
HAR	HIST	DH	HM-02	425553.48	5476964.72	401.00	183.49	140	-50	6.0	9.9	14.5	
HAR	HIST	DH	HM-05	425717.00	5477125.31	400.00	103.63	140	-50	16.4		15.5	
HAR	HIST	DH	HM-10	425670.14	5477006.21	407.00	110.28	140	-50	18.5		18.5	
HAR	HIST	DH	HM-11	425625.72	5476967.65	406.00	128.03	140	-50	18.4	5.8	18.4	
HAR	HIST	DH	HM-12	425613.03	5477062.34	399.00	187.45	140	-50	23.2		21.3	
HAR	HIST	DH	HM-13	425562.76	5476836.34	409.00	109.73	140	-50	14.6		14.6	
HAR	HIST	DH	HM-14	425697.96	5477143.86	398.00	121.92	140	-55	24.1		23.6	
HAR	HIST	DH	HM-15	425545.67	5476785.58	409.00	80.47	140	-50	6.6		6.4	
HAR	HIST	DH	HM-16	425588.63	5477002.79	401.00	146.91	140	-50	13.0		13.0	
HAR	HIST	DH	HM-18	425516.87	5476735.79	406.00	121.92	140	-50				
HAR	HIST	DH	NC-HM-03	425491.00	5476992.54	393.00	274.32	140	-52	12.9		12.2	
HAR	HIST	DH	NC-HM-04	425749.22	5477175.59	396.00	102.72	140	-50	2.9		2.9	
HAR	HIST	DH	NC-HM-06	425661.36	5477261.01	382.00	138.07	140	-50	15.9	1.5	15.9	
HAR	HIST	DH	NC-HM-07	425598.88	5477158.99	387.00	224.04	140	-60	12.7		12.8	
HAR	HIST	DH	NC-HM-08	425614.01	5477305.43	377.00	121.63	140	-50	8.1		8.1	
HAR	HIST	DH	NC-HM-09	425589.60	5477332.27	375.00	111.86	45	-50	4.3	0.3	3.4	
HAR	HIST	DH	NC-HM-17	425523.71	5476872.95	404.00	182.88	140	-50	10.9		10.7	
HAR	NEW	CH	Harricana-16-CH-1	425892.38	5477137.12	398.43	6.70	47	0	4.0	0.7	6.7	
HAR	NEW	CH	Harricana-16-CH-2	425852.81	5477114.60	401.35	15.85	102	0	13.1	0.6	15.9	1.0
HAR	NEW	CH	Harricana-16-CH-3A	425804.44	5477069.25	411.78	4.45	125	0	2.1		4.5	1.0
HAR	NEW	CH	Harricana-16-CH-3B	425805.97	5477067.86	411.75	5.27	123	0	3.0		5.3	
HAR	NEW	CH	Harricana-16-CH-3C	425808.00	5477066.04	411.07	7.11	127	0	4.0	1.1	7.1	1.0
HAR	NEW	CH	Harricana-16-CH-4A	425777.64	5477041.61	414.57	0.94	136	0			0.9	
HAR	NEW	CH	Harricana-16-CH-4B	425778.24	5477040.93	414.97	8.26	135	0	6.4	0.7	8.3	1.0
HAR	NEW	CH	Harricana-16-CH-5	425738.34	5477016.59	416.05	5.90	130	0	3.2	0.9	5.9	1.0
HAR	NEW	CH	Harricana-16-CH-6	425725.80	5476958.73	417.69	6.69	143	0	3.1		6.7	1.0
HAR	NEW	CH	Harricana-16-CH-7	425631.61	5476733.41	417.44	3.10	135	0	1.0		3.1	
HAR	NEW	CH	HW-CH-11-01	425688.73	5477235.97	386.57	0.60	56	0	0.6		0.6	0.6
HAR	NEW	CH	HW-CH-11-02	425739.70	5477159.61	399.63	0.62	87	0	0.6		0.6	0.6
HAR	NEW	CH	HW-CH-11-03	425753.61	5477069.89	412.74	0.48	60	0	0.5		0.5	0.5
HAR	NEW	CH	HW-CH-11-04	425739.06	5477015.91	417.17	0.59	103	0	0.6		0.6	0.6
HAR	NEW	CH	HW-CH-11-05	425681.80	5476959.80	412.54	0.60	131	0	0.6		0.6	0.6
HAR	NEW	CH	HW-CH-11-06	425658.50	5476822.17	418.17	0.50	120	0		0.5	0.5	0.5
HAR	NEW	CH	HW-CH-11-07	425738.01	5476932.24	419.77	0.40	110	0	0.4		0.4	0.4
HAR	NEW	CH	HW-CH-11-08	425713.27	5476916.42	419.41	0.50	24	0	0.5		0.5	
HAR	NEW	CH	HW-CH-11-09	425735.36	5476972.98	420.29	0.56	126	0	0.6		0.6	0.6
HAR	NEW	CH	HW-CH-11-10	425766.38	5477023.38	417.11	0.35	130	0	0.4		0.4	0.4
HAR	NEW	CH	HW-CH-11-11	425806.74	5477072.56	412.12	0.70	160	0	0.7		0.7	0.7
HAR	NEW	CH	West-16-CH-1	425679.09	5477409.06	371.09	6.15	95	0	4.2		6.2	1.1
HAR	NEW	CH	West-16-CH-1W	425671.32	5477386.98	373.99	3.43	133	0	1.6		3.4	
HAR	NEW	CH	West-16-CH-2	425651.38	5477319.36	378.05	9.97	67	0	6.0		10.0	1.0
HAR	NEW	CH	West-16-CH-3A	425661.25	5477277.68	380.72	6.69	59	0	5.7		6.7	1.2
HAR	NEW	CH	West-16-CH-3B	425665.84	5477281.83	381.74	0.96	36	0			1.0	
HAR	NEW	CH	West-16-CH-3W	425650.42	5477264.68	381.24	2.95	36	0	1.1		3.0	
HAR	NEW	CH	West-16-CH-4	425686.74	5477235.60	387.03	9.59	53	0	7.5		9.6	1.0
HAR	NEW	CH	West-16-CH-5	425722.92	5477183.31	396.77	12.29	55	0	10.3		12.3	1.0
HAR	NEW	CH	West-16-CH-7	425592.76	5476889.28	410.25	6.05	164	0	3.6		6.1	
HAR	NEW	CH	West-16-CH-8	425544.15	5476856.60	405.02	5.77	138	0	3.7		5.8	1.7
HAR	NEW	DH	HW-11-01	425662.50	5477134.20	396.00	122.00	94	-55	9.5		13.5	3.1
HAR	NEW	DH	HW-11-02	425680.90	5477071.80	403.00	173.40	104	-55	13.6	1.4	26.8	3.0
HAR	NEW	DH	HW-11-03	425598.00	5476906.10	409.00	151.90	135	-50	12.3	0.4	18.7	2.8
HAR	NEW	DH	HW-11-04	425640.70	5477114.50	396.00	224.00	103	-60	8.7	5.0	23.7	2.0
HAR	NEW	DH	HW-11-05	425647.60	5477206.10	386.00	140.00	80	-60	9.4		16.3	3.0
HAR	NEW	DH	HW-11-06	425616.00	5477244.40	379.00	155.00	80	-50	8.0	7.2	15.4	3.1
LIN	HIST	DH	HM-19	426219.54	5477096.33	405.05	142.95	110	-50	25.2		25.2	
LIN	HIST	DH	HM-23	426203.58	5477021.18	409.49	95.10	110	-50	24.7		24.7	
LIN	HIST	DH	HM-26	426169.41	5476962.45	409.42	110.64	110	-50	9.6	6.1	15.7	
LIN	HIST	DH	HM-27	426166.28	5476902.12	413.58	109.42	110	-50	18.5		18.5	
LIN	HIST	DH	HM-29	426113.80	5476858.98	415.78	149.66	110	-50	17.7		17.7	
LIN	HIST	DH	HM-30	426157.32	5477058.55	403.54	137.17	110	-50	25.6		24.7	
LIN	HIST	DH	HM-32	426118.32	5477018.84	400.38	172.52	110	-50	7.0	0.3	7.0	
LIN	HIST	DH	HM-33	426113.33	5476952.64	409.51	207.26	110	-53	13.4	8.8	19.8	
LIN	HIST	DH	HM-34	426107.89	5476911.52	409.79	217.93	110	-51	8.2	9.3	8.2	
LIN	HIST	DH	NC-HM-20	426230.35	5477190.29	394.26	164.59	110	-50	26.9		26.9	
LIN	HIST	DH	NC-HM-21	426246.05	5477247.90	393.76	181.98	110	-50	23.0		20.1	
LIN	HIST	DH	NC-HM-22	426271.82	5477304.88	387.12	154.53	110	-50	8.9		8.9	
LIN	HIST	DH	NC-HM-24	426310.95	5477353.41	377.38	140.82	110	-50	16.2		16.1	
LIN	HIST	DH	NC-HM-25	426133.44	5477190.38	389.79	248.41	110	-50		16.2		
LIN	HIST	DH	NC-HM-28	426176.58	5477142.35	397.98	152.41	110	-50	27.9		27.9	
LIN	HIST	DH	NC-HM-31	426297.60	5477431.32	370.00	65.23	140	-50	7.6	0.6	7.6	
LIN	HIST	DH	NC-HM-35	426338.57	5477465.36	370.00	75.90	140	-50	4.0	0.6	4.0	
LIN	HIST	DH	NC-HM-36	426254.73	5477394.42	372.12	150.89	110	-50	4.6	3.7	5.8	
LIN	HIST	DH	NC-HM-37	426190.36	5477227.00	388.88	156.68	110	-50	37.5	2.9	21.6	
LIN	HIST	DH	NC-HM-38	426209.56	5477279.77	385.88	149.35	110	-50	19.8	1.2	19.8	
LIN	HIST	DH	NC-HM-39	426226.24	5477347.48	378.94	149.96	110	-50	14.3	0.3	14.0	
LIN	HIST	DH	NC-HM-40	426142.52	5477269.59	381.61	171.91	110	-50				
LIN	NEW	CH	L60-11-C01	426330.99	5477326.95	383.62	14.66	105	0	13.7		14.7	1.0
LIN	NEW	CH	L60-11-C02	426332.92	5477273.36	391.13	7.91	110	0	6.8		7.9	1.0
LIN	NEW	CH	L60-11-C03	426300.59	5477240.50	394.65	8.35	114	0	6.8		8.4	1.0
LIN	NEW	CH	L60-11-C04	426295.51	5477203.13	398.38	15.86	110	0	13.9		15.9	1.0

AREA	PROG	TYPE	DHID	EASTING	NORTHING	ELEV	EODH	AZIM	INCL	SPD PEG [m]	PEG [m]	Li_2O [m]	DENS [m]
LIN	NEW	CH	L60-11-C04B	426295.51	5477203.13	398.38	3.01	133	0	1.2		3.0	1.2
LIN	NEW	CH	L60-11-C05A	426259.11	5477178.68	397.61	7.09	98	0	5.1		7.1	1.0
LIN	NEW	CH	L60-11-C05B	426259.11	5477178.68	397.61	11.22	103	0	10.7		11.2	1.1
LIN	NEW	CH	L60-11-C05C	426259.11	5477178.68	397.61	3.92	96	0	1.8		3.9	1.0
LIN	NEW	CH	L60-11-C05D	426259.11	5477178.68	397.61	3.26	95	0	1.4		3.3	0.5
LIN	NEW	CH	L60-11-C06	426257.02	5477134.84	402.78	24.01	105	0	21.8		24.0	1.1
LIN	NEW	CH	L60-11-C07A	426254.24	5477093.33	407.52	22.14	110	0	19.1		22.1	2.0
LIN	NEW	CH	L60-11-C07B	426254.24	5477093.33	407.52	2.49	80	0	0.8		2.5	0.8
LIN	NEW	CH	L60-11-C07C	426254.24	5477093.33	407.52	3.11	72	0	1.2		3.1	1.2
LIN	NEW	CH	L60-11-C08	426249.00	5477059.91	411.18	23.18	105	0	21.3		23.2	1.9
LIN	NEW	CH	L60-11-C09	426247.08	5477012.83	413.57	24.91	109	0	21.0	2.1	24.9	2.0
LIN	NEW	CH	L60-11-C10	426236.32	5476959.09	415.55	22.74	100	0	17.0	4.0	22.7	2.1
LIN	NEW	CH	L60-16-CH-11	426223.24	5476918.23	414.49	25.12	97	0	24.1		25.1	2.1
LIN	NEW	CH	L60-16-CH-11East	426276.88	5476905.50	418.49	3.55	173	0		1.6	3.6	
LIN	NEW	CH	L60-16-CH-12A	426207.76	5476883.55	413.42	7.90	114	0	3.1	2.0	7.9	
LIN	NEW	CH	L60-16-CH-12B	426213.98	5476879.48	416.58	2.06	120	0	2.1		2.1	
LIN	NEW	CH	L60-16-CH-12C	426215.68	5476878.57	416.51	10.50	118	0	10.5		10.5	0.7
LIN	NEW	CH	L60-16-CH-12D	426223.22	5476872.60	417.72	3.06	111	0	3.1		3.1	
LIN	NEW	CH	L60-16-CH-12E	426227.65	5476874.84	419.76	2.04	101	0	2.0		2.0	1.0
LIN	NEW	CH	L60-16-CH-12F	426229.42	5476875.37	419.08	1.03	101	0			1.0	
LIN	NEW	CH	L60-16-CH-13A	426230.31	5476837.64	422.65	15.44	108	0	13.4		15.4	2.4
LIN	NEW	CH	L60-16-CH-13B	426241.04	5476835.13	422.10	4.55	102	0	4.6		4.6	
LIN	NEW	CH	L60-16-CH-13C	426244.00	5476831.00	418.30	0.87	82	0			0.9	
LIN	NEW	CH	L60-16-CH-13West	426215.61	5476837.31	418.28	6.07	106	0	3.0	1.0	6.1	1.0
LIN	NEW	CH	L60-16-CH-14NA	426369.48	5477367.39	374.10	0.92	159	0			0.9	0.9
LIN	NEW	CH	L60-16-CH-14NB	426371.72	5477367.41	374.33	4.49	160	0	4.5		4.5	
LIN	NEW	CH	L60-16-CH-14NC	426374.35	5477363.95	375.10	0.98	157	0			1.0	
LIN	NEW	CH	L60-16-CH-15NA	426421.39	5477380.54	371.73	3.62	135	0	2.8	0.9	3.6	
LIN	NEW	CH	L60-16-CH-15NB	426423.21	5477378.64	372.30	0.87	145	0			0.9	
LIN	NEW	CH	L60-16-CH-15NW	426414.86	5477384.08	370.08	1.60	157	0		0.6	1.6	
LIN	NEW	CH	L6-CH-11-01	426312.51	5477291.39	390.15	0.50	116	0	0.5		0.5	0.5
LIN	NEW	CH	L6-CH-11-02	426305.49	5477241.38	396.61	0.55	40	0	0.6		0.6	0.6
LIN	NEW	CH	L6-CH-11-03	426288.85	5477169.74	403.97	0.40	98	0	0.4		0.4	0.4
LIN	NEW	CH	L6-CH-11-04	426265.02	5477088.73	410.83	0.52	99	0	0.5		0.5	0.5
LIN	NEW	CH	L6-CH-11-05	426253.09	5477014.24	417.69	0.50	80	0	0.5		0.5	0.5
LIN	NEW	DH	L60-11-01	426237.30	5477249.50	391.00	125.00	100	-55	14.4	2.1	28.0	3.0
LIN	NEW	DH	L60-11-02	426175.50	5477149.10	395.00	160.70	102	-55	21.1		23.1	2.0
LIN	NEW	DH	L60-11-03	426215.65	5477289.68	384.24	203.00	105	-60	14.8	21.5	42.6	4.0
LIN	NEW	DH	L60-11-04	426189.06	5477225.72	389.49	203.00	104	-46	18.1	7.0	29.3	3.0
LIN	NEW	DH	L60-11-05	426321.04	5477383.74	373.28	116.00	110	-60	12.0	10.5	18.0	2.7
CON	HIST	DH	CON-01	427451.65	5478400.52	372.67	57.97	120	-45	13.4		13.4	
CON	HIST	DH	CON-02	427467.15	5478489.39	370.00	84.43	120	-45	8.1	0.1	8.1	
CON	HIST	DH	CON-03	427499.46	5478513.81	370.00	64.16	120	-50	6.2	0.5	6.6	
CON	HIST	DH	CON-04	427522.35	5478541.53	370.02	53.95	120	-50	4.9	0.0	4.8	
CON	HIST	DH	CON-05	427544.56	5478573.29	370.41	90.22	120	-45	4.0	0.5	4.3	
CON	HIST	DH	CON-06	427421.36	5478363.33	378.69	48.16	120	-45	12.7		12.7	
CON	HIST	DH	CON-07	427407.88	5478296.00	388.92	39.93	120	-45	10.7	0.2	10.7	
CON	HIST	DH	CON-08	427385.00	5478275.00	391.00	46.94	120	-45	10.7	0.3	10.7	
CON	HIST	DH	CON-09	427363.46	5478234.51	398.95	45.72	120	-45	12.3	0.0	12.3	
CON	HIST	DH	CON-10	427325.76	5478201.31	400.00	46.33	120	-45	9.0	0.4	9.0	
CON	HIST	DH	CON-11	427307.58	5478156.13	399.77	54.25	120	-45	6.5	2.1	8.0	
CON	HIST	DH	CON-12	427241.63	5478124.75	389.56	75.59	120	-50	11.0	1.9	10.8	
CON	HIST	DH	CON-13	427216.05	5478089.61	386.09	77.72	120	-50	10.5	2.3	11.0	
CON	HIST	DH	CON-14	427180.38	5478071.22	382.29	96.01	120	-45	13.1	2.3	14.6	
CON	HIST	DH	CON-15	427191.80	5478002.04	383.16	63.40	120	-50	12.6		12.6	
CON	NEW	CH	CW-11-C01	427476.60	5478355.23	377.88	11.11	310	0	8.7		11.1	1.3
CON	NEW	CH	CW-11-C02	427445.32	5478344.49	382.11	3.77	246	0	2.7		3.8	1.0
CON	NEW	CH	CW-11-C03A	427446.62	5478325.71	382.59	7.18	122	0	7.2		7.2	1.1
CON	NEW	CH	CW-11-C03B	427446.62	5478325.71	382.59	6.15	122	0	4.9		5.3	0.9
CON	NEW	CH	CW-16-CH-4A	427431.11	5478296.28	391.59	0.91	107	0			0.9	0.9
CON	NEW	CH	CW-16-CH-4B	427432.93	5478294.83	392.23	5.78	113	0	5.0	0.8	5.8	
CON	NEW	CH	CW-16-CH-4C	427438.51	5478292.68	393.76	4.01	114	0	4.0		4.0	
CON	NEW	CH	CW-16-CH-4D	427439.14	5478287.14	395.19	3.16	116	0	3.2		3.2	
CON	NEW	CH	CW-16-CH-4E	427442.75	5478285.01	394.66	1.20	116	0			1.2	
CON	NEW	CH	CW-16-CH-5	427414.11	5478240.26	396.85	5.39	130	0	4.4		5.4	1.5
CON	NEW	CH	CW-16-CH-5E	427417.50	5478236.52	397.93	0.96	122	0			1.0	
CON	NEW	CH	CW-16-CH-5W	427407.38	5478249.00	395.26	7.03	144	0	6.0		7.0	1.0
CON	NEW	CH	CW-16-CH-6	427268.06	5477877.99	389.99	4.50	89	0	2.5		4.5	
CON	NEW	CH	CW-16-CH-8N	427483.19	5478380.43	371.99	6.28	140	0	5.3		6.3	1.0
CON	NEW	CH	CW-16-CH-8NW	427487.86	5478369.61	371.73	1.01	128	0			1.0	
CON	NEW	CH	CW-CH-11-01	427397.79	5478214.56	401.26	0.40	124	0	0.4		0.4	0.4
CON	NEW	CH	CW-CH-11-02	427430.65	5478282.82	396.97	0.45	70	0	0.5		0.5	0.5
CON	NEW	CH	CW-CH-11-03	427453.11	5478318.41	387.83	0.50	6	0	0.5		0.5	0.5
CON	NEW	CH	CW-CH-11-04	427457.62	5478345.92	380.86	0.55	80	0	0.6		0.6	0.6
CON	NEW	CH	CW-CH-11-05	427467.71	5478348.70	382.66	0.67	72	0	0.7		0.7	0.7
CON	NEW	CH	CW-CH-11-06	427472.22	5478363.83	381.71	0.60	42	0	0.6		0.6	0.6
CON	NEW	CH	CW-CH-11-06A	427470.19	5478370.44	386.92	0.20	127	0	0.2		0.2	0.2
CON	NEW	DH	CON-09-01	427385.00	5478275.00	391.00	57.00	120	-45	11.4	0.4	13.4	
CON	NEW	DH	CON-09-02	427385.00	5478275.00	391.00	70.00	120	-60	12.5		14.9	
CON	NEW	DH	CW-11-01	427302.50	5478322.10	385.00	169.70	121	-55	8.6	5.4	27.4	2.1
CON	NEW	DH	CW-11-02	427235.00	5478130.10	390.00	130.83	119	-55	21.0		28.8	1.0
CON	NEW	DH	CW-11-03	427299.30	5478208.20	396.00	101.00	119	-55	10.9	1.5	16.4	2.0
CON	NEW	DH	CW-11-04	427196.30	5478029.80	381.00	154.30	120	-55	12.8	1.2	20.2	2.0
CON	NEW	DH	CW-11-05	427195.80	5477997.20	381.00	170.00	120	-50	10.6		15.2	3.0
CON	NEW	DH	CW-11-06	427116.90	5478098.80	374.00	275.00	120	-45				

AREA	PROG	TYPE	DHID	EASTING	NORTHING	ELEV	EODH	AZIM	INCL	SPD PEG [m]	PEG [m]	Li_2O [m]	DENS [m]
CON	NEW	DH	CW-11-07	427226.10	5478178.10	388.00	157.40	120	-45	15.6		19.6	3.0
CON	NEW	DH	CW-11-08	427257.00	5478289.40	384.00	218.20	120	-50	10.7	1.6	16.0	2.0
CON	NEW	DH	CW-11-09	427365.40	5478355.60	382.00	152.00	120	-50	11.3	0.9	18.1	2.0
CON	NEW	DH	CW-11-10	427426.40	5478414.00	372.00	107.00	120	-45	6.3	4.0	16.3	2.0
CON	NEW	DH	CW-11-11	427376.80	5478443.40	369.00	220.00	120	-45	5.5	3.2	14.3	2.0
CON	NEW	DH	CW-11-12	427411.30	5478509.80	369.00	176.90	120	-45	2.2	3.4	11.6	2.1
CON	NEW	DH	CW-11-13	427445.80	5478551.90	369.00	152.00	110	-50	1.6	1.3	7.0	1.8

Table 45 Table summarizing collar location, length and dip of historical and recent drill holes and channels (CH) located within the SSPA.

Name	Holeheld	East	North	Depth	Dip	Date
McVittie	MV-01	425705	5473207	64.9	-45	10. Okt 55
McVittie	MV-02	425707	5473197	105.2	-60	14. Okt 55
McVittie	MV-03	425712	5473285	93.3	-60	19. Okt 55
McVittie	MV-04	425713	5473278	59.1	-45	23. Okt 55
McVittie	MV-05	425715	5473370	73.5	-45	26. Okt 55
McVittie	MV-06	425715	5473363	96.6	-60	29. Okt 55
McVittie	MV-07	425728.32	5473467.94	111.6	-60	09. Nov 55
McVittie	MV-08	425730	5473560	79.6	-45	16. Nov 55
McVittie	MV-09	425730	5473550	110.9	-60	25. Nov 55
McVittie	MV-10	425728.51	5473651.02	116.1	-60	07. Jan 56
McVittie	MV-11	425730	5473730	80.5	-45	13. Jan 56
McVittie	MV-12	425730	5473740	102.1	-60	22. Jan 56
McVittie	NV-87-1	425831.4	5473398.8	65.8	-45	07. Aug 87
McVittie	NV-87-2	425830.6	5473434.2	84.1	-45	10. Aug 87
McVittie	McVittie-16-CH-1	425760.63	5473311.356	4.93	0	23. Okt 16
McVittie	McVittie-16-CH-1W	425759.643	5473309.643	1.92	0	22. Okt 16
McVittie	McVittie-16-CH-2	425748.635	5473322.635	8.35	0	23. Okt 16
McVittie	McVittie-16-CH-3	425737.819	5473353.895	8.59	0	24. Okt 16
McVittie	McVittie-16-CH-3W	425736.932	5473354.03	1.06	0	23. Okt 16
McVittie	McVittie-16-CH-4	425724.309	5473391.53	6.09	0	24. Okt 16
McVittie	McVittie-16-CH-4E	425729.43	5473395.022	1.02	0	24. Okt 16
McVittie	McVittie-16-CH-4W	425723.199	5473391.829	0.85	0	24. Okt 16
McVittie	McVittie-16-CH-5	425713.587	5473420.665	5.76	0	25. Okt 16
McVittie	McVittie-16-CH-5E	425719.236	5473421.05	2.29	0	26. Okt 16
Jean Lake	JL-01	432462.45	5471917.1	54.90	-50	12. Nov 55
Jean Lake	JL-02	432443.35	5471917.99	66.45	-50	16. Nov 55
Jean Lake	JL-03	432424.77	5471920.39	91.44	-50	20. Nov 55
Jean Lake	JL-04	432406.3	5471921.01	62.50	-50	22. Nov 55
Jean Lake	JL-05	432471.11	5471916.68	145.10	-50	27. Nov 55
Jean Lake	JL-06	432471.11	5471916.68	55.80	-50	02. Dez 55
Jean Lake	JL-07	432445.83	5471966.56	61.30	-45	04. Dez 55

Name	Holeheld	East	North	Depth	Dip	Date
Jean Lake	JL-08	432535.6	5471904	78.03	-50	11. Dez 55
Jean Lake	JL-09	432558.38	5471926.53	126.20	-45	14. Dez 55
Jean Lake	JL-10	432481.38	5471943.16	137.20	-45	21. Dez 55
Jean Lake	JL-11	432569.54	5471938.36	212.80	-33	17. Jan 56
Jean Lake	JL-12	432499.22	5471953.68	91.44	-45	21. Jan 56
Jean Lake	JL-13	432499.22	5471953.68	79.90	-45	24. Jan 56
Jean Lake	JL-14	432431.93	5471957.67	46.94	-45	27. Jan 56
Jean Lake	JL-15	432433.05	5471926.08	46.02	-45	29. Jan 56
Jean Lake	JL-16	432532.08	5471910.5	68.30	-45	02. Feb 56
Jean Lake	JL-17	432464.72	5471843.3	214.00	-50	15. Feb 56
Jean Lake	JL-18	432446.88	5471844.57	209.40	-50	27. Feb 56
Jean Lake	JL-19	432419.23	5471848.83	217.32	-50	04. Mrz 56
Jean Lake	JL-20	432391.49	5471856.19	240.50	-50	12. Mrz 56
Jean Lake	JL-21	432364.9	5471869	245.10	-50	21. Mrz 56
Jean Lake	JL-22	432512.06	5471832.85	241.71	-50	23. Mrz 56
Jean Lake	JL-23	432531.69	5471838.92	202.70	-50	05. Apr 56
Jean Lake	JL-24	432557.34	5471820.73	266.70	-50	13. Apr 56
Jean Lake	JL-25	432583.81	5471802.4	355.10	-50	23. Apr 56
Jean Lake	JL-26	432549.39	5471755.27	359.70	-50	03. Mai 56
Jean Lake	JL-27	432434.62	5471751.18	445.01	-50	23. Mai 56
Jean Lake	JL-28	432324.82	5471784.62	471.83	-50	02. Jul 56
Jean Lake	A-01	431725.48	5472029.67	70.10	-50	17. Mrz 56
Jean Lake	A-02	431761.59	5472032.39	77.42	-50	20. Mrz 56
Jean Lake	A-03	431796.33	5472033.07	93.57	-50	22. Mrz 56
Jean Lake	A-04	431744.55	5471984.02	120.70	-50	26. Mrz 56
Jean Lake	A-05	431742.51	5472031.03	110.34	-45	31. Mrz 56
Jean Lake	A-06	431688.69	5472027.62	92.05	-50	04. Apr 56
Jean Lake	A-07	431779.30	5471982.66	121.92	-50	06. Apr 56
Jean Lake	A-08	431709.81	5471980.62	154.23	-50	11. Apr 56
Jean Lake	A-09	431879.44	5472030.35	153.31	-50	17. Apr 56
Jean Lake	A-10	431792.24	5472143.43	105.16	-45	06. Jun 56
Jean Lake	B-01	432007.51	5471943.15	167.94	-45	26. Apr 56
Jean Lake	B-02	432036.80	5471851.86	138.38	-45	02. Mai 56
Jean Lake	B-03	432066.10	5471763.30	157.58	-45	07. Mai 56
Jean Lake	B-04	432109.02	5471675.42	109.73	-45	18. Mai 56
Jean Lake	B-05	432051.79	5471808.26	122.22	-35	25. Mai 56
Jean Lake	B-06	431968.68	5471980.62	106.68	-60	01. Jun 56
Jean Lake	FL-1-89	436323.66	5473671.96	48.77	-55	12. Okt 89
Jean Lake	T-06	436768.83	5473813.15	118.57	-45	22. Sep 56
Jean Lake	T-07	436800.39	5473817.04	118.57	-45	25. Sep 56
Jean Lake	T-08	436919.59	5473827.95	152.40	-65	30. Sep 56
Jean Lake	T-09	436724.03	5473922.61	93.73	-45	04. Okt 56

Name	Holeheld	East	North	Depth	Dip	Date
Jean Lake	T-10	436659.37	5473931.18	114.91	-45	06. Okt 56
Jean Lake	PL-11-01	432557.40	5471810.00	221.96	-50	11. Mrz 11
Jean Lake	PL-11-02	432624.60	5471837.50	224.00	-45	13. Mrz 11
Jean Lake	PL-16-CH-01	432451.00	5471967.00	3.01	0	06. Nov 16
Jean Lake	PL-16-CH-02	432441.94	5471960.97	4.22	0	06. Nov 16
Jean Lake	PL-17-01	432451.34	5471919.4	77.00	-50	10.04.2017
Jean Lake	PL-17-02	432447.56	5471885.1	203.00	-50	12.04.2017
Jean Lake	PL-17-03	432447.56	5471885.1	221.00	-60	14.04.2017
Jean Lake	PL-17-04	432405.8	5471880.23	236.00	-45	17.04.2017
Jean Lake	PL-17-05	432405.8	5471880.23	221.00	-55	19.04.2017
Jean Lake	PL-17-06	432372.2	5471884.66	221.00	-55	22.04.2017
Jean Lake	PL-17-07	432372.2	5471884.66	209.00	-45	24.04.2017
Newkirk	D-20	430828.9372	5464098.499	92.66	-60	17. Okt 55
Newkirk	D-21	430830.7375	5464236.223	50.29	-50	19. Okt 55
Newkirk	D-22	430830.7375	5464236.223	77.11	-90	22. Okt 55
Newkirk	D-23	430795.6313	5464245.225	54.25	-40	27. Okt 55
Newkirk	D-24	430795.6313	5464245.225	74.98	-90	30. Okt 55
Newkirk	D-25	430765.026	5464254.226	67.06	-40	04. Nov 55
Newkirk	D-26	430765.026	5464254.226	69.04	-90	06. Nov 55
Newkirk	D-27	430728.1195	5464228.122	46.94	-40	10. Nov 55
Newkirk	D-28	430691.213	5464225.421	53.64	-40	14. Nov 55
Newkirk	D-29	430607.2283	5464268.269	83.85	-40	21. Nov 55
Newkirk	D-30	430670.2394	5464221.1	73.30	-50	26. Nov 55
Newkirk	D-31	430614.4296	5464261.608	62.18	-40	04. Dez 55
Newkirk	D-32	430625.2315	5464240.904	40.39	-40	07. Sep 55
Newkirk	D-33	430594.6261	5464272.409	31.70	-40	10. Dez 55
Newkirk	D-34	430560.4201	5464268.809	31.55	-40	13. Dez 55
Newkirk	D-35	430529.8147	5464276.91	30.94	-40	14. Dez 55
Newkirk	D-36	430557.7196	5464246.305	92.66	-40	20. Dez 55
Newkirk	D-37	430699.9446	5464375.027	128.02	-35	30. Dez 56
Newkirk	D-38	430502.81	5464290.413	37.80	-40	03. Jan 56
Newkirk	D-39	430502.81	5464290.413	39.62	-40	05. Jan 56
Newkirk	D-40	430447.9004	5464313.817	36.42	-40	07. Jan 56
Newkirk	D-41	430476.7054	5464303.015	43.59	-90	10. Jan 56
Newkirk	D-42	430418.1952	5464330.02	29.11	-40	13. Jan 56
Newkirk	D-43	430699.9446	5464375.027	147.83	-90	18. Jan 56
Newkirk	D-44	430386.6896	5464345.322	62.48	-40	21. Jan 56
Newkirk	D-45	430356.9844	5464358.825	44.50	-40	24. Jan 56
Newkirk	D-46	430311.9766	5464378.628	57.52	not recorded	27. Jan 56
Newkirk	D-49	430557.4496	5464182.394	154.84	-35	09. Feb 56
Newkirk	D-50	430557.4496	5464063.573	155.45	-35	14. Feb 56
Newkirk	D-51	430621.3608	5464178.793	76.20	-45	20. Feb 56

Name	Holeheld	East	North	Depth	Dip	Date
Newkirk	D-52	430555.6493	5463950.153	156.97	-35	29. Feb 56
Newkirk	D-53	430561.05	5463817.65	not recorded	not recorded	not recorded
Newkirk	D-54	430562.85	5463693.43	not recorded	not recorded	not recorded
Newkirk	D-55	429873.33	5463715.83	not recorded	not recorded	not recorded
Newkirk	N-01	430947.4621	5464096.917	153.01	-41	23. Sep 55
Newkirk	N-02	430948.2193	5464117.362	49.07	-44	25. Sep 55
Newkirk	N-03	430950.4909	5464164.057	46.33	-45	27. Sep 55
Newkirk	N-04	430954.0246	5464256.185	106.68	-53	05. Okt 55
Newkirk	N-05	430979.0128	5464117.16	106.68	-44	14. Okt 55
Newkirk	N-06	430980.0224	5464162.341	60.96	-60	15. Okt 55
Newkirk	N-07	430977.4983	5464317.419	124.97	-60	22. Okt 55
Newkirk	N-08	431082.2466	5464275.267	121.92	-60	29. Okt 55
Newkirk	NK-11-C03	430950.21	5464072.32	8.03	0	22. Okt 11
Newkirk	NK-11-C02	430923.75	5464076.82	11.04	0	20. Okt 11
Newkirk	NK-11-C01	430884.35	5464085.76	5.91	0	20. Okt 11
Newkirk	NK-11-CH-1N-16Ex	430885.07	5464088.04	7.23	0	11. Nov 16
Newkirk	NK-11-CH-1S-16Ex	430883.94	5464081.78	1.5	0	11. Nov 16
Newkirk	NK-11-CH-2-16Ex	430926.15	5464081.07	5.26	0	10. Nov 16
Newkirk	NK-11-CH-2S-16Ex	430923.46	5464071.02	1.68	0	11. Nov 16
Newkirk	NK-16-CH-04	430840.85	5464101.89	4.67	0	10. Nov 16
Newkirk	NK-16-CH-4S	430841.83	5464099.74	0.94	0	10. Nov 16
Aumacho	AM-10-01	427311	5461477.4	71	-50	09. Dez 10
Aumacho	AM-10-02	427297.3	5461431.1	58.75	-50	10. Dez 10
Aumacho	AM-10-03	427565.5	5461593.6	70.65	-50	13. Dez 10
Aumacho	A-01	427315.5834	5461459.07	78.9	-45	not recorded
Aumacho	A-02	427325.2305	5461442.838	123.4	-45	not recorded
Aumacho	A-03	427325.2305	5461442.838	77.1	-45	not recorded
Aumacho	A-04	427315.1213	5461459.879	114.6	-78	not recorded
Aumacho	A-05	427272.8939	5461427.645	137.5	-45	not recorded
Aumacho	A-06	427273.876	5461426.143	62.8	-40	not recorded
Aumacho	A-07	427304.0879	5461421.522	107.7	-45	not recorded
Aumacho	A-08	427304.0879	5461421.522	85	-60	not recorded
Aumacho	A-09	427304.0879	5461421.522	75.9	-45	21. Nov 55
Aumacho	A-10	427273.876	5461426.143	82	-77	27. Nov 55
Aumacho	A-11	427283.7078	5461418.114	74.1	-45	02. Dez 55
Aumacho	A-12	427303.5218	5461420.771	60	-80	07. Dez 55
Aumacho	A-13	427283.7078	5461418.114	69.5	-85	12. Dez 55
Aumacho	AM-09-01	427316	5461454	55	-45	09. Nov 09
Aumacho	AM-09-02	427318	5461459	55	-45	11. Nov 09
Aumacho	AM-09-03	427271	5461411	65	-60	14. Nov 09
Aumacho	AM-11-CH-01	427217.29	5461360.37	5.35	0	29. Jun 11
Aumacho	AM-16-CH-01	427320.28	5461356.233	5.49	0	27. Okt 16

Name	Holeheld	East	North	Depth	Dip	Date
Aumacho	AM-16-CH-02	427336.553	5461448.622	7.13	0	27. Okt 16
Aumacho	AM-16-CH-03	427325.543	5461478.648	6.1	0	29. Okt 16
Aumacho	Unknown 1	427574	5461577	not recorded	-45	not recorded
Aumacho	Unknown 2	427698	5461559	not recorded	-85	not recorded
Aumacho	Unknown 3	427695	5461533	not recorded	-67	not recorded
Aumacho	AM-17-01	427289.76	5461423.1	92	-45	04.05.2017
Aumacho	AM-17-02	427289.76	5461423.1	80	-85	05.05.2017
Aumacho	AM-17-03	427253.4	5461435.43	86	-60	06.05.2017
Aumacho	AM-17-04	427269.59	5461437.1	80	-50	07.01.2017
Aumacho	AM-17-05	427268.1	5461435	92	-50	07.05.2017
Aumacho	AM-17-06	427285.11	5461453.56	74	-55	09.05.2017
Aumacho	AM-17-07	427309.13	5461476.04	80	-50	10.05.2017
MNW	MNW-01	426943.3	5453920.7	46	-45.3	1956
MNW	MNW-02	426934.4	5453933.576	63.4	-46.9	1956
MNW	MNW-03	426944.6751	5453954.29	48.8	-45	1956
MNW	MNW-04	426941.9362	5453972.606	61.6	-44.4	1956
MNW	MNW-05	426946.0446	5453990.237	53.6	-44.5	1956
MNW	MNW-06	426946.3869	5454007.698	52.1	-43.5	1956
MNW	MNW-07	426944.6751	5454025.33	55.8	-43.8	1956
MNW	MNW-08	426947.0716	5454041.934	53	-44.2	1956
MNW	MNW-09	427002.1999	5453988.348	41.8	-43.8	1956
MNW	MNW-13	426997.6819	5453846.603	62.2	-42.2	1956
MNW	MNW-14	426983.9608	5453766.452	78.6	-40.3	1956

APPENDIX 2: ASSESSMENT FILES USED FOR THIS REPORT

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
52H08NE0009	Nama Creek Mines Ltd	Nama Creek Mines Ltd	line cutting, trenching, sampling	1955	Main Zone/North, Main Zone/Southwest, South Zone/West Dyke 1
52H08NE0006	Nama Creek Mines Ltd	Nama Creek Mines Ltd	drilling	1955	Main Zone/North, Main Zone/Southwest, South Zone/West Dyke 1
52H08NE0005	New Highridge Mining Company Ltd./Nama Creek Mines Ltd.	New Highridge Mining Company Ltd.	drilling	1955	South Zone/West Dyke 1, Southeast Zone/Line 60 Dyke
42E05NW0006	Caral Mines Ltd.	Spencer	geological survey	1955	Caral Dyke
52H08NE0004	Caral Mines Ltd.	Caral Mines Ltd.	drilling	1956	Caral Dyke
52H08NE0007	Kenogamisis Gold Mines Ltd.	Boyles Bros.	drilling	1955	Kenogamisis Dyke

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
52H08NE0003	United Montauben Mines	United Montauben Mines	drilling	1957	No. 1 Dyke
42E05NW0008	E.S.Conway	E.S.Conway	drilling	1958	No. 4 Dyke, Norland Dyke, Conway Dyke
2.44004	James Bay Midarctic Developments Inc.	James Bay Midarctic Developments Inc.	drilling	2009	Conway Dyke

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
52H08NE9236	Noranda Mines Ltd.	Noranda Mines Ltd.	drilling	1955/56	
52H08NE0002	Noranda Mines Ltd.	Noranda Mines Ltd.	line cutting, ground magnetic and electromagnetic surveys	1978/79	
42E05SW0006	Armeno Resources Inc.	MPH Consulting Ltd.	geological mapping, line cutting, ground geophysical surveying	1984	
42E05SW0005	Armeno Resources Inc.	NVC Engineering Ltd.	soil geochemical survey	1986	

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
52H08NE9234	Armeno Resources Inc.	Norwescon Development Ltd.	drilling	1987	

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
42E05NW0009	Jean Lake Lithium Mines Ltd.	Cameron Drilling Company Ltd.	drilling	1955/56	No. 4 Dyke/ Parole Lake, No. 1, 3 and 5 Dykes
42E05NW0007	Assad Group	Goldale Mines Ltd. Et al.	line cutting, prospecting, stripping, mapping and a magnetometer survey	1956	NW shore of Jean Lake
42E05NW0010	Towagmac Exploration Company Ltd.	Goldale Syndicate	drilling	1956	Giles Lithium Dyke, Foster Dyke, NE shore of Jean Lake
42E05NW0011	Sutherland & Associates	Sutherland & Associates	drilling	1965	Pomace Creek Dyke
42E05NW0004	Hudson Bay Exploration and Development Company Ltd.	Hudson Bay Exploration and Development Company Ltd.	line cutting and a VLF electromagnetic survey	1986	Foster Property
42E05NW0002	Armeno Resources Inc.	Phantom Exploration Services Ltd.	line cutting and geological survey	1987	Foster-Lew Property
42E05NW0003	Armeno Resources Inc.	Phantom Exploration Services Ltd.	proton magnetometer and VLF electromagnetic surveys	1987	Foster-Lew Property
42E05NW0005	Armeno Resources Inc.	Phantom Exploration Services Ltd.	stripping	1987	Foster-Lew Property
42E05NW0012	Armeno Resources Inc.	Phantom Exploration Services Ltd.	line cutting, photon magnetometer and VLF electromagnetic surveys	1988	No. 1, 3, 4 (Parole Lake) and 5 Dykes

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
52H01NE0003	Armeno Resources Inc.	Phantom Exploration Services Ltd.	geochemistry, mineralogy	1988	Jean Lake
42E05NW0001	Armeno Resources Inc.	Phantom Exploration Services Ltd.	drilling	1989	Foster-Lew Property
2.44004	James Bay Midarctic Developments Inc.	James Bay Midarctic Developments Inc.	drilling	2009	Foster-Lew Property

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
42E05SW0028	Canadian Lithium Mining Corporation Ltd.	Geo-Technical Development Company Ltd.	line cutting and geological survey	1955	S of Blay Lake
42E05SW0027	Mogul Mining Corporation Ltd.	Mogul Mining Corporation Ltd.	line cutting, magnetometer survey and geological survey	1956	E of Blay Lake, N of Abner Lake
42E05SW0018	Aumacho River Mines Ltd.	Aumacho River Mines Ltd.	drilling	1955	Brink
42E05SW0011	Georgia Lakes Lithium Mines Ltd.	Boyles Bros. Drilling Ltd.	drilling	1955/56	Georgia Lake
52H08SE0002	Canadian Lithium Mining Corporation Ltd.	Boyles Bros. Drilling Ltd.	drilling	1956	W of Blay Lake
42E05SW0019	Ontario Lithium Company Ltd.	Ontario Lithium Company Ltd.	drilling	1957	E of Abner Lake
42E05SE0012	Audrey M. Hayes	James R. B. Parres and Brent Parres	line cutting and magnetometer survey	1994	Brink
42E05SW0002	Audrey M. Hayes	James R. B. Parres and Brent Parres	VLF electromagnetic survey	1995	Brink
2.44004	James Bay Midarctic Developments Inc.	James Bay Midarctic Developments Inc.	drilling	2009	Brink

Assessment file/work report number	Owner	Performed by	Type of work	Year	Zone
	Consolidated Mining and Smelting Company of Canada Ltd.	Consolidated Mining and Smelting Company of Canada Ltd.	drilling, mapping, trenching and sampling	1956	
	Bird River Mines	Bird River Mines	mapping and sampling	1974	
52H01NE0006	John Donner	Bird River Mines/ University of Manitoba	trenching, sampling	1978	
52H01NE0005	Boris Zayachkivsky	Boris Zayachkivsky	geological survey	1986	
52H01NE0004	Armeno Resources Inc.	Phantom Exploration Services Ltd.	line cutting, proton magnetometer and VLF electromagnetic surveys	1987	
52H01NE0003	Armeno Resources Inc.	Phantom Exploration Services Ltd.	analytical and mineralogical survey	1988	
52H01NE0002	Armeno Resources Inc.	Phantom Exploration Services Ltd.	radiometric and VLF electromagnetic surveys	1989	
52H01NE0001	Armeno Resources Inc.	Phantom Exploration Services Ltd.	geochemical analysis	1989	

Assessment file/work report number	Owner	Performed by	Type of work	Time	Zone
42E05SW0009	Slush Lake Group	Boyles Bros. Drilling Ltd.	drilling	1955	Newkirk
42E05SW0012	Dunvegan Mines Ltd.	Midwest Diamond Drilling Co.	drilling	1955	Vegan No. 2
42E05SW0025	Newkirk Mining Corporation Ltd.	Geotechnical Development Company Ltd.	line cutting, geological survey	1955	Newkirk
42E05SW0006	Armeno Resources Inc.	MPH Consulting Ltd.	line cutting, magnetic survey, VLF electromagnetic survey, gradiometry survey and geological mapping	1984	Newkirk and Vegan

Assessment file/work report number	Owner	Performed by	Type of work	Time	Zone
42E05SW0005	Armeno Resources Inc.	NVC Engineering	geochemical soil survey	1986	Newkirk and Vegan
42E05SW0003	Armeno Resources Inc.	Phantom Exploration Services Ltd.	proton magnetometer and VLF electromagnetic surveys	1987	Newkirk and Vegan
2E05SW0004	Armeno Resources Inc.	Phantom Exploration Services Ltd.	stripping	1988	Newkirk
42E05SW8331	Armeno Resources Inc.	Phantom Exploration Services Ltd.	geological survey	1988	Newkirk and Vegan

Additionally used literature

Breaks, F.W., Selway, J.B. and Tindle, A.G. (2008): The Georgia Lake rare-element pegmatite field and related S-type, peraluminous granite, Quetico Subprovince, north-central Ontario; Ontario Geological Survey, Open File Report 6199, 176p.

Černý, P., Trueman, D.L., Ziehlke, D.V., Goad, B.E. and Paul, B.J. (1981): The Cat Lake-Winnipeg River and the Wekusko Lake pegmatite fields, Manitoba. Manitoba Department of Energy and Mines, Mineral Resources Division, Economic Geology Report ER80-1, 216 p.

Hewitt, D.F. (1972); Rocks and Minerals of Ontario, Geological Circular 13, Ontario Department of Mines and Northern Affairs, Toronto

Selway, J, Baker, J., Peshkepia, A, Magyarosi, Z. and Dixon. A. (2011b): Independent Technical Report and Estimated Resources for Georgia Lake Lithium Property, Beardmore, Ontario, Canada, prepared for Rock Tech Lithium Inc., dated Nov. 2, 2011.

Selway, J. and Dixon, A. (2012a): Channel and Drill Report for Phase 1: 2011 Summer Channel and Phase 2: 2011 Fall Channel and Drill Program: Nama Creek, Conway, Aumacho, Nama Creek, Newkirk-Vegan, Georgia Lake pegmatite field, Ontario, Canada, NTS sheets: 42E05NW and 52H08NE, prepared for Rock Tech Lithium Inc., submission date Aug. 17, 2012, MNDMF assessment file number pending.

Selway, J., Magyarosi, Z, Ronacher, E., Tucker, M., Peshkepia, A., and McKenzie, J. (2011a): Independent Technical Report, Georgia Lake Lithium Property, Beardmore, Ontario, Canada, prepared for Rock Tech Lithium Inc., dated Mar. 25, 2011.

Zayachivsky, B. (1985): Granitoids and rare-earth element pegmatites of the Georgia Lake area, northwestern Ontario; unpublished M.Sc. thesis, Lakehead University, Thunder Bay, Ontario, 234p

APPENDIX 3: CLAIM NUMBER HISTORY

Old Claim Number	new Claim Number	Township/Area
4283894	344757	Kilkenny
4253781, 4253782	334733	Kilkenny
4253781	331861	Kilkenny
4253781	327329	Kilkenny
4253781	314571	Kilkenny
4253781	314570	Kilkenny
4253785	308917	Kilkenny
4283894	305332	Kilkenny
4253783	296585	Kilkenny
4253782, 4283894	293192	Kilkenny
4283894	293191	Kilkenny
4253783	288479	Kilkenny
4253783	288478	Kilkenny
4253781, 4253782 , 4283894	284578	Kilkenny
4253781	278007	Kilkenny
4253781	277987	Kilkenny
4253781	260008	Kilkenny
4253781	260007	Kilkenny
4253782	258045	Kilkenny
4253782	258032	Kilkenny
4253782	257144	Kilkenny
4253790	254946	Kilkenny
4253790, 4253791	244210	Kilkenny
4253782	239854	Kilkenny
4253782	239853	Kilkenny
4253785	231904	Kilkenny
4253781	230728	Kilkenny
4253781	224064	Kilkenny
4253781, 4253782	222058	Kilkenny
4283894	218621	Kilkenny
4283894	218620	Kilkenny
4253781	212013	Kilkenny
4253782	210049	Kilkenny
4253781, 4253782	210027	Kilkenny
4253790	207141	Kilkenny
4253790	207140	Kilkenny
4253781	203998	Kilkenny
4253785	194468	Kilkenny
4253790	187622	Kilkenny
4253786, 4253790, 4253791	184915	Kilkenny
4253786, 4253790	184914	Kilkenny
4253786, 4253790	184064	Kilkenny
4253790	180171	Kilkenny
4253782	172048	Kilkenny
4283894	169789	Kilkenny
4253785, 4253786	165287	Kilkenny
4253782	157397	Kilkenny
4253782	157396	Kilkenny
4253783	146069	Kilkenny
4253781, 4253782	145282	Kilkenny
4253790	141079	Kilkenny
4253781	139846	Kilkenny
4253790	132071	Kilkenny
4253790	132070	Kilkenny
4253783	128670	Kilkenny
4253782, 4253783	128669	Kilkenny

Old Claim Number	new Claim Number	Township/Area
4253781, 4283894	125308	Kilkenny
4253794, 4253795	344793	Lake Jean Area
4253787, 4283894	344758	Lake Jean Area
3005434	342241	Lake Jean Area
3005434, 3009087	342240	Lake Jean Area
4253792, 4253795	340997	Lake Jean Area
4253787, 4253792	340996	Lake Jean Area
4253791	339163	Lake Jean Area
4253791	339162	Lake Jean Area
4253788	335846	Lake Jean Area
3009087, 3009122	335845	Lake Jean Area
3009122	335844	Lake Jean Area
3009087	334731	Lake Jean Area
4253788, 4253789	333104	Lake Jean Area
4253787, 4253789	332147	Lake Jean Area
4253789, 4253793	331609	Lake Jean Area
4253787	328730	Lake Jean Area
4253787	328711	Lake Jean Area
3009122	327111	Lake Jean Area
4253793	326591	Lake Jean Area
4253795	323418	Lake Jean Area
4253795	323417	Lake Jean Area
4253795, 4253796	323416	Lake Jean Area
3005434, 3009087	320172	Lake Jean Area
4253793	313831	Lake Jean Area
4253789, 4253793	313830	Lake Jean Area
3009087	312610	Lake Jean Area
3009121	310367	Lake Jean Area
4253787	309239	Lake Jean Area
3009122	307702	Lake Jean Area
3009087	305793	Lake Jean Area
3009087	305792	Lake Jean Area
4253794	305369	Lake Jean Area
4253795, 4253796	304416	Lake Jean Area
3005434	303480	Lake Jean Area
3005434, 4253786	303459	Lake Jean Area
3005434, 4253786	303458	Lake Jean Area
4253790, 4253791	300299	Lake Jean Area
4253794	293232	Lake Jean Area
4283894	293190	Lake Jean Area
3005434, 3009087	291323	Lake Jean Area
4253794	285135	Lake Jean Area
3009087	275993	Lake Jean Area
3009121	273068	Lake Jean Area
3009121, 3009122	266435	Lake Jean Area
4253786	263715	Lake Jean Area
4253793	258772	Lake Jean Area
4253793	258771	Lake Jean Area
4253796	258678	Lake Jean Area
3009087	257142	Lake Jean Area
3009087	257141	Lake Jean Area
4253794, 4253795	255987	Lake Jean Area
3009087, 4283894	255953	Lake Jean Area
4253795	255225	Lake Jean Area
3009121, 3009122	254950	Lake Jean Area
4253792, 4253793	253549	Lake Jean Area
4253788, 4253789	253224	Lake Jean Area
3009122, 4253788	248431	Lake Jean Area
3009087, 3009122	248430	Lake Jean Area
4253796	247210	Lake Jean Area
4253787, 4253792	246696	Lake Jean Area
4253786	244181	Lake Jean Area

Old Claim Number	new Claim Number	Township/Area
3009087, 3009122	240377	Lake Jean Area
3009087	239838	Lake Jean Area
3009087	239837	Lake Jean Area
3009087	239836	Lake Jean Area
3005434, 3009087	236143	Lake Jean Area
3005434	236125	Lake Jean Area
3005434	235294	Lake Jean Area
4253792, 4253793, 4253795, 4253796	234029	Lake Jean Area
4253794	226614	Lake Jean Area
4253787	225448	Lake Jean Area
4253787, 4253789	225352	Lake Jean Area
4253789	225351	Lake Jean Area
4253796	222709	Lake Jean Area
4253796	222708	Lake Jean Area
4253791, 4253794	218658	Lake Jean Area
4253791, 4253792, 4253794, 4253795	218657	Lake Jean Area
4283894	218619	Lake Jean Area
3005434, 4253787	216677	Lake Jean Area
4253792, 4253793	215448	Lake Jean Area
4253792	215447	Lake Jean Area
4253789, 4253793	211252	Lake Jean Area
4253795, 4253796	208593	Lake Jean Area
3009087, 3009121, 3009122	207150	Lake Jean Area
4253788	205988	Lake Jean Area
3009122, 4253788	204435	Lake Jean Area
4253796	202635	Lake Jean Area
4253795	200580	Lake Jean Area
3005434	199694	Lake Jean Area
4253791	197101	Lake Jean Area
4253786	197074	Lake Jean Area
4253794	189326	Lake Jean Area
4253794, 4253795	189325	Lake Jean Area
4253791, 4253794	189324	Lake Jean Area
4283894	189290	Lake Jean Area
4253785, 4253786	187449	Lake Jean Area
3005434	187448	Lake Jean Area
4253791, 4253792	186743	Lake Jean Area
4253794	183301	Lake Jean Area
4253794	183300	Lake Jean Area
4253794	183299	Lake Jean Area
3005434	180689	Lake Jean Area
3005434	180688	Lake Jean Area
4253792	179970	Lake Jean Area
4253791	177563	Lake Jean Area
4253786	177537	Lake Jean Area
4253786	177536	Lake Jean Area
4253786, 4253787, 4253787	177535	Lake Jean Area
4283894	169790	Lake Jean Area
4253789	165473	Lake Jean Area
4253793	158092	Lake Jean Area
4253793, 4253796	158011	Lake Jean Area
3009087	157382	Lake Jean Area
4253795	155959	Lake Jean Area
4253795	155958	Lake Jean Area
4253794	153229	Lake Jean Area
3009087, 4283894	153196	Lake Jean Area
3009087, 4283894	153195	Lake Jean Area
4253786, 4253787, 4253791, 4253792	150200	Lake Jean Area
4253791	148976	Lake Jean Area
4253791	148959	Lake Jean Area
4253786, 4253791	148958	Lake Jean Area
3009122	145105	Lake Jean Area

Old Claim Number	new Claim Number	Township/Area
3005435, 4253786, 4253787	141545	Lake Jean Area
3009121	141085	Lake Jean Area
4253793, 4253796	138507	Lake Jean Area
3009087	137835	Lake Jean Area
4253794	137312	Lake Jean Area
4253794	137311	Lake Jean Area
3005435, 4253788	135509	Lake Jean Area
3005435, 3009087, 3009122, 4253788	135508	Lake Jean Area
3005434	135487	Lake Jean Area
3009121	135004	Lake Jean Area
4253792	134754	Lake Jean Area
3005435, 4253787, 4253788, 4253789	123503	Lake Jean Area
4253792, 4253795	122741	Lake Jean Area
3009121	122214	Lake Jean Area
4253786, 4253791	120898	Lake Jean Area
4253786	120897	Lake Jean Area
4253796	112545	Lake Jean Area
3009087	112402	Lake Jean Area
3009122	111996	Lake Jean Area
4253788	111427	Lake Jean Area
3009121	110212	Lake Jean Area
4253786	109132	Lake Jean Area
4253791, 4253792	106179	Lake Jean Area
4253792	106178	Lake Jean Area
4253787, 4253789, 4253792, 4253793	106177	Lake Jean Area
4283894	105386	Lake Jean Area
4283668	340907	Barbara Lake Area
3009119, 4283895	336485	Barbara Lake Area
3009119	330035	Barbara Lake Area
3009119	330034	Barbara Lake Area
3009119, 4281236, 4283895	323525	Barbara Lake Area
4283668	318849	Barbara Lake Area
4283896	318123	Barbara Lake Area
4283895, 4283896	318122	Barbara Lake Area
4283896	318121	Barbara Lake Area
4283896	318120	Barbara Lake Area
4283668	302202	Barbara Lake Area
4283895	296171	Barbara Lake Area
4283668	281965	Barbara Lake Area
4283668	281964	Barbara Lake Area
4283668	281963	Barbara Lake Area
4283896	252231	Barbara Lake Area
4283895, 4283896	245372	Barbara Lake Area
4283668	233958	Barbara Lake Area
4283896	233225	Barbara Lake Area
4283896	233224	Barbara Lake Area
3009119	232827	Barbara Lake Area
4281236, 4283895	227673	Barbara Lake Area
4283895, 4283896	225207	Barbara Lake Area
4283895	211644	Barbara Lake Area
3009119, 4281236, 4283895	208806	Barbara Lake Area
3009119, 4281236	200775	Barbara Lake Area
4283895	194433	Barbara Lake Area
4283895	192885	Barbara Lake Area
4283668	186148	Barbara Lake Area
4283896	185423	Barbara Lake Area
4283668	179398	Barbara Lake Area
4283896	178633	Barbara Lake Area
4283896	178632	Barbara Lake Area
4283896	178631	Barbara Lake Area
3009119, 4283895	174909	Barbara Lake Area
4281236, 4283895	171517	Barbara Lake Area

Old Claim Number	new Claim Number	Township/Area
4283668	166643	Barbara Lake Area
4283668	166642	Barbara Lake Area
4283668	166641	Barbara Lake Area
3009119	162012	Barbara Lake Area
4283668	150096	Barbara Lake Area
4283896	148869	Barbara Lake Area
4283668	134146	Barbara Lake Area
4283668	134145	Barbara Lake Area
4283668	134144	Barbara Lake Area
4283668	134143	Barbara Lake Area
4283896	133440	Barbara Lake Area
4283895, 4283896	133439	Barbara Lake Area
4283896	133438	Barbara Lake Area
4283895, 4283896	121403	Barbara Lake Area
4283896	121402	Barbara Lake Area
4281236, 4283895	115825	Barbara Lake Area
3009119	114362	Barbara Lake Area
4283896	106704	Barbara Lake Area
4283668	106094	Barbara Lake Area
4283668	106093	Barbara Lake Area
4283668	106092	Barbara Lake Area
3009119	330036	Keemile Lake Area
3009119	232828	Keemile Lake Area
3009119	216184	Keemile Lake Area
3009119	196118	Keemile Lake Area
3009119	196117	Keemile Lake Area
4283878	336306	Cosgrave Lake Area
4283878	308691	Cosgrave Lake Area
4283878	296565	Cosgrave Lake Area
4283878	296564	Cosgrave Lake Area
4283878	288460	Cosgrave Lake Area
4283878	248844	Cosgrave Lake Area
4283878	248843	Cosgrave Lake Area
4283878	248842	Cosgrave Lake Area
4283878	229857	Cosgrave Lake Area
4283878	229856	Cosgrave Lake Area
4283878	192621	Cosgrave Lake Area
4283878	192620	Cosgrave Lake Area
4283878	162596	Cosgrave Lake Area
4283878	140097	Cosgrave Lake Area
4283878	140096	Cosgrave Lake Area

ANNEX A2:U2779_RT_MCE_G_TESTING UCS JULY 2018 V01

DMT GmbH & Co. KG
Mining Consulting & Engineering
Rock Mechanics
Test Laboratory VMPA-B-2184

Essen, 15.08.2018
Tel.: +49 201/172-1327
Fax: +49 201/172-1012

Test Report
No. U2779 RT-MCE-G

Project: Testing of compressive strength and density of rock samples from Rock Tech Lithium Inc., Canada according to Recommendation 1 by DGGT

Client: Rock Tech Lithium Inc.
Canada

Engineer responsible: Dipl.-Ing. B. Pflüger
Tel.: 0201/172 - 1327

Internal Auditor: Dipl.-Ing. Axel Studeny

Number of pages: 4
Number of Enclosures: 30

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2. Test material 3

3. Preparation of specimen and testing 3

4. Test results 4

1. Definition of project and testing competence

Rock Tech Lithium Inc. provided DMT with drilling-cores from rock samples for the determination of both, density and unconfined compressive strength (UCS).

DMT operates within its business segment Mining Engineering a laboratory for Rock Mechanics including expert bodies for building material technology and support technique.

A certified test lab for concrete (Certificate No. VMPA-B-2184) is an essential element of DMT's testing expertise.

2. Test material

Test material were 12 drilling cores with an overall length between 0,18 m and 0,30 m (see Annex 1). Due to the wedge-shaped ends the true length for preparing specimen was often much shorter.

Another two hand-pieces (grab samples) were provided as well. Due to the irregular shape and some cracks, the extraction of drilling-cores was limited.

Annexes 2 to 29 show photographs of the test material. The photographs show each drilling-core before preparation, the specimen and finally the specimen after testing UCS.

3. Preparation of specimen and testing

Each drilling core was cut by wet-diamond-cutting after determining an individual cutting-scheme. This determination was done considering cracks and the aim of gaining long specimen. It was aimed to keep the length to diameter ratio as close to 2 as possible.

Most of the drilling-cores had a diameter of 47.6 mm; two samples had a slightly larger diameter of 50.7 mm. Grab pieces were drilled in the laboratory with 51.7 core-diameter.

After cutting, the specimen were ground by wet-diamond-grinding in order to achieve smooth and parallel end-planes for vertical load-impact in the test frame.

Before performing the UCS-test, each specimen was dried, weighed and measured.

As testing machine, a class 1 (highest possible accuracy) calibrated test frame of the type ToniTechnik 600 kN was used. Due to its two-column compact design, the test-frame represents a stiff construction.

DGGT (German Association of Geotechnique) provides with Recommendation 1 a suitable test procedure for testing UCS of soil and hard rock. Each test was run displacement- controlled at a deformation rate of 0.001 mm/s with a duration of more than 15 min (minimum recommended duration of testing.

4. Test results

A graphic evaluation of test results is shown in Annex 30.

For the evaluation of test results, Recommendation 1 by DGGT provides correction values depending on the shape factor above and below 2.0. These correction values are already applied in the table of annex 30, values for UCS.

(Pflüger)

Internal Auditor: (Studeny)

Rock Tech Lithium Inc.
Test Results – basic data

Running sample ID	DHID	Drillhole sample ID	DEFR [m]	DETO [m]	Length [m]	Rock Type	Test
881851	NC-11-16	1	14,00	14,18	0,18	Paragneiss	UCS
881852	NC-11-16	2	116,00	116,19	0,19	Diabase	UCS
881853	NC-11-16	3	195,20	195,50	0,30	Pegmatite	UCS
881854	NC-11-16	4	289,79	290,00	0,21	Paragneiss	UCS
881855	NC-11-16	5	292,04	292,30	0,26	Pegmatite	UCS
881856	NC-11-16	6	354,21	354,50	0,29	Pegmatite	UCS
881857	L6-11-03	1	23,83	24,08	0,25	Paragneiss	UCS
881858	L6-11-03	2	125,00	125,26	0,26	Paragneiss	UCS
881859	CW-11-09	1	6,54	6,77	0,23	Diabase	UCS
881860	CW-11-09	2	90,97	91,25	0,28	Paragneiss	UCS
881861	NC-11-16	7	110,00	110,22	0,22	Diabase	UCS
881862	NC-11-16	8	120,95	121,18	0,23	Diabase	UCS
881863	Grab piece					ore body	UCS
881864	Grab piece					ore body	UCS

Rock Tech Lithium Inc.

Rock Samples – Specimen 881851

Rock Tech Lithium Inc.

Specimen 881851/1 and 2 after UCS-testing

Rock Tech Lithium Inc.

Rock Samples – Specimen 881852

Rock Tech Lithium Inc.

Specimen 881852/1 and 2 after UCS-testing

Rock Tech Lithium Inc.

Rock Samples – Specimen 881853

Rock Tech Lithium Inc.

Specimen 881853/1 and 2 after UCS-testing

Rock Tech Lithium Inc.

Rock Samples – Specimen 881854

Rock Tech Lithium Inc.

Specimen 881854/ 2 after UCS-testing

Specimen
881854/1
disintegrated at
max. load
No photograph

Rock Tech Lithium Inc.

Rock Samples – Specimen 881855

Rock Tech Lithium Inc.

Specimen 881855/1 and 2 after UCS-testing

Rock Tech Lithium Inc.

Rock Samples – Specimen 881856

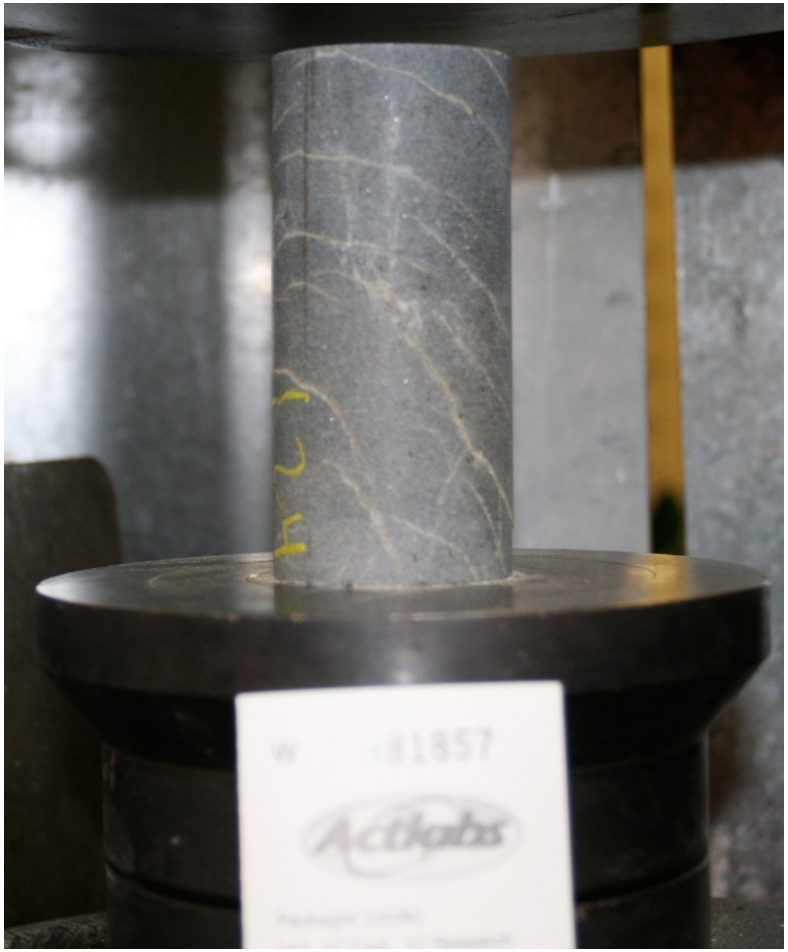
Rock Tech Lithium Inc.

Specimen 881856/1 and 2 after UCS-testing

Rock Tech Lithium Inc.
Rock Samples – Specimen 881857

Rock Tech Lithium Inc.

Specimen 881857/1 and 2 after UCS-testing



Rock Tech Lithium Inc.

Rock Samples – Specimen 881858

Rock Tech Lithium Inc.

Specimen 881858/1 and 2 after UCS-testing

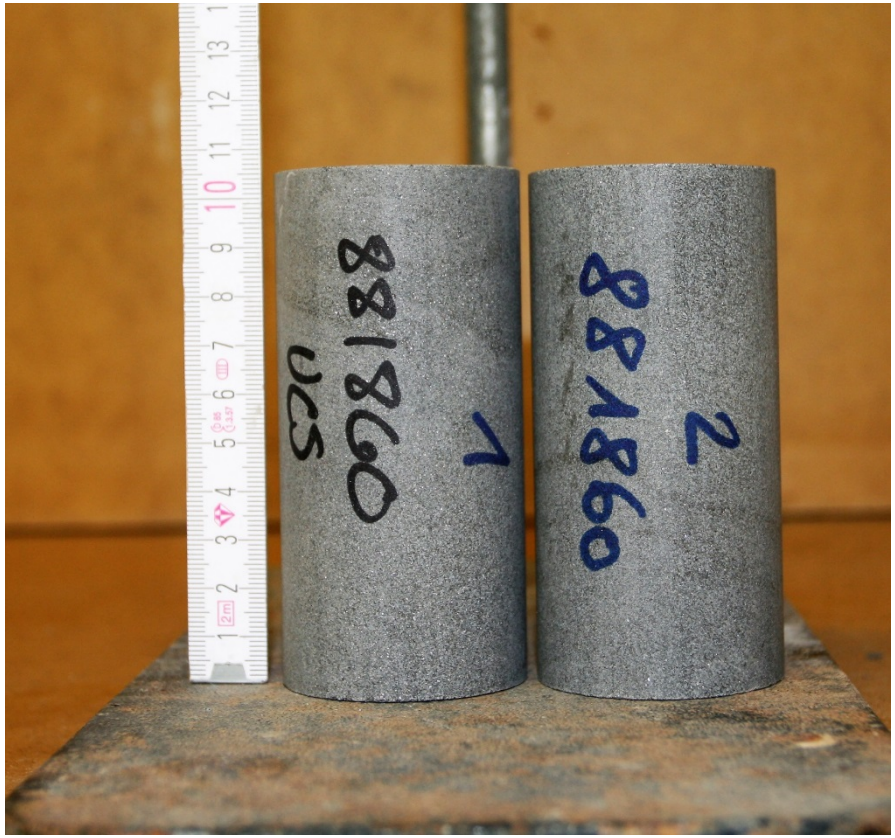
Rock Tech Lithium Inc.

Rock Samples – Specimen 881859

Rock Tech Lithium Inc.

Specimen 881859/1 and 2 after UCS-testing

Rock Tech Lithium Inc.
Rock Samples – Specimen 881860



Rock Tech Lithium Inc.

Specimen 881860/1 and 2 after UCS-testing

Rock Tech Lithium Inc.
Rock Samples – Specimen 881861

Rock Tech Lithium Inc.

Specimen 881861/1 and 2 after UCS-testing

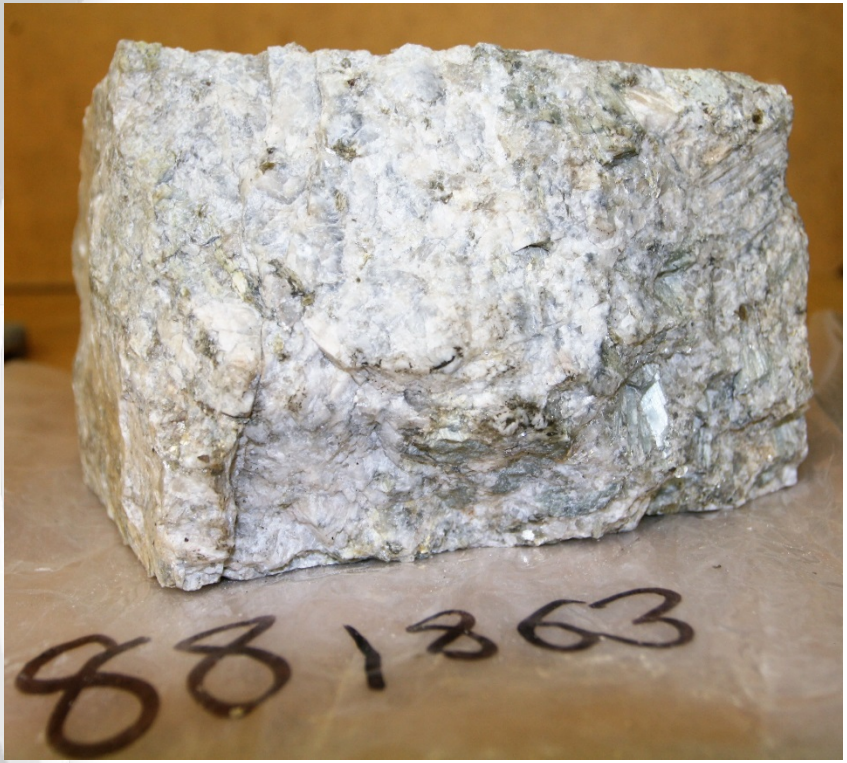
Rock Tech Lithium Inc.
Rock Samples – Specimen 881862

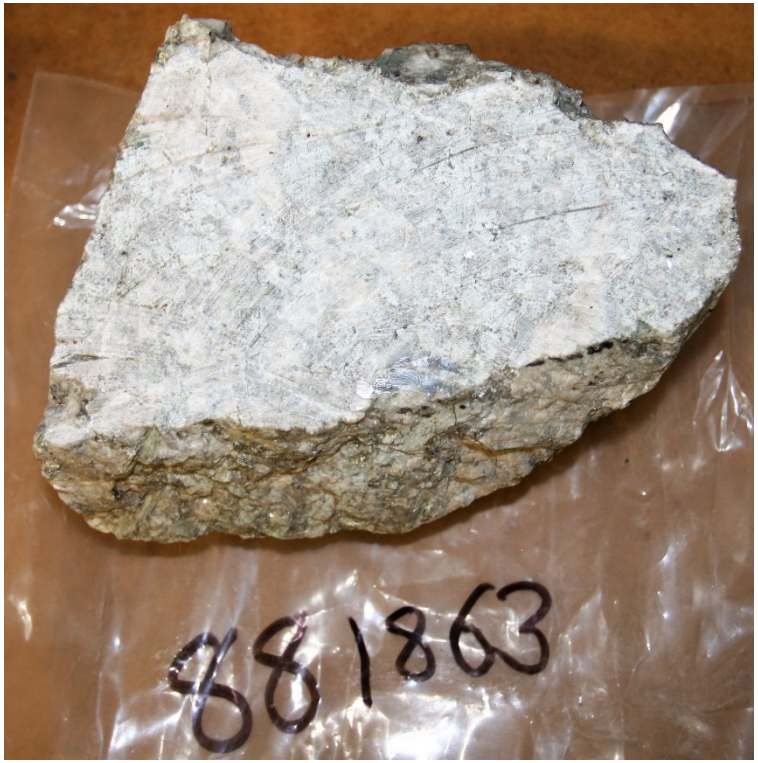
Rock Tech Lithium Inc.
Specimen 881862/1 after UCS-testing

Specimen
881862/2
disintegrated at
max. load
No photograph

Rock Tech Lithium Inc.

Rock Samples – Grab piece 881863

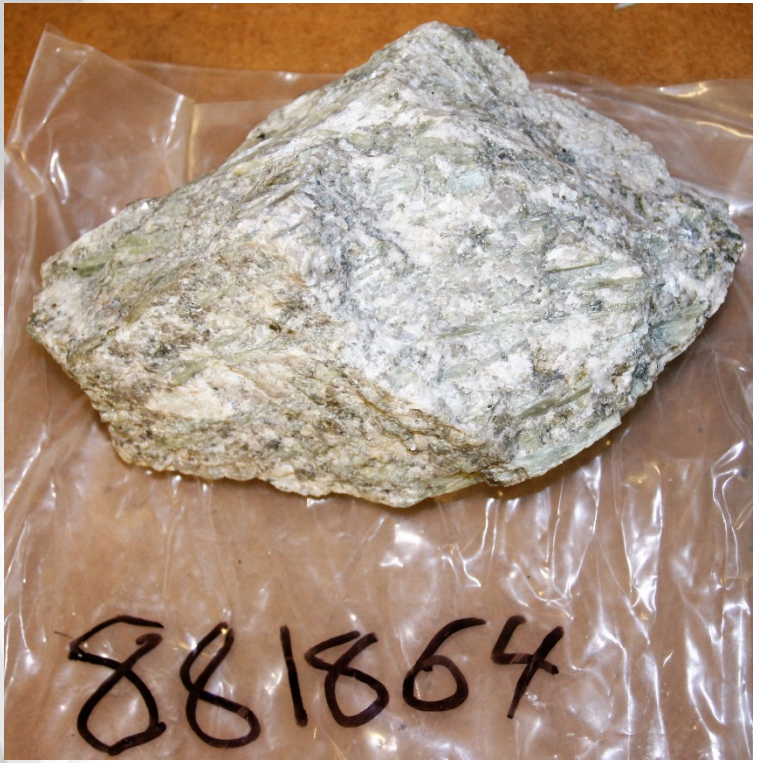


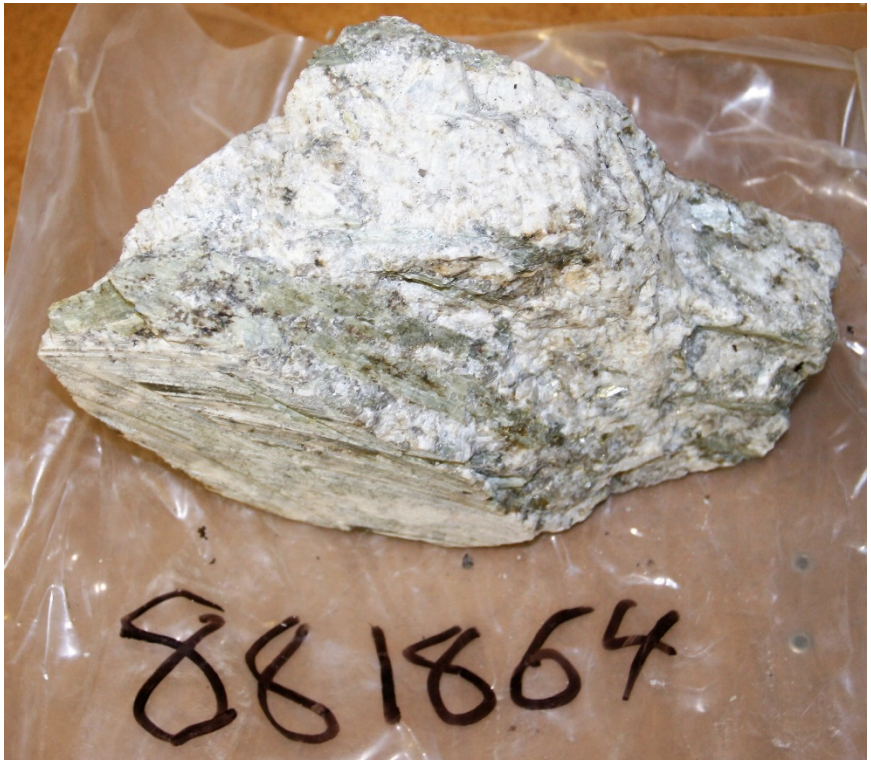


Rock Tech Lithium Inc.
Rock Samples – Specimen 881863

Rock Tech Lithium Inc.

Rock Samples – Grab piece 881864





Rock Tech Lithium Inc.
Rock Samples – Specimen 881864

Rock Tech Lithium Inc.
Test Results

Number of Samples	UCS sample 1	UCS sample 2	Average UCS	Standard Deviation UCS	Density sample 1	Density sample 2	Average Density
2	124,3	117,3	120,8	4,9	2,7	2,7	2,7
2	133,5	98,9	116,2	24,5	3,0	3,0	3,0
2	88,8	120,5	104,7	22,4	2,6	2,6	2,6
2	244,8	131,5	188,2	80,1	2,8	2,8	2,8
2	150,3	113,2	131,8	26,2	2,6	2,6	2,6
2	65,1	71,0	68,1	4,2	2,6	2,6	2,6
2	162,0	107,5	134,8	38,5	2,7	2,7	2,7
2	119,4	121,5	120,5	1,5	2,7	2,7	2,7
2	71,6	201,2	136,4	91,6	2,9	2,9	2,9
2	107,2	101,7	104,5	3,9	2,8	2,8	2,8
2	114,1	79,4	96,8	24,5	3,1	3,1	3,1
2	94,6	147,7	121,2	37,5	3,1	3,1	3,1
1	88,6		88,6	n.a.	2,7		2,7
1	80,7		80,7	n.a.	2,7		2,7

ANNEX A3: SGS REPORTS 12607-001 FINAL OCT12 2011 AND 12607-002 FINAL AUG 26 2011

An Investigation into

THE RECOVERY OF SPODUMENE FROM GEORGIA LAKE PROJECT

prepared for

ROCK TECH LITHUM INC.

Project 12607-001
October 11, 2011

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Terminology

K_{80}:	Opening size of the screen that 80% of the material will pass through
DMS:	Dense media separation
3030C:	Amine collector for mica flotation manufactured by Cytec.
Armac C:	Amine collector for mica flotation manufactured by Akzo Nobel
Armac T:	Amine collector for mica flotation manufactured by Akzo Nobel
LR19 and LR19B:	Collectors for spodumene flotation
FA-2:	Fatty acid collector for spodumene flotation manufactured by Arizona Chemical
D30LR:	Fatty acid collector manufactured by Arizona Chemical
Aero 801:	Reagent used as a promoter in spodumene flotation manufactured by Cytec
Aero 855:	Petroleum sulfonate used as a promoter in spodumene flotation manufactured by Cytec
D618:	Calcium lignosulfonate used as dispersant manufactured by Lignotech
Desliming:	Separation of slimes (particles normally less than 20 micron) from the feed
Pulp density:	the amount of solids in the pulp expressed in weight percentage
High density conditioning:	Conditioning of the pulp with the flotation reagent(s) at high pulp density

Executive Summary

Three outcrop and one drill core samples from the Georgia Lake deposit were submitted to SGS Minerals Services for metallurgical testwork by Rock Tech Lithium Inc. A single composite sample was prepared from these drill cores. The analytical assays of the composite head sample are as follows:

	Assay %														
Sample ID	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	Sum
Head Comp	0.69	1.49	75.6	16.0	0.48	0.11	0.26	3.92	2.01	< 0.01	0.16	0.08	0.01	< 0.01	0.65	99.3

Recovery of spodumene by gravity separation was evaluated by heavy liquid separation tests. The results indicated that if the head grade is about 1.5% Li_2O, DMS can most likely reject 40% of the mass as gangue and can produce high grade concentrate, 6.0% Li_2O or higher, with an expected Li recovery of 70% or higher. Flotation testing was also conducted on the head sample and on the combined heavy liquid middlings and undersize fractions (-0.5 mm). In a locked cycle test, a spodumene concentrate was generated with a grade of 6.2% Li_2O and 81.5% (global) Li recovery. Primary coarse grinding with the closing size of 300 μm, desliming, scrubbing and fatty acid conditioning are some of the key factors to a successful mica and spodumene flotation operation.

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

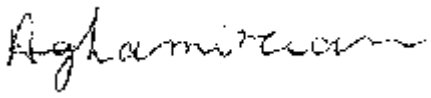
- Mica pre-flotation was required since the mica grade in the head sample was 7.2% ;
- LR19, LR19B and FA-2 were suitable and strong spodumene collectors. LR19 collector is a mixture of FA-2 and Aero 855, MIBC and fuel oil. LR19B collector is a mixture of FA-2 and D30, R801, kerosene, pine oil;
- Stage grinding with the closing size of 300 μm and K_{80} of around 210 μm was used to minimize the Li loss in the primary desliming;
- Scrubbing with lignin sulfonate at pH 11 followed by desliming was also beneficial;
- Implementing 3 or 4 cleaning stages was a suitable flotation strategy. The iron content of the lithium concentrate was high (over 1% Fe_2O_3). It was only possible to remove a portion of the iron using magnetic separation. A large portion of the iron is expected to be in the spodumene crystal structure as solid solution as confirmed in other similar projects.

In this report, details of the mineral beneficial development testwork are discussed. In addition, recommendations for the development of a commercial process are provided.

Introduction

A request was received by Mr. Afzaal Pirzada, of Rock Tech Lithium Inc., for a scoping test program on a sample from the Georgia Lake Property. The objective was to develop a flowsheet to produce spodumene concentrate with a moderate grade of 6.0 to 6.5% Li_2O, suitable for hydrometallurgy operation. The complete flowsheet was expected to include spodumene concentrate preparation, concentrate roasting followed by acid roasting, and hydrometallurgical operations to extract lithium from acid roasted material to produce lithium carbonate with high purity (more than 99.5%) as the final product.

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



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Testwork Summary

1. Sample Receipt, Description, Preparation and Characterization

Three outcrop samples and one drill core sample were received in February 2011 in a single shipment weighing approximately 770 kg. A receipt number, 0357-Feb11, was assigned to these samples. The outcrop and drill core samples were initially crushed to -1/2". About 75 kg from each sample was riffled out and further crushed to 6M. About 15 kg from each sample was also set aside for future heavy liquid tests and Bond work index testing. A subsample from each material was supplied for chemical analysis, and the results are provided in Table 1.

Table 1: Chemical Analysis of the Outcrop and Drill Core Samples

	Assay %															
Sample ID	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	Sum
Spodumene Drill Cores	0.32	0.69	74.5	15.7	0.69	0.09	0.31	4.44	2.99	< 0.01	0.17	0.05	0.02	< 0.01	0.68	99.7
Spodumene Bulk #1	0.76	1.64	76.0	16.3	0.56	0.07	0.28	4.08	1.71	< 0.01	0.19	0.08	0.02	< 0.01	0.49	99.8
Spodumene Bulk #2	0.88	1.89	75.6	15.9	0.69	0.09	0.26	3.54	1.64	< 0.01	0.18	0.06	0.02	< 0.01	0.57	98.6
Spodumene Bulk #3	0.79	1.70	74.5	15.9	0.62	0.10	0.23	3.68	1.97	< 0.01	0.19	0.10	0.02	< 0.01	0.57	97.9

The assays of the outcrop samples were similar. The drill core sample, however, had a comparatively low lithium assay. After consulting with the client, it was decided to mix the samples, based on the equal weights, to make a composite head sample. The assays of this composite are illustrated in Table 2. All of the flotation development testwork was conduced on this composite sample.

Table 2: Chemical Analysis of the Composite Head Sample

	Assay %															
Sample ID	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	Sum
Head Comp	0.69	1.49	75.6	16.0	0.48	0.11	0.26	3.92	2.01	< 0.01	0.16	0.08	0.01	< 0.01	0.65	99.3

A sub-sample from the head composite was submitted for semi quantitative XRD analysis and the results are provided in Appendix A. Details of the XRD analysis are reported in Appendix A.

Table 3: Mineralogy Analysis of the Composite Head Sample

Mineral	Spod Head Composite (wt %)
Quartz	32.4
Albite	34.4
Spodumene (Monoclinic)	19.0
Microcline	7.10
Muscovite	7.20
TOTAL	100

2. Bond Mill Work Index

2.1. Bond Rod Mill Work Index

The minus ½" spodumene composite sample was submitted for Bond rod work index (RWI) determination as per the standard procedure using a product D_{100} size of 1180 µm. A RWI of 12.5 kWh/t (metric) was obtained for the composite. Details of the test are included in the Appendix B.

3. Heavy Liquid Separation (HLS) Test Results

A number of heavy liquid separation tests were conducted on the composite head sample and original head samples with the objective to evaluate the potential for Dense Media Separation (DMS) to produce the final concentrate. The first test was conducted on the composite sample. The test flowsheet is illustrated in Figure 1. The feed was primary crushed to -3/8" and then processed by heavy liquid with a density of 2.65 g/cm^3 to reject the gangue minerals in the float stream. The sink product, which contained almost all the spodumene, was further crushed to 10 mesh and processed sequentially by heavy liquids with densities of 3.0 g/cm^3, 2.95 g/cm^3, 2.90 g/cm^3 and 2.65 g/cm^3. The summary of results is illustrated in Table 4. All the heavy liquid test data are presented in Appendix C.

The following conclusions can be made from these results:

- By using the flowsheet shown in Figure 1, at a density of 2.65 g/cm^3, 43% of the head sample mass was rejected at a lithium loss of 0.9%. Following each crushing stage, gangue silicates were floated at a density of 2.65 g/cm^3 . This should be considered a significant success of the HLS testing, indicating the amenability of pre-concentration of this material by DMS. The reduced mass to the grinding circuit is expected to result in a significant decrease in grinding circuit capital and operating costs;

- A high grade concentrate of 6.8% Li_2O was produced at a Li recovery of 56% by separating at a density of 3.00 g/cm^3 followed by low intensity magnetic separation. The high iron content of the concentrate, 0.9% Fe_2O_3, would render it unsuitable for the ceramic industry. It is believed that the majority of the iron occurs in the solid solution in the spodumene crystal, and further analysis is required in this regard;

- A concentrate grading 6.59% Li_2O was produced as the sink product at a density of 2.90 g/cm^3. The global recovery of the lithium in this product was 62%. Further processing with low intensity magnetic separation increased the concentrate grade to 6.61% Li_2O. Thus, magnetic separation did not increase the concentrate grade to any significant level;

- The middlings (float) product from the HLS tests at a density of 2.90 g/cm^3 combined with the minus 0.5 mm fines represents the flotation test feed material. This flotation feed contained 37% of the Li at a grade of 1.4% Li_2O representing 42% of the original feed mass. The grade of this flotation feed is similar to that of the original head sample.

In conclusion, in a multistage DMS processing operation with an initial feed size of -3/8" followed by secondary crushing to 10 mesh (2 mm), it is expected to:

- Discard about 42.8% of the mass as gangue before further grinding;
- Produce a spodumene concentrate with 6.6% Li assay or higher. About 62.1% of spodumene is expected to deport to this stream. The efficiency of the DMS process might be as high as 90% relative to the DMS feed. This efficiency is considered extremely high;
- Generate a flotation feed with a grade of 1.38% Li_2O of the original feed mass. about 42.4%;
- Have a very low Li loss in the DMS process of about 0.9% Li (global distribution).

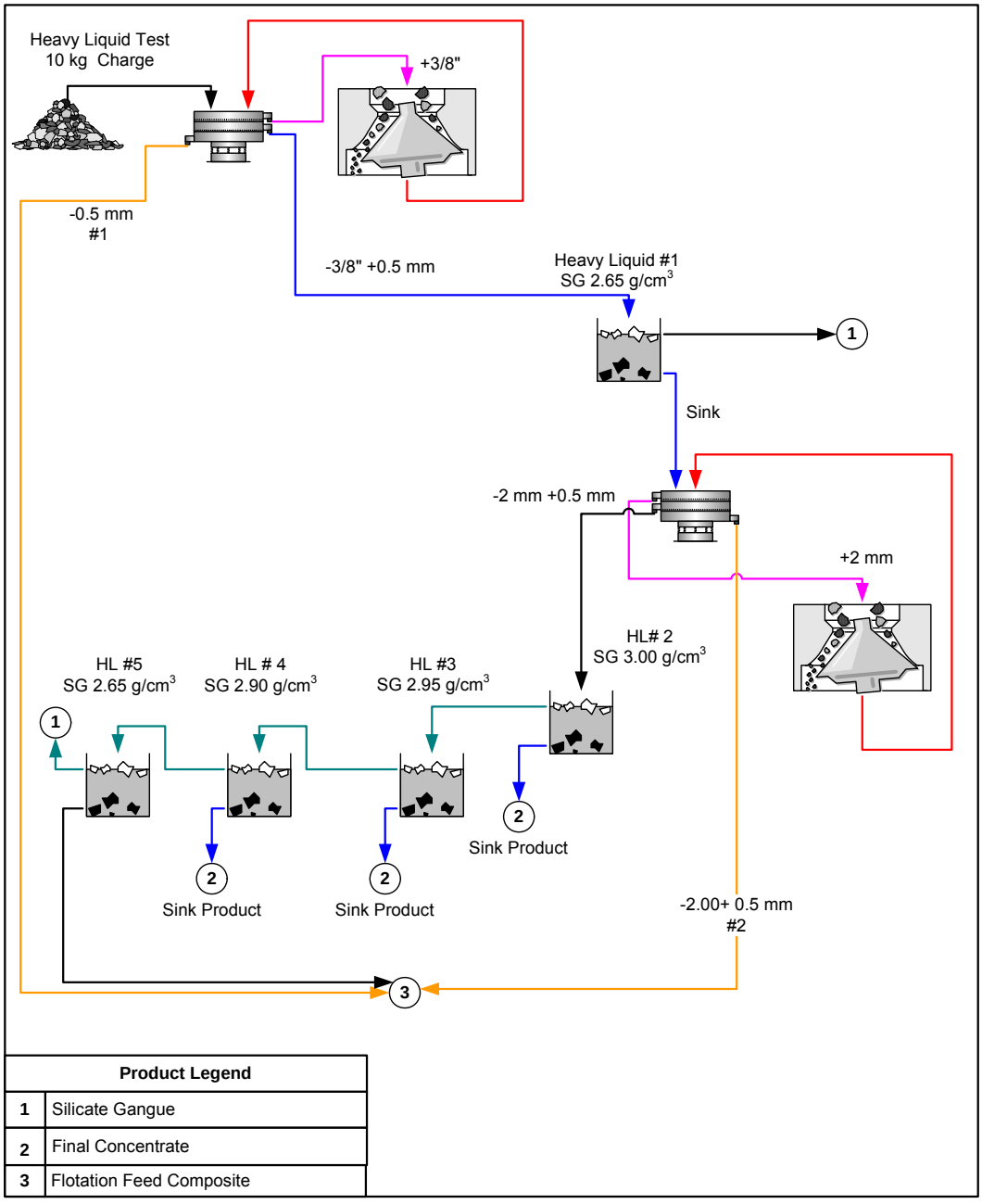


Figure 1: Flowsheet of HLS Test 1

Table 4: Heavy Liquid Test Results – Test 1

Products / Sample ID	Heavy Liquid Density g/cm³	Weight		Assays %								Distribution %						
		g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8" + 0.5mm Fraction, Float 2.65 SG	2.65	3,269	33.3	0.02	0.04	75.5	14.3	0.16	6.08	3.16	0.27	0.77	33.9	29.2	10.8	49.5	51.4	33.3
-2 + 0.5mm Fraction, Float 2.65 SG	2.65	934	9.50	0.01	0.03	80.5	11.9	0.15	6.47	0.82	0.22	0.16	10.3	6.95	2.89	15.0	3.81	7.76
Float 2.90 SG	2.90	1,425	14.5	0.32	0.69	76.8	14.6	0.76	2.32	2.65	0.28	6.33	15.0	13.0	22.3	8.23	18.8	15.1
Sink 2.90 SG, Non Mag	2.90	69.9	0.71	2.25	4.84	67.1	21.9	0.86	1.90	1.36	0.42	2.18	0.64	0.96	1.24	0.33	0.47	1.11
Sink 2.90 SG, Mag & Middlings	2.95	2.20	0.02	0.94	2.02	55.1	21.4	3.94	2.20	2.92	3.25	0.03	0.02	0.03	0.18	0.01	0.03	0.27
Sink 2.95 SG	2.95	107	1.08	2.44	5.25	65.8	23.1	0.93	1.45	1.19	0.42	3.61	0.96	1.54	2.04	0.38	0.63	1.69
Sink 3.00 SG, Non Mag	3.00	1,268	12.9	3.17	6.82	63.8	26.4	0.91	0.45	0.45	0.16	55.8	11.1	20.9	23.8	1.42	2.84	7.66
Sink 3.00 SG, Middlings	3.00	10.4	0.11	2.93	6.31	60.1	25.1	2.28	0.63	0.95	1.35	0.42	0.09	0.16	0.49	0.02	0.05	0.53
Sink 3.00 SG, Mag	3.00	5.00	0.05	0.94	2.02	42.9	16.9	12.4	1.08	0.96	2.50	0.07	0.03	0.05	1.28	0.01	0.02	0.47
2nd -0.5 mm Fraction		1,630	16.6	1.06	2.28	73.4	16.5	0.64	3.33	1.23	0.31	24.0	16.4	16.8	21.5	13.5	9.98	19.1
1st -0.5 mm Fraction		1,109	11.3	0.43	0.93	75.3	14.9	0.59	4.18	2.16	0.31	6.62	11.5	10.3	13.5	11.5	11.9	13.0
Global Feed (Calc.)		9,828	100	0.73	1.58	74.1	16.3	0.49	4.09	2.04	0.27	100	100	100	100	100	100	100

Products / Sample ID	Heavy Liquid Density g/cm³	Product		Assay %								Distribution %						
		wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Silicate Gangue (-3/8+ 0.5 mm Float 2.65 SG)	2.65	4,203	42.8	0.02	0.03	76.6	13.8	0.16	6.17	2.64	0.26	0.93	44.2	36.2	13.7	64.5	55.2	41.1
Final HL Spodumene Concentrate	2.90	1,462	14.9	3.06	6.59	64.0	25.9	0.96	0.60	0.56	0.21	62.1	12.8	23.7	29.0	2.18	4.05	11.7
Flotation Feed		4,164	42.4	0.64	1.38	75.1	15.4	0.67	3.21	1.96	0.30	36.9	42.9	40.2	57.3	33.3	40.7	47.2
Feed (Calc.)		9,828	100	0.73	1.58	74.1	16.3	0.49	4.09	2.04	0.27	100	100	100	100	100	100	100

Products / Sample ID	Heavy Liquid Density g/cm³	Product		Assay %								Distribution %						
		wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Final HL Spodumene Concentrate -Non Mag		1,444	14.7	3.07	6.61	64.1	25.9	0.91	0.59	0.55	0.19	61.6	12.7	23.4	27.1	2.14	3.94	10.5
Final HL Spodumene Concentrate -Mag		17.6	0.18	2.12	4.55	54.6	22.3	5.36	0.95	1.20	1.91	0.52	0.13	0.25	1.95	0.04	0.11	1.27
Final HL Spodumene Concentrate		1,462	14.9	3.06	6.59	64.0	25.9	0.96	0.60	0.56	0.21	62.1	12.8	23.7	29.0	2.18	4.05	11.7

Flotation Feed Components / Sample ID	Weight		Assay %								Distribution %						
	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	1,425	34.2	0.32	0.69	76.8	14.6	0.76	2.32	2.65	0.28	17.1	35.0	32	39	24.7	46	32
2nd -0.5 mm Fraction	1,630	39.1	1.06	2.28	73.4	16.5	0.64	3.33	1.23	0.31	64.9	38.3	42	38	40.6	25	40
1st -0.5 mm Fraction	1,109	26.6	0.43	0.93	75.3	14.9	0.59	4.18	2.16	0.31	17.9	26.7	26	24	34.7	29	28
Flotation Feed (Calc.)	4,164	100	0.64	1.38	75.1	15.4	0.67	3.21	1.96	0.30	100	100	100	100	100	100	100

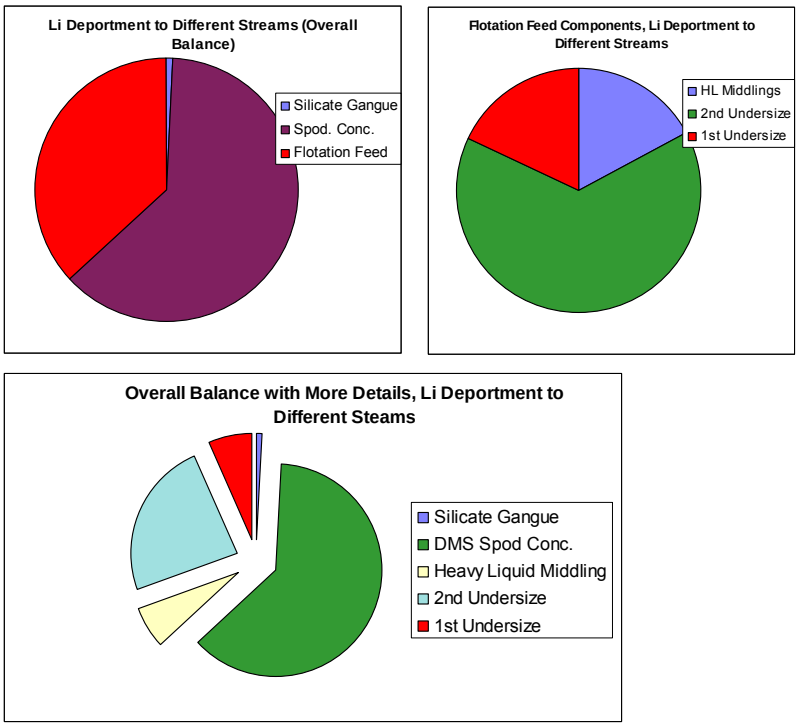


Figure 2: Plots of Li Deportment in Different Products- HLS Test 1

The results from the first HLS test clearly indicated that the major source of lithium deportment to the flotation feed is in the second undersize fraction that has been generated after secondary crushing. Thus, in a second heavy liquid test, it was decided to increase the crushing size to 6 mesh (3.35 mm). The flowsheet for this test is shown in Figure 3. This flowsheet was fairly similar to first heavy liquid test, with the major difference in secondary crushing size. The results of this test are shown in Table 5. Lithium deportment to different products are shown in Figure 4.

The following conclusions can be made from the 2nd heavy liquid test:

- By performing the primary crushing at -3/8" followed by secondary crushing to -6 mesh (3.35 mm), and separating -0.5 mm fraction, in the heavy liquid separation with the density of 2.65, about 43.6% of the head sample can be rejected with the lithium loss of 1.10% Li. Gangue silicates were floated after each crushing stage with the heavy liquid, SG of 2.65 g/cm³. It was very beneficial to repass the material through 2.65 SG heavy liquid after the second crushing since more silicate gangue minerals were liberated after second crushing and could be rejected. This should be considered as a significant success of the DMS operation. As a result, the grinding operating cost is expected to decrease significantly due to the large mass pull of the silicate gangues before entering to the grinding circuit;

- It was possible to produce a high grade concentrate of 6.85% Li_2O at a recovery of 61.3% using a heavy liquid with SG of 3.00 g/cm^3. Due to the absence of any significant amount of magnetic particles in the concentrate and failure of magnetic separation in improving the concentrate grade, it was decided to halt further magnetic separation testwork on the heavy liquid concentrates;

- A combined lithium concentrate with a grade of 6.21% Li_2O was produced as the sink product at a density of 2.85 g/cm^3. The global recovery of the Li in this product was about 75.5%. The heavy liquid efficiency (based on the HL feed) was 88.9%;

- The middling float product from HL (with SG of 2.85 g/cm^3) and the undersize material (-0.5 mm) are to be combined and further processed by flotation. This flotation feed contained about 23.4% of the lithium with the grade of 0.98% Li_2O and the mass fraction of 37.4% of the original feed. The grade of the flotation feed is close to the head sample.

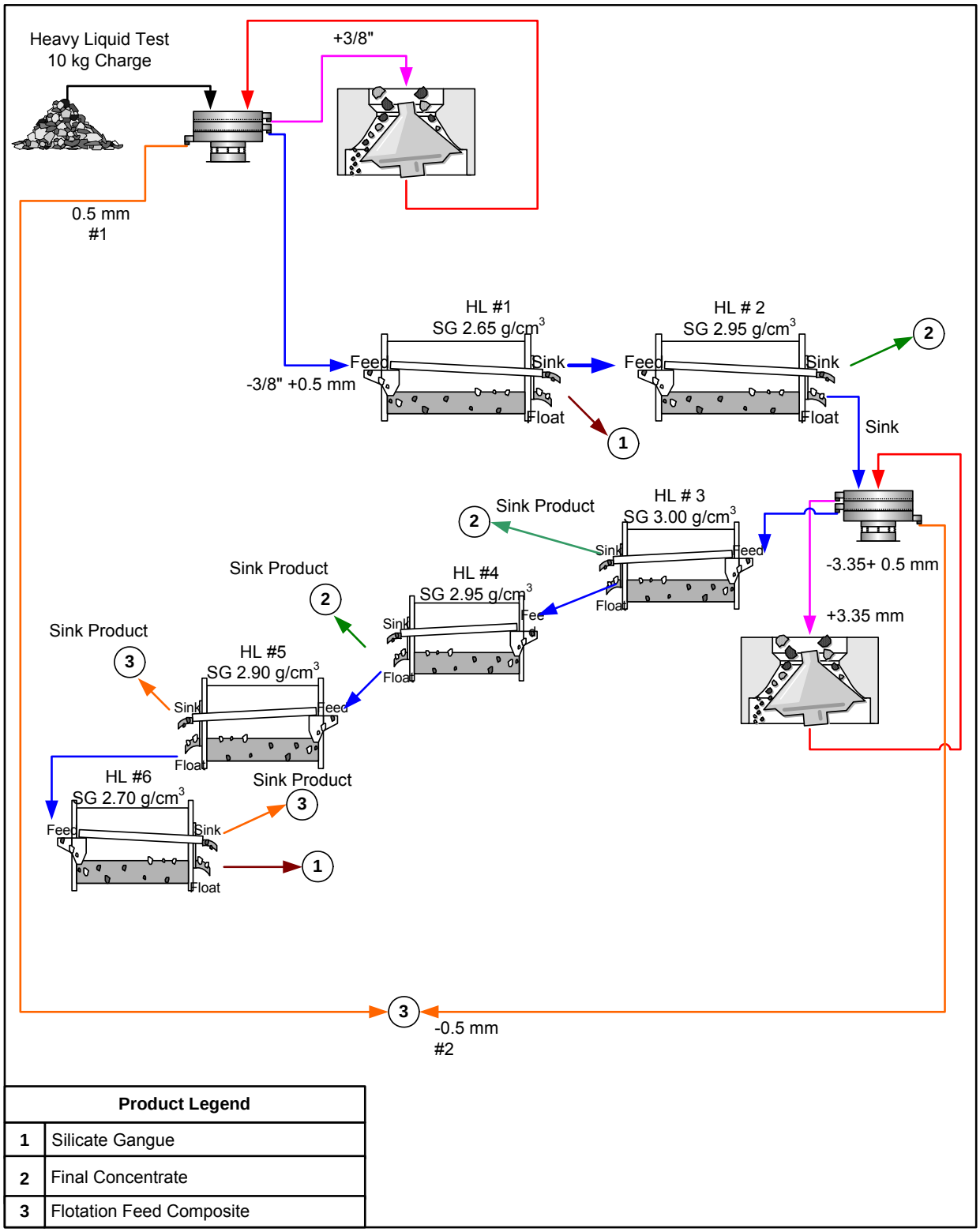
In conclusion, in a multistage DMS processing operation with the initial feed size of -3/8" followed by secondary crushing to 6 mesh (3.35 mm), it is expected to:

- Discard about 43.6% of the mass as gangue before further grinding;

- Produce a spodumene concentrate with 6.2% Li assay. About 75.5% of spodumene is expected to deport to this stream. The efficiency of the DMS might be as high as 89% based on the DMS feed. This efficiency is extremely high;

- Generate a flotation feed with the grade of 0.98% Li_2O with the mass fraction of about 37.4%;

- Have a very low Li loss in the DMS process of about 1.1% (global distribution).



Product Legend	
1	Silicate Gangue
2	Final Concentrate
3	Flotation Feed Composite

Figure 3: Flowsheet of HLS Test 2

Table 5: Heavy Liquid Test Results – Test 2

Products / Sample ID	Heavy Liquid Density g/cm³	Weight		Assays %								Distribution %						
		g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8" + 0.5mm, 2.65 SG Float	2.65	3,284	33.1	0.02	0.04	75.9	14.0	0.15	5.81	3.07	0.27	0.78	33.8	28.6	8.15	48.6	49.4	32.3
-3.35 + 0.5mm, 2.65 SG Float	2.65	1,048	10.6	0.02	0.05	82.1	11.0	0.27	5.71	0.87	0.22	0.32	11.7	7.18	4.68	15.2	4.47	8.41
Sink 2.65 SG	2.65	1,869	18.8	0.32	0.69	75.7	15.3	0.78	3.05	2.53	0.29	8.31	19.2	17.8	24.1	14.5	23.2	19.8
Sink 2.85 SG	2.85	127	1.28	1.57	3.38	68.5	20.1	0.94	2.10	1.68	0.43	2.76	1.18	1.59	1.97	0.68	1.04	1.99
Sink 2.90 SG	2.90	175	1.76	2.01	4.33	68.2	21.0	0.90	1.78	1.16	0.40	4.88	1.61	2.28	2.60	0.79	0.99	2.55
Sink 2.95 SG	2.95	195	1.96	2.42	5.21	66.7	22.6	0.91	1.30	0.96	0.32	6.56	1.76	2.74	2.94	0.65	0.92	2.28
Sink 3.00 SG	3.00	1,386	14.0	3.18	6.85	63.9	25.6	1.16	0.48	0.45	0.19	61.3	12.0	22.1	26.6	1.70	3.06	9.61
2nd -0.5 mm Fraction		700	7.05	0.85	1.83	73.6	15.8	1.61	3.38	1.35	0.41	8.27	6.98	6.89	18.6	6.03	4.63	10.5
1st -0.5mm Fraction		1,150	11.6	0.43	0.93	75.9	15.1	0.54	4.02	2.17	0.30	6.87	11.8	10.8	10.3	11.8	12.2	12.6
Global Feed (Calc.)		9,934	100	0.72	1.56	74.3	16.2	0.61	3.95	2.05	0.28	100	100	100	100	100	100	100

Products / Sample ID	Heavy Liquid Density g/cm³	Product		Assay %								Distribution %						
		wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	4,332	43.6	0.02	0.04	77.4	13.3	0.18	5.79	2.54	0.26	1.10	45.4	35.8	12.8	63.9	53.9	40.8
Final HL Spodumene Concentrate	2.85	1,883	19.0	2.88	6.21	64.9	24.5	1.10	0.79	0.65	0.24	75.5	16.6	28.7	34.1	3.81	6.02	16.4
Flotation Feed		3,719	37.4	0.45	0.98	75.4	15.3	0.86	3.41	2.20	0.32	23.4	38.0	35.5	53.0	32.3	40.1	42.8
Feed (Calc.)		9,934	100	0.72	1.56	74.3	16.2	0.61	3.95	2.05	0.28	100	100	100	100	100	100	100

Flotation Feed Components / Sample ID	Weight		Assay %								Distribution %						
	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	1,869	50.3	0.32	0.69	75.7	15.3	0.78	3.05	2.53	0.29	35.4	50.5	50.1	45.5	44.9	57.9	46.2
2nd -0.5 mm Fraction	700	18.8	0.85	1.83	73.6	15.8	1.61	3.38	1.35	0.41	35.3	18.4	19.4	35.2	18.6	11.6	24.4
1st -0.5mm Fraction	1,150	30.9	0.43	0.93	75.9	15.1	0.54	4.02	2.17	0.30	29.3	31.1	30.5	19.4	36.4	30.6	29.4
Flotation Feed (Calc.)	3,719	100	0.45	0.98	75.4	15.3	0.86	3.41	2.20	0.32	100	100	100	100	100	100	100

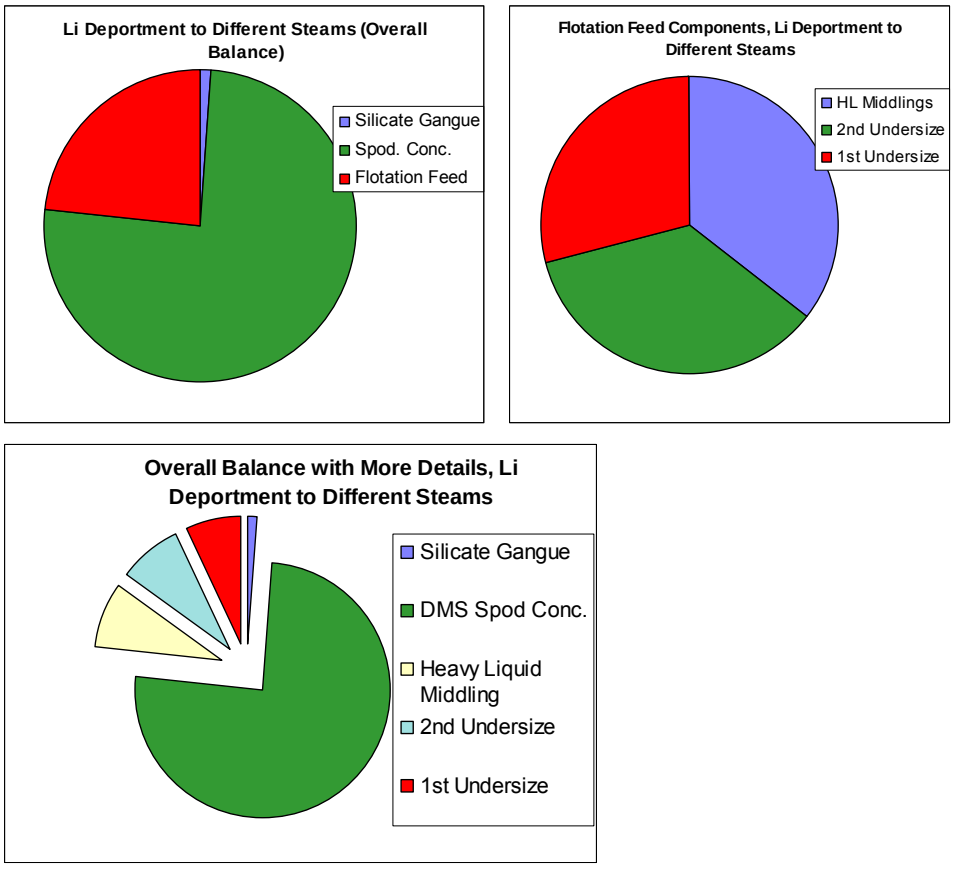


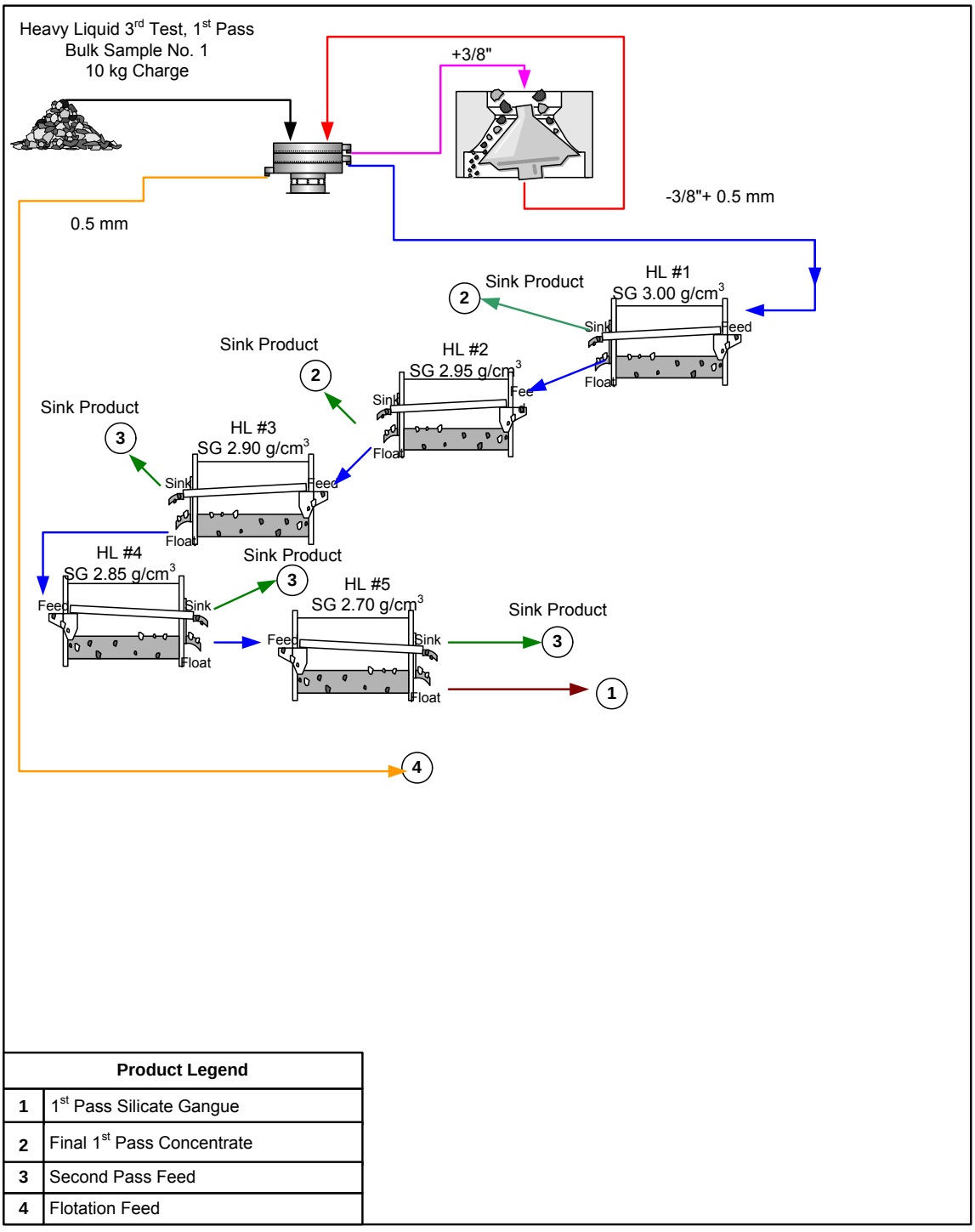
Figure 4: Plots of Li Deportment in Different Products- HLS Test 2

Due to the successful results of the second heavy liquid test, it was decided to perform another heavy liquid test without any secondary crushing as illustrated in Figure 5. The remaining tasks were performed in a similar manner to the second heavy liquid test. Once this test was completed, and due to the high lithium deportations to the sink products from heavy liquids with densities of 2.70 g/cm^3, 2.85 g/cm^3 and 2.90 g/cm^3, it was decided to further process these products as illustrated in Figure 6. These three streams were combined and further crushed to -6 mesh and then passed through a series of heavy liquids. The following results are illustrated in Table 6. Lithium deportations to different products are shown in Figure 7. The following conclusions can be made from this test:

- By performing the primary crushing at -3/8" in the first pass, and separating -0.5 mm fraction, in the heavy liquid separation test with the density of 2.70, about 44% of the feed sample mass can be rejected with the lithium loss of 4.37% Li in the 1st pass and a further 8% of the mass rejected at 0.71% Li loss in the second pass. It was possible to produce a high grade concentrate 6.18% Li_2O at a recovery of 46.5%. Thus, it is clear that secondary crushing is required to improve the lithium recovery;
- After further crushing the sink products from 2.70 and 2.85 g/cm^3 and 2.90 g/cm^3 heavy liquids and further processing it was possible to separate another 8.28% of the mass as silicate gangue at a lithium loss of 0.71%. A lithium concentrate was generated with a grade of 5.77% Li_2O and a Li recovery of 30.8%;
- In total, 77.3% of the lithium deported to the combined concentrates with a grade of 6.01% Li_2O. The cut point was 2.95 g/cm^3 in the 1st pass and 2.95 g/cm^3 in the 2nd pass;
- The middlings float product from the HLS tests and the undersize material (-0.5 mm fraction) are to be combined and further processed by flotation. This flotation feed contained about 17.7% of the lithium at a grade of 1.20% Li_2O , representing 25.5% of the original feed mass;
- By crushing the sample only to -3/8”, it would not be expected to achieve more than 46.5% lithium recovery in DMS operation. To increase the lithium recovery, it is required to further crush the ore sample to -6 mesh.

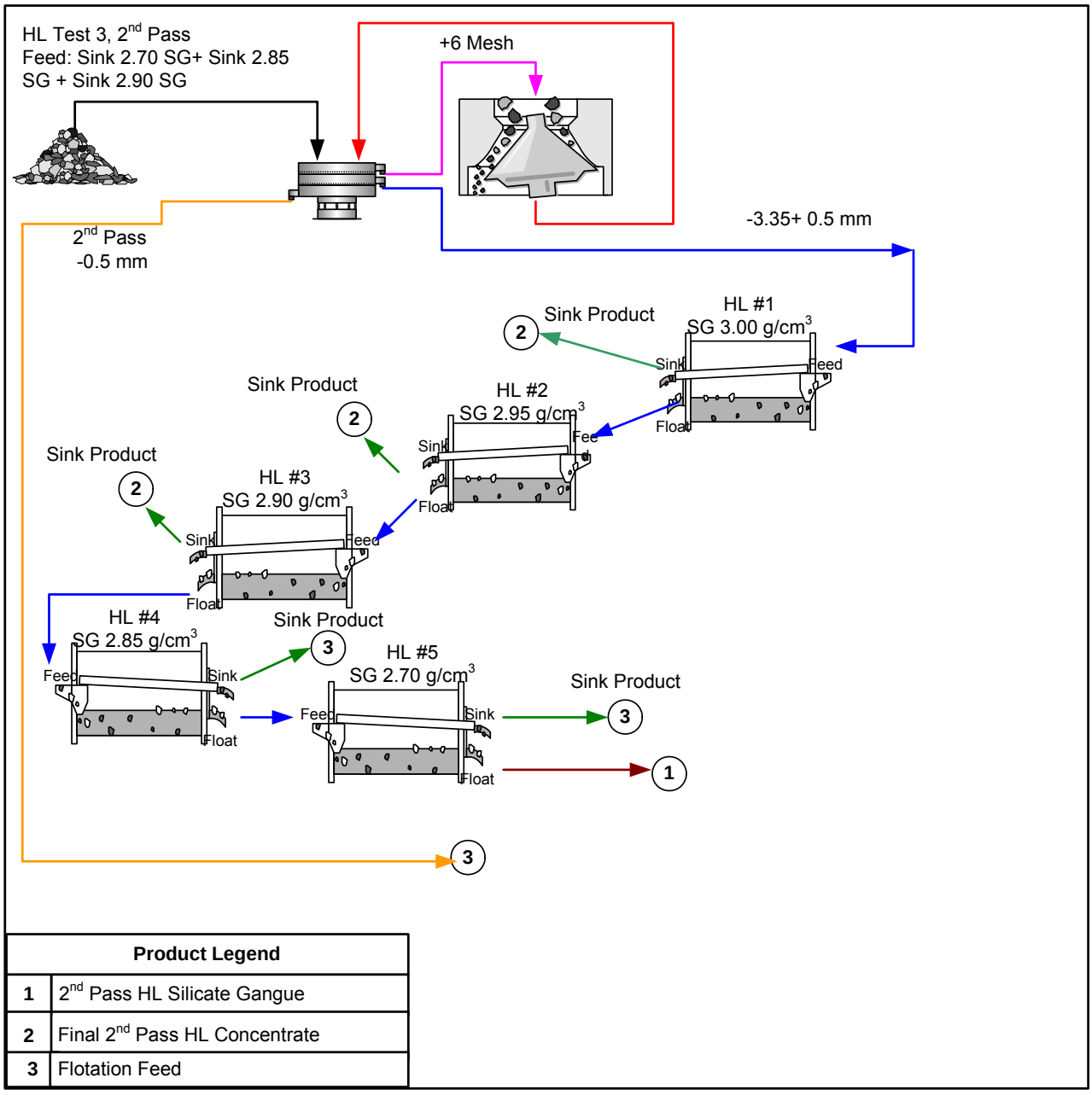
In conclusion, in a multistage DMS processing operation with the initial feed size of -3/8" followed by secondary crushing to 6 mesh (3.35 mm), it is expected to:

- Discard 52.3% of the mass as gangue before further crushing or grinding;
- Produce a spodumene concentrate with 6.0% Li_2O assay. About 77.3% of spodumene is expected to deport to this stream. The efficiency of the DMS might be as high as 79.8% relative to the DMS feed;
- Generate a flotation feed with a grade of 1.20% Li_2O with the mass fraction of about 25.5% of the original feed mass;
- Have a low Li loss in the DMS process of about 5.08% (global distribution).



Product Legend	
1	1st Pass Silicate Gangue
2	Final 1st Pass Concentrate
3	Second Pass Feed
4	Flotation Feed

Figure 5: Flowsheet of the HLS Test 3



Product Legend	
1	2nd Pass HL Silicate Gangue
2	Final 2nd Pass HL Concentrate
3	Flotation Feed

Figure 6: Flowsheet of the HLS Test 3b

Table 6: Heavy Liquid Test Results – Test 3

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Float 2.65 SG, 1st pass	2.65	3,374	34.2	0.03	0.07	76.1	14.2	0.20	6.05	2.73	0.27	1.45	35.2	28.9	14.1	50.8	48.0	31.7
Sink 2.65 SG, 1st pass	2.65	968	9.80	0.24	0.52	78.7	13.3	0.47	4.41	1.54	0.34	2.93	10.4	7.77	9.48	10.6	7.76	11.5
Float 2.65 SG, 2nd pass	2.65	483	4.89	0.02	0.04	81.7	11.4	0.14	5.97	0.69	0.22	0.12	5.40	3.32	1.41	7.17	1.73	3.69
Sink 2.65 SG, 2nd pass	2.65	336	3.40	0.14	0.30	84.3	9.69	0.39	3.40	0.93	0.22	0.59	3.87	1.96	2.73	2.84	1.62	2.57
Sink 2.70 SG, 2nd pass	2.70	715	7.24	0.75	1.61	71.0	19.1	0.79	3.01	2.57	0.39	6.75	6.95	8.24	11.8	5.36	9.56	9.70
Sink 2.85 SG, 2nd pass	2.85	166	1.68	1.64	3.53	69.7	20.4	0.79	2.21	1.57	0.40	3.42	1.58	2.04	2.73	0.91	1.35	2.31
Sink 2.90 SG, 2nd pass	2.90	205	2.08	1.90	4.09	69.7	21.4	0.76	1.79	1.12	0.37	4.91	1.96	2.65	3.25	0.91	1.20	2.64
Sink 2.95 SG, 2nd pass	2.95	158	1.60	2.48	5.34	68.0	23.1	0.76	1.39	0.99	0.32	4.93	1.47	2.20	2.50	0.55	0.81	1.76
Sink 3.00 SG, 2nd pass	3.00	549	5.56	3.03	6.52	65.2	25.7	0.85	0.62	0.56	0.19	21.0	4.90	8.52	9.73	0.85	1.60	3.63
Sink 2.95 SG, 1st pass	2.95	351	3.55	2.34	5.04	69.5	21.5	0.73	1.16	0.75	0.24	10.3	3.34	4.56	5.34	1.01	1.37	2.93
Sink 3.00 SG, 1st pass	3.00	934	9.46	3.07	6.61	65.0	25.0	0.85	0.61	0.50	0.17	36.1	8.32	14.1	16.5	1.42	2.43	5.53
2nd pass-0.5mm Spod. Out Crop 1		443	4.49	0.80	1.72	76.4	15.6	0.57	3.22	1.32	0.30	4.47	4.64	4.17	5.27	3.55	3.05	4.63
1st pass -0.5mm Spod. Out Crop 1		1,195	12.1	0.20	0.43	73.1	16.0	0.61	4.69	3.14	0.42	3.01	12.0	11.5	15.2	14.0	19.5	17.5
Global Feed (Calc.)		9,876	100	0.80	1.73	73.9	16.8	0.49	4.07	1.94	0.29	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.70, Silicate Gangue, 1st Pass	2.70	4,342	44.0	0.08	0.17	76.7	14.0	0.26	5.68	2.46	0.29	4.37	45.6	36.7	23.5	61.5	55.7	43.2
-3/8+ 0.5 mm Float 2.65, Silicate Gangue, 2nd Pass	2.70	818	8.28	0.07	0.15	82.8	10.7	0.24	4.92	0.79	0.22	0.71	9.27	5.28	4.14	10.0	3.36	6.26
Final HL Spod Conc. 1st Pass	2.95	1,285	13.0	2.87	6.18	66.2	24.0	0.82	0.76	0.57	0.19	46.5	11.7	18.7	21.9	2.43	3.80	8.46
Final HL Spod Conc. 2nd Pass	2.90	912	9.24	2.68	5.77	66.7	24.3	0.81	1.02	0.76	0.25	30.8	8.33	13.4	15.5	2.31	3.61	8.03
Combined HL Concentrates		2,197	22.2	2.79	6.01	66.4	24.1	0.82	0.87	0.65	0.22	77.3	20.0	32.0	37.4	4.74	7.41	16.5
Flotation Feed		2,518	25.5	0.56	1.20	72.9	17.1	0.67	3.79	2.55	0.39	17.7	25.1	26.0	35.0	23.8	33.5	34.1
Feed (Calc.)		9,876	122	0.80	1.73	73.9	16.8	0.49	4.07	1.94	0.29	100	100	100	100	100	100	100

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	880	8.92	0.92	1.97	70.8	19.3	0.79	2.86	2.38	0.39	10.2	8.53	10.3	14.5	6.27	10.9	12.0
1st pass -0.5mm Spod. Out Crop 1	1,195	12.1	0.20	0.43	73.1	16.0	0.61	4.69	3.14	0.42	3.01	12.0	11.5	15.2	14.0	19.5	17.5
2nd pass-0.5mm Spod. Out Crop 1	443	4.49	0.80	1.72	76.4	15.6	0.57	3.22	1.32	0.30	4.47	4.64	4.17	5.27	3.55	3.05	4.63
Flotation Feed (Calc.)	2,518	25.5	0.56	1.20	72.9	17.1	0.67	3.79	2.55	0.39	17.7	25.1	26.0	35.0	23.8	33.5	34.1

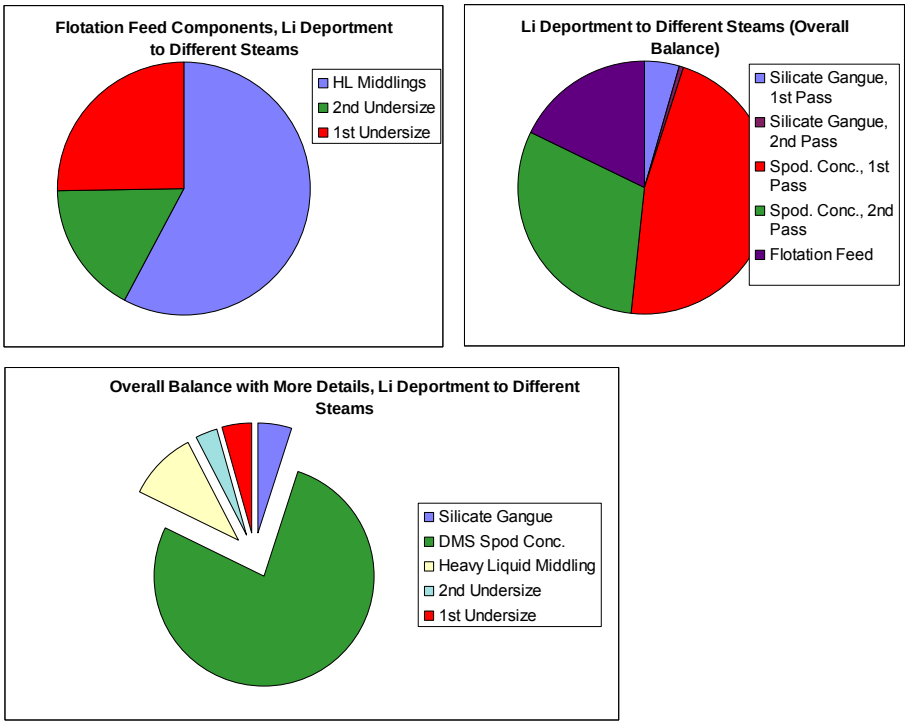


Figure 7: Plots of Li Deportment in Different Products- HLS Test 3

Clearly the results of all the HLS tests are very encouraging and suggest that there is high potential that DMS will be successful to produce high grade concentrate with a recovery of 70% or higher. In order to have more confidence on the performance of the heavy liquid tests, two variability tests were conducted. These tests were conducted on the low grade drill cores. As illustrated in Figure 8, the flowsheet was very similar to the second test. The combined results are shown in Table 7 and Figure 9. The following conclusions can be made from this test with the low feed grade.

Even with such a low grade feed, 0.75% Li_2O, it was possible to produce a high grade concentrate of 6.29% Li_2O at a Li recovery of 60.8% using a heavy liquid with a density of 3.00 g/cm^3. Higher recovery can probably be achieved by reducing the liquid density to 2.95 g/cm^3. By performing the primary crushing at -3/8" and secondary crushing to -6 mesh (Figure 8) and separating -0.5 mm fraction, in the heavy liquid separation test with the density of 2.65 g/cm^3, 68.2% of the mass of the head sample can be rejected with the lithium loss of 4.55% Li.

The middling float product from HLS tests and the undersize material (-0.5 mm fraction) are to be combined and further processed by flotation. This flotation feed contained about 34.6% of the lithium with the grade of 1.06% Li_2O and the mass fraction of 24.6% of the original feed. It is obvious that the original feed grade, 0.75% Li_2O, has been upgraded by the heavy liquid.

In conclusion, in a multistage DMS processing operation with the initial feed size of -3/8" followed by secondary crushing to 6 mesh (3.35 mm), it is expected to:

- Discard 68.2% of the mass as gangue before further crushing or grinding;
- Produce a spodumene concentrate with 6.29% Li assay. About 60.8% of spodumene is expected to deport to this stream. The efficiency of the DMS might be as high as 77.7% based on the DMS feed;
- Generate a flotation feed with the grade of 1.06% Li_2O with the mass fraction of about 24.6%;
- Have a low Li loss in the DMS process of about 4.55% (global distribution).

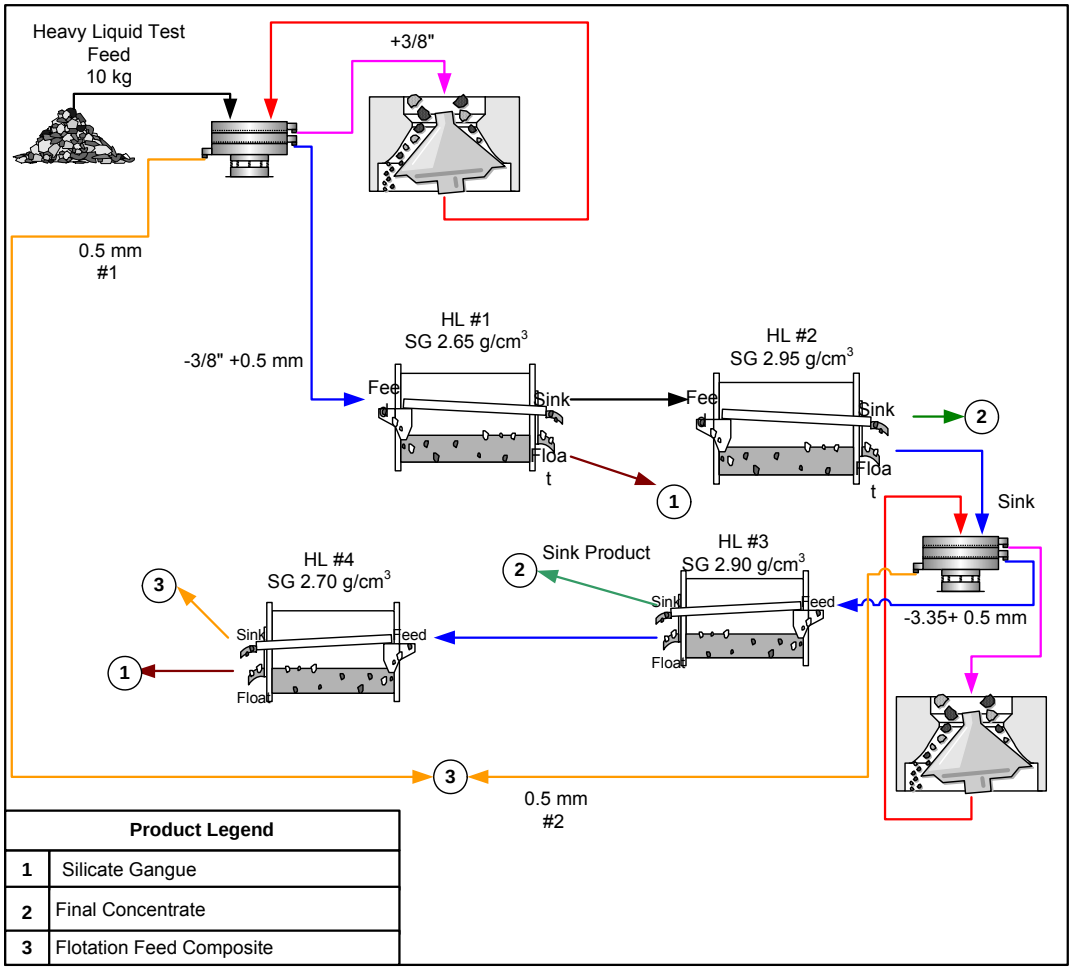


Figure 8: Flowsheet of HLS Test 4

Table 7: Heavy Liquid Test Results, Test 4

Products / Sample ID	Heavy Liquid Density g/cm³	Weight		Assays %								Distribution %						
		g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8"+0.5 mm, HLS Float 2.65 SG	2.65	4,424	48.8	0.02	0.03	74.6	14.2	0.20	6.06	3.01	0.27	2.10	49.1	44.5	21.3	61.8	57.9	47.6
-6M +0.5 mm, HL Float 2.65 SG	2.65	608	6.72	0.01	0.03	77.2	13.9	0.17	6.96	1.19	0.26	0.27	6.99	6.00	2.48	9.76	3.15	6.31
HL Sink 2.65 SG	2.65	1,143	12.6	0.06	0.13	82.8	10.5	0.43	3.47	1.47	0.22	2.17	14.1	8.51	11.8	9.14	7.31	10.0
HL Sink 2.70 SG	2.70	686	7.58	0.42	0.90	63.1	23.7	1.38	2.44	5.13	0.36	9.14	6.44	11.5	22.7	3.86	15.3	9.86
HL Sink 2.85 SG	2.85	85.8	0.95	1.37	2.95	66.2	22.5	1.20	1.95	2.89	0.38	3.73	0.84	1.37	2.47	0.39	1.08	1.30
HL Sink 2.90 SG	2.90	74.2	0.82	1.99	4.28	68.0	22.0	1.03	1.71	1.61	0.34	4.68	0.75	1.16	1.84	0.29	0.52	1.01
HL Sink 2.95 SG	2.95	81.4	0.90	2.37	5.10	66.6	23.3	1.03	1.31	1.30	0.31	6.11	0.81	1.34	2.01	0.25	0.46	1.01
HL Sink 3.00 SG	3.00	501	5.53	3.15	6.78	64.0	26.1	1.15	0.49	0.65	0.21	50.0	4.77	9.27	13.8	0.57	1.42	4.20
-0.5mm 1st Pass		845	9.33	0.41	0.88	76.2	15.0	0.50	4.45	1.96	0.33	11.0	9.58	8.99	10.1	8.67	7.20	11.1
-0.5mm 2nd Pass		608	6.71	0.56	1.21	73.5	16.9	0.78	3.76	2.13	0.31	10.8	6.65	7.29	11.4	5.27	5.63	7.52
Global Feed (Calc.)		9,057	100	0.35	0.75	74.2	15.6	0.46	4.79	2.54	0.28	100	100	100	100	100	100	100

Products / Sample ID	Heavy Liquid Density g/cm³	Product		Assay %								Distribution %						
		wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	6,175	68.2	0.02	0.05	76.4	13.5	0.24	5.67	2.55	0.26	4.55	70.2	59.0	35.5	80.7	68.4	64.0
Final HL Spodumene Concentrate	2.90	657	7.25	2.92	6.29	64.8	25.3	1.12	0.73	0.84	0.24	60.8	6.33	11.8	17.7	1.10	2.40	6.21
Flotation Feed		2,225	24.6	0.49	1.06	71.0	18.5	0.87	3.55	3.02	0.34	34.6	23.5	29.2	46.8	18.2	29.2	29.8
Feed (Calc.)		9,057	100	0.35	0.75	74.2	15.6	0.46	4.79	2.54	0.28	100	100	100	100	100	100	100

Flotation Feed Components / Sample ID	Weight		Assay %								Distribution %						
	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
HL Sink 2.70 SG	686	30.8	0.42	0.90	63.1	23.7	1.38	2.44	5.13	0.36	26.4	27.4	39.5	48.6	21.2	52.4	33.1
HL Sink 2.85 SG	85.8	3.86	1.37	2.95	66.2	22.5	1.20	1.95	2.89	0.38	10.8	3.59	4.69	5.29	2.12	3.69	4.36
-0.5mm 1st Pass	845	38.0	0.41	0.88	76.2	15.0	0.50	4.45	1.96	0.33	31.7	40.7	30.8	21.7	47.7	24.6	37.3
-0.5mm 2nd Pass	608	27.3	0.56	1.21	73.5	16.9	0.78	3.76	2.13	0.31	31.2	28.3	25.0	24.4	29.0	19.3	25.2
Flotation Feed (Calc.)	2,225	100	0.49	1.06	71.0	18.5	0.87	3.55	3.02	0.34	100	100	100	100	100	100	100

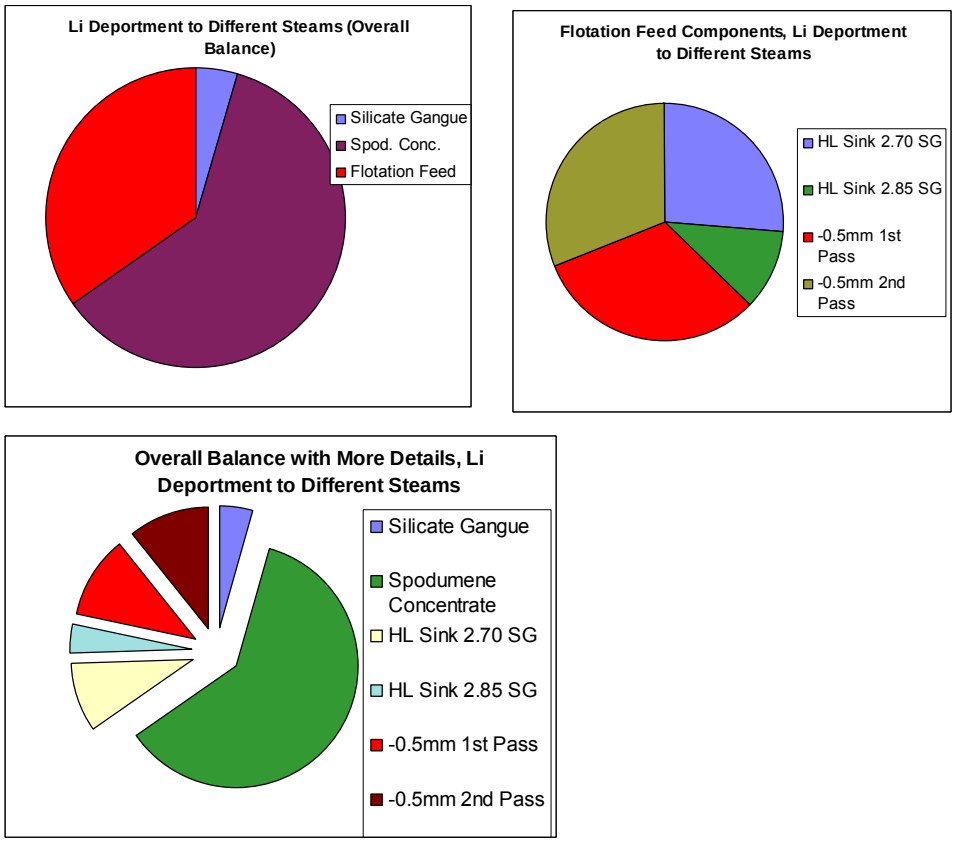


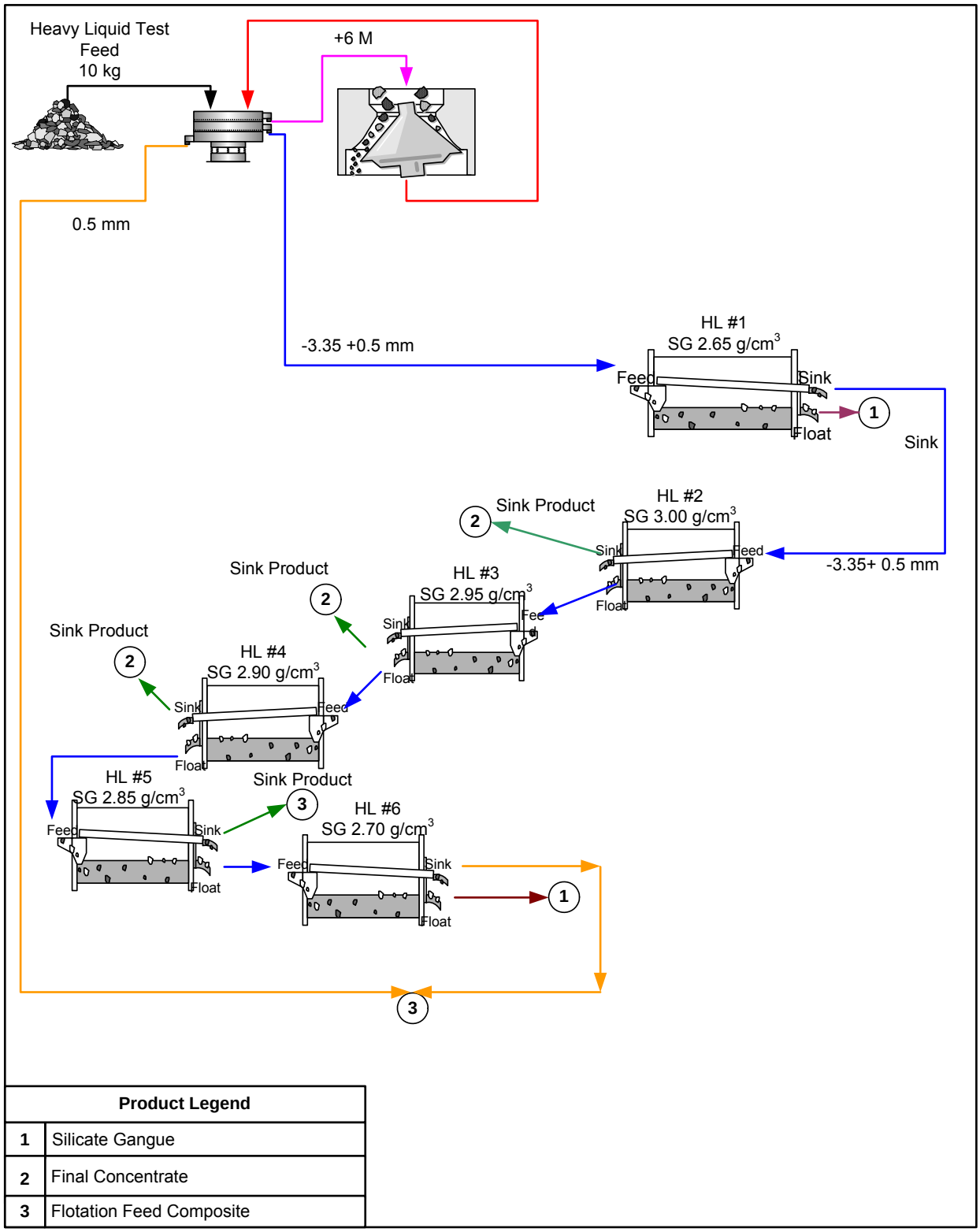
Figure 9: Plots of Li Deportment in Different Products- HLS Test 4

The last heavy liquid test was conducted on the second outcrop sample. The objective of this test was to perform another variability test with the higher feed grade, 1.96% Li_2O and to identify the recovery of one stage of crushing. The test flowsheet is illustrated and the results are illustrated in Table 8 and Figure 11. The following conclusions were made from this test:

- By performing only one crushing stage at -6 mesh, and separating -0.5 mm fraction, and using a heavy liquid with a density of 2.70 g/cm^3, it was possible to reject 45.8% of the mass with the lithium loss of 2.27% Li;
- By further processing the sample with the heavy liquid with SG of 2.90 g/cm^3, it was possible to produce a high grade concentrate, 6.34% Li_2O, with the recovery of 73.6.%.;
- The middling float product from HLS with SG of 2.90 g/cm^3, and the undersize material -0.5 mm are to be combined and further processed by flotation. This flotation feed contained about 24.1% of the lithium with the grade of 1.50% Li_2O and the mass fraction of 31.4%. The grade of the flotation feed is less than the sample head grade.

In conclusion, in a multistage DMS processing operation with the primary crushing to 6 mesh (3.35 mm), it is expected:

- To reject about 45.8% of the mass as gangue before any further grinding;
- To produce a spodumene concentrate with a grade of 6.34% Li. About 73.6% of spodumene is expected to deport to this stream. The efficiency of the DMS might be as high as 84.3% based on the DMS feed;
- To generate a flotation feed with the grade of 1.50% Li_2O with the mass fraction of about 31.4%;
- Have a very low Li loss in the DMS process of about 2.27% (global distribution).



Product Legend	
1	Silicate Gangue
2	Final Concentrate
3	Flotation Feed Composite

Figure 10: Flowsheet of HLS Test 5

Table 8: Heavy Liquid Test Results, Test 5

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Float 2.65 SG	2.65	2,718	35.7	0.02	0.05	78.6	12.7	0.17	5.78	2.16	0.22	0.82	37.7	27.2	10.8	57.4	46.3	31.5
Sink 2.65 SG	2.65	769	10.1	0.13	0.28	84.7	9.16	0.32	3.12	1.00	0.25	1.44	11.5	5.54	5.77	8.76	6.06	10.1
Sink 2.70 SG	2.70	855	11.2	0.65	1.40	68.5	20.1	0.98	2.94	3.19	0.36	8.02	10.3	13.5	19.6	9.18	21.5	16.2
Sink 2.85 SG	2.85	140	1.84	1.67	3.59	69.5	20.7	0.89	2.06	1.60	0.41	3.38	1.72	2.28	2.92	1.05	1.77	3.02
Sink 2.90 SG	2.90	149	1.96	2.11	4.54	69.2	20.9	0.84	1.69	0.87	0.33	4.55	1.83	2.46	2.94	0.92	1.02	2.60
Sink 2.95 SG	2.95	157	2.06	2.45	5.27	68.3	22.5	0.88	1.25	0.68	0.31	5.55	1.89	2.78	3.24	0.72	0.84	2.56
Sink 3.00 SG	3.00	1,424	18.7	3.09	6.65	64.7	26.2	1.00	0.50	0.35	0.16	63.5	16.3	29.4	33.4	2.60	3.93	12.0
HL Test 5 -35M Fraction		1,395	18.3	0.63	1.36	76.1	15.4	0.65	3.81	1.69	0.30	12.7	18.7	16.9	21.3	19.4	18.6	22.0
Global Feed (Calc.)		7,606	100	0.91	1.96	74.5	16.7	0.56	3.60	1.67	0.25	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.70	3,486	45.8	0.05	0.10	79.9	11.9	0.20	5.19	1.90	0.23	2.27	49.2	32.7	16.6	66.1	52.3	41.6
Final HL Spodumene Concentrate	2.90	1,730	22.7	2.95	6.34	65.4	25.4	0.98	0.67	0.42	0.19	73.6	20.0	34.6	39.6	4.24	5.80	17.2
Flotation Feed		2,390	31.4	0.70	1.50	73.0	17.4	0.78	3.40	2.22	0.33	24.1	30.8	32.7	43.8	29.6	41.9	41.3
Feed (Calc.)		7,606	100	0.91	1.96	74.5	16.7	0.56	3.60	1.67	0.25	100	100	100	100	100	100	100

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	995	13.1	0.79	1.71	68.6	20.2	0.97	2.82	2.97	0.37	11.4	12.1	15.8	22.6	10.2	23.3	19.2
HL Test 5 -35M Fraction	1,395	18.3	0.63	1.36	76.1	15.4	0.65	3.81	1.69	0.30	12.7	18.7	16.9	21.3	19.4	18.6	22.0
Flotation Feed (Calc.)	2,390	31.4	0.70	1.50	73.0	17.4	0.78	3.40	2.22	0.33	24.1	30.8	32.7	43.8	29.6	41.9	41.3

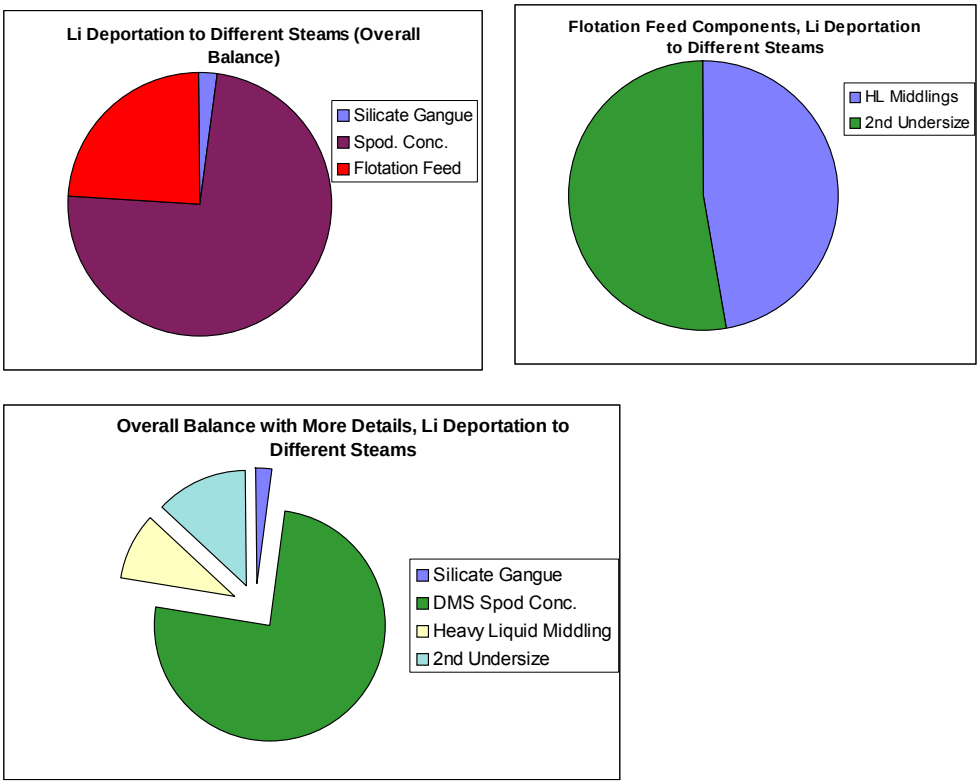


Figure 11: Plots of Li Deportment in Different Products- HLS Test 5

All the heavy liquid test results are summarized in Table 9. The head grade appears to play an important role in the amount of mass rejected as gangue steam.

Table 9: Summary of the Heavy Liquid Separation Test Results

Heavy Liquid Test No.	Products Sample ID	Heavy Liquid Density g/cm³	Product		Assay %								Distribution %						
			wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
1	Silicate Gangue (-3/8+ 0.5 mm Float 2.65 SG)	2.65	4,203	42.8	0.02	0.03	76.6	13.8	0.16	6.17	2.64	0.26	0.93	44.2	36.2	13.7	64.5	55.2	41.1
	Final HL Spodumene Concentrate	3.00	1,462	14.9	3.06	6.59	64.0	25.9	0.96	0.60	0.56	0.21	62.1	12.8	23.7	29.0	2.18	4.05	11.7
	Flotation Feed		4,164	42.4	0.64	1.38	75.1	15.4	0.67	3.21	1.96	0.30	36.9	42.9	40.2	57.3	33.3	40.7	47.2
	Feed (Calc.)		9,828	100	0.73	1.58	74.1	16.3	0.49	4.09	2.04	0.27	100	100	100	100	100	100	100
2	-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	4,332	43.6	0.02	0.04	77.4	13.3	0.18	5.79	2.54	0.26	1.10	45.4	35.8	12.8	63.9	53.9	40.8
	Final HL Spodumene Concentrate	3.00	1,883	19.0	2.88	6.21	64.9	24.5	1.10	0.79	0.65	0.24	75.5	16.6	28.7	34.1	3.81	6.02	16.4
	Flotation Feed		3,719	37.4	0.45	0.98	75.4	15.3	0.86	3.41	2.20	0.32	23.4	38.0	35.5	53.0	32.3	40.1	42.8
	Feed (Calc.)		9,934	100	0.72	1.56	74.3	16.2	0.61	3.95	2.05	0.28	100	100	100	100	100	100	100
3	-3/8+ 0.5 mm Float 2.70, Silicate Gangue, 1st Pass	2.70	4,342	44.0	0.08	0.17	76.7	14.0	0.26	5.68	2.46	0.29	4.37	45.6	36.7	23.5	61.5	55.7	43.2
	-3/8+ 0.5 mm Float 2.65, Silicate Gangue, 2nd Pass	2.70	818	8.28	0.07	0.15	82.8	10.7	0.24	4.92	0.79	0.22	0.71	9.27	5.28	4.14	10.0	3.36	6.26
	Final HL Spod Conc. 1st Pass	2.95	1,285	13.0	2.87	6.18	66.2	24.0	0.82	0.76	0.57	0.19	46.5	11.7	18.7	21.9	2.43	3.80	8.46
	Final HL Spod Conc. 2nd Pass	2.90	912	9.24	2.68	5.77	66.7	24.3	0.81	1.02	0.76	0.25	30.8	8.33	13.4	15.5	2.31	3.61	8.03
	Flotation Feed		2,518	25.5	0.56	1.20	72.9	17.1	0.67	3.79	2.55	0.39	17.7	25.1	26.0	35.0	23.8	33.5	34.1
	Feed (Calc.)		9,876	100	0.80	1.73	73.9	16.8	0.49	4.07	1.94	0.29	100	100	100	100	100	100	100
4	-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	6,175	68.2	0.02	0.05	76.4	13.5	0.24	5.67	2.55	0.26	4.55	70.2	59.0	35.5	80.7	68.4	64.0
	Final HL Spodumene Concentrate	3.00	657	7.25	2.92	6.29	64.8	25.3	1.12	0.73	0.84	0.24	60.8	6.33	11.8	17.7	1.10	2.40	6.21
	Flotation Feed		2,225	24.6	0.49	1.06	71.0	18.5	0.87	3.55	3.02	0.34	34.6	23.5	29.2	46.8	18.2	29.2	29.8
	Feed (Calc.)		9,057	100	0.35	0.75	74.2	15.6	0.46	4.79	2.54	0.28	100	100	100	100	100	100	100
5	-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.70	3,486	45.8	0.05	0.10	79.9	11.9	0.20	5.19	1.90	0.23	2.27	49.2	32.7	16.6	66.1	52.3	41.6
	Final HL Spodumene Concentrate	2.90	1,730	22.7	2.95	6.34	65.4	25.4	0.98	0.67	0.42	0.19	73.6	20.0	34.6	39.6	4.24	5.80	17.2
	Flotation Feed		2,390	31.4	0.70	1.50	73.0	17.4	0.78	3.40	2.22	0.33	24.1	30.8	32.7	43.8	29.6	41.9	41.3
	Feed (Calc.)		7,606	100	0.91	1.96	74.5	16.7	0.56	3.60	1.67	0.25	100	100	100	100	100	100	100

4. Flotation Test Results

As it was stated before, the composite sample mainly contained spodumene (19%), quartz (32.4%), albite (34.4%), microcline (7.1%), and muscovite (7.2%). Flotation is one of the main methods to produce spodumene concentrate.

All of the flotation batch tests were conducted either in a Denver D12 or a 10 kg flotation cell. All the stage grinding was performed in a 10 kg mill. Steel rods were used as the grinding media. After stage grinding in the 10 kg mill with the closing screen size of 300 microns (in most cases), a 50 mm Mozley cyclone with 4.5 mm apex and 11.5 mm vortex finder operating at 20 psi was used to remove the primary slimes. Scrubbing was performed in a Denver flotation cell at pH around 11 and in the presence of lignin sulfonate for 10 minutes. Slime separation after scrubbing was performed by settling and decanting in an acrylic container. All the equipment used in this testwork is illustrated in Figure 12.

Flotation tests have shown that the Georgia deposit ore sample is amenable to flotation beneficiation to produce concentrate for the hydrometallurgical operation. Higher grade concentrate (above 6.5% Li_2O) can be produced but the iron content of spodumene, mainly in the solid solution, was fairly high (about 1%). Flotation tests were mainly designed to generate spodumene concentrate with higher than 6.0%

Li_2O grades and above 80% Li recovery. Production of a bulk feldspar concentrate or separate Na and K feldspar concentrates was not examined. All the flotation test data are presented in Appendix D.

4.1. Grinding and Desliming

Based on past experience and expected liberation size, stage grinding (100% passing 48 mesh) was performed on 10 kg charges as follows:

- The feed was primary screened at 48 mesh with the over size used as feed for the next stage;
- After each grinding stage the product was screened on a 48 mesh (300 micron) screen;
- The pulp density in each grinding stage was roughly adjusted to ~60% solids;
- The first grinding stage was performed for 10 minutes and thereafter the grinding time was adjusted to 10 minutes and then another 10 minutes in a 10 kg mill. The last over size fraction was ground in a 2 kg mill for another 10 minutes;
- The combined undersize was deslimed using a 50 mm Mozley cyclone with a 4.5 mm apex at 20 psi;
- The cyclone underflow was split into 5 charges, for the subsequent flotation tests in the 2 or 4 kg flotation cell or kept as one 10 kg charge in a 10 kilogram flotation cell;
- In all tests, the primary desliming was performed after grinding and by hydrocyclone. This slime is referred to as cyclone overflow (O/F).

The desliming results are illustrated in Table 10, Table 11 and Figure 13. From these results it can be concluded that:

- On average, 3.5% of the head sample mass reported to the cyclone overflow, which corresponds to a lithium loss of 2.4%. The cyclone overflow fineness (K_{80}) was less than 10 μm;
- For the charges prepared by combining the undersize fraction -35 mesh and the heavy liquid middling products, an average 3.3% of the mass reported to the cyclone overflow with a lithium loss of 1.9%;
- On average 3.3% of the lithium reported to the slimes collected ahead of mica flotation with a mass fraction of 4.5%. On average 1.1% of the lithium reported to the second slime collected after scrubbing in 1.4% of the mass;
- In total, 6.5% of lithium reported to the combined cyclone overflow and both slimes collected before mica flotation and after scrubbing, with a mass fraction of 9.0%. As indicated in Figure 13, there is a linear relation between the total mass fraction of combined slimes and the lithium loss. The grinding procedure for lithium flotation was stage grinding with a closing size of 300 μm (48 mesh) and a K_{80} of about 200 to 230 μm.

A: 10 kg Grinding Mill

B: 2 kg Grinding Mill

C: Scrubbing in a Denver Flotation Cell

D: Denver Flotation Cell

E: Mozley Hydrocyclone

F: 10 kg Flotation Cell

Figure 12: Lab Equipment Used for Flotation Testwork

Table 10: Cyclone Overflow Analysis

Grinding Test No.	Product	Weight %	Assays %							Distribution %					
			Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
1	Cyclone O/F	3.3	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.17	2.96	3.79	5.26	3.49	8.98
2	Cyclone O/F	3.5	0.56	1.21	68.0	17.4	3.65	3.53	1.82	2.83	3.16	3.74	6.17	1.55	10.21
3	Cyclone O/F	4.4	0.45	.96	68.2	18.8	3.33	4.42	1.54	2.71	4.03	5.07	7.31	5.08	12.08
5	Cyclone O/F	3.0	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.32	2.75	3.56	4.79	3.15	6.36
8	Cyclone O/F	3.3	0.42	0.90	65.6	19.4	3.60	4.21	1.77	2.00	2.95	4.02	5.84	3.45	9.86
	Average	3.5	0.49	1.05	67.3	18.6	3.43	4.12	1.71	2.41	3.17	4.04	5.88	3.34	9.50

Grinding Test No.	Product	Weight %	Assays %							Distribution %					
			Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
4	F-12 HL Test 1&2	3.55	0.37	0.80	67.7	18.3	3.29	3.45	2.09	2.39	3.17	4.24	5.73	3.75	9.58
7	Mixed HL Sample Cyclone O/F	3.11	0.27	0.58	62.4	21.7	4.38	3.71	2.03	1.47	2.66	3.92	5.58	3.19	7.59
	Average	3.33	0.32	0.69	65.1	20.0	3.84	3.58	2.06	1.93	2.91	4.08	5.66	3.47	8.58

Grinding Test No.	Product	Weight %	Assays %							Distribution %					
			Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
6	Fine Grinding-Cyclone O/F	8.83	0.47	1.01	68.0	18.2	3.16	4.29	1.84	6.08	8.12	9.85	13.4	9.66	17.5

Table 11: 1st Slime Assays

Test No.	Product	Weight %	Assays %							Distribution %					
			Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F1	F1 Slimes #1	5.3	0.54	1.16	71.9	16.9	2.62	4.39	1.09	3.74	5.10	5.53	6.80	5.86	9.20
F2	F2 Slimes #1	5.5	0.55	1.18	71.4	16.9	2.69	4.34	1.08	4.15	5.34	5.82	7.25	6.15	9.59
F3	F3 Slimes #1	3.8	0.52	1.12	71.6	17.0	2.66	4.23	1.12	2.68	3.71	4.02	5.21	4.29	6.34
F4	F4 Slimes #1	5.9	0.55	1.18	71.6	17.0	2.60	4.22	1.04	4.53	5.68	6.21	7.66	6.50	9.12
F5	F5 Slimes #1	4.7	0.54	1.16	71.6	16.8	2.65	4.35	1.06	3.45	4.50	4.90	6.17	5.22	7.42
F6	F6 Slimes #1	4.8	0.52	1.12	71.4	17.1	2.63	4.19	1.09	3.47	4.63	5.08	6.41	5.35	8.10
F7	F7 Slimes #1	1.8	0.59	1.27	69.8	17.6	2.75	4.02	1.35	1.36	1.64	1.91	2.39	1.81	3.48
F8	F8 Slimes #1	3.5	0.46	0.99	71.8	16.9	2.62	4.42	0.88	2.33	3.35	3.64	4.44	1.94	4.95
F9	F9 Slimes #1	4.4	0.53	1.14	71.7	17.1	2.67	4.21	1.04	3.21	4.23	4.61	5.85	4.83	8.16
F10	F10 Slimes #1	3.6	0.54	1.16	71.2	17.3	2.73	4.14	1.11	2.63	3.44	3.81	4.89	3.93	6.75
F11	F11 Slimes #1	4.2	0.42	0.90	73.8	15.6	2.32	3.59	1.24	3.19	4.12	4.30	4.82	4.55	6.23
F12	F12 Slimes #1	3.7	0.41	0.88	74.0	15.7	2.32	3.51	1.21	2.77	3.62	3.80	4.22	3.99	5.80
F13	F13 Slimes #1	5.0	0.51	1.10	71.9	16.8	2.61	4.45	0.98	3.86	4.85	5.28	6.31	5.51	6.00
F14	F14 Slimes #1	8.8	0.51	1.10	72.6	16.6	2.53	4.22	1.07	6.56	8.62	8.93	10.6	9.45	10.1
F15	F15 Slimes #1	8.2	0.53	1.14	73.3	16.7	2.54	4.22	1.12	6.35	8.13	8.46	9.91	8.82	11.1
F16	F16 Slimes #1	4.0	0.54	1.16	70.6	17.1	2.70	4.20	1.28	2.96	3.81	4.21	5.38	4.37	8.67
F17	F17 Slimes #1	1.3	0.48	1.03	71.2	16.9	2.72	4.54	1.01	0.84	1.21	1.31	1.65	1.44	2.01
F18	F18 Slimes #1	4.1	0.50	1.08	70.9	16.9	2.71	4.47	1.04	2.85	3.96	4.32	5.51	4.64	6.74
F19	F19 Slimes #1	3.9	0.48	1.03	70.4	16.9	2.70	4.46	1.04	2.70	3.73	4.12	5.21	4.35	6.66
F20	F20 Slimes #1	4.3	0.49	1.05	71.0	17.3	2.80	4.54	1.07	3.00	4.16	4.69	6.02	4.87	8.17
F21	F21 Slimes #1	4.2	0.36	0.77	68.1	18.9	3.36	4.08	1.34	2.62	3.90	4.57	5.74	4.70	6.71
F22	F22 Slimes #1	4.5	0.50	1.08	70.9	17.2	2.81	4.49	1.08	3.23	4.32	4.84	6.17	4.98	8.14
F23	F23 Slimes #1	4.2	0.35	0.75	67.9	18.5	3.30	4.13	1.27	2.51	4.01	4.58	5.68	4.82	6.22
Average		4.5	0.50	1.07	71.3	17.0	2.70	4.24	1.11	3.26	4.35	4.74	5.84	4.89	7.21

Table 12: 2nd Slime Assays

Test No.	Product	Weight %	Assays %							Distribution %					
			Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F1	F1 Slimes #2	1.5	0.65	1.40	69.6	18.1	2.80	3.79	1.60	1.28	1.41	1.69	2.08	1.45	3.86
F2	F2 Slimes #2	0.5	0.45	0.97	67.2	15.6	2.43	4.21	1.15	0.29	0.43	0.46	0.56	0.51	0.87
F3	F3 Slmies #2	1.5	0.60	1.29	60.2	14.3	1.93	2.92	1.35	1.21	1.23	1.33	1.48	1.16	3.00
F4	F4 Slimes #2	1.3	0.65	1.40	64.3	14.9	2.01	3.20	1.37	1.19	1.13	1.21	1.32	1.10	2.67
F5	F5 Slimes #2	1.4	0.65	1.40	71.3	17.0	2.47	3.82	1.39	1.20	1.30	1.44	1.66	1.33	2.82
F6	F6 Slimes #2	0.6	0.57	1.23	66.7	19.1	3.14	3.70	1.83	0.49	0.56	0.74	0.99	0.61	1.77
F7	F7 Slimes #2	3.9	0.53	1.14	71.7	17.1	2.68	4.32	1.11	2.69	3.70	4.07	5.11	4.27	6.27
F8	F8 Slimes #2	0.2	0.22	0.47	59.2	22.0	4.03	2.92	2.24	0.07	0.18	0.31	0.45	0.08	0.83
F9	F9 Slimes #2	1.8	0.58	1.25	71.2	17.0	2.47	4.07	1.29	1.47	1.76	1.92	2.27	1.95	4.24
F10	F10 Slimes #2	0.9	0.64	1.38	69.8	18.1	2.73	3.66	1.58	0.77	0.83	0.98	1.20	0.86	2.36
F11	F11 Slimes #2	1.2	0.44	0.95	69.2	16.9	2.70	3.25	1.71	0.97	1.12	1.35	1.62	1.19	2.49
F12	F12 Slimes #2	1.4	0.47	1.01	73.2	15.8	2.20	3.34	1.30	1.19	1.34	1.43	1.50	1.42	2.33
F13	F13 Slimes #2	1.4	0.67	1.44	70.6	17.3	2.50	3.76	1.56	1.45	1.36	1.56	1.73	1.33	2.74
F14	F14 Slimes #2	2.2	0.67	1.44	75.3	15.3	1.88	3.58	1.27	2.18	2.27	2.09	2.00	2.03	3.05
F15	F15 Slimes #2	2.4	0.68	1.46	75.7	14.8	1.79	3.69	1.27	2.38	2.45	2.19	2.04	2.25	3.68
F16	F16 Slimes #2	1.2	0.73	1.57	71.3	16.9	2.23	4.12	1.46	1.19	1.14	1.24	1.32	1.28	2.94
F17	F17 Slimes #2	3.3	0.53	1.14	66.9	17.8	2.96	4.15	1.31	2.38	2.94	3.57	4.62	3.40	6.71
F18	F18 Slimes #2	1.1	0.69	1.49	70.6	16.4	2.10	4.29	1.37	1.01	1.01	1.08	1.10	1.14	2.28
F19	F19 Slimes #2	0.4	0.59	1.27	69.9	17.0	2.43	4.34	1.36	0.32	0.36	0.41	0.46	0.41	0.85
F20	F20 Slimes #2	0.5	0.57	1.23	70.9	16.9	2.54	4.42	1.24	0.41	0.49	0.54	0.64	0.56	1.11
F21	F21 Slimes #2	0.6	0.39	0.84	71.2	16.6	2.48	4.38	1.24	0.38	0.54	0.53	0.56	0.67	0.82
F22	F22 Slimes #2	0.7	0.63	1.36	76.4	14.6	1.58	4.63	0.87	0.62	0.71	0.62	0.53	0.78	1.00
Average		1.4	0.57	1.2	69.7	16.80	2.46	3.84	1.40	1.14	1.28	1.40	1.60	1.35	2.67

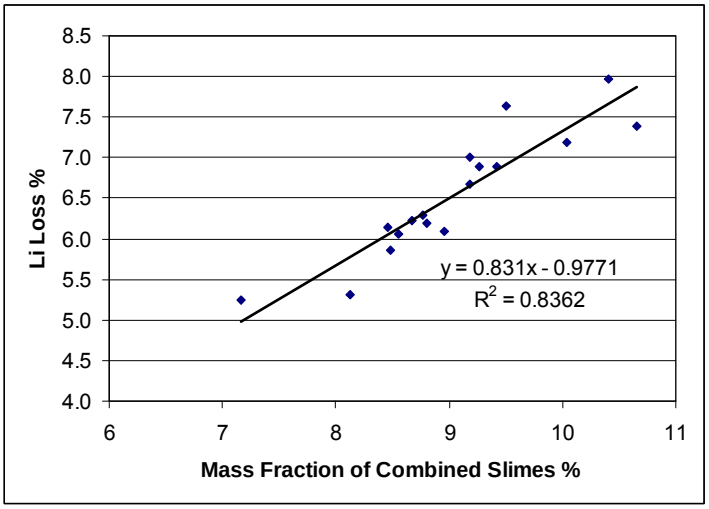


Figure 13: Correlation between Slime Mass Fractions and Lithium Losses

4.2. Development of a Spodumene Flotation Scheme

The main variables investigated in the flotation testwork were spodumene and muscovite collector types, different kinds of depressants and reagent dosages. Unless stated otherwise, grinding unit (rod mill), grinding media (stainless steel rods) and grinding size (stage grinding to -48 mesh) were kept constant. It was found that collector (fatty acid) conditioning, temperature in conditioning, and desliming had strong

effects on lithium flotation. Desliming after scrubbing was performed in a 18L acrylic cylinder. For 2 kg charges, it was initially filled to 11L, allowed to settle for 8 or 10 minutes, and decanted by siphon to above 1" distance from the top of the bed of thickened pulp. For the 10-kg charges, the cylinder was initially filled to 18L, allowed to settle for the same time and decanted by siphon to 9L. The whole procedure (filling the acrylic container with water, mixing for 30 seconds, waiting for certain period and siphoning to the desired level) was repeated twice. A typical flowsheet for the primary tests is shown in Figure 14.

A number of reagents were used in this testwork as follows:

- Armac T , Armac C and Aero 3030 C collectors for muscovite flotation;
- Kerosene as a promoter for coarse mica flotation;
- LR19, LR19B, FAT1, FAT2 and FA-2 were the main collectors used in this testwork for spodumene flotation. LR19 was prepared by combining FA-2 from Arizona Chemical (62 %), Aero 855 (27%), fuel oil (9%) and MIBC (2%). LR19B was made by mixing D30LR from Arizona Chemical (62 %), Aero 801 from Cytec (27 %), Kerosene (9 %) and Pine oil (2 %). FAT1: 80% of F-5 collector + 15% Aero 801 + 5% Diesel. FAT2 Collector was prepared by mixing FA-2 65%+ D 30 LR 35%;
- Marasperse D618 is a calcium lignin sulfonate reagent used for improving the dispersion of fine particles and improving desliming;
- Mainly sulphuric acid was used to reduce pulp pH before amine conditioning (Armac C or T) in some of the mica flotation;
- Sodium hydroxide was used in the scrubbing to improve slime suspension and to clean the spodumene grains from iron contaminants;
- Soda ash is the preferred pH regulator in spodumene flotation;
- Multivalent metal ions, like ferric nitrate and lead nitrate, were tested in the attempt to activate spodumene and mica;
- Magnetic separation did not improve the final concentrate grade to any significant level. Due the files and burrs generated from the grinding media and the mill wall during stage grinding, and since most probably low intensity magnetic separation will be included in the final flotation circuit design, flotation concentrates were passed through wet low intensity magnetic separation.

Due to its large amount in the flotation feed, mica was floated prior to spodumene. Sodium fluorosilicate may also be used to depress mica in the cleaning stages but it has not been utilized in this testwork.

The results of all the flotation tests performed in this investigation are shown in Table 13. The flowsheet development results are outlined in the following sections.

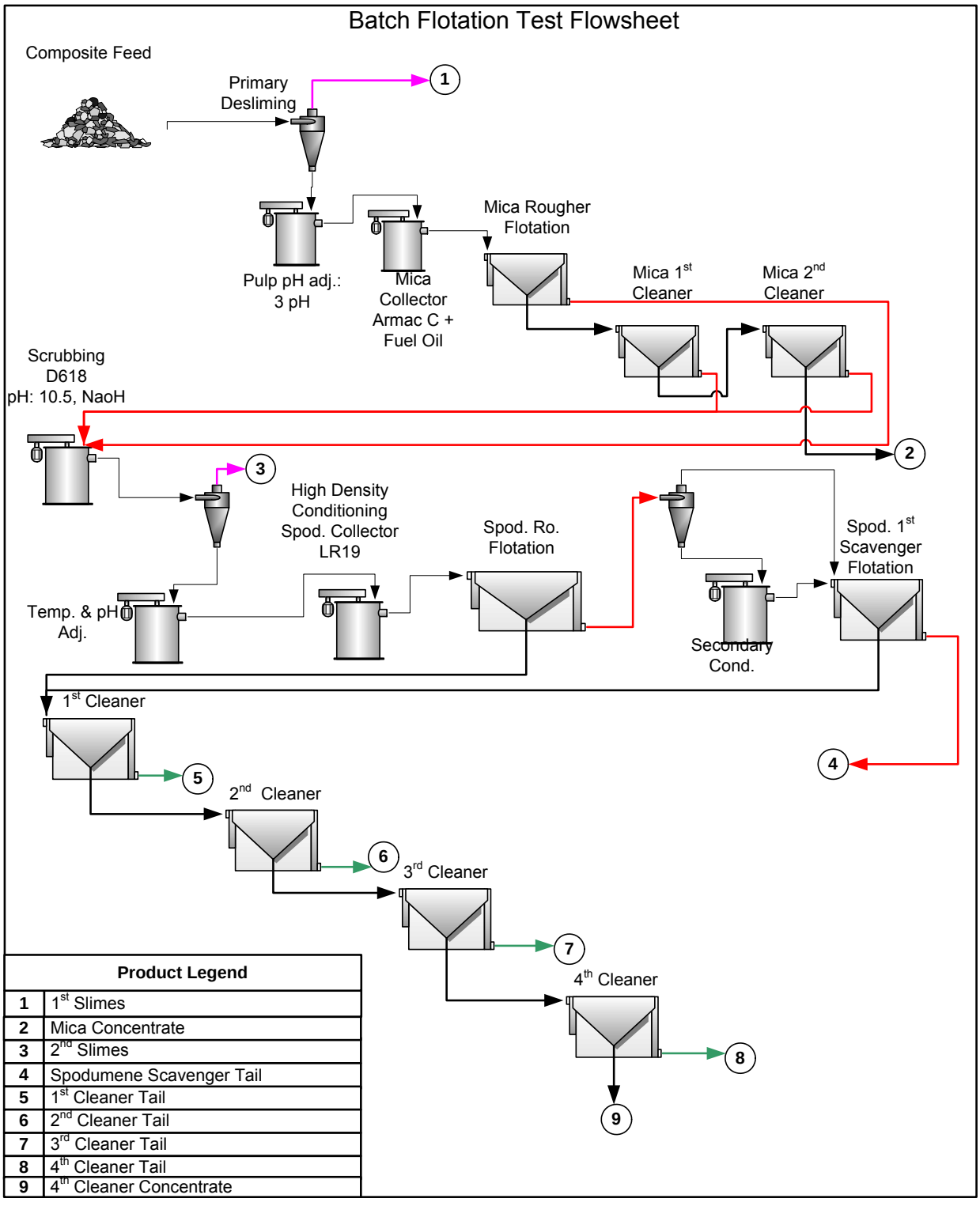


Figure 14: Typical Flotation Flowsheet for the Primary Flotation Tests

Table 13: Summary of the Spodumene Flotation Test Results

Objective	Product	Weight		Assays %							Distribution %					
		g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F1	Li Non Mag Final Conc.	671	19.1	3.05	6.57	61.0	26.2	1.09	0.58	1.00	76.6	15.7	31.1	10.3	2.81	30.7
	Li 3rd Cl Conc	685	19.5	3.03	6.52	60.6	26.0	1.09	0.58	1.45	77.6	15.9	31.6	10.5	2.85	45.4
	Li 2nd Cl Conc	705	20.1	2.99	6.43	60.9	25.9	1.10	0.65	1.44	78.8	16.4	32.3	10.9	3.29	46.2
	Li 1st Cl Conc	724	20.6	2.94	6.32	61.2	25.6	1.11	0.74	1.42	79.5	17.0	32.9	11.3	3.86	46.9
To recover spodumene based on the flowsheet developed on the previous programs	Li Ro Conc.	848	24.2	2.53	5.44	64.1	23.6	1.19	1.38	1.23	80.3	20.8	35.5	14.2	8.48	47.8
	F1 Li Ro Tail	2,009	57.3	0.12	0.26	81.7	11.2	1.62	5.11	0.10	9.03	63.0	39.9	45.7	74.2	9.18
	F1 Slimes #2	52.8	1.51	0.65	1.40	69.6	18.1	2.80	3.79	1.60	1.28	1.41	1.69	2.08	1.45	3.86
	F1 Mica 2nd Cl Conc	298	8.51	0.31	0.67	58.8	25.8	6.20	3.02	1.54	3.46	6.73	13.6	26.0	6.51	21.0
	F1 Slimes #1	185	5.27	0.54	1.16	71.9	16.9	2.62	4.39	1.09	3.74	5.10	5.53	6.80	5.86	9.20
	Cyclone OIF	114	3.26	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.17	2.96	3.79	5.26	3.49	8.98
	Head (calc.)	3,508	100	0.76	1.64	74.3	16.1	2.03	3.94	0.62	100	100	100	100	100	100
	Direct	0.00	0.00	0.69	1.49	75.6	16.0	2.01	3.92	0.48	0.00	0.00	0.00	0.00	0.00	0.00
	Flotation Feed	3,209	91.5	0.78	1.62	69.2	13.6	1.38	3.63	0.41	97.8	97.0	96.2	94.7	96.5	91.0
	Spod Flot Feed	1,013	59.8	0.93	2.00	77.0	15.7	2.10	3.32	0.54	73.8	61.5	58.3	47.1	60.6	35.4
F2	Li Non Mag Final Conc.	679	19.0	2.98	6.41	61.5	25.9	0.95	0.67	0.97	77.8	15.9	30.8	8.85	3.28	29.8
	Li 3rd Cl Conc	691	19.4	2.96	6.37	61.1	25.7	0.94	0.67	1.44	78.6	16.1	31.2	8.95	3.31	45.0
	Li 2nd Cl Conc	724	20.3	2.91	6.26	61.2	25.6	1.00	0.74	1.43	80.9	16.9	32.5	9.93	3.88	46.7
	Li 1st Cl Conc	765	21.4	2.81	6.06	61.7	25.3	1.05	0.91	1.40	82.7	18.0	33.9	11.0	4.99	48.2
	Li Ro Conc.	961	26.9	2.28	4.90	65.2	22.6	1.18	1.77	1.15	84.1	23.9	38.0	15.6	12.2	49.8
	F2 Li Ro Tail	2,003	56.1	0.08	0.17	80.6	11.0	1.62	5.07	0.11	6.16	61.5	38.6	44.5	73.2	9.96
To investigate the addition of the Li collector to the scrubber	F2 Slimes #2	16.7	0.47	0.45	0.97	67.2	15.6	2.43	4.21	1.15	0.29	0.43	0.46	0.56	0.51	0.87
	F2 Mica 2nd Cl Conc	278	7.79	0.29	0.62	56.5	27.5	7.06	2.23	1.66	3.10	5.98	13.4	26.9	4.47	20.9
	F2 Slimes #1	197	5.51	0.55	1.18	71.4	16.9	2.69	4.34	1.08	4.15	5.34	5.82	7.25	6.15	9.59
	Cyclone OIF	114	3.21	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.23	2.94	3.75	5.15	3.48	8.88
	Head (calc.)	3,569	100	0.73	1.57	73.6	16.0	2.04	3.89	0.62	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,258	91.3	0.75	1.56	69.1	13.5	1.36	3.66	0.41	97.8	97.1	96.2	94.9	96.5	91.1
	Spod Flot Feed	2,963	83.0	0.79	1.71	75.6	14.8	1.48	4.00	0.45	90.5	85.7	77.0	60.7	85.9	60.7
F3	Li Non Mag Final Conc.	577	16.1	3.23	6.95	63.2	26.6	0.43	0.48	1.01	69.6	13.7	26.3	3.52	2.03	23.9
	Li 2nd Cl. Conc.	587	16.4	3.21	6.91	62.8	26.4	0.43	0.48	1.42	70.4	13.9	26.6	3.57	2.06	34.2
	Li 1st Cl Conc	616	17.2	3.15	6.77	63.2	26.2	0.46	0.57	1.40	72.3	14.6	27.7	4.04	2.60	35.3
	Li Ro Conc.	762	21.3	2.60	5.60	66.4	23.5	0.67	1.35	1.18	74.0	19.0	30.8	7.27	7.53	37.1
	F3 Li Ro Tail	2,062	57.6	0.17	0.37	81.0	11.6	1.58	4.81	0.22	13.1	62.7	41.0	46.2	72.9	18.6
To repeat test F1 but replacing LR19 collector with FA-2	F3 Mica 2nd Cl Conc	327	9.12	0.30	0.65	60.0	25.7	5.94	3.02	1.55	3.66	7.36	14.4	27.6	7.25	20.8
	F3 Slimes #2	54.2	1.51	0.60	1.29	60.2	14.3	1.93	2.92	1.35	1.21	1.23	1.33	1.48	1.16	3.00
	F3 Mica Clnr Scav Conc	126	3.51	0.67	1.44	65.7	22.2	3.88	3.64	1.18	3.14	3.10	4.78	6.92	3.36	6.09
	F3 Slimes #1	138	3.85	0.52	1.12	71.6	17.0	2.66	4.23	1.12	2.68	3.71	4.02	5.21	4.29	6.34
	Cyclone OIF	114	3.19	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.17	2.90	3.67	5.32	3.54	8.07
	Head (calc.)	3,583	100	0.75	1.61	74.3	16.3	1.97	3.80	0.68	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,331	93.0	0.76	1.59	72.2	15.3	1.75	3.63	0.58	95.2	93.4	92.3	89.5	92.2	85.6
	Spod Flot Feed	2,824	78.8	0.86	1.85	84.0	17.8	2.02	4.22	0.66	87.1	81.7	71.8	53.5	80.4	55.7
F4	Li Non Mag Final Conc.	469	13.0	2.84	6.11	63.0	25.6	0.96	0.73	1.08	51.5	11.0	20.5	6.22	2.47	20.8
	Li 2nd Cl Conc	478	13.3	2.82	6.06	62.6	25.5	0.97	0.73	1.51	52.0	11.1	20.8	6.38	2.50	29.7
	Li 1st Cl Conc	523	14.5	2.71	5.84	63.5	24.8	1.00	0.88	1.44	54.7	12.3	22.2	7.18	3.33	30.9
	Li Ro Conc.	671	18.6	2.27	4.89	66.8	22.5	1.11	1.55	1.20	58.8	16.6	25.9	10.3	7.50	33.1
	F4 Li Ro Tail	2,106	58.4	0.32	0.69	81.0	12.2	1.46	4.60	0.25	26.0	63.3	43.9	42.4	69.9	21.6
To repeat test F1 but replacing LR19 collector with LR19B	F4 Slimes #2	47.5	1.32	0.65	1.40	64.3	14.9	2.01	3.20	1.37	1.19	1.13	1.21	1.32	1.10	2.67
	F4 Mica 2nd Cl Conc	455	12.6	0.41	0.88	61.4	24.6	5.28	3.53	1.36	7.20	10.4	19.1	33.1	11.6	25.4
	F4 Slimes #1	214	5.92	0.55	1.18	71.6	17.0	2.60	4.22	1.04	4.53	5.68	6.21	7.66	6.50	9.12
	Cyclone OIF	114	3.17	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.24	2.87	3.66	5.17	3.48	8.07
	Head (calc.)	3,607	100	0.72	1.55	74.7	16.2	2.01	3.84	0.68	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,493	96.8	0.72	1.49	70.5	15.1	1.81	3.57	0.58	97.8	97.1	96.3	94.8	96.5	91.9
	Spod Flot Feed	2,824	78.3	0.79	1.70	77.3	14.7	1.39	3.85	0.50	93.2	91.5	90.1	87.2	90.0	82.8
F5	Li Non Mag Final Conc.	608	17.1	2.96	6.37	61.1	26.6	1.14	0.56	1.06	69.1	14.0	28.3	9.69	2.45	27.1
	Li 3rd Cl Conc	617	17.4	2.94	6.33	60.7	26.4	1.13	0.56	1.51	69.6	14.1	28.6	9.76	2.48	39.0
	Li 2nd Cl Conc	633	17.8	2.91	6.26	60.8	26.4	1.14	0.60	1.49	70.7	14.5	29.2	10.1	2.73	39.7
	Li 1st Cl Conc	659	18.6	2.86	6.16	61.1	26.1	1.15	0.70	1.46	72.4	15.2	30.2	10.6	3.30	40.5
	Li Ro Conc.	776	21.9	2.47	5.32	63.8	24.1	1.23	1.33	1.28	73.6	18.7	32.7	13.3	7.44	41.7
To repeat test F1 but replacing LR19 collector with D30LR and FA-2	F5 Li Ro Tail	2,140	60.3	0.19	0.41	81.3	11.6	1.60	4.91	0.23	15.6	65.7	43.5	47.9	75.7	20.7
	F5 Slimes #2	48.2	1.36	0.65	1.40	71.3	17.0	2.47	3.82	1.39	1.20	1.30	1.44	1.66	1.33	2.82
	F5 Mica 2nd Cl Conc	305	8.59	0.33	0.71	59.8	25.6	6.03	3.12	1.49	3.87	6.89	13.7	25.7	6.86	19.1
	F5 Slimes #1	167	4.69	0.54	1.16	71.6	16.8	2.65	4.35	1.06	3.45	4.50	4.90	6.17	5.22	7.42
	Cyclone OIF	114	3.22	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.23	2.92	3.75	5.24	3.48	8.26
	Head (calc.)	3,551	100	0.73	1.58	74.6	16.1	2.01	3.91	0.67	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,270	92.1	0.75	1.55	69.4	13.6	1.38	3.59	0.47	97.8	97.1	96.2	94.8	96.5	91.7
	Spod Flot Feed	2,917	82.1	0.80	1.72	76.6	14.9	1.50	3.96	0.51	90.5	85.7	77.7	62.9	84.5	65.2
F6	Li Non Mag Final Conc.	532	15.0	2.99	6.44	62.1	27.1	0.82	0.42	1.09	61.8	12.5	24.9	6.20	1.66	25.1
	Li 3rd Cl Conc	544	15.3	2.96	6.36	61.5	26.8	0.82	0.42	1.92	62.5	12.6	25.2	6.30	1.69	45.3
	Li 2nd Cl Conc	604	17.0	2.90	6.23	61.4	26.9	0.99	0.46	1.87	67.9	14.0	28.1	8.49	2.07	48.9
	Li 1st Cl Conc	657	18.5	2.83	6.08	61.5	26.8	1.13	0.55	1.82	72.1	15.2	30.5	10.5	2.67	51.7
	Li Ro Conc.	755	21.2	2.58	5.56	62.9	25.7	1.25	0.95	1.66	75.7	17.9	33.6	13.4	5.35	54.2
To repeat test F1 but with the addition of caustic starch to the mica flotation	F6 Li Ro Tail	2,333	65.6	0.18	0.39	80.6	11.9	1.65	4.82	0.13	16.3	71.0	48.0	54.6	83.6	13.1
	F6 Slimes #2	22.3	0.63	0.57	1.23	66.7	19.1	3.14	3.70	1.83	0.49	0.56	0.74	0.99	0.61	1.77
	F6 Mica 2nd Cl Conc	160	4.50	0.28	0.60	50.0	32.2	8.48	1.25	2.07	1.74	3.02	8.92	19.3	1.49	14.3
	F6 Slimes #1	172	4.83	0.52	1.12	71.4	17.1	2.63	4.19	1.09	3.47	4.63	5.08	6.41	5.35	8.10
	Cyclone OIF	114	3.22	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.26	2.91	3.70	5.32	3.59	8.50
	Head (calc.)	3,557	100	0.72	1.56	74.5	16.3	1.98	3.78	0.65	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,271	92.0	0.74	1.57	72.5	14.6	1.49	3.68	0.49	97.7	97.1	96.3	94.7	96.4	91.5
	Spod Flot Feed	3,088	86.8	0.77	1.65	76.3	15.3	1.55	3.87	0.50	92.5	89.4	82.3	69.0	89.6	69.1

Table 14: Summary of the Spodumene Flotation Test Results……Cont'd

Objective	Product	Weight		Assays %							Distribution %					
		g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F7	Li Non Mag Final Conc.	656	18.3	3.10	6.67	62.6	26.5	0.86	0.58	1.11	74.9	15.4	30.1	7.82	2.73	29.9
	Li 2nd Cl. Conc.	664	18.6	3.08	6.64	62.3	26.4	0.86	0.58	1.44	75.4	15.5	30.3	7.88	2.76	39.1
	Li 1st Cl Conc	684	19.1	3.03	6.51	62.6	26.1	0.88	0.68	1.42	76.2	16.0	30.9	8.33	3.33	39.9
	Li Ro Conc.	799	22.3	2.62	5.64	65.1	24.1	0.99	1.32	1.25	77.0	19.5	33.3	11.0	7.56	40.9
	F7 Li Ro Tail	2,237	62.5	0.17	0.37	81.1	11.6	1.64	5.00	0.25	14.0	67.9	44.9	50.8	80.3	22.9
To test the FAT-1 collector for spodumene flotation	F7 Slimes #2	138	3.85	0.53	1.14	71.7	17.1	2.68	4.32	1.11	2.69	3.70	4.07	5.11	4.27	6.27
	F7 Mica 2nd Cl Conc	228	6.37	0.33	0.71	51.8	30.8	8.08	1.58	1.96	2.77	4.42	12.1	25.5	2.58	18.3
	F7 Slimes #1	62.8	1.75	0.59	1.27	69.8	17.6	2.75	4.02	1.35	1.36	1.64	1.91	2.39	1.81	3.48
	Cyclone OIF	114	3.20	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.14	2.89	3.70	5.19	3.46	8.06
	Head (calc.)	3,579	100	0.76	1.63	74.7	16.2	2.02	3.89	0.68	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,464	96.8	0.77	1.63	73.6	15.8	1.93	3.81	0.62	97.9	97.1	96.3	94.8	96.5	91.9
	Spod Flot Feed	3,036	84.8	0.84	1.81	80.1	15.7	1.59	4.23	0.56	91.0	87.4	78.2	61.8	87.9	63.9
F8	Li 4th Cl. Conc.	1,405	14.5	2.97	6.39	61.4	26.4	0.56	0.41	1.89	63.2	12.0	23.9	3.98	0.75	44.6
	Li 3rd Cl. Conc.	1,529	15.8	2.94	6.32	61.4	26.4	0.65	0.44	1.84	67.9	13.1	26.0	5.04	0.88	47.2
	Li 2nd Cl Conc	1,662	17.2	2.88	6.20	61.4	26.5	0.79	0.49	1.79	72.5	14.2	28.3	6.66	1.06	50.1
	Li 1st Cl Conc	1,823	18.9	2.81	6.04	61.7	26.3	0.89	0.59	1.73	77.4	15.6	30.8	8.25	1.42	52.9
To repeat Test F1 on 10 kg charge, Also testing the addition of NaF to the Li rougher scavenger	Li Ro. Conc	2,054	21.3	2.60	5.60	63.0	25.3	0.99	0.90	1.61	80.8	18.0	33.4	10.3	2.42	55.4
	F8 Li Ro Scav Tls	6,294	65.1	0.10	0.21	80.7	11.6	1.70	11.2	0.08	9.11	70.7	47.0	54.0	92.4	8.45
	F8 Slimes #2	22.1	0.23	0.22	0.47	59.2	22.0	4.03	2.92	2.24	0.07	0.18	0.31	0.45	0.08	0.83
	F8 Mica Conc	623	6.45	0.51	1.10	53.4	29.9	7.80	1.95	1.93	4.81	4.63	12.0	24.6	1.59	20.2
	F8 Slimes #1	335	3.47	0.46	0.99	71.8	16.9	2.62	4.42	0.88	2.33	3.35	3.64	4.44	1.94	4.95
	Cyclone OIF	334	3.46	0.56	1.21	68.0	17.4	3.65	3.53	1.82	2.83	3.16	3.74	6.17	1.55	10.2
	Head (calc.)	9,664	100	0.68	1.47	74.4	16.1	2.05	7.90	0.62	100	100	100	100	100	100
	Direct			0.80	1.72	75.3	16.2	2.69	3.35	0.99						
	Flotation Feed	8,371	86.6	0.71	1.53	76.2	14.9	1.52	8.65	0.45	94.8	93.5	92.6	89.4	96.5	84.8
	Spod Flot Feed	8,349	86.4	0.71	1.53	76.4	15.0	1.52	8.67	0.46	89.9	88.7	80.3	64.3	94.8	63.8
F9	Li Non Mag Final Conc.	702	19.4	3.00	6.46	63.0	26.4	0.74	0.59	0.95	80.2	16.4	31.4	7.15	2.98	32.9
	Li 2nd Cl. Conc.	718	19.8	2.98	6.41	62.7	26.3	0.75	0.59	1.28	81.4	16.7	32.0	7.36	3.04	45.2
	Li 1st Cl Conc	742	20.5	2.92	6.29	63.0	26.1	0.77	0.67	1.26	82.6	17.3	32.8	7.90	3.60	46.0
	Li Ro Conc.	779	21.5	2.81	6.06	63.6	25.5	0.81	0.84	1.22	83.5	18.4	33.7	8.72	4.73	46.7
	F9 Li Ro Tail	2,130	58.8	0.09	0.20	81.4	11.3	1.61	5.04	0.08	7.54	64.2	40.8	47.2	77.3	8.39
To repeat test F1 but replacing Armac C with Armac T	F9 Slimes #2	66.7	1.84	0.58	1.25	71.2	17.0	2.47	4.07	1.29	1.47	1.76	1.92	2.27	1.95	4.24
	F9 Mica 2nd Cl Conc	191	5.26	0.13	0.28	47.3	34.6	9.74	0.84	2.12	0.94	3.34	11.2	25.5	1.15	19.9
	F9 Slimes #1	159	4.40	0.53	1.14	71.7	17.1	2.67	4.21	1.04	3.21	4.23	4.61	5.85	4.83	8.16
	Cyclone OIF	160	4.41	0.45	.96	68.2	18.8	3.33	4.42	1.54	2.71	4.03	5.07	7.31	5.08	12.08
	Head (calc.)	3,621	100	0.73	1.56	74.5	16.3	2.01	3.84	0.56	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,302	91.2	0.75	1.60	72.3	14.1	1.35	3.74	0.37	94.1	91.7	90.3	86.8	90.1	79.8
	Spod Flot Feed	915	25.3	2.63	5.67	256	49.8	4.69	13.2	1.23	84.1	22.4	36.4	11.9	9.70	47.2
F10	Li Non Mag Final Conc.	710	19.9	3.00	6.46	63.3	26.3	0.66	0.65	0.89	80.6	16.9	32.0	6.53	3.41	29.9
	Li 2nd Cl. Conc.	721	20.2	2.98	6.42	62.9	26.2	0.66	0.65	1.35	81.4	17.1	32.3	6.65	3.45	46.2
	Li 1st Cl Conc	750	21.0	2.92	6.29	63.3	25.9	0.68	0.74	1.33	83.1	17.9	33.3	7.16	4.11	47.2
	Li Ro Conc.	786	22.0	2.83	6.08	63.8	25.4	0.72	0.89	1.29	84.2	18.9	34.3	7.87	5.20	47.9
	F10 Li Ro Tail	2,032	56.9	0.07	0.16	81.9	10.9	1.59	5.01	0.06	5.62	62.6	38.0	45.0	75.3	5.77
To repeat test F1 but using Lead Nitrate as mica activator	F10 Slimes #2	31.6	0.89	0.64	1.38	69.8	18.1	2.73	3.66	1.58	0.77	0.83	0.98	1.20	0.86	2.36
	F10 Mica 2nd Cl Conc	303	8.49	0.30	0.65	54.9	29.3	7.29	2.15	1.75	3.44	6.26	15.2	30.8	4.82	25.1
	F10 Slimes #1	128	3.60	0.54	1.16	71.2	17.3	2.73	4.14	1.11	2.63	3.44	3.81	4.89	3.93	6.75
	Cyclone OIF	160	4.48	0.45	.96	68.2	18.8	3.33	4.42	1.54	2.69	4.10	5.14	7.40	5.22	11.61
	Head (calc.)	3,569	100	0.74	1.59	74.5	16.3	2.01	3.79	0.59	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,281	91.9	0.76	1.58	69.8	13.5	1.24	3.55	0.36	94.7	92.5	91.1	87.7	90.8	81.6
	Spod Flot Feed	915	25.6	2.61	5.62	248	47.7	4.37	12.6	1.25	84.9	22.8	36.8	10.7	9.9	48.4
F11	Li Non Mag Final Conc.	581	16.2	2.11	4.54	57.8	27.7	3.33	0.65	1.31	61.9	12.5	29.5	26.7	3.18	25.4
	Li 4th Cl. Conc.	597	16.6	2.08	4.48	57.0	27.3	3.27	0.64	2.40	62.7	12.6	29.8	26.9	3.24	47.7
	Li 3rd Cl Conc	637	17.7	2.03	4.38	57.7	26.9	3.23	0.71	2.33	65.4	13.6	31.4	28.4	3.78	49.5
	Li 2nd Cl. Conc.	691	19.2	1.96	4.21	58.8	26.2	3.18	0.81	2.23	68.2	15.1	33.2	30.2	4.69	51.3
	Li 1st Cl Conc	781	21.7	1.80	3.88	61.0	24.8	2.99	1.03	2.04	71.0	17.7	35.4	32.2	6.75	53.1
To repeat test F10 but using Armac T instead of Armac C, without any Pb or Fe Salt	Li Ro Conc.	979	27.2	1.48	3.19	65.2	22.1	2.68	1.57	1.70	73.2	23.7	39.5	36.1	13.0	55.5
	F11 Ro Tail	2,124	59.1	0.18	0.39	82.4	10.3	0.99	4.28	0.19	19.3	65.0	40.1	29.0	76.5	13.5
	F11 Slimes #2	43.6	1.21	0.44	0.95	69.2	16.9	2.70	3.25	1.71	0.97	1.12	1.35	1.62	1.19	2.49
	F11 Mica Conc	167	4.66	0.11	0.24	47.2	34.2	9.82	0.81	2.40	0.93	2.93	10.5	22.7	1.14	13.4
	F11 Slimes #1	151	4.19	0.42	0.90	73.8	15.6	2.32	3.59	1.24	3.19	4.12	4.30	4.82	4.55	6.23
	F-11 Cyclone O/F	128	3.55	0.37	0.80	67.7	18.3	3.29	3.45	2.09	2.38	3.20	4.27	5.78	3.70	8.89
	Head (calc.)	3,592	100	0.55	1.19	75.0	15.2	2.02	3.31	0.83	100	100	100	100	100	100
	Flotation Feed	3,314	92.3	0.56	1.21	75.4	15.1	1.96	3.29	0.77	94.4	92.7	91.4	89.4	91.8	84.9
	Spod Flot Feed	3,103	86.4	0.59	1.27	77.0	14.0	1.52	3.43	0.67	92.5	88.6	79.6	65.1	89.4	69.0
F12	Li Non Mag Final Conc.	464	12.9	2.64	5.68	59.8	27.1	1.93	0.61	1.24	62.1	10.2	22.8	12.2	2.41	20.7
	Li 4th Cl. Conc.	479	13.3	2.59	5.58	58.8	26.6	1.90	0.60	2.59	62.9	10.3	23.1	12.4	2.46	44.5
	Li 3rd Cl Conc	512	14.2	2.55	5.48	59.1	26.6	1.96	0.65	2.51	66.1	11.1	24.7	13.7	2.84	46.1
	Li 2nd Cl. Conc.	556	15.5	2.47	5.32	59.5	26.4	2.03	0.73	2.40	69.7	12.1	26.6	15.4	3.48	48.1
	Li 1st Cl Conc	601	16.7	2.38	5.11	60.2	25.9	2.04	0.86	2.30	72.3	13.3	28.2	16.8	4.40	49.6
	Li Ro Conc.	670	18.6	2.19	4.72	62.0	24.7	2.00	1.11	2.11	74.4	15.2	30.0	18.3	6.34	50.9
To repeat test F11, but including scav flotation on the +400 fraction only	F12 Li Ro Tail	2,040	56.7	0.14	0.30	84.8	9.54	0.83	4.24	0.04	14.5	63.4	35.3	23.1	73.8	2.93
	F12 Slimes #2	49.8	1.39	0.47	1.01	73.2	15.8	2.20	3.34	1.30	1.19	1.34	1.43	1.50	1.42	2.33
	F12 Slimes #3 - 400m	36.6	1.02	0.06	0.13	81.8	10.5	0.99	4.85	0.30	0.11	1.10	0.70	0.50	1.51	0.39
	F12 Mica Conc	326	9.07	0.12	0.26	47.4	34.3	9.75	0.80	2.28	1.98	5.67	20.3	43.4	2.23	26.7
	F12 Slimes #1	133	3.71	0.41	0.88	74.0	15.7	2.32	3.51	1.21	2.77	3.62	3.80	4.22	3.99	5.80
	F-12 HL Test 1&2 Cyclone O/F	128	3.55	0.37	0.80	67.7	18.3	3.29	3.45	2.09	2.39	3.17	4.24	5.73	3.75	9.58
	Head (calc.)	3,595	100	0.55	1.18	75.8	15.3	2.04	3.26	0.77	100	100	100	100	100	100
	Flotation Feed	3,009	83.7	0.61	1.31	79.4	13.1	1.13	3.51	0.54	92.9	87.5	71.7	46.6	90.0	57.9
	Spod Flot Feed	882	24.5	1.72	3.71	67.0	21.3	1.78	1.77	1.65	77.1	21.7	34.2	21.5	13.3	52.2

Table 15: Summary of the Spodumene Flotation Test Results……Cont'd

Objective	Product	Weight		Assays %							Distribution %					
		g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F13	Li Non Mag Final Conc.	602	16.9	3.07	6.61	62.9	26.3	0.69	0.51	0.96	78.3	14.3	27.9	5.62	2.12	19.8
	Li 3rd Cl. Conc.	617	17.4	3.00	6.46	61.8	25.8	0.69	0.51	2.60	78.6	14.4	28.0	5.76	2.19	54.9
	Li 2nd Cl Conc	636	17.9	2.96	6.37	61.9	25.7	0.73	0.57	2.55	79.9	14.9	28.8	6.32	2.52	55.6
	Li 1st Cl Conc	670	18.8	2.85	6.15	62.5	25.3	0.80	0.74	2.45	81.1	15.8	29.8	7.21	3.42	56.3
	Li Ro. Conc	871	24.5	2.23	4.80	66.7	22.2	1.00	1.74	1.92	82.3	21.9	34.0	11.8	10.5	57.2
To study the effect of FA-2 and silica flotation on the spod concentrate	F13 Li 1st Ro Tail	2,003	56.3	0.06	0.13	81.5	11.3	1.73	5.20	0.13	5.10	61.6	39.8	46.9	72.1	8.92
	F13 Slimes #2	51.2	1.44	0.67	1.44	70.6	17.3	2.50	3.76	1.56	1.45	1.36	1.56	1.73	1.33	2.74
	F13 Mica Conc	344	9.68	0.34	0.73	58.4	26.0	6.10	3.10	1.59	4.96	7.58	15.8	28.4	7.39	18.8
	F13 Slimes #1	179	5.03	0.51	1.10	71.9	16.8	2.61	4.45	0.98	3.86	4.85	5.28	6.31	5.51	6.00
	Cyclone OIF	108	3.04	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.32	2.75	3.56	4.79	3.15	6.36
	Head (calc.)	3,556	100	0.66	1.43	74.6	16.0	2.08	4.06	0.82	100	100	100	100	100	100
	Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,269	91.9	0.68	1.38	68.8	13.1	1.37	3.71	0.62	93.8	92.4	91.2	88.9	91.3	87.6
	Spod Flot Feed	871	24.5	2.37	5.10	254	48.2	4.98	13.7	2.22	82.3	21.9	34.0	11.8	10.5	57.2
F14	Li Non Mag Final Conc.	837	22.0	2.22	4.78	70.4	20.5	0.72	0.90	0.90	71.5	20.9	27.6	7.58	5.04	21.3
	Li 4th Cl. Conc.	850	22.3	2.20	4.74	69.9	20.4	0.72	0.90	1.46	72.1	21.1	27.8	7.67	5.10	35.2
	Li 3rd Cl Conc	953	25.0	2.00	4.31	71.7	19.0	0.74	1.04	1.34	73.5	24.3	29.2	8.88	6.63	36.3
	Li 2nd Cl. Conc.	1,045	27.4	1.84	3.97	73.3	18.0	0.76	1.17	1.24	74.1	27.2	30.2	9.94	8.20	36.8
	Li 1st Cl Conc	1,151	30.2	1.68	3.62	74.6	17.0	0.79	1.37	1.14	74.5	30.5	31.6	11.4	10.5	37.3
	Li Ro Conc.	1,351	35.5	1.44	3.10	75.8	16.1	0.90	1.88	1.02	74.9	36.4	35.0	15.3	17.0	38.9
	F14 Li Ro Tail	1,286	33.7	0.07	0.16	78.4	13.0	1.88	5.93	0.33	3.66	35.8	26.9	30.4	51.1	12.0
Similar to F1 but on with finer flotation feed	F14 Li Ro Scav #3 -400m slimes	34.3	0.90	0.07	0.15	76.5	13.9	2.18	6.20	0.29	0.09	0.93	0.77	0.94	1.42	0.28
	F14 Slimes #2	84.7	2.22	0.67	1.44	75.3	15.3	1.88	3.58	1.27	2.18	2.27	2.09	2.00	2.03	3.05
	F14 Mica Conc	384	10.1	0.44	0.95	57.9	26.7	5.66	3.64	1.66	6.50	7.89	16.5	27.3	9.36	18.1
	F14 Slimes #1	334	8.78	0.51	1.10	72.6	16.6	2.53	4.22	1.07	6.56	8.62	8.93	10.6	9.45	10.1
	Fine Grinding-Cyclone O/F	336	8.83	0.47	1.01	68.0	18.2	3.16	4.29	1.84	6.08	8.12	9.85	13.4	9.66	17.5
	Head (calc.)	3,811	100	0.68	1.47	73.9	16.3	2.09	3.92	0.93	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,140	82.4	0.72	1.56	74.7	16.1	1.92	3.85	0.81	87.4	83.3	81.2	76.0	80.9	72.3
	Spod Flot Feed	2,637	69.2	0.77	1.67	77.1	14.6	1.38	3.85	0.68	78.6	72.2	61.9	45.7	68.1	50.9
F15	Li Non Mag Final Conc.	552	14.6	2.79	6.01	66.1	24.1	0.67	0.86	0.92	59.3	13.0	21.6	4.63	3.19	16.2
	Li 4th Cl. Conc.	566	15.0	2.75	5.93	65.4	23.8	0.67	0.86	1.83	60.1	13.2	21.9	4.73	3.26	33.0
	Li 3rd Cl Conc	662	17.5	2.60	5.59	66.7	23.0	0.72	1.03	1.67	66.2	15.7	24.8	5.98	4.58	35.3
	Li 2nd Cl. Conc.	731	19.3	2.42	5.20	68.0	22.0	0.76	1.22	1.55	68.1	17.7	26.2	7.00	5.99	36.2
	Li 1st Cl Conc	829	21.9	2.18	4.69	69.8	20.7	0.82	1.49	1.40	69.5	20.6	27.9	8.57	8.32	37.0
	Li Ro Conc.	1,090	28.8	1.69	3.63	72.9	18.3	1.00	2.25	1.11	70.8	28.3	32.5	13.6	16.5	38.5
Similar to F14, with reduced collector and increased Na silicate	F15 Li Ro Scav #4	50.0	1.32	0.36	0.77	80.1	12.0	1.42	4.41	0.27	0.69	1.43	0.98	0.89	1.48	0.43
	F15 Li Ro Tail	1,505	39.8	0.10	0.22	80.5	11.6	1.65	5.19	0.12	5.80	43.2	28.4	31.1	52.5	5.76
	F15 Slimes #2	90.9	2.40	0.68	1.46	75.7	14.8	1.79	3.69	1.27	2.38	2.45	2.19	2.04	2.25	3.68
	F15 Mica Conc	400	10.6	0.51	1.10	58.1	26.8	5.79	3.26	1.63	7.87	8.29	17.5	29.1	8.77	20.8
	F15 Slimes #1	311	8.22	0.53	1.14	73.3	16.7	2.54	4.22	1.12	6.35	8.13	8.46	9.91	8.82	11.1
	Fine Grinding-Cyclone O/F	336	8.89	0.47	1.01	68.0	18.2	3.16	4.29	1.84	6.09	8.15	9.96	13.3	9.69	19.7
	Head (calc.)	3,784	100	0.69	1.48	74.1	16.2	2.11	3.93	0.83	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,137	82.9	0.72	1.56	74.9	16.0	1.95	3.87	0.69	87.6	83.7	81.6	76.8	81.5	69.2
	Spod Flot Feed	1,140	30.1	1.63	3.50	73.3	18.1	1.02	2.35	1.07	71.5	29.8	33.5	14.5	18.0	38.9
F16	Li Non Mag Final Conc.	374	19.5	2.97	6.39	62.5	26.1	0.71	0.72	0.97	79.4	16.4	31.4	6.91	3.66	32.1
	Li 4th Cl. Conc.	381	19.9	2.95	6.36	62.2	26.0	0.71	0.72	1.28	80.4	16.7	31.8	7.08	3.72	42.9
	Li 3rd Cl Conc	393	20.5	2.90	6.23	62.5	25.8	0.74	0.83	1.25	81.2	17.3	32.5	7.60	4.40	43.5
	Li 2nd Cl. Conc.	409	21.3	2.81	6.04	62.9	25.4	0.78	1.00	1.22	81.9	18.1	33.3	8.31	5.52	44.1
	Li 1st Cl Conc	463	24.1	2.54	5.46	64.4	24.2	0.88	1.49	1.11	83.8	21.0	36.0	10.60	9.37	45.4
	Li Ro and Scav. Conc.	574	29.9	2.06	4.42	67.5	21.8	1.01	2.27	0.90	84.3	27.2	40.2	15.1	17.7	45.9
Testing the effect of 3030 C collector on mica floatation at neutral pH	F16 Li Ro Tail	867	45.2	0.01	0.02	81.9	10.6	1.62	5.10	0.04	0.50	49.9	29.5	36.5	60.0	3.06
	F16 Slimes #2	22.8	1.19	0.73	1.57	71.3	16.9	2.23	4.12	1.46	1.19	1.14	1.24	1.32	1.28	2.94
	F16 Mica Conc	302	15.7	0.38	0.82	67.0	21.0	4.53	2.97	1.10	8.19	14.2	20.3	35.5	12.2	29.3
	F16 Slimes #1	76.7	4.00	0.54	1.16	70.6	17.1	2.70	4.20	1.28	2.96	3.81	4.21	5.38	4.37	8.67
	Cyclone O/F	77	3.99	0.52	1.13	68.6	18.3	3.08	4.26	1.50	2.87	3.70	4.51	6.1	4.44	10.1
	Head (calc.)	1,919	100	0.73	1.57	74.1	16.2	2.00	3.84	0.59	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	1,766	92.0	0.79	1.71	80.6	17.6	2.18	4.17	0.64	94.2	92.5	91.3	88.5	91.2	81.2
	Spod Flot Feed	1,441	75.1	0.82	1.77	76.1	15.1	1.38	3.97	0.38	84.8	77.1	69.7	51.6	77.8	48.9
F17	Li Non Mag Final Conc.	845	22.5	2.69	5.79	59.8	27.0	1.88	0.61	1.11	83.7	18.2	37.4	20.3	3.45	39.3
	Li 4th Cl. Conc.	853	22.7	2.67	5.76	59.5	26.9	1.87	0.61	1.47	83.9	18.3	37.6	20.4	3.48	52.6
	Li 3rd Cl Conc	895	23.8	2.62	5.64	59.6	26.8	1.95	0.68	1.47	86.2	19.2	39.3	22.3	4.05	55.0
	Li 2nd Cl. Conc.	928	24.7	2.57	5.53	59.8	26.6	1.97	0.77	1.46	87.8	20.0	40.5	23.3	4.76	56.7
	Li 1st Cl Conc	963	25.7	2.51	5.40	60.2	26.3	1.97	0.89	1.43	88.9	20.9	41.6	24.2	5.76	57.9
Similar to F16 but reducing mica collector in the rougher and adding mica scavenger flotation	Li Ro Conc.	1,015	27.0	2.40	5.16	61.0	25.7	1.97	1.11	1.38	89.6	22.3	42.8	25.5	7.56	58.8
	F17 Li Ro Tail	2,122	56.5	0.03	0.07	81.9	11.0	1.69	5.31	0.12	2.42	62.5	38.3	45.8	75.4	10.68
	F17 Slimes #2	122	3.25	0.53	1.14	66.9	17.8	2.96	4.15	1.31	2.38	2.94	3.57	4.62	3.40	6.71
	F17 Mica Conc	110	2.94	0.22	0.47	50.8	31.1	8.56	1.39	2.32	0.89	2.01	5.63	12.0	1.03	10.73
	F17 Slimes #1	47.4	1.26	0.48	1.03	71.2	16.9	2.72	4.54	1.01	0.84	1.21	1.31	1.65	1.44	2.01
	Cyclone O/F	160	4.25	0.52	1.13	68.6	18.3	3.08	4.26	1.50	3.08	3.94	4.80	6.3	4.56	10.0
	Head (calc.)	3,755	100	0.72	1.56	74.1	16.2	2.09	3.98	0.64	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,548	94.5	0.74	1.58	74.4	16.1	2.03	3.96	0.59	96.1	94.8	93.9	92.1	94.0	88.0
	Spod Flot Feed	1,194	31.8	2.06	4.43	63.9	23.7	1.95	1.77	1.20	90.4	27.4	46.4	29.7	14.1	59.9
F18	Li Non Mag Final Conc.	780	20.5	3.03	6.52	62.9	25.5	0.59	0.79	0.94	85.7	17.4	32.3	5.94	4.06	30.2
	Li 4th Cl. Conc.	790	20.8	3.01	6.48	62.6	25.4	0.59	0.79	1.29	86.2	17.5	32.5	6.02	4.12	41.9
	Li 3rd Cl Conc	807	21.2	2.96	6.37	62.8	25.2	0.62	0.88	1.28	86.6	18.0	33.0	6.48	4.67	42.4
	Li 2nd Cl. Conc.	836	22.0	2.87	6.18	63.2	24.9	0.67	1.04	1.25	87.0	18.8	33.7	7.21	5.73	43.0
	Li 1st Cl Conc	886	23.3	2.72	5.85	64.1	24.2	0.73	1.31	1.19	87.3	20.1	34.8	8.37	7.66	43.5
Repeat test F16 with higher pH at scrubbing and addition of mica cleaner scavenger	Li Ro Conc.	1,104	29.1	2.19	4.71	67.4	21.7	0.92	2.15	0.98	87.5	26.4	38.8	13.1	15.7	44.4
	F18 Li Ro Tail	1,934	50.9	0.02	0.04	82.0	10.9	1.64	5.30	0.13	1.26	56.2	34.2	40.9	67.4	10.34
	F18 Slimes #2	40.5	1.07	0.69	1.49	70.6	16.4	2.10	4.29	1.37	1.01	1.01	1.08	1.10	1.14	2.28
	F18 Mica Conc	403	10.6	0.30	0.65	59.6	25.7	6.33	2.50	1.59	4.38	8.52	16.8	33.0	6.64	26.4
	F18 Slimes #1	158	4.15	0.50	1.08	70.9	16.9	2.71	4.47	1.04	2.85	3.96	4.32	5.51	4.64	6.74
	Cyclone O/F	160	4.20	0.52	1.13	68.6	18.3	3.08	4.26	1.50	3.03	3.89	4.75	6.4	4.48	9.8
	Head (calc.)	3,799	100	0.73	1.56	74.2	16.2	2.04	4.00	0.64	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,640	95.8	0.74	1.58	74.5	16.1	1.99	3.99	0.60	97.0	96.1	95.3	93.6	95.5	90.2
	Spod Flot Feed	3,038	80.0	0.81	1.74	76.7	14.8	1.38	4.16	0.44	88.7	82.6	73.0	54.1	83.1	54.8

Table 16: Summary of the Spodumene Flotation Test Results……Cont'd

Objective	Product	Weight		Assays %							Distribution %					
		g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F19	Li Non Mag Final Conc.	830	21.5	2.83	6.09	61.9	26.0	1.06	0.78	1.14	87.3	18.0	34.8	11.2	4.17	40.0
	Li 4th Cl. Conc.	842	21.8	2.82	6.07	61.6	25.9	1.05	0.78	1.47	88.3	18.2	35.1	11.3	4.21	52.4
	Li 3rd Cl Conc	867	22.5	2.76	5.94	61.9	25.7	1.08	0.90	1.45	89.0	18.8	35.9	11.9	5.01	53.2
	Li 2nd Cl. Conc.	902	23.4	2.67	5.75	62.4	25.3	1.11	1.08	1.41	89.7	19.7	36.8	12.8	6.25	53.9
	Li 1st Cl Conc	957	24.8	2.53	5.45	63.2	24.7	1.16	1.37	1.35	90.1	21.2	38.0	14.1	8.41	54.5
Repeat Test F18, with elimination of NaOH addition in scrubbing	Li Ro Conc.	1,221	31.7	1.99	4.28	66.8	21.8	1.28	2.28	1.08	90.4	28.6	42.9	19.8	17.9	55.6
	F19 Li Ro Tail	2,036	52.9	0.01	0.02	81.9	10.8	1.65	5.30	0.07	0.83	58.4	35.4	42.8	69.5	6.03
	F19 Slimes #2	14.8	0.38	0.59	1.27	69.9	17.0	2.43	4.34	1.36	0.32	0.36	0.41	0.46	0.41	0.85
	F19 Mica Conc	268	6.97	0.26	0.56	55.0	28.6	7.45	2.01	1.83	2.60	5.16	12.4	25.5	3.47	20.8
	F19 Slimes #1	151	3.93	0.48	1.03	70.4	16.9	2.70	4.46	1.04	2.70	3.73	4.12	5.21	4.35	6.66
	Cyclone O/F	160	4.15	0.52	1.13	68.6	18.3	3.08	4.26	1.50	3.11	3.84	4.72	6.3	4.38	10.1
	Head (calc.)	3,851	100	0.70	1.50	74.2	16.1	2.04	4.03	0.61	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	3,692	95.9	0.71	1.52	74.4	16.0	1.99	4.02	0.58	96.9	96.2	95.3	93.7	95.6	89.9
	Spod Flot Feed	3,257	84.6	0.75	1.62	76.2	14.9	1.51	4.17	0.45	91.3	86.9	78.4	62.6	87.4	61.6
F20	Li Non Mag Final Conc.	403	20.5	3.00	6.46	62.6	25.7	0.76	0.78	.92	86.6	17.3	32.9	7.71	3.95	33.2
	Li 4th Cl. Conc.	410	20.8	2.97	6.40	62.2	25.5	0.76	0.79	1.31	87.2	17.5	33.2	7.82	4.08	48.0
	Li 3rd Cl Conc	423	21.5	2.92	6.28	62.3	25.4	0.83	0.87	1.30	88.2	18.0	34.1	8.87	4.61	49.2
	Li 2nd Cl. Conc.	440	22.4	2.83	6.10	62.6	25.2	0.91	0.99	1.28	89.1	18.8	35.2	10.09	5.49	50.3
	Li 1st Cl Conc	471	23.9	2.67	5.75	63.3	24.6	1.03	1.26	1.23	90.0	20.4	36.8	12.14	7.46	51.6
	Li Ro Conc.	570	29.0	2.23	4.80	65.8	22.7	1.21	2.00	1.06	91.0	25.7	41.0	17.3	14.30	54.1
Repeat Test F18, Replacing LR19 with FA-2 Collector	F20 Li Ro Tail	1,118	56.9	0.02	0.03	81.5	10.8	1.65	5.30	0.06	1.20	62.5	38.4	46.4	74.5	6.00
	F20 Slimes #2	10.0	0.51	0.57	1.23	70.9	16.9	2.54	4.42	1.24	0.41	.49	.54	.64	.56	1.11
	F20 Mica Conc	84.2	4.28	0.23	0.50	50.6	31.6	8.58	1.34	2.04	1.39	2.92	8.45	18.18	1.42	15.36
	F20 Mica 2nd Cl Tail	34.8	1.77	0.44	0.95	58.2	26.8	6.37	2.32	1.53	1.10	1.39	2.96	5.58	1.01	4.76
	F20 Slimes #1	85.4	4.34	0.49	1.05	71.0	17.3	2.80	4.54	1.07	3.00	4.16	4.69	6.02	4.87	8.17
	Cyclone O/F	64	3.27	0.41	.89	65.6	19.5	3.64	4.18	1.82	1.90	3	4	6	3	10
	Head (calc.)	1,966	100	0.71	1.53	74.18	16.01	2.02	4.05	0.57	100	100	100	100	100	100
	Head (Dir.)	0.00	0.00	0.69	1.49	75.6	16.0	2.01	3.92	0.48	0.00	0.00	0.00	0.00	0.00	0.00
	Flotation Feed	1,817	92.4	0.73	1.57	74.63	15.82	1.93	4.02	0.50	95.1	93.0	91.3	88.1	91.8	81.3
	Spod Flot Feed	1,688	85.8	0.76	1.64	76.19	14.80	1.50	4.19	0.40	92.2	88.2	79.4	63.7	88.8	60.1
F21	Li Non Mag Final Conc.	368	17.3	2.30	4.95	58.8	27.7	2.71	0.74	1.51	69.6	14.0	27.8	19.2	3.54	31.4
	Li 4th Cl. Conc.	375	17.6	2.28	4.91	58.4	27.5	2.70	0.74	1.83	70.2	14.1	28.1	19.5	3.59	38.7
	Li 3rd Cl Conc	433	20.3	2.12	4.57	58.3	27.7	3.06	0.84	1.79	75.5	16.3	32.6	25.5	4.74	43.8
	Li 2nd Cl. Conc.	455	21.4	2.10	4.51	58.7	27.4	3.00	0.94	1.76	78.3	17.2	34.0	26.2	5.54	45.2
	Li 1st Cl Conc	479	22.5	2.05	4.41	59.3	27.0	2.93	1.06	1.72	80.6	18.3	35.3	27.0	6.59	46.4
	Li Ro Conc.	515	24.2	1.97	4.24	60.2	26.4	2.84	1.26	1.65	83.4	20.0	37.0	28.1	8.40	47.9
Repeat Test F18, using combined HL middlings and undersize fraction	F21 Li Ro Tail	973	45.8	0.02	0.04	84.3	9.97	1.03	4.96	0.10	1.44	52.9	26.5	19.3	62.7	5.49
	F21 Slimes #2	11.7	0.55	0.39	0.84	71.2	16.6	2.48	4.38	1.24	0.38	0.54	0.53	0.56	0.67	0.82
	F21 Mica Conc	190	8.95	0.19	0.41	51.4	31.0	8.38	1.41	2.15	2.97	6.30	16.1	30.7	3.49	23.1
	F21 Mica Conc 2nd Cl Tail	52.3	2.46	0.40	0.86	63.4	22.8	4.87	3.03	1.37	1.72	2.14	3.25	4.90	2.06	4.04
	F21 Slimes #1	88.7	4.17	0.36	0.77	68.1	18.9	3.36	4.08	1.34	2.62	3.90	4.57	5.74	4.70	6.71
	Mixed HL Smpl Cyclone O/F	66.2	3.11	0.27	0.58	62.4	21.7	4.38	3.71	2.03	1.47	2.66	3.92	5.58	3.19	7.59
	Head (calc.)	2,126	100	0.57	1.23	72.9	17.2	2.44	3.62	0.83	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	1,971	92.7	0.56	1.21	66.5	13.3	1.39	3.35	0.52	95.9	93.4	91.5	88.7	92.1	85.7
	Spod Flot Feed	1,717	80.8	0.61	1.31	20.5	8.39	0.89	0.52	0.51	90.8	84.5	71.6	52.5	85.9	57.8
F22	Li Ro Conc 1	392	20.3	2.68	5.77	64.0	24.1	0.70	1.32	1.10	78.1	17.6	30.6	6.93	6.61	37.4
	Li Ro Conc 1-2	445	23.0	2.51	5.40	64.9	23.4	0.77	1.61	1.04	83.1	20.3	33.7	8.67	9.17	40.2
	Li 1st Ro Scav. Conc 1	36.8	1.90	0.16	0.34	79.4	11.8	1.44	5.29	0.24	0.44	2.05	1.41	1.34	2.49	0.77
	Li 1st Ro Scav. Conc 1-2	70.2	3.63	0.12	0.25	79.8	11.7	1.49	5.39	0.23	0.61	3.93	2.66	2.65	4.84	1.38
	Li 1st Ro Scav. Conc 1-3	107	5.55	0.09	0.20	80.1	11.6	1.51	5.47	0.21	0.72	6.03	4.03	4.10	7.51	1.92
	Li 2nd Ro Scav Conc. 1	31.9	1.65	0.03	0.06	80.3	11.6	1.50	5.79	0.20	0.07	1.80	1.20	1.21	2.36	0.55
	Li 2nd Ro Scav Conc. 1-2	61.9	3.20	0.02	0.05	80.1	11.7	1.58	5.92	0.19	0.11	3.48	2.36	2.47	4.69	1.02
	Li Ro & 1st Scav Conc.	553	28.6	2.04	4.39	67.9	21.1	0.91	2.36	0.88	83.9	26.3	37.7	12.8	16.7	42.2
	Li Ro & Both Scav Conc.	614	31.8	1.84	3.95	69.1	20.1	0.98	2.72	0.81	84.0	29.8	40.1	15.2	21.4	43.2
	Mica Ro Conc 1	44.0	2.27	0.43	0.93	59.8	24.6	5.66	2.70	1.44	1.41	1.85	3.51	6.30	1.52	5.50
	Mica Ro Conc 1-2	160	8.28	0.34	0.74	56.5	27.2	6.83	2.16	1.64	4.09	6.35	14.1	27.6	4.43	22.85
	Mica Ro Conc 1-3	189	9.77	0.38	0.82	58.0	26.1	6.35	2.39	1.57	5.35	7.69	16.0	30.3	5.78	25.70
Kinetics Tests	Mica Ro Conc 1-4	200	10.3	0.40	0.85	58.5	25.6	6.15	2.48	1.54	5.88	8.21	16.6	31.0	6.34	26.78
	Mica Ro Scav 1	31.4	1.62	0.81	1.74	68.7	19.2	2.87	3.78	1.05	1.89	1.51	1.95	2.28	1.52	2.86
	Mica Ro Scav 1-2	49.3	2.55	0.82	1.76	69.3	18.9	2.73	3.86	1.05	3.00	2.40	3.02	3.40	2.43	4.48
	Mica Ro Scav 1-3	62.2	3.22	0.81	1.75	69.4	18.5	2.63	3.91	1.04	3.76	3.03	3.73	4.13	3.11	5.60
	Mica Ro & Scav	262	13.5	0.49	1.06	61.1	24.0	5.31	2.82	1.42	9.63	11.2	20.3	35.2	9.44	32.39
	F22 Li Ro Tail	894	46.2	0.01	0.02	81.3	10.4	1.64	5.25	0.07	0.53	51.0	30.1	37.0	60.0	5.43
	F22 Slimes #2	13.2	0.68	0.63	1.36	76.4	14.6	1.58	4.63	0.87	0.62	0.71	0.62	0.53	0.78	1.00
	F22 Slimes #1	86.8	4.49	0.50	1.08	70.9	17.2	2.81	4.49	1.08	3.23	4.32	4.84	6.17	4.98	8.14
	Cyclone O/F	64.2	3.32	0.42	0.90	65.6	19.4	3.60	4.21	1.77	2.00	2.95	4.02	5.84	3.45	9.86
	Head (calc.)	1,934	100	0.69	1.50	73.7	16.0	2.05	4.04	0.60	100	100	100	100	100	100
	Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92							
	Flotation Feed	1,870	96.7	0.70	1.52	73.9	15.8	1.99	4.04	0.55	98.0	97.0	96.0	94.2	96.5	90.1
	Spod Flot Feed	1,770	91.5	0.71	1.54	74.1	15.8	1.95	4.02	0.53	94.0	92.1	90.4	87.3	90.9	80.9
F23	Li Non Mag Final Conc.	291	13.8	2.80	6.03	62.1	25.1	0.95	1.00	1.16	65.6	12.0	20.3	5.35	3.82	18.6
	Li 3rd Cl Conc	295	14.0	2.78	5.99	61.8	25.0	0.95	1.00	1.34	66.1	12.1	20.5	5.44	3.87	21.7
	Li 2nd Cl. Conc.	311	14.8	2.70	5.82	62.2	24.7	0.98	1.13	1.31	67.8	12.8	21.4	5.90	4.62	22.5
Repeat Test F21 with some modifications to better float mica and depress it in the spodumene flotation	Li 1st Cl Conc	339	16.1	2.56	5.52	63.2	24.1	1.01	1.37	1.26	69.9	14.2	22.7	6.63	6.11	23.5
	Li Ro Conc.	391	18.6	2.27	4.88	65.3	22.6	1.03	1.84	1.13	71.5	17.0	24.6	7.83	9.43	24.4
	F23 Li Ro Tail	740	35.2	0.05	0.11	81.6	9.70	1.00	4.88	0.21	2.92	40.1	20.0	14.3	47.4	8.55
	F23 Mica Conc	816	38.8	0.33	0.71	67.0	20.7	4.22	3.28	1.19	21.7	36.2	46.9	66.6	35.1	53.4
	F23 Slimes #1	89.0	4.23	0.35	0.75	67.9	18.5	3.30	4.13	1.27	2.51	4.01	4.58	5.68	4.82	6.22
	Mixed HL Smpl Cyclone O/F	66.2	3.15	0.27	0.58	62.4	21.7	4.38	3.71	2.03	1.44	2.74	3.99	5.61	3.22	7.40
	Head (calc.)	2,102	100	0.59	1.27	71.7	17.1	2.46	3.62	0.86	100	100	100	100	100	100
	Head (Dir.)	0.00	0.00	0.69	1.49	75.6	16.0	2.01	3.92	0.48						
	Flotation Feed	2,036	96.9	0.60	1.29	72.0	17.0	2.40	3.62	0.83	98.6	97.3	96.0	94.4	96.8	92.6
	Spod Flot Feed	2,036	96.9	0.59	1.26	69.1	16.2	2.25	3.44	0.77	98.6	97.3	96.0	94.4	96.8	92.6

4.3. Mica Flotation

Since the ore sample contained a significant amount of mica, the common practice is to float mica first to prevent further dilution/complication of the spodumene flotation. Three mica collectors, Armac C, Armac T and Aero3030C, were tested. Desliming was performed ahead of mica flotation. It was found that not all the slimes would be separated in the cyclone overflow and that performing an extra desliming stage ahead of mica flotation would improve mica separation.

Conditioning and flotation with both Armac collectors were performed in an acidic environment. Aero 3030C was conditioned in a neutral pH environment and without the use of any pulp pH regulator. That is considered as an advantage of using 3030C. Fuel oil was used as the mica flotation promoter. The results of mica flotation are shown in Table 17 and Figure 15. The test conditions are presented in Table 18. The results indicated that:

- In the majority of the mica flotation tests performed with Armac C, lithium distributing in the mica concentrate was slightly higher in comparison with those cases that Armac T or 3030C were used with similar mass pull. Thus, it was concluded that Armac C was a less selective mica collector;
- Fuel oil addition with Armac or 3030 C collector is expected to improve coarse mica recovery;
- Further addition of the Armac or 3030C collector in the cleaning stages was found to be helpful to keep high mica recovery in the cleaning stages;
- Scavenger flotation was found to be necessary when Armac T or 3030C collector was tried;
- It was possible to control the lithium loss to about 2-3% by using Aero 3030C. A small portion of Li is expected to be in mica lattice and not due to the spodumene deportation to the mica concentrate;
- Those tests that were conducted on the heavy liquid middling and undersize fractions contained more mica than the normal charge. High recovery of mica to the flotation concentrate would directly affect spodumene flotation;
- There is a linear relation, although not very strongly, between lithium losses in the mica concentrates and the concentrate mass pulls;
- Poor mica recovery and very low mass pull of the mica concentrate can lead to the low grade spodumene concentrate and failure to reach to the target lithium grade;
- Collector 3030C can be the preferred collector for mica flotation since it performed reasonably well at neutral environment and eliminated acid consumption without jeopardizing lithium recovery;
- Addition of lead nitrate to the mica rougher conditioner with the objective to improve mica recovery did not have any pronounced effect on the mica flotation.

Table 17: Mica Pre-Flotation Results

Product	Weight		Assays %							Distribution %					
	gr	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
F1 Mica 2nd Cl Conc	298	8.51	0.31	0.67	58.8	25.8	6.20	3.02	1.54	3.46	6.73	13.6	26.0	6.51	21.0
F2 Mica 2nd Cl Conc	278	7.79	0.29	0.62	56.5	27.5	7.06	2.23	1.66	3.10	5.98	13.4	26.9	4.47	20.9
F3 Mica 2nd Cl Conc	327	9.12	0.30	0.65	60.0	25.7	5.94	3.02	1.55	3.66	7.36	14.4	27.6	7.25	20.8
F3 Mica Clnr Scav Conc	126	3.51	0.67	1.44	65.7	22.2	3.88	3.64	1.18	3.14	3.10	4.78	6.92	3.36	6.09
F4 Mica 2nd Cl Conc	455	12.6	0.41	0.88	61.4	24.6	5.28	3.53	1.36	7.20	10.4	19.1	33.1	11.6	25.4
F5 Mica 2nd Cl Conc	305	8.59	0.33	0.71	59.8	25.6	6.03	3.12	1.49	3.87	6.89	13.7	25.7	6.86	19.1
F6 Mica 2nd Cl Conc	160	4.50	0.28	0.60	50.0	32.2	8.48	1.25	2.07	1.74	3.02	8.92	19.3	1.49	14.3
F7 Mica 2nd Cl Conc	228	6.37	0.33	0.71	51.8	30.8	8.08	1.58	1.96	2.77	4.42	12.1	25.5	2.58	18.3
F8 Mica Conc	623	6.45	0.51	1.10	53.4	29.9	7.80	1.95	1.93	4.81	4.63	12.0	24.6	1.59	20.2
F10 Mica 2nd Cl Conc	303	8.49	0.30	0.65	54.9	29.3	7.29	2.15	1.75	3.44	6.26	15.2	30.8	4.82	25.1
F13 Mica Conc	344	9.68	0.34	0.73	58.4	26.0	6.10	3.10	1.59	4.96	7.58	15.8	28.4	7.39	18.8
F14 Mica Conc	384	10.1	0.44	0.95	57.9	26.7	5.66	3.64	1.66	6.50	7.89	16.5	27.3	9.36	18.1
F15 Mica Conc	400	10.6	0.51	1.10	58.1	26.8	5.79	3.26	1.63	7.87	8.29	17.5	29.1	8.77	20.8
F9 Mica 2nd Cl Conc	191	5.26	0.13	0.28	47.3	34.6	9.74	0.84	2.12	0.94	3.34	11.2	25.5	1.15	19.9
F11 Mica Conc	167	4.66	0.11	0.24	47.2	34.2	9.82	0.81	2.40	0.93	2.93	10.5	22.7	1.14	13.4
F12 Mica Conc	326	9.07	0.12	0.26	47.4	34.3	9.75	0.80	2.28	1.98	5.67	20.3	43.4	2.23	26.7
F16 Mica Conc	302	15.7	0.38	0.82	67.0	21.0	4.53	2.97	1.10	8.19	14.2	20.3	35.5	12.2	29.3
F17 Mica Conc	110	2.94	0.22	0.47	50.8	31.1	8.56	1.39	2.32	0.89	2.01	5.63	12.0	1.03	10.7
F18 Mica Conc	403	10.6	0.30	0.65	59.6	25.7	6.33	2.50	1.59	4.38	8.52	16.8	33.0	6.64	26.4
F19 Mica Conc	268	6.97	0.26	0.56	55.0	28.6	7.45	2.01	1.83	2.60	5.16	12.4	25.5	3.47	20.8
F20 Mica Conc	84.2	4.28	0.23	0.50	50.6	31.6	8.58	1.34	2.04	1.39	2.92	8.45	18.2	1.42	15.4
F21 Mica Conc	190	8.95	0.19	0.41	51.4	31.0	8.38	1.41	2.15	2.97	6.30	16.1	30.7	3.49	23.1
F23 Mica Conc	816	38.8	0.33	0.71	67.0	20.7	4.22	3.28	1.19	21.7	36.2	46.9	66.6	35.1	53.4

Table 18: Mica Flotation Conditions

			Collector Dasage g/t				Fuel Oil
Test	pH	Collector	Rougher	Scav.	Cl 1	Cl 2	g/t
F1	3.0	Armac C	50		10		25
F2	3-4	Armac C	50		10		25
F3	3.0	Armac C	50	25	10		25+5
F4	3.0	Armac C	50	25	25		25
F5	3.0	Armac C	80		10		25
F6	3.0	Armac C	80		10		25
F7	3.0	Armac C	50		10		25
F8	3.0	Armac C	80		10		25+15
F9	3.0	Armac T	80		10		25
F10	3.0	Armac C	80		10		25
F11	3.0	Armac T	80		10		25
F12	3.0	Armac T	80		25	25	25
F13	3.0	Armac C	80		10		25
F14	3.0	Armac C	80		25	25	25
F15	3.0	Armac C	80		25	25	25
F16	7.8	Aero 3030C	80		10	10	25
F17	7.8	Aero 3030C	70	20	10	10	25
F18	7.8	Aero 3030C	80	20	10	10	25
F19	7.8	Aero 3030C	80	20	10	10	25
F20	7.8	Aero 3030C	80	20	10	10	50
F21	7.8	Aero 3030C	80	20	10	10	50
F22	8.0	Aero 3030C	80	20		25	50
F23	8.0	Aero 3030C	80	70	30		60

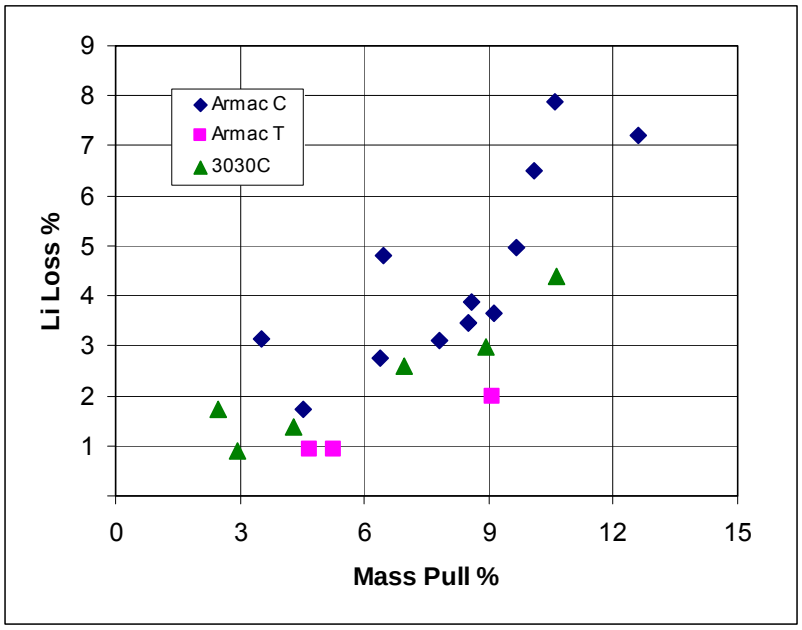


Figure 15: Lithium Distributions and Mass Pulls in the Mica Concentrates

4.4. Primary Spodumene Flotation Tests

In all the flotation tests, mica was first floated with Armac C or 3030C collectors in acidic or neutral pulp pH. The pulp was then conditioned with NaOH at pH 11 and D618 at 250 g/t, followed by desliming through decantation. Soda ash (Na_2CO_3) was used to readjust the pH in the spodumene conditioning and collector, LR19 or LR19B or FA2, was added at normally around 600 g/t and conditioned at high pulp density for 5 minutes followed by spodumene rougher flotation. Further spodumene scavenger flotation was also performed on the rougher tailings. The concentrates from the spodumene rougher and scavenger flotation were combined and cleaned in 3 or 4 stages. The flotation kinetics was fast and the majority of the spodumene floated within 1 minute. The majority of the flotation tests were conducted on the original composite. A few tests were also conducted on the heavy liquid middling combined with undersize fraction (-0.5 mm). The results of the first test were attractive. Final spodumene concentrate contained 6.57% Li_2O with a recovery of 76.6% Li. After the 1st cleaner, the concentrate grade reached to 6.32% Li_2O with a Li recovery of 79.5%. The major impurities in the final concentrate were probably from both k-feldspar and mica. As a result K_2O assay in the concentrate rose to 1.09% K_2O. The major lithium loss was in the scavenger tail (9%), followed by 1st slime (3.7%), mica concentrate (3.5%), cyclone O/F (2.2%) and finally 2nd slime after scrubbing (1.3%). Thus, the main area for lithium flotation improvement will be increased lithium recovery in the rougher flotation.

4.4.1. Effect of Collector Addition to the Scrubber

Since in future locked cycle tests, some of the collector in the pulp will be re-circulated to the scrubbing tank, it was decided to evaluate the effect of collector addition to the scrubber. A large amount of collector (2500 g/t) was added to the scrubber followed by desliming. No collector was then added to the rougher spodumene flotation. The results are illustrated in Figure 16. Although the rougher flotation concentrate grade was less than in test F1, after the 1st cleaner the concentrate grade reached 6.02% K_2O at a Li recovery of 83%. Thus, it does not seem that residual collector transported to the scrubber will have any serious negative effect on the flotation performance.

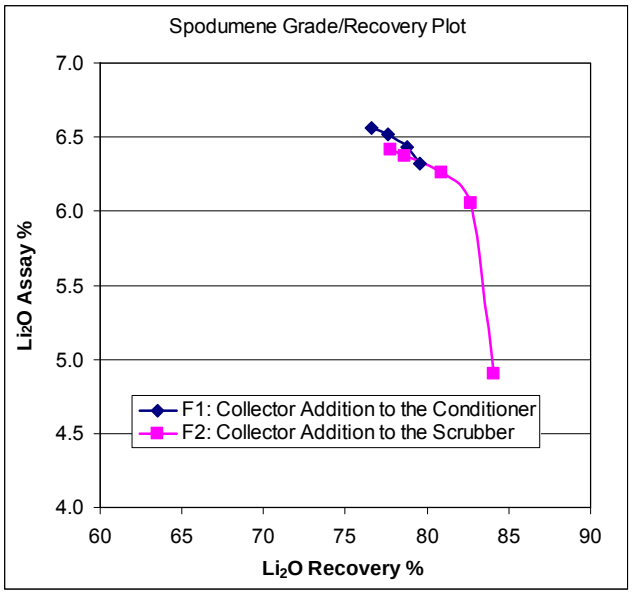


Figure 16: Effects of Collector Addition to the Scrubber

4.4.2. Effect of Different Types of Collectors

A number of spodumene collectors were tested on this sample and the results are provided in Figure 17. Among all the collectors tested, LR19, FAT-1 and FA-2 provided the best results. The loss of lithium to the scavenger concentrate was high, 26%, in test F4 with LR19B collector.

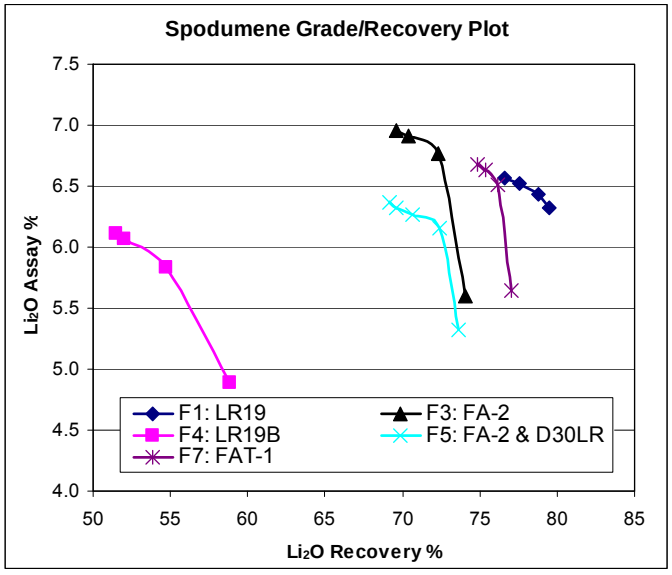


Figure 17: Effect of Different Types of Collectors on Spodumene Flotation

4.4.3. Effect of Starch Addition to the Spodumene Flotation

A minor amount of starch (50 g/t) was added ahead of mica flotation to reduce spodumene deportation to the mica concentrate. The depressing effect of starch on the spodumene continued even after scrubbing and desliming. As a result, spodumene recovery was lower in test F6 in comparison with test F1. In test F6, about 14% of the lithium deported to the scavenger tail which can be partially attributed to the depressing effect of starch on spodumene. The lithium loss in mica concentrate decreased to 2.77%, from 3.74% in test F1. This can be also partially attributed to the reduced mass pull in test F7 at 6.4%, in comparison with 8.5% in test F1.

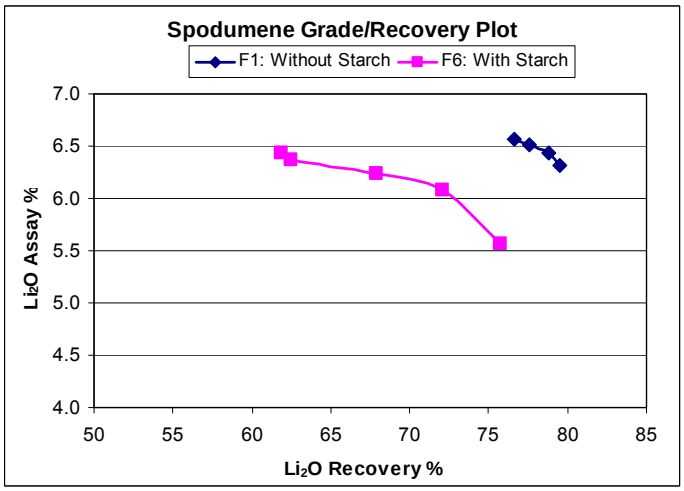


Figure 18: Effect of Starch Addition ahead of Mica Flotation on Spodumene Flotation

4.4.4. Effect of NaF Addition

In test F8, which was performed in a 10 kg cell, NaF was added to the lithium rougher scavenger flotation with the objective to reduce the lithium loss to the tailings. The lithium losses in the tailings were still 9.1%. After the 1st cleaner, the concentrate grade reached 6.04% Li_2O at a Li recovery of 77.4%. NaF might have a minor benefit to improve lithium recovery.

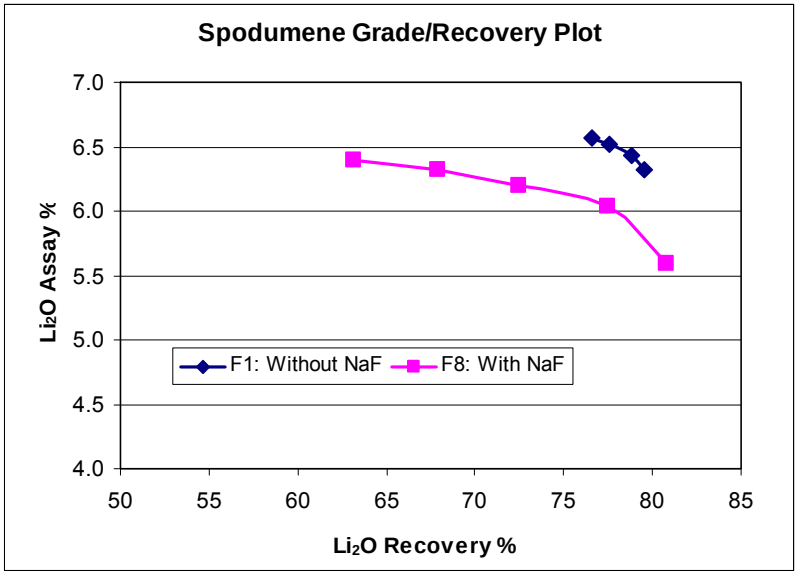


Figure 19: Effect of NaF Addition on Spodumene Flotation

4.4.5. Effect of Finer Primary Grinding

The effect of finer primary grind size was investigated and the results are shown in Figure 20. These tests were performed by stage grinding a 10 kg charges in a 10kg rod mill with closing sizes of 150 µm in tests F14 and F15, and to 300 µm in test F1. Test F15 was conducted similar to test F14, but spodumene collector was added in the scavenger flotation in stages and a minor amount of sodium silicate (50 g/t) was added to the last cleaner. It is clear that finer grinding did not improve Li recovery or concentrate grade. In test F15, it was possible to generate a concentrate grading 6.01% Li_2O but with the recovery of 59.3% or lower. In both tests, F14 and F15, the majority of the Li losses were in the mica concentrate, cyclone overflow and 1st slime. In test F15, the lithium losses were 7.87%, 6.35% and 6.09% in the mica concentrate, 1st slime and Cyclone O/F, respectively. Thus, lithium losses in the finer fraction and concentrate upgrading are the major problems with the fine grinding, in addition to the higher grinding cost.

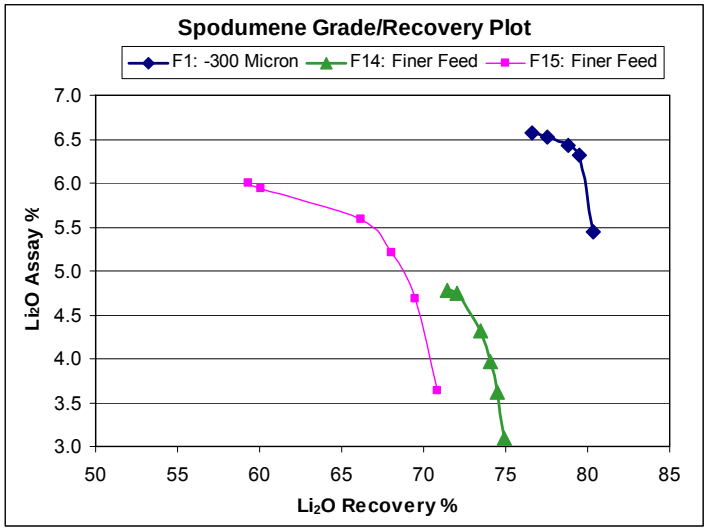


Figure 20: Effect of Grinding Fineness on Spodumene Flotation

4.4.6. Evaluating the Effects of a 2kg Feed vs. 10 kg Feed Charge

Test F13 was performed similar to test F1 but in a 10 kg flotation cell and FA-2 was used as the main lithium collector compared with LR19 in F1. The results are illustrated in Figure 21. The remaining conditions were similar in both tests. The results do not indicate that changing the flotation cell size or the collector will negatively affect spodumene flotation. In both cases, it was possible to generate a concentrate with 6.3% Li_2O grade and 80% Li recovery.

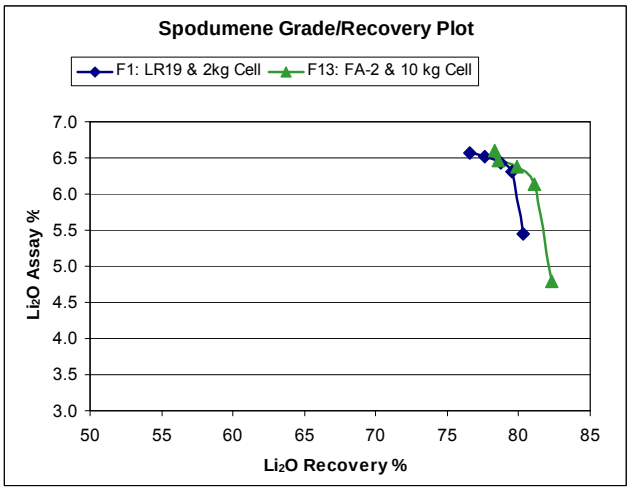


Figure 21 Effect of 2kg feed vs. 10 kg feed

4.4.7. Effect of 3030C Mica Collector

This collector can be used in neutral or slightly alkaline environment to float mica. Floating mica at natural pH is the preferred practice since it will eliminate acid consumption. Thus, in test F16, 3030C collector

was used to pre-float mica at natural pH. Test F17 was performed similar to test F16, but further mica rougher scavengers were implemented within the mica flotation circuit. The results are provided in Figure 22 . In test F16, the spodumene final concentrate contained 6.39% Li_2O at a lithium recovery of 77.4%. Thus, it was concluded that floating mica at neutral pH with 3030C collector would provide similar results to those tests in which Armac C was used at low pH. Test F17 did not, however, provide attractive results. The lithium recovery of the final concentrate generated in this test reached 81.4% but with a concentrate grade of 5.79% Li_2O. It was not clear why the concentrate grade did not reach an acceptable level, but this was likely a result of experimental conditions and not a technical problem.

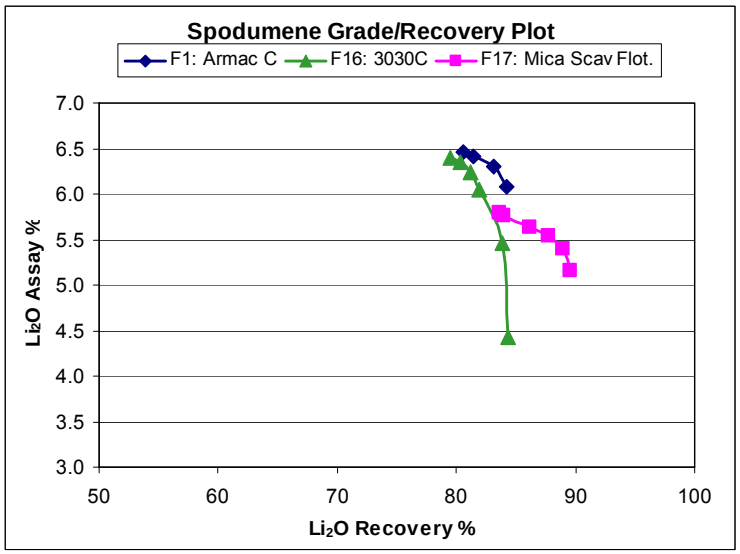


Figure 22 Effect of Mica Flotation at Neutral pH of Spodumene Flotation

4.4.8. Effect of Pulp pH at Scrubbing Stage

The effect of pulp pH at the scrubbing stage was investigated in tests F18, F19 and F20. Test F18 was performed with pulp pH adjusted to 11.5, whereas in test F19, no NaOH was added to pulp and scrubbing was performed at natural pH, 7.8. Pulp pH in scrubber was raised to 11.0 in test F20, and LR19 collector was substituted by FA-2 collector. The results are presented in Figure 23. it appears that higher pH improved final concentrate grade.

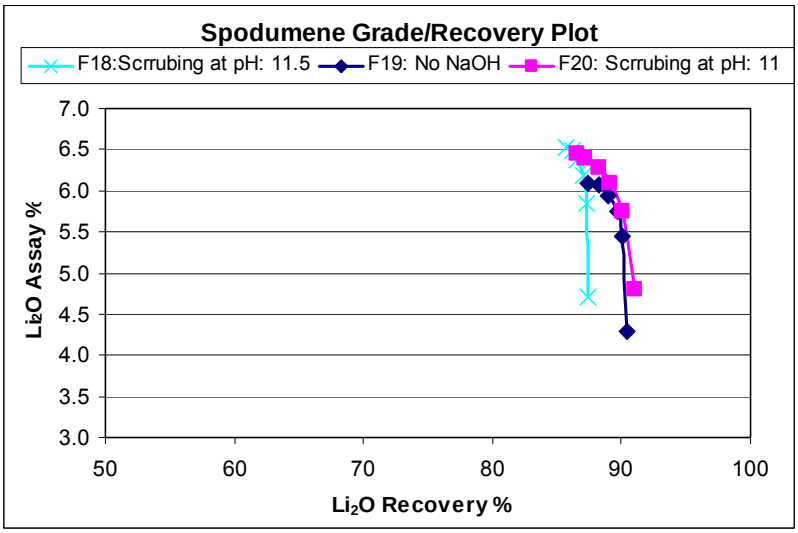


Figure 23 Effect of Scrubbing pH on Spodumene Flotation

4.4.9. Flotation Results on the Heavy Liquid Middling and Undersize Fraction

Four flotation tests, F11, F12, F21, and F23, were conducted on the combined undersize, -0.5 mm fraction, and the heavy liquid middlings. Their results are illustrated in Figure 24. The final concentrates of tests F11 and F21 had the lowest grades. In tests F11 and F12, Armac T was used, and in tests F21 and F22, 3030C was used as mica collector. Test F12 was performed similar to test F11, but further Armac T was added to the 1st and 2nd mica cleaner. Test F23 was performed similar to test F21, but mica was further floated in a number of rougher scavengers. In test F23, after three cleaners the final concentrate grade reached 6.03% at a Li recovery of 65.6%. A large portion of the lithium, 21.7%, deported to the mica combined concentrates due to the addition of large amount of mica collector. Such high collector dosages can be decreased significantly. It thus appears that the middlings from the heavy liquid tests (DMS in the real practice) can be processed to provide a high grade concentrate with a good Li recovery, but further development tests are necessary to optimize the flotation reagent dosages.

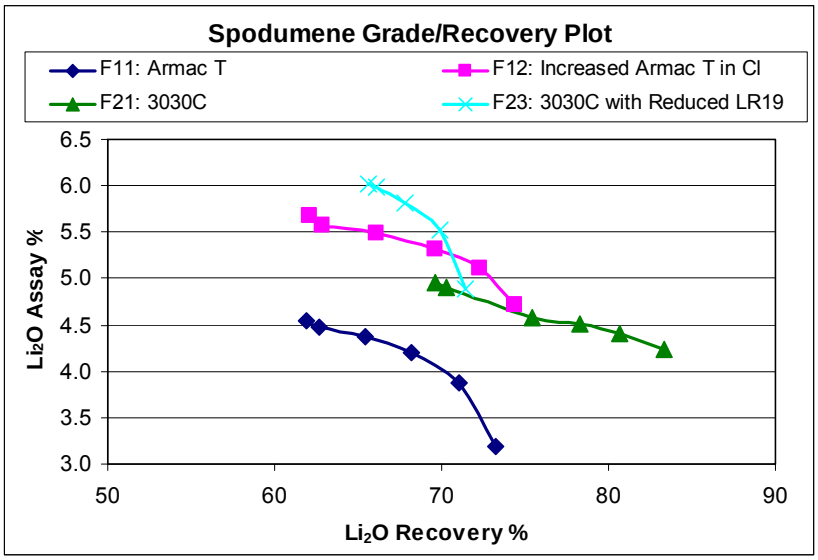


Figure 24: Results of the flotation Tests on the Heavy Liquid Middling Samples

4.4.10. Mica and Spodumene Flotation Kinetics

One rougher kinetics test, test F22, was performed on the composite head sample similar to the rougher flotation conditions as in test F20, but collector FA-2 was replaced with LR19A. The feed was stage ground to -300 µm. Collector 3030C and fuel oil were added to the mica conditioner at 80 and 50 g/t, respectively. Scrubbing was performed at pH 11 with the addition of 250 g/t D618. Collector LR19 was added to the spodumene rougher flotation. Timed rougher concentrates were collected from both mica and spodumene rougher flotation for 0.5, 0.5, 1, and 1 minute. The results are illustrated in Figure 25: Mica and Spodumene Flotation Kinetics. More than 78% of lithium was recovered within the first 30 seconds of rougher flotation. The addition of the lithium scavenger flotation did not improve the recovery to any significant level. Lithium recovery only increased by 0.9% by combining rougher and scavenger concentrates, at a decreased rougher concentrate grade from 5.4% to 3.95% Li_2O. Thus, it was concluded that scavenger flotation would not be necessary. The main indicator for mica flotation evaluation is K_2O. From Figure 25, it can be concluded that in the mica rougher flotation, after 1 minute, about 30% of the K_2O deported to the concentrate. In the scavenger flotation another 5% of K_2O was recovered. This was at the expense of 4% Li deportation to the mica concentrate. A large portion of the lithium, however, can be recovered in the cleaning stages.

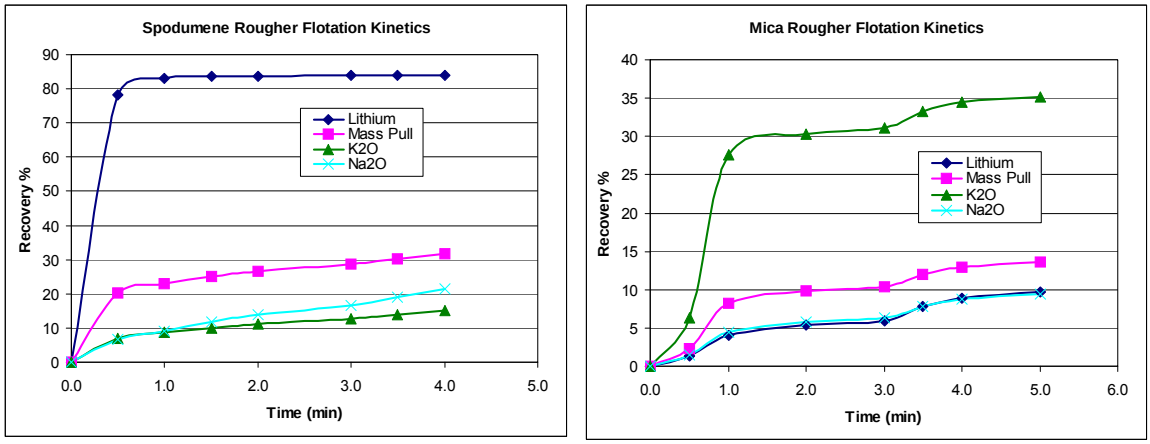


Figure 25: Mica and Spodumene Flotation Kinetics

4.5. Locked Cycle Test

One locked cycle flotation test was conducted on the composite head sample. This locked cycle test was mainly conducted to produce spodumene concentrate for hydrometallurgy testwork. It was conducted after the initial tests, and the conditions and the flowsheet had not been optimised at that time. The flowsheet for the locked cycle test is illustrated in Figure 26. The feed charge, -6 mesh, was stage ground to -48 mesh followed by a desliming process. Scrubbing was performed on the combined mica rougher and cleaner tailings. A second stage of de-sliming was then performed on the scrubbed material followed by high density conditioning at ambient temperature (22ºC). After rougher flotation, the rougher tailing was conditioned at high density with additional collector. Rougher and rougher scavenger concentrates were combined and cleaned in three cleaning stages. The scavenger tailings were considered as final tailings. The cleaner tailings were re-circulated to the previous stage as illustrated in the Figure 26. Seven cycles were completed in order to ensure stability and to generate adequate concentrate. LR19 was used as the collector (600 g/t in rougher and 300 g/t in scavenger) and D618 (250 g/t) as the dispersion agent in the scrubbing stage after adjusting pulp pH to about 10.5 to 11.

The locked cycle test details are illustrated in the Appendix E. Figure 27 demonstrates good stability of the locked cycle test for both mass and lithium inflow and outflow early in the test. The projected mass balance is indicated in Table 19. Conclusions from this LCT testing are as follows:

- After three cleaners, the final concentrate grade was 6.15% Li_2O at a recovery of 81.5%. The scavenger tailings grade was 0.07% Li_2O with a lithium distribution of 6.36%;
- The major Li loss was in the spodumene scavenger tail, at 6.36%. The Li loss in the mica flotation was 5.14% and in the cyclone overflow was 2.47%;
- Three slimes were generated as cyclone overflow, decantation before mica flotation, and after scrubbing the spodumene flotation feed. The lithium loss in the cyclone overflow and 1st and 2nd slimes were 2.47%, 2.90% and 1.59%, respectively. In total, 7% of the lithium deported to the slimes;

- The grinding fineness (K_{80}) of the rougher scavenger tailings was 190 µm. The cyclone overflow was 100% less than 20 µm.

As it was stated before, there is little improvement in the Li recovery by collecting scavenger concentrate. Furthermore, only one mica cleaner flotation might be sufficient. Thus, the flowsheet can be simplified considering these points.

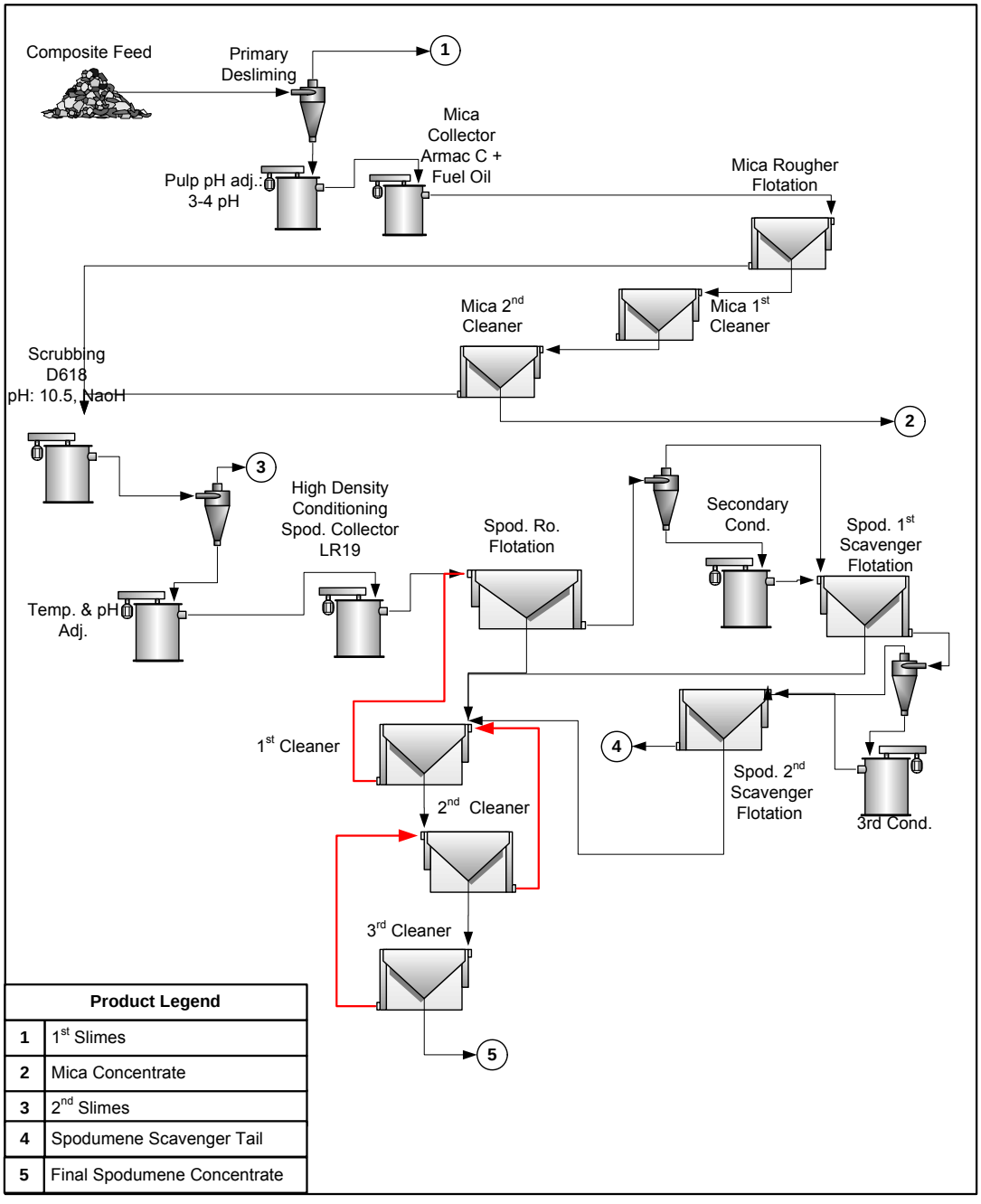


Figure 26: Locked Cycle Test Flowsheet

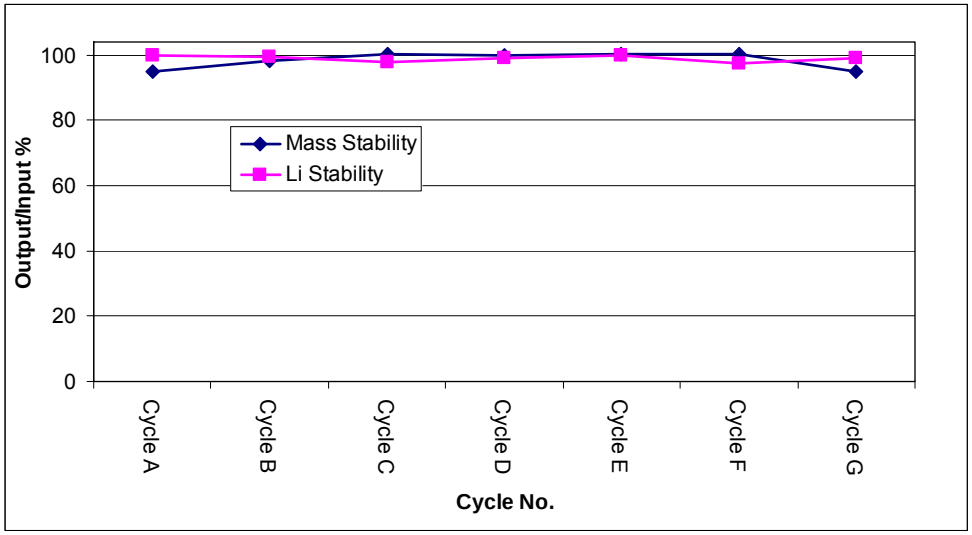


Figure 27: Locked Cycle Stability Plot

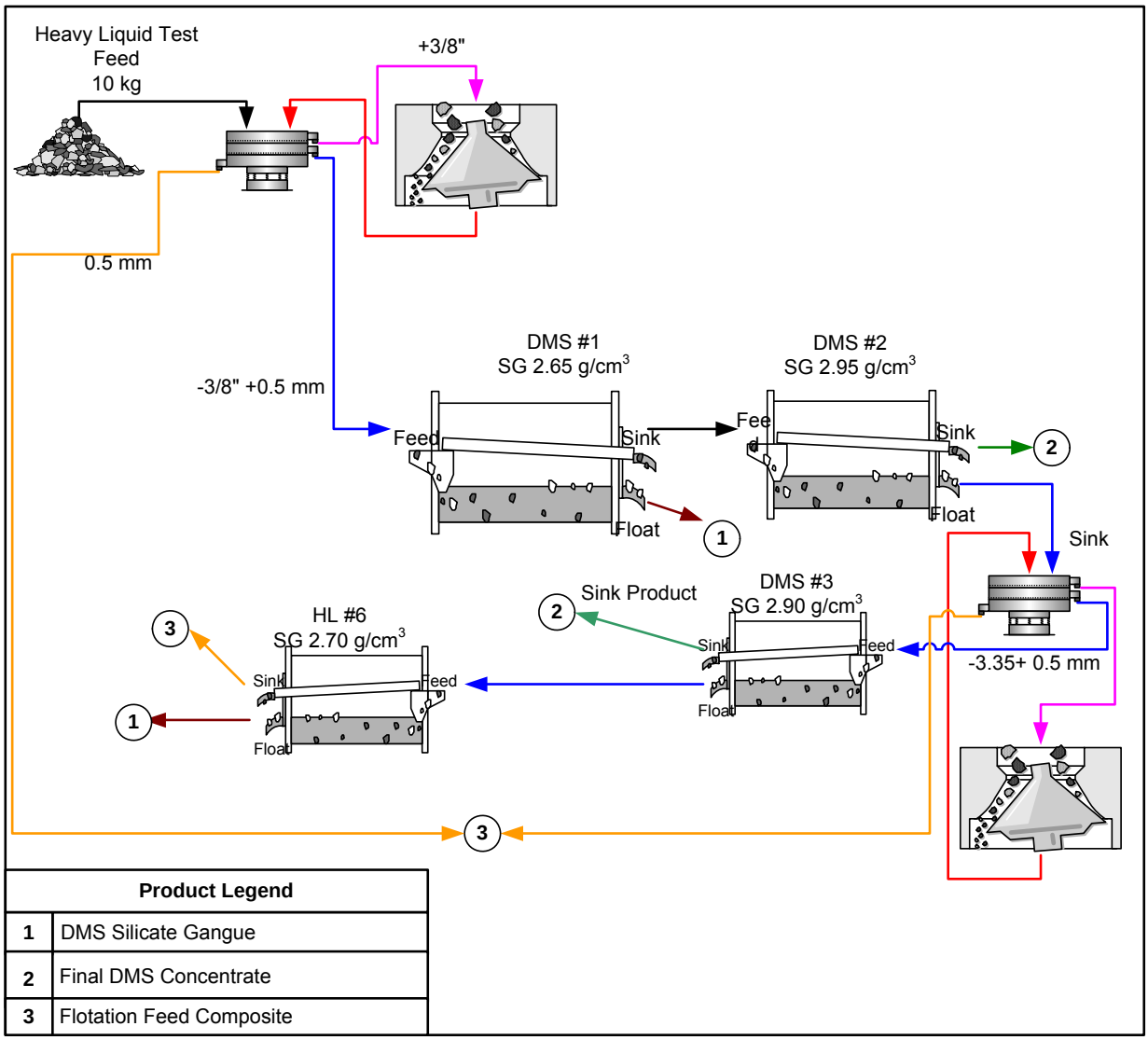
Table 19: Projected Mass Balance Based on Cycles D to H

Projected Balance Cycles D to H															
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 3rd Cl Conc	4,953	20.1	2.86	6.15	63.0	25.7	1.14	0.92	1.50	81.5	17.2	32.4	11.3	4.77	14.4
Li Scav. Tail	14,939	60.7	0.07	0.16	81.6	11.2	1.60	5.07	1.60	6.36	67.1	42.7	48.0	79.1	46.4
Slime 2	417	1.69	0.66	1.43	71.1	17.3	2.49	3.92	2.49	1.59	1.63	1.84	2.08	1.71	2.01
Mica Conce.	2,266	9.20	0.39	0.85	59.0	26.4	6.01	3.01	6.01	5.14	7.36	15.2	27.3	7.11	26.4
Slime 1	934	3.79	0.54	1.16	71.8	17.0	2.62	4.38	2.62	2.90	3.69	4.06	4.91	4.27	4.73
Cyclone Overflow	819	3.32	0.52	1.13	67.2	18.1	3.87	3.61	3.87	2.47	3.03	3.77	6.35	3.08	6.13
Total Feed	24,327	98.8	0.70	1.52	73.7	15.9	2.02	3.89	2.10	100	100	100	100	100	100

4.5.1. Recommended Flowsheet

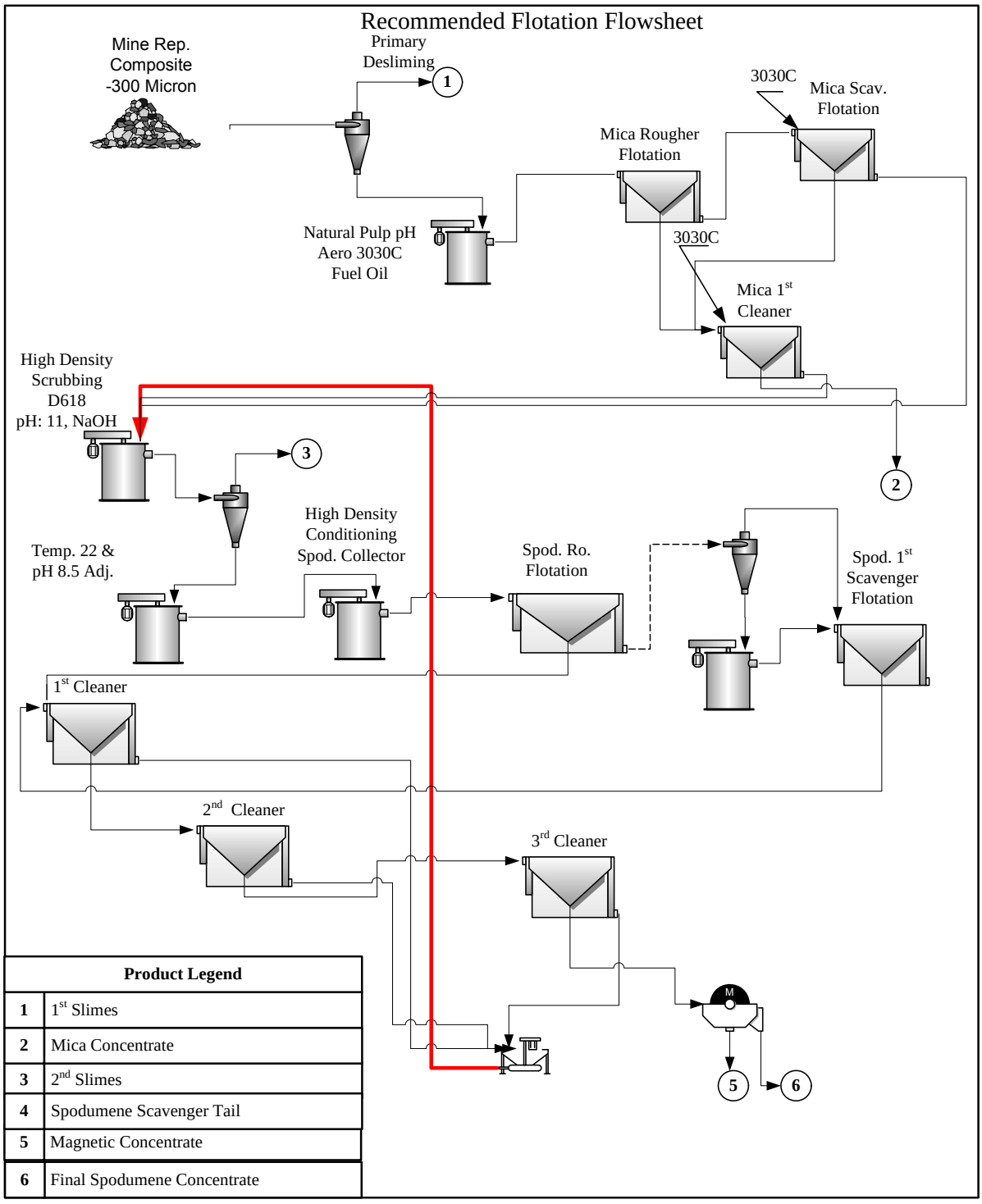
Based on the results of the heavy liquid and flotation tests, the flowsheets illustrated in, Figure 28 and Figure 29 are recommended. These flowsheets should be confirmed with new representative sample(s) to be taken from the deposit. Due to the attractive results of the heavy liquid tests, it is suggested to couple these flowsheets. In other words, dense media separation (DMS) is expected to be a basic part of the process. The DMS middlings and undersize fractions may be combined and further processed in a flotation plant to increase the total lithium recovery.

In the flotation flowsheet, it might not be necessary to do the lithium scavenger flotation if the recovery of the rougher flotation is adequate. Furthermore, the first cleaner tail might be discarded if the lithium assay is low in this stream.



Product Legend	
1	DMS Silicate Gangue
2	Final DMS Concentrate
3	Flotation Feed Composite

Figure 28: Recommended DMS Flowsheet



Product Legend	
1	1st Slimes
2	Mica Concentrate
3	2nd Slimes
4	Spodumene Scavenger Tail
5	Magnetic Concentrate
6	Final Spodumene Concentrate

Figure 29: Recommended Flotation Flowsheet

Conclusions and Recommendations

Metallurgical testwork was conducted on a drill core composite sample taken from the Georgia Lake Property. Conclusions of the testwork and recommendations are outlined below.

Heavy Liquid Testwork

- By performing primary crushing at -3/8" in the first pass and secondary crushing at -6 mesh and separating the -0.5 mm fraction prior to heavy liquid separation at a density of 2.65 to 2.70 g/cm^3, it was possible to reject more than 43% of the mass with only 1 to 4% lithium losses;
- A lithium concentrate was produced by further processing the sink product at a heavy liquid density of 2.90 to 3.00 g/cm^3. With the exception of the low grade sample (head grade of 0.75% Li_2O), the grade of the final lithium concentrate was above 6.0% Li_2O at approximately 75% recovery;
- In case of the low grade sample, it was possible to reject 68% of the mass with lithium losses of 4.5%. The final concentrate graded 6.29% Li_2O at 60.8% lithium recovery;
- The undersize fraction and the heavy liquid separation middling were combined and further processed by flotation. Depending on the sample head grade and target for the mass rejection and its acceptable lithium loss, the flotation feed might have a lower grade than the original sample. Flotation recovery, however, is expected be at least 70% or higher;
- From the heavy liquid tests conducted on different samples, it was concluded that this deposit would be amenable to dense media separation. A large portion of lithium (at least 60%) is expected to report to the DMS concentrate.

Flotation Testwork

- Primary crushing to nominal 6 mesh and stage-grinding with a closing size of 48 mesh (300 µm) were used to minimize slime generation in the flotation tests;
- Optimum flotation results were obtained with a flotation feed K_{80} of about 200 to 220 µm;
- Desliming and scrubbing are necessary to achieve high quality and reproducible flotation results. The Li loss in the slimes is a function of the slime mass fraction;
- Lignin sulfonate, D618, and NaOH additions to the scrubber were beneficial in terms of slime dispersion and separation;
- Primary mica separation is necessary since the mica content in the head sample was significant and mica deportation in the spodumene concentrate could be better controlled by separating mica ahead of the mica flotation;
- To reduce lithium losses in the mica concentrate, two cleaning stages were performed on the mica concentrate. It was found that even one cleaning stage could be sufficient and the lithium loss to the first cleaner concentrate might be acceptable. The cleaner tailings were combined with the mica rougher tailings and used as spodumene flotation feed. Normally about 3 to 5% of the lithium deported to the mica concentrate. A minor amount of lithium is expected to be in the mica crystal structure as solid solution;
- A flotation concentrate with 6.15% Li_2O grade at a recovery of 81.5% Li was achieved in the locked cycle test which was executed in the early stage of flotation program to generate concentrate for hydrometallurgy testwork. Better results are expected using the final developed flotation flowsheet and conditions;

- The best collector combination for spodumene flotation was LR19, LR19B and FA-2. It was added to the pulp in normally two stages, at 600 g/t in the rougher flotation and 200 g/t in the rougher scavenger flotation. Lignin sulfonate (Marasperse) was added to the scrubber at 250 g/t and pH was adjusted with NaOH to 10.5 to 11.0;
- Soda ash was used as the pH regulator in spodumene flotation. Pulp pH was kept in the range of 8.0 to 8.5 during the rougher and cleaner flotation stages;
- From the limited fractional size analysis performed on the flotation tailings, it was concluded that the main losses of spodumene were in the coarse fractions.

Recommendations:

- More detailed mineralogy on the selected spodumene head sample should be conducted;
- More variability studies looking at samples covering a range in head grade and location to identify the effect of lithium head grade and mineralogy on overall DMS and flotation performance should be conducted;
- Confirmatory heavy liquid tests on the main composite (to be used for DMS pilot plant) should be conducted;
- A DMS pilot plant should be conducted on the mine representative sample(s) to generate engineering data for construction of a large scale DMS plant and to generate enough concentrate for hydrometallurgy batch and pilot plant testwork;
- Confirmatory flotation tests on the main composite sample and combined heavy liquid middlings and undersize fractions should be conducted;
- Further locked cycle tests with LR19 and FA-2 collectors and mica flotation in natural pH should be examined;
- The effect of in-situ mine water on flotation performance should be examined;
- The nature of spodumene losses in different fractions of the tailings, and mica concentrate should be identified;
- Performing feldspar bulk flotation on the spodumene tailings with the objective to generate feldspar and quartz concentrates.

Appendix A – Mineralogy

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Rock Tech Lithium Inc*
Project Number/ LIMS No.	*12607-001/MI4520-AUG11*
Reporting Date:	*September 2, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
Spod Head Composite	plagioclase, quartz	spodumene *(Monoclinic)*	potassium-feldspar, mica	-

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

Mineral	Composition
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	$LiAlSi_2O_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	Spod Head Composite (wt %)
Quartz	32.4
Albite	34.4
Spodumene (Monoclinic)	19.0
Microcline	7.10
Muscovite	7.20
TOTAL	100

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.

Chemical Balance

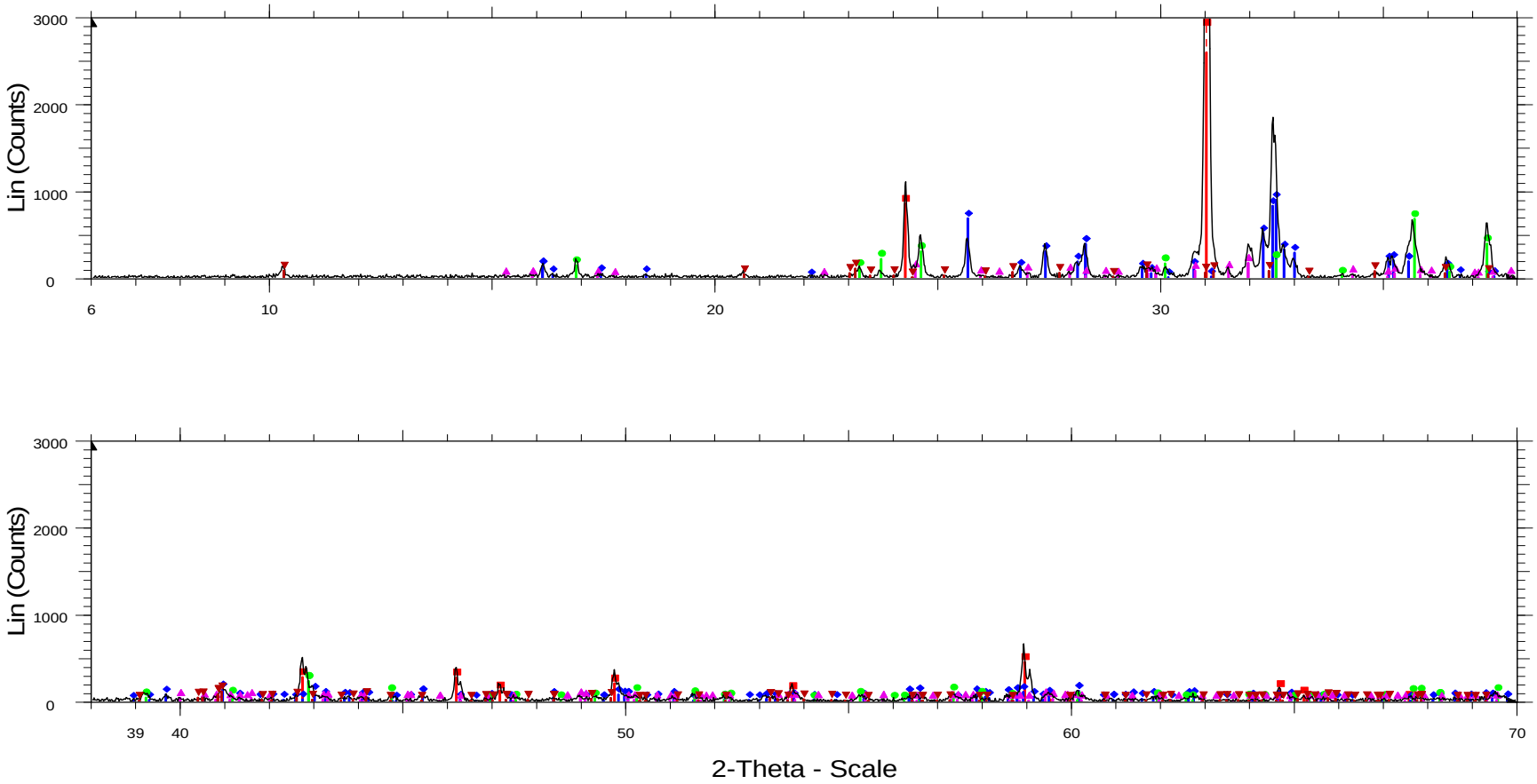
Spod Head Composite

Name	Assay[1]	SQD[2]	Delta	Status
SiO2	75.6	76.1	-0.52	Both
Al2O3	16.0	15.9	0.07	Both
Na2O	3.92	4.06	-0.14	Both
K2O	2.01	2.04	-0.03	Both
Li2O	1.49	1.53	-0.04	Both
Fe2O3	0.48	-	0.48	XRF
CaO	0.26	-	0.26	XRF
P2O5	0.16	-	0.16	XRF
MgO	0.11	-	0.11	XRF
MnO	0.08	-	0.08	XRF
H2O	-	0.32	0.32	SQD

1. Values measured by chemical assay.

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

Spod Head Composite



Appendix B – Bond Ball Work Indices

SGS Minerals Services

Standard Bond Rod Mill Grindability Test

Project No.: 12607-001 Date: 30-Mar-11
Sample.: spod -3/8 head comp

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):	2,519	grams
Equivalent to :	2,015	kg/m³ at Minus 6 mesh
Weight % of the undersize material in the rod mill feed:		17.9%
Weight of undersize product for 100% circulating load:		1,260 grams

Results: Gram per Rev Average for the Last Three Stages = **12.51 g**
Circulation load = **107%**

CALCULATION OF A BOND WORK INDEX

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

P1 = 100% passing size of the product	1,180	microns
Grp = Grams per revolution	12.51	grams
P_{80} = 80% passing size of product	966	microns
F_{80} = 80% passing size of the feed	9,852	microns

RWI = **11.4** kWh/t (imperial)

RWI = **12.5** 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	2,519	450	810	992	542	10.84
2	100	992	177	1,082	1,435	1,258	12.58
3	80	1,435	256	1,003	1,318	1,062	13.27
4	77	1,318	235	1,024	1,240	1,005	13.05
5	80	1,240	221	1,038	1,354	1,133	14.16
6	72	1,354	242	1,018	1,142	900	12.50
7	84	1,142	204	1,056	1,268	1,064	12.67
8	82	1,268	226	1,033	1,239	1,013	12.35
		Average for Last Three Stages =			1,216 g		12.51 g

SGS Minerals Services

Standard Bond Rod Mill Grindability Test

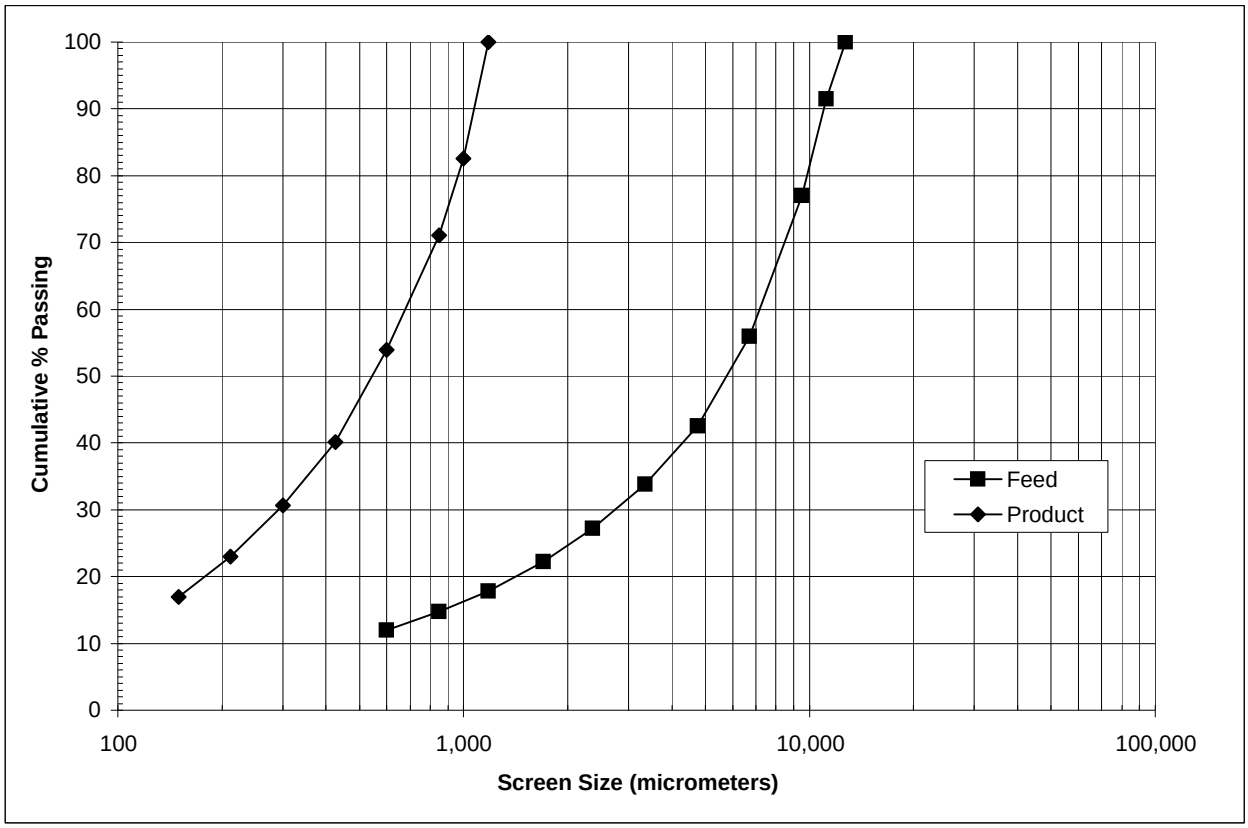
Project No.: 12607-001 Date: 30-Mar-11
Sample.: spod -3/8 head comp

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	115.8	8.53	8.53	91.5
3/8"	9,500	196.2	14.5	23.0	77.0
3	6,700	286.0	21.1	44.1	55.9
4	4,750	181.5	13.4	57.4	42.6
6	3,350	118.6	8.74	66.2	33.8
8	2,360	90.2	6.65	72.8	27.2
10	1,700	66.9	4.93	77.7	22.3
14	1,180	59.8	4.41	82.1	17.9
18	1,000	-	-	-	-
20	850	42.4	3.12	85.3	14.7
28	600	37.2	2.74	88.0	12.0
35	425				
48	300				
65	212				
100	150				
Pan		162.8	12.0	100.0	-
Total	**-**	**1357.4**	**100.0**	**F_{80}:**	**9,852**

Product Particle Size Analysis

Weight	% Retained		% Passing
grams	Individual	Cumulative	Cumulative
0.00	0.00	0.00	100.0
43.6	17.4	17.4	82.6
28.7	11.5	28.9	71.1
42.8	17.1	46.0	54.0
34.5	13.8	59.8	40.2
23.8	9.52	69.4	30.6
19.1	7.64	77.0	23.0
15.0	6.00	83.0	17.0
42.5	17.0	100.0	-
250.0	**100.0**	**P_{80}:**	**966**



Appendix C – HL Testwork

All the calculations are based on the Head Sample
Combined Heavy Liquid Test Results with Undersize Fractions

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8" + 0.5mm Fraction, Float 2.65 SG	2.65	3,269	33.3	0.02	0.04	75.5	14.3	0.16	6.08	3.16	0.27	0.77	33.9	29.2	10.8	49.5	51.4	33.3
-2 + 0.5mm Fraction, Float 2.65 SG	2.65	934	9.50	0.01	0.03	80.5	11.9	0.15	6.47	0.82	0.22	0.16	10.3	6.95	2.89	15.0	3.81	7.76
Float 2.90 SG	2.90	1,425	14.5	0.32	0.69	76.8	14.6	0.76	2.32	2.65	0.28	6.33	15.0	13.0	22.3	8.23	18.8	15.1
Sink 2.90 SG, Non Mag	2.90	69.9	0.71	2.25	4.84	67.1	21.9	0.86	1.90	1.36	0.42	2.18	0.64	0.96	1.24	0.33	0.47	1.11
Sink 2.90 SG, Mag & Middlings	2.95	2.20	0.02	0.94	2.02	55.1	21.4	3.94	2.20	2.92	3.25	0.03	0.02	0.03	0.18	0.01	0.03	0.27
Sink 2.95 SG	2.95	107	1.08	2.44	5.25	65.8	23.1	0.93	1.45	1.19	0.42	3.61	0.96	1.54	2.04	0.38	0.63	1.69
Sink 3.00 SG, Non Mag	3.00	1,268	12.9	3.17	6.82	63.8	26.4	0.91	0.45	0.45	0.16	55.8	11.1	20.9	23.8	1.42	2.84	7.66
Sink 3.00 SG, Middlings	3.00	10.4	0.11	2.93	6.31	60.1	25.1	2.28	0.63	0.95	1.35	0.42	0.09	0.16	0.49	0.02	0.05	0.53
Sink 3.00 SG, Mag	3.00	5.00	0.05	0.94	2.02	42.9	16.9	12.4	1.08	0.96	2.50	0.07	0.03	0.05	1.28	0.01	0.02	0.47
2nd -0.5 mm Fraction		1,630	16.6	1.06	2.28	73.4	16.5	0.64	3.33	1.23	0.31	24.0	16.4	16.8	21.5	13.5	9.98	19.1
1st -0.5 mm Fraction		1,109	11.3	0.43	0.93	75.3	14.9	0.59	4.18	2.16	0.31	6.62	11.5	10.3	13.5	11.5	11.9	13.0
Global Feed (Calc.)		9,828	100	0.73	1.58	74.1	16.3	0.49	4.09	2.04	0.27	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Silicate Gangue (-3/8+ 0.5 mm Float 2.65 SG)	2.65	4,203	42.8	0.02	0.03	76.6	13.8	0.16	6.17	2.64	0.26	0.93	44.2	36.2	13.7	64.5	55.2	41.1
Final HL Spodumene Concentrate	2.90	1,462	14.9	3.06	6.59	64.0	25.9	0.96	0.60	0.56	0.21	62.1	12.8	23.7	29.0	2.18	4.05	11.7
Flotation Feed		4,164	42.4	0.64	1.38	75.1	15.4	0.67	3.21	1.96	0.30	36.9	42.9	40.2	57.3	33.3	40.7	47.2
Feed (Calc.)		9,828	100	0.73	1.58	74.1	16.3	0.49	4.09	2.04	0.27	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Final HL Spodumene Concentrate -Non Mag		1,444	14.7	3.07	6.61	64.1	25.9	0.91	0.59	0.55	0.19	61.6	12.7	23.4	27.1	2.14	3.94	10.5
Final HL Spodumene Concentrate -Mag		17.6	0.18	2.12	4.55	54.6	22.3	5.36	0.95	1.20	1.91	0.52	0.13	0.25	1.95	0.04	0.11	1.27
Final HL Spodumene Concentrate		1,462	14.9	3.06	6.59	64.0	25.9	0.96	0.60	0.56	0.21	62.1	12.8	23.7	29.0	2.18	4.05	11.7

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	1,425	34.2	0.32	0.69	76.8	14.6	0.76	2.32	2.65	0.28	17.1	17.1	35.0	32.4	38.9	24.7	46.2
2nd -0.5 mm Fraction	1,630	39.1	1.06	2.28	73.4	16.5	0.64	3.33	1.23	0.31	64.9	64.9	38.3	41.9	37.5	40.6	24.5
1st -0.5 mm Fraction	1,109	26.6	0.43	0.93	75.3	14.9	0.59	4.18	2.16	0.31	17.9	17.9	26.7	25.7	23.5	34.7	29.3
Flotation Feed (Calc.)	4,164	100	0.64	1.38	75.1	15.4	0.67	3.21	1.96	0.30	100	100	100	100	100	100	100

All the calculations are based on the Head Sample
Combined Heavy Liquid Test Results with Undersize Fractions

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8" + 0.5mm, 2.65 SG Float	2.65	3,284	33.1	0.02	0.04	75.9	14.0	0.15	5.81	3.07	0.27	0.78	33.8	28.6	8.15	48.6	49.4	32.3
-3.35 + 0.5mm, 2.65 SG Float	2.65	1,048	10.6	0.02	0.05	82.1	11.0	0.27	5.71	0.87	0.22	0.32	11.7	7.18	4.68	15.2	4.47	8.41
Sink 2.65 SG	2.65	1,869	18.8	0.32	0.69	75.7	15.3	0.78	3.05	2.53	0.29	8.31	19.2	17.8	24.1	14.5	23.2	19.8
Sink 2.85 SG	2.85	127	1.28	1.57	3.38	68.5	20.1	0.94	2.10	1.68	0.43	2.76	1.18	1.59	1.97	0.68	1.04	1.99
Sink 2.90 SG	2.90	175	1.76	2.01	4.33	68.2	21.0	0.90	1.78	1.16	0.40	4.88	1.61	2.28	2.60	0.79	0.99	2.55
Sink 2.95 SG	2.95	195	1.96	2.42	5.21	66.7	22.6	0.91	1.30	0.96	0.32	6.56	1.76	2.74	2.94	0.65	0.92	2.28
Sink 3.00 SG	3.00	1,386	14.0	3.18	6.85	63.9	25.6	1.16	0.48	0.45	0.19	61.3	12.0	22.1	26.6	1.70	3.06	9.61
2nd -0.5 mm Fraction		700	7.05	0.85	1.83	73.6	15.8	1.61	3.38	1.35	0.41	8.27	6.98	6.89	18.6	6.03	4.63	10.5
1st -0.5mm Fraction		1,150	11.6	0.43	0.93	75.9	15.1	0.54	4.02	2.17	0.30	6.87	11.8	10.8	10.3	11.8	12.2	12.6
Global Feed (Calc.)		9,934	100	0.72	1.56	74.3	16.2	0.61	3.95	2.05	0.28	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	4,332	43.6	0.02	0.04	77.4	13.3	0.18	5.79	2.54	0.26	1.10	45.4	35.8	12.8	63.9	53.9	40.8
Final HL Spodumene Concentrate	2.85	1,883	19.0	2.88	6.21	64.9	24.5	1.10	0.79	0.65	0.24	75.5	16.6	28.7	34.1	3.81	6.02	16.4
Flotation Feed		3,719	37.4	0.45	0.98	75.4	15.3	0.86	3.41	2.20	0.32	23.4	38.0	35.5	53.0	32.3	40.1	42.8
Feed (Calc.)		9,934	100	0.72	1.56	74.3	16.2	0.61	3.95	2.05	0.28	100	100	100	100	100	100	100

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	1,869	50.3	0.32	0.69	75.7	15.3	0.78	3.05	2.53	0.29	35.4	35.4	50.5	50.1	45.5	44.9	57.9
2nd -0.5 mm Fraction	700	18.8	0.85	1.83	73.6	15.8	1.61	3.38	1.35	0.41	35.3	35.3	18.4	19.4	35.2	18.6	11.6
1st -0.5mm Fraction	1,150	30.9	0.43	0.93	75.9	15.1	0.54	4.02	2.17	0.30	29.3	29.3	31.1	30.5	19.4	36.4	30.6
Flotation Feed (Calc.)	3,719	100	0.45	0.98	75.4	15.3	0.86	3.41	2.20	0.32	100	100	100	100	100	100	100

All the calculations are based on the Head Sample
Combined Heavy Liquid Test Results with Undersize Fractions 2.8706

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Float 2.65 SG, 1st pass	2.65	3,374	34.2	0.03	0.07	76.1	14.2	0.20	6.05	2.73	0.27	1.45	35.2	28.9	14.1	50.8	48.0	31.7
Sink 2.65 SG, 1st pass	2.65	968	9.80	0.24	0.52	78.7	13.3	0.47	4.41	1.54	0.34	2.93	10.4	7.77	9.48	10.6	7.76	11.5
Float 2.65 SG, 2nd pass	2.65	483	4.89	0.02	0.04	81.7	11.4	0.14	5.97	0.69	0.22	0.12	5.40	3.32	1.41	7.17	1.73	3.69
Sink 2.65 SG, 2nd pass	2.65	336	3.40	0.14	0.30	84.3	9.69	0.39	3.40	0.93	0.22	0.59	3.87	1.96	2.73	2.84	1.62	2.57
Sink 2.70 SG, 2nd pass	2.70	715	7.24	0.75	1.61	71.0	19.1	0.79	3.01	2.57	0.39	6.75	6.95	8.24	11.8	5.36	9.56	9.70
Sink 2.85 SG, 2nd pass	2.85	166	1.68	1.64	3.53	69.7	20.4	0.79	2.21	1.57	0.40	3.42	1.58	2.04	2.73	0.91	1.35	2.31
Sink 2.90 SG, 2nd pass	2.90	205	2.08	1.90	4.09	69.7	21.4	0.76	1.79	1.12	0.37	4.91	1.96	2.65	3.25	0.91	1.20	2.64
Sink 2.95 SG, 2nd pass	2.95	158	1.60	2.48	5.34	68.0	23.1	0.76	1.39	0.99	0.32	4.93	1.47	2.20	2.50	0.55	0.81	1.76
Sink 3.00 SG, 2nd pass	3.00	549	5.56	3.03	6.52	65.2	25.7	0.85	0.62	0.56	0.19	21.0	4.90	8.52	9.73	0.85	1.60	3.63
Sink 2.95 SG, 1st pass	2.95	351	3.55	2.34	5.04	69.5	21.5	0.73	1.16	0.75	0.24	10.3	3.34	4.56	5.34	1.01	1.37	2.93
Sink 3.00 SG, 1st pass	3.00	934	9.46	3.07	6.61	65.0	25.0	0.85	0.61	0.50	0.17	36.1	8.32	14.1	16.5	1.42	2.43	5.53
2nd pass-0.5mm Spod. Out Crop 1		443	4.49	0.80	1.72	76.4	15.6	0.57	3.22	1.32	0.30	4.47	4.64	4.17	5.27	3.55	3.05	4.63
1st pass -0.5mm Spod. Out Crop 1		1,195	12.1	0.20	0.43	73.1	16.0	0.61	4.69	3.14	0.42	3.01	12.0	11.5	15.2	14.0	19.5	17.5
Global Feed (Calc.)		9,876	100	0.80	1.73	73.9	16.8	0.49	4.07	1.94	0.29	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.70, Silicate Gangue, 1st Pass	2.70	4,342	44.0	0.08	0.17	76.7	14.0	0.26	5.68	2.46	0.29	4.37	45.6	36.7	23.5	61.5	55.7	43.2
-3/8+ 0.5 mm Float 2.65, Silicate Gangue, 2nd Pass	2.70	818	8.28	0.07	0.15	82.8	10.7	0.24	4.92	0.79	0.22	0.71	9.27	5.28	4.14	10.0	3.36	6.26
Final HL Spod Conc. 1st Pass	2.95	1,285	13.0	2.87	6.18	66.2	24.0	0.82	0.76	0.57	0.19	46.5	11.7	18.7	21.9	2.43	3.80	8.46
Final HL Spod Conc. 2nd Pass	2.90	912	9.24	2.68	5.77	66.7	24.3	0.81	1.02	0.76	0.25	30.8	8.33	13.4	15.5	2.31	3.61	8.03
Combined HL Concentrates		2,197	22.2	2.79	6.01	66.4	24.1	0.82	0.87	0.65	0.22	77.3	20.0	32.0	37.4	4.74	7.41	16.5
Flotation Feed		2,518	25.5	0.56	1.20	72.9	17.1	0.67	3.79	2.55	0.39	17.7	25.1	26.0	35.0	23.8	33.5	34.1
Feed (Calc.)		9,876	122	0.80	1.73	73.9	16.8	0.49	4.07	1.94	0.29	100	100	100	100	100	100	100

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	880	8.92	0.92	1.97	70.8	19.3	0.79	2.86	2.38	0.39	10.2	8.53	10.3	14.5	6.27	10.9	12.0
1st pass -0.5mm Spod. Out Crop 1	1,195	12.1	0.20	0.43	73.1	16.0	0.61	4.69	3.14	0.42	3.01	12.0	11.5	15.2	14.0	19.5	17.5
2nd pass-0.5mm Spod. Out Crop 1	443	4.49	0.80	1.72	76.4	15.6	0.57	3.22	1.32	0.30	4.47	4.64	4.17	5.27	3.55	3.05	4.63
Flotation Feed (Calc.)	2,518	25.5	0.56	1.20	72.9	17.1	0.67	3.79	2.55	0.39	17.7	25.1	26.0	35.0	23.8	33.5	34.1

All the calculations are based on the Head Sample
Combined Heavy Liquid Test Results with Undersize Fractions 41.94

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8"+0.5 mm, HLS Float 2.65 SG	2.65	4,424	48.8	0.02	0.03	74.6	14.2	0.20	6.06	3.01	0.27	2.10	49.1	44.5	21.3	61.8	57.9	47.6
-6M +0.5 mm, HL Float 2.65 SG	2.65	608	6.72	0.01	0.03	77.2	13.9	0.17	6.96	1.19	0.26	0.27	6.99	6.00	2.48	9.76	3.15	6.31
HL Sink 2.65 SG	2.65	1,143	12.6	0.06	0.13	82.8	10.5	0.43	3.47	1.47	0.22	2.17	14.1	8.51	11.8	9.14	7.31	10.0
HL Sink 2.70 SG	2.70	686	7.58	0.42	0.90	63.1	23.7	1.38	2.44	5.13	0.36	9.14	6.44	11.5	22.7	3.86	15.3	9.86
HL Sink 2.85 SG	2.85	85.8	0.95	1.37	2.95	66.2	22.5	1.20	1.95	2.89	0.38	3.73	0.84	1.37	2.47	0.39	1.08	1.30
HL Sink 2.90 SG	2.90	74.2	0.82	1.99	4.28	68.0	22.0	1.03	1.71	1.61	0.34	4.68	0.75	1.16	1.84	0.29	0.52	1.01
HL Sink 2.95 SG	2.95	81.4	0.90	2.37	5.10	66.6	23.3	1.03	1.31	1.30	0.31	6.11	0.81	1.34	2.01	0.25	0.46	1.01
HL Sink 3.00 SG	3.00	501	5.53	3.15	6.78	64.0	26.1	1.15	0.49	0.65	0.21	50.0	4.77	9.27	13.8	0.57	1.42	4.20
-0.5mm 1st Pass		845	9.33	0.41	0.88	76.2	15.0	0.50	4.45	1.96	0.33	11.0	9.58	8.99	10.1	8.67	7.20	11.1
-0.5mm 2nd Pass		608	6.71	0.56	1.21	73.5	16.9	0.78	3.76	2.13	0.31	10.8	6.65	7.29	11.4	5.27	5.63	7.52
Global Feed (Calc.)		9,057	100	0.35	0.75	74.2	15.6	0.46	4.79	2.54	0.28	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.65	6,175	68.2	0.02	0.05	76.4	13.5	0.24	5.67	2.55	0.26	4.55	70.2	59.0	35.5	80.7	68.4	64.0
Final HL Spodumene Concentrate	2.90	657	7.25	2.92	6.29	64.8	25.3	1.12	0.73	0.84	0.24	60.8	6.33	11.8	17.7	1.10	2.40	6.21
Flotation Feed		2,225	24.6	0.49	1.06	71.0	18.5	0.87	3.55	3.02	0.34	34.6	23.5	29.2	46.8	18.2	29.2	29.8
Feed (Calc.)		9,057	100	0.35	0.75	74.2	15.6	0.46	4.79	2.54	0.28	100	100	100	100	100	100	100

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
HL Sink 2.70 SG	686	30.8	0.42	0.90	63.1	23.7	1.38	2.44	5.13	0.36	26.4	26.4	27.4	39.5	48.6	21.2	52.4
HL Sink 2.85 SG	85.8	3.86	1.37	2.95	66.2	22.5	1.20	1.95	2.89	0.38	10.8	10.8	3.59	4.69	5.29	2.12	3.69
-0.5mm 1st Pass	845	38.0	0.41	0.88	76.2	15.0	0.50	4.45	1.96	0.33	31.7	31.7	40.7	30.8	21.7	47.7	24.6
-0.5mm 2nd Pass	608	27.3	0.56	1.21	73.5	16.9	0.78	3.76	2.13	0.31	31.2	31.2	28.3	25.0	24.4	29.0	19.3
Flotation Feed (Calc.)	2,225	100	0.49	1.06	71.0	18.5	0.87	3.55	3.02	0.34	100	100	100	100	100	100	100

All the calculations are based on the Head Sample
Combined Heavy Liquid Test Results with Undersize Fractions

Products	Heavy Liquid	Weight		Assays %								Distribution %						
Sample ID	Density g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Float 2.65 SG	2.65	2,718	35.7	0.02	0.05	78.6	12.7	0.17	5.78	2.16	0.22	0.82	37.7	27.2	10.8	57.4	46.3	31.5
Sink 2.65 SG	2.65	769	10.1	0.13	0.28	84.7	9.16	0.32	3.12	1.00	0.25	1.44	11.5	5.54	5.77	8.76	6.06	10.1
Sink 2.70 SG	2.70	855	11.2	0.65	1.40	68.5	20.1	0.98	2.94	3.19	0.36	8.02	10.3	13.5	19.6	9.18	21.5	16.2
Sink 2.85 SG	2.85	140	1.84	1.67	3.59	69.5	20.7	0.89	2.06	1.60	0.41	3.38	1.72	2.28	2.92	1.05	1.77	3.02
Sink 2.90 SG	2.90	149	1.96	2.11	4.54	69.2	20.9	0.84	1.69	0.87	0.33	4.55	1.83	2.46	2.94	0.92	1.02	2.60
Sink 2.95 SG	2.95	157	2.06	2.45	5.27	68.3	22.5	0.88	1.25	0.68	0.31	5.55	1.89	2.78	3.24	0.72	0.84	2.56
Sink 3.00 SG	3.00	1,424	18.7	3.09	6.65	64.7	26.2	1.00	0.50	0.35	0.16	63.5	16.3	29.4	33.4	2.60	3.93	12.0
HL Test 5 -35M Fraction		1,395	18.3	0.63	1.36	76.1	15.4	0.65	3.81	1.69	0.30	12.7	18.7	16.9	21.3	19.4	18.6	22.0
Global Feed (Calc.)		7,606	100	0.91	1.96	74.5	16.7	0.56	3.60	1.67	0.25	100	100	100	100	100	100	100

Products	Heavy Liquid	Product		Assay %								Distribution %						
Sample ID	Density g/cm^3	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
-3/8+ 0.5 mm Float 2.65, Silicate Gangue	2.70	3,486	45.8	0.05	0.10	79.9	11.9	0.20	5.19	1.90	0.23	2.27	49.2	32.7	16.6	66.1	52.3	41.6
Final HL Spodumene Concentrate	2.90	1,730	22.7	2.95	6.34	65.4	25.4	0.98	0.67	0.42	0.19	73.6	20.0	34.6	39.6	4.24	5.80	17.2
Flotation Feed		2,390	31.4	0.70	1.50	73.0	17.4	0.78	3.40	2.22	0.33	24.1	30.8	32.7	43.8	29.6	41.9	41.3
Feed (Calc.)		7,606	100	0.91	1.96	74.5	16.7	0.56	3.60	1.67	0.25	100	100	100	100	100	100	100

Flotation Feed Components	Weight		Assay %								Distribution %						
Sample ID	wt g	wt %	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO	Li	SiO_2	Al_2O_3	Fe_2O_3	Na_2O	K_2O	CaO
Heavy Liquid Middling	995	13.1	0.79	1.71	68.6	20.2	0.97	2.82	2.97	0.37	11.4	12.1	15.8	22.6	10.2	23.3	19.2
HL Test 5 -35M Fraction	1,395	18.3	0.63	1.36	76.1	15.4	0.65	3.81	1.69	0.30	12.7	18.7	16.9	21.3	19.4	18.6	22.0
Flotation Feed (Calc.)	2,390	31.4	0.70	1.50	73.0	17.4	0.78	3.40	2.22	0.33	24.1	30.8	32.7	43.8	29.6	41.9	41.3

Appendix D – Flotation Testwork

Test: F-1 **Project No.:** 12607-001 **Operator**: JG **Date:** 04-Apr-11

Purpose: To recover spodumene based on the flowsheet developed on the previous programs

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using Collector LR19

LR19: FA-2 (62 %), Aero 855 (27%), fuel oil (9%) and MIBC (2%)

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	NaF	D618	LR19	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%			5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	125	50	25									3		3.0	25
Mica Ro (4kg cell)	300											1	4	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															
Mica Clnr #1	175	10										1	3.5	3.0	20
Mica Clnr #2	125											1	3	3.0	23
1st and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						6.25				500		5		8.0	
Li Rougher 1 (4kg cell)						62.5							3	8.0	23
Li Rougher Scav 1						12.5		50	200	100			1.5	8.0	22
Li Rougher Scav 2						12.5				100			1.5	8.0	22.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	2	8.0	25
Total	1225	60	25		188	131			450	750	0	24	23		

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 WHIMS Non Mags	671	19.1	3.05	6.57	61.0	26.2	1.09	0.58	1.00	76.6	15.7	31.1	10.3	2.81	30.7
F1 WHIMS Mags	13.6	0.39	1.94	4.18	41.9	18.5	1.00	0.37	23.7	0.99	0.22	0.45	0.19	0.04	14.7
F1 Li 3rd Cl Tail	20.0	0.57	1.60	3.44	69.2	20.0	1.44	3.08	0.94	1.20	0.53	0.71	0.40	0.45	0.86
F1 Li 2nd Cl Tail	19.3	0.55	1.03	2.22	72.8	17.4	1.60	4.04	0.71	0.74	0.54	0.59	0.43	0.56	0.63
F1 Li 1st Cl Tail	125	3.55	0.17	0.37	80.9	11.8	1.66	5.13	0.16	0.79	3.87	2.60	2.90	4.62	0.91
F1 Li Ro Tail	2,009	57.3	0.12	0.26	81.7	11.2	1.62	5.11	0.10	9.03	63.0	39.9	45.7	74.2	9.18
F1 Slimes #2	52.8	1.51	0.65	1.40	69.6	18.1	2.80	3.79	1.60	1.28	1.41	1.69	2.08	1.45	3.86
F1 Mica 2nd Cl Conc	298	8.51	0.31	0.67	58.8	25.8	6.20	3.02	1.54	3.46	6.73	13.6	26.0	6.51	21.0
F1 Slimes #1	185	5.27	0.54	1.16	71.9	16.9	2.62	4.39	1.09	3.74	5.10	5.53	6.80	5.86	9.20
Cyclone OIF	114	3.26	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.17	2.96	3.79	5.26	3.49	8.98
Head (calc.)	3,508	100	0.76	1.64	74.3	16.1	2.03	3.94	0.62	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,209	91.5	0.78	1.62	69.2	13.6	1.38	3.63	0.41	97.8	97.0	96.2	94.7	96.5	91.0
Spod Flot Feed	2,858	81.5	0.84	1.80	76.5	14.9	1.49	4.00	0.44	90.6	85.2	77.0	62.0	84.1	60.8

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.	671	19.1	3.05	6.57	61.0	26.2	1.09	0.58	1.00	76.6	15.7	31.1	10.3	2.81	30.7
Li 3rd Cl Conc	685	19.5	3.03	6.52	60.6	26.0	1.09	0.58	1.45	77.6	15.9	31.6	10.5	2.85	45.4
Li 2nd Cl Conc	705	20.1	2.99	6.43	60.9	25.9	1.10	0.65	1.44	78.8	16.4	32.3	10.9	3.29	46.2
Li 1st Cl Conc	724	20.6	2.94	6.32	61.2	25.6	1.11	0.74	1.42	79.5	17.0	32.9	11.3	3.86	46.9
Li Ro Conc.	848	24.2	2.53	5.44	64.1	23.6	1.19	1.38	1.23	80.3	20.8	35.5	14.2	8.48	47.8

Test: F2 **Project No.:** 12607-001 **Operator:** JG **Date:** 04-Apr-11

Purpose: **To investigate the addition of the Li collector to the scrubber**

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using Collector LR19

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**NaF**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**			**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	500													4.0	25
Condition	125	50	25									3		3.0	25
Mica Ro (4kg cell)	0											1	4	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															
Mica Clnr #1	125	10										1	3	3.0	20
Mica Clnr #2	125											1	2	3.0	23
1st and 2nd clnr Tails to Li float															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					312.5				250	2500		10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						0				0		5		8.0	
Li Rougher 1 (4kg cell)						0							3.5	8.0	27
Li Rougher Scav 1						25		0	0	100			2	8.0	26
Li Rougher Scav 2						25				100			1	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						25				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						25						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	0.5	8.0	25
Total	875	60	25		313	113			250	2750	0	24	21		

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 WHIMS Non Mags	679	19.0	2.98	6.41	61.5	25.9	0.95	0.67	0.97	77.8	15.9	30.8	8.85	3.28	29.8
F2 WHIMS MAGS	12.1	0.34	1.78	3.83	38.7	16.4	0.61	0.40	27.9	0.83	0.18	0.35	0.10	0.03	15.3
F2 Li 3rd Cl Tail	32.3	0.91	1.80	3.87	63.8	23.5	2.21	2.41	1.16	2.23	0.78	1.33	0.98	0.56	1.69
F2 Li 2nd Cl Tail	41.1	1.15	1.17	2.52	69.9	19.1	1.93	3.77	0.80	1.85	1.09	1.37	1.09	1.12	1.49
F2 Li 1st Cl Tail	196	5.49	0.18	0.39	79.1	12.1	1.70	5.13	0.18	1.36	5.90	4.15	4.57	7.24	1.59
F2 Li Ro Tail	2,003	56.1	0.08	0.17	80.6	11.0	1.62	5.07	0.11	6.16	61.5	38.6	44.5	73.2	9.96
F2 Slimes #2	16.7	0.47	0.45	0.97	67.2	15.6	2.43	4.21	1.15	0.29	0.43	0.46	0.56	0.51	0.87
F2 Mica 2nd Cl Conc	278	7.79	0.29	0.62	56.5	27.5	7.06	2.23	1.66	3.10	5.98	13.4	26.9	4.47	20.9
F2 Slimes #1	197	5.51	0.55	1.18	71.4	16.9	2.69	4.34	1.08	4.15	5.34	5.82	7.25	6.15	9.59
Cyclone OIF	114	3.21	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.23	2.94	3.75	5.15	3.48	8.88
Head (calc.)	3,569	100	0.73	1.57	73.6	16.0	2.04	3.89	0.62	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,258	91.3	0.75	1.56	69.1	13.5	1.36	3.66	0.41	97.8	97.1	96.2	94.9	96.5	91.1
Spod Flot Feed	2,963	83.0	0.79	1.71	75.6	14.8	1.48	4.00	0.45	90.5	85.7	77.0	60.7	85.9	60.7

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.	679	19.0	2.98	6.41	61.5	25.9	0.95	0.67	0.97	77.8	15.9	30.8	8.85	3.28	29.8
Li 3rd Cl Conc	691	19.4	2.96	6.37	61.1	25.7	0.94	0.67	1.44	78.6	16.1	31.2	8.95	3.31	45.0
Li 2nd Cl Conc	724	20.3	2.91	6.26	61.2	25.6	1.00	0.74	1.43	80.9	16.9	32.5	9.93	3.88	46.7
Li 1st Cl Conc	765	21.4	2.81	6.06	61.7	25.3	1.05	0.91	1.40	82.7	18.0	33.9	11.0	4.99	48.2
Li Ro Conc.	961	26.9	2.28	4.90	65.2	22.6	1.18	1.77	1.15	84.1	23.9	38.0	15.6	12.2	49.8

Test: F-3 **Project No.:** 12607-001 **Operator:** JG **Date:** 04-Apr-11

Purpose: To repeat test F1 but replacing LR19 collector with FA-2

Procedure: As outlined below.
Feed: 4kg charge
Grind: Stage ground to -48 mesh -300 µm.
Regrind: None
Conditions: Using Collector FA-2

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	NaF	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%			5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	375													3.0	25
Condition	375	50	25									3		3.0	25
Mica Ro (4kg cell)	325											1	3	3.0	22
Mica Ro Scav	175	25	5									2	2.5	3.0	20
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															
Mica Clnr #1 (1kg cell)	375	10										1	3	3.0	20
Mica Clnr #2 (1kg cell)	75											1	2	3.0	23
Mica Clnr Scav (clnr 1+2 Tail)	175	25	10									1	1	3.0	20
1st and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 10 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						37.5				500		5		8.0	
Li Rougher 1 (4kg cell)						187.5							2.5	8.0	27
Li Rougher Scav 1						12.5		0	200	100			1.5	8.0	26
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2	8.0	21
Li 2nd Cleaner (2kg cell)						25						1	2	8.0	24
Total	1875	110	40		188	288			450	750	0	26	21		

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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 WHIMS Non-Mags	577	16.1	3.23	6.95	63.2	26.6	0.43	0.48	1.01	69.6	13.7	26.3	3.52	2.03	23.9
F3 WHIMS Mags	10.3	0.29	2.07	4.46	42.8	17.8	0.33	0.36	24.4	0.80	0.17	0.31	0.05	0.03	10.3
F3 Li 2nd Cl Tail	28.5	0.80	1.82	3.92	69.8	20.8	1.16	2.55	0.92	1.94	0.75	1.02	0.47	0.53	1.08
F3 Li 1st Cl Tail	146	4.07	0.31	0.67	79.9	12.4	1.56	4.60	0.29	1.69	4.38	3.10	3.23	4.93	1.74
F3 Li Ro Tail	2,062	57.6	0.17	0.37	81.0	11.6	1.58	4.81	0.22	13.1	62.7	41.0	46.2	72.9	18.6
F3 Mica 2nd Cl Conc	327	9.12	0.30	0.65	60.0	25.7	5.94	3.02	1.55	3.66	7.36	14.4	27.6	7.25	20.8
F3 Slimes #2	54.2	1.51	0.60	1.29	60.2	14.3	1.93	2.92	1.35	1.21	1.23	1.33	1.48	1.16	3.00
F3 Mica Clnr Scav Conc	126	3.51	0.67	1.44	65.7	22.2	3.88	3.64	1.18	3.14	3.10	4.78	6.92	3.36	6.09
F3 Slimes #1	138	3.85	0.52	1.12	71.6	17.0	2.66	4.23	1.12	2.68	3.71	4.02	5.21	4.29	6.34
Cyclone OIF	114	3.19	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.17	2.90	3.67	5.32	3.54	8.07
Head (calc.)	3,583	100	0.75	1.61	74.3	16.3	1.97	3.80	0.68	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,331	93.0	0.76	1.59	72.2	15.3	1.75	3.63	0.58	95.2	93.4	92.3	89.5	92.2	85.6
Spod Flot Feed	2,824	78.8	0.86	1.85	84.0	17.8	2.02	4.22	0.66	87.1	81.7	71.8	53.5	80.4	55.7

	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
Tag	Li Non Mag Final Conc.	577	16.1	3.23	6.95	63.2	26.6	0.43	0.48	1.01	69.6	13.7	26.3	3.52	2.03	23.9
1	Li 2nd Cl. Conc.	587	16.4	3.21	6.91	62.8	26.4	0.43	0.48	1.42	70.4	13.9	26.6	3.57	2.06	34.2
2	Li 1st Cl Conc	616	17.2	3.15	6.77	63.2	26.2	0.46	0.57	1.40	72.3	14.6	27.7	4.04	2.60	35.3
3	Li Ro Conc.	762	21.3	2.60	5.60	66.4	23.5	0.67	1.35	1.18	74.0	19.0	30.8	7.27	7.53	37.1
6																

Test: F-4 **Project No.:** 12607-001 **Operator:** JG **Date:** 04-Apr-11

Purpose: **To repeat test F1 but replacing LR19 collector with LR19B**

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: D30LR (62 %), Aero 801 from Cytec (27 %), Kerosene (9 %) and Pine oil (2 %);

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**NaF**	**D618**	**LR19B**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**			**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	375	50	25									3		3.0	25
Mica Ro (4kg cell)	375											1	3	3.0	22
Mica Ro Scav	250	25											2		
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															
Mica Clnr #1	250	25										1	3	3.0	20
Mica Clnr #2	250											1	2	3.0	23
1st and 2nd clnr Tails to Li float															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 10 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						87.5				500		5		8.0	
Li Rougher 1 (4kg cell)						62.5							3	8.0	27
Li Rougher Scav 1						0		0	200	100			1.5	8.0	26
Li Rougher Scav 2						0				100			1.5	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						0				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						25						1	2	8.0	24
Total	2000	100	25		188	175			450	750	0	23	21		

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
WHIMS Non Mags	469	13.0	2.84	6.11	63.0	25.6	0.96	0.73	1.08	51.5	11.0	20.5	6.22	2.47	20.8
WHIMS Mags	8.80	0.24	1.62	3.49	40.9	17.5	1.31	0.51	24.6	0.55	0.13	0.26	0.16	0.03	8.89
Li 2nd Cl Tail	44.7	1.24	1.58	3.40	73.0	18.4	1.31	2.55	0.67	2.73	1.21	1.41	0.81	0.82	1.23
Li 1st Cl Tail	148	4.11	0.71	1.53	78.3	14.4	1.52	3.90	0.36	4.07	4.31	3.65	3.11	4.17	2.19
F4 Li Ro Tail	2,106	58.4	0.32	0.69	81.0	12.2	1.46	4.60	0.25	26.0	63.3	43.9	42.4	69.9	21.6
F4 Slimes #2	47.5	1.32	0.65	1.40	64.3	14.9	2.01	3.20	1.37	1.19	1.13	1.21	1.32	1.10	2.67
F4 Mica 2nd Cl Conc	455	12.6	0.41	0.88	61.4	24.6	5.28	3.53	1.36	7.20	10.4	19.1	33.1	11.6	25.4
F4 Slimes #1	214	5.92	0.55	1.18	71.6	17.0	2.60	4.22	1.04	4.53	5.68	6.21	7.66	6.50	9.12
Cyclone OIF	114	3.17	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.24	2.87	3.66	5.17	3.48	8.07
Head (calc.)	3,607	100	0.72	1.55	74.7	16.2	2.01	3.84	0.68	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,493	96.8	0.72	1.49	70.5	15.1	1.81	3.57	0.58	97.8	97.1	96.3	94.8	96.5	91.9
Spod Flot Feed	2,824	78.3	0.79	1.70	77.3	14.7	1.39	3.85	0.50	93.2	91.5	90.1	87.2	90.0	82.8

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
Tag Li Non Mag Final Conc.	469	13.0	2.84	6.11	63.0	25.6	0.96	0.73	1.08	51.5	11.0	20.5	6.22	2.47	20.8
1 Li 2nd Cl Conc	478	13.3	2.82	6.06	62.6	25.5	0.97	0.73	1.51	52.0	11.1	20.8	6.38	2.50	29.7
2 Li 1st Cl Conc	523	14.5	2.71	5.84	63.5	24.8	1.00	0.88	1.44	54.7	12.3	22.2	7.18	3.33	30.9
3 Li Ro Conc.	671	18.6	2.27	4.89	66.8	22.5	1.11	1.55	1.20	58.8	16.6	25.9	10.3	7.50	33.1
6															

Test: F-5 **Project No.:** 12607-001 **Operator**: JG **Date**: 04-Apr-11

Purpose: **To repeat test F1 but replacing LR19 collector with D30LR and FA-2**

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using FAT2 Collector: FA-2 65%+ D 30 LR 35%

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**NaF**	**D618**	**FAT 2**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**			**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	250	80	25									3		3.0	25
Mica Ro (4kg cell)	625											1	4	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															
Mica Clnr #1	250	10										1	3	3.0	20
Mica Clnr #2	125											1	2	3.0	23
1st and 2nd clnr Tails to Li float															
Mica Rougher Tail to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 10 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						50				500		5		8.0	
Li Rougher 1 (4kg cell)						37.5							4	8.0	27
Li Rougher Scav 1						0		0	200	100			2	8.0	26
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						37.5				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	1.5	8.0	25
Total	1750	90	25		188	163			450	750	0	24	23		

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
WHIMS NON MAGS	608	17.1	2.96	6.37	61.1	26.6	1.14	0.56	1.06	69.1	14.0	28.3	9.69	2.45	27.1
WHIMS MAGS	8.60	0.24	1.47	3.16	33.9	14.2	0.57	0.51	33.0	0.49	0.11	0.21	0.07	0.03	11.9
Li 3rd Cl Tail	15.6	0.44	1.75	3.77	65.5	23.9	1.42	2.18	1.03	1.05	0.39	0.65	0.31	0.24	0.68
Li 2nd Cl Tail	26.5	0.75	1.69	3.64	68.1	20.8	1.47	3.00	0.74	1.72	0.68	0.97	0.54	0.57	0.82
Li 1st Cl Tail	117	3.31	0.28	0.60	78.9	12.5	1.65	4.89	0.24	1.26	3.50	2.57	2.71	4.13	1.18
F5 Li Ro Tail	2,140	60.3	0.19	0.41	81.3	11.6	1.60	4.91	0.23	15.6	65.7	43.5	47.9	75.7	20.7
F5 Slimes #2	48.2	1.36	0.65	1.40	71.3	17.0	2.47	3.82	1.39	1.20	1.30	1.44	1.66	1.33	2.82
F5 Mica 2nd Cl Conc	305	8.59	0.33	0.71	59.8	25.6	6.03	3.12	1.49	3.87	6.89	13.7	25.7	6.86	19.1
F5 Slimes #1	167	4.69	0.54	1.16	71.6	16.8	2.65	4.35	1.06	3.45	4.50	4.90	6.17	5.22	7.42
Cyclone OIF	114	3.22	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.23	2.92	3.75	5.24	3.48	8.26
Head (calc.)	3,551	100	0.73	1.58	74.6	16.1	2.01	3.91	0.67	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,270	92.1	0.75	1.55	69.4	13.6	1.38	3.59	0.47	97.8	97.1	96.2	94.8	96.5	91.7
Spod Flot Feed	2,917	82.1	0.80	1.72	76.6	14.9	1.50	3.96	0.51	90.5	85.7	77.7	62.9	84.5	65.2

	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
Tag	Li Non Mag Final Conc.	608	17.1	2.96	6.37	61.1	26.6	1.14	0.56	1.06	69.1	14.0	28.3	9.69	2.45	27.1
1	Li 3rd Cl Conc	617	17.4	2.94	6.33	60.7	26.4	1.13	0.56	1.51	69.6	14.1	28.6	9.76	2.48	39.0
2	Li 2nd Cl Conc	633	17.8	2.91	6.26	60.8	26.4	1.14	0.60	1.49	70.7	14.5	29.2	10.1	2.73	39.7
3	Li 1st Cl Conc	659	18.6	2.86	6.16	61.1	26.1	1.15	0.70	1.46	72.4	15.2	30.2	10.6	3.30	40.5
4	Li Ro Conc.	776	21.9	2.47	5.32	63.8	24.1	1.23	1.33	1.28	73.6	18.7	32.7	13.3	7.44	41.7
6																

Test: F6 **Project No.:** 12607-001 **Operator**: Yasha C and Massoud **Date:** 04-Apr-11

Purpose: **To repeat test F1 but with the addition of caustic starch to the mica flotation**

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	Caustic Starch	D618	LR19	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%			5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
														9.0	21
Cond. deslimed product at high density															
pH Adjustment	145													3.0	25
Condition	100	80	25					50				3		3.0	25
Mica Ro (4kg cell)												1	4	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30
Mica Clnr #1	125	10										1	3	3.0	29
Mica Clnr #2	100											1	2	3.0	27
1st and 2nd clnr Tails to Li float															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						125				500		5		8.0	
Li Rougher 1 (4kg cell)						25							2.5	8.0	27
Li Rougher Scav 1						25			200	100			1.5	8.0	26
Li Rougher Scav 2						25				100			1.5	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	1.5	8.0	25
Total	470	90	25		188	238			450	750	0	24	20		

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
WHIMS Non Mags	532	15.0	2.99	6.44	62.1	27.1	0.82	0.42	1.09	61.8	12.5	24.9	6.20	1.66	25.1
WHIMS Mags	11.5	0.32	1.38	2.97	32.4	14.2	0.66	0.33	40.5	0.62	0.14	0.28	0.11	0.03	20.2
Li 3rd Cl Tail	59.8	1.68	2.35	5.06	60.8	27.8	2.58	0.86	1.40	5.46	1.37	2.87	2.19	0.38	3.62
Li 2nd Cl Tail	53.3	1.50	2.03	4.37	62.1	25.9	2.66	1.51	1.21	4.20	1.25	2.39	2.01	0.60	2.79
Li 1st Cl Tail	97.8	2.75	0.95	2.04	72.2	18.1	2.05	3.68	0.59	3.61	2.66	3.06	2.85	2.68	2.50
F6 Li Ro Tail	2,333	65.6	0.18	0.39	80.6	11.9	1.65	4.82	0.13	16.3	71.0	48.0	54.6	83.6	13.1
F6 Slimes #2	22.3	0.63	0.57	1.23	66.7	19.1	3.14	3.70	1.83	0.49	0.56	0.74	0.99	0.61	1.77
F6 Mica 2nd Cl Conc	160	4.50	0.28	0.60	50.0	32.2	8.48	1.25	2.07	1.74	3.02	8.92	19.3	1.49	14.3
F6 Slimes #1	172	4.83	0.52	1.12	71.4	17.1	2.63	4.19	1.09	3.47	4.63	5.08	6.41	5.35	8.10
Cyclone OIF	114	3.22	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.26	2.91	3.70	5.32	3.59	8.50
Head (calc.)	3,557	100	0.72	1.56	74.5	16.3	1.98	3.78	0.65	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,271	92.0	0.74	1.57	72.5	14.6	1.49	3.68	0.49	97.7	97.1	96.3	94.7	96.4	91.5
Spod Flot Feed	3,088	86.8	0.77	1.65	76.3	15.3	1.55	3.87	0.50	92.5	89.4	82.3	69.0	89.6	69.1

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.	532	15.0	2.99	6.44	62.1	27.1	0.82	0.42	1.09	61.8	12.5	24.9	6.20	1.66	25.1
Li 3rd Cl Conc	544	15.3	2.96	6.36	61.5	26.8	0.82	0.42	1.92	62.5	12.6	25.2	6.30	1.69	45.3
Li 2nd Cl Conc	604	17.0	2.90	6.23	61.4	26.9	0.99	0.46	1.87	67.9	14.0	28.1	8.49	2.07	48.9
Li 1st Cl Conc	657	18.5	2.83	6.08	61.5	26.8	1.13	0.55	1.82	72.1	15.2	30.5	10.5	2.67	51.7
Li Ro Conc.	755	21.2	2.58	5.56	62.9	25.7	1.25	0.95	1.66	75.7	17.9	33.6	13.4	5.35	54.2

Test: F-7 **Project No.:** 12607-001 **Operator**: JG **Date:** 04/?/2011

Purpose: **To test the FAT-1 collector for spodumene flotation**

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm.

Regrind: None

Conditions: FAT-1: 80% of F-5 collector + 15% Aero 801 + 5% Diesel

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Caustic Starch**	**D618**	**FAT 1**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**			**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	250	50	25									3		3.0	25
Mica Ro (4kg cell)	375											1	3	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															
Mica Clnr #1	300	10						75				1	3	3.0	20
Mica Clnr #2	250											1	2	3.0	23
1st and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						125				500		5		8.0	
Li Rougher 1 (4kg cell)						12.5							3	8.0	23
Li Rougher Scav 1						12.5		50	200	100			1.5	8.0	22
Li Rougher Scav 2						12.5				100			1.5	8.0	22.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						25				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						25						1	1.5	8.0	24
Total	1675	60	25		188	213			450	750	0	23	18		

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
WHIMS Non Mags	656	18.3	3.10	6.67	62.6	26.5	0.86	0.58	1.11	74.9	15.4	30.1	7.82	2.73	29.9
WHIMS Mags	8.00	0.22	1.75	3.77	40.0	16.8	0.62	0.45	28.1	0.52	0.12	0.23	0.07	0.03	9.22
Li 2nd Cl Tail	19.9	0.56	1.10	2.37	72.8	17.8	1.60	4.00	1.04	0.81	0.54	0.61	0.44	0.57	0.85
Li 1st Cl Tail	115	3.20	0.20	0.43	79.9	12.2	1.68	5.14	0.21	0.84	3.43	2.42	2.67	4.23	0.99
F7 Li Ro Tail	2,237	62.5	0.17	0.37	81.1	11.6	1.64	5.00	0.25	14.0	67.9	44.9	50.8	80.3	22.9
F7 Slimes #2	138	3.85	0.53	1.14	71.7	17.1	2.68	4.32	1.11	2.69	3.70	4.07	5.11	4.27	6.27
F7 Mica 2nd Cl Conc	228	6.37	0.33	0.71	51.8	30.8	8.08	1.58	1.96	2.77	4.42	12.1	25.5	2.58	18.3
F7 Slimes #1	62.8	1.75	0.59	1.27	69.8	17.6	2.75	4.02	1.35	1.36	1.64	1.91	2.39	1.81	3.48
Cyclone OIF	114	3.20	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.14	2.89	3.70	5.19	3.46	8.06
Head (calc.)	3,579	100	0.76	1.63	74.7	16.2	2.02	3.89	0.68	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,464	96.8	0.77	1.63	73.6	15.8	1.93	3.81	0.62	97.9	97.1	96.3	94.8	96.5	91.9
Spod Flot Feed	3,036	84.8	0.84	1.81	80.1	15.7	1.59	4.23	0.56	91.0	87.4	78.2	61.8	87.9	63.9

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.	656	18.3	3.10	6.67	62.6	26.5	0.86	0.58	1.11	74.9	15.4	30.1	7.82	2.73	29.9
Li 2nd Cl. Conc.	664	18.6	3.08	6.64	62.3	26.4	0.86	0.58	1.44	75.4	15.5	30.3	7.88	2.76	39.1
Li 1st Cl Conc	684	19.1	3.03	6.51	62.6	26.1	0.88	0.68	1.42	76.2	16.0	30.9	8.33	3.33	39.9
Li Ro Conc.	799	22.3	2.62	5.64	65.1	24.1	0.99	1.32	1.25	77.0	19.5	33.3	11.0	7.56	40.9

Test: F8 **Project No.:** 12607-001 **Operator:** YC **Date:** 13/04/2011

Purpose: **To repeat Test F1 on 10 kg charge, Also testing the addition of NaF to the Li rougher scavenger**

Procedure: As outlined below.

Feed: 10kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes					
Stage	H2SO4	Armac C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	NaF	D618	LR19	Grind	Cond.	Froth	pH	Temp°C	
	10%	5%		2%	25%	5%			5%	100%	Stage Grinding					
Mica Rougher Feed to Deslime																
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 12 minutes decant by syphon 1 time*													8.5	15	Deslimed only -150M fraction
														9.0	21	
Cond. deslimed product at high density																Conditioning tried in deslime cc
pH Adjustment	325													3.0	25	Finally conditioning was done i
Condition	0	80	25									3		3.0	25	Fuel oil used was less than that
Mica Ro (4kg cell)												1	4	3.0	24	
Clean Mica Conc. and return the tailing to the rougher tailings																
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30	
Mica Clnr #1	50	10										1	11	3.0	29	
Mica Clnr #2	50											1	12	3.0	27	
Mica Clnr #3	45		15										5			
1st, 2nd and 3rd clnr Tails to Li float.																
Mica Rougher Feed to Scrub					262.5											
Scrub in 4 kg cell (high density)									350			12		10.5		
Deslime	*Use 16 L acrylic cylinder fill to 15L settle 12 minutes decant by syphon 2 times*															
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30	
Condition 2 (10kg cell) 1500 RPM										600		5		8.0		
Li Rougher 1 (4kg cell)													8	8.0	27	
Li Rougher Scav 1				50		25		200	200	200			4	8.0	26	
Li Rougher Scav 2						25				100			3	8.0	25.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.																
Combine Rougher Concentrate and Rougher Scav#1.																
Li 1st Cleaner (4kg cell)						12.5				50		3	8	8.0	21	
Li 2nd Cleaner (4kg cell)						12.5			50			1	5	8.0	24	
Li 3rd Cleaner (4kg cell)						12.5			25			1	5	8.0	25	
Li 4th Cleaner (4kg cell)													5			
Total	470	90	40		263	88			625	950	0	28	70			

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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 Li 4th Clnr Conc	1,405	14.5	2.97	6.39	61.4	26.4	0.56	0.41	1.89	63.2	12.0	23.9	3.98	0.75	44.6
F8 Li 4th Clnr Tls	124	1.28	2.55	5.49	61.7	27.0	1.70	0.77	1.29	4.77	1.06	2.14	1.06	0.12	2.67
F8 Li 3rd Clnr Tls	133	1.38	2.24	4.82	61.1	27.0	2.40	1.03	1.26	4.52	1.13	2.32	1.62	0.18	2.82
F8 Li 2nd Clnr Tls	161	1.67	2.04	4.39	64.4	24.3	1.95	1.71	1.05	4.98	1.45	2.52	1.59	0.36	2.84
F8 Li 1st Clnr Tls	231	2.39	0.97	2.09	73.4	17.0	1.76	3.30	0.64	3.39	2.36	2.52	2.05	1.00	2.48
F8 Li Ro Scav Tls	6,294	65.1	0.10	0.21	80.7	11.6	1.70	11.2	0.08	9.11	70.7	47.0	54.0	92.4	8.45
F8 Slimes #2	22.1	0.23	0.22	0.47	59.2	22.0	4.03	2.92	2.24	0.07	0.18	0.31	0.45	0.08	0.83
F8 Mica Conc	623	6.45	0.51	1.10	53.4	29.9	7.80	1.95	1.93	4.81	4.63	12.0	24.6	1.59	20.2
F8 Slimes #1	335	3.47	0.46	0.99	71.8	16.9	2.62	4.42	0.88	2.33	3.35	3.64	4.44	1.94	4.95
Cyclone OIF	334	3.46	0.56	1.21	68.0	17.4	3.65	3.53	1.82	2.83	3.16	3.74	6.17	1.55	10.2
Head (calc.)	9,664	100	0.68	1.47	74.4	16.1	2.05	7.90	0.62	100	100	100	100	100	100
Direct			0.80	1.72	75.3	16.2	2.69	3.35	0.99						
Flotation Feed	8,371	86.6	0.71	1.53	76.2	14.9	1.52	8.65	0.45	94.8	93.5	92.6	89.4	96.5	84.8
Spod Flot Feed	8,349	86.4	0.71	1.53	76.4	15.0	1.52	8.67	0.46	89.9	88.7	80.3	64.3	94.8	63.8

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 4th Cl. Conc.	1,405	14.5	2.97	6.39	61.4	26.4	0.56	0.41	1.89	63.2	12.0	23.9	3.98	0.75	44.6
Li 3rd Cl. Conc.	1,529	15.8	2.94	6.32	61.4	26.4	0.65	0.44	1.84	67.9	13.1	26.0	5.04	0.88	47.2
Li 2nd Cl Conc	1,662	17.2	2.88	6.20	61.4	26.5	0.79	0.49	1.79	72.5	14.2	28.3	6.66	1.06	50.1
Li 1st Cl Conc	1,823	18.9	2.81	6.04	61.7	26.3	0.89	0.59	1.73	77.4	15.6	30.8	8.25	1.42	52.9
Li Ro. Conc	2,054	21.3	2.60	5.60	63.0	25.3	0.99	0.90	1.61	80.8	18.0	33.4	10.3	2.42	55.4

Test: F9 **Project No.:** 12607-001 **Operator**: JG **Date**: 15/04/2011

Purpose: **To repeat test F1 but replacing Armac C with Armac T**

Procedure: As outlined below.

Feed: About 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes					
Stage	**H2SO4**	**Armac T**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**	
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**					
Mica Rougher Feed to Deslime																
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*															Deslimed only -150M fraction
														9.0	21	
Cond. deslimed product at high density																Conditioning tried in deslime co
pH Adjustment	375													3.0	25	Finally conditioning was done ir
Condition	125	80	25					0				3		3.0	25	Fuel oil used was less than that r
Mica Ro (4kg cell)	375											1	3.5	3.0	24	
Clean Mica Conc. and return the tailing to the rougher tailings																
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30	
Mica Clnr #1	250	10										1	2	3.0	29	
Mica Clnr #2	250											1	2	3.0	27	
1st, and 2nd clnr Tails to Li float																
Mica Rougher Feed to Scrub																
Scrub in 4 kg cell (high density)									250			10		10.5		
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*															
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30	
Condition 2 (4kg cell) 1500 RPM						125				600		5		8.0		
Li Rougher 1 (4kg cell)						25							3	8.0	27	
Li Rougher Scav 1				50		100			200	200			2	8.0	26	
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.																
Combine Rougher Concentrate and Rougher Scav#1 and #2.																
Li 1st Cleaner (2kg cell)						25				50		1	2	8.0	21	
Li 2nd Cleaner (2kg cell)						25						1	1.5	8.0	24	
Li 3rd Cleaner (2kg cell)						12.5						1	1.5	8.0	25	
Total	1375	90	25		0	325			450	950	0	24	19			

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
Whims Non Mags	702	19.4	3.00	6.46	63.0	26.4	0.74	0.59	0.95	80.2	16.4	31.4	7.15	2.98	32.9
Whims Mags	15.4	0.43	2.00	4.31	48.4	21.0	1.00	0.48	16.2	1.17	0.28	0.55	0.21	0.05	12.3
Li 3rd Cl Tail	24.6	0.68	1.30	2.80	71.0	19.3	1.58	3.17	0.72	1.22	0.65	0.80	0.53	0.56	0.87
Li 2nd Cl Tail	36.7	1.01	0.61	1.31	76.0	15.3	1.62	4.28	0.39	0.85	1.03	0.95	0.82	1.13	0.71
Li 1st Cl Tail	136	3.75	0.13	0.28	80.5	11.9	1.68	5.09	0.07	0.67	4.04	2.73	3.13	4.97	0.47
F9 Li Ro Tail	2,130	58.8	0.09	0.20	81.4	11.3	1.61	5.04	0.08	7.54	64.2	40.8	47.2	77.3	8.39
F9 Slimes #2	66.7	1.84	0.58	1.25	71.2	17.0	2.47	4.07	1.29	1.47	1.76	1.92	2.27	1.95	4.24
F9 Mica 2nd Cl Conc	191	5.26	0.13	0.28	47.3	34.6	9.74	0.84	2.12	0.94	3.34	11.2	25.5	1.15	19.9
F9 Slimes #1	159	4.40	0.53	1.14	71.7	17.1	2.67	4.21	1.04	3.21	4.23	4.61	5.85	4.83	8.16
Cyclone OIF	160	4.41	0.45	0.96	68.2	18.8	3.33	4.42	1.54	2.71	4.03	5.07	7.31	5.08	12.1
Head (calc.)	3,621	100	0.73	1.56	74.5	16.3	2.01	3.84	0.56	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,302	91.2	0.75	1.60	72.3	14.1	1.35	3.74	0.37	94.1	91.7	90.3	86.8	90.1	79.8
Spod Flot Feed	915	25.3	2.63	5.67	256	49.8	4.69	13.2	1.23	84.1	22.4	36.4	11.9	9.70	47.2

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.	702	19.4	3.00	6.46	63.0	26.4	0.74	0.59	0.95	80.2	16.4	31.4	7.15	2.98	32.9
Li 2nd Cl. Conc.	718	19.8	2.98	6.41	62.7	26.3	0.75	0.59	1.28	81.4	16.7	32.0	7.36	3.04	45.2
Li 1st Cl Conc	742	20.5	2.92	6.29	63.0	26.1	0.77	0.67	1.26	82.6	17.3	32.8	7.90	3.60	46.0
Li Ro Conc.	779	21.5	2.81	6.06	63.6	25.5	0.81	0.84	1.22	83.5	18.4	33.7	8.72	4.73	46.7

Test: F10 **Project No.:** 12607-001 **Operator**: JG **Date**: 15/04/2011

Purpose: **To repeat test F1 but using Lead Nitrate as mica activator**

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes					
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**	
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**					
Mica Rougher Feed to Deslime																
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*															Deslimed only -150M fraction
														9.0	21	
Cond. deslimed product at high density																Conditioning tried in deslime co
pH Adjustment	375													3.0	25	Finally conditioning was done in
Condition	125	80	25					25				3		3.0	25	Fuel oil used was less than that
Mica Ro (4kg cell)	250											1	3	3.0	24	
Clean Mica Conc. and return the tailing to the rougher tailings																
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30	
Mica Clnr #1	250	10										1	2.5	3.0	29	
Mica Clnr #2	150											1	2.5	3.0	27	
1st. and 2nd clnr Tails to Li float																
Mica Rougher Feed to Scrub																
Scrub in 4 kg cell (high density)									250			10		10.5		
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*															
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30	
Condition 2 (4kg cell) 1500 RPM						125				600		5		8.0		
Li Rougher 1 (4kg cell)						37.5							3	8.0	27	
Li Rougher Scav 1				50		25			200	200			2.5	8.0	26	
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.																
Combine Rougher Concentrate and Rougher Scav#1.																
Li 1st Cleaner (2kg cell)						25				50		1	2.5	8.0	21	
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24	
Li 3rd Cleaner (2kg cell)						12.5						1	2	8.0	25	
Total	1150	90	25		0	250			450	950	0	24	22			

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
Whims Non Mags	710	19.9	3.00	6.46	63.3	26.3	0.66	0.65	0.89	80.6	16.9	32.0	6.53	3.41	29.9
Whims Mags	11.5	0.32	1.80	3.87	40.5	17.0	0.75	0.51	29.9	0.78	0.18	0.34	0.12	0.04	16.3
Li 3rd Cl Tail	29.4	0.82	1.54	3.31	71.4	19.9	1.25	3.01	0.73	1.72	0.79	1.00	0.51	0.65	1.02
Li 2nd Cl Tail	35.8	1.00	0.76	1.64	75.8	15.4	1.43	4.12	0.41	1.03	1.02	0.95	0.71	1.09	0.70
Li 1st Cl Tail	128	3.60	0.14	0.30	80.6	11.6	1.57	4.97	0.08	0.68	3.89	2.55	2.81	4.72	0.49
F10 Li Ro Tail	2,032	56.9	0.07	0.16	81.9	10.9	1.59	5.01	0.06	5.62	62.6	38.0	45.0	75.3	5.77
F10 Slimes #2	31.6	0.89	0.64	1.38	69.8	18.1	2.73	3.66	1.58	0.77	0.83	0.98	1.20	0.86	2.36
F10 Mica 2nd Cl Conc	303	8.49	0.30	0.65	54.9	29.3	7.29	2.15	1.75	3.44	6.26	15.2	30.8	4.82	25.1
F10 Slimes #1	128	3.60	0.54	1.16	71.2	17.3	2.73	4.14	1.11	2.63	3.44	3.81	4.89	3.93	6.75
Cyclone OIF	160	4.48	0.45	0.96	68.2	18.8	3.33	4.42	1.54	2.69	4.10	5.14	7.40	5.22	11.6
Head (calc.)	3,569	100	0.74	1.59	74.5	16.3	2.01	3.79	0.59	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,281	91.9	0.76	1.58	69.8	13.5	1.24	3.55	0.36	94.7	92.5	91.1	87.7	90.8	81.6
Spod Flot Feed	915	25.6	2.61	5.62	248	47.7	4.37	12.6	1.25	84.9	22.8	36.8	10.7	9.92	48.4

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.	710	19.9	3.00	6.46	63.3	26.3	0.66	0.65	0.89	80.6	16.9	32.0	6.53	3.41	29.9
Li 2nd Cl. Conc.	721	20.2	2.98	6.42	62.9	26.2	0.66	0.65	1.35	81.4	17.1	32.3	6.65	3.45	46.2
Li 1st Cl Conc	750	21.0	2.92	6.29	63.3	25.9	0.68	0.74	1.33	83.1	17.9	33.3	7.16	4.11	47.2
Li Ro Conc.	786	22.0	2.83	6.08	63.8	25.4	0.72	0.89	1.29	84.2	18.9	34.3	7.87	5.20	47.9

Test: F11 **Project No.:** 12607-001 **Operator:** JG **Date:** 05-May-11

Purpose: **To repeat test F10 but using Armac T instead of Armac C, without any Pb or Fe Salt**

Procedure: As outlined below.

Feed: HL test 1 and 2: Middlings+Undersizes

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac T**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
														7.5	21
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	250	80	25									3		3.0	25
Mica Ro (4kg cell)	250											1	3.5	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good.)														8.0	30
Mica Clnr #1	250	10										1	1.5	3.0	29
Mica Clnr #2	250											1	1.5	3.0	27
1st, and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					250				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						187.5				600		5		8.0	
Li Rougher 1 (4kg cell)						125							3	8.0	27
Li Rougher Scav 1						25			200	200			2	8.0	26
Li Rougher Scav 2						25				100			1.5	8.0	25.0
Li Rougher Scav 3						25				100			1	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	2	8.0	25
Li 4th Cleaner (2kg cell)						12.5						1	1.5	8.0	
Total	1500	90	25		250	438			450	1050	0	25	22		

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
F11 WHIMS Non Mags	581	16.2	2.11	4.54	57.8	27.7	3.33	0.65	1.31	61.9	12.5	29.5	26.7	3.18	25.4
F11 WHIMS Mags	15.7	0.44	0.97	2.09	27.2	11.0	0.96	0.44	42.7	0.77	0.16	0.32	0.21	0.06	22.4
F11 4th Cl Tail	39.7	1.11	1.34	2.88	67.9	21.5	2.74	1.63	1.29	2.69	1.00	1.56	1.50	0.54	1.71
F11 3rd Cl Tail	54.2	1.51	1.03	2.22	71.4	18.5	2.48	1.99	1.01	2.82	1.44	1.84	1.85	0.91	1.83
F11 2nd Cl Tail	89.7	2.50	0.62	1.33	78.4	13.6	1.59	2.74	0.61	2.81	2.61	2.23	1.97	2.07	1.82
F11 1st Cl Tail	198	5.52	0.22	0.47	81.5	11.3	1.44	3.72	0.37	2.20	5.99	4.10	3.93	6.20	2.45
F11 Ro Tail	2,124	59.1	0.18	0.39	82.4	10.3	0.99	4.28	0.19	19.3	65.0	40.1	29.0	76.5	13.5
F11 Slimes #2	43.6	1.21	0.44	0.95	69.2	16.9	2.70	3.25	1.71	0.97	1.12	1.35	1.62	1.19	2.49
F11 Mica Conc	167	4.66	0.11	0.24	47.2	34.2	9.82	0.81	2.40	0.93	2.93	10.5	22.7	1.14	13.4
F11 Slimes #1	151	4.19	0.42	0.90	73.8	15.6	2.32	3.59	1.24	3.19	4.12	4.30	4.82	4.55	6.23
F-11 Cyclone O/F	128	3.55	0.37	0.80	67.7	18.3	3.29	3.45	2.09	2.38	3.20	4.27	5.78	3.70	8.89
Head (calc.)	3,592	100	0.55	1.19	75.0	15.2	2.02	3.31	0.83	100	100	100	100	100	100
Flotation Feed	3,314	92.3	0.56	1.21	75.4	15.1	1.96	3.29	0.77	94.4	92.7	91.4	89.4	91.8	84.9
Spod Flot Feed	3,103	86.4	0.59	1.27	77.0	14.0	1.52	3.43	0.67	92.5	88.6	79.6	65.1	89.4	69.0

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.	581	16.2	2.11	4.54	57.8	27.7	3.33	0.65	1.31	61.9	12.5	29.5	26.7	3.18	25.4
Li 4th Cl. Conc.	597	16.6	2.08	4.48	57.0	27.3	3.27	0.64	2.40	62.7	12.6	29.8	26.9	3.24	47.7
Li 3rd Cl Conc	637	17.7	2.03	4.38	57.7	26.9	3.23	0.71	2.33	65.4	13.6	31.4	28.4	3.78	49.5
Li 2nd Cl. Conc.	691	19.2	1.96	4.21	58.8	26.2	3.18	0.81	2.23	68.2	15.1	33.2	30.2	4.69	51.3
Li 1st Cl Conc	781	21.7	1.80	3.88	61.0	24.8	2.99	1.03	2.04	71.0	17.7	35.4	32.2	6.75	53.1
Li Ro Conc.	979	27.2	1.48	3.19	65.2	22.1	2.68	1.57	1.70	73.2	23.7	39.5	36.1	13.0	55.5

Test: F12 **Project No.:** 12607-001 **Operator:** JG **Date:** 05-May-11

Purpose: **To repeat test F11, but including scav flotation on the +400 fraction only**

Procedure: As outlined below.

Feed: HL test 3 and 4: Middlings+Undersizes

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac T**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
														9.0	21
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	375	80	25									3		3.0	25
Mica Ro (4kg cell)	250											1	3.5	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30
Mica Clnr #1	200	25										1	2.5	3.0	29
Mica Clnr #2	125	25										1	2	3.0	27
1st. and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)									250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						87.5				600		5		8.0	
Li Rougher 1 (4kg cell)						37.5							3	8.0	27
Screen the Tailings on the 400 Mesh Screen, only condition the +400 fraction															
Li Rougher Scav 1						37.5			200	200			2.5	8.0	26
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						25				50		1	2	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	1.5	8.0	25
Li 4th Cleaner (2kg cell)						12.5						1	1.5	8.0	
Li 5th Cleaner (2kg cell)						12.5						1	1.5	8.0	
Total	1450	130	25		0	13			450	950	0	26	24		

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
F12 WHIMS Non Mags	464	12.9	2.64	5.68	59.8	27.1	1.93	0.61	1.24	62.1	10.2	22.8	12.2	2.41	20.7
F12 WHIMS Mags	14.8	0.41	1.03	2.22	28.5	11.7	0.83	0.38	44.8	0.77	0.15	0.31	0.17	0.05	23.8
F12 Li 5th Cl Tail	33.1	0.92	1.91	4.11	62.4	25.8	2.86	1.35	1.33	3.20	0.76	1.55	1.29	0.38	1.58
F12 Li 4th Cl Tail	44.5	1.24	1.60	3.44	64.3	23.9	2.83	1.68	1.24	3.61	1.05	1.93	1.72	0.64	1.98
F12 Li 3rd Cl Tail	44.4	1.23	1.18	2.54	69.8	19.7	2.23	2.43	0.93	2.66	1.14	1.59	1.35	0.92	1.48
F12 Li 2nd Cl Tail	69.1	1.92	0.59	1.27	77.5	14.5	1.62	3.28	0.52	2.07	1.96	1.82	1.53	1.93	1.29
F12 Li 1st Cl Tail	212	5.91	0.25	0.54	82.8	10.8	1.10	3.84	0.18	2.69	6.45	4.17	3.19	6.96	1.37
F12 Li Ro Tail	2,040	56.7	0.14	0.30	84.8	9.54	0.83	4.24	0.04	14.5	63.4	35.3	23.1	73.8	2.93
F12 Slimes #2	49.8	1.39	0.47	1.01	73.2	15.8	2.20	3.34	1.30	1.19	1.34	1.43	1.50	1.42	2.33
F12 Slimes #3 - 400m	36.6	1.02	0.06	0.13	81.8	10.5	0.99	4.85	0.30	0.11	1.10	0.70	0.50	1.51	0.39
F12 Mica Conc	326	9.07	0.12	0.26	47.4	34.3	9.75	0.80	2.28	1.98	5.67	20.3	43.4	2.23	26.7
F12 Slimes #1	133	3.71	0.41	0.88	74.0	15.7	2.32	3.51	1.21	2.77	3.62	3.80	4.22	3.99	5.80
F-12 HL Test 1&2 Cyclone O/F	128	3.55	0.37	0.80	67.7	18.3	3.29	3.45	2.09	2.39	3.17	4.24	5.73	3.75	9.58
Head (calc.)	3,595	100	0.55	1.18	75.8	15.3	2.04	3.26	0.77	100	100	100	100	100	100
Flotation Feed	3,009	83.7	0.61	1.31	79.4	13.1	1.13	3.51	0.54	92.9	87.5	71.7	46.6	90.0	57.9
Spod Flot Feed	882	24.5	1.72	3.71	67.0	21.3	1.78	1.77	1.65	77.1	21.7	34.2	21.5	13.3	52.2

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.	464	12.9	2.64	5.68	59.8	27.1	1.93	0.61	1.24	62.1	10.2	22.8	12.2	2.41	20.7
Li 4th Cl. Conc.	479	13.3	2.59	5.58	58.8	26.6	1.90	0.60	2.59	62.9	10.3	23.1	12.4	2.46	44.5
Li 3rd Cl Conc	512	14.2	2.55	5.48	59.1	26.6	1.96	0.65	2.51	66.1	11.1	24.7	13.7	2.84	46.1
Li 2nd Cl. Conc.	556	15.5	2.47	5.32	59.5	26.4	2.03	0.73	2.40	69.7	12.1	26.6	15.4	3.48	48.1
Li 1st Cl Conc	601	16.7	2.38	5.11	60.2	25.9	2.04	0.86	2.30	72.3	13.3	28.2	16.8	4.40	49.6
Li Ro Conc.	670	18.6	2.19	4.72	62.0	24.7	2.00	1.11	2.11	74.4	15.2	30.0	18.3	6.34	50.9

Test: F13 **Project No.:** 12607-001 **Operator:** JG **Date:** 13/05/2011

Purpose: To study the effect of FA-2 and silica flotation on the spod concentrate

Procedure: As outlined below.

Feed: 4kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes					
Stage	H2SO4	Armac C	Fuel Oil	HF	NaOH	Na_2CO_3	Na Silicate	NaF	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C	
	10%			2%	25%	5%		5%	5%	100%	Stage Grinding					
Mica Rougher Feed to Deslime																
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*															Deslimed only -150M fraction
														9.0		
Cond. deslimed product at high density																Conditioning tried in deslime co
pH Adjustment	375													3.0		Finally conditioning was done i
Condition	125	80	25									3		3.0		Fuel oil used was less than that
Mica Ro (4kg cell)	300											1	4	3.0		
Clean Mica Conc. and return the tailing to the rougher tailings																
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0		
Mica Clnr #1	125	10										1	3	3.0		
Mica Clnr #2	125											1	2.5	3.0		
1st, and 2nd clnr Tails to Li float.																
Mica Rougher Feed to Scrub																
Scrub in 4 kg cell (high density)					187.5				250			10		10.5		
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*															
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0		
Condition 2 (4kg cell) 1500 RPM						125				500		5		8.0		
Li Rougher 1 (4kg cell)						62.5							3.5	8.0		
Li Rougher Scav 1						37.5			200	100			1.5	8.0		
Li Rougher Scav 2						37.5				100			1	8.0		
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.																
Combine Rougher Concentrate and Rougher Scav#1.																
Li 1st Cleaner (2kg cell)						37.5				50		1	1.5	8.0		
Li 2nd Cleaner (2kg cell)						12.5						1	1.5	8.0		
Li 3rd Cleaner (2kg cell)						12.5						1	1.5	8.0		
Washing the final concentrate with H2SO4 at pH 2 or lower: Transfer the spod concentrate to a 1 kg cell and decant the water as much as possible and then add a bit of fresh water with H2SO4 to reduce pH to 2, then condition for																
5 minutes, then decant the water and repeat washing the froth 1 more time, then do the silica flotation																
Silica Flotation																
Decant Water																
Condition 1	2250			750								10		2.0		
Condition 2		250	50									3				
Bulk Silica Flotation	0												2.5	2.0		
The final Tail is the spodumene Concentrate, pass that through Mag Sep.																
Total	3300	340	75		188	325			450	750	0	37	23			

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
F13 WHIMS Non-Mags	602	16.9	3.07	6.61	62.9	26.3	0.69	0.51	0.96	78.3	14.3	27.9	5.62	2.12	19.8
F13 WHIMS Mags	15.5	0.44	0.41	0.88	19.4	6.96	0.65	0.56	66.1	0.27	0.11	0.19	0.14	0.06	35.1
F13 Li 3rd Cl Tail	18.9	0.53	1.56	3.36	65.5	22.2	2.18	2.54	1.05	1.25	0.47	0.74	0.56	0.33	0.68
F13 Li 2nd Cl Tail	33.8	0.95	0.87	1.87	72.8	17.3	1.96	3.84	0.60	1.25	0.93	1.03	0.90	0.90	0.69
F13 Li 1st Cl Tail	201	5.64	0.14	0.30	80.8	11.9	1.70	5.11	0.14	1.19	6.12	4.20	4.62	7.10	0.96
F13 Li 1st Ro Tail	2,003	56.3	0.06	0.13	81.5	11.3	1.73	5.20	0.13	5.10	61.6	39.8	46.9	72.1	8.92
F13 Slimes #2	51.2	1.44	0.67	1.44	70.6	17.3	2.50	3.76	1.56	1.45	1.36	1.56	1.73	1.33	2.74
F13 Mica Conc	344	9.68	0.34	0.73	58.4	26.0	6.10	3.10	1.59	4.96	7.58	15.8	28.4	7.39	18.8
F13 Slimes #1	179	5.03	0.51	1.10	71.9	16.8	2.61	4.45	0.98	3.86	4.85	5.28	6.31	5.51	6.00
Cyclone OIF	108	3.04	0.51	1.09	67.5	18.7	3.28	4.22	1.72	2.32	2.75	3.56	4.79	3.15	6.36
Head (calc.)	3,556	100	0.66	1.43	74.6	16.0	2.08	4.06	0.82	100	100	100	100	100	100
Direct			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,269	91.9	0.68	1.38	68.8	13.1	1.37	3.71	0.62	93.8	92.4	91.2	88.9	91.3	87.6
Spod Flot Feed	871	24.5	2.37	5.10	254	48.2	4.98	13.7	2.22	82.3	21.9	34.0	11.8	10.5	57.2

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.	602	16.9	3.07	6.61	62.9	26.3	0.69	0.51	0.96	78.3	14.3	27.9	5.62	2.12	19.8
Li 3rd Cl. Conc.	617	17.4	3.00	6.46	61.8	25.8	0.69	0.51	2.60	78.6	14.4	28.0	5.76	2.19	54.9
Li 2nd Cl Conc	636	17.9	2.96	6.37	61.9	25.7	0.73	0.57	2.55	79.9	14.9	28.8	6.32	2.52	55.6
Li 1st Cl Conc	670	18.8	2.85	6.15	62.5	25.3	0.80	0.74	2.45	81.1	15.8	29.8	7.21	3.42	56.3
Li Ro. Conc	871	24.5	2.23	4.80	66.7	22.2	1.00	1.74	1.92	82.3	21.9	34.0	11.8	10.5	57.2

Test: F14 **Project No.:** 12607-001 **Operator**: JG **Date:** 05-25-2011

Purpose: **Similar to F1 but on with finer flotation feed**

Procedure: As outlined below.

Feed: Spod Charge, Grind to 117 micron

Grind: Stage ground to -117 µm

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Armac C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	LR19	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
														9.0	
Cond. deslimed product at high density															
pH Adjustment	500													3.0	25
Condition	250	80	25									3		3.0	25
Mica Ro (4kg cell)	250											1	3.5	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30
Mica Clnr #1	250	25										1	2	3.0	29
Mica Clnr #2	125	25										1	2	3.0	27
1st, and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					220				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						250				600		5		8.0	
Li Rougher 1 (4kg cell)						37.5							3.5	8.0	27
High Density Conditioning															
Li Rougher Scav 1						87.5			200	200			1.5	8.0	26
Li Rougher Scav 2						0				100			1.5	8.0	25.0
Screen on 400 mesh filter -400m send + to scav#3															
Li Rougher Scav 3						62.5				100			1.5		
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						37.5				50		1	3	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	2	8.0	25
Li 4th Cleaner (2kg cell)						0	25					1	1.5	8.5	
Total	1375	130	25		220	13			450	1050	0	25	24		

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
WHIMS non-mags	837	22.0	2.22	4.78	70.4	20.5	0.72	0.90	0.90	71.5	20.9	27.6	7.58	5.04	21.3
WHIMS mags	12.9	0.34	1.18	2.54	35.3	11.1	0.56	0.62	37.9	0.59	0.16	0.23	0.09	0.05	13.9
Li 4th Cl Tail	104	2.72	0.36	0.77	87.0	8.21	0.93	2.20	0.38	1.44	3.21	1.37	1.21	1.53	1.12
Li 3rd Cl Tail	91.3	2.40	0.16	0.34	89.2	6.93	0.92	2.58	0.20	0.56	2.89	1.02	1.06	1.58	0.52
Li 2nd Cl Tail	107	2.80	0.10	0.22	87.5	7.78	1.09	3.28	0.16	0.41	3.31	1.33	1.46	2.34	0.48
Li 1st Cl Tail	200	5.25	0.06	0.12	82.6	10.6	1.56	4.83	0.28	0.45	5.87	3.41	3.93	6.47	1.59
F14 Li Ro Tail	1,286	33.7	0.07	0.16	78.4	13.0	1.88	5.93	0.33	3.66	35.8	26.9	30.4	51.1	12.0
F14 Li Ro Scav #3 -400m slimes	34.3	0.90	0.07	0.15	76.5	13.9	2.18	6.20	0.29	0.09	0.93	0.77	0.94	1.42	0.28
F14 Slimes #2	84.7	2.22	0.67	1.44	75.3	15.3	1.88	3.58	1.27	2.18	2.27	2.09	2.00	2.03	3.05
F14 Mica Conc	384	10.1	0.44	0.95	57.9	26.7	5.66	3.64	1.66	6.50	7.89	16.5	27.3	9.36	18.1
F14 Slimes #1	334	8.78	0.51	1.10	72.6	16.6	2.53	4.22	1.07	6.56	8.62	8.93	10.6	9.45	10.1
Fine Grinding-Cyclone O/F	336	8.83	0.47	1.01	68.0	18.2	3.16	4.29	1.84	6.08	8.12	9.85	13.4	9.66	17.5
Head (calc.)	3,811	100	0.68	1.47	73.9	16.3	2.09	3.92	0.93	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,140	82.4	0.72	1.56	74.7	16.1	1.92	3.85	0.81	87.4	83.3	81.2	76.0	80.9	72.3
Spod Flot Feed	2,637	69.2	0.77	1.67	77.1	14.6	1.38	3.85	0.68	78.6	72.2	61.9	45.7	68.1	50.9

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.	837	22.0	2.22	4.78	70.4	20.5	0.72	0.90	0.90	71.5	20.9	27.6	7.58	5.04	21.3
Li 4th Cl. Conc.	850	22.3	2.20	4.74	69.9	20.4	0.72	0.90	1.46	72.1	21.1	27.8	7.67	5.10	35.2
Li 3rd Cl Conc	953	25.0	2.00	4.31	71.7	19.0	0.74	1.04	1.34	73.5	24.3	29.2	8.88	6.63	36.3
Li 2nd Cl. Conc.	1,045	27.4	1.84	3.97	73.3	18.0	0.76	1.17	1.24	74.1	27.2	30.2	9.94	8.20	36.8
Li 1st Cl Conc	1,151	30.2	1.68	3.62	74.6	17.0	0.79	1.37	1.14	74.5	30.5	31.6	11.4	10.5	37.3
Li Ro Conc.	1,351	35.5	1.44	3.10	75.8	16.1	0.90	1.88	1.02	74.9	36.4	35.0	15.3	17.0	38.9

Test: F15 **Project No.:** 12607-001 **Operator:** JG **Date:** 27-May-11

Purpose: **Similar to F14, with reduced collector and increased Na silicate**

Procedure: As outlined below.

Feed: Spod Charge, Grind to 117 micron

Grind: Stage ground to -117 µm

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate 'N'**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
														9.0	
Cond. deslimed product at high density															
pH Adjustment	375													3.0	25
Condition	125	80	25									3		3.0	25
Mica Ro (4kg cell)	250											1	3	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														8.0	30
Mica Clnr #1	250	25										1	2	3.0	29
Mica Clnr #2	150	25										1	1.5	3.0	27
1st and 2nd clnr Tails to Li float															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					220				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						187.5				400		5		8.0	
Li Rougher 1 (4kg cell)						62.5							3	8.0	27
High Density Conditioning															
Li Rougher Scav 1						50			200	200			1.5	8.0	26
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0
Li Rougher Scav 3						12.5				100			1.5	8.0	
Li Rougher Scav 4 (filtered)						12.5				100			1.5	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	2	8.0	25
Li 4th Cleaner (2kg cell)						12.5	50					1	1.5	8.5	
Total	1150	130	25		220	13			450	950	0	25	24		

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
F15 WHIMS Non Mags	552	14.6	2.79	6.01	66.1	24.1	0.67	0.86	0.92	59.3	13.0	21.6	4.63	3.19	16.2
F15 WHIMS Mags	14.1	0.37	1.36	2.93	40.0	12.9	0.55	0.75	37.5	0.74	0.20	0.30	0.10	0.07	16.9
F15 Li 4th Cl Tail	95.7	2.53	1.66	3.57	74.1	18.1	1.04	2.06	0.74	6.12	2.53	2.82	1.25	1.32	2.26
F15 Li 3rd Cl Tail	69.4	1.83	0.71	1.53	80.7	12.3	1.17	3.02	0.42	1.90	2.00	1.39	1.02	1.41	0.93
F15 Li 2nd Cl Tail	97.7	2.58	0.38	0.82	83.1	10.9	1.28	3.55	0.25	1.43	2.89	1.73	1.57	2.33	0.78
F15 Li 1st Cl Tail	261	6.91	0.13	0.28	82.9	10.9	1.55	4.65	0.18	1.31	7.72	4.64	5.08	8.16	1.50
F15 Li Ro Scav #4	50.0	1.32	0.36	0.77	80.1	12.0	1.42	4.41	0.27	0.69	1.43	0.98	0.89	1.48	0.43
F15 Li Ro Tail	1,505	39.8	0.10	0.22	80.5	11.6	1.65	5.19	0.12	5.80	43.2	28.4	31.1	52.5	5.76
F15 Slimes #2	90.9	2.40	0.68	1.46	75.7	14.8	1.79	3.69	1.27	2.38	2.45	2.19	2.04	2.25	3.68
F15 Mica Conc	400	10.6	0.51	1.10	58.1	26.8	5.79	3.26	1.63	7.87	8.29	17.5	29.1	8.77	20.8
F15 Slimes #1	311	8.22	0.53	1.14	73.3	16.7	2.54	4.22	1.12	6.35	8.13	8.46	9.91	8.82	11.1
Fine Grinding-Cyclone O/F	336	8.89	0.47	1.01	68.0	18.2	3.16	4.29	1.84	6.09	8.15	9.96	13.3	9.69	19.7
Head (calc.)	3,784	100	0.69	1.48	74.1	16.2	2.11	3.93	0.83	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,137	82.9	0.72	1.56	74.9	16.0	1.95	3.87	0.69	87.6	83.7	81.6	76.8	81.5	69.2
Spod Flot Feed	1,140	30.1	1.63	3.50	73.3	18.1	1.02	2.35	1.07	71.5	29.8	33.5	14.5	18.0	38.9

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.	552	14.6	2.79	6.01	66.1	24.1	0.67	0.86	0.92	59.3	13.0	21.6	4.63	3.19	16.2
Li 4th Cl. Conc.	566	15.0	2.75	5.93	65.4	23.8	0.67	0.86	1.83	60.1	13.2	21.9	4.73	3.26	33.0
Li 3rd Cl Conc	662	17.5	2.60	5.59	66.7	23.0	0.72	1.03	1.67	66.2	15.7	24.8	5.98	4.58	35.3
Li 2nd Cl. Conc.	731	19.3	2.42	5.20	68.0	22.0	0.76	1.22	1.55	68.1	17.7	26.2	7.00	5.99	36.2
Li 1st Cl Conc	829	21.9	2.18	4.69	69.8	20.7	0.82	1.49	1.40	69.5	20.6	27.9	8.57	8.32	37.0
Li Ro Conc.	1,090	28.8	1.69	3.63	72.9	18.3	1.00	2.25	1.11	70.8	28.3	32.5	13.6	16.5	38.5

Test: F16 **Project No.:** 12607-001 **Operator**: JG **Date:** 30-May-11

Purpose: **Testing the effect of 3030 C collector on mica floatation at neutral pH**

Procedure: As outlined below.

Feed: Head Composite Charge, 2kg

Grind: Stage ground to -300 µm

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Aero 3030C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate 'N'**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**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															
Condition		80	25									3		Natural	25
Mica Ro (4kg cell)												1	2	7.8	
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															30
Mica Clnr #1		10										1	1	7.7	
Mica Clnr #2		10										1	1		
1st, and 2nd clnr Tails to Li float															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					47				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (4kg cell) 1500 RPM						6.25				600		5		8.0	
Li Rougher 1 (4kg cell)						12.5							1.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						12.5			200	200			1	8.0	
Li Rougher Scav 2						6.25				100			1	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						6.25				50		1	1	8.0	
Li 2nd Cleaner (2kg cell)						3.125						1	1	8.0	
Li 3rd Cleaner (2kg cell)						3.125						1	1	8.0	
Li 4th Cleaner (2kg cell)						3.125	50					1	1	8.5	
Total	0	100	25		47	13			450	950	0	25	12		

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
F16 WHIMS Non-Mags	374	19.5	2.97	6.39	62.5	26.1	0.71	0.72	0.97	79.4	16.4	31.4	6.91	3.66	32.1
F16 WHIMS Mags	6.60	0.34	2.00	4.31	48.0	19.8	0.99	0.67	18.6	0.94	0.22	0.42	0.17	0.06	10.8
F16 Li 4th Cl Tail	11.8	0.61	1.03	2.22	70.9	18.8	1.71	4.27	0.59	0.87	0.59	0.71	0.52	0.68	0.62
F16 Li 3rd Cl Tail	15.7	0.82	0.57	1.23	73.7	16.2	1.72	5.26	0.39	0.64	0.81	0.82	0.70	1.12	0.54
F16 Li 2nd Cl Tail	54.4	2.83	0.50	1.08	75.5	15.2	1.62	5.21	0.27	1.94	2.89	2.65	2.29	3.85	1.30
F16 Li 1st Cl Tail	111	5.80	0.06	0.13	80.1	11.9	1.57	5.53	0.05	0.48	6.27	4.25	4.54	8.36	0.49
F16 Li Ro Tail	867	45.2	0.01	0.02	81.9	10.6	1.62	5.10	0.04	0.50	49.9	29.5	36.5	60.0	3.06
F16 Slimes #2	22.8	1.19	0.73	1.57	71.3	16.9	2.23	4.12	1.46	1.19	1.14	1.24	1.32	1.28	2.94
F16 Mica Conc	302	15.7	0.38	0.82	67.0	21.0	4.53	2.97	1.10	8.19	14.2	20.3	35.5	12.2	29.3
F16 Slimes #1	76.7	4.00	0.54	1.16	70.6	17.1	2.70	4.20	1.28	2.96	3.81	4.21	5.38	4.37	8.67
Cyclone O/F	76.6	3.99	0.52	1.13	68.6	18.3	3.08	4.26	1.50	2.87	3.70	4.51	6.14	4.44	10.1
Head (calc.)	1,919	100	0.73	1.57	74.1	16.2	2.00	3.84	0.59	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	1,766	92.0	0.79	1.71	80.6	17.6	2.18	4.17	0.64	94.2	92.5	91.3	88.5	91.2	81.2
Spod Flot Feed	1,441	75.1	0.82	1.77	76.1	15.1	1.38	3.97	0.38	84.8	77.1	69.7	51.6	77.8	48.9

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.	374	19.5	2.97	6.39	62.5	26.1	0.71	0.72	0.97	79.4	16.4	31.4	6.91	3.66	32.1
Li 4th Cl. Conc.	381	19.9	2.95	6.36	62.2	26.0	0.71	0.72	1.28	80.4	16.7	31.8	7.08	3.72	42.9
Li 3rd Cl Conc	393	20.5	2.90	6.23	62.5	25.8	0.74	0.83	1.25	81.2	17.3	32.5	7.60	4.40	43.5
Li 2nd Cl. Conc.	409	21.3	2.81	6.04	62.9	25.4	0.78	1.00	1.22	81.9	18.1	33.3	8.31	5.52	44.1
Li 1st Cl Conc	463	24.1	2.54	5.46	64.4	24.2	0.88	1.49	1.11	83.8	21.0	36.0	10.6	9.37	45.4
Li Ro and Scav. Conc.	574	29.9	2.06	4.42	67.5	21.8	1.01	2.27	0.90	84.3	27.2	40.2	15.1	17.7	45.9

Test: F17 **Project No.:** 12607-001 **Operator:** JG **Date:** 06-Jun-11

Purpose: **Similar to F16 but reducing mica collector in the rougher and adding mica scavenger flotation**

Procedure: As outlined below.

Feed: Head Composite Charge, 4kg

Grind: Stage ground to -300 µm

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Aero 3030C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate 'N'**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime														7.8	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		70	25									3		7.8	25
Mica Ro (4kg cell)												1	4	7.8	
Mica Ro Scav #1		10											2	7.8	
Mica Ro Scav #2		10											2	7.8	
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															30
Mica Clnr #1		10										1	5		
Mica Clnr #2		10										1	5		
1st, and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					187.5				250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (4kg cell) 1500 RPM						50				600		5		8.0	
Li Rougher 1 (4kg cell)						12.5							2.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						25			200	200			2	8.0	
Li Rougher Scav 2						12.5				100			2	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	
Li 3rd Cleaner (2kg cell)						12.5						1	1.5	8.0	
Li 4th Cleaner (2kg cell)						12.5						1	1.5	8.5	
Li 5th Cleaner (2kg cell)						0						1	1.5	8.5	
Total	0	110	25		188	13			450	950	0	25	32		

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
F17 WHIMS Non-Mags	845	22.5	2.69	5.79	59.8	27.0	1.88	0.61	1.11	83.7	18.2	37.4	20.3	3.45	39.3
F17 WHIMS Mags	7.80	0.21	0.91	1.96	30.0	12.3	1.04	0.61	40.5	0.26	0.08	0.16	0.10	0.03	13.2
F17 Li 5th Cl Tail	41.6	1.11	1.48	3.19	60.6	25.3	3.53	2.04	1.40	2.27	0.91	1.73	1.87	0.57	2.44
F17 Li 4th Cl Tail	33.0	0.88	1.30	2.80	66.8	21.7	2.43	3.20	1.20	1.58	0.79	1.17	1.02	0.71	1.66
F17 Li 3rd Cl Tail	35.7	0.95	0.86	1.85	70.6	18.3	2.13	4.18	0.82	1.13	0.91	1.07	0.97	1.00	1.23
F17 Li 2nd Cl Tail	51.6	1.37	0.37	0.80	76.2	14.5	1.94	5.22	0.41	0.70	1.41	1.23	1.28	1.80	0.89
F17 Li 1st Cl Tail	179	4.76	0.12	0.26	80.0	12.3	1.81	5.50	0.14	0.79	5.14	3.61	4.13	6.58	1.05
F17 Li Ro Tail	2,122	56.5	0.03	0.07	81.9	11.0	1.69	5.31	0.12	2.42	62.5	38.3	45.8	75.4	10.7
F17 Slimes #2	122	3.25	0.53	1.14	66.9	17.8	2.96	4.15	1.31	2.38	2.94	3.57	4.62	3.40	6.71
F17 Mica Conc	110	2.94	0.22	0.47	50.8	31.1	8.56	1.39	2.32	0.89	2.01	5.63	12.0	1.03	10.7
F17 Slimes #1	47.4	1.26	0.48	1.03	71.2	16.9	2.72	4.54	1.01	0.84	1.21	1.31	1.65	1.44	2.01
Cyclone O/F	160	4.25	0.52	1.13	68.6	18.3	3.08	4.26	1.50	3.08	3.94	4.80	6.28	4.56	10.0
Head (calc.)	3,755	100	0.72	1.56	74.1	16.2	2.09	3.98	0.64	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,548	94.5	0.74	1.58	74.4	16.1	2.03	3.96	0.59	96.1	94.8	93.9	92.1	94.0	88.0
Spod Flot Feed	1,194	31.8	2.06	4.43	63.9	23.7	1.95	1.77	1.20	90.4	27.4	46.4	29.7	14.1	59.9

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.	845	22.5	2.69	5.79	59.8	27.0	1.88	0.61	1.11	83.7	18.2	37.4	20.3	3.45	39.3
Li 4th Cl. Conc.	853	22.7	2.67	5.76	59.5	26.9	1.87	0.61	1.47	83.9	18.3	37.6	20.4	3.48	52.6
Li 3rd Cl Conc	895	23.8	2.62	5.64	59.6	26.8	1.95	0.68	1.47	86.2	19.2	39.3	22.3	4.05	55.0
Li 2nd Cl. Conc.	928	24.7	2.57	5.53	59.8	26.6	1.97	0.77	1.46	87.8	20.0	40.5	23.3	4.76	56.7
Li 1st Cl Conc	963	25.7	2.51	5.40	60.2	26.3	1.97	0.89	1.43	88.9	20.9	41.6	24.2	5.76	57.9
Li Ro Conc.	1,015	27.0	2.40	5.16	61.0	25.7	1.97	1.11	1.38	89.6	22.3	42.8	25.5	7.56	58.8

Test: F18 **Project No.:** 12607-001 **Operator:** JG **Date:** 07-Jun-11

Purpose: **Repeat test F16 with higher pH at scrubbing and addition of mica cleaner scavenger**

Procedure: As outlined below.
Feed: Head Composite Charge,
Grind: Stage ground to -300 µm
Regrind: None
Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Aero 3030C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate 'N'**	**Pb(NO3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime														8.0	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		80	25									3		8.0	25
Mica Ro (4kg cell)												1	3.5	7.8	
Mica Ro Scav		20											2		
Clean Mica Conc. and return the tailing to the rougher tailings															30
Mica Cleaner Use appropriate Cell (maybe 2kg is good)												1	3	8.0	
Mica Clnr #1		10										1	2	7.8	
Mica Clnr #1 Scav		10													
Mica Clnr #2		10													
Mica Clnr #2 Scav		6			312.5				250			10		11.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (4kg cell) 1500 RPM						50				600		5		8.0	
Li Rougher 1 (4kg cell)						12.5							3.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						25			200	200			2	8.0	
Li Rougher Scav 2						25				100			1.5	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	2	8.0	
Li 2nd Cleaner (2kg cell)						12.5						1	1.5	8.0	
Li 3rd Cleaner (2kg cell)						12.5						1	1	8.0	
Li 4th Cleaner (2kg cell)						6.125						1	1	8.5	
Total	0	136	25		313	13			450	950	0	25	23		

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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
F18 WHIMS Non-Mags	780	20.5	3.03	6.52	62.9	25.5	0.59	0.79	0.94	85.7	17.4	32.3	5.94	4.06	30.2
F18 WHIMS Mags	9.50	0.25	1.41	3.04	42.0	14.6	0.62	0.98	30.0	0.49	0.14	0.23	0.08	0.06	11.7
F18 Li 4th Cl Tail	17.0	0.45	0.68	1.46	71.3	17.6	2.11	4.97	0.76	0.42	0.43	0.49	0.46	0.56	0.53
F18 Li 3rd Cl Tail	29.4	0.77	0.41	0.88	74.6	15.3	1.91	5.44	0.49	0.44	0.78	0.73	0.72	1.05	0.59
F18 Li 2nd Cl Tail	49.8	1.31	0.14	0.30	78.1	13.3	1.80	5.89	0.24	0.25	1.38	1.07	1.16	1.93	0.49
F18 Li 1st Cl Tail	218	5.74	0.03	0.05	80.9	11.4	1.69	5.57	0.10	0.20	6.26	4.04	4.76	8.00	0.90
F18 Li Ro Tail	1,934	50.9	0.02	0.04	82.0	10.9	1.64	5.30	0.13	1.26	56.2	34.2	40.9	67.4	10.3
F18 Slimes #2	40.5	1.07	0.69	1.49	70.6	16.4	2.10	4.29	1.37	1.01	1.01	1.08	1.10	1.14	2.28
F18 Mica Conc	403	10.6	0.30	0.65	59.6	25.7	6.33	2.50	1.59	4.38	8.52	16.8	33.0	6.64	26.4
F18 Slimes #1	158	4.15	0.50	1.08	70.9	16.9	2.71	4.47	1.04	2.85	3.96	4.32	5.51	4.64	6.74
Cyclone O/F	160	4.20	0.52	1.13	68.6	18.3	3.08	4.26	1.50	3.03	3.89	4.75	6.35	4.48	9.83
Head (calc.)	3,799	100	0.73	1.56	74.2	16.2	2.04	4.00	0.64	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,640	95.8	0.74	1.58	74.5	16.1	1.99	3.99	0.60	97.0	96.1	95.3	93.6	95.5	90.2
Spod Flot Feed	3,038	80.0	0.81	1.74	76.7	14.8	1.38	4.16	0.44	88.7	82.6	73.0	54.1	83.1	54.8

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.	780	20.5	3.03	6.52	62.9	25.5	0.59	0.79	0.94	85.7	17.4	32.3	5.94	4.06	30.2
Li 4th Cl. Conc.	790	20.8	3.01	6.48	62.6	25.4	0.59	0.79	1.29	86.2	17.5	32.5	6.02	4.12	41.9
Li 3rd Cl Conc	807	21.2	2.96	6.37	62.8	25.2	0.62	0.88	1.28	86.6	18.0	33.0	6.48	4.67	42.4
Li 2nd Cl. Conc.	836	22.0	2.87	6.18	63.2	24.9	0.67	1.04	1.25	87.0	18.8	33.7	7.21	5.73	43.0
Li 1st Cl Conc	886	23.3	2.72	5.85	64.1	24.2	0.73	1.31	1.19	87.3	20.1	34.8	8.37	7.66	43.5
Li Ro Conc.	1,104	29.1	2.19	4.71	67.4	21.7	0.92	2.15	0.98	87.5	26.4	38.8	13.1	15.7	44.4

Test: F19 **Project No.:** 12607-001 **Operator**: JG **Date:** 07-Jun-11

Purpose: Repeat Test F18, with elimination of NaOH addition in scrubbing

Procedure: As outlined below.
Feed: Head Composite Charge,
Grind: Stage ground to -300 µm
Regrind: None
Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Aero 3030C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	LR19	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime														8.0	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		80	25									3		8.0	25
Mica Ro (4kg cell)												1	2	8.0	
Mica Ro Scav		20													
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)															30
Mica Clnr #1		10										1	3	8.0	
Mica Clnr #1 Scav		10											2	7.8	
Mica Clnr #2		10										1	2		
Mica Clnr #2 Scav		6											1.5		
1st, and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)					None				250			10		7.5-8	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (4kg cell) 1500 RPM						50				600		5		8.0	
Li Rougher 1 (4kg cell)						25							3.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						25			200	200			2	8.0	
Li Rougher Scav 2						12.5				100			1.5	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						12.5				50		1	1.5	8.0	
Li 2nd Cleaner (2kg cell)						12.5						1	1.5	8.0	
Li 3rd Cleaner (2kg cell)						12.5						1	1	8.0	
Li 4th Cleaner (2kg cell)						12.5						1	1	8.5	
Total	0	136	25		0	13			450	950	0	25	23		

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
WHIMS non-mags	830	21.5	2.83	6.09	61.9	26.0	1.06	0.78	1.14	87.3	18.0	34.8	11.2	4.17	40.0
WHIMS mags	11.9	0.31	2.07	4.46	43.7	18.1	0.59	0.50	24.7	0.92	0.18	0.35	0.09	0.04	12.4
Li 4th Cl Tail	25.6	0.66	0.80	1.72	70.8	18.2	2.00	4.90	0.69	0.76	0.63	0.75	0.65	0.81	0.75
Li 3rd Cl Tail	34.4	0.89	0.50	1.08	73.5	16.3	1.92	5.60	0.49	0.64	0.88	0.90	0.84	1.24	0.71
Li 2nd Cl Tail	55.4	1.44	0.20	0.43	76.8	14.1	1.85	6.05	0.24	0.41	1.49	1.26	1.31	2.16	0.56
Li 1st Cl Tail	265	6.87	0.04	0.08	79.9	11.5	1.71	5.57	0.10	0.35	7.40	4.90	5.76	9.49	1.12
F19 Li Ro Tail	2,036	52.9	0.01	0.02	81.9	10.8	1.65	5.30	0.07	0.83	58.4	35.4	42.8	69.5	6.03
F19 Slimes #2	14.8	0.38	0.59	1.27	69.9	17.0	2.43	4.34	1.36	0.32	0.36	0.41	0.46	0.41	0.85
F19 Mica Conc	268	6.97	0.26	0.56	55.0	28.6	7.45	2.01	1.83	2.60	5.16	12.4	25.5	3.47	20.8
F19 Slimes #1	151	3.93	0.48	1.03	70.4	16.9	2.70	4.46	1.04	2.70	3.73	4.12	5.21	4.35	6.66
Cyclone O/F	160	4.15	0.52	1.13	68.6	18.3	3.08	4.26	1.50	3.11	3.84	4.72	6.26	4.38	10.1
Head (calc.)	3,851	100	0.70	1.50	74.2	16.1	2.04	4.03	0.61	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	3,692	95.9	0.71	1.52	74.4	16.0	1.99	4.02	0.58	96.9	96.2	95.3	93.7	95.6	89.9
Spod Flot Feed	3,257	84.6	0.75	1.62	76.2	14.9	1.51	4.17	0.45	91.3	86.9	78.4	62.6	87.4	61.6

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.	830	21.5	2.83	6.09	61.9	26.0	1.06	0.78	1.14	87.3	18.0	34.8	11.2	4.17	40.0
Li 4th Cl. Conc.	842	21.8	2.82	6.07	61.6	25.9	1.05	0.78	1.47	88.3	18.2	35.1	11.3	4.21	52.4
Li 3rd Cl Conc	867	22.5	2.76	5.94	61.9	25.7	1.08	0.90	1.45	89.0	18.8	35.9	11.9	5.01	53.2
Li 2nd Cl. Conc.	902	23.4	2.67	5.75	62.4	25.3	1.11	1.08	1.41	89.7	19.7	36.8	12.8	6.25	53.9
Li 1st Cl Conc	957	24.8	2.53	5.45	63.2	24.7	1.16	1.37	1.35	90.1	21.2	38.0	14.1	8.41	54.5
Li Ro Conc.	1,221	31.7	1.99	4.28	66.8	21.8	1.28	2.28	1.08	90.4	28.6	42.9	19.8	17.9	55.6

Test: F20 **Project No.:** 12607-001 **Operator**: JG **Date**: 07-Dec-11

Purpose: **Repeat Test F18 but replacing LR19 with FA-2 collector**

Procedure: As outlined below.
Feed: Head Composite Charge, 2 kg sample
Grind: Stage ground to -300 µm, charge #13
Regrind: None
Conditions: Using FA-2 collector

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Aero 3030C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate 'N'	$Pb(NO3)2$	D618	FA-2	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime														8.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		80	50									3		8.0	25
Mica Ro (2kg cell)												1	2	7.8	
Mica Ro Scav		20											1.5	8.0	
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 1kg is good)															
Mica Clnr #1		10										1	3	8.0	
Mica Clnr #2		10										1	2	7.8	
Scrubbing					175				250			10		11.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (2kg cell) 1500 RPM						75				600		5		8.0	25
Li Rougher 1 (2kg cell)													1.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						25			200	200			1	8.0	
Li Rougher Scav 2										100			1	8.0	
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (1kg cell)						0				50		1	1.5	8.0	
Li 2nd Cleaner (1kg cell)						6.25						1	1	8.0	
Li 3rd Cleaner (1kg cell)						6.25						1	1	8.0	
Li 4th Cleaner (1kg cell)						0						1	1	8.5	
Total	0	120	50		175	13			450	950	0	25	17		

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
F20 Non-Mags	403	20.5	3.00	6.46	62.6	25.7	0.76	0.78	0.92	86.6	17.3	32.9	7.71	3.95	33.2
F20 WHIMS Mags	6.50	0.33	1.30	2.80	37.4	13.2	0.71	1.53	25.6	0.61	0.17	0.27	0.12	0.12	14.9
F20 Li 4th Cl Tail	12.9	0.66	1.10	2.37	65.0	22.2	3.23	3.28	1.00	1.02	0.57	0.91	1.05	0.53	1.15
F20 Li 3rd Cl Tail	17.0	0.86	0.75	1.61	69.4	19.6	2.84	4.15	0.72	0.91	0.81	1.06	1.21	0.89	1.09
F20 Li 2nd Cl Tail	31.3	1.59	0.39	0.84	73.6	16.8	2.61	5.00	0.48	0.87	1.58	1.67	2.06	1.97	1.34
F20 Li 1st Cl Tail	98.9	5.03	0.14	0.30	77.6	13.3	2.07	5.50	0.28	0.99	5.26	4.18	5.15	6.84	2.48
F20 Li Ro Tail	1,118	56.9	0.02	0.03	81.5	10.8	1.65	5.30	0.06	1.20	62.5	38.4	46.4	74.5	6.00
F20 Slimes #2	10.0	0.51	0.57	1.23	70.9	16.9	2.54	4.42	1.24	0.41	0.49	0.54	0.64	0.56	1.11
F20 Mica Conc	84.2	4.28	0.23	0.50	50.6	31.6	8.58	1.34	2.04	1.39	2.92	8.45	18.2	1.42	15.4
F20 Mica 2nd Cl Tail	34.8	1.77	0.44	0.95	58.2	26.8	6.37	2.32	1.53	1.10	1.39	2.96	5.58	1.01	4.76
F20 Slimes #1	85.4	4.34	0.49	1.05	71.0	17.3	2.80	4.54	1.07	3.00	4.16	4.69	6.02	4.87	8.17
Cyclone O/F	64.3	3.27	0.41	0.89	65.6	19.5	3.64	4.18	1.82	1.90	2.89	3.98	5.88	3.38	10.5
Head (calc.)	1,966	100	0.71	1.53	74.2	16.0	2.02	4.05	0.57	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	1,817	92.4	0.73	1.57	74.6	15.8	1.93	4.02	0.50	95.1	93.0	91.3	88.1	91.8	81.3
Spod Flot Feed	1,688	85.8	0.76	1.64	76.2	14.8	1.50	4.19	0.40	92.2	88.2	79.4	63.7	88.8	60.1

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.	403	20.5	3.00	6.46	62.6	25.7	0.76	0.78	0.92	86.6	17.3	32.9	7.71	3.95	33.2
Li 4th Cl. Conc.	410	20.8	2.97	6.40	62.2	25.5	0.76	0.79	1.31	87.2	17.5	33.2	7.82	4.08	48.0
Li 3rd Cl Conc	423	21.5	2.92	6.28	62.3	25.4	0.83	0.87	1.30	88.2	18.0	34.1	8.87	4.61	49.2
Li 2nd Cl. Conc.	440	22.4	2.83	6.10	62.6	25.2	0.91	0.99	1.28	89.1	18.8	35.2	10.1	5.49	50.3
Li 1st Cl Conc	471	23.9	2.67	5.75	63.3	24.6	1.03	1.26	1.23	90.0	20.4	36.8	12.1	7.46	51.6
Li Ro Conc.	570	29.0	2.23	4.80	65.8	22.7	1.21	2.00	1.06	91.0	25.7	41.0	17.3	14.3	54.1

Test: F21 **Project No.:** 12607-001 **Operator**: JG **Date**: 13/07/2011

Purpose: **Repeat Test F18, using combined HL middlings and undersize fraction**

Procedure: As outlined below.

Feed: Combined HL middlings and undersize fraction, 2 kg sample

Grind: Stage ground to -300 µm,

Regrind: None

Conditions: Using LR-19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**3030C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate 'N'**	**Pb(NO3)2**	**D618**	**LR-19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime														8.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		80	50									3			
Mica Ro (2kg cell)												1	3	7.8	
Mica Ro Scav		20											2		
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 1kg is good)															
Mica Clnr #1		10										1	4		
Mica Clnr #2		10										1	3.5		
Scrubbing					175				250			10		11.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (2kg cell) 1500 RPM						75				600		5		8.0	25
Li Rougher 1 (2kg cell)													1.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						25			200	200			1.5	8.0	
Li Rougher Scav 2						25				100			1	8.0	
Submit Scav 2 for analysis															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (1kg cell)						6.25				50		1	1.5	8.0	
Li 2nd Cleaner (1kg cell)						0						1	1	8.0	
Li 3rd Cleaner (1kg cell)						0						1	1	8.0	
Li 4th Cleaner (1kg cell)						6.25						1	1	8.5	
Total	0	120	50		175	13			450	950	0	25	21		

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
F21 Li Conc Non Mags	368	17.3	2.30	4.95	58.8	27.7	2.71	0.74	1.51	69.6	14.0	27.8	19.2	3.54	31.4
F21 Mags	6.30	0.30	1.16	2.50	36.7	17.7	2.16	0.60	20.6	0.60	0.15	0.30	0.26	0.05	7.33
F21 Li 6th Cl Tail	57.8	2.72	1.10	2.37	57.7	28.6	5.37	1.53	1.56	5.23	2.15	4.51	5.97	1.15	5.09
F21 Li 5th Cl Tail	22.2	1.04	1.56	3.36	66.9	21.8	1.78	2.76	1.08	2.85	0.96	1.32	0.76	0.80	1.35
F21 Li 4th Cl Tail	24.0	1.13	1.17	2.52	69.5	19.9	1.70	3.36	0.92	2.31	1.08	1.30	0.79	1.05	1.25
F21 Li 3rd Cl Tail	36.0	1.69	0.93	2.00	72.1	17.9	1.57	3.88	0.74	2.75	1.67	1.76	1.09	1.82	1.50
F21 Li 2nd Cl Tail	55.0	2.59	0.57	1.23	76.0	15.0	1.34	4.55	0.47	2.58	2.70	2.25	1.42	3.25	1.46
F21 Li 1st Cl Tail	121	5.68	0.29	0.62	79.0	12.6	1.15	4.89	0.30	2.88	6.15	4.15	2.67	7.68	2.05
F21 Li Ro Scav 2	53.2	2.50	0.13	0.28	79.7	12.0	1.05	5.51	0.28	0.57	2.73	1.74	1.08	3.81	0.84
F21 Li Ro Tail	973	45.8	0.02	0.04	84.3	9.97	1.03	4.96	0.10	1.44	52.9	26.5	19.3	62.7	5.49
F21 Slimes #2	11.7	0.55	0.39	0.84	71.2	16.6	2.48	4.38	1.24	0.38	0.54	0.53	0.56	0.67	0.82
F21 Mica Conc	190	8.95	0.19	0.41	51.4	31.0	8.38	1.41	2.15	2.97	6.30	16.1	30.7	3.49	23.1
F21 Mica Conc 2nd Cl Tail	52.3	2.46	0.40	0.86	63.4	22.8	4.87	3.03	1.37	1.72	2.14	3.25	4.90	2.06	4.04
F21 Slimes #1	88.7	4.17	0.36	0.77	68.1	18.9	3.36	4.08	1.34	2.62	3.90	4.57	5.74	4.70	6.71
Mixed HL Smpl Cyclone O/F	66.2	3.11	0.27	0.58	62.4	21.7	4.38	3.71	2.03	1.47	2.66	3.92	5.58	3.19	7.59
Head (calc.)	2,126	100	0.57	1.23	72.9	17.2	2.44	3.62	0.83	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	1,971	92.7	0.56	1.21	66.5	13.3	1.39	3.35	0.52	95.9	93.4	91.5	88.7	92.1	85.7
Spod Flot Feed	1,717	80.8	0.61	1.31	20.5	8.39	0.89	0.52	0.51	90.8	84.5	71.6	52.5	85.9	57.8

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.	368	17.3	2.30	4.95	58.8	27.7	2.71	0.74	1.51	69.6	14.0	27.8	19.2	3.54	31.4
Li 4th Cl. Conc.	375	17.6	2.28	4.91	58.4	27.5	2.70	0.74	1.83	70.2	14.1	28.1	19.5	3.59	38.7
Li 3rd Cl Conc	433	20.3	2.12	4.57	58.3	27.7	3.06	0.84	1.79	75.5	16.3	32.6	25.5	4.74	43.8
Li 2nd Cl. Conc.	455	21.4	2.10	4.51	58.7	27.4	3.00	0.94	1.76	78.3	17.2	34.0	26.2	5.54	45.2
Li 1st Cl Conc	479	22.5	2.05	4.41	59.3	27.0	2.93	1.06	1.72	80.6	18.3	35.3	27.0	6.59	46.4
Li Ro Conc.	515	24.2	1.97	4.24	60.2	26.4	2.84	1.26	1.65	83.4	20.0	37.0	28.1	8.40	47.9

Test: F22 **Project No.:** 12607-001 **Operator**: JG **Date:** 14/07/2011

Purpose: Kinetics Tests

Procedure: As outlined below.
Feed: Head Sample, 2 kg sample
Grind: Stage ground to -300 µm,
Regrind: None
Conditions: Using LR-19 collector

	Reagents added, g/t										Time, minutes				
Stage	H2SO4	Aero 3030C	Fuel Oil	$Fe_2(S04)_3$	NaOH	Na_2CO_3	Na Silicate	Pb(NO3)2	D618	LR-19	Grind	Cond.	Froth	pH	Temp°C
	10%			2%	25%	5%		1%	5%	100%	Stage Grinding				
Mica Rougher Feed to Deslime														8.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		80	50									3			
Mica Ro 1												1	0.5		
Mica Ro 2													0.5		
Mica Ro 3													1		
Mica Ro 4													1		
Mica Ro Scav 1		20										2	0.5		
Mica Ro Scav 2													0.5		
Mica Ro Scav 3													1		
Scrubbing					175				250			10		11.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*														
Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (2kg cell) 1500 RPM						75				600		5		8.0	25
Li Rougher 1													0.5	8.0	
Li Rougher 2													0.5		
Li Rougher 3													1		
Li Rougher 4													1		
High Density Conditioning															
1st Li Rougher Scav 1						25			200	200			0.5	8.0	
1st Li Rougher Scav 2													0.5	8.0	
1st Li Rougher Scav 3													1		
2nd Li Rougher Scav 1										100			0.5	8.0	
2nd Li Rougher Scav 2													0.5	8.0	
2nd Li Rougher Scav 3													1	8.0	
Total	0	100	50		175	13			450	900	0	21	12		

Metallurgical Balance

Product	Weight		Time	Assays %							Distribution %					
	g	%	Min	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
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
F22 Li Ro 1	392	20.3	0.50	2.68	5.77	64.0	24.1	0.70	1.32	1.10	78.1	17.6	30.6	6.93	6.61	37.4
F22 Li Ro 2	53.3	2.76	0.50	1.26	2.71	71.8	17.9	1.29	3.75	0.60	5.00	2.69	3.09	1.74	2.56	2.78
F22 Li Ro Scav. 1	36.8	1.90	0.50	0.16	0.34	79.4	11.8	1.44	5.29	0.24	0.44	2.05	1.41	1.34	2.49	0.77
F22 Li Ro Scav. 2	33.4	1.73	0.50	0.07	0.15	80.3	11.6	1.55	5.50	0.21	0.18	1.88	1.26	1.31	2.35	0.61
F22 Li Ro Scav. 3	37.1	1.92	1.00	0.04	0.09	80.5	11.4	1.55	5.63	0.17	0.11	2.10	1.37	1.45	2.67	0.55
F22 Li 2nd Ro Scav. 1	31.9	1.65	0.50	0.03	0.06	80.3	11.6	1.50	5.79	0.20	0.07	1.80	1.20	1.21	2.36	0.55
F22 Li 2nd Ro Scav. 2	30.0	1.55	0.50	0.02	0.04	79.9	11.9	1.66	6.06	0.18	0.04	1.68	1.16	1.26	2.32	0.47
F22 Li Ro Tail	894	46.2		0.01	0.02	81.3	10.4	1.64	5.25	0.07	0.53	51.0	30.1	37.0	60.0	5.43
F22 Slimes #2	13.2	0.68		0.63	1.36	76.4	14.6	1.58	4.63	0.87	0.62	0.71	0.62	0.53	0.78	1.00
F22 Mica Ro Conc 1	44.0	2.27	0.50	0.43	0.93	59.8	24.6	5.66	2.70	1.44	1.41	1.85	3.51	6.30	1.52	5.50
F22 Mica Ro Conc 2	116	6.00	0.50	0.31	0.67	55.3	28.2	7.27	1.96	1.72	2.68	4.51	10.6	21.3	2.91	17.3
F22 Mica Ro Conc 3	28.8	1.49	1.00	0.59	1.27	66.1	20.0	3.69	3.68	1.14	1.26	1.34	1.87	2.69	1.35	2.85
F22 Mica Ro Conc 4	10.9	0.56	1.00	0.65	1.40	68.1	17.4	2.65	3.98	1.14	0.53	0.52	0.61	0.73	0.55	1.08
F22 Mica Ro Scav 1	31.4	1.62	0.50	0.81	1.74	68.7	19.2	2.87	3.78	1.05	1.89	1.51	1.95	2.28	1.52	2.86
F22 Mica Ro Scav 2	17.9	0.93	0.50	0.83	1.79	70.3	18.4	2.49	4.00	1.04	1.11	0.88	1.07	1.13	0.92	1.62
F22 Mica Ro Scav 3	12.9	0.67	1.00	0.79	1.70	69.8	17.1	2.23	4.08	1.00	0.76	0.63	0.71	0.73	0.67	1.12
F22 Slimes #1	86.8	4.49		0.50	1.08	70.9	17.2	2.81	4.49	1.08	3.23	4.32	4.84	6.17	4.98	8.14
Cyclone O/F	64.2	3.32		0.42	0.90	65.6	19.4	3.60	4.21	1.77	2.00	2.95	4.02	5.84	3.45	9.9
Head (calc.)	1,934	100		0.69	1.50	73.7	16.0	2.05	4.04	0.60	100	100	100	100	100	100
Head (Dir.)				0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	1,870	96.7		0.70	1.52	73.9	15.8	1.99	4.04	0.55	98.0	97.0	96.0	94.2	96.5	90.1
Spod Flot Feed	1,770	91.5		0.71	1.54	74.1	15.8	1.95	4.02	0.53	94.0	92.1	90.4	87.3	90.9	80.9
											83.1					

Combined	Weight		Time	Assays %							Distribution %					
Product	g	%	Min	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
			0.00													
Li Ro Conc 1	392	20.3	0.50	2.68	5.77	64.0	24.1	0.70	1.32	1.10	78.1	17.6	30.6	6.93	6.61	37.4
Li Ro Conc 1-2	445	23.0	1.00	2.51	5.40	64.9	23.4	0.77	1.61	1.04	83.1	20.3	33.7	8.67	9.17	40.2
Li 1st Ro Scav. Conc 1	36.8	1.90	0.50	0.16	0.34	79.4	11.8	1.44	5.29	0.24	0.44	2.05	1.41	1.34	2.49	0.77
Li 1st Ro Scav. Conc 1-2	70.2	3.63	1.00	0.12	0.25	79.8	11.7	1.49	5.39	0.23	0.61	3.93	2.66	2.65	4.84	1.38
Li 1st Ro Scav. Conc 1-3	107	5.55	2.00	0.09	0.20	80.1	11.6	1.51	5.47	0.21	0.72	6.03	4.03	4.10	7.51	1.92
Li 2nd Ro Scav Conc. 1	31.9	1.65	0.50	0.03	0.06	80.3	11.6	1.50	5.79	0.20	0.07	1.80	1.20	1.21	2.36	0.55
Li 2nd Ro Scav Conc. 1-2	61.9	3.20	1.00	0.02	0.05	80.1	11.7	1.58	5.92	0.19	0.11	3.48	2.36	2.47	4.69	1.02
Li Ro & 1st Scav Conc.	553	28.6	3.00	2.04	4.39	67.9	21.1	0.91	2.36	0.88	83.9	26.3	37.7	12.8	16.7	42.2
Li Ro & Both Scav Conc.	614	31.8	4.00	1.84	3.95	69.1	20.1	0.98	2.72	0.81	84.0	29.8	40.1	15.2	21.4	43.2
Mica Ro Conc 1	44.0	2.27	0.50	0.43	0.93	59.8	24.6	5.66	2.70	1.44	1.41	1.85	3.51	6.30	1.52	5.50
Mica Ro Conc 1-2	160	8.28	1.00	0.34	0.74	56.5	27.2	6.83	2.16	1.64	4.09	6.35	14.1	27.6	4.43	22.9
Mica Ro Conc 1-3	189	9.77	2.00	0.38	0.82	58.0	26.1	6.35	2.39	1.57	5.35	7.69	16.0	30.3	5.78	25.7
Mica Ro Conc 1-4	200	10.3	3.00	0.40	0.85	58.5	25.6	6.15	2.48	1.54	5.88	8.21	16.6	31.0	6.34	26.8
Mica Ro Scav 1	31.4	1.62	0.50	0.81	1.74	68.7	19.2	2.87	3.78	1.05	1.89	1.51	1.95	2.28	1.52	2.86
Mica Ro Scav 1-2	49.3	2.55	1.00	0.82	1.76	69.3	18.9	2.73	3.86	1.05	3.00	2.40	3.02	3.40	2.43	4.48
Mica Ro Scav 1-3	62.2	3.22	2.00	0.81	1.75	69.4	18.5	2.63	3.91	1.04	3.76	3.03	3.73	4.13	3.11	5.60
Mica Ro & Scav	262	13.5	5.00	0.49	1.06	61.1	24.0	5.31	2.82	1.42	9.63	11.2	20.3	35.2	9.44	32.4
Li Ro tail	956	49.4		0.01	0.02	81.2	10.5	1.64	5.29	0.08	0.64	54.5	32.5	39.5	64.7	6.46

Combined	Weight		Time	Assays %							Distribution %					
Product	g	%	Min	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
		0.00	0.00								**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
Li Conc. 1	392	20.3	0.50	2.68	5.77	64.0	24.1	0.70	1.32	1.10	78.1	17.6	30.6	6.93	6.61	37.4
Li Conc 1-2	445	23.0	1.00	2.51	5.40	64.9	23.4	0.77	1.61	1.04	83.1	20.3	33.7	8.67	9.17	40.2
Li Conc 1-3	482	24.9	1.50	2.33	5.02	66.0	22.5	0.82	1.89	0.98	83.6	22.3	35.1	10.0	11.7	41.0
Li Conc 1-4	515	26.6	2.00	2.18	4.70	67.0	21.8	0.87	2.13	0.93	83.8	24.2	36.4	11.3	14.0	41.6
Li Conc 1-5	553	28.6	3.00	2.04	4.39	67.9	21.1	0.91	2.36	0.88	83.9	26.3	37.7	12.8	16.7	42.2
Li Conc 1-6	584	30.2	3.50	1.93	4.16	68.5	20.6	0.95	2.55	0.84	83.9	28.1	38.9	14.0	19.0	42.7
Li Conc 1-7	614	31.8	4.00	1.84	3.95	69.1	20.1	0.98	2.72	0.81	84.0	29.8	40.1	15.2	21.4	43.2
		0.00	0.00								**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
Mica Conc 1	44.0	2.27	0.50	0.43	0.93	59.8	24.6	5.66	2.70	1.44	1.41	1.85	3.51	6.30	1.52	5.50
Mica Conc 1-2	160	8.28	1.00	0.34	0.74	56.5	27.2	6.83	2.16	1.64	4.09	6.35	14.1	27.6	4.43	22.9
Mica Conc 1-3	189	9.77	2.00	0.38	0.82	58.0	26.1	6.35	2.39	1.57	5.35	7.69	16.0	30.3	5.78	25.7
Mica Conc 1-4	200	10.3	3.00	0.40	0.85	58.5	25.6	6.15	2.48	1.54	5.88	8.21	16.6	31.0	6.34	26.8
Mica Conc 1-5	231	12.0	3.50	0.45	0.97	59.9	24.8	5.70	2.66	1.48	7.77	9.72	18.5	33.3	7.86	29.6
Mica Conc 1-6	249	12.9	4.00	0.48	1.03	60.7	24.3	5.47	2.75	1.44	8.88	10.6	19.6	34.5	8.77	31.3
Mica Conc 1-7	262	13.5	5.00	0.49	1.06	61.1	24.0	5.31	2.82	1.42	9.63	11.2	20.3	35.2	9.44	32.4

Test: F23 **Project No.:** 12607-001 **Operator**: JG **Date:** 29-Jul-11

Purpose: **Repeat Test F21 with some modifications to better float mica and depress it in the spodumene flotation**

Procedure: As outlined below.

Feed: Combined HL middlings and undersize fraction, 2 kg sample

Grind: Stage ground to -300 µm,

Regrind: None

Conditions: Using LR-19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**3030 C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na2F6Si**	**Pb(NO3)2**	**D618**	**LR-19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime														8.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
Condition		80	60									3			
Mica Ro (2kg cell)												1	3	7.7	
Mica Ro Scav 1		20											2		
Mica Ro Scav 2		20											2		
Mica Ro Scav 3		20											1.5		
Mica Ro Scav 4		10											1		
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 1kg is good)															
Mica Clnr #1 (2kg cell)		30										1	5	7.8	
Mica Clnr #2 (1kg cell)		25										1	4		
Scrubbing					175				250			10		11.0	
Deslime	*Use 16 L acrylic cylinder fill to 11L settle 8 minutes decant by syphon 2 times*							**Deslime not needed**							
Condition 1 (2kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	
Condition 2 (2kg cell) 1500 RPM						50				400		5		8.0	25
Li Rougher 1 (2kg cell)													1.5	8.0	
High Density Conditioning															
Li Rougher Scav 1						25			200	100			1.5	8.0	
Combine Rougher Concentrate and Rougher Scav#1															
Li 1st Cleaner (1kg cell)						25	20			50		1	1	8.0	
Li 2nd Cleaner (1kg cell)						25	20					1	1	8.0	
Li 3rd Cleaner (1kg cell)						0	0					1	1	8.0	
Add Extra Cleaner if required, with sodium fluorosilicate															
Total	0	205	60		175	13			450	550	0	24	25		

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
F23 Whims Non Mags	291	13.8	2.80	6.03	62.1	25.1	0.95	1.00	1.16	65.6	12.0	20.3	5.35	3.82	18.6
F23 Whims Mags	4.00	0.19	1.50	3.23	43.3	17.6	1.13	0.92	14.1	0.48	0.11	0.20	0.09	0.05	3.10
F23 Li 3rd Cl Tail	16.3	0.78	1.30	2.80	68.8	19.1	1.47	3.50	0.88	1.71	0.74	0.87	0.46	0.75	0.79
F23 Li 2nd Cl Tail	27.4	1.30	0.95	2.04	74.9	17.4	1.38	4.16	0.66	2.10	1.36	1.33	0.73	1.50	1.00
F23 Li 1st Cl Tail	52.6	2.50	0.36	0.77	78.9	12.7	1.18	4.80	0.33	1.53	2.75	1.86	1.20	3.31	0.96
F23 Li Ro Tail	740	35.2	0.05	0.11	81.6	9.70	1.00	4.88	0.21	2.92	40.1	20.0	14.3	47.4	8.55
F23 Mica Conc	816	38.8	0.33	0.71	67.0	20.7	4.22	3.28	1.19	21.7	36.2	46.9	66.6	35.1	53.4
F23 Slimes #1	89.0	4.23	0.35	0.75	67.9	18.5	3.30	4.13	1.27	2.51	4.01	4.58	5.68	4.82	6.22
Mixed HL Smpl Cyclone O/F	66.2	3.15	0.27	0.58	62.4	21.7	4.38	3.71	2.03	1.44	2.74	3.99	5.61	3.22	7.40
Head (calc.)	2,102	100	0.59	1.27	71.7	17.1	2.46	3.62	0.86	100	100	100	100	100	100
Head (Dir.)			0.69	1.49	75.6	16.0	2.01	3.92	0.48						
Flotation Feed	2,036	96.9	0.60	1.29	72.0	17.0	2.40	3.62	0.83	98.6	97.3	96.0	94.4	96.8	92.6
Spod Flot Feed	2,036	96.9	0.59	1.26	69.1	16.2	2.25	3.44	0.77	98.6	97.3	96.0	94.4	96.8	92.6
										66.6					

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.	291	13.8	2.80	6.03	62.1	25.1	0.95	1.00	1.16	65.6	12.0	20.3	5.35	3.82	18.6
Li 3rd Cl Conc	295	14.0	2.78	5.99	61.8	25.0	0.95	1.00	1.34	66.1	12.1	20.5	5.44	3.87	21.7
Li 2nd Cl. Conc.	311	14.8	2.70	5.82	62.2	24.7	0.98	1.13	1.31	67.8	12.8	21.4	5.90	4.62	22.5
Li 1st Cl Conc	339	16.1	2.56	5.52	63.2	24.1	1.01	1.37	1.26	69.9	14.2	22.7	6.63	6.11	23.5
Li Ro Conc.	391	18.6	2.27	4.88	65.3	22.6	1.03	1.84	1.13	71.5	17.0	24.6	7.83	9.43	24.4

Appendix E – Locked Cycle Testwork

Test: F10 **Project No.**: 12607-001 **Operator**: JG and YC **Date**: 20-Apr-11

Purpose: **To Duplicate test F10 For LCT No. 1**

Procedure: As outlined below.

Feed: 10kg charge

Grind: Stage ground to -48 mesh -300 µm,

Regrind: None

Conditions: Using LR19 collector

	Reagents added, g/t										Time, minutes				
Stage	**H2SO4**	**Armac C**	**Fuel Oil**	**$Fe_2(S04)_3$**	**NaOH**	**Na_2CO_3**	**Na Silicate**	**Pb(NO 3)2**	**D618**	**LR19**	**Grind**	**Cond.**	**Froth**	**pH**	**Temp°C**
	10%			**2%**	**25%**	**5%**		**1%**	**5%**	**100%**	**Stage Grinding**				
Mica Rougher Feed to Deslime															
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 6 minutes decant by syphon 1 time*														
Cond. deslimed product at high density															
pH Adjustment	375													3.0	25
Condition	125	80	25					25				3		3.0	25
Mica Ro (4kg cell)	250											1	3	3.0	24
Clean Mica Conc. and return the tailing to the rougher tailings															
Mica Cleaner Use appropriate Cell (maybe 2kg is good)														3.0	
Mica Clnr #1	250	10										1	2.5	3.0	
Mica Clnr #2	150											1	2.5	3.0	
1st, and 2nd clnr Tails to Li float.															
Mica Rougher Feed to Scrub															
Scrub in 4 kg cell (high density)									250			10		10.5	
Deslime	*Use 16 L acrylic cylinder fill to 13L settle 8 minutes decant by syphon 2 times*														
Condition 1 (4kg cell) 1500 RPM Raise pulp Temp to at least 25'C														8.0	30
Condition 2 (4kg cell) 1500 RPM						125				600		5		8.0	
Li Rougher 1 (4kg cell)						37.5							3	8.0	27
Li Rougher Scav 1				50		25			200	200			2.5	8.0	26
Li Rougher Scav 2						12.5				100			1.5	8.0	25.0
Please carefully examine the tailings and add scav if required. We need to keep the loss to a very low level.															
Combine Rougher Concentrate and Rougher Scav#1.															
Li 1st Cleaner (2kg cell)						25				50		1	2.5	8.0	21
Li 2nd Cleaner (2kg cell)						12.5						1	2	8.0	24
Li 3rd Cleaner (2kg cell)						12.5						1	2	8.0	25
Total	1150	90	25	50	0	250			450	950	0	24	22	.	

Metallurgical Balance

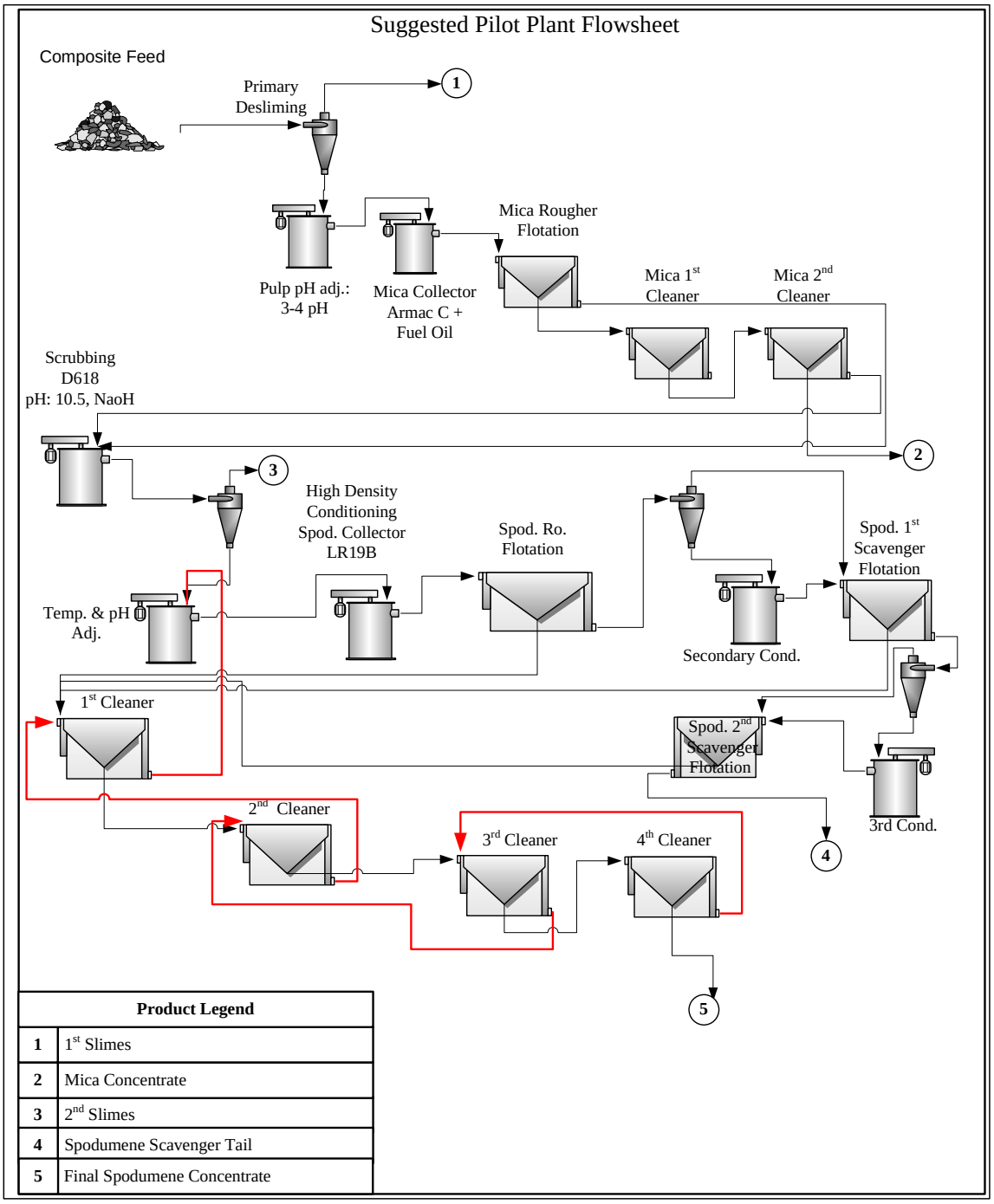
Product	Weight		Assays %							Distribution %					
	g	%	Li	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3	Li_2O	SiO_2	Al_2O_3	K_2O	Na_2O	Fe_2O_3
Li Final Conc A	654	2.66	3.14	6.76	63.8	25.8	0.62	0.76	1.39	11.7	2.27	4.25	0.80	0.51	1.74
Li Final Conc B	729	2.96	2.85	6.13	63.3	25.3	1.00	1.05	1.42	11.8	2.51	4.65	1.45	0.79	1.98
Li Final Conc C	681	2.77	3.02	6.50	62.8	25.9	0.84	0.78	1.51	11.7	2.33	4.45	1.13	0.55	1.97
Li Final Conc D	751	3.05	2.77	5.96	63.7	25.3	1.07	0.98	1.48	11.8	2.60	4.78	1.59	0.76	2.13
Li Final Conc E	759	3.08	2.77	5.96	62.9	25.6	1.28	0.98	1.48	12.0	2.59	4.89	1.93	0.77	2.15
Li Final Conc F	634	2.58	2.89	6.22	62.1	26.1	1.18	0.84	1.64	10.4	2.14	4.17	1.48	0.55	1.99
Li Final Conc G	687	2.79	3.02	6.50	63.1	25.7	1.01	0.87	1.42	11.8	2.36	4.44	1.37	0.62	1.87
Li 3rd Clnr Tls G	46.9	0.19	1.13	2.43	72.0	18.2	1.79	3.68	0.66	0.30	0.18	0.21	0.17	0.18	0.06
Li 2nd Cln Tls G	89.4	0.36	0.68	1.46	75.2	16.0	1.79	4.47	0.41	0.35	0.37	0.36	0.32	0.41	0.07
Li 1st Clnr Tls G	258	1.05	0.21	0.45	79.9	12.6	1.71	5.09	0.16	0.31	1.12	0.82	0.87	1.35	0.08
Li Ro Scav. Tls A	1,911	7.76	0.03	0.05	82.1	10.9	1.63	5.15	1.63	0.27	8.53	5.25	6.17	10.1	5.97
Li Ro Scav. Tls B	2,045	8.31	0.03	0.06	82.2	10.7	1.52	5.13	1.52	0.31	9.14	5.51	6.16	10.8	5.96
Li Ro Scav. Tls C	2,176	8.84	0.04	0.08	82.0	11.2	1.66	5.13	1.66	0.47	9.70	6.14	7.16	11.5	6.92
Li Ro Scav. Tls D	2,080	8.45	0.04	0.08	82.0	11.0	1.61	5.12	1.61	0.45	9.27	5.76	6.64	11.0	6.42
Li Ro Scav. Tls E	2,098	8.52	0.04	0.08	81.8	10.9	1.58	5.11	1.58	0.44	9.33	5.76	6.57	11.0	6.35
Li Ro Scav. Tls F	2,268	9.21	0.14	0.30	81.0	11.7	1.61	5.01	1.61	1.81	9.99	6.68	7.24	11.7	7.00
Li Ro Scav. Tls G	2,091	8.49	0.08	0.16	81.5	11.2	1.61	5.05	1.61	0.89	9.26	5.90	6.67	10.9	6.45
Slimes #2 A	33.6	0.14	0.69	1.49	70.7	17.3	2.38	3.85	2.38	0.13	0.13	0.15	0.16	0.13	0.15
Slimes #2 B	36.8	0.15	0.65	1.40	73.2	16.0	2.16	3.96	2.16	0.14	0.15	0.15	0.16	0.15	0.15
Slimes #2 C	32.4	0.13	0.64	1.38	68.8	18.5	2.84	3.77	2.84	0.12	0.12	0.15	0.18	0.13	0.18
Slimes #2 D	48.9	0.20	0.69	1.49	69.7	18.0	2.62	3.78	2.62	0.19	0.19	0.22	0.25	0.19	0.25
Slimes #2 E	54.2	0.22	0.67	1.44	70.2	17.5	2.53	3.89	2.53	0.21	0.21	0.24	0.27	0.22	0.26
Slimes #2 F	67.7	0.27	0.65	1.40	73.2	16.4	2.24	4.02	2.24	0.25	0.27	0.28	0.30	0.28	0.29
Slimes #2 G	67.6	0.27	0.65	1.40	70.9	17.5	2.60	3.95	2.60	0.25	0.26	0.30	0.35	0.28	0.34
Mica Conc A	488	1.98	0.50	1.08	61.7	24.3	5.21	2.96	5.21	1.39	1.64	2.99	5.04	1.49	4.87
Mica Conc B	412	1.67	0.52	1.12	59.2	26.1	5.71	2.70	5.71	1.22	1.33	2.71	4.66	1.15	4.51
Mica Conc C	382	1.55	0.43	0.93	59.6	25.6	5.72	3.02	5.72	0.94	1.24	2.47	4.34	1.19	4.19
Mica Conc D	396	1.61	0.43	0.93	60.6	25.4	5.61	3.04	5.61	0.97	1.30	2.53	4.40	1.24	4.25
Mica Conc E	363	1.47	0.41	0.88	61.1	24.9	5.49	3.20	5.49	0.85	1.21	2.28	3.95	1.20	3.82
Mica Conc F	278	1.13	0.37	0.80	56.8	27.8	6.53	2.94	6.53	0.59	0.86	1.95	3.60	0.84	3.48
Mica Conc G	258	1.05	0.34	0.73	55.8	28.4	6.81	2.76	6.81	0.50	0.78	1.85	3.49	0.73	3.37
Slimes #1 A	144	0.58	0.53	1.14	71.7	17.0	2.61	4.41	2.61	0.43	0.56	0.62	0.74	0.65	0.72
Slimes #1 B	119	0.48	0.52	1.12	71.7	17.1	2.65	4.41	2.65	0.35	0.46	0.51	0.63	0.54	0.60
Slimes #1 C	136	0.55	0.52	1.12	71.7	17.1	2.66	4.39	2.66	0.40	0.53	0.58	0.71	0.61	0.69
Slimes #1 D	116	0.47	0.54	1.16	71.8	17.0	2.60	4.39	2.60	0.36	0.45	0.50	0.60	0.52	0.58
Slimes #1 E	136	0.55	0.55	1.18	71.4	16.9	2.63	4.34	2.63	0.43	0.53	0.58	0.71	0.61	0.69
Slimes #1 F	161	0.65	0.53	1.14	72.2	17.1	2.60	4.40	2.60	0.49	0.63	0.69	0.83	0.73	0.80
Slimes #1 G	120	0.49	0.54	1.16	71.7	17.2	2.65	4.38	2.65	0.37	0.47	0.52	0.63	0.54	0.61
Cyclone Overflow A	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Cyclone Overflow B	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Cyclone Overflow C	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Cyclone Overflow D	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Cyclone Overflow E	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Cyclone Overflow F	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Cyclone Overflow G	117	0.47	0.52	1.13	67.2	18.1	3.87	3.61	3.87	0.35	0.43	0.53	0.90	0.43	0.87
Head (calc.)	24,626	100	0.71	1.53	74.7	16.1	2.05	3.94	2.12	100	100	100	100	100	100
Direct															
Flotation Feed	23,808	96.7	0.72	1.55	74.9	16.1	1.99	3.95	2.06	97.6	97.0	96.3	93.7	97.0	93.9
	4,104														

Combined Products

Combined	**Weight**		**Assays %**							**Distribution %**					
Product	g	%	**Li**	**Li_2O**	**SiO_2**	**Al_2O_3**	**K_2O**	**Na_2O**	**Fe_2O_3**	**Li_2O**	**SiO_2**	**Al_2O_3**	**K_2O**	**Na_2O**	**Fe_2O_3**
Li 3rd Cl Conc	4,895	19.9	2.92	6.28	63.1	25.7	1.01	0.90	1.48	81.4	16.8	31.6	9.76	4.54	13.8
Li Scav. Tail	14,669	59.6	0.06	0.12	81.8	11.1	1.60	5.10	1.60	4.65	65.2	41.0	46.6	77.1	45.1
Slime 2	341	1.39	0.66	1.42	71.1	17.3	2.47	3.90	2.47	1.29	1.32	1.48	1.67	1.37	1.62
Mica Conce.	2,577	10.5	0.44	0.95	59.6	25.8	5.77	2.95	5.77	6.45	8.35	16.8	29.5	7.83	28.5
Slime 1	932	3.78	0.53	1.15	71.8	17.1	2.63	4.39	2.63	2.83	3.64	4.00	4.85	4.21	4.69
Cyclone Overflow	819	3.32	0.52	1.13	67.2	18.1	3.87	3.61	3.87	2.44	2.99	3.72	6.27	3.04	6.07
Li 3rd Cleaner Conc.	4,895	20.6	2.82	6.07	61.0	24.8	0.97	0.87	1.43	81.4	16.8	31.6	9.76	4.54	13.8
Li 2nd Cleaner Conc.	5,271	22.1	2.70	5.81	61.6	24.3	1.03	1.06	1.37	83.8	18.3	33.4	11.1	5.96	14.3
Li 1st Cleaner Conc.	5,986	25.1	2.45	5.28	63.0	23.2	1.11	1.45	1.25	86.6	21.2	36.2	13.6	9.25	14.9
Li Rougher Conc.	8,048	33.8	1.88	4.04	66.6	20.4	1.25	2.34	0.97	89.0	30.2	42.8	20.6	20.1	15.5

Projected Balance Cycles D to H

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 3rd Cl Conc	4,953	20.1	2.86	6.15	63.0	25.7	1.14	0.92	1.50	81.5	17.2	32.4	11.3	4.77	14.4
Li Scav. Tail	14,939	60.7	0.07	0.16	81.6	11.2	1.60	5.07	1.60	6.36	67.1	42.7	48.0	79.1	46.4
Slime 2	417	1.69	0.66	1.43	71.1	17.3	2.49	3.92	2.49	1.59	1.63	1.84	2.08	1.71	2.01
Mica Conce.	2,266	9.20	0.39	0.85	59.0	26.4	6.01	3.01	6.01	5.14	7.36	15.2	27.3	7.11	26.4
Slime 1	934	3.79	0.54	1.16	71.8	17.0	2.62	4.38	2.62	2.90	3.69	4.06	4.91	4.27	4.73
Cyclone Overflow	819	3.32	0.52	1.13	67.2	18.1	3.87	3.61	3.87	2.47	3.03	3.77	6.35	3.08	6.13
Total Feed	24,327	98.8	0.70	1.52	73.7	15.9	2.02	3.89	2.10	100	100	100	100	100	100



Product Legend	
1	1st Slimes
2	Mica Concentrate
3	2nd Slimes
4	Spodumene Scavenger Tail
5	Final Spodumene Concentrate

An Investigation into

THE FEASIBILITY OF GENERATING A HIGH GRADE LITHIUM CARBONATE SAMPLE FROM THE GEORGIA LAKE ORE

prepared for

ROCK TECH LITHIUM INC.

Project 12607-002 Final Report
August 26, 2011

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Executive Summary

Rock Tech Lithium Inc. (Rock Tech) is a Canadian based resource company that is involved in the exploration and development of potential hard rock lithium resources. In particular, Rock Tech owns a property in the Thunder Bay Mining District of north-western Ontario (Georgia Lake Project (11,481 hectares)) that has a historic resource of 9.8 million tonnes of lithium oxide (Li_2O) at a grade of 1.18%.

Roch Tech in cooperation with SGS Minerals Services (SGS) conducted a scoping study on a sample from the Georgia Lake property. The first part of the scoping study involved mineral processing testing that focused on the comminution and beneficiation processes to produce a concentrate of sufficient Li grade required for lithium carbonate production. This work produced a concentrate that was then subjected to the standard hydrometallurgical flow sheet for the production of lithium carbonate (Li_2CO_3). The objective of the second part was to demonstrate that a high grade lithium carbonate product could be obtained for the concentrate of Rock Tech's Georgia Lake ore.

The comminution and beneficiation testing was conducted by Mineral Processing Department of SGS and will be reported separately. The second part of the study involved a multi-step hydrometallurgical process. This work was performed by Hydrometallurgy Department of SGS and the details are presented in this report.

Process optimization was not a direct objective of the hydrometallurgical program. The data obtained during this program will be used to define a future process development and optimization study for the Georgia Lake ore. The hydrometallurgical processes used in the program are fairly robust and may be applied without further process optimization to achieve the objective of producing a market sample of Li_2CO_3.

This program did determine that a high grade Li_2CO_3 product could be produced from Rock Tech's Georgia Lake ore. A final product having an average grade of 99.988% was produced. A standard hydrometallurgical flow sheet was used to achieve this result.

This program did not identify any distinct process advantages for one concentrate over the other concentrate samples.

The simplified process diagram of the hydrometallurgical approach is provided in Figure 1. This flow diagram was developed at SGS (Lakefield site). Some aspects of the process have not been finalized or optimized. Aspects to be optimized or finalized prior to entering a pilot plant program may include, but are not limited to, 1) the solid-liquid addition ratio in the Concentrate Leach process, 2) whether a solid-liquid separation stage is required prior to Primary Impurity Removal, 3) retention time in all stages prior

to Ion Exchange (IX), 4) pH optimization in the impurity removal stages, and 5) resin selection for IX. The key findings and results of each process unit are summarized below.

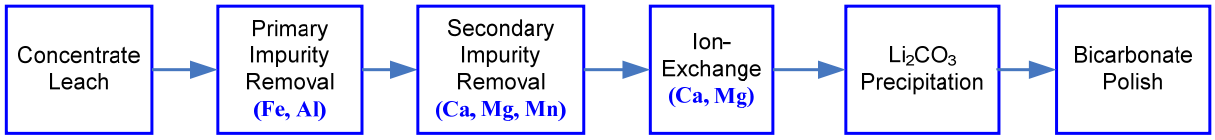

Figure 1 Simplified Flow sheet for the Production of Li_2CO_3

Concentrate Leach

The application of the standard concentrate leach to the three concentrate samples produced good lithium (Li) extractions in the range 96 to 97%. The Li loss for each of the concentrate leaches of the Low and High iron (Fe) Concentrate (Con) samples was 3.0%. The Li loss was higher for the concentrate leach of the Heavy Media Con sample at 3.7%.

Primary Impurity Removal (PIR)

The process objectives of the PIR stage were achieved for each of the concentrate samples that were tested. This demonstrated the robust nature of the PIR process. The PIR tests determined that a pH target of 5.8 is a suitable endpoint for the precipitation process. The Li % precipitation was lowest for the Low Fe Con at 0.15%. It was observed to be the highest for the Heavy Media Con. Any Li units precipitated during the PIR stage are not readily recovered and represent a potential loss to the process.

The lime consumptions during the PIR of the Low and High Fe Cons were similar at ~0.15 kg dry lime per kg lithium carbonate equivalent (LCE). This is lower than the typical consumption value for this process. The lime consumption for the Heavy Media Con sample was 0.25 kg dry lime per kg LCE. This is the typical lime consumption rate for similar systems.

Secondary Impurity Removal (SIR)

The process objectives of the SIR were not completely realized. The SIR step did not effectively remove Ca from the Li process solution.

The reagent consumptions of the SIR process were 0.0011 kg sodium hydroxide (NaOH)/ kg LCE and 0.016 kg (sodium carbonate) Na_2CO_3/ kg LCE.

Ion-Exchange (IX)

Mixed results were obtained when the standard IX parameters, which have been developed to treat Li process solutions, were applied to the Rock Tech's SIR product solution. The ion-exchange process

effectively reduced the magnesium (Mg) tenor but not the calcium (Ca) tenor in the Li process solution to below 10 mg/L. This is attributed to the fact that the IX feed solution had a greater than expected Ca tenor. The standard IX parameters that were used could not accommodate the higher Ca level.

The Li loss during the IX step was ~0.8%. The Li units that were absorbed by the media can be recovered as a lithium chloride (LiCl) solution that is generated during the Acid Strip IX cycle.

Lithium Carbonate Precipitation

The lithium carbonate that was produced from the precipitation reaction had an approximate grade of 99.8% based on the contained impurities. This sample contained sodium (Na) and Ca at levels above the maximum allowable limits for battery grade Li_2CO_3. As a result the product had to be polished in order to remove the remaining impurities. The average precipitation efficiency for Li was observed to be 76%.

Bicarbonate Polishing

The Bicarbonate Polishing procedure was successfully used to upgrade the initial Li_2CO_3 product. The first bicarbonate polishing test produced Li_2CO_3 having a grade of 99.976%. The solids met all of the product specification for battery grade Li_2CO_3 except for Fe (7 g/t). The Fe was introduced to the process during the bicarbonate step. The source was most likely the stainless steel (316) heating coils that were used.

The solids were subjected to a second bicarbonate polishing step. The resulting solids met all of the product specifications and had a Li_2CO_3 grade of 99.988%.

Introduction

Rock Tech Lithium Inc. contracted SGS Minerals Services (SGS) to evaluate the application of the standard lithium carbonate process flow sheet to a sample from their Georgia Lake deposit. This program focused on the Hydrometallurgical part of the overall flow sheet, though a discussion of the conversion of the concentrate of the ore to the feed for the down stream processes is presented. Three different concentrate samples were examined. These samples were subjected to the standard hydrometallurgical flow sheet with the objective of determining if a high grade product could be produced. This report discusses the experiments and results of the hydrometallurgical program that was conducted from April through to July 2011 at SGS (Lakefield site). The Rock Tech Lithium representative for the program was Mr. Afzaal Pirzada.

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Test Programme Summary

1. Programme Description

The objective of the Rock Tech Lithium laboratory program was to determine if the standard lithium carbonate process flow sheet could be applied to the concentrate generated from the Georgia Lake Project deposit to produce high grade Li_2CO_3. The program examined three concentrate samples. Each sample was separately subjected to the following pyro- and hydrometallurgical process steps:

- Phase Transition;
- Acid Bake;
- Concentrate Leach;
- Primary Impurity Removal (PIR).

After the PIR step the three Li process solutions were combined to produce one solution for the balance of the test program.

2. Concentrate Samples

Three concentrate samples were provided for the Hydrometallurgical test program. Two of the concentrates (Con) were produced using flotation methods – Low Fe Con and High Fe Con. The Low Fe Con sample had one additional processing step through magnetic separation to remove the magnetic iron particles. The third sample (Heavy Media Con) was produced using heavy liquid separation techniques. The elemental composition of each concentrate is summarized in Table 1.

Table 1 Elemental Composition of the Concentrate Samples

Concentrate	Li %	Al g/t	Fe g/t	Si %	Ba g/t	Be g/t	Ca g/t	Co g/t	Cr g/t	Cu g/t
Low Fe Con	3.06	136000	6410	28.8	47	272	4260	< 4	49	13.1
High Fe Con	2.79	132000	9570	29.5	63	409	3450	7	754	32.7
Heavy Media Con	2.92	130000	7680	30.2	44.5	65.2	1920	< 4	213	6.7
Sample ID	K g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Sn g/t	Sr g/t	Ti g/t
Low Fe Con	6170	382	1660	< 10	3330	< 20	1120	142	49.2	48
High Fe Con	8400	387	1770	79	6480	389	1340	157	48.6	45.2
Heavy Media Con	5300	296	1560	< 5	4720	< 20	474	145	36	30
Sample ID	Y g/t	Zn g/t	S %	Elements Assaying below Detection						
Low Fe Con	1.1	81	< 0.01	Ag	As	Bi	Cd	Pb	Sb	Se
High Fe Con	1.3	58	0.04	Tl	U	V				
Heavy Media Con	< 0.4	72	< 0.01							

The Low Fe Con sample contained 33% less Fe than the High Fe Con. The Low and High Fe designations are used to identify the two concentrate samples specific to this program. It should not be interpreted that a level of 9570 g/t Fe is overly high and may possibly cause downstream processing issues. Concentrates having Fe levels of ≤12000 g/t are readily treated by the Primary Impurity Removal circuit. The absolute and percentage decrease values of the assayed content for select elements in the magnetic separation concentrate (Low Fe Con) are summarized in Table 2. Significant decreases in the concentrate's content of chromium (Cr), molybdenum (Mo), nickel (Ni) and sulphur (S) were observed.

Table 2 Decrease in Assayed Content as a Result of Magnetic Separation

Element	Absolute Decrease in Content, Δg/t	% Decrease, %
Fe	3160	33.0
Be	137	33.5
Cr	705	93.5
K	2230	26.5
Mo	69	87.3
Na	3150	48.6
Ni	369	94.9
S, Δ%	0.03	75.0

The removal of the magnetic fraction to produce the Low Fe Con resulted in an overall mass reduction of the sample. This may explain the higher %Li in the Low Fe Con sample (3.06%) as compared to the High Fe Con sample (2.79%).[1]

3. Preparation of the Acid Roasted β-Spodumene Samples

The contained lithium in the concentrate samples was as spodumene (monoclinic) or α-spodumene. Lithium in α-spodumene is unavailable to acidic extraction. The α-spodumene must be converted to β-spodumene (spodumene having a tetragonal crystal lattice) in order to extract the lithium using acid. The phase transition from the monoclinic to tetragonal form of spodumene is achieved by heating the concentrate at 1050 °C for ~15 minutes. As a result roasting the crystal lattice has expanded and this is accompanied by a decrease in the specific gravity of the spodumene from 3.1 g/cm^3 (α-spodumene) to ~2.4 g/cm^3 (β-spodumene). In this less dense form the contained lithium may be chemically extracted.

The extraction process involves mixing treating the cooled β-spodumene with concentrated sulphuric acid and baked at a temperature at 250 °C. The acid reacts with the lithium aluminum silicate to form lithium sulphate, which is water soluble, and hydrogen aluminum silicate.

[1] *Refer to the separate Report on the Mineral Processing part of the overall flow sheet for the details of the Magnetic Separation testing.*

The main mineralogical differences between the three concentrate samples are summarized in Table 3. The relative amount of α-spodumene in concentrate is significantly lower in the High Fe Con as compared to the other two samples. The High Fe Con also has the highest amount of quartz. All three concentrates should be considered as low Fe samples.

Table 3 Select Semi-Quantitative XRD Data for the Concentrate Samples

Mineral	Chemical Composition	Low Fe Con (Head) (wt %)	High Fe Con (Head) (wt %)	Heavy Media Con (Head) (wt %)
Quartz	SiO_2	3.3	10	6.8
Spodumene (Monoclinic)	$LiAlSi_2O_6$	85.6	69.4	81.7
Hematite	Fe_2O_3	-	0.8	
Muscovite	$KAl_2(AlSi_3O_{10})(F,OH)_2$	4.7	8.2	
Albite	$NaAlSi_3O_8$	2.9	3.7	5.7
Goethite	αFeO·OH			0.8

Each of the concentrate samples were subjected to the high temperature roast (Phase Transition), acid roasting and concentrate leach. The changes in the Li mineralogy were tracked by semi-quantitative X-ray diffraction (SQ XRD). This will be illustrated for the Low Fe Con.

The SQ XRD data tracking the mineralogical changes in the Low Fe Con as it is processed through the Phase Transition, Acid Roast and Concentrate Leach are summarized in Table 4. Only the Li-bearing minerals and hydrogen aluminum silicates are listed. The complete SQ XRD report is provided in the Appendices.

The Low Fe Con was determined to be 85.6% by weight α-spodumene (Low Fe Con (Head). After the phase transition the Li in the solids are now 74.3% β-spodumene and 13% lithium aluminum silicate (hexagonal crystal lattice). The data indicate that the phase transition was 100% efficient for the conversion of α-spodumene. When the phase transition solids are treated with sulphuric acid both Li minerals are converted to lithium sulphate monohydrate. Subjecting the acid roasted solids to the concentrate leach produces a leach residue that is ~88% hydrogen aluminum silicate (hexagonal and tetragonal). No Li-bearing minerals were detected in the residue.

Table 4 SQ XRD for the Low Fe Concentrate

Mineral	Chemical Composition	Low Fe Con (Head) (wt %)	Low Fe Con Phase Transfer Product (wt %)	Low Fe Con Acid Roast Product (wt %)	Low Fe Con Concentrate Leach Residue (wt %)
Spodumene (Monoclinic)	$LiAlSi_2O_6$	85.6	-	-	
Spodumene Beta (Tetragonal)	$LiAlSi_2O_6$	-	74.3	-	
Lithium Aluminum Silicate (Hexagonal)	$LiAlSi_2O_6$	-	13	-	
Lithium Sulfate Hydrate (Monoclinic)	Li_2SO_4•H2O	-	-	25.9	
Hydrogen Alumninum Silicate (Hexagonal)	$H(AlSi_2)O_6$	-	-	0.8	4.8
Hydrogen Alumninum Silicate (Tetragonal)	$H(AlSi_2)O_6$	-	-	64.6	84.1
TOTAL		**100.0**	**100.0**	**100.0**	**100.1**

A similar exercise was conducted for the High Fe and Heavy Media concentrates. The details of which are provided in the Appendices.

4. Initial Product[2]

Rock Tech required a small quantity of high grade Li_2CO_3 in advance of the established program schedule. The main program was stopped and work focused on meeting Rock Tech's request. A quantity of the Low Fe Con was used to produce the initial sample of Li_2CO_3. Standard procedures were used for the production of the sample. In order to meet the required schedule minimal assaying was conducted during the production of the initial sample. Only key process solutions and solids were assayed. Where assay data were not available best estimates were used in order to advance the process.

The Low Fe AR β-spodumene was subjected to a standard leach with water (50% solids; 15 minutes reaction time). The pregnant leach solution (PLS) and the first wash fraction were combined and this solution was used going forward in the process to produce the sample of Li_2CO_3. The combined solution was only assayed for Li, Al and Fe. The assay data are provided in Table 5.

Table 5 Initial Product: Concentrate Leach Process Solution Assay Data

Concentrate Leach Process Solution	Solution Tenor, mg/L		
	Li	Al	Fe
Combined PLS + Wash 1	24500	2270	583

The combined solution was then subjected to the series of steps leading up to Li_2CO_3 precipitation. These steps are Primary Impurity Removal (PIR), Secondary Impurity Removal (SIR) and Ion-exchange (IX).

The PIR process involved modifying the pH of the solution to ~5.8 using lime while air sparging the solution to maintain the reduction potential of the solution above ~250 mV. The objective of the PIR step is to remove the main impurities in solution (Al and Fe). The select assay data of the product solutions of each of the three steps are provided in Table 6. The objective of the PIR step was met; both Al and Fe were removed from solution.

Table 6 Initial Product: PIR Product Solution - Select Assay Data

	Solution Tenor, mg/L				
	Li	Al	Fe	Ca	Mg
PIR Product Solution	25700	< 2	< 0.05	428	45.8
SIR Product Solution	20000	< 2	< 0.05	208	20.6
IX Product Solution	20800	n.a.	n.a.	117	4.28

n.a. - not assayed

[2] *All test details and data are provided in the Appendices.*

SIR focused on the reduction of the divalent metal cation impurities (Ca, Mg, Mn) in solution. The SIR step involved first modifying the pH of the Li process solution to 10 using sodium hydroxide (NaOH) and then adding sodium carbonate (Na_2CO_3) as 25% by weight aqueous solution. The Ca and Mg levels each decreased by ~50%.

The SIR product solution was then passed through an IX column packed with Purolite S950. The objective of this step is to reduce the tenors of Ca and Mg to below 10 mg/L. This was achieved for Mg, which had a post-IX tenor of 4.3 mg/L. The Ca tenor in the IX product solution was ~12 times above the required target at 117 mg/L. This will result in elevated Ca levels in the Li_2CO_3 that will be precipitated from the IX product solution.

Li_2CO_3 precipitation involved heating the IX product solution to ~95 °C and then adding a solution of Na_2CO_3. The Li_2CO_3 immediately precipitates from solution. The solids were recovered by vacuum filtration and washed with deionized water at ~95 °C to remove the entrained process solution. Based on the Ca tenor in the precipitation feed solution it was decided only to assay for Ca. The solids had a Ca content of 1480 g/t. This exceeded the specification for Ca of < 100 g/t for battery grade Li_2CO_3. The solids had to be polished in order to reduce the Ca content.

One standard method for polishing Li_2CO_3 is the Bicarbonate process. This involves dissolving the Li_2CO_3 in deionized water as lithium bicarbonate ($LiHCO_3$ (aq)) by sparging the slurry of Li_2CO_3 with CO_2(g). Once the majority of the solids have dissolved the solution is filtered. The filtrate is then heated to ~95°C. The $LiHCO_3$ (aq) decomposes to Li_2CO_3, which precipitates from solution. The assay data of the polished product are summarized in Table 7. The assay data are compared to the specifications for battery grade Li_2CO_3. The product met the specifications for all of the analytes except Ca. The polishing step reduced the Ca level from 1480 g/t to 205 g/t. Due to Rock Tech's schedule no further polishing was conducted on these solids. This product was shipped to Rock Tech.

Table 7 Initial Product: Bicarbonate Polishing – Initial Product Assays

Analyte	Spec	Rock Tech Lithium - First Product		
		Sample 1	Sample 2	**Average**
Na	< 400 ppm	<20	<20	**<20**
Sulphur (S)	< 200 ppm	100	200	**150**
Chlorides (Cl)	< 100 ppm	<10	<10	**<10**
Ca	< 100 ppm	210	200	**205**
Mg	< 100 ppm	15.2	17	**16**
K	< 50 ppm	<20	<20	**<20**
B	< 10 ppm	< 4	< 4	**< 4**
Fe	< 5 ppm	<2	<2	**<2**
Cr	< 5 ppm	< 1	< 1	**< 1**
Ni	< 5 ppm	< 5	< 5	**< 5**
Cu	< 5 ppm	1	1	**1**
Pb	< 5 ppm	< 1	< 1	**< 1**
Al	< 5 ppm	<8	<8	**<8**
Zn	< 5 ppm	< 5	< 5	**< 5**
Mn	< 5 ppm	0.4	0.4	**0.4**
Li_2CO_3 Grade, %	> 99.5%	99.9673	99.9582	**99.9628**

Note: Li_2CO_3 grade determined by difference

	Assay value not on spec

The success of this process demonstrated that a high grade Li_2CO_3 product could be produced from the Georgia Lake ore. The elevated Ca level in the initial Li_2CO_3 product can be addressed by examining specific process units (SIR, IX) within the flow sheet.

5. Main Market Sample of Li_2CO_3

Once the initial Li_2CO_3 product was produced the program refocused on processing the three concentrates through the standard Li_2CO_3 flow sheet.

6. Concentrate Leach

The objective of the Concentrate Leach was to extract the lithium in the Acid Roasted (AR) β-spodumene as lithium sulphate. The general procedure involved mixing AR β-spodumene with an equal mass of water. The resulting slurry was mixed for approximately 60 minutes and then vacuum filtered to recover the Pregnant Leach Solution (PLS) and the residue.

The solution tenors of each Concentrate Leach PLS are summarized in Table 8. The Li tenors ranged from 22 to 24 g/L. The first wash fraction contained a significant amount of Li (~7 – 8 g/L). In this program the PLS and Wash 1 fraction were combined and used as the Li process solution for the subsequent process (PIR).

The main differences in the compositions of the three PLS samples were as follows:

- The concentrate leach of the High Fe Con produced a PLS having the highest Fe tenor (1280 mg/L);

- The Al tenor of the concentrate leach PLS from the Heavy Media concentrate was the highest at 3910 mg/L;

These differences should not cause any process issues going forward. The elevated levels in Fe and Al may lead to higher reagent consumption during the PIR stage.

Table 8 Concentrate Leach – PLS Solution Tenors for Each Concentrate Sample

Concentrate	Concentrate Leach PLS - Solution Tenors, mg/L													
	Li	Si	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn
Low Fe Con	24500	84.5	2420	731	557	6.3	341	18.6	164	1790	352	1.01	0.2	6.2
High Fe Con	22400	63.1	2630	1280	548	74.5	267	21.2	206	2660	268	0.82	0.4	6.9
Heavy Media Con	22900	107	3910	994	320	1.6	316	15.9	137	1970	99	0.57	< 0.2	12.9

Select assay data of the Concentrate Leach residues are provided in Table 9. The %Li levels in the residues are fall at the low end of the typical range observed for similar systems.

Table 9 Concentrate Leach – Select Assays of the Residue

Concentrate	Residue Elemental Levels, %				
	Li	Si	Al	Fe	Ca
Low Fe Con	0.08	30.3	14.1	0.6	0.2
High Fe Con	0.09	30.6	13.5	0.8	0.2
Heavy Media Con	0.11	31.4	13.8	0.4	0.1

The extraction data of the Concentrate Leach are summarized in Table 10. The Li extraction values are typical and indicate no significant difference between the samples.

Table 10 Concentrate Leach – Select Elemental Extraction Data

Concentrate	Elemental Extraction, %				
	Li	Si	Al	Fe	Ca
Low Fe Con	97.1	0.0	1.1	7.9	17.1
High Fe Con	97.1	0.0	1.5	11.3	15.6
Heavy Media Con	96.3	0.0	2.5	19.2	31.0

7. Primary Impurity Removal

The PIR step was designed to remove the major impurities from the feed solution (Al and Fe). The procedure involved an air oxidation and pH modification using lime slurry (20 wt %) to precipitate the impurities as a mixed metal hydroxide.

Each PIR test was conducted in a suitably sized Pyrex™ glass reactor equipped with pH and ORP probes, a fritted glass gas sparging tube and an overhead mixer with a Teflon pitched blade impeller. The reaction was conducted at ambient temperature (average test temperature 23 °C). The test involved sparging the feed solution for a set period of time (5 minutes) and then pH modification of the feed solution using lime slurry to a target pH endpoint of 5.8.

The assay data for the Feed, Product Solution and Precipitate for each of the PIR tests are summarized in Table 11. The low Li tenor of the Heavy Media Con Feed Solution (13.4 g/L) was due to the dilution that occurred when the concentrate leach PLS was combined with the first wash fraction. In this case the volume of the wash fraction was slightly greater than that of the PLS.

Table 11 PIR Feed, Product Solution and Precipitate Assay Data

Concentrate	PIR Feed Solutions - Assay Data, mg/L											
	Li	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	Zn
Low Fe Con	18500	2390	541	480	6	297	28	143	1760	325	0.9	5.7
High Fe Con	19300	2114	637	484	5.3	265	18.1	140	1473	281	0.86	5.4
Heavy Media Con	13400	2560	660	222	1	214	11	91	1280	64	0	8
Concentrate	PIR Product Solutions - Assay Data, mg/L											
	Li	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	Zn
Low Fe Con	17500	< 0.8	< 0.2	461	< 0.1	233	48.6	97	1220	< 5	< 0.02	< 0.7
High Fe Con	18400	< 0.8	0.30	496	< 0.2	250	51.7	106	1421	< 5	< 0.02	< 0.7
Heavy Media Con	12500	< 0.8	0	465	< 0.2	199	52	53	1240	< 5	< 0.02	< 0.7
Concentrate	PIR Precipitate - Assay Data, %											
	Li	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	Zn
Low Fe Con	0.077	5.4	1.7	20.1	0.016	0.011	0.026	0.07	0.07	0.8	0.004	0.015
High Fe Con	0.32	5.3	2.5	16.6	0.091	0.013	0.037	0.07	0.08	0.5	0.003	0.014
Heavy Media Con	0.70	5.4	1.3	18.7	0.002	<.02	0.024	0.08	0.10	0.1	0.002	0.016

The % precipitation data are summarized in Table 12. The main impurities (Al, Fe, Cr, P, Ti and Zn) were removed completely by the PIR step. The Li % precipitation values ranged from 0.15 to 2.4%. The precipitated Li units can not be readily recovered and would represent a loss in a commercial operation. The PIR of the Low Fe Con had the lowest Li %precipitation.

Table 12 PIR – Percent Precipitation

Concentrate	PIR - Elemental % Precipitation, %											
	Li	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	Zn
Low Fe Con	0.15	100	100	94.2	100	1.8	16.6	21.8	2.1	100	100	100
High Fe Con	0.86	100	100	95.1	100	2.9	29.4	27.6	3.0	100	100	100
Heavy Media Con	2.4	100	100	95.1	100	0.0	18.5	40.8	3.9	100	100	100

The final slurry pH and lime consumption data are provided in Table 13 for the PIR tests. The pH endpoint of 5.8 was demonstrated to be a suitable value to achieve the objectives of the PIR process. The lime consumption is presented as kg of dry lime consumed per kg of lithium carbonate equivalent (LCE). Lime consumption is typically in the range 0.20 – 0.25 kg lime per kg LCE. The low consumption values observed for the Low and High Fe Con PIR tests are encouraging and if proven reproducible will represent some reagent cost savings in the commercial operation.

Table 13 PIR – Final Slurry pH and Lime Reagent Consumption

Concentrate	Final pH	kg Lime per kg LCE
Low Fe Con	5.8	0.14
High Fe Con	5.8	0.16
Heavy Media Con	5.9	0.25

8. Secondary Impurity Removal

The purpose of the Secondary Impurity Removal (SIR) process is to reduce the solution tenors of the remaining divalent metal ions in the Li process solution (*e.g.* Ca, Mg, Mn, *etc*.). It is especially important to reduce the Ca tenor as much as possible prior to the ion-exchange step.

The Li process solutions going forward from the PIR step have very similar compositions. The initial chemical distinction between the three concentrate samples is no longer apparent. As a result the three PIR product solutions were combined to produce the Li process solution that was used a feed for the SIR test.

The SIR process involved modifying the pH of the Li process solution to 10 with 2 M NaOH. This was followed by the addition of Na_2CO_3 as a 25 wt% solution. The addition of Na_2CO_3 is based on the solution concentrations of the targeted metal ions. The reaction is conducted at ambient temperature.

The assay data of the feed and product solutions and the resulting precipitate are provided in Table 14. The % precipitation values are also provided. A minimum amount of Li was lost in this process (Li % precipitation of 0.01%). Poor elemental accountability was achieved for Ca (68.8%). The low % precipitation value for Ca of 11.9% reflects the poor accountability. If the Ca %precipitation is calculated based on solution assay data then the value increases to 39%. This is still rather low for this process and therefore the SIR process should be examined in more detail.

Table 14 SIR Assay and Precipitation Data

SIR Process	Li	Ca	Fe	K	Mg	Mn	Na
SIR Feed Solution, mg/L	16200	461	4.5	220	50	84.7	1620
SIR Product Solution, mg/L	16500	280	0.2	224	42.4	<0.04	2300
SIR Precipitate, %	0.41	14.6	1.38	0.0478	1.69	22.4	0.189
% Precipitation, %	0.01	11.9	94.7	0.0	9.4	100.0	0.0

The main impurities that remained in the SIR product solution were Ca and Mg. The solution tenors of these Group 2 metal ions were too high to proceed directly to Li_2CO_3 precipitation. The use of this solution to produce Li_2CO_3 would produce a product that contained levels of Ca and Mg above the maximum allowable limits for a high grade Li product. The application of IX methodologies to the SIR product solution was required to reduce the Ca and Mg tenors in the Li process solution.

The reagent consumptions were 0.0011 kg NaOH/ kg LCE for NaOH and 0.016 kg Na_2CO_3/ kg LCE for Na_2CO_3 for the SIR test.

9. Ion-Exchange

Ion-exchange (IX) methods were applied to the SIR product solution with the objective of reducing the Ca and Mg tenors to < 10 mg/L. Two IX columns, packed with Purolite S950, were used in a lead-lag configuration. The IX parameters used in this test are summarized in Table 15.

Table 15 IX Operational Parameters

Column Bed Volume, mL:	250			
Feed:	SIR Product Soluiton	Loading Flowrate:	10	BV/h
Wash-1:	Deionized Water	Wash Flowrate:	10	BV/h
Strip Sol'n:	2 M HCl	Stripping Flowrate:	5	BV/h
Wash-2:	Deionized Water	Wash Flowrate:	5	BV/h
Regeneration Sol'n:	2 M NaOH	Regen Flowrate:	5	BV/h
Final Wash:	Deionized Water	Wash Flowrate:	10	BV/h

The loading profile for Li is provided in Figure 2. The profile indicates that some Li did load on to the resin.

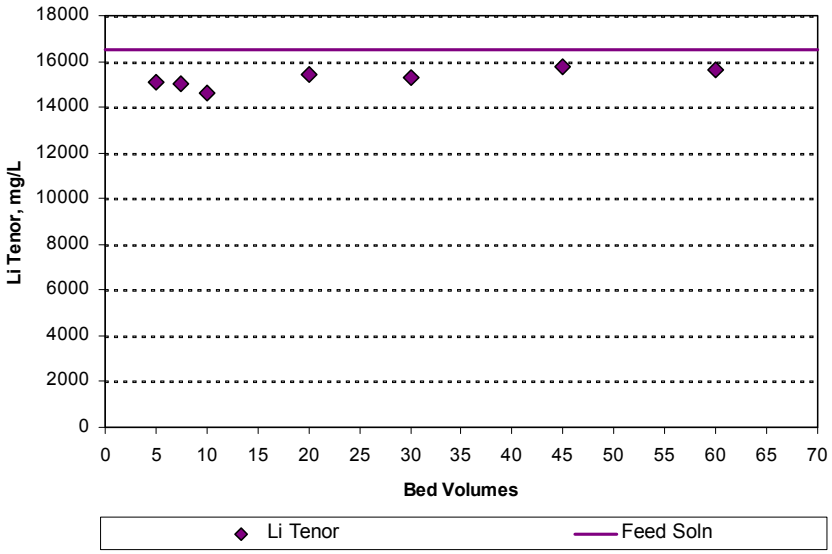


Figure 2 IX Loading Profile for Li

The loading profiles for Ca and Mg are provided in Figure 3. The breakthrough points for both Ca and Mg occurred below 10 bed volumes (BV). The compositions of the IX feed and product solutions are summarized in Table 16. The Mg tenor in the product was on target at 4.2 mg/L. The Ca tenor was above the 10 mg/L maximum for this impurity in solution. Levels of Ca > 10 mg/L generally produce a Li_2CO_3 product with a Ca content > 100 g/t, which is the maximum allowable level for a high grade product.

Table 16 IX – Composition of the IX Feed and Product Solutions

Process Solution	Solution Tenor, mg/L								
	Li	Ba	Ca	Cu	Fe	K	Mg	Mo	Na
IX Feed	16500	0.091	280	0.2	< 0.2	224	42.4	1.4	2300
IX Product	14400	0.018	38.6	< 0.1	< 0.2	210	4.19	1.5	3210

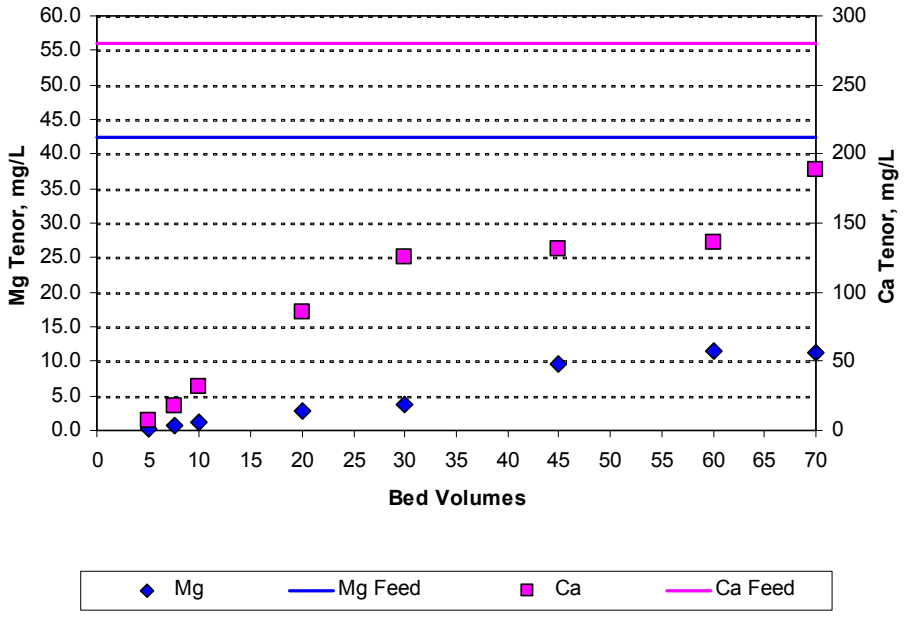


Figure 3 IX Loading Profile for Ca and Mg

The Acid Strip profiles for Li and Ca (Figure 4) indicated that Li eluted from the column before Ca. The Li profile reached a maximum at ~1.25 BV's. The maximum of the Ca profile was ~1.7 BV's. The Acid Strip profile clearly indicates that Li does load on to the resin. The mass balance calculation for the IX process indicated that only 0.8% of the contained Li in the feed solution loads on to the resin. This represent a potential lose if the Li can not be returned to the system.

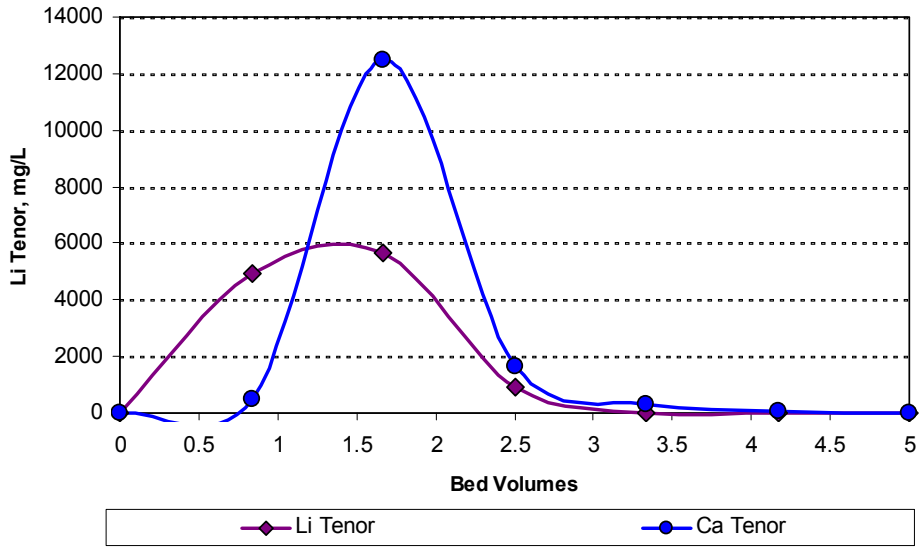


Figure 4 IX Stripping Profile for Li and Ca

10. Lithium Carbonate Precipitation

The IX product solution was the feed for the Lithium Carbonate precipitation step. The precipitation reaction involved heating the feed solution to T ~ 95 °C. Lithium carbonate was precipitated from solution by adding an aqueous solution of sodium carbonate (25 wt% Na_2CO_3, 4% stoichiometric excess). No seed Li_2CO_3 was used to promote crystallization. The resulting solution was stirred at temperature for 40 minutes. After 40 minutes the reaction slurry was vacuum filtered. Three displacement washes using distilled water at T ≥ 95 °C were conducted on the filter cake. The washed filter cake was dried in an oven (T ~ 105 - 110 °C) for ~16 – 20 h.

Two Li_2CO_3 precipitation tests were conducted using the IX product solution. The average Li precipitation efficiency that was achieved was 76%. The assay data of the Li_2CO_3 product are summarized in Table 17. Included in this table is a list of the elements that assayed below the respective detection limits on the spectroscopic technique. The chemical composition of the Li_2CO_3 product is compared to the specifications for battery grade Li_2CO_3 in Table 18. The product met the specifications for all analytes except for Na and Ca. The solids were not assayed for chlorine (Cl), boron (B) or sulphur (S).

Table 17 Assay Data for the Li_2CO_3 Product

Li %	Na %	K %	Mg g/t	Ca g/t	Sr g/t	Ba g/t	Cu g/t	Ti g/t
19.1	0.15	<0.002	35	260	6.02	1.4	0.7	0.9
Elements that assayed below the detection limits								
Ag	Al	As	Be	Bi	Cd	Co	Cr	Fe
Mn	Mo	Ni	P	Pb	Sb	Se	Sn	Tl
U	V	Y	Zn					

In order to improve the grade of the Li_2CO_3 the solids required further processing through the bicarbonate polishing step.

11. Bicarbonate Polishing

The Bicarbonate Polishing step is a method that is used to upgrade lithium carbonate. The bicarbonate polishing procedure involves dissolving the lithium carbonate as lithium bicarbonate ($LiHCO_3$) by sparging the aqueous slurry of the precipitate with CO_2 (g) (**Equation 1**). The resulting solution is then filtered to remove any solids from solution. The filtrate is then heated to T ≥ 95 °C to decompose the dissolved bicarbonate (**Equation 2**). Lithium carbonate precipitates from solution as the decomposition product.

$$\text{Equation 1} \quad CO_2(g) + H_2O \rightarrow H_2CO_3(aq)$$

$$\text{Equation 2} \quad H_2CO_3(aq) + Li_2CO_3(s) \rightarrow 2\ LiHCO_3(aq)$$

The assay data of the Bicarbonate Polished solids are summarized in Table 18 . Included in this table are the assay data of the original solids and of the solids after the first bicarbonate polishing step. The product produced from the first bicarbonate polishing step met the product specifications for high grade Li_2CO_3 for all analytes except Fe. During the first bicarbonate polishing step the process solution was contaminated with a trace amount of Fe. This Fe reported to the resulting solids. Duplicate samples of the polished Li_2CO_3 were submitted for analysis. The Fe level of these solids was reported as 11 g/t and 3 g/t (average 7 g/t). This is above the maximum specified level for Fe. The solids were then subjected to a second bicarbonate polishing step in order to remove the Fe.

The second bicarbonate product now met the product specifications for all of the analytes.

The grade of the Li_2CO_3 increased from 99.8204% in the precipitated product to 99.9762% in the first bicarbonate product and finally to 99.9878% in the second bicarbonate product.

Table 18 Summary of the Li_2CO_3 Assay Data

Analyte	Spec	First Li_2CO_3 Precipitation Product	First Bicarbonate Polish		Second Bicarbonate Polish		
			RT Bulk BCB DC T1 Prod	RT BCB DC T2 Prod	Final Sample - Assay Cut 1	Final Sample - Assay Cut 2	Second Bicarbonate Polish Average Assay Values
Na	< 400 ppm	1500	<20	<20	40	20	**30**
Sulphur (S)	< 200 ppm	n.a.	190	<100	<100	<100	**<100**
Chlorides (Cl)	< 100 ppm	n.a.	73	48	26	22	**24**
Ca	< 100 ppm	260	58	75	62	60	**61**
Mg	< 100 ppm	35	11.3	4.6	3.2	3.1	**3**
K	< 50 ppm	<20	<20	<20	<20	<20	**20**
B	< 10 ppm	n.a.	<4	<4	6	< 4	**6**
Fe	< 5 ppm	<4	11	3	< 2	< 2	**< 2**
Cr	< 5 ppm	<4	<1	<1	< 1	< 1	**< 1**
Ni	< 5 ppm	<20	<5	<5	< 5	< 5	**< 5**
Cu	< 5 ppm	0.7	1	<1	< 1	< 1	**< 1**
Pb	< 5 ppm	< 20	<1	<1	< 1	< 1	**< 1**
Al	< 5 ppm	< 20	<8	<8	< 8	< 8	**< 8**
Zn	< 5 ppm	<2	<5	<5	< 5	< 5	**< 5**
Mn	< 5 ppm	<0.3	0.7	0.4	0.4	0.4	**0.4**
Li_2CO_3 Grade, %	> 99.5%	*99.8204*	99.9655	99.9869	99.9862	99.9895	**99.9878**

Note: Li_2CO_3 grade determined by difference

Conclusions and Recommendations

Conclusions

The main objective of the test program was to determine if a high grade lithium carbonate sample could be produced for the concentrate generated from the ore obtained from Rock Tech Lithium's Georgia Lake property. The first attempt produced a Li_2CO_3 product with an average grade of 99.963%. After processing the sample through the bicarbonate polishing step a Li_2CO_3 product having the average grade of 99.988% was produced. This clearly demonstrated that a high grade Li_2CO_3 product could be produced from the Rock Tech ore.

Concentrate Leach

- The application of the standard concentrate leach to the three concentrate samples produced good Li extractions in the range 96 to 97%;
- The Li loss for the concentrate leaches of the Low and High Fe Concentrate samples was 3.0%. The Li loss was higher for the concentrate leach of the Heavy Media Con sample at 3.7%.

Primary Impurity Removal

- The process objectives of the PIR stage were achieved for each of the concentrate samples that were tested. This demonstrated the robust nature of the PIR process;
- The PIR tests determined that a pH target of 5.8 is a suitable endpoint for the precipitation process;
- The Li % precipitation was lowest for the Low Fe Concentrate at 0.15%. It was observed to be the highest for the Heavy Media Concentrate.
- The lime consumptions during the PIR of the Low and High Fe Cons were similar at ~0.15 kg dry lime per kg lithium carbonate equivalent (LCE). This is lower than the typical consumption value for this process;
- The lime consumption for the Heavy Media Con sample was 0.25 kg dry lime per kg LCE. This is the typical consumption rate for similar systems.

Secondary Impurity Removal

- The process objectives of the SIR were not completely realized. The SIR step did not effectively remove Ca from the Li process solution;
- The reagent consumptions of the SIR process were 0.0011 kg NaOH/ kg LCE and 0.016 kg Na_2CO_3/ kg LCE.

Ion-Exchange

- The ion-exchange process effectively reduced the Mg tenor in the Li process solution to below 10 mg/L;

- The ion-exchange process did not reduce the Ca tenor in the Li process solution to below 10 mg/L. This is attributed to the fact that the IX feed solution (SIR product solution) had a greater than expected Ca tenor. The standard IX parameters that were used could not accommodate the higher Ca level;

- The Li loss during the IX step was ~0.8%. The Li units that were absorbed by the media can be recovered as a LiCl solution that is generated during the Acid Strip IX cycle.

Lithium Carbonate Precipitation

- The lithium carbonate that was produced from the precipitation reaction had an approximate grade of 99.8% based on the contained impurities;

- The lithium carbonate contained Na and Ca at levels above the maximum allowable limits for battery grade Li_2CO_3. As a result the product had to polished in order to remove the remaining impurities;

- The average precipitation efficiency for Li was observed to be 76%.

Bicarbonate Polishing

- The Bicarbonate Polishing procedure was successfully used to upgrade the initial Li_2CO_3 product. The first bicarbonate polishing test produced Li_2CO_3 having a grade of 99.976%. The solids met all of the product specification for battery grade Li_2CO_3 except for Fe (7 g/t);

- The solids were subjected to a second bicarbonate polishing step. The resulting solids met all of the product specifications and had a Li_2CO_3 grade of 99.988%.

Recommendations

The optimization of the process parameters was not an objective of the current program. It is recommended that each of the hydrometallurgical steps be examined with the objective to determine the optimum operational conditions. If possible the recommended laboratory program should include a preliminary Liquid-Solid Separation test series to examine the settling and thickening characteristics of all of the process slurries generated by the hydrometallurgical flow sheet. More specific recommendations related to further lab-scale test work are provided below.

Concentrate Leach

- An economic cost benefit analysis should be conducted to determine if the mass reduction of the concentrate due to the magnetic separation step significantly decreases the acid requirement of the concentrate leach. Does the savings in reagent cost offset the capital and operation costs of the magnetic separation circuit?

- Combining the Concentrate Leach and PIR processes should be considered and tested.

Primary Impurity Removal

- A pH isotherm should be completed in order to determine the optimum pH target for the PIR process;

- Once the PIR process parameters have been optimized the lime consumption should be examined to determine if the values achieved in this program are reproducible.

- Recycling a portion of the PIR solids back to the PIR process could be considered and tested. This approach may or may not recover a portion of the lithium lost to PIR solids and could reduce overall lime consumption.

Secondary Impurity Removal

- The low %precipitation achieved for Ca during the SIR step is of concern. The main objective of this step is to precipitate at least 70% of the contained Ca in solution. The resulting solution would have a Ca tenor that can be easily treated by IX to reduce it to below 10 mg/L. It is recommended that the SIR process be examined in detail to determine the parameters required to maximize the removal of Ca.

Ion-Exchange

- The IX operational parameters must be optimized in order to improve the removal of Ca from the Li process solution. It is recommended that a series of commercial resins be examined in order to determine the optimum one for the Rock Tech solution;

- The Stripping stage should be examined to determine if the difference in elution times between Li and Ca can be exploited to produce a LiCl(aq) process stream can be separated from the Ca (and Mg) chloride rich strip solution.

Lithium Carbonate Precipitation

- The precipitation step should be optimized to improve the efficiency of this step;

- Other routes to produce Li_2CO_3 should be examined.

Bicarbonate Polishing Step

- A corrosion coupon test should be conducted to determine the compatibility of the bicarbonate process solution with various materials;

- The CO_2 consumption, optimum process conditions and reaction kinetics should be determined;

The following assays are recommended to be included when examining future Li_2CO_3 products – loss of drying (LOD), loss on ignition (LOI) and percentage acid insolubles (HCl). These analyses can help confirm assumptions about the purity of the product and are commonly found in the technical specifications of battery-grade Li_2CO_3.

Appendix A – Concentrate Samples

Assay Data

CA02346-MAY11

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %
RT Con. L Fe	61.6	26.3	0.91	0.1	0.54	0.54	0.76	< 0.01	0.36	0.21	0.01	< 0.01	0.96
	Sum %												
	92.3												

CA02347-MAY11

Sample ID	Li %	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t
RT Con. L Fe	3.06	< 2	136000	< 30	47	272	< 20	4260	< 2	< 4	49	13.1	6410
	K g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t
	6170	382	1660	< 10	3330	< 20	1120	< 20	< 10	< 30	142	49.2	48
	Tl g/t	U g/t	V g/t	Y g/t	Zn g/t	S %							
	< 30	< 20	< 2	1.1	81	< 0.01							

CA02669-MAY11

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %
RT PT Con L Fe	62.3	26.4	0.91	0.09	0.55	0.57	0.79	< 0.01	0.37	0.21	< 0.01	< 0.01	0.33
	Sum %												
	92.6												

CA02668-MAY11

Sample ID	Li %	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t
RT PT Con L Fe	3.04	< 2	117000	< 30	30.7	233	< 20	3240	< 2	< 4	7	15.7	5950
	K g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t
	6070	333	1520	< 10	3570	< 20	1050	< 20	< 10	< 30	119	40.6	38.7
	Tl g/t	U g/t	V g/t	Y g/t	Zn g/t	S %							
	< 30	< 20	< 2	< 0.6	57	0.01							

CA02299-JUN11

Sample ID	Li %	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %
RT AR L Fe	2.35	48.7	20.5	0.76	0.08	0.42	0.46	0.64	< 0.01	0.29	0.17	< 0.01	< 0.01
	LOI %	Sum %	Buck wt. g										
	21.7	93.7	15										

CA02300-JUN11

Sample ID	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t	K g/t
RT AR L Fe	< 2	114000	< 30	39.9	198	< 20	3020	< 2	< 4	110	16.9	5670	4640
	Li g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Tl g/t
	25600	323	1310	< 10	3080	20	1220	< 20	< 10	< 30	< 200	38.3	< 30
	U g/t	V g/t	Y g/t	Zn g/t	S %	Buck wt. g							
	< 20	< 2	0.7	60	7.1	15							

CA02077-MAY11

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %
RT Con. H Fe	63.1	25.4	1.31	0.07	0.52	0.9	1.02	< 0.01	0.36	0.24	0.1	< 0.01	1.16
	Sum %												
	94.1												

CA02078-MAY11

Sample ID	Li %	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t
RT Con. H Fe	2.79	< 2	132000	< 30	63	409	< 20	3450	< 2	7	754	32.7	9570
	K g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t
	8400	387	1770	79	6480	389	1340	< 20	< 20	< 30	157	48.6	45.2
	Tl g/t	U g/t	V g/t	Y g/t	Zn g/t	S %							
	< 30	< 20	< 4	1.3	58	0.04							

CA02641-MAY11

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %
RT PT Con H Fe	64	26	1.39	0.05	0.52	0.9	1.0	< 0.01	0.32	0.24	0.1	< 0.01	0.12
	Sum %												
	94.7												

CA02640-MAY11

Sample ID	Li %	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t
RT PT Con H Fe	2.74	< 2	130000	< 30	63	409	< 20	4930	< 10	< 10	806	28.8	10600
	K g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t	Tl g/t
	9130	1830	78	7200	329	1170	< 20	< 20	< 30	116	60.1	40.8	< 30
	U g/t	V g/t	Y g/t	Zn g/t	S %								
	< 20	< 4	0.8	75	0.01								

CA02344-JUN11

Sample ID	Li %	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %
RT AR H Fe	2.14	50.4	20.4	1.12	0.06	0.4	0.75	0.84	< 0.01	0.28	0.2	0.09	< 0.01
	LOI %	Sum %	Buck wt. g										
	19.9	94.4	11										

CA02349-JUN11

Sample ID	Li %	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t
RT AR H Fe	2.26	< 2	117000	< 30	49.4	315	< 20	3820	< 2	< 10	367	33	8070
	K g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t
	7360	335	1450	78	5410	309	1130	< 20	< 10	< 30	93	46.2	83.4
	Tl g/t	U g/t	V g/t	Y g/t	Zn g/t	S %	Buck wt. g						
	< 30	< 20	< 2	0.6	67	6.78	15.4						

CA02145-MAY11

Sample ID	Li %	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %
RT HM Con	2.92	64.7	24.7	0.97	0.09	0.19	0.73	0.57	< 0.01	0.11	0.17	0.03	< 0.01
	LOI %	Sum %	Buck wt. g										
	0.66	93	31.6										

CA02144-MAY11

Sample ID	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t	K g/t
RT HM Con	< 2	130000	< 30	44.5	65.2	< 20	1920	< 2	< 4	213	6.7	7680	5300
	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t	Tl g/t
	296	1560	< 5	4720	< 20	474	< 20	< 10	< 30	145	36	30	< 30
	U g/t	V g/t	Y g/t	Zn g/t	S %	Buck wt. g	Li %						
	< 20	< 2	< 0.4	72	< 0.01	26.7	3.08						

CA02371-JUN11

Sample ID	Li %	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %
RT PT HL Con	2.83	65.9	25.2	0.94	0.07	0.2	0.71	0.51	< 0.01	0.11	0.19	< 0.01	< 0.01
	LOI %	Sum %	Buck wt. g										
	0.24	94	16										

CA02372-JUN11

Sample ID	Li %	Ag g/t	Al g/t	As g/t	Ba g/t	Be g/t	Bi g/t	Ca g/t	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe g/t
RT PT HL Con	3.04	< 2	131000	< 30	38.9	49.2	< 20	2470	< 2	< 4	34	8	6620
	K g/t	Mg g/t	Mn g/t	Mo g/t	Na g/t	Ni g/t	P g/t	Pb g/t	Sb g/t	Se g/t	Sn g/t	Sr g/t	Ti g/t
	5210	296	1150	< 10	5320	< 20	331	< 20	< 10	< 30	110	40.8	13.8
	Tl g/t	U g/t	V g/t	Y g/t	Zn g/t	S %	Buck wt. g						
	< 30	< 20	< 2	< 0.5	61	0.01	14.6						

CA02297-JUN11

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %
RT WL HL T1 Re	65.3	24.8	0.8	0.09	0.13	0.39	0.55	< 0.01	0.07	0.14	< 0.01	< 0.01	7.16
	Sum %	Buck wt. g											
	99.4	81.1											

Appendix B – Mineralogical Data

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Rock Tech Lithium Inc.*

Project Number/ LIMS No. *12607-001/ MI4507-MAY11*

Reporting Date: *June 22, 2011*

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)

Jennifer LaBelle-Brown, A.Sc.T
Technologist, XRD

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT Con. L Fe	spodumene *(monoclinic)*	-	mica, quartz, plagioclase	*pyrrhotite, *potassium feldspar

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

Mineral	Composition
Mica	$K(Mg,Fe)Al_2Si_3AlO_{10}(OH)_2$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Pyrrhotite	$Fe_{(1-x)}S$
Quartz	SiO_2
Spodumene	$LiAlSi_2O_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT Con. L Fe (wt %)
Spodumene (Monoclinic)	85.6
Muscovite	4.7
Quartz	3.3
Albite	2.9
Biotite	1.8
Pyrrhotite	1.1
Orthoclase	0.6
TOTAL	100.0

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.

Chemical Balance

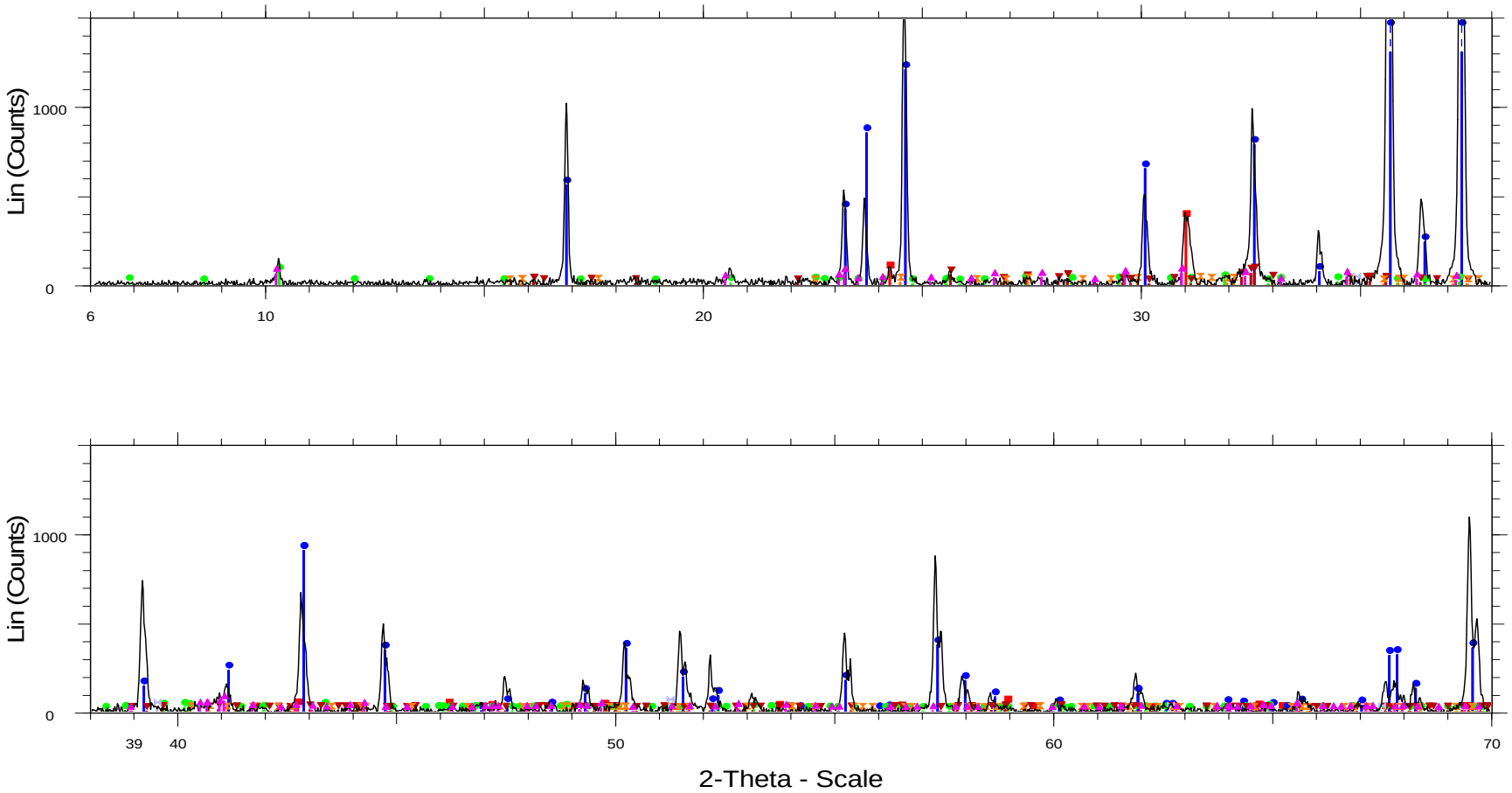
RT Con. L Fe

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	46.2	50.6	-4.47	Both
Silicon	28.8	29.9	-1.07	Both
Aluminum	13.9	13.8	0.08	Both
Lithium	3.06	3.19	-0.13	Both
Iron	0.64	0.66	-0.03	Both
Potassium	0.63	0.72	-0.09	Both
Sodium	0.40	0.25	0.15	Both
Calcium	0.39	-	0.39	XRF
Manganese	0.16	-	0.16	XRF
Phosphorus	0.16	-	0.16	XRF
Magnesium	0.06	0.31	0.25	Both
Hydrogen	-	0.03	0.03	SQD
Fluorine	-	0.08	0.08	SQD
Sulfur	-	0.40	0.40	SQD

1. Values measured by chemical assay.

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

RT Con. L Fe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2-Theta - Scale
RT Con. L Fe - File: May4507-1.raw
Operations: Background 1.000,1.000 | Import
01-079-1910 (C) - Quartz - SiO2
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01-084-1305 (C) - Muscovite - KAl3Si3O10(OH)2
01-073-1661 (C) - Biotite - KMg3AlSi3O10OHF
01-083-1466 (C) - Albite low - Na0.986(Al1.005Si2.995O8)
01-075-1190 (C) - Orthoclase - K(AlSi3)O8
00-020-0535 (D) - Pyrrhotite 1C, syn - Fe0.95S

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Met Ops*

Project Number/ LIMS No. *12607-001/MI4511-MAY11*

Reporting Date: *June 17, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown 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) database and released on software as 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.

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)
RT PT Con. L Fe	Spodumene beta (Tetragonal)	Lithium Aluminum Silicate (Hexagonal)	quartz, potassium-feldspar	*hematite

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

Mineral	Composition
Quartz	SiO_2
Lithium Aluminum Silicate	$LiAlSi_2O_6$ (Hexagonal)
Spodumene beta	$LiAlSi_2O_6$ (Tetragonal)
Orthoclase	$KAlSi_3O_8$
Hematite	Fe_2O_3

Semi-Quantitative X-ray Diffraction Results

Mineral	RT PT Con. L Fe (wt %)
Quartz	8.4
Lithium Aluminum Silicate (Hexagonal)	13.0
Spodumene beta (Tetragonal)	74.3
Orthoclase	3.7
Hematite	0.6
TOTAL	100.0

Chemical Balance

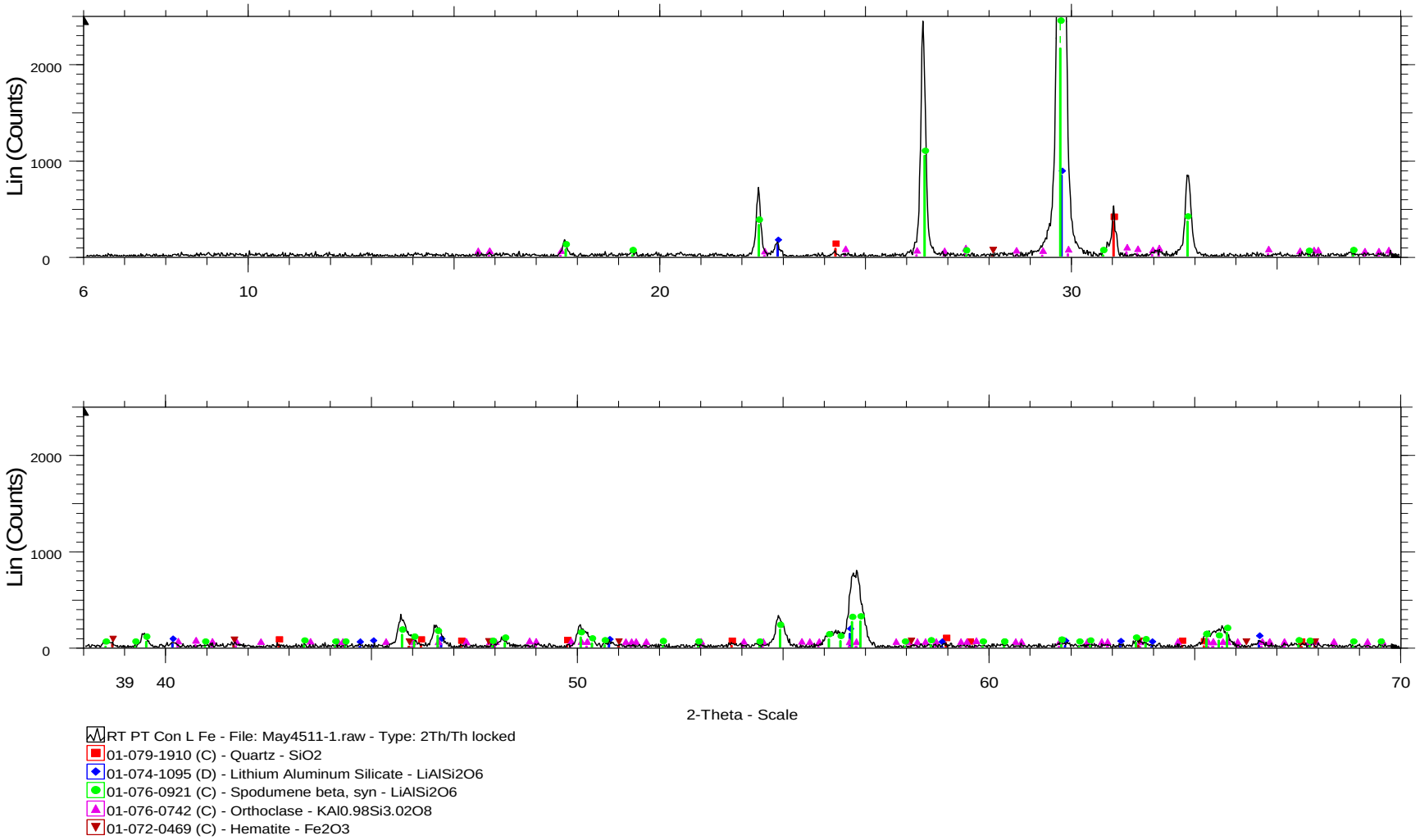
RT PT Con. L Fe

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	46.6	51.4	-4.78	Both
Silicon	29.1	31.4	-2.27	Both
Aluminum	14.0	13.0	0.96	Both
Lithium	3.04	3.26	-0.22	Both
Potassium	0.66	0.52	0.14	Both
Iron	0.64	0.43	0.20	Both
Sodium	0.42	-	0.42	XRF
Calcium	0.39	-	0.39	XRF
Manganese	0.16	-	0.16	XRF
Phosphorus	0.16	-	0.16	XRF
Magnesium	0.05	-	0.05	XRF

1. Values measured by chemical assay.

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

RT PT Con L Fe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2-Theta - Scale
RT PT Con L Fe - File: May4511-1.raw - Type: 2Th/Th locked
01-079-1910 (C) - Quartz - SiO2
01-074-1095 (D) - Lithium Aluminum Silicate - LiAlSi2O6
01-076-0921 (C) - Spodumene beta, syn - LiAlSi2O6
01-076-0742 (C) - Orthoclase - KAl0.98Si3.02O8
01-072-0469 (C) - Hematite - Fe2O3

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Met Ops*

Project Number/ LIMS No. *12607-001/MI4505-JUN11*

Reporting Date: *June 17, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown 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) database and released on software as 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.

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)
RT AR L Fe	Hydrogen Aluminum Silicate (Tetragonal)	Lithium Sulfate Hydrate (Monoclinic)	quartz, potassium-feldspar	*Hydrogen Aluminum Silicate (Hexagonal)

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

Mineral	Composition
Hydrogen Aluminum Silicate	$H(AlSi_2O_6)$ (Hexagonal)
Hydrogen Aluminum Silicate	$H(AlSi_2)O_6$ (Tetragonal)
Lithium Sulfate Hydrate	$Li_2(SO_4)(H_2O)$ (Monoclinic)
Orthoclase	$KAlSi_3O_8$
Quartz	SiO_2

Semi-Quantitative X-ray Diffraction Results

Mineral	RT AR L Fe (wt %)
Quartz	4.6
Lithium Sulfate Hydrate (Monoclinic)	25.9
Hydrogen Aluminum Silicate (Hexagonal)	0.8
Orthoclase	4.1
Hydrogen Aluminum Silicate (Tetragonal)	64.6
TOTAL	100.0

Chemical Balance

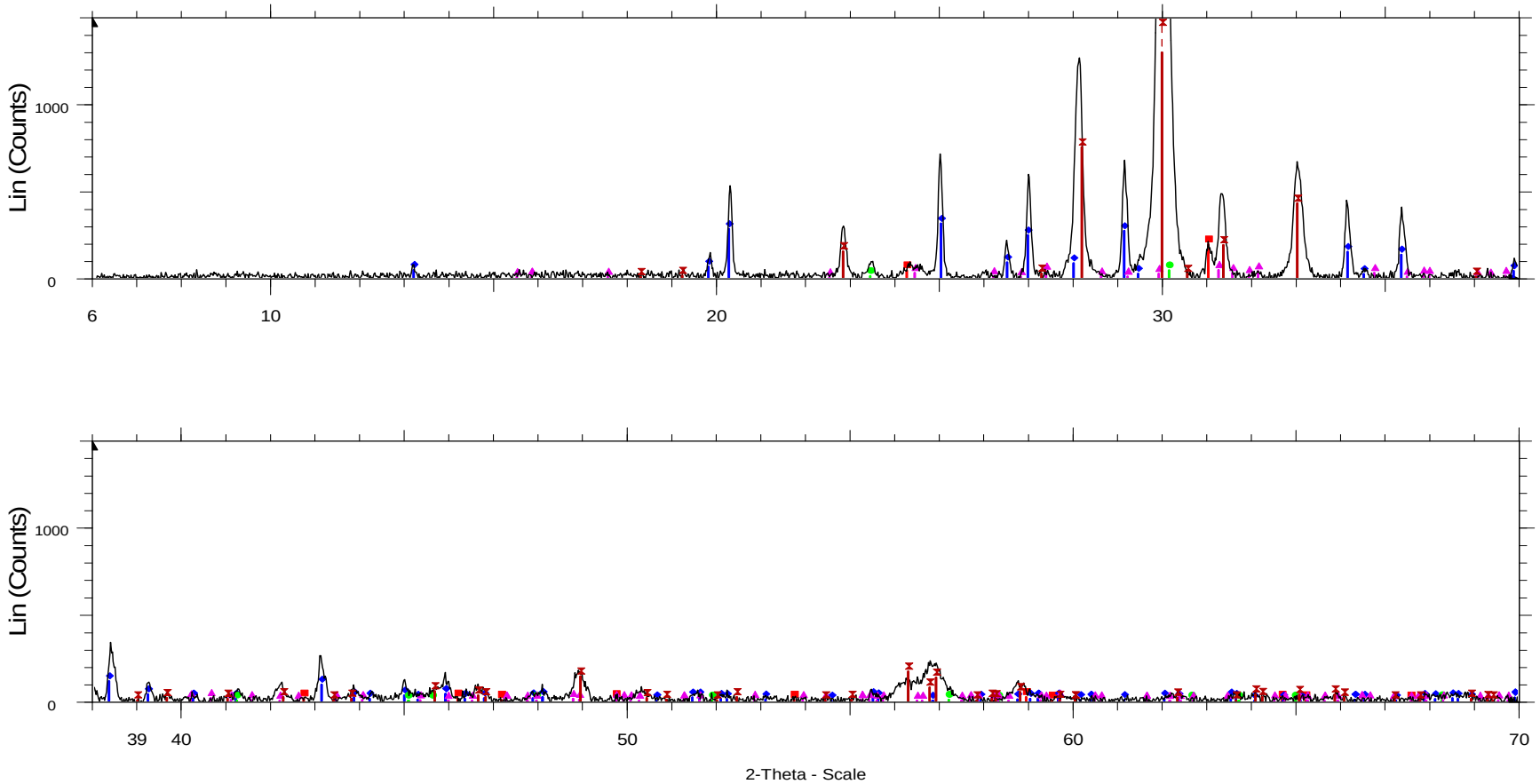
RT AR L Fe

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	36.4	55.4	-19.0	Both
Silicon	22.8	23.8	-1.01	Both
Aluminum	10.8	10.2	0.66	Both
Lithium	2.35	2.81	-0.46	Both
Iron	0.53	-	0.53	XRF
Potassium	0.53	0.57	-0.04	Both
Sodium	0.34	-	0.34	XRF
Calcium	0.30	-	0.30	XRF
Manganese	0.13	-	0.13	XRF
Phosphorus	0.13	-	0.13	XRF
Magnesium	0.05	-	0.05	XRF
Hydrogen	-	0.77	0.77	SQD
Sulfur	-	6.50	6.50	SQD

1. Values measured by chemical assay.

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

RT AR L Fe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Lin (Counts)
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2-Theta - Scale
RT AR L Fe - File: Jun4505-1.raw - Type: 2Th/Th locked
01-079-1910 (C) - Quartz - SiO2
01-078-0955 (C) - Lithium Sulfate Hydrate - Li2(SO4)(H2O)
01-080-1387 (C) - Hydrogen Aluminum Silicate - H(AlSi2O6)
01-086-0437 (C) - Orthoclase - K(AlSi3O8)
01-079-1982 (C) - Hydrogen Aluminum Silicate - H(AlSi2)O6

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Met Ops*

Project Number/ LIMS No. *12607-002/MI4518-JUN11*

Reporting Date: *July 28, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT L Fe CL COMP RES	Hydrogen Aluminum Silicate *(Tetragonal)*	-	potassium-feldspar, Hydrogen Aluminum Silicate *(Hexagonal)*, quartz, plagioclase	*goethite

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

Mineral	Composition
Quartz	SiO_2
Hydrogen Aluminum Silicate (Tetragonal)	$H(AlSi_2)O_6$
Hydrogen Aluminum Silicate (Hexagonal)	$H(AlSi_2O_6)$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Goethite	$\alpha FeO{\cdot}OH$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT L Fe CL COMP RES (wt %)
Quartz	2.7
Hydrogen Aluminum Silicate (Tetragonal)	84.1
Hydrogen Aluminum Silicate (Hexagonal)	4.8
Albite low	2.3
Microcline	5.1
Goethite	1.1
TOTAL	100.1

Note: Albite is a plagioclase feldspar

Chemical Balance

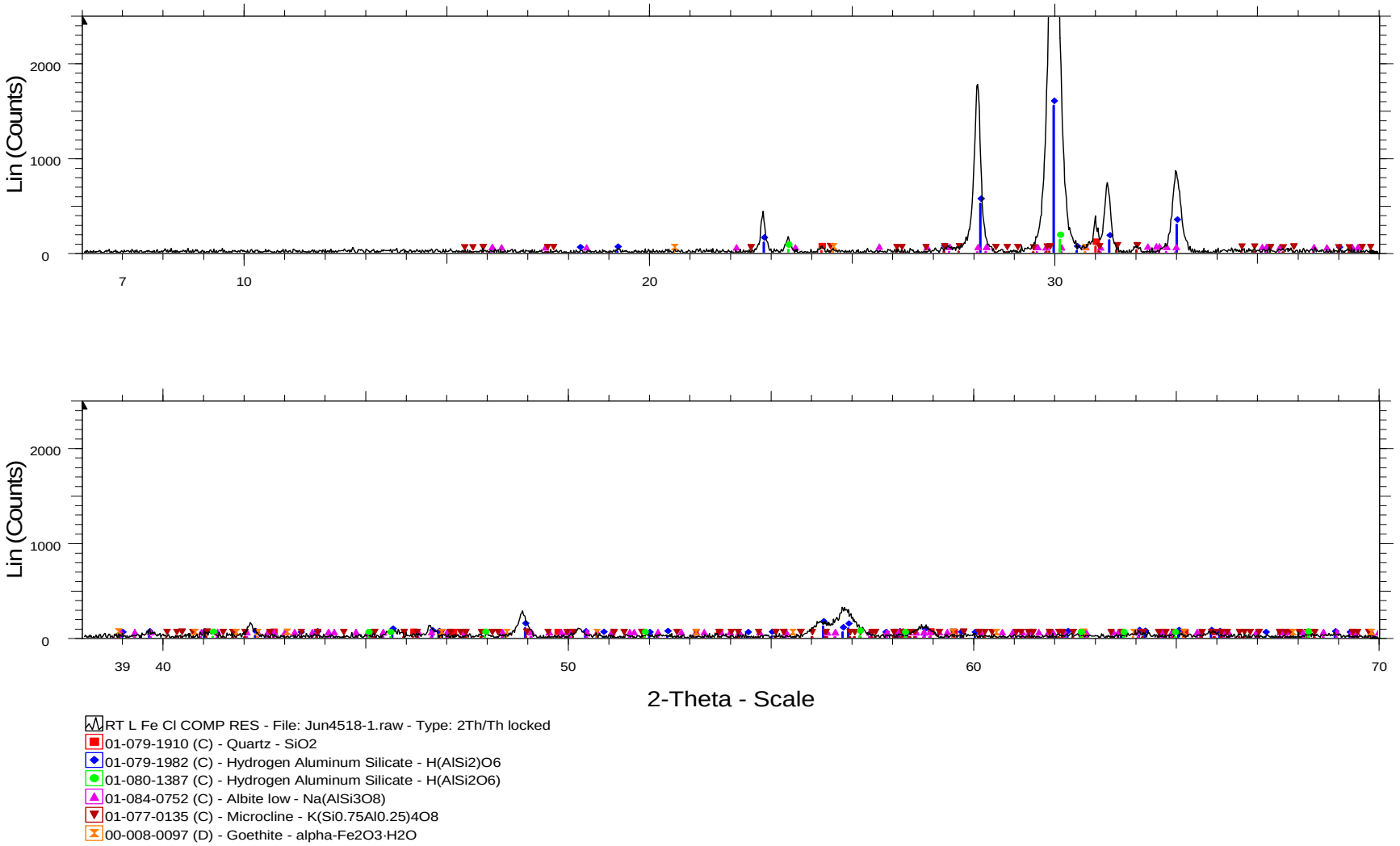
RT L Fe CL COMP RES

Name	Assay[1]	SQD[2]	Delta	Status
SiO2	64.8	66.8	-2.02	Both
Al2O3	26.6	26.5	0.08	Both
Fe2O3	0.81	0.98	-0.17	Both
K2O	0.78	0.86	-0.08	Both
SO3	0.77	-	0.77	XRF
P2O5	0.26	-	0.26	XRF
CaO	0.25	-	0.25	XRF
Na2O	0.21	0.27	-0.06	Both
MnO	0.18	-	0.18	XRF
Li2O	0.17	-	0.17	XRF
MgO	0.10	-	0.10	XRF
H2O	-	4.55	4.55	SQD

1. Values measured by chemical assay.

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

RT L Fe Cl COMP RES



Semi-Quantitative X-ray Diffraction Results

Low Fe Con

		Low Fe Concentrate			
Mineral	**Chemical Composition**	**HEAD** (wt %)	**Phase Transition Product** (wt %)	**Acid Roast Product** (wt %)	**Concentrate Leach Product** (wt %)
Quartz	SiO_2	3.3	8.4	4.6	2.7
Spodumene (Monoclinic)	$LiAlSi_2O_6$	85.6	-	-	
Spodumene Beta (Tetragonal)	$LiAlSi_2O_6$	-	74.3	-	
Lithium Aluminum Silicate (Hexagonal)	$LiAlSi_2O_6$	-	13	-	
Lithium Sulfate Hydrate (Monoclinic)	Li_2SO_4•H2O	-	-	25.9	
Hydrogen Alumninum Silicate (Hexagonal)	$H(AlSi_2)O_6$	-	-	0.8	4.8
Hydrogen Alumninum Silicate (Tetragonal)	$H(AlSi_2)O_6$	-	-	64.6	84.1
Hematite	Fe_2O_3	-	0.6	-	
Muscovite	$KAl_2(AlSi_3O_{10})(F,OH)_2$	4.7	-	-	
Albite	$NaAlSi_3O_8$	2.9	-	-	2.3
Biotite	$K(Fe,Mg)_3AlSi_3O_{10}(F,OH)_2$	1.8	-	-	
Pyrrhotite	$Fe_{(1-x)}S$	1.1	-	-	
Orthoclase	$KAlSi_3O_8$	0.6	3.7	4.1	
Microcline (Potassium Feldspar)	$KAlSi_3O_8$				5.1
Goethite	αFeO·OH				1.1
TOTAL		**100.0**	**100.0**	**100.0**	**100.1**

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Roch Tech Lithium Inc.*

Project Number/ LIMS No. *12607-001/ MI4504-MAY11*

Reporting Date: *June 22, 2011*

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)

Jennifer LaBelle-Brown, A.Sc.T
Technologist, XRD

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT Con. H Fe	spodumene *(monoclinic)*	quartz	mica, plagioclase	*mellite, *potassium feldspar, *kaolinite, *hematite

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

Mineral	Composition
Hematite	Fe_2O_3
Kaolinite	$Al_2Si_2O_5(OH)_4$
Mellite	$Al_2C_{12}H_{12}•18(H_2O)$
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	$LiAlSi_2O_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT Con. H Fe (wt %)
Spodumene (Monoclinic)	69.4
Quartz	10.0
Muscovite	8.2
Albite	3.7
Anorthite	3.2
Mellite	1.6
Orthoclase	1.5
Kaolinite	1.4
Hematite	0.8
TOTAL	99.8

Chemical Balance

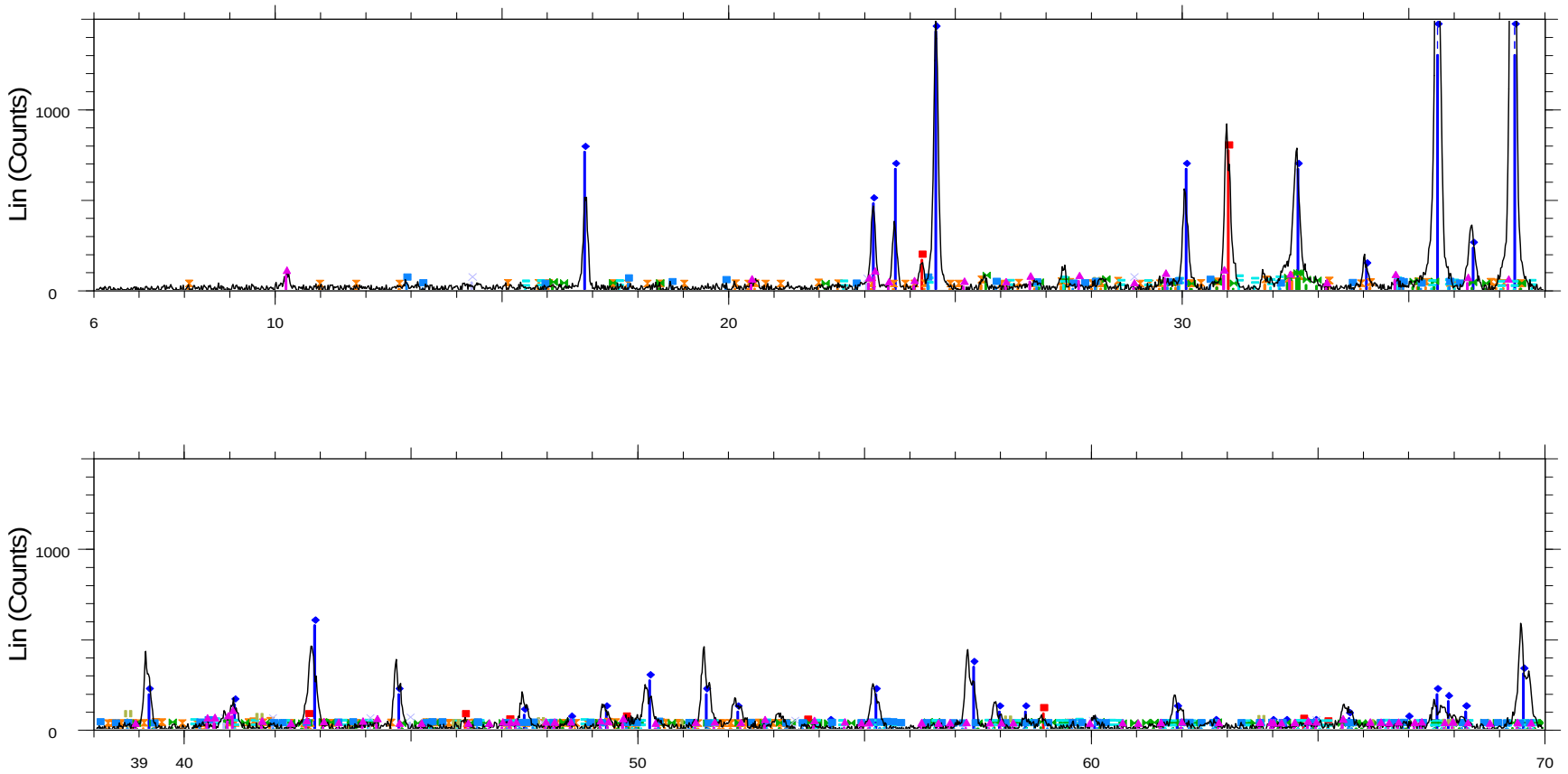
RT Con. H Fe

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	46.8	51.2	-4.39	Both
Silicon	29.5	30.0	-0.50	Both
Aluminum	13.4	13.3	0.13	Both
Lithium	2.79	2.59	0.20	Both
Iron	0.92	0.59	0.32	Both
Potassium	0.85	1.01	-0.17	Both
Sodium	0.67	0.33	0.34	Both
Calcium	0.37	0.47	-0.09	Both
Manganese	0.19	-	0.19	XRF
Phosphorus	0.16	-	0.16	XRF
Chromium	0.07	-	0.07	XRF
Magnesium	0.04	-	0.04	XRF
Hydrogen	-	0.14	0.14	SQD
Carbon	-	0.35	0.35	SQD

1. Values measured by chemical assay.

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

RT Con H Fe



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Operations: Background 1.000,1.000 | Import
01-079-1910 (C) - Quartz - SiO2
00-033-0786 (*) - Spodumene - LiAlSi2O6
01-084-1305 (C) - Muscovite - KAl3Si3O10(OH)2
01-076-0948 (C) - Anorthite - Ca(Al2Si2O8)
01-076-1819 (C) - Albite low - Na(AlSi3O8)
01-086-0439 (C) - Orthoclase - K(AlSi3O8)
00-033-0664 (*) - Hematite, syn - Fe2O3
00-006-0221 (D) - Kaolinite 1Md - Al2Si2O5(OH)4
00-042-1501 (*) - Mellite - C12Al2O12·16H2O/Al2(C6(COO)6)·16H2O

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Rock Tech Lithium Inc.*

Project Number/ LIMS No. *12607-001/ MI4509-MAY11*

Reporting Date: *June 22, 2011*

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
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4) Chemical Balance(s)
5) XRD Pattern(s)

Jennifer LaBelle-Brown, A.Sc.T
Technologist, XRD

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT PT Con. H Fe	lithium aluminum silicate *(tetragonal)*	-	quartz, mellite	*plagioclase, *potassium feldspar, *maghemite, *aluminum oxide, *montmorillonite, *spodumene *(monoclinic)*

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

Mineral	Composition
Aluminum Oxide	Al_2O_3
Lithium Aluminum Silicate	beta-$LiAlSi_2O_6$
Maghemite	γ-Fe_2O_3
Mellite	$Al_2C_{12}H_{12}$•18(H_2O)
Montmorillonite	$(Al,Mg)_8(Si_4O_{10})_3(OH)_{10}$*12$H_2O$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Quartz	SiO_2
Spodumene	$LiAlSi_2O_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT PT Con. H Fe (wt %)
Lithium Aluminum Silicate (Tetragonal)	86.7
Quartz	4.2
Mellite	3.3
Albite	1.4
Orthoclase	1.1
Maghemite	1.0
Aluminum Oxide	0.8
Montmorillonite	0.6
Anorthite	0.6
Spodumene (Spodumene)	0.3
TOTAL	100.0

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.

Chemical Balance

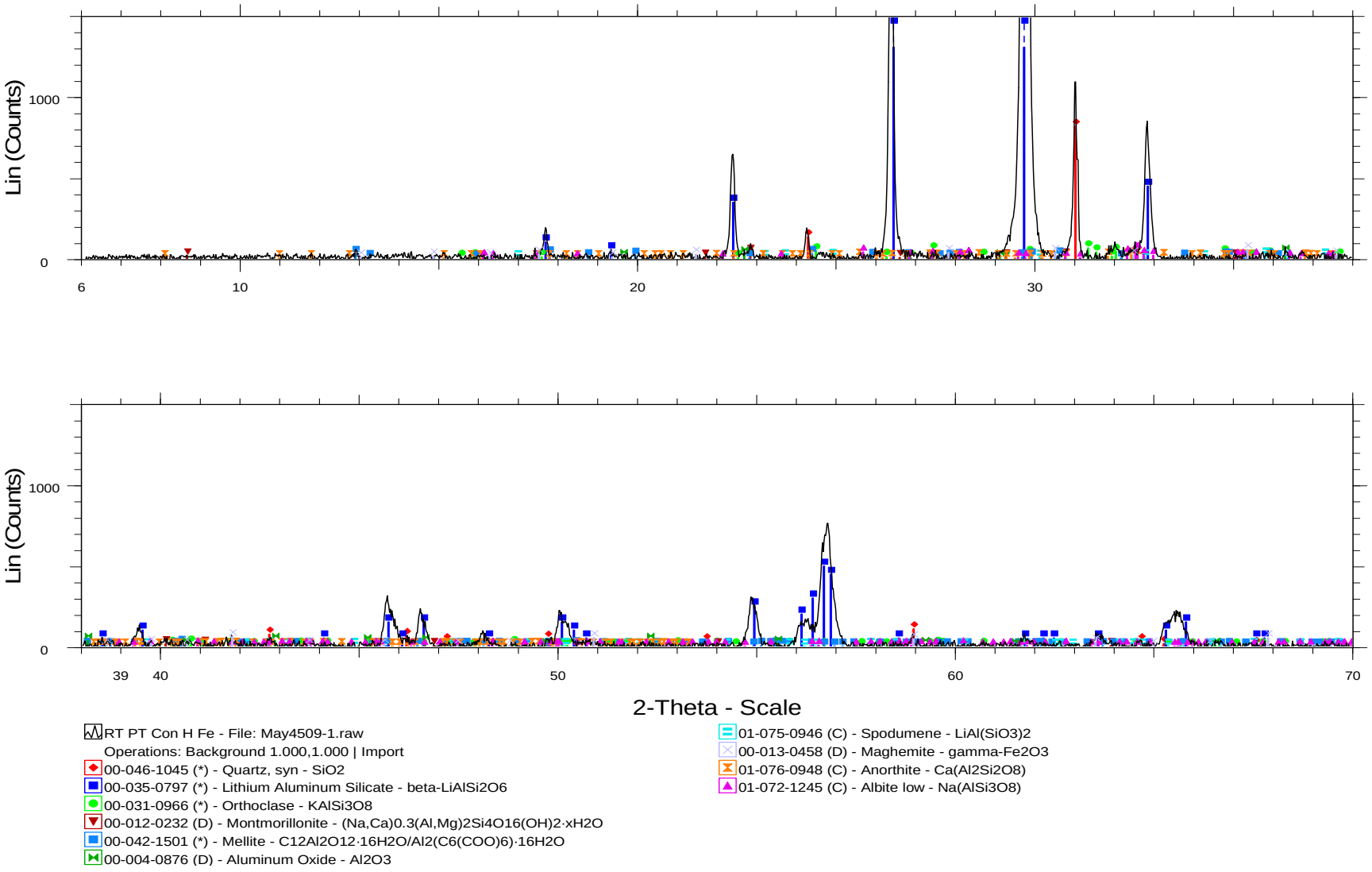
RT PT Con. H Fe

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	47.6	51.8	-4.22	Both
Silicon	29.9	29.3	0.62	Both
Aluminum	13.8	13.7	0.07	Both
Lithium	2.74	3.25	-0.51	Both
Iron	0.97	0.70	0.27	Both
Potassium	0.83	0.15	0.68	Both
Sodium	0.67	0.13	0.54	Both
Calcium	0.37	0.09	0.28	Both
Manganese	0.19	-	0.19	XRF
Phosphorus	0.14	-	0.14	XRF
Chromium	0.07	-	0.07	XRF
Magnesium	0.03	0.03	0.00	Both
Hydrogen	-	0.16	0.16	SQD
Carbon	-	0.70	0.70	SQD

1. Values measured by chemical assay.

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

RT PT Con H Fe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00-046-1045 (*) - Quartz, syn - SiO2
00-035-0797 (*) - Lithium Aluminum Silicate - beta-LiAlSi2O6
00-031-0966 (*) - Orthoclase - KAlSi3O8
00-012-0232 (D) - Montmorillonite - (Na,Ca)0.3(Al,Mg)2Si4O16(OH)2·xH2O
00-042-1501 (*) - Mellite - C12Al2O12·16H2O/Al2(C6(COO)6)·16H2O
00-004-0876 (D) - Aluminum Oxide - Al2O3
01-075-0946 (C) - Spodumene - LiAl(SiO3)2
00-013-0458 (D) - Maghemite - gamma-Fe2O3
01-076-0948 (C) - Anorthite - Ca(Al2Si2O8)
01-072-1245 (C) - Albite low - Na(AlSi3O8)

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Rock Tech Lithium Ltd.*
Project Number/ LIMS No.	*12607-001/ MI4506-JUN11*
Reporting Date:	*June 22, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT AR H Fe	hydrogen aluminum silicate *(tetragonal)*	lithium sulfate hydrate *(Monoclinic)*	potassium feldspar, plagioclase, quartz, lithium aluminum silicate *(Orthorombic),*	*goethite

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

Mineral	Composition
Quartz	SiO_2
Hydrogen Aluminum Silicate	$H(AlSi_2)O_6$
Lithium Aluminum Silicate	Li_3AlSiO_5
Lithium Sulfate Hydrate	$Li_2SO_4 \cdot H_2O$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Goethite	$\alpha FeO \cdot OH$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT AR H Fe (wt %)
Quartz	6.5
Hydrogen Aluminum Silicate (Tetragonal)	58.5
Lithium Sulfate Hydrate (Monoclinic)	18.0
Orthoclase	5.4
Lithium Aluminum Silicate (Orthorombic)	3.4
Albite	6.9
Goethite	1.2
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.

Chemical Balance

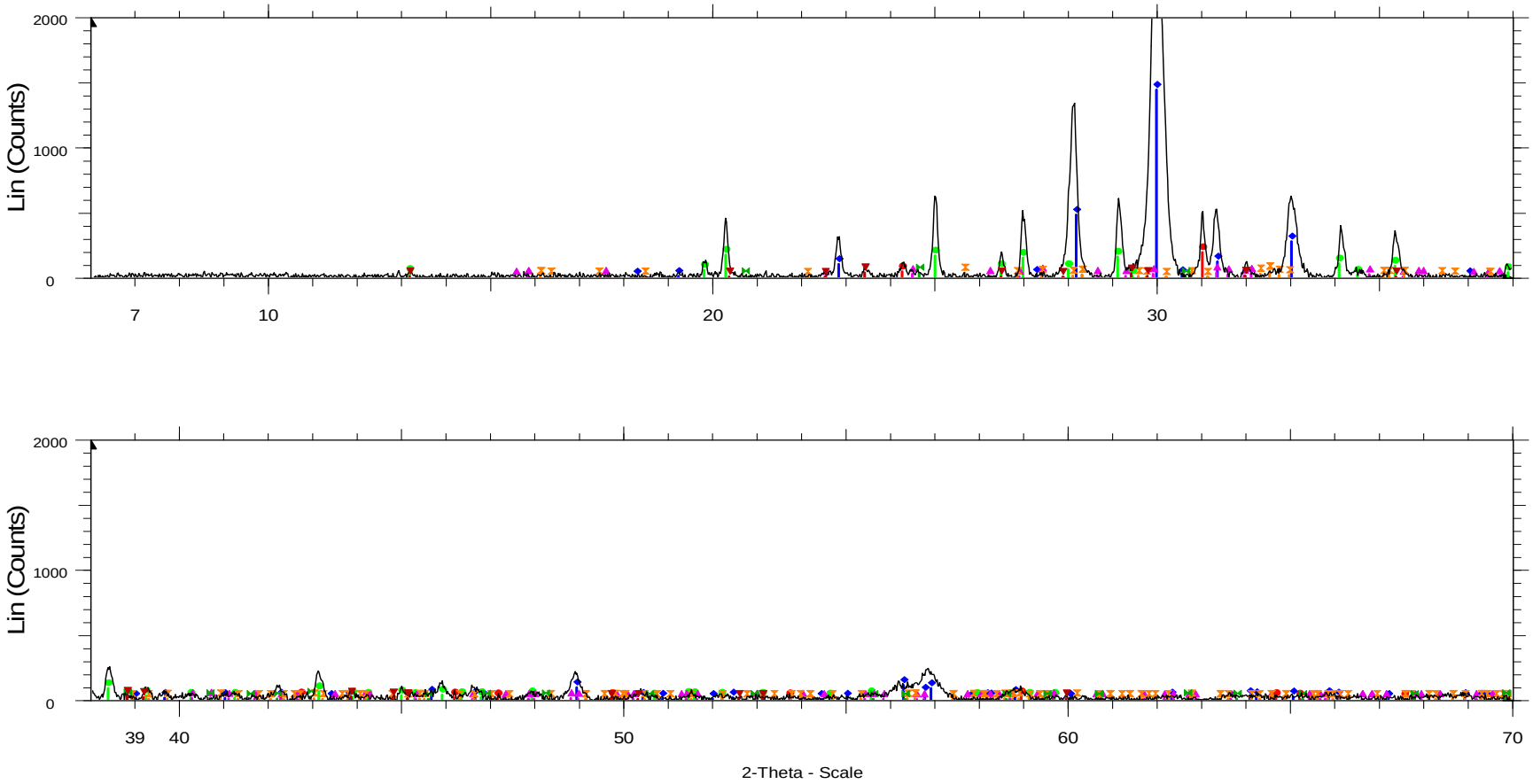
RT AR H Fe

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	37.5	54.0	-16.5	Both
Silicon	23.6	25.8	-2.20	Both
Aluminum	10.8	10.6	0.20	Both
Lithium	2.14	2.41	-0.27	Both
Iron	0.78	0.76	0.02	Both
Potassium	0.70	0.76	-0.06	Both
Sodium	0.56	0.61	-0.05	Both
Calcium	0.29	-	0.29	XRF
Manganese	0.16	-	0.16	XRF
Phosphorus	0.12	-	0.12	XRF
Chromium	0.06	-	0.06	XRF
Magnesium	0.04	-	0.04	XRF
Hydrogen	-	0.61	-0.61	SQD
Sulfur	-	4.51	-4.51	SQD
Deuterium	-	0.03	-0.03	SQD

1. Values measured by chemical assay.

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

RT AR H Fe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2-Theta - Scale
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01-079-1910 (C) - Quartz - SiO2
01-079-1982 (C) - Hydrogen Aluminum Silicate - H(AlSi2)O6
00-015-0873 (I) - Lithium Sulfate Hydrate - Li2SO4·H2O
01-075-1190 (C) - Orthoclase - K(AlSi3)O8
00-042-0295 (D) - Lithium Aluminum Silicate - Li3AlSiO5
01-074-0603 (C) - Albite low - Na(AlSi3O8)
01-074-2195 (C) - Goethite, syn - FeOOD

P

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Met Ops*

Project Number/ LIMS No. *12607-002/MI4520-JUN11*

Reporting Date: *July 28, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

P

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
RT H Fe CL COMP RES	Hydrogen Aluminum Silicate *(Tetragonal)*	-	potassium-feldspar, Hydrogen Aluminum Silicate *(Hexagonal),* quartz, plagioclase	*goethite

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

Mineral	Composition
Quartz	SiO_2
Hydrogen Aluminum Silicate (Tetragonal)	$H(AlSi_2)O_6$
Hydrogen Aluminum Silicate (Hexagonal)	$H(AlSi_2O_6)$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Goethite	αFeO·OH

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Mineral	RT H Fe CL COMP RES (wt %)
Quartz	4.2
Hydrogen Aluminum Silicate (Tetragonal)	78.4
Hydrogen Aluminum Silicate (Hexagonal)	6.1
Albite	3.8
Microcline	6.3
Goethite	1.2
TOTAL	100.0

P

RT H Fe CL COMP RES

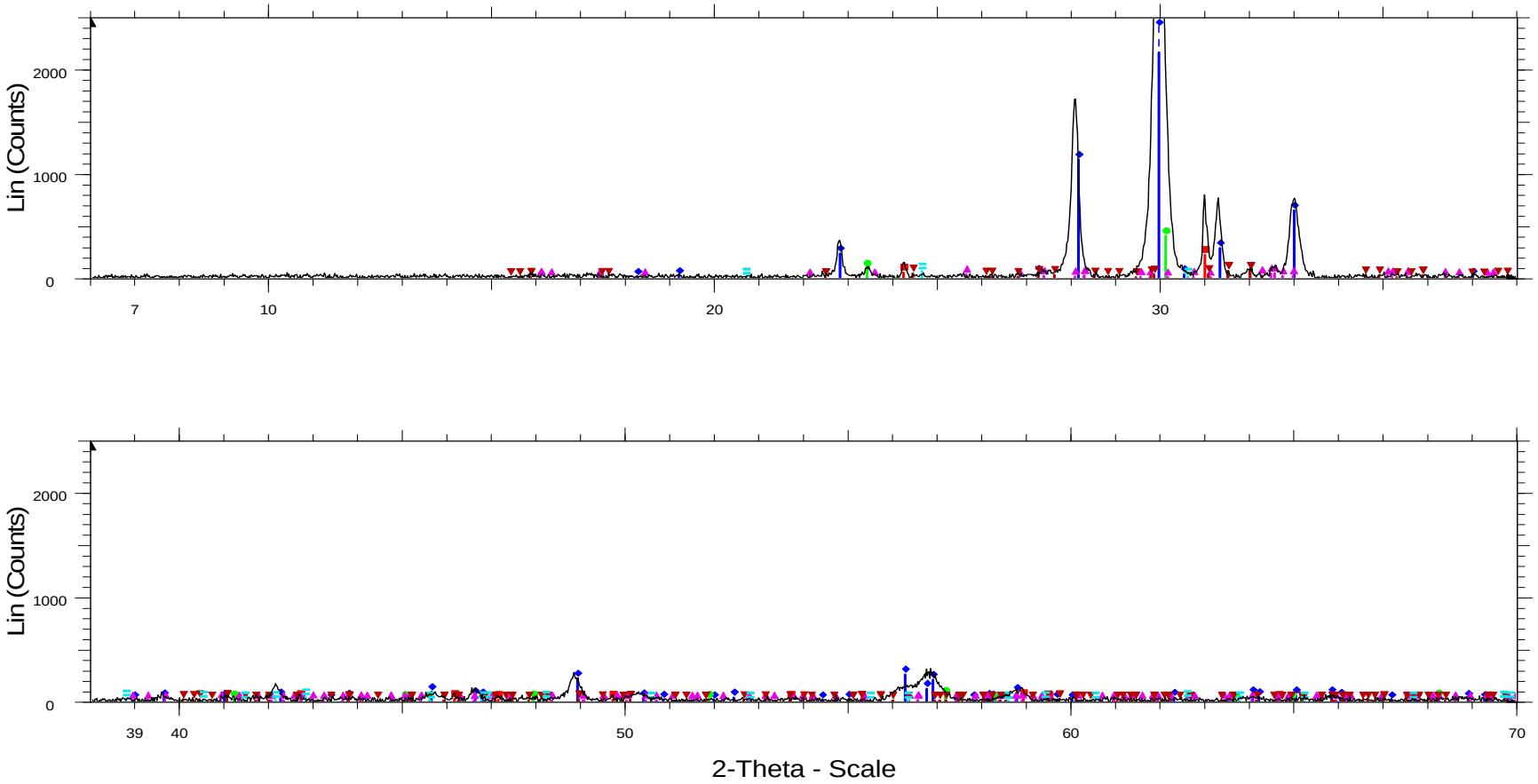
Name	Assay[1]	SQD[2]	Delta	Status
SiO2	65.5	67.2	-1.73	Both
Al2O3	25.6	25.8	-0.21	Both
Fe2O3	1.11	1.10	0.01	Both
K2O	1.04	1.06	-0.02	Both
SO3	0.45	-	0.45	XRF
Na2O	0.44	0.45	-0.01	Both
CaO	0.32	-	0.32	XRF
P2O5	0.27	-	0.27	XRF
MnO	0.22	-	0.22	XRF
Li2O	0.19	-	0.19	XRF
Cr2O3	0.10	-	0.10	XRF
MgO	0.09	-	0.09	XRF
H2O	-	4.35	4.35	SQD

1. Values measured by chemical assay.

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

P

RT H Fe Cl COMP RES



Semi-Quantitative X-ray Diffraction Results
High Fe Con

Mineral	Chemical Composition	High Fe Concentrate			
		HEAD (wt %)	Phase Transition Product (wt %)	Acid Roast Product (wt %)	Concentrate Leach Product (wt %)
Quartz	SiO_2	10	4.2	6.5	4.2
Spodumene (Monoclinic)	$LiAlSi_2O_6$	69.4	0.3	-	
Spodumene Beta (Tetragonal)	$LiAlSi_2O_6$	-	86.7	-	
Lithium Aluminum Silicate (Orthorombic)	Li_3AlSiO_5	-	-	3.4	
Lithium Sulfate Hydrate (Monoclinic)	Li_2SO_4•H2O	-	-	18	
Hydrogen Alumninum Silicate (Hexagonal)	$H(AlSi_2)O_6$				6.1
Hydrogen Alumninum Silicate (Tetragonal)	$H(AlSi_2)O_6$	-	-	58.5	78.4
Hematite	Fe_2O_3	0.8	-	-	
Muscovite	$KAl_2(AlSi_3O_{10})(F,OH)_2$	8.2	-	-	
Albite	$NaAlSi_3O_8$	3.7	1.4	6.9	3.8
Orthoclase	$KAlSi_3O_8$	1.5	1.1	5.4	
Anorthite	$CaAl_2Si_2O_8$	3.2	0.6	-	
Mellite	$Al_2C_{12}H_{12}$•$18(H_2O)$	1.6	3.3	-	
Kaolinite	$Al_2Si_2O_5(OH)_4$	1.4	-	-	
Maghemite	γ-Fe_2O_3	-	1.0	-	
Aluminum Oxide	Al_2O_3	-	0.8	-	
Montmorillonite	$(Al,Mg)_8(Si_4O_{10})_3(OH)_{10}$*$12H_2O$	-	0.6	-	
Microcline (Potassium Feldspar)	$KAlSi_3O_8$				6.3
Goethite	αFeO·OH	-	-	1.2	1.2
TOTAL		**99.8**	**100.0**	**99.9**	**100.0**

Missing RT. Con. H Fe Water Leach

Sample ID	Minerology LIMS	VRA LIMS	Li LIMS
RT H Fe CL COMP RES	MI4520-JUN11	02852-JUN11	

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Met Ops*

Project Number/ LIMS No. *12607-001/MI4500-JUN11*

Reporting Date: *June 17, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown 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) database and released on software as Powder Diffraction Files (PDF).

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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.

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)
RT HM Con	spodumene (Monoclinic)	-	quartz, mica, plagioclase	*goethite

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

Mineral	Composition
Goethite	αFeO·OH
Mica	$K(Mg,Fe)Al_2Si_3AlO_{10}(OH)_2$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Quartz	SiO_2
Spodumene	$LiAlSi_2O_6$ (Monoclinic)

Semi-Quantitative X-ray Diffraction Results

Mineral	RT HM Con (wt %)
Quartz	6.8
Spodumene (Monoclinic)	81.7
Albite	5.7
Muscovite	5.1
Goethite	0.8
TOTAL	100.1

Chemical Balance

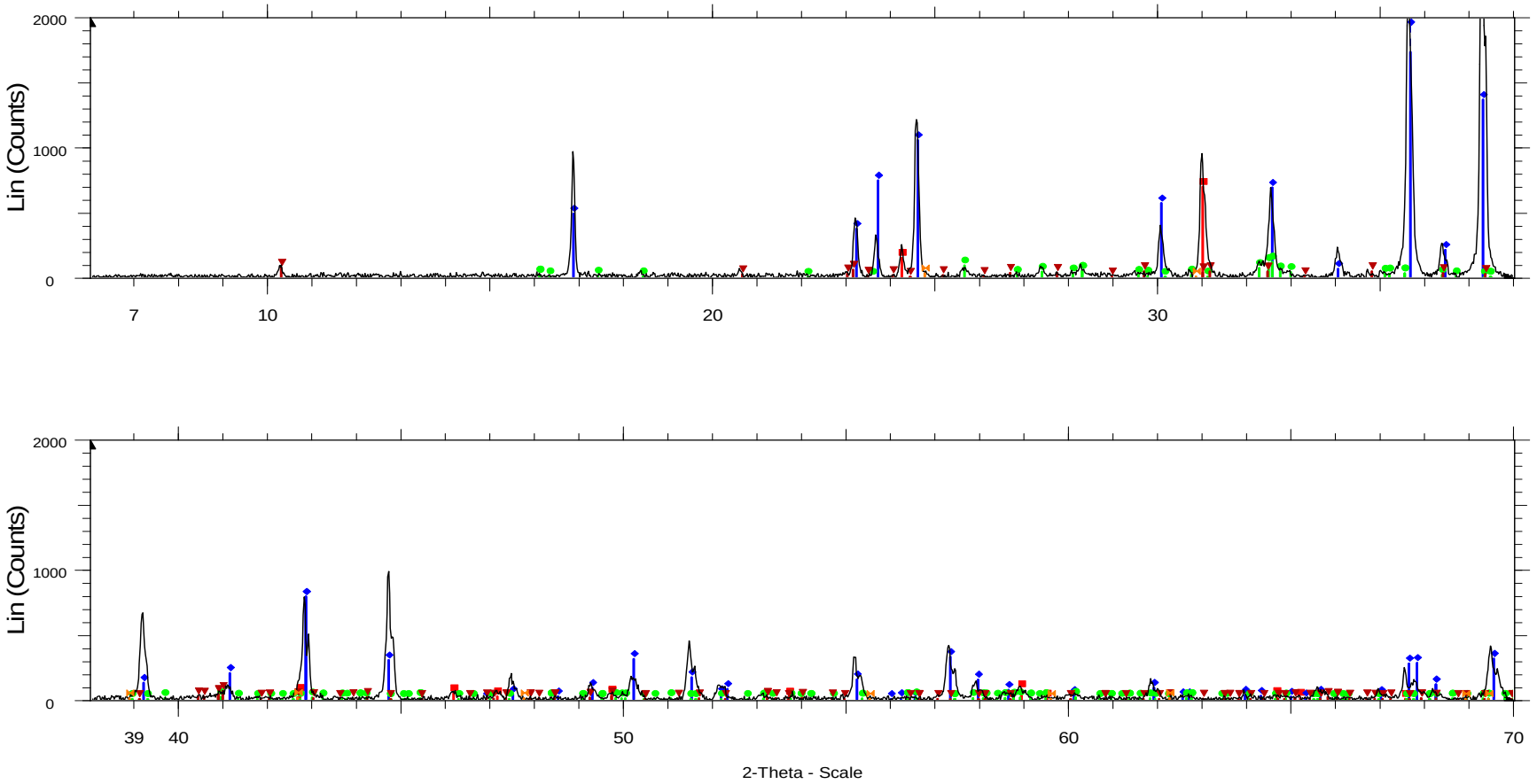
RT HM Con

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	46.9	51.1	-4.25	Both
Silicon	30.2	30.7	-0.50	Both
Aluminum	13.1	13.4	-0.38	Both
Lithium	2.92	3.05	-0.13	Both
Iron	0.68	0.67	0.01	Both
Sodium	0.54	0.55	-0.01	Both
Potassium	0.47	0.41	0.06	Both
Calcium	0.14	-	0.14	XRF
Manganese	0.13	-	0.13	XRF
Magnesium	0.05	-	0.05	XRF
Phosphorus	0.05	-	0.05	XRF
Chromium	0.02	-	0.02	XRF
Hydrogen	-	0.03	0.03	SQD

1. Values measured by chemical assay.

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

RT HM Con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2-Theta - Scale
RT HM Con - File: Jun4500-1.raw - Type: 2Th/Th locked
01-079-1910 (C) - Quartz - SiO2
01-075-1091 (C) - Spodumene - LiAlSi2O6
01-084-0752 (C) - Albite low - Na(AlSi3O8)
01-080-0742 (C) - Muscovite 2M1 - (K0.82Na0.18)(Fe0.03Al1.97)(AlSi3)O10(OH)2
00-003-0251 (D) - Goethite - Fe+3O(OH)

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Rock Tech Resources Inc*

Project Number/ LIMS No. *12607-001/MI4511-Jun11*

Reporting Date: *June 15, 2011*

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
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4) Chemical Balance(s)
5) XRD Pattern(s)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown 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) database and released on software as Powder Diffraction Files (PDF).

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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)
RT PT HL Con	spodumene beta *(Tetragonal)*	-	quartz, lithium aluminum silicate *(Hexagonal)*, plagioclase	*goethite,*siderite

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

Mineral	Composition
Albite	$NaAlSi_3O_8$
Lithium Aluminum Silicate	$LiAlSi_2O_6$
Quartz	SiO_2
Siderite	$FeCO_3$
Spodumene beta	$LiAlSi_2O_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT PT HL Con (wt %)
Quartz	9.9
Spodumene beta (Tetragonal)	72.9
Lithium Aluminum Silicate (Hexagonal)	9.6
Siderite	0.4
Albite	6.5
Goethite	0.8
TOTAL	100.1

Chemical Balance

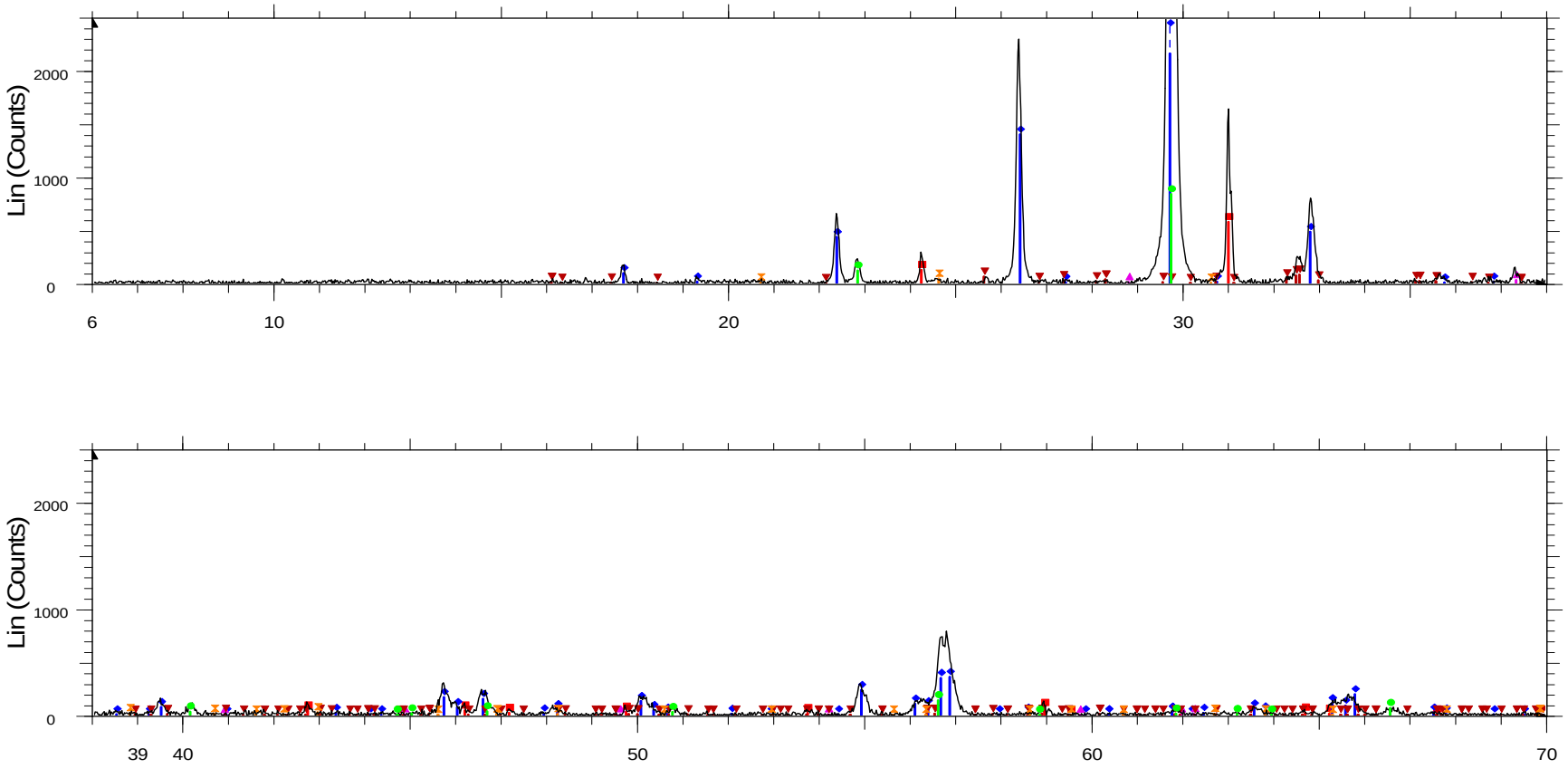
RT PT HL Con

Name	Assay[1]	SQD[2]	Delta	Status
Oxygen	47.7	51.4	-3.71	Both
Silicon	30.8	31.6	-0.80	Both
Aluminum	13.3	12.6	0.71	Both
Lithium	2.83	3.07	-0.24	Both
Iron	0.66	0.67	-0.01	Both
Sodium	0.53	0.56	-0.04	Both
Potassium	0.42	-	0.42	XRF
Manganese	0.15	-	0.15	XRF
Calcium	0.14	-	0.14	XRF
Phosphorus	0.05	-	0.05	XRF
Magnesium	0.04	-	0.04	XRF
Carbon	-	0.04	0.04	SQD
Deuterium	-	0.02	0.02	SQD

1. Values measured by chemical assay.

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

RT PT HL Con



RT PT HL Con - File: Jun4517-2.raw - Type: 2Th/Th locked - Start: 6.000 ° - End: 70.006 ° - Step: 0.019 ° - Step time: 12. s - Temp.: 25 °C (Room) - Time Started: 17 s - 2-Theta: 6.000 ° - Th
01-079-1910 (C) - Quartz - SiO2
01-076-0921 (C) - Spodumene beta, syn - LiAlSi2O6
01-074-1095 (D) - Lithium Aluminum Silicate - LiAlSi2O6
01-083-1764 (C) - Siderite - Fe(CO3)
01-083-1466 (C) - Albite low - Na0.986(Al1.005Si2.995O8)
01-074-2195 (C) - Goethite, syn - FeOOD

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Rock Tech Lithium Inc*

Project Number/ LIMS No. *12607-001/MI4528-JUN11*

Reporting Date: *July 10, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT AR HL Con	Hydrogen Aluminum Silicate (Tetragonal)	Lithium Sulfate Hydrate (Monoclinic)	quartz, plagioclase, potassium-feldspar,	*goethite

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

Mineral	Composition
Goethite	αFeO·OH
Hydrogen Aluminum Silicate	$H(AlSi_2)O_6$
Lithium Sulfate Hydrate	$Li_2SO_4·H_2O$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Quartz	SiO_2

Semi-Quantitative X-ray Diffraction Results

Mineral	RT AR HL Con (wt %)
Quartz	4.6
Hydrogen Aluminum Silicate (Tetragonal)	64.0
Lithium Sulfate Hydrate (Monoclinic)	22.2
Microcline	3.2
Goethite	1.0
Albite	5.0
TOTAL	100.0

Chemical Balance

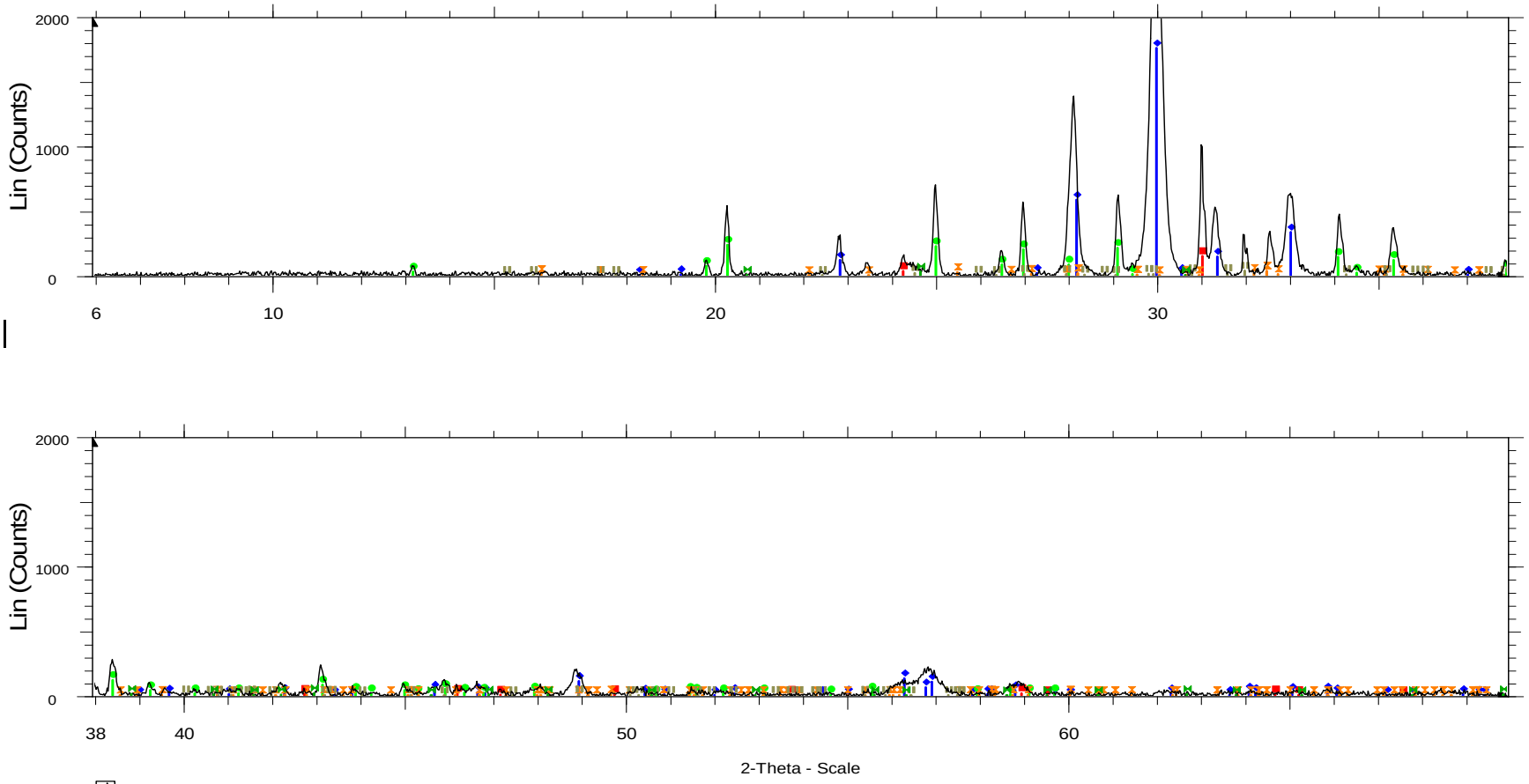
RT AR HL Con

Name	Assay[1]	SQD[2]	Delta	Status
SiO2	50.7	52.8	-2.09	Both
Al2O3	19.4	19.7	-0.28	Both
Li2O	5.06	5.19	-0.13	Both
Fe2O3	0.70	0.85	-0.15	Both
Na2O	0.56	0.60	-0.04	Both
K2O	0.44	0.55	-0.11	Both
CaO	0.16	-	0.16	XRF
MnO	0.14	-	0.14	XRF
P2O5	0.08	-	0.08	XRF
MgO	0.06	-	0.06	XRF
H2O	-	6.33	6.33	SQD
SO3	-	13.9	13.9	SQD
D2O	-	0.11	0.11	SQD

1. Values measured by chemical assay.

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

RT AR HL Con


Lin (Counts)
2000
1000
0
6
10
20
30
Lin (Counts)
2000
1000
0
38
40
50
60
2-Theta - Scale
RT AR HL Con - File: Jun4528-1.raw - Type: 2Th/Th locked
01-079-1910 (C) - Quartz - SiO2
01-079-1982 (C) - Hydrogen Aluminum Silicate - H(AlSi2)O6
00-015-0873 (I) - Lithium Sulfate Hydrate - Li2SO4·H2O
01-074-2195 (C) - Goethite, syn - FeOOD
01-076-0897 (C) - Albite low - Na(AlSi3O8)
00-022-0687 (D) - Microcline, ordered - KAlSi3O8

Semi-Quantitative X-Ray Diffraction

Report Prepared for: *Roch Tech Lithium Inc*

Project Number/ LIMS No. *12607-002/MI4526-JUN11*

Reporting Date: *July 10, 2011*

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)

Bernie C. Yeung, B. Sc.
Mineralogist

Huyun Zhou, Ph.D.
Senior Mineralogist

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.

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)
RT HM Fe CL COMP RES	hydrogen aluminum silicate *(Tetragonal)*	-	hydrogen aluminum silicate *(Hexagonal)*, quartz, spodumene *(Monoclinic)*, plagioclase, potassium-feldspar	*pyrite, *goethite

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

Mineral	Composition
Goethite	αFeO·OH
Hydrogen Aluminum Silicate *(Hexagonal)*	$H(AlSi_2O_6)$
Hydrogen Aluminum Silicate *(Tetragonal)*	$H(AlSi_2)O_6$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Potassium Feldspar	$KAlSi_3O_8$
Pyrite	FeS_2
Quartz	SiO_2
Spodumene *(Monoclinic)*	$LiAlSi_2O_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	RT HM Fe CL COMP RES (wt %)
Quartz	9.4
Hydrogen Aluminum Silicate (Tetragonal)	76.5
Spodumene (Monoclinic)	3.0
Albite	2.1
Hydrogen Aluminum Silicate (Hexagonal)	5.7
Orthoclase	2.4
Pyrite	0.7
Goethite	0.1
TOTAL	99.9

Chemical Balance

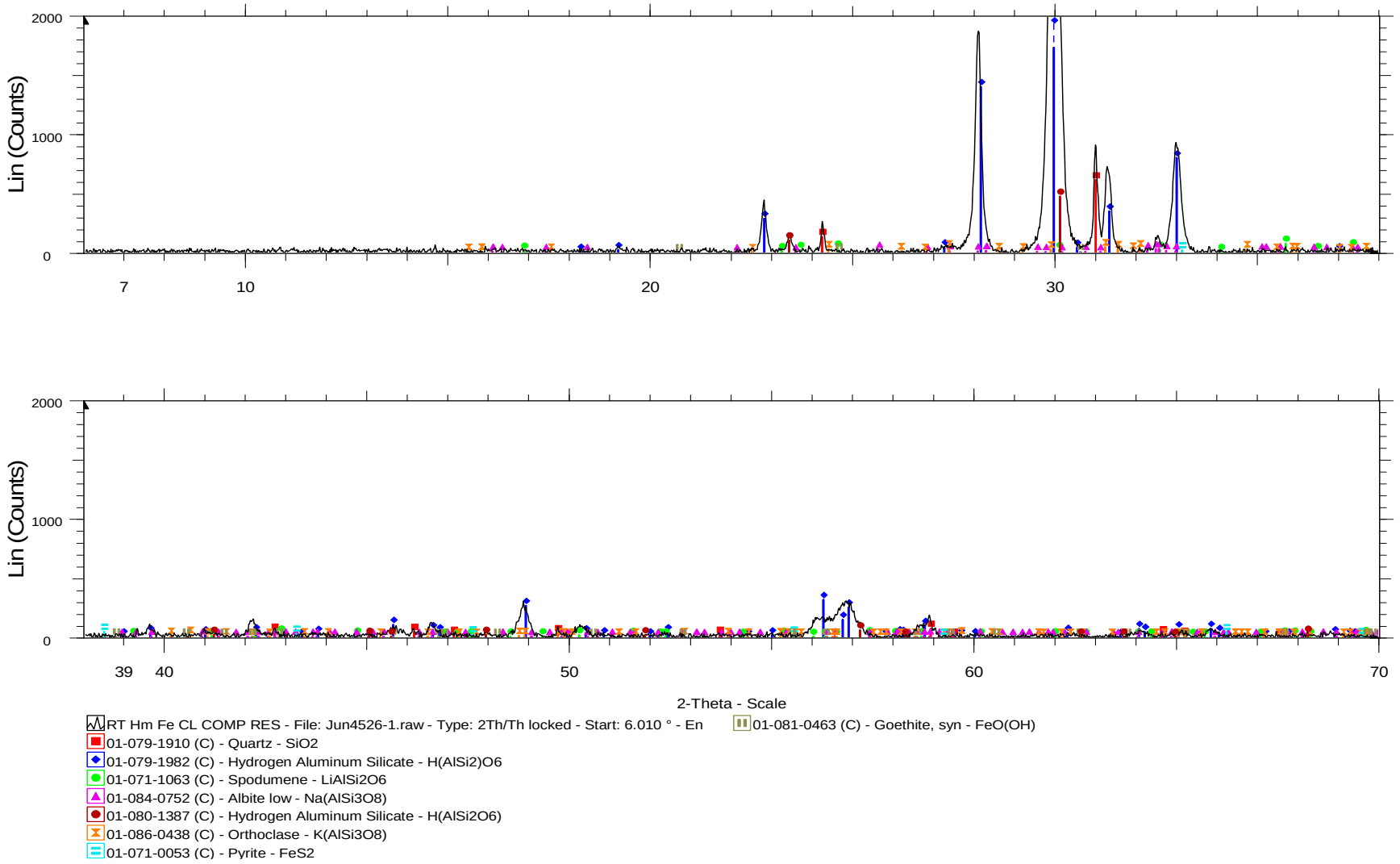
RT HM Fe CL COMP RES

Name	Assay[1]	SQD[2]	Delta	Status
SiO2	67.2	69.2	-2.04	Both
Al2O3	26.0	25.0	1.03	Both
SO3	1.95	0.92	1.03	Both
Fe2O3	0.54	0.54	0.00	Both
K2O	0.43	0.41	0.02	Both
Li2O	0.24	0.25	-0.01	Both
Na2O	0.23	0.25	-0.02	Both
MnO	0.11	-	0.11	XRF
CaO	0.09	-	0.09	XRF
P2O5	0.06	-	0.06	XRF
MgO	0.05	-	0.05	XRF
H2O	-	4.12	4.12	SQD

1. Values measured by chemical assay.

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

RT Hm Fe CL COMP RES



Semi-Quantitative X-ray Diffraction Results

Heavy Media/Heavy Liquid Con

		Heavy Media Concentrate				
Mineral	**Chemical Composition**	**Head** (wt %)	**Phase Transition Product** (wt %)	**Phase Transition Product #2** (wt %)	**Acid Roast Product** (wt %)	**Concentrate Leach Product** (wt %)
Quartz	SiO_2	6.8	9.9	9	4.6	9.5
Spodumene (Monoclinic)	$LiAlSi_2O_6$	81.7	-	-	-	3.1
Spodumene beta (Tetragonal)	$LiAlSi_2O_6$	-	72.9	84.3	-	-
Lithium Aluminum Silicate (Hexagonal)	$LiAlSi_2O_6$	-	9.6	1	-	-
Lithium Sulfate Hydrate (Monoclinic)	Li_2SO_4•H2O	-	-	-	22.2	-
Hydrogen Alumninum Silicate (Hexagonal)	$H(AlSi_2)O_6$	-	-	-	64	5.8
Hydrogen Alumninum Silicate (Tetragonal)	$H(AlSi_2)O_6$	-	-	-	-	77.1
Muscovite	$KAl_2(AlSi_3O_{10})(F,OH)_2$	5.1	-	-	-	-
Albite	$NaAlSi_3O_8$	5.7	6.5	3.9	5	2.1
Petalite	$Li(AlSi_4O_{10})$	-	-	-	-	-
Magnetite	Fe_3O_4	-	-	-	-	-
Orthoclase	$KAlSi_3O_8$	-	-	1.8	-	2.4
Microcline	$KAlSi_3O_8$	-	-	-	3.2	-
Siderite	$FeCO_3$	-	0.4	-	-	-
Goethite	αFeO·OH	0.8	0.8	-	1	-
TOTAL		**100.1**	**100.1**	**100.0**	**100.0**	**100.0**

Appendix C – Initial Li_2CO_3 Product

Test Details and Data

Project: CALR-12607-002 **Project Tech.:** A. Ferguson (2290)
Client: Rock Tech **Test Date:** 17-May-11

Test Name: **Water Leach @ 50% Solids**
Test No.: **RT L Fe WL T1** **Project Manager:** Stephen Mackie (ext.2058)

Feed/Head: **Acid Roasted β-Spodumene - Variable Acid Dose Tests**
(%Excess Acid of Acid Bake = 30 %)

Test Objective: To conduct a water leach on AR β-Spod that was produced in Muffle furnace.

Test Summary: The feed sample is subjected to a water leach for 60 minutes.

Reagents:

Conc.	Chemical(s)
100%	deioized water (Rocky Ridge)

**review all pertanant MSDS prior to handling reagents and ALWAYS wear proper PPE.*

Apparatus:

Qty.	Equipment
1x	~6kg balance, x.xxg ***NOTE: all glassware acid washed before use.***
various	120mL containers
various	1L containers
1x	1 L reactor
1x	Caframo mixer
1x	15cm Buchner filter funnel
1x	glass drying pan
1x	teflon impeller
1x	pH meter w/ATC
1x	Whatman No. 1 filter paper, 12.5cm
1x	500 mL vacuum flask
1x	spatula

****various connectors, fittings, clamps, tools etc.***

Procedure:

1. Measure out equal charges of the AR β-Spod and deionized water.
2. Combine the solids and water in a reactor. Stir for 60 minutes.
3. Measure & record initial and final solution pH and ORP.
4. Filter sample through buchner filter. Recover the filtrate. Measure & record mass and solution density.
5. Conduct three displacement washes of the filter cake, each wash is of ~1.8 kg.
6. Measure & record mass and solution density of WSH fraction.
7. Recover the Final RES, record the NET wet & dry mass, dry in oven, submit to bucking room.
8. Perform analysis on Final FIL and WSH samples from each step; submit samples to login.

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
AR β-Spod Head	RT L Fe WL T1 Head	Li-AAS, ICP-OES	***CA02299-JUN11***
Final Filtrate + Wash 1	RT L Fe WL T1 Final PLS + WSH 1	Li-AAS, ICP-OES	***CA02857-MAY11***

Project: **CALR-12607-002** **Project Tech.:** **A. Ferguson (2290)**
Client: **Rock Tech** **Test Date:** **17-May-11**

Test Name: **Water Leach @ 50% Solids**
Test No.: **RT L Fe WL T1** **Project Engineer:** Stephen Mackie (ext.2058)

Test Data:

	Mass, g		
	tare	Initial	Final
Reactor	431.4	7207.8	7178
CLSW		3387.6	
Solids		3387.6	

Test Notes:

TIME	NOTES					
	pH	Temp.(°C)	ORP (mV)	RPM	Ref. START time:	14:14
14:14:00						
14:31:00	2.1	28.1				
15:14:00	Stop					
0:17:00	:Elapsed Time					

Sample Data:

Final Reaction Slurry

	Solution			Solid	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
RT L Fe WL T1 Final PLS	2049.90	1.1691	1753					
RT L Fe WL T1 WSH 1	2257.60	1.117	2021					
RT L Fe WL T1 WSH 2, & 3	4169.20	1.0109	4124					
RT L Fe WL T1 FIN RES				1623.0	5112.8	4293.0	3489.8	2670.0

Mass Red.% 21.18

Test No.:		RT L Fe CL		
Test Parameters		**T, °C**	**pH**	**ORP**
		24	1.95	543
		Assays, % or mg/L		
	mass or vol, g or mL	**Li**	Al	Fe
RT L Fe WL T1 Head	3388	2.46	11.40	0.57
di H2O	3388			
RT L Fe WL T1 Final PLS+ WSH1	3775	24500	2270	583
RT L Fe WL T1 FIN RES	2670	Residue not assayed		
IN (Elemental Masses)				
RT L Fe WL T1 Head		83.17	386.19	19.21
di H2O				
Total IN, g		**83.17**	**386.19**	**19.21**
OUT (Elemental Masses)				
RT L Fe WL T1 Final PLS+ WSH1		92.48	8.57	2.20
RT L Fe WL T1 FIN RES			0.000	0.00
Total OUT, g		**92.48**	**8.57**	**2.20**
Relative Distribution				
RT L Fe WL T1 Final PLS+ WSH1		100.0	100.0	100.0
RT L Fe WL T1 FIN RES		0.0	0.0	0.0
Total, %		**100**	**100**	**100**
%OUT/IN		111.2	2.2	11.5
% Extraction, %		100.0	100.0	100.0

Cal'd Head		2.730	0.253	0.065
% Mass Reduction, %	**21.18**			

Project No.: CALR-12607-002 **Project Technologist:** A. Ferguson (2290)
Client: Rock Tech Lithium **Date of Test:** 18-May-11

Test: **RT L Fe S1 T1**
Feed Feed Solution = Combine the filtrate and WSH-1 fraction of Bulk Water leach
Lime 20 wt%
Apparatus Bulk reactor flask with lid, overhead mixer and impeller, condenser, pH/ORP meter and pH and ORP probes Air Sparging line with in-line flow meter, heating mantle and YSI controller

Test Summary:

To modify the pH of the Feed solution to a specific target within the pH isotherm.

Procedure Feed Solution Prep

Combine the WL filtrates and Wash 1 fractions as discussed. Stir for ~10 minutes.
Measure & record mass and solution density of the resulting solution.
reab a sample of the Feed Sol'n for assay.

Feed Soln	Mass, g	Sol'n Den., g/L	V, mL
	1118.50	1157.4	966.4

Bulk Reactor Set-up

Assemble the reactor with the impeller, lid (ports with septa).(10L pail)
Record the mass of the assembled reactor (this will be the reactor tare).
Standardize pH and ORP probes.

Lime slurry Preparation

Dry Lime Requirement, g/L Feed Sol'n				25.65
Vol of Feed Sol'n, mL				966.39
Approximate mass of Dry Lime required, g				24.8
20% Lime Slurry	Theo.	Actual	4 Portions	
Mass of dry Lime, g	31.0	34.3		
Mass of CLSW, g	123.9	137.3		

Check
20

Prepare a 20% solids slurry of lime based on the above data. Use Can. Li Site Water to prepare the lime slurry.

Stage 1 Test - Air Sparging & pH Modification

Add the Full amount of the combined solution to the tared reactor. Measure & record mass.
Maintain the reaction solution temperature above 15 °C.
Record pH and ORP of the solution.
Start sparging the solution with air and then immediately start pH modification. **Reaction Time = 0.**
Use the Lime slurry to modify the pH of the solution to the specified target values.
Add the lime in ~ 4 portions slowly. Measure the mass of each lime addition charge.
Monitor & record the reaction solution pH & ORP every 5 minutes.
As the reaction slurry pH approaches the target reduce the size of lime slurry charges are added.
pH Target: **5.8**

Sample Preparation

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate; measure & record the mass and soltion density of the filtrate.
Collect a sample of the filtrate for analysis .
Repulp wash the filter cake (900 mL CLSW). Use some of the wash water to rinse out the reactor.
Filter the wash slurry - conduct 2 displacement washes (900 mL/wash).
Collect the Wash fraction; measure & record the mass and soltion density of the combined Wash fraction. Dry the filter cake.

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
Feed	RT L Fe S1 T1 FEED	assayed on WL	CA02857-MAY11
FINAL FIL	RT L Fe S1 T1 FIL	Li,Fe,Al,Ca,Mg	CA02912-MAY11
WASH	RT L Fe S1 T1 WSH	HOLD	
Final RES	RT L Fe S1 T1 RES	HOLD	

Project No.: CALR-12607-002 **Project Technologist:** A. Ferguson (2290)
Client: Rock Tech Lithium **Date of Test:** 18-May-11

Test: **RT L Fe S1 T1**

Data

Combined Feed Solution

Weight (g):	1118.50
SG (g/mL):	1.1574
Volume (mL):	966.39

	Mass, g	
Reactor	Gross	Net
tare	**2661.00**	
Initial	3779.50	**1118.50**
Final	3842.00	**1181.00**

Reaction Time, min	Air Flow, L/min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of lime slurry added, g	Comment/Observations
<0	0.00	19	1.89	541	0.00	Before sparging. RPM 350
11:06	2.00	19.0	1.87	541		Adding LS
11:10	2.00	20.8	2.90	525	20.5	Solution starting to go a tan colour. RPM 450
11:12	2.00	21.3	3.90	458	13.00	Slurry is Tan colour
11:15	2.00	22.4	4.54	461	32.20	Buffer Region between 4.0-4.5. RPM 500
11:20	2.00	22.5	5.61	380	5.5	
11:35	2.00	21.8	5.77	355		
11:40	2.00	21.7	5.80	351		Test complete

Average Temperature (°C):	21.36	Total Lime Added (g):	71.20	dry lime (g)	14.24

lime slurry was 20%w/w made from 92%CaO

Final reaction Slurry

	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, L	Tare	Wet	Dry	Wet	Dry
Filtrate	1000.70	1.132	884.01					
Wash	634.40	1.0278	617.24					
Residue				529	672	571.9	143	42.9

Project No.: CALR-12607-002 **Project Technologist:** A. Ferguson (2290)
Client: Rock Tech Lithium **Date of Test:** 18-May-11

Test: **BL RT L Fe S1 T1**
Feed Feed Solution = Combine the filtrate and WSH-1 fraction of Bulk Water leach
Lime 20 wt%
Apparatus Bulk reactor flask with lid, overhead mixer and impeller, condenser, pH/ORP meter and pH and ORP probes Air Sparging line with in-line flow meter, heating mantle and YSI controller

Test Summary:

To modify the pH of the Feed solution to a specific target within the pH isotherm.

Procedure Feed Solution Prep

Combine the WL filtrates and Wash 1 fractions as discussed. Stir for ~10 minutes.
Measure & record mass and solution density of the resulting solution.
reab a sample of the Feed Sol'n for assay.

Feed Soln	Mass, g	Sol'n Den., g/L	V, mL
	3069.4	1157.4	2652.0

Bulk Reactor Set-up

Assemble the reactor with the impeller, lid (ports with septa).(10L pail)
Record the mass of the assembled reactor (this will be the reactor tare).
Standardize pH and ORP probes.

Lime slurry Preparation

Dry Lime Requirement, g/L Feed Sol'n				14.74	
Vol of Feed Sol'n, mL				2651.98	
Approximate mass of Dry Lime required, g				39.1	
20% Lime Slurry		Theo.	Actual	4 Portions	
Mass of dry Lime, g		48.9	48.9		Check
Mass of CLSW, g		195.5	195.5		20

Prepare a 20% solids slurry of lime based on the above data. Use Can. Li Site Water to prepare the lime slurry.

Stage 1 Test - Air Sparging & pH Modification

Add the Full amount of the combined solution to the tared reactor. Measure & record mass.
Maintain the reaction solution temperature above 15 °C.
Record pH and ORP of the solution.
Start sparging the solution with air and then immediately start pH modification. **Reaction Time = 0.**
Use the Lime slurry to modify the pH of the solution to the specified target values.
Add the lime in ~ 4 portions slowly. Measure the mass of each lime addition charge.
Monitor & record the reaction solution pH & ORP every 5 minutes.
As the reaction slurry pH approaches the target reduce the size of lime slurry charges are added.
pH Target: **5.8**

Sample Preparation

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate; measure & record the mass and soltion density of the filtrate.
Collect a sample of the filtrate for analysis .
Repulp wash the filter cake (900 mL CLSW). Use some of the wash water to rinse out the reactor.
Filter the wash slurry - conduct 2 displacement washes (900 mL/wash).
Collect the Wash fraction; measure & record the mass and soltion density of the combined Wash fraction. Dry the filter cake.

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
Feed	BL RT L Fe S1 T1 FEED	Li,Al,Fe -ASS	CA02857-MAY11
FINAL FIL	BL RT L Fe S1 T1 FIL	Li,Al,Fe,Ca,Mg -ASS	CA02912-MAY11
WASH	BL RT L Fe S1 T1 WSH		
Final RES	BL RT L Fe S1 T1 RES		

Project No.: CALR-12607-002 **Project Technologist:** A. Ferguson (2290)
Client: Rock Tech Lithium **Date of Test:** 18-May-11

Test: **BL RT L Fe S1 T1**

Project Technologist: A. Ferguson (2290)
Test: **BL RT L Fe S1 T1** Date of Test: 18-May-11

Data

Combined Feed Solution

Weight (g):	3069
SG (g/mL):	1.1574
Volume (mL):	2652

	Mass, g	
Reactor	Gross	Net
tare	**174.50**	
Initial	3243.80	**3069**
Final	3426.10	**3252**

Reaction Time, min	Air Flow, L/min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of lime slurry added, g	Comment/Observations
<0	0.00	18.7	1.90	534	0.00	Before sparging. RPM 500
13:09	4.00	18.7	1.83	537		Adding LS
13:13	4.00	20.6	2.72	525	53.4	
13:19	4.00	21.2	3.86	451	32.50	
13:24	4.00	22.6	4.61	397	99.40	
13:28	4.00	22.6	5.08	257	8.7	
13:30	4.00	22.6	5.38	365	0.90	
13:33	4.00	22.3	5.58	365	1.50	
13:47	4.00	21.8	5.79	347	0.50	
13:52	4.00	21.8	5.79	345	0.00	Test complete

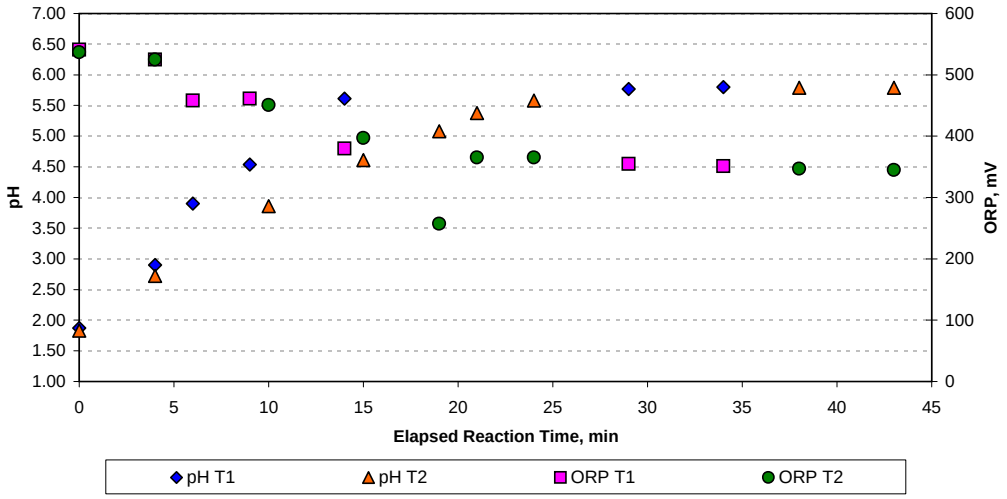
Average Temperature (°C): **21.6** Total Lime Added (g): 196.9 dry lime (g) 39.4

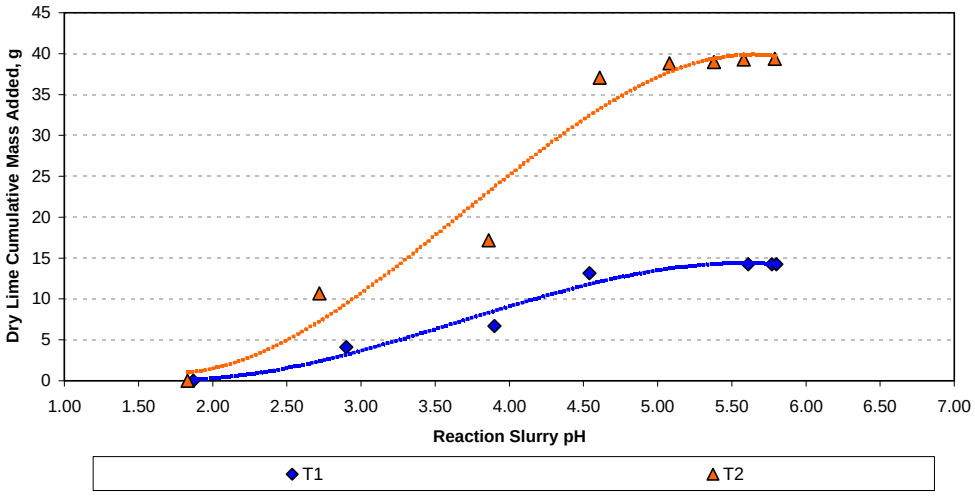
lime slurry was 20%w/w made from 92%CaO

Final reaction Slurry

	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	2629.5	1.1312	2325					
Wash	830.7	1.0619	782					
Residue				2213.8	2662.3	2330.1	448.5	116.3
				Filter Cake %moisture, %			Reaction Slurry %Solids, %	
				74.1			3.6	

Project No.: CALR-12607-002
Client: Rock Tech Lithium





Test No.:	BL RT L Fe S1 T1			
Test Parameters		**T, °C**		
		22		
		Assays, % or mg/L		
	mass or vol, g or mL	**Li**	Fe	Al
BL RT L Fe S1 T1 FEED	2652	24500	583	2270
BL RT L Fe S1 T1 FIL	2324.5	25700	< 0.05	< 2
BL RT L Fe S1 T1 WSH				
BL RT L Fe S1 T1 RES				
IN - Elemental Masses, g				
BL RT L Fe S1 T1 FEED		64.97	1.55	6.02
Total IN, g		**64.97**	**1.55**	**6.02**
OUT - Elemental Masses, g				
BL RT L Fe S1 T1 FIL		59.74	0.00	0.00
BL RT L Fe S1 T1 WSH				
BL RT L Fe S1 T1 RES				
Total OUT, g		**59.74**	**0.00**	**0.00**
%OUT/IN		91.9	0.0	0.0

Cal'd Head		22527	0	0

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 20-May-11
Test No.: **RT L Fe S2 T1**

Feed **BL RT L Fe S1 FIL** LIMS No.: ***CA02912-MAY11***

Reagents 2 M NaOH
50 wt % Na_2CO_3 Solution
Rocky Ridge Water

Apparatus 10 L reactor flask with lid, overhead mixer and impeller, pH/ORP probes & meter

Test Summary: Stage 2 Precipitation - This test is designed to remove the remaining minor impurites from the Li-solution. Stoichiometric addition of Na_2CO_3

Procedure

Reagent Prep.

Prepare the Na_2CO_3 solution as defined.

25	wt% Na_2CO_3(aq) Sol'n
mass Na_2CO_3, g	4.6
mass di H_2O, g	13.9

Test Procedure

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solution therefore ensure that everything that is in contact with the solution is clean.
2 Measure & record the tare of the reactor.
3 Obtain the Feed solution. Add **3154** mL of the solution to the reactor.
4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH, ORP & T of the stirred solution.
6 Modify the pH of the solution to pH = 10 using 2 M NaOH. Add slowly so the pH target is not overshot.
7 Record the mass of NaOH used after each addition. Record the pH, ORP, T and elapsed time.
8 Once the pH is stable at 10 add **18.48 g** of the 25 wt % Na2CO3(aq) solution.
9 Stir the resulting solution for 60 minutes. Measure & record the pH, ORP & T every 5 minutes.
10 After 60 minutes the reaction is complete.
11 Measure & record the final mass of the reactor and reaction slurry.
12 Vacuum filter the reaction slurry. Recover the filtrate. Displacement wash the filter cake (3 X 50 mL RRW). Combine the wash fractions.
13 Recover the filter cake - record wet mass and dry O/N in an oven. When dry record the dry mass.

Sample Preparation

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate and record its mass and solution density.
Collect a sample of the filtrate for analysis .
Conduct 3 displacement washes (~50 mL CLSW /wash).
Collect the Wash fraction - record mass and solution density.
Dry the filter cake.

Analytical Requirements:

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
BL RT L Fe S1 FIL	**BL RT L Fe S1 FIL**	**Li, Al, Fe, Ca, Mg**	**CA02912-MAY11**
Filtrate	**RT L Fe S2 T1 FIL**	**Li, Al, Fe, Ca, Mg**	***CA03034-MAY11***
Wash	**RT L Fe S2 T1 WSH**	**No**	
Residue	**RT L Fe S2 T1 RES**	**No**	

Data

RT L Fe S2 T1

Weight (g):	3569.2
SG (g/mL):	1.1319
Volume (mL):	3153.3

	Mass, g	
Reactor	**Gross**	**Net**
tare	**139.4**	
Initial	**3708.6**	3569.2
Final	**3740.3**	3600.9

2 M NaOH	S.G., g/mL	

Reaction Time, min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of 2 M NaOH added, g	Comment/Observations
9:45	20.7	5.87	296	0	Starting to npH adjust. RPM 600
9:48	20.7	9.46	-212	3.40	Solution is going a brown colour
9:51	20.7	10.03	-349	9.00	
10:02	20.7	10.04	-199	2.40	Solution is really dark brown
10:09	20.7	9.90	-179	0.00	
10:21	20.7	10.06	-187	1.60	
10:23	20.7	10.01	-179	0.00	
10:36	20.7	10.13	-141	1.40	
10:41	20.7	10.03	-169		adding Na2CO3 (18.2g)
10:46	20.7	10.08	-164		
11:03	20.7	10.01	-129		
11:24	20.7	9.96	-85		
11:44	20.7	9.92	-62		Test complete

Average/*Total*

20.70	9.65	17.80

Final reaction Slurry

	Solution			Solid			
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Net Dry
Filtrate, kg	3527.70	1.1318	3116.89				
Wash, g							
Residue				18.3	31.4	21.5	3.2

***NOTE: S.G. on polished (0.45µm) sample.

Rxn Slurry % Solids, % 0.09

LIMS DATA

CA02912-MAY11

Sample ID	Li mg/L	Fe mg/L	Al mg/L	Ca mg/L	Mg mg/L
BL RT L Fe S1 T1	25700	< 0.05	< 2	428	45.8

CA03034-MAY11

Sample ID	Li mg/L	Al mg/L	Fe mg/L	Mg mg/L	Ca mg/L
RT L Fe S2 T1	20000	< 2	< 0.05	20.6	208

Project No.: 12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 20-May-11

Test No.: RT L Fe IX T1
Feed: RT L Fe S2 T1
Resin: S950 purolite

Experimental Procedures:

Review MSDS for resins and every chemical reagent requied in this test.

Solution Preparation:

1 Prepare 600 mL of dilute HCl solution (2N). (500 mL water + 100 mL HCl)
2 Prepare 500 mL of dilute NaOH solution (2N). (40 g NaOH in 500 mL vol. flask)

IX Column Setup:

1 IX test is conducted in a chromatographic column (25 mm ID X 300 mm).
2 Install a back pressure regulator (set at ~5psi) for the discharge tubings conneted to the column.
3 Insert a felt plug into the column and locate it at the bottom of the column.
4 Transfer the resin (100 mL) as a slurry to the column and remove any bubbles by gently tapping the column sides. (Note: Always maintain the solution above the resin bed surface.)
5 Seal the column with a stopper.

Procedure:

Loading: (using one column)

1 The solution generated from the pilot plant (10 L) is passed through the column.
2 The loading test applies down-flow mode at the flowrate of 10 BV/h.
3 Kinetic samples are taken every 5 BV (30 min). (Each sample ~30 mL)
4 Submit selected samples; lable and store the rest samples.

Feed Wash:

1 After loading, 2 BV of DI water is passed through the column.
2 The DI water flows from the top to bottom at 10 BV/h.
3 Kinetic samples are taken every 0.2 BV (20 mL, 1.2 min).

Acid Strip:

1 After Feed Wash, 3 BV of 2 N HCl solution is passed through the column for stripping.
2 The stripping test applies down-flow mode at the flowrate of 5 BV/h.
3 Kinetic samples are taken every 0.3 BV (30 mL, 3.6 min).
4 Submit 10 samples for assay.

Acid Wash:

1 Five BVs of deionized water are passed through the column to replace the acid solution.
2 The wash process applies down-flow mode at the flowrate of 10 BV/h.
3 Kinetic samples are taken every 1 BV (100 mL, 6 min).
4 Label and store samples.

Regeneration:

1 After wash, 300 mL (3 BV) of NaOH solution (2 N) is passed through the column for regeneration.
2 The regeneration process applies down-flow mode at the flowrate of 5 BV/h.
3 Kinetic samples are taken every 0.5 BV (50 mL, 7.2 min/sample).
4 Label and store samples.

Regeneration Wash:

1 300 mL (3 BVs) deionized water is passed through the column to replace the caustic solution.
2 The wash process applies down-flow mode at the flowrate of 10 BV/h.
3 Kinetic samples are taken every 0.5 BV (50 mL, 3 min/sample).
4 Label and store samples.

Note: The resin need to strip and regenerate prior to the loading process.

	Mass, g	SG	Vol, mL
Start	3527.70	1.1318	3116.9
End	3485.8	1.1284	3089.2

CA03034-MAY11

Sample ID	Li mg/L	Al mg/L	Fe mg/L	Mg mg/L	Ca mg/L
RT L Fe S2 T1	20000	< 2	< 0.05	20.6	208

CA03064-MAY11

Sample ID	Li mg/L	Mg mg/L	Ca mg/L
RT L Fe IX T1	20800	4.28	117

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 24-May-11

Li_2CO_3 Precipitation
Test No. RT L Fe Li2CO3 PPT T1
Objective: To conduct Li_2CO_3 PPT followed by Bicarb Polishing

Feed IX-Product

Reagents	Li_2CO_3 (99.3%*)	Rocky Ridge Water
	25 wt. % Na_2CO_3(aq)	(This solution was preheated to 45 °C)

Apparatus One 4 L reactor with lid, overhead mixer and impeller, with pH and ATC probes

Test Summary:

To produce Li2CO3 from a Spodumene sample.

Stoichiometric Excess of Na_2CO_3	**1.04**

	Required
	25 wt% Na_2CO_3(aq) Sol'n
mass Na_2CO_3, g	250.9
mass RR water H_2O, g	752.6
Total Mass of 25% Na_2CO_3(aq), g	**1003.5**

Procedure

Reagent Prep.

Prepare the Na_2CO_3 solution as defined.

Mass, g	1723

Li2CO3 Precipitation Test

1 Weigh out the required amount of feed solution.
2 Add solution to the 4 L reactor. Pre heat the feed solution to ~90 - 95 °C.
3 Prepare the Na2CO3 solution. Pre-heat to 45 °C.
4 Measure & record the pH & temp of the feed solution.
5 Start the addition of Na2CO3(aq). Add total volume of Na2CO3(aq) over 10 minutes. Note the time when all of the solution has been added. This is T = 0 min.
6 Stir vigourously for 20 minutes.
7 After 20 minutes record final pH. Record mass of reaction slurry.
8 Vacuum filter the reaction slurry.

NOTE: Theoretically 168.2 g of Li_2CO_3 will be produced.

Product

Recover the Product Filtrate. Measure & record the mass and solution density.
Wash the Product solids with hot d.i. water (T≥ 95 °C) (ca. 3 x 150 mL).
Combine Wash fractions. Measure & Record mass and solution density of combined wash fraction.
Dry the filter cake.
Record the dry mass of the product filter cake.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
Feed	RT L Fe Li2CO3 PPT T1 Feed	Li,Mg-AAS, Ca-ICP	
Product Filtrate	RT L Fe Li2CO3 PPT T1 FIL	HOLD	
Product Wash	RT L Fe Li2CO3 PPT T1 WSH	HOLD	
PROD	RT L Fe Li2CO3 PPT T1 PROD	HOLD	

Test No. **RT L Fe Li2CO3 PPT T1**

Project Technologist: A.Ferguson
Date of Test: **24-May-11**

Initial Data

Feed Solution

Mass, g	**1723.0**
Soln Den., g/mL	1.1284
Cal'd V, mL	1526.9

	Mass, g	
Reactor	Gross	Net
tare	**2773.80**	
Initial	4496.80	**1723.00**
Final	5489.00	**2715.20**

Reaction Time, min	Precipitation Test Data			
	Sol'n T, °C	Sol'n pH	Mass of Na2CO3(aq) Added, g	Comment
13:26	17.1	9.68	0.0	heat on, initial readings, rpm: 400
13:52	58.8	8.49	0.0	
14:10	94.9	7.47	0.0	adding Na2CO3
14:19	88.9	8.25		rpm: 500
14:23	89.9	8.39	1006.1	finished adding Na2CO3
14:43	90.6	8.45	0.0	test complete
Average:	84.6	8.21		

Mass of Na_2CO_3(aq) Added, g	1006.10

Final Reaction Slurry

	Solution			Solids	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	2470.1	1.1404	2166.0					
Wash	274.1	1.0341	265.1					
Product				1623.5	1774.8	1750.3	151.3	126.8

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	3	
Filtration time:	<5	min
Clarity of filtrate:	CC	
Colour of filtrate:	WW	
Cake thickness:	~30	mm

Washing time:	<3	min
Clarity of wash:	CC	
Volume of wash:	~300	mL
Colour of wash:	WW	

LIMS Data

CA03064-MAY11

Sample ID	Li mg/L	Mg mg/L	Ca mg/L
RT L Fe IX T1	20800	4.28	117

CA03564-MAY11

Sample ID	Ca g/t
RT L Fe Li2CO3 T1 Pr	1480

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 24-May-11
Li_2CO_3 Precipitation
Test No. RT L Fe Li2CO3 PPT T2
Objective: To conduct Li_2CO_3 PPT followed by Bicarb Polishing

Feed IX-Product

Reagents	Li_2CO_3 (99.3%$^+$)	Rocky Ridge Water
	25 wt. % Na_2CO_3(aq)	(This solution was preheated to 45 °C)

Apparatus One 4 L reactor with lid, overhead mixer and impeller, with pH and ATC probes

Test Summary:

To produce Li2CO3 from a Spodumene sample.

Stoichiometric Excess of Na_2CO_3	**1.04**

	Required
25	wt% Na_2CO_3(aq) Sol'n
mass Na_2CO_3, g	250.9
mass RR water H_2O, g	752.6
Total Mass of 25% Na_2CO_3(aq), g	**1003.5**

Procedure

Reagent Prep.

Prepare the Na_2CO_3 solution as defined.

Mass, g	1714

Li2CO3 Precipitation Test

1 Weigh out the required amount of feed solution.
2 Add solution to the 4 L reactor. Pre heat the feed solution to ~90 - 95°C.
3 Prepare the Na2CO3 solution. Pre-heat to 45°C.
4 Measure & record the pH & temp of the feed solution.
5 Start the addition of Na2CO3(aq). Add total volume of Na2CO3(aq) over 10 minutes. Note the time when all of the solution has been added. This is T = 0 min.
6 Stir vigourously for 20 minutes.
7 After 20 minutes record final pH. Record mass of reaction slurry.
8 Vacuum filter the reaction slurry.

NOTE: Theoretically 168.2 g of Li_2CO_3 will be produced.

Product

Recover the Product Filtrate. Measure & record the mass and solution density.
Wash the Product solids with hot d.i. water (T≥ 95 °C) (ca. 3 x 150 mL).
Combine Wash fractions. Measure & Record mass and solution density of combined wash fraction.
Dry the filter cake.
Record the dry mass of the product filter cake.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
Feed	RT L Fe Li2CO3 PPT T2 Feed	Li,Mg-AAS, Ca-ICP	
Product Filtrate	RT L Fe Li2CO3 PPT T2 FIL	HOLD	
Product Wash	RT L Fe Li2CO3 PPT T2 WSH	HOLD	
PROD	RT L Fe Li2CO3 PPT T2 PROD	HOLD	

Test No. **RT L Fe Li2CO3 PPT T2**

Project Technologist: A.Ferguson
Date of Test: **24-May-11**

Initial Data

Feed Solution

Mass, g	**1714.2**
Soln Den., g/mL	1.1284
Cal'd V, mL	1519.1

	Mass, g	
Reactor	Gross	Net
tare	**2814.10**	
Initial	4528.30	**1714.20**
Final		

Reaction Time, min	Precipitation Test Data			
	Sol'n T, °C	Sol'n pH	Mass of Na2CO3(aq) Added, g	Comment
13:30	17.6	9.90		Heat On
14:03	95.6	8.06		Adding Na2CO3
14:18	87.0	8.55	1006.3	All Na2CO3 in. RPM 500
14:20	88.7	8.55		T=0
14:27	97.7	8.33		
14:40	100.0	8.56		Test complete
Average:	93.8	8.41		

Mass of Na_2CO_3(aq) Added, g	1006.30

Final Reaction Slurry

	Solution			Solids	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	2435.7	1.1414	2134.0					
Wash	346.0	1.0246	337.7					
Product				2214.5	2361.6	2341.2	147.1	126.7

Final Filtration:

Diameter of filtration paper:	150	mm
type of paper (Whatman ##):	3	
Filtration time:	<5	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	50.0	mm

Washing time:	<1	min
Clarity of wash:	cc	
Volume of wash:	338	mL
Colour of wash:	ww	

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 26-May-11
Test No.: **RT L Fe BCB T1**

Feed **RT L Fe Li2CO3 PPT T1**

Reagents CO_2(g) Rocky Ridge Distilled Water

Apparatus 3 L Reactor, overhead mixer and impeller, pH probe & meter, glass frit sparger, bubbler with frit
CO_2 regulator, small CO_2 Cylinder

Test Summary: A charge of Li_2CO_3 is converted to $LiHCO_3$ using CO_2(g).

$CO_2(g) + H_2O \longrightarrow H_2CO_3(aq)$

$Li_2CO_3(s) + H_2CO_3(aq) \longrightarrow 2\ LiHCO_3(aq)$

Procedure

A) Bicarbonate conversion

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solids therefore ensure that everything that is in contact with the reaction solutions and solids is clean.
2 Measure & record the tare of the reactor.
3 Prepare a 5.0% solids slurry of the Li_2CO_3 product in distilled water.

Solution %Solids, %	**5**	Mass of Li_2CO_3, g	**121**
		Mass of Distilled Water, g	**2299**

4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH of the stirred solution.
6 **Measure & record the initial mass of the CO_2 cylinder.**
7 Sparge the solution with CO_2(g) (~2.5 L/min)
8 Monitor pH and note any changes to the appearance of the solution.
9 Once pH stabilizes. Run for an additional 60min
10 Measure & record the final mass of the reactor and reaction slurry.
11 Vacuum filter the reaction slurry. Recover the filtrate.
12 Recover the filter cake - record wet mass and dry O/N in an oven (105 - 110 °C). When dry record the dry mass.
13 **Measure & record the final mass of the CO_2 cylinder.**

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
RT L Fe Li2CO3 PPT T1	RT L Fe BCB T1 HEAD		
Filtrate	RT L Fe BCB T1 FIL	Li-AAS	
Residue	RT L Fe BCB T1 RES		

Data

A) Bicarbonate conversion

	Mass, g
Stage PPT Prod (FEED)	121.0
Distilled Water	2299
CO_2 used	325.5

	Mass, g		
Reactor	Gross	Net	CO_2
tare	**2908.10**		
Initial	5328.10	**2420.00**	22.471
Final	5371.80	**2463.70**	**21.82**

Reaction Time, min	Sol'n T, °C	Sol'n pH	Flow CO_2(g), L/min	Comment/Observations
9:26	19.9	11.39	0	Initial Reading , RPM 500, Cloudy white
9:28	19.9	11.04	2.5	adding CO2
9:30	20.2	10.15	2.5	bubbling
9:33	20.5	9.53	3.0	
9:38	21.7	8.80	3.0	increased CO2 Flow
9:46	23.0	8.20	3.0	
9:58	23.9	7.85	3.0	
10:26	23.1	7.80	3.0	
10:53	22.2	7.78	2.5	Flow dropping, Sparger maybe plugging
11:03	21.9	7.78	2.5	
11:10	21.7	7.78	2.5	Test complete
Average/*Total*	21.8		2.8	

Reaction Slurry

	Solution			Solids	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	2455.10	1.054	2329.32					
Wash								
Residue				18.9	22.8	20.4	3.9	1.5

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	milli	
Filtration time:	<10	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	<5	mm
Temperature of Drying Oven, °C	105	

Washing time:	n/a	min
Clarity of wash:	n/a	
Volume of wash:	n/a	mL
Colour of wash:	n/a	

****Solids not seedling in reactor like normal '

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 26-May-11
Test No.: **RT L Fe BCB T1-2**

Feed **RT L Fe Li2CO3 PPT T1**

Reagents CO_2(g) Rocky Ridge Distilled Water

Apparatus 2 L Reactor, overhead mixer and impeller, pH probe & meter, glass frit sparger, bubbler with frit
CO_2 regulator, small CO_2 Cylinder

Test Summary: A charge of Li_2CO_3 is converted to $LiHCO_3$ using CO_2(g).

CO2(g) + H2O ⟶ H_2CO_3(aq)

Li_2CO_3(s) + H_2CO_3(aq) ⟶ 2 LiHCO3(aq)

Procedure

A) Bicarbonate conversion

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solids therefore ensure that everything that is in contact with the reaction solutions and solids is clean.
2 Measure & record the tare of the reactor.
3 Prepare a 5.0% solids slurry of the Li_2CO_3 product in distilled water.

Solution %Solids, %	**5**	Mass of Li_2CO_3, g	**122.8**
		Mass of Distilled Water, g	**2333**

4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH of the stirred solution.
6 **Measure & record the initial mass of the CO_2 cylinder.**
7 Sparge the solution with CO_2(g) (~2.5 L/min)
8 Monitor pH and note any changes to the appearance of the solution.
9 Once pH stabilizes. Run for an additional 60min
10 Measure & record the final mass of the reactor and reaction slurry.
11 Vacuum filter the reaction slurry. Recover the filtrate.
12 Recover the filter cake - record wet mass and dry O/N in an oven (105 - 110 °C). When dry record the dry mass.
13 **Measure & record the final mass of the CO_2 cylinder.**

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
RT L Fe Li2CO3 PPT T1	RT L Fe BCB T1-2 HEAD		
Filtrate	RT L Fe BCB T1-2 FIL	Li - AAS	
Residue	RT L Fe BCB T1-2 RES		

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **RT L Fe BCB T1-2**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Data

A) Bicarbonate conversion

	Mass, g
Stage PPT Prod (FEED)	122.8
Distilled Water	2333
CO_2 used	325.5

	Mass, g		
Reactor	Gross	Net	CO_2
tare	**2816.30**		
Initial	5272.00	**2455.70**	22.471
Final	5336.60	**2520.30**	

Reaction Time, min	Sol'n T, °C	Sol'n pH	Flow CO_2(g), L/min	ORP	Comment/Observations
9:42	19.6	10.83	0		CO_2 ON
9:43	19.9	9.87	3.0		
9:44	20.2	9.54	3.0		
9:47	21.0	8.88	3.0		
9:50	21.7	8.51	3.0		
9:52	22.2	8.26	3.0		
10:30	23.1	7.62	3.0		
11:00	22.2	7.61	3.0		
11:37	21.1	7.60	3.0		
11:50	20.8	7.60	3.0		Test complete
Average/*Total*	21.4		3.0		

Reaction Slurry

	Solution			**Solids**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	2507.30	1.0552	2376.14					
Wash								
Residue				19.0	21.6	20.2	2.6	1.2

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	milli	
Filtration time:	<10	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	<5	mm
Temperature of Drying Oven, °C	105	

Washing time:	N/A	min
Clarity of wash:	N/A	
Volume of wash:	N/A	mL
Colour of wash:	N/A	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **BCB DC RT L Fe T1**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Feed **RT L Fe BCB T1 FIL**

Reagents Rocky Ridge Distilled Water

Apparatus 3 L reactor flask with lid, overhead mixer and impeller, heating mantle, YSI controller, T probe

Test Summary: A charge of LiHCO3 is heated to ~95°C to decompose the bicarbonate to the carbonate. The solids are recovered and washed with distilled water at 95 °C.

Procedure

Bicarbonate decomposition - Li_2CO_3 production

1 Add the blance of the bicarbonate solution to the 2 L reactor. Measure the initial pH of the solution.
3 Heat the solution to ~ 95 °C. Note the changes in the solution.
3 At the end of the reaction recover the solids as per procedure below.

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate and record its mass and solution density. When cool measure final pH.
Collect a sample of the filtrate for analysis .
Conduct 2 displacement washes (~75 mL Rocky H2O /wash @ ambient T).
Collect Wash fraction as one - record mass and solution density.
Remove the filter paper from the wet cake. Dry the solids in a clean glass container.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
PLS Head	BCB DC RT L Fe T1 HEAD		
Filtrate	BCB DC RT L Fe T1 FIL		
CMB WSH	BCB DC RT L Fe T1 CMB WSH		
Solids	BCB DC RT L Fe T1 PROD	ANA2010-0475	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **BCB DC RT L Fe T1**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Data
B) Bicarbonate decomposition - Li_2CO_3 production

Project Technologist: **A.Ferguson**
Date of Test: **26-May-11**

	Mass, g
RT L Fe BCB T1 FIL	5384.90
Solution Den., g/mL	1.054
Li_2CO_3 Seed, g	--

	Mass, g	
Reactor	Gross	Net
tare	**2947.50**	
Initial	5384.90	**2437.40**
Final	5322.10	**2374.60**

Reaction Time, min	Sol'n T, °C	Sol'n pH	Comment/Observations
12:02	21.3	7.81	Heat On
13:02	85.3	7.58	Lots of bubbling
13:05	90.0	7.63	T=0
14:06	95.8	7.92	
14:21	95.8	7.96	
14:30	93.9	7.99	Test complete

	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
FILTRATE	2169.7	1.0132	2141.4					
CMB Wash	512.8	1.0071	509.2					
Solids				529.1	637.8	611.2	108.7	82.10

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	3	
Filtration time:	<3	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	40.0	mm
Temperature of Drying Oven, °C	105	

Washing time:	<1	min
Clarity of wash:	cc	
Volume of wash:	509	mL
Colour of wash:	ww	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **BCB DC RT L Fe T1-2**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Feed **RT L Fe BCB T1 FIL**

Reagents Rocky Ridge Distilled Water

Apparatus 3 L reactor flask with lid, overhead mixer and impeller, heating mantle, YSI controller, T probe

Test Summary: A charge of LiHCO3 is heated to ~95°C to decompose the bicarbonate to the carbonate. The solids are recovered and washed with distilled water at 95 °C.

Procedure

Bicarbonate decomposition - Li_2CO_3 production

1 Add the blance of the bicarbonate solution to the 2 L reactor. Measure the initial pH of the solution.
3 Heat the solution to ~ 95 °C. Note the changes in the solution.
3 At the end of the reaction recover the solids as per procedure below.

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate and record its mass and solution density. When cool measure final pH.
Collect a sample of the filtrate for analysis .
Conduct 2 displacement washes (~75 mL Rocky H2O /wash @ ambient T).
Collect Wash fraction as one - record mass and solution density.
Remove the filter paper from the wet cake. Dry the solids in a clean glass container.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
PLS Head	BCB DC RT L Fe T1-2 HEAD		
Filtrate	BCB DC RT L Fe T1-2 FIL		
CMB WSH	BCB DC RT L Fe T1-2 CMB WSH		
Solids	BCB DC RT L Fe T1-2 PROD	ANA2010-0475	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **BCB DC RT L Fe T1-2**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Data
B) Bicarbonate decomposition - Li_2CO_3 production

Project Technologist: **A.Ferguson**
Date of Test: **26-May-11**

	Mass, g
RT L Fe BCB T1 FIL	2471.30
Solution Den., g/mL	1.0552
Li_2CO_3 Seed, g	

	Mass, g	
Reactor	Gross	Net
tare	**2873.30**	
Initial	5344.60	**2471.30**
Final	5276.90	**2403.60**

Reaction Time, min	Sol'n T, °C	Sol'n pH	Comment/Observations
12:34	25.1	7.92	Started initial reading, Heat ON, RPM = 450
13:06	68.9	8.07	Bubbles from solution
13:22	90.0	8.36	T=0
13:55	94.5	8.66	T=33
14:20	93.8	8.70	T=58
14:52	95.1	8.73	T=90 End Test

	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
FILTRATE	2255.3	1.0136	2225.0					
CMB Wash	610.1	1.0067	606.0					
Solids				1628.4	1732.1	1710.6	103.7	82.20

Final Filtration:

Diameter of filtration paper:	125	mm			
type of paper (Whatman ##):	3		Washing time:	<1	min
Filtration time:	<5	min	Clarity of wash:	CC	
Clarity of filtrate:	CC		Volume of wash:	606	mL
Colour of filtrate:	White **		Colour of wash:	WW	
Cake thickness:	50.0	mm			
Temperature of Drying Oven, °C	105				

****Note: pH meter cracked some of the blue fluid leaked into the rxn, added extra wash to remove contaminiants**

	Spec	Rock Tech Lithium - First Product		
		Sample 1	Sample 2	**Average**
Na	< 400 ppm	<20	<20	**<20**
Sulphur (S)	< 200 ppm	100	200	**150**
Chlorides (Cl)	< 100 ppm	<10	<10	**<10**
Ca	< 100 ppm	210	200	**205**
Mg	< 100 ppm	15.2	17	**16**
K	< 50 ppm	<20	<20	**<20**
B	< 10 ppm	< 4	< 4	**< 4**
Fe	< 5 ppm	<2	<2	**<2**
Cr	< 5 ppm	< 1	< 1	**< 1**
Ni	< 5 ppm	< 5	< 5	**< 5**
Cu	< 5 ppm	1	1	**1**
Pb	< 5 ppm	< 1	< 1	**< 1**
Al	< 5 ppm	<8	<8	**<8**
Zn	< 5 ppm	< 5	< 5	**< 5**
Mn	< 5 ppm	0.4	0.4	**0.4**
Li_2CO_3 Grade, %	> 99.5%	99.9673	99.9582	**99.9628**

Note: Li_2CO_3 grade determined by difference

	Assay value not on spec

Appendix D – Bulk Li_2CO_3 Sample

Test Details and Data

Project: CALR-12607-002 **Project Tech.:** D.Lustig
Client: Rock Tech Lithium **Test Date:** 13-Jun-11

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT L Fe CL T1** **Project Manager:** Stephen Mackie (ext.2058)

Feed/Head: **Acid Roasted β-Spodumene - Variable Acid Dose Tests**
(%Excess Acid of Acid Bake = 30 %)

Test Objective: To conduct a concentrate leach on AR β-Spod that was produced in Muffle furnace.

Test Summary: The feed sample is subjected to a concentrate leach using deioinized water for 60 minutes.

Reagents:

Conc.	Chemical(s)
100%	Deionized Water (Rocky Ridge)

**review all pertanant MSDS prior to handling reagents and ALWAYS wear proper PPE.*

Apparatus:

Qty.	Equipment
1x	~6kg balance, x.xxg ***NOTE: all glassware acid washed before use.***
various	120mL containers
various	1L containers
1x	1 L reactor
1x	Caframo mixer
1x	15cm Buchner filter funnel
1x	glass drying pan
1x	teflon impeller
1x	pH meter w/ATC
1x	Whatman No. 1 filter paper, 12.5cm
1x	500 mL vacuum flask
1x	spatula

****various connectors, fittings, clamps, tools etc.***

Procedure:

1. Measure out equal charges of the AR β-Spod and deionized water.
2. Combine the solids and water in a reactor. Stir for 60 minutes.
3. Measure & record initial and final solution **mass, pH and ORP**.
4. Filter sample through buchner filter. Recover the filtrate. Measure & record mass and solution density of filtrate.
5. Conduct three displacement washes of the filter cake. **Recover the Wash 1 fraction separately.**
6. Measure & record mass and solution density of WSH 1 and combined WSH 2&3 fractions.
7. Recover the Final RES, record the NET wet & dry mass, dry in oven, submit to bucking room.
8. Perform analyses as summarized below. Keep all process streams until assays have been reviewed.

NOTE: After the assay data have been reviewed and confirmed please make composite samples of the following test process streams:

Test Series	**RT L Fe CL**	Composite Sample		Comment
Process Stream		Required physical data	ANA	
All PLS and WSH 1 solutions		Mass, Soln Den	Li, ICP, Si, S	Feed Soln for PIR Tests
All RES		Mass	Li, ICP, WRA, S, XRD, s.g.	keep ~ 1 kg of COMP sample and dispose of the balance
All WSH 2& 3		Mass, Soln Den	Li	Dispose once Assay reviewed & confirmed

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
AR β-Spod Head	RT L Fe CL T1 Head	Li-AAS, ICP-OES	***CA02300-JUN11***
Final Filtrate	RT L Fe CL T1 Final PLS	Li, Al, Fe-AAS	***CA02790-JUN11***
Disp. Wash 1	RT L Fe CL T1 WSH 1	Li-AAS	***CA02790-JUN11***
Disp. Wash 2, & 3	RT L Fe CL T1 WSH 2&3	HOLD	***CA03290-JUN11***
Final RES	RT L Fe CL T1 RES	Li, Al, Fe-AAS	***CA02851-JUN11***

Project: **CALR-12607-002** **Project Tech.:** **D.Lustig**
Client: **Rock Tech Lithium** **Test Date:** **13-Jun-11**

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT L Fe CL** **Project Engineer:** Stephen Mackie (ext.2058)

Test Data:

	Mass, g		
	tare	**Initial**	**Final**
Reactor	973	8259	8229
di H_2O		3648	
Solids		3640	

Test Notes:

TIME	NOTES					
	pH	**Temp.(°C)**	**ORP (mV)**	**RPM**	**Ref. START time:**	14:14
8:07:00	2.04	25.5	550	240		
9:07:00	1.86	22.6	535	250	Increased RPM 240 -> 250RPM @ 8:14	
1:00:00	**:Elapsed Time**					

Average	**1.95**	**24.05**	**542.5**

Comments:
Wash # 1 = 2500 mL
I stopped the PLS filtration at 20 mins (the samples was dripping very slowly) but for the second wash I allowed twice as much time to filter
in order to recover Li

Sample Data:

Final Reaction Slurry

	Solution			Solid	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
RT L Fe CL T1 Final PLS	2121.80	1.1641	1823					
RT L Fe CL T1 WSH 1	3398.70	1.1076	3069					
RT L Fe CL T1 WSH 2&3	4567.90	1.009	4527					
RT L Fe CL T1 RES				3817.7	7532.6	6571.8	3714.9	2754.1

Final Filtration:

Diameter of filtration paper:	1185	mm			
type of paper (Whatman ##):	3		Washing time:	60	min
Filtration time:	30<	min	Clarity of wash:	Clear	
Clarity of filtrate:	Little Bit Cloudy		Volume of wash:	2500	mL
Colour of filtrate:	Yellowish Green		Colour of wash:	Yellowish	
Cake thickness:	130.0	mm			

Test No.:		RT L Fe CL														
Test Parameters		T, °C	pH	ORP, mV												
		24	1.95	543												
		Assays, % or mg/L														
	mass or vol, g or mL	Li	Si	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn	S
RT L Fe CL T1 Head	3640	2.35	22.76	11.40	0.57	0.30	0.01	0.46	0.03	0.13	0.31	0.12	0.00	0.00	0.01	7.10
di H2O	3648															
RT L Fe CL T1 Final PLS	1823	24500	84.50	2420.00	731.00	557.00	6.30	341.00	18.60	164.00	1790.00	352.00	1.01	0.20	6.20	64500
RT L Fe CL T1 WSH 1	3069	7770			428.00											
RT L Fe CL T1 WSH 2&3	4527	723														
RT L Fe CL T1 RES	2754	0.08	30.29	14.08	0.57	0.18	0.01	0.65	0.06	0.14	0.16	0.11	0.01	0.01	0.01	0.31
IN (Elemental Masses)																
RT L Fe CL T1 Head		85.54	828.61	414.96	20.64	10.99	0.40	16.89	1.18	4.77	11.21	4.44	0.00	0.01	0.22	258.44
di H2O																
Total IN, g		**85.54**	**828.61**	**414.96**	**20.64**	**10.99**	**0.40**	**16.89**	**1.18**	**4.77**	**11.21**	**4.44**	**0.00**	**0.01**	**0.22**	**258.44**
OUT (Elemental Masses)																
RT L Fe CL T1 Final PLS		44.66	0.15	4.41	1.33	1.02	0.01	0.62	0.03	0.30	3.26	0.64	0.00	0.00	0.01	117.56
RT L Fe CL T1 WSH 1		23.84			1.31											
RT L Fe CL T1 WSH 2&3		3.27														
RT L Fe CL T1 RES		2.176	834.211	387.72	15.60	4.921	0.188	17.833	1.661	3.839	4.291	0.000	0.165	0.154	0.187	8.538
Total OUT, g		**73.95**	**834.36**	**392.14**	**18.25**	**5.94**	**0.20**	**18.45**	**1.69**	**4.14**	**7.55**	**0.64**	**0.17**	**0.15**	**0.20**	**126.10**
Relative Distribution																
RT L Fe CL T1 Final PLS		60.4	0.0	1.1	7.3	17.1	5.7	3.4	2.0	7.2	43.2	100.0	1.1	0.2	5.7	93.2
RT L Fe CL T1 WSH 1		32.2			7.2											
RT L Fe CL T1 WSH 2&3		4.4														
RT L Fe CL T1 RES		2.9	100.0	98.9	85.5	82.9	94.3	96.6	98.0	92.8	56.8	0.0	98.9	99.8	94.3	6.8
Total, %		**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**
%OUT/IN		86.4	100.7	94.5	88.4	54.0	49.9	109.3	144.1	86.8	67.4	14.4		2124.2	90.9	48.8
% Extraction, %		97.1	0.0	1.1	14.5	17.1	5.7	3.4	2.0	7.2	43.2	100.0	1.1	0.2	5.7	93.2
Cal'd Head		2.032	22.922	10.773	0.501	0.163	0.005	0.507	0.047	0.114	0.208	0.018	0.005	0.004	0.005	3.464
% Mass Reduction, %	24.34															

Project No.: CALR-12607-002 **Project Technologist:** A. Ferguson (2290)
Client: Rock Tech Lithium **Date of Test:** 15-Jun-11

Test: **RT LF PIR T1 Bulk Test**
Feed Feed Solution = Combine the filtrate and WSH-1 fraction of Low Fe Con Concentrate Leach
Lime 20 wt%
Apparatus Bulk reactor flask with lid, overhead mixer and impeller, condenser, pH/ORP meter and pH and ORP probes Air Sparging line with in-line flow meter, heating mantle and YSI controller

Test Summary:

To modify the pH of the Feed solution to a specific target within the pH isotherm.

Procedure Feed Solution Prep
Combine the WL filtrates and Wash 1 fractions as discussed. Stir for ~10 minutes.
Measure & record mass and solution density of the resulting solution.
reab a sample of the Feed Sol'n for assay.

Feed Soln	Mass, g	Sol'n Den., g/L	V, mL
	5340.10	1129.1	4729.5

Please take a picture of the set-up and process slurries/solutions

Bulk Reactor Set-up
Assemble the reactor with the impeller, lid (ports with septa).(10L pail)
Record the mass of the assembled reactor (this will be the reactor tare).
Standardize pH and ORP probes.

Lime slurry Preparation

Dry Lime Requirement, g/L Feed Sol'n				14.8
Vol of Feed Sol'n, mL				4729.52
Approximate mass of Dry Lime required, g				70.0
20% Lime Slurry	Theo.	Actual	4 Portions	
Mass of dry Lime, g	87.5	87.52	21.88	
Mass of DI H_2O, g	349.8	349.8	87.45	

Check
20

Prepare a 20% solids slurry of lime based on the above data. Use Rocky Ridge di Water to prepare the lime slurry.

PIR Test - Air Sparging & pH Modification
Add the Full amount of the combined solution to the tared reactor. Measure & record mass.
Maintain the reaction solution temperature above 15 °C.
Record pH and ORP of the solution.
Start sparging the solution with air and then immediately start pH modification. **Reaction Time = 0.**
Use the Lime slurry to modify the pH of the solution to the specified target values.
Add the lime in ~ 4 portions slowly. Measure the mass of each lime addition charge.
Monitor & record the reaction solution pH & ORP every 5 minutes.
As the reaction slurry pH approaches the target reduce the size of lime slurry charges are added.
pH Target: **5.8**

Sample Preparation

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate; measure & record the mass and soltion density of the filtrate.
Collect a sample of the filtrate for analysis .
Conduct three displaccement washes of the filter cake (deionized water). Use some of the wash water to rinse out the reactor. Combine wash fractions.
Measure & record the mass and solution density of the combined Wash solution.
Collect the filter cake. Measure & record wet mass. Dry the filter cake. Measure & record the dry solids mass.

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
Feed	RT LF PIR T1 FEED	Li, ICP	CA02855-JUN11
FINAL FIL	RT LF PIR T1 FIL	Li, ICP	CA02855-JUN11
WASH	RT LF PIR T1 WSH	Li	CA02855-JUN11
Final RES	RT LF PIR T1 RES	Li, ICP	CA03550-JUN11

Test: **RT LF PIR T1**

Data

Combined Feed Solution

Weight (g):	5340
SG (g/mL):	1.1291
Volume (mL):	4730

	Mass, g	
Reactor	Gross	Net
tare	**604.30**	
Initial	5944.40	**5340.10**
Final	6272.70	**5668.40**

Reaction Time, min	Air Flow, L/min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of lime slurry added, g	Comment/Observations
<0	0.00	19.2	1.64	541	0.00	Before sparging. RPM
0	4.00	19.2	1.59	539	0	**Impeller Type: pitch 4 blade**
3	4.00	21.4	2.15	532	108.2	adding LS
5	4.00	21.4	2.15	531	0.00	
7	4.00	22.8	3.94	439	108.30	tan colour
10	4.00	22.8	3.99	435	0	
13	4.00	23.4	4.32	415	76.60	tan colour
15	4.00	23.4	4.35	428	0.00	
17	4.00	23.6	4.81	425	31.80	
20	4.00	23.6	5.00	433	0.00	
25	4.00	23.5	5.68	355	5.70	
30	4.00	23.4	5.77	349	0.80	
35	4.00	23.3	5.80	346	0.20	
40	4.00	23.1	5.81	345	0.00	Test complete

Average Temperature (°C): 22.68 | Total Lime Added (g): 331.60 | dry lime (g) 66.32

lime slurry was 20%w/w made from 92%CaO

Final reaction Slurry

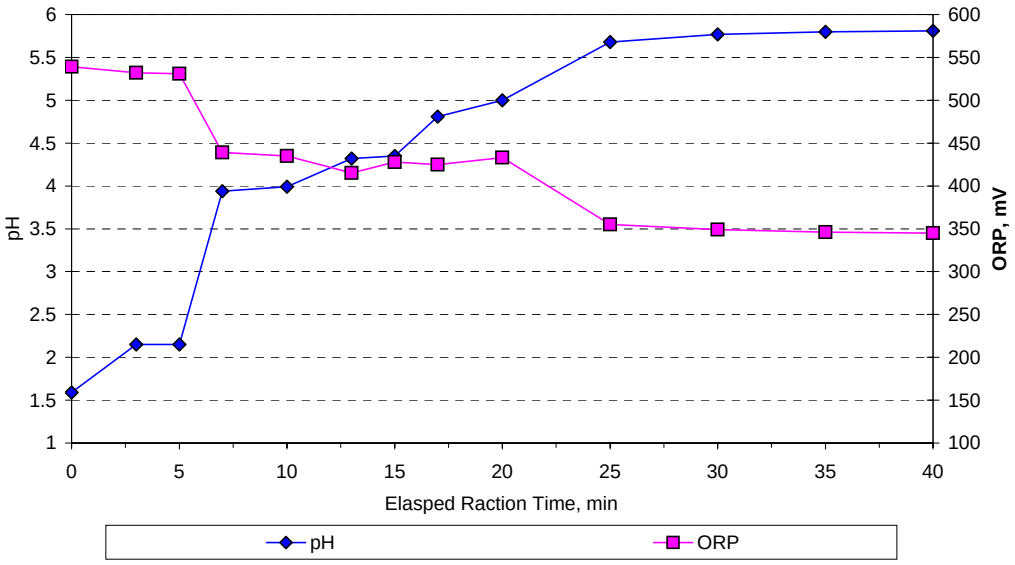
	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, L	Tare	Wet	Dry	Wet	Dry
Filtrate	4845.60	1.1077	4374					
Wash	1696.90	1.0367	1637					
Residue				1624	2199.8	1786.8	575.8	162.8
				Filter Cake %moisture, %			Reaction Slurry %solids, %	
				71.7			2.9	

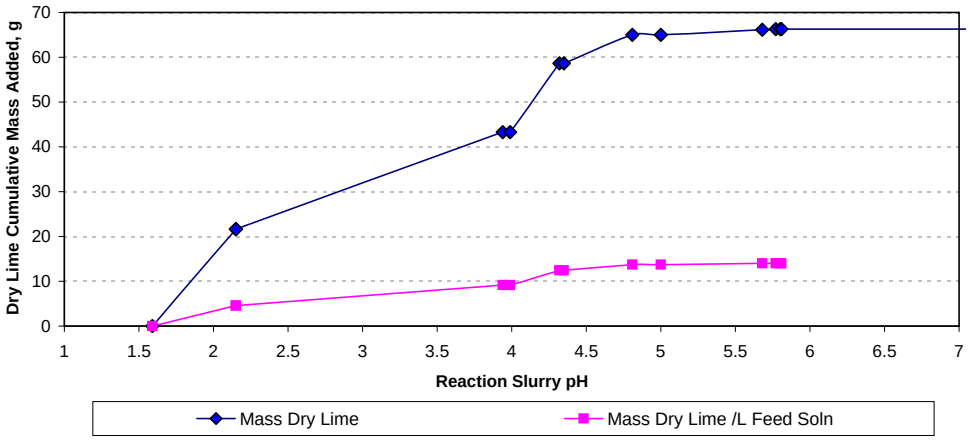
Final Filtration:

Diameter of filtration paper:	185	mm		
type of paper (Whatman ##):	3		Washing time:	30 min
Filtration time:	<40	min	Clarity of wash:	cc
Clarity of filtrate:	cc		Volume of wash:	1637 mL
Colour of filtrate:	ww		Colour of wash:	ww
Cake thickness:	40.0	mm		

Test No.:	RT LF PIR T1													
Test Parameters		**T, °C**	**Final pH**	**Final ORP**										
		23	5.8	345										
					Assays, % or mg/L									
	mass or vol, g or mL	**Li**	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn
RT LF PIR T1 FEED	4730	18500	2390	541	480	6	297	28	143	1760	325	0.9	< 0.2	5.7
CaO	66		0.05	0.03	52			0.36						
RT LF PIR T1 FIL	4374	17500	< 0.8	< 0.2	461	< 0.1	233	48.6	97	1220	< 5	< 0.02	< 0.2	< 0.7
RT LF PIR T1 WSH	1637	5780												
RT LF PIR T1 RES	163	0.077	5.4	1.7	20	0.02	0.01	0.03	0.07	0.07	1	0.00	0.00	0.02
IN - Elemental Masses, g														
RT LF PIR T1 FEED		87.50	11.30	2.56	2.27	0.03	1.40	0.13	0.68	8.32	1.54	0.00	0.00	0.03
CaO		0.00	0.03	0.02	34.60	0.00	0.00	0.24	0.00	0.00	0.00	0.00	0.00	0.00
Total IN, g		**87.50**	**11.34**	**2.58**	**36.87**	**0.03**	**1.40**	**0.37**	**0.68**	**8.32**	**1.54**	**0.00**	**0.00**	**0.03**
OUT - Elemental Masses, g														
RT LF PIR T1 FIL		76.55	0.00	0.00	2.02	0.00	1.02	0.21	0.43	5.34	0.00	0.00	0.00	0.00
RT LF PIR T1 WSH		9.46												
RT LF PIR T1 RES		0.125	8.807	2.719	32.723	0.026	0.018	0.042	0.119	0.116	1.231	0.007	0.001	0.024
Total OUT, g		**86.14**	**8.81**	**2.72**	**34.74**	**0.03**	**1.04**	**0.25**	**0.54**	**5.45**	**1.23**	**0.01**	**0.00**	**0.02**
Relative Distribution														
RT LF PIR T1 FIL		99.9	0.0	0.0	5.8	0.0	98.2	83.4	78.2	97.9	0.0	0.0	0.0	0.0
Residue		0.1	100.0	100.0	94.2	100.0	1.8	16.6	21.8	2.1	100.0	100.0	100.0	100.0
Total, %		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
%OUT/IN		98.4	77.7	105.4	94.2	96.0	73.9	68.3	80.5	65.5	80.1	166.6		90.6
% Precipitation, %		0.15	100.0	100.0	94.2	100.0	1.8	16.6	21.8	2.1	100.0	100.0	100.0	100.0
Cal'd Head		18213	1862	575	7345	5	219	54	115	1153	260	1	0	5

Project No.: CALR-12607-002
Client: Rock Tech Lithium





Project: CALR-12607-002 **Project Tech.:** D.Lustig (2290)
Client: Rock Tech Lithium **Test Date:** 13-Jun-11

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT H Fe CL T1** **Project Manager:** Stephen Mackie (ext.2058)

Feed/Head: **Acid Roasted β-Spodumene - Variable Acid Dose Tests**
(%Excess Acid of Acid Bake = 30 %)

Test Objective: To conduct a concentrate leach on AR β-Spod that was produced in Muffle furnace.

Test Summary: The feed sample is subjected to a concentrate leach using deioinized water for 60 minutes.

Reagents:

Conc.	Chemical(s)
100%	Deionized Water (Rocky Ridge)

**review all pertanant MSDS prior to handling reagents and ALWAYS wear proper PPE.*

Apparatus:

Qty.	Equipment
1x	~6kg balance, x.xxg ***NOTE: all glassware acid washed before use.***
various	120mL containers
various	1L containers
1x	1 L reactor
1x	Caframo mixer
1x	15cm Buchner filter funnel
1x	glass drying pan
1x	teflon impeller
1x	pH meter w/ATC
1x	Whatman No. 1 filter paper, 12.5cm
1x	500 mL vacuum flask
1x	spatula

****various connectors, fittings, clamps, tools etc.***

Procedure:

1. Measure out equal charges of the AR β-Spod and deionized water.
2. Combine the solids and water in a reactor. Stir for 60 minutes.
3. Measure & record initial and final solution **mass, pH and ORP**.
4. Filter sample through buchner filter. Recover the filtrate. Measure & record mass and solution density of filtrate.
5. Conduct three displacement washes of the filter cake. **Recover the Wash 1 fraction separately.**
6. Measure & record mass and solution density of WSH 1 and combined WSH 2&3 fractions.
7. Recover the Final RES, record the NET wet & dry mass, dry in oven, submit to bucking room.
8. Perform analyses as summarized below. Keep all process streams until assays have been reviewed.

NOTE: After the assay data have been reviewed and confirmed please make composite samples of the following test process streams:

Test Series	**RT H Fe CL**	Composite Sample		Comment
Process Stream		Required physical data	ANA	
All PLS and WSH 1 solutions		Mass, Soln Den	Li, ICP, Si, S	Feed Soln for PIR Tests
All RES		Mass	Li, ICP, WRA, S, XRD, s.g.	keep ~ 1 kg of COMP sample and dispose of the balance
All WSH 2& 3		Mass, Soln Den	Li	Dispose once Assay reviewed & confirmed

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
AR β-Spod Head	RT H Fe CL T1 Head	~~Li-AAS, ICP-OES~~	***CA02344-JUN11***
Final Filtrate	RT H Fe CL T1 Final PLS	Li, Al, Fe-AAS	***CA02788-JUN11***
Disp. Wash 1	RT H Fe CL T1 WSH 1	Li-AAS	***CA02788-JUN11***
Disp. Wash 2, & 3	RT H Fe CL T1 WSH 2&3	HOLD	***CA03290-JUN11***
Final RES	RT H Fe CL T1 RES	Li, Al, Fe-AAS	***CA02852-JUN11***

Project: **CALR-12607-002** **Project Tech.: D.Lustig (2290)**
Client: **Rock Tech Lithium** **Test Date: 13-Jun-11**

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT H Fe CL** **Project Engineer:** Stephen Mackie (ext.2058)

Test Data:

	Mass, g		
	tare	**Initial**	**Final**
Reactor	973	9962	9929
di H_2O		4503	
Solids		4495	

Test Notes:

	pH	**Temp.(°C)**	**ORP (mV)**	**RPM**	**Ref. START time:**	14:14
10:33:00	2.07	27.1	495	290		
11:33:00	2.02	24	445	285		
1:00:00	**:Elapsed Time**					

Average	**2.045**	**25.55**	**470**

Comments:
used tap DI for washes since there was no Rocky Ridge Water left
split sample into two purex dishes

Sample Data:

Final Reaction Slurry

	Solution			Solid	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
RT H Fe CL T1 Final PLS	3143.60	1.1564	2718.44					
RT H Fe CL T1 WSH 1	2295.30	1.1316	2028.37					
RT H Fe CL T1 WSH 2&3	5939.50	1.02	5823.04					
RT H Fe CL T1 RES				3819.3	9463.5	7331.2	5644.2	3511.9

Final Filtration:

Diameter of filtration paper:	185	mm
type of paper (Whatman ##):	3	
Filtration time:	<120	min
Clarity of filtrate:	Little bit cloudy	
Colour of filtrate:	Dark Green	
Cake thickness:	145.0	mm

Washing time:	55	min
Clarity of wash:	clear	
Volume of wash:	2000	mL
Colour of wash:	Dark Green (but lighter then the PLS)	

Test No.:		#REF!														
Test Parameters		**T, °C**	**pH**	**ORP**												
		26	2.05	470												
		Assays, % or mg/L														
	mass or vol, g or mL	**Li**	Si	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn	S
RT H Fe CL T1 Head	4495	2.20	23.56	11.70	0.81	0.38	0.04	0.74	0.03	0.15	0.54	0.11	0.01	0.00	0.01	6.78
DI H2O	4503															
RT H Fe CL T1 Final PLS	2718	22400	63	2630	1280	548	75	267	21	206	2660	268	1	0	7	6640
RT H Fe CL T1 WSH 1	2028	17100														
RT H Fe CL T1 WSH 2&3	5823	2190														
RT H Fe CL T1 RES	3512	0.09	30.6	13.5	0.8	0.2	0.1	0.9	0.1	0.2	0.3	0.1	0.01	0.01	0.01	0.2
IN (Elemental Masses)																
RT H Fe CL T1 Head		98.89	1058.96	525.92	36.27	17.17	1.65	33.08	1.51	6.52	24.32	5.08	0.37	0.01	0.30	304.76
DI H2O																
Total IN, g		**98.89**	**1058.96**	**525.92**	**36.27**	**17.17**	**1.65**	**33.08**	**1.51**	**6.52**	**24.32**	**5.08**	**0.37**	**0.01**	**0.30**	**304.76**
OUT (Elemental Masses)																
RT H Fe CL T1 Final PLS		60.89	0.17	7.15	3.48	1.49	0.20	0.73	0.06	0.56	7.23	0.73	0.00	0.00	0.02	18.05
RT H Fe CL T1 WSH 1		34.69	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
RT H Fe CL T1 WSH 2&3		12.75	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
RT H Fe CL T1 RES		3.20	1075.24	475.82	27.27	8.03	2.40	30.32	1.91	5.98	11.46	4.14	0.21	0.00	0.25	6.32
Total OUT, g		**111.53**	**1075.41**	**482.97**	**30.74**	**9.52**	**2.61**	**31.05**	**1.96**	**6.54**	**18.69**	**4.87**	**0.21**	**0.00**	**0.26**	**24.37**
Relative Distribution																
RT H Fe CL T1 Final PLS		54.6	0.0	1.5	11.3	15.6	7.8	2.3	2.9	8.6	38.7	15.0	1.0	100.0	7.1	74.1
RT H Fe CL T1 WSH 1		31.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RT H Fe CL T1 WSH 2&3		11.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RT H Fe CL T1 RES		2.9	100.0	98.5	88.7	84.4	92.2	97.7	97.1	91.4	61.3	85.0	99.0	0.0	92.9	25.9
Total, %		**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**
%OUT/IN		112.8	101.6	91.8	84.8	55.5	157.9	93.8	130.4	100.4	76.9	95.8	56.7	12.1	87.9	8.0
% Extraction, %		97.1	0.0	1.5	11.3	15.6	7.8	2.3	2.9	8.6	38.7	15.0	1.0	100.0	7.1	74.1

Cal'd Head		2.481	23.925	10.745	0.684	0.212	0.058	0.691	0.044	0.146	0.416	0.108	0.005	0.000	0.006	0.542
% Mass Reduction, %	**21.87**															

Project No.: CALR-12607-002 **Project Technologist:** D.Lustig (2290)
Client: Rock Tech Lithium **Date of Test:** 15-Jun-11

Test: **RT HF PIR T1 Bulk**
Feed Feed Solution = Combine the filtrate and WSH-1 fraction of High Fe Con Concentrate Leach
Lime 20 wt%
Apparatus Bulk reactor flask with lid, overhead mixer and impeller, condenser, pH/ORP meter and pH and ORP probes Air Sparging line with in-line flow meter, heating mantle and YSI controller

Test Summary:

To modify the pH of the Feed solution to a specific target within the pH isotherm.

Procedure Feed Solution Prep

Combine the WL filtrates and Wash 1 fractions as discussed. Stir for ~10 minutes.
Measure & record mass and solution density of the resulting solution.
reab a sample of the Feed Sol'n for assay.

Feed Soln	Mass, g	Sol'n Den., g/L	V, mL
	5287.60	1148.6	4603.5

Please take a picture of the set-up and process slurries/solutions

Bulk Reactor Set-up

Assemble the reactor with the impeller, lid (ports with septa).(10L pail)
Record the mass of the assembled reactor (this will be the reactor tare).
Standardize pH and ORP probes.

Lime slurry Preparation

Dry Lime Requirement, g/L Feed Sol'n			**20.0**	**This is an estimate.**
Vol of Feed Sol'n, mL			4603.52	
Approximate mass of Dry Lime required, g			92.1	
20% Lime Slurry	Theo.	Actual	4 Portions	
Mass of dry Lime, g	115.1		28.77	
Mass of DI H_2O, g	460.4		115.09	

Check
20

Prepare a 20% solids slurry of lime based on the above data. Use Rocky Ridge di Water to prepare the lime slurry.

PIR Test - Air Sparging & pH Modification

Add the Full amount of the combined solution to the tared reactor. Measure & record mass.
Maintain the reaction solution temperature above 15 °C.
Record pH and ORP of the solution.
Start sparging the solution with air and then immediately start pH modification. **Reaction Time = 0.**
Use the Lime slurry to modify the pH of the solution to the specified target values.
Add the lime in ~ 4 portions slowly. Measure the mass of each lime addition charge.
Monitor & record the reaction solution pH & ORP every 5 minutes.
As the reaction slurry pH approaches the target reduce the size of lime slurry charges are added.
pH Target: **5.8**

Sample Preparation

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate; measure & record the mass and soltion density of the filtrate.
Collect a sample of the filtrate for analysis .
Conduct three displaccement washes of the filter cake (deionized water). Use some of the wash water to rinse out the reactor. Combine wash fractions.
Measure & record the mass and solution density of the combined Wash solution.
Collect the filter cake. Measure & record wet mass. Dry the filter cake. Measure & record the dry solids mass.

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
Feed	RT HF PIR T1 FEED	Li, ICP	CA02943-JUN11
FINAL FIL	RT HF PIR T1 FIL	Li, ICP	CA02943-JUN11
WASH	RT HF PIR T1 WSH	Li	CA02943-JUN11
Final RES	RT HF PIR T1 RES	Li, ICP	CA03092-JUN11

Project No.: CALR-12607-002 **Project Technologist:** D.Lustig (2290)
Client: Rock Tech Lithium **Date of Test:** 15-Jun-11

Test: **RT HF PIR** **T1** **Bulk**

Test: **RT HF PIR** **T1**

Data

Combined Feed Solution

Weight (g):	5288
SG (g/mL):	1.1485
Volume (mL):	4604

	Mass, g	
Reactor	Gross	Net
tare	**604.30**	
Initial	5892.40	**5288.10**
Final	6129.80	**5525.50**

Reaction Time, min	Air Flow, L/min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of lime slurry added, g	Comment/Observations	
<0	2.50	19.1	1.74	448	0.00	Before sparging. RPM	110
0:00	2.50	19.2	1.72	449	0	**Impeller Type:**	**pitch 4 blade**
0:03	2.50	21.5	2.50	413	143.2		
0:05	2.50	21.8	3.42	377	0		
0:07	2.50	22.1	3.70	351	44.70		
0:09	2.50	22.2	3.82	339	0.00		
0:12	2.50	22.5	3.94	318	76.1		
0:15	4.00	22.8	4.04	299	24.20		
0:20	4.00	23.2	4.23	270	59.40		
0:23	4.00	23.4	4.38	240	0.00		
0:25	4.00	23.4	4.45	230	9.60		
0:27	4.00	23.4	4.52	217	0.00		
0:30	4.00	23.4	4.61	196	8.10		
0:33	4.00	23.4	4.81	174	0.00		
0:35	4.00	23.3	5.34	97	5.00		
0:37	4.00	23.3	5.28	117	0.00		
0:40	4.00	23.3	5.63	62	3.00		
0:43	4.00	23.2	5.51	93	0.00		
0:45	4.00	23.2	5.71	54	1.80		
0:50	4.00	23.2	5.63	83	6.00		
0:55	4.00	23.1	5.81	64	2.20		
1:00	4.00	23	5.74	29	0.00		
1:03	4.00	22.9	5.87	55	1.2		
1:05	4.00	22.9	5.84	71	0		
1:10	4.00	22.8	5.8	81	0		

Average Temperature (°C):	22.77	Total Lime Added (g):	384.50	dry lime (g)	76.90

lime slurry was 20%w/w made from 92%CaO

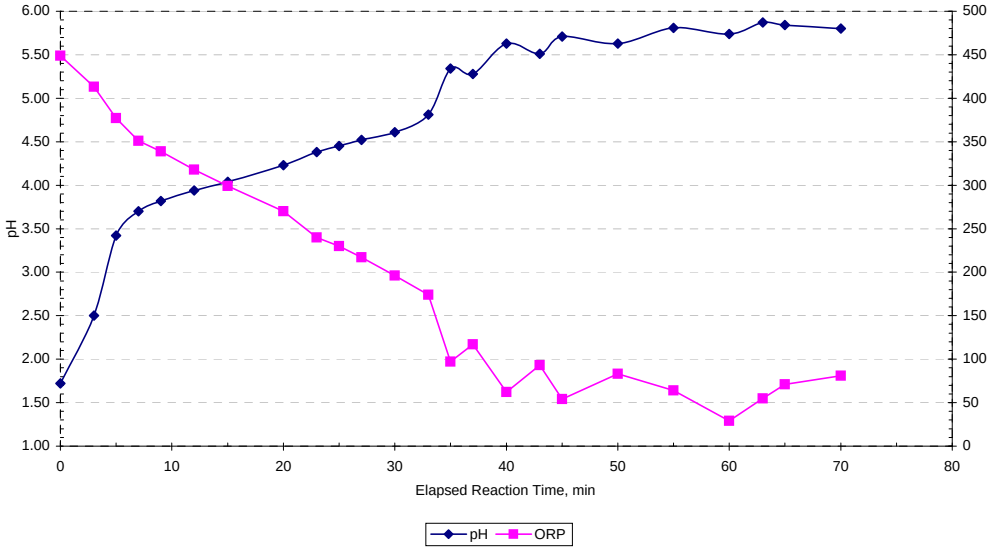
Final reaction Slurry

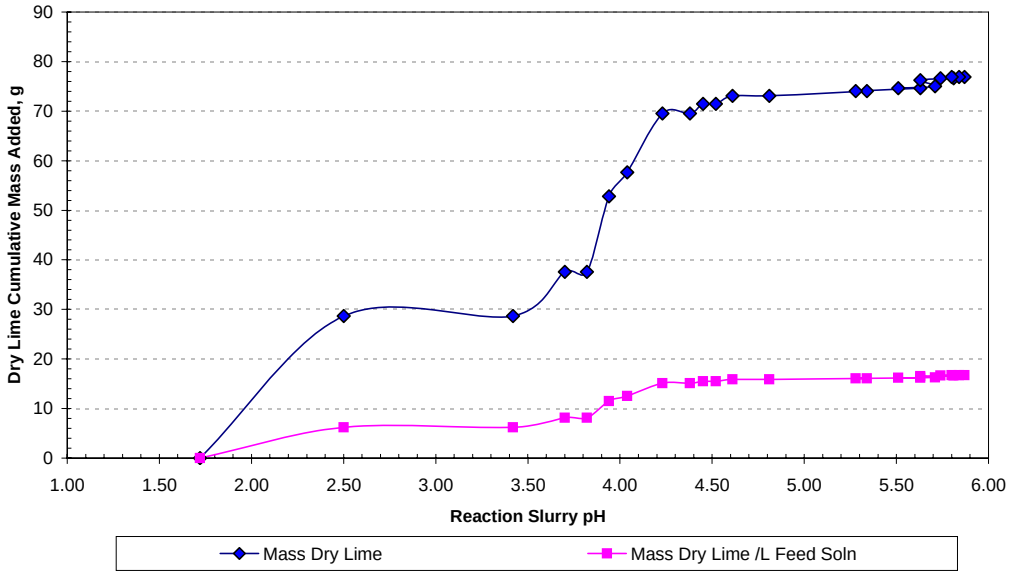
	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, kg	SG, g/mL	Vol, L	Tare	Wet	Dry	Wet	Dry
Filtrate	4486.30	1.1202	4004.91					
Wash	2209.00	1.0439	2116.10					
Residue				1625	2495.3	1859.4	870.3	234.4
				Filter Cake %moisture, %			Reaction Slurry %solids, %	
				73.1			4.2	

Final Filtration:

Diameter of filtration paper:	185	mm			
type of paper (Whatman ##):	3		Washing time:	150	min
Filtration time:	<120	min	Clarity of wash:	clear	
Clarity of filtrate:	clear		Volume of wash:	2116	mL
Colour of filtrate:	WW		Colour of wash:	colourless	
Cake thickness:	15.0	mm			

Project No.: CALR-12607-002
Client: Rock Tech Lithium





Test No.:	RT LF PIR T1													
Test Parameters		**T, °C**	**Final pH**	**Final ORP**										
		23	5.8	81										
					Assays, % or mg/L									
	mass or vol, g or mL	**Li**	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn
RT HF PIR T1 FEED	4604	19300	2114	637	484	5.3	265	18.1	140	1473	281	0.86	< 0.5	5.4
CaO	77		0.05	0.03	52.17			0.36						
RT HF PIR T1 FIL	4005	18400	< 0.8	0.30	496	< 0.2	250	51.7	106	1421	< 5	< 0.02	< 0.5	< 0.7
PR HF PIR T1 WSH	2116	6100												
RT HF PIR T1 Res	234	0.32	5.31	2.52	16.6	0.0911	0.0126	0.0367	0.0692	0.076	0.519	0.00282	0.0009	0.0141
					IN - Elemental Masses, g									
RT HF PIR T1 FEED		88.85	9.73	2.93	2.23	0.02	1.22	0.08	0.64	6.78	1.29	0.00	0.00	0.02
CaO		0.00	0.04	0.02	40.12	0.00	0.00	0.28	0.00	0.00	0.00	0.00	0.00	0.00
Total IN, g		**88.85**	**9.77**	**2.96**	**42.35**	**0.02**	**1.22**	**0.36**	**0.64**	**6.78**	**1.29**	**0.00**	**0.00**	**0.02**
					OUT - Elemental Masses, g									
RT HF PIR T1 FIL		73.69	0.00	0.00	1.99	0.00	1.00	0.21	0.42	5.69	0.00	0.00	0.00	0.00
PR HF PIR T1 WSH		12.91												
RT HF PIR T1 Res		0.750	12.447	5.907	38.910	0.214	0.030	0.086	0.162	0.178	1.217	0.007	0.002	0.033
Total OUT, g		**87.35**	**12.45**	**5.91**	**40.90**	**0.21**	**1.03**	**0.29**	**0.59**	**5.87**	**1.22**	**0.01**	**0.00**	**0.03**
Relative Distribution														
RT HF PIR T1 FIL		99.1	0.0	0.0	4.9	0.0	97.1	70.6	72.4	97.0	0.0	0.0	0.0	0.0
Residue		0.9	100.0	100.0	95.1	100.0	2.9	29.4	27.6	3.0	100.0	100.0	100.0	100.0
Total, %		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
%OUT/IN		98.3	127.4	199.9	96.6	875.2	84.5	81.4	91.0	86.6	94.0	167.0		133.0
% Precipitation, %		0.86	100.0	100.0	95.1	100.0	2.9	29.4	27.6	3.0	100.0	100.0	100.0	100.0
Cal'd Head		18974	2704	1283	8884	46	224	64	127	1275	264	1	0	7

Project: CALR-12607-002
Client: Rock Tech Lithium

Project Tech.: A. Ferguson (2290)
Test Date: 23-Jun-11

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT HM Fe Cl T1**

Project Manager: Stephen Mackie (ext.2058)

Feed/Head: **Acid Roasted β-Spodumene - Variable Acid Dose Tests**
(%Excess Acid of Acid Bake = 30 %)

Test Objective: To conduct a concentrate leach on AR β-Spod that was produced in Muffle furnace.

Test Summary: The feed sample is subjected to a concentrate leach using deioinized water for 60 minutes.

Reagents:

Conc.	Chemical(s)
100%	Deionized Water (Rocky Ridge)

**review all pertanant MSDS prior to handling reagents and ALWAYS wear proper PPE.*

Apparatus:

Qty.	Equipment
1x	~6kg balance, x.xxg ***NOTE: all glassware acid washed before use.***
various	120mL containers
various	1L containers
1x	10 L reactor
1x	Caframo mixer
1x	15cm Buchner filter funnel
1x	glass drying pan
1x	teflon impeller
1x	pH meter w/ATC
1x	Whatman No. 1 filter paper, 12.5cm
1x	4 L vacuum flask
1x	spatula

****various connectors, fittings, clamps, tools etc.***

Procedure:

1. Measure out equal charges of the AR β-Spod and deionized water.
2. Combine the solids and water in a reactor. Stir for 60 minutes.
3. Measure & record initial and final solution **mass, pH and ORP**.
4. Filter sample through buchner filter. Recover the filtrate. Measure & record mass and solution density of filtrate.
5. Conduct three displacement washes of the filter cake. **Recover the Wash 1 fraction separately.**
6. Measure & record mass and solution density of WSH 1 and combined WSH 2&3 fractions.
7. Recover the Final RES, record the NET wet & dry mass, dry in oven, submit to bucking room.
8. Perform analyses as summarized below. Keep all process streams until assays have been reviewed.

NOTE: After the assay data have been reviewed and confirmed please make composite samples of the following test process streams:

Test Series	**RT HM Fe CL**	Composite Sample		Comment
Process Stream		Required physical data	ANA	
All PLS and WSH 1 solutions		Mass, Soln Den	Li, ICP, Si, S	Feed Soln for PIR Tests
All RES		Mass	Li, ICP, WRA, S, XRD, s.g.	keep ~ 1 kg of COMP sample and dispose of the balance
All WSH 2& 3		Mass, Soln Den	Li	Dispose once Assay reviewed & confirmed

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
AR β-Spod Head	RT HM Fe CL T1 Head	~~Li-AAS, ICP-OES~~	
Final Filtrate	RT HM Fe CL T1 Final PLS	Li, Al, Fe-AAS	
Disp. Wash 1	RT HM Fe CL T1 WSH 1	Li-AAS	
Disp. Wash 2, & 3	RT HM Fe CL T1 WSH 2, & 3	HOLD	
Final RES	RT HM Fe CL T1 RES	Li, Al, Fe-AAS	

Project: **CALR-12607-002** **Project Tech.: A. Ferguson (2290)**
Client: **Rock Tech Lithium** **Test Date: 23-Jun-11**

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT HM Fe CL** **Project Engineer:** Stephen Mackie (ext.2058)

Test Data:

	Mass, g		
	tare	Initial	Final
Reactor	430.1	7433.1	7401.9
di H_2O		3500	
Solids		3503	

Test Notes:

TIME	NOTES			
	pH	Temp.(°C)	ORP (mV)	RPM
13:35:00	1.76	28.9	701	140
14:35:00	1.61	26.9	688	140
1:00:00	**:Elapsed Time**			

Average	**1.685**	**27.9**	**694.5**

Sample Data:

Final Reaction Slurry

	Solution			Solid	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
RT HM Fe CL T1 Final PLS	2713.30	1.1736	2312					
RT HM Fe CL T1 WSH 1	2809.30	1.0638	2641					
RT HM Fe CL T1 WSH 2, & 3	4175.80	1.0220	4086					
RT HM Fe CL T1 RES				1624.7	5072.8	4269.0	3448.1	2644.3

Final Filtration:

Diameter of filtration paper:	185	mm
type of paper (Whatman ##):	3	
Filtration time:	>30	min
Clarity of filtrate:	cloudy	
Colour of filtrate:	yellowish green	
Cake thickness:		mm

Washing time:	>30	min
Clarity of wash:	cloudy	
Volume of wash:	2500	mL
Colour of wash:	pale yellow	

Project: CALR-12607-002 **Project Tech.:** D.Lustig (2290)
Client: Rock Tech Lithium **Test Date:** 23-Jun-11

Test Name: **Concentrate Leach @ 50% Solids**
Test No.: **RT HM Fe Cl T2** **Project Manager:** Stephen Mackie (ext.2058)

Feed/Head: **Acid Roasted β-Spodumene - Variable Acid Dose Tests**
(%Excess Acid of Acid Bake = 30 %)

Test Objective: To conduct a concentrate leach on AR β-Spod that was produced in Muffle furnace.

Test Summary: The feed sample is subjected to a concentrate leach using deioinized water for 60 minutes.

Reagents:

Conc.	Chemical(s)
100%	Deionized Water (Rocky Ridge)

**review all pertanant MSDS prior to handling reagents and ALWAYS wear proper PPE.*

Apparatus:

Qty.	Equipment
1x	~6kg balance, x.xxg ***NOTE: all glassware acid washed before use.***
various	120mL containers
various	1L containers
1x	10 L reactor
1x	Caframo mixer
1x	15cm Buchner filter funnel
1x	glass drying pan
1x	teflon impeller
1x	pH meter w/ATC
1x	Whatman No. 1 filter paper, 12.5cm
1x	4 L vacuum flask
1x	spatula

****various connectors, fittings, clamps, tools etc.***

Procedure:

1. Measure out equal charges of the AR β-Spod and deionized water.
2. Combine the solids and water in a reactor. Stir for 60 minutes.
3. Measure & record initial and final solution **mass, pH and ORP**.
4. Filter sample through buchner filter. Recover the filtrate. Measure & record mass and solution density of filtrate.
5. Conduct three displacement washes of the filter cake. **Recover the Wash 1 fraction separately.**
6. Measure & record mass and solution density of WSH 1 and combined WSH 2&3 fractions.
7. Recover the Final RES, record the NET wet & dry mass, dry in oven, submit to bucking room.
8. Perform analyses as summarized below. Keep all process streams until assays have been reviewed.

NOTE: After the assay data have been reviewed and confirmed please make composite samples of the following test process streams:

Test Series	**RT HM Fe CL**	Composite Sample		Comment
Process Stream		Required physical data	ANA	
All PLS and WSH 1 solutions		Mass, Soln Den	Li, ICP, Si, S	Feed Soln for PIR Tests
All RES		Mass	Li, ICP, WRA, S, XRD, s.g.	keep ~ 1 kg of COMP sample and dispose of the balance
All WSH 2& 3		Mass, Soln Den	Li	Dispose once Assay reviewed & confirmed

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
AR β-Spod Head	RT HM Fe CL T2 Head	~~Li-AAS, ICP-OES~~	
Final Filtrate	RT HM Fe CL T2 Final PLS	Li, Al, Fe-AAS	
Disp. Wash 1	RT HM Fe CL T2 WSH 1	Li-AAS	
Disp. Wash 2, & 3	RT HM Fe CL T2 WSH 2, & 3	HOLD	
Final RES	RT HM Fe CL T2 RES	Li, Al, Fe-AAS	

Project: CALR-12607-002 **Project Tech.:** D.Lustig (2290)
Client: Rock Tech Lithium **Test Date:** 23-Jun-11

Test Name: Concentrate Leach @ 50% Solids
Test No.: RT HM Fe CL **Project Engineer:** Stephen Mackie (ext.2058)

Test Data:

	Mass, g		
	tare	Initial	Final
Reactor	430	7721.6	7684.7
di H_2O		3645.9	
Solids		3645.7	

Test Notes:

TIME	NOTES			
	pH	Temp.(°C)	ORP (mV)	RPM
13:43:00	1.76	31.1	700	240
14:43:00	1.6	28.4	697	240
1:00:00	:Elapsed Time			

Average	1.68	29.75	698.5

Sample Data:

Final Reaction Slurry

	Solution			Solid	Gross Mass, g		Net Mass, g	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
RT HM Fe CL T2 Final PLS	3021.70	1.1722	2578					
RT HM Fe CL T2 WSH 1	2819.30	1.0600	2660					
RT HM Fe CL T2 WSH 2, & 3	4061.60	1.0155	4000					
RT HM Fe CL T2 RES				1629.4	5187.0	4394.8	3557.6	2765.4

Final Filtration:

Diameter of filtration paper:	185	mm			
type of paper (Whatman ##):	3		Washing time:	>30	min
Filtration time:	>30	min	Clarity of wash:	Cloudy	
Clarity of filtrate:	Cloudy		Volume of wash:	2500	mL
Colour of filtrate:	Yellowish Green		Colour of wash:	Pale Yellow	
Cake thickness:	--	mm			

Test No.:		RT HM Fe CL														
Test Parameters		**T, °C**	**pH**	**ORP**												
		28	1.7	696												
		Assays, % or mg/L														
	mass or vol, g or mL	**Li**	Si	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn	S
RT AR HL Con	7149	2.27	30.34	10.70	0.57	0.12	0.01	0.33	0.03	0.11	0.32	0.04	0.00	0.00	0.01	7.11
DI Water	7146															
RT HM CL Comp PLS	4890	22900	107	3910	994	320	2	316	16	137	1970	99	1	0	13	0
RT HM CL Comp WSH 1	5301	7640														
RT HM CL Comp WSH 2&3	8086															
RT HM Fe CL COMP RES	5410	0.11	31	13.6	0.44	0.11	0.00	0.42	0.03	0.15	0.29	0.03	0.0	0.0	0.01	0.78
IN (Elemental Masses)																
RT AR HL Con		162.3	2168.7	764.9	40.4	8.8	0.8	23.2	2.2	7.9	22.9	2.7	0.1	0.0	0.6	508.3
DI Water		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.00
Total IN, g		**162.28**	**2168.67**	**764.91**	**40.39**	**8.79**	**0.81**	**23.23**	**2.18**	**7.86**	**22.95**	**2.67**	**0.11**	**0.01**	**0.59**	**509.27**
OUT (Elemental Masses)																
RT HM CL Comp PLS		111.98	0.52	19.12	4.86	1.56	0.01	1.55	0.08	0.67	9.63	0.48	0.00	0.00	0.06	0.00
RT HM CL Comp WSH 1		40.50														
RT HM CL Comp WSH 2&3																
RT HM Fe CL COMP RES		5.9	1699	735.7	24.0	5.8	0.0	22.6	1.5	8.0	15.6	1.9	0.1	0.0	0.6	42.2
Total OUT, g		**158.37**	**1699.80**	**754.84**	**28.88**	**7.41**	**0.03**	**24.16**	**1.54**	**8.68**	**25.21**	**2.37**	**0.10**	**0.01**	**0.62**	**42.20**
Relative Distribution																
RT HM CL Comp PLS		70.7	0.0	2.5	16.8	21.1	26.6	6.4	5.0	7.7	38.2	20.5	2.7	0.0	10.2	0.0
RT HM CL Comp WSH 1		25.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RT HM CL Comp WSH 2&3		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RT HM Fe CL COMP RES		3.7	100.0	97.5	83.2	78.9	73.4	93.6	95.0	92.3	61.8	79.5	97.3	0.0	89.8	0.0
Total, %		**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**0**	**100**	**0**
%OUT/IN		97.6	78.4	98.7	71.5	84.2	3.6	104.0	70.8	110.3	109.9	88.5	93.9	75.7	105.8	8.3
% Extraction, %		96.4	21.6	3.8	40.5	33.6	97.3	2.7	32.8	0.0	32.1	29.6	8.6	24.3	4.9	91.7
Cal'd Head		2.215	23.778	10.559	0.404	0.104	0.000	0.338	0.022	0.121	0.353	0.033	0.001	0.000	0.009	0.590
% Mass Reduction, %	**24.33**															

Project No.: CALR-12607-002 **Project Technologist:** D.Lustig
Client: Rock Tech Lithium **Date of Test:** 23-Jun-11

Test: **RT HM PIR T1 Bulk**
Feed Feed Solution = Combine the filtrate and WSH-1 fraction of Heavy Media Con Concentrate Leach
Lime 20 wt%
Apparatus Bulk reactor flask with lid, overhead mixer and impeller, condenser, pH/ORP meter and pH and ORP probes Air Sparging line with in-line flow meter, heating mantle and YSI controller

Test Summary:

To modify the pH of the Feed solution to a specific target within the pH isotherm.

Procedure Feed Solution Prep
Combine the WL filtrates and Wash 1 fractions as discussed. Stir for ~10 minutes.
Measure & record mass and solution density of the resulting solution.
reab a sample of the Feed Sol'n for assay.

Feed Soln	Mass, g	Sol'n Den., g/L	V, mL
	11216.00	1117.3	10038.5

Please take a picture of the set-up and process slurries/solutions

Bulk Reactor Set-up
Assemble the reactor with the impeller, lid (ports with septa).(10L pail)
Record the mass of the assembled reactor (this will be the reactor tare).
Standardize pH and ORP probes.

Lime slurry Preparation

Dry Lime Requirement, g/L Feed Sol'n			**20.0**	**This is an estimate.**
Vol of Feed Sol'n, mL			10038.5	
Approximate mass of Dry Lime required, g			200.8	
20% Lime Slurry	Theo.	Actual	4 Portions	
Mass of dry Lime, g	251.0		62.740535	
Mass of DI H_2O, g	1003.8		250.96214	

Check
20

Prepare a 20% solids slurry of lime based on the above data. Use Rocky Ridge di Water to prepare the lime slurry.

PIR Test - Air Sparging & pH Modification
Add the Full amount of the combined solution to the tared reactor. Measure & record mass.
Maintain the reaction solution temperature above 15 °C.
Record pH and ORP of the solution.
Start sparging the solution with air and then immediately start pH modification. **Reaction Time = 0.**
Use the Lime slurry to modify the pH of the solution to the specified target values.
Add the lime in ~ 4 portions slowly. Measure the mass of each lime addition charge.
Monitor & record the reaction solution pH & ORP every 5 minutes.
As the reaction slurry pH approaches the target reduce the size of lime slurry charges are added.
pH Target: **5.8**

Sample Preparation

Final Reaction Slurry:
Filter the reaction slurry.
Recover the filtrate; measure & record the mass and soltion density of the filtrate.
Collect a sample of the filtrate for analysis .
Conduct three displaccement washes of the filter cake (deionized water). Use some of the wash water to rinse out the reactor. Combine wash fractions.
Measure & record the mass and solution density of the combined Wash solution.
Collect the filter cake. Measure & record wet mass. Dry the filter cake. Measure & record the dry solids mass.

Assays:

Sample	Sample ID(s)	Analysis	LIMS No.
Feed	RT HM PIR T1 FEED	Li, ICP	***CA03377-JUN11***
FINAL FIL	RT HM PIR T1 FIL	Li, ICP	***CA03377-JUN11***
WASH	RT HM PIR T1 WSH	Li	***CA03377-JUN11***
Final RES	RT HM PIR T1 RES	Li, ICP	***CA03460-JUN11***

Project No.: CALR-12607-002 **Project Technologist:** D.Lustig
Client: Rock Tech Lithium **Date of Test:** 23-Jun-11

Test: **RT HM PIR T1** **Bulk**

Data

Combined Feed Solution

Weight (g):	11249
SG (g/mL):	1.1173
Volume (mL):	10068

	Mass, kg	
Reactor	Gross	Net
tare	**0.97**	
Initial	12.18	**11.22**
Final	13.03	**12.06**

Reaction Time, min	Air Flow, L/min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of lime slurry added, g	Comment/Observations
<0	5.00	21.6	1.59	721	0.00	Before sparging. RPM
9:45	5.00	21.6	1.58	770	0	**Impeller Type: pitch 4 blade**
9:50	5.00	24.3	2.35	803	310.8	
9:55	5.00	25.9	4.01	729	311.60	solution turned tan
10:00	5.00	26.7	4.69	613	268.60	solution is light brown
10:05	5.00	26.6	4.95	590	0	
10:10	5.00	26.5	5.35	553	5.20	
10:15	5.00	26.4	5.75	528	7.00	
10:20	4.50	26.2	5.73	509	0.00	
10:25	4.50	26.1	5.80	489	0.90	
10:30	4.50	26.0	5.83	474	0.00	
10:35	4.50	25.8	5.86	460	0.00	

Average Temperature (°C): 25.65 Total Lime Added (g): 904.10 dry lime (g) 180.82

lime slurry was 20%w/w made from 92%CaO

Final reaction Slurry

	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, kg	SG, g/mL	Vol, L	Tare	Wet	Dry	Wet	Dry
Filtrate	10.10	1.0914	9.25					
Wash	2.56	1.041	2.46					
Residue				1625	3107.2	2074.3	1482.2	449.3

Final Filtration:

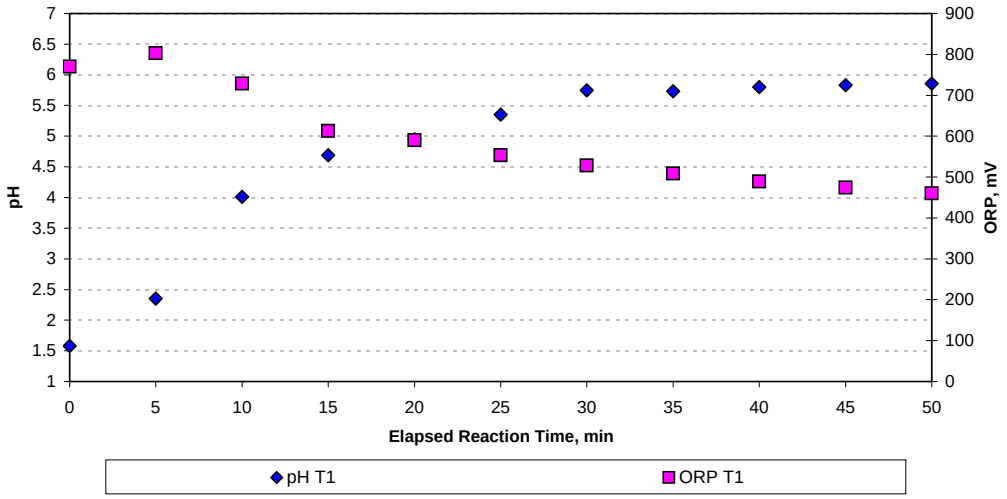
Diameter of filtration paper:	185	mm
type of paper (Whatman ##):	3	
Filtration time:	<70	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	50 x 2	mm

Washing time:	40	min
Clarity of wash:	cc	
Volume of wash:	2462	mL
Colour of wash:	ww	

Test No.:		RT HM PIR T1											
Test Parameters		**T, °C**											
		26											
		Assays, % or mg/L											
	mass or vol, g or mL	**Li**	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	Zn
RT HM PIR T1 FEED	10068	13400	2560	660	222	1	214	11	91	1280	64	0	8
Lime	181				52			0.356					
FINAL FIL	9252	12500	< 0.8	0	465	< 0.2	199	52	53	1240	< 5	< 0.02	< 0.7
WASH	2462	5340											
Final RES	449	0.7	5	1	19	0	<0	0	0	0	0	0	0
IN - Elemental Masses, g													
RT HM PIR T1 FEED		134.91	25.77	6.64	2.24	0.01	2.15	0.11	0.91	12.89	0.64	0.00	0.08
Lime					94.34			0.64					
Total IN, g		**134.91**	**25.77**	**6.64**	**96.57**	**0.01**	**2.15**	**0.75**	**0.91**	**12.89**	**0.64**	**0.00**	**0.08**
OUT - Elemental Masses, g													
FINAL FIL		115.65	0.00	0.00	4.30	0.00	1.84	0.48	0.49	11.47	0.00	0.00	0.00
WASH		13.15											
Final RES		3.145	24.211	6.021	84.019	0.008	0.000	0.109	0.337	0.467	0.580	0.009	0.071
Total OUT, g		**131.95**	**24.21**	**6.02**	**88.32**	**0.01**	**1.84**	**0.59**	**0.83**	**11.94**	**0.58**	**0.01**	**0.07**
Relative Distribution													
FINAL FIL		97.6	0.0	0.0	4.9	0.0	100.0	81.5	59.2	96.1	0.0	0.0	0.0
Residue		2.4	100.0	100.0	95.1	100.0	0.0	18.5	40.8	3.9	100.0	100.0	100.0
Total, %		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
%OUT/IN		97.8	93.9	90.6	91.5	69.0	85.5	78.1	90.8	92.7	90.0	239.6	86.5
% Precipitation, %		2.4	100.0	100.0	95.1	100.0	0.0	18.5	40.8	3.9	100.0	100.0	100.0

Cal'd Head		13106	2405	598	8772	1	183	58	82	1186	58	1	7

Project No.: CALR-12607-002
Client: Rock Tech Lithium



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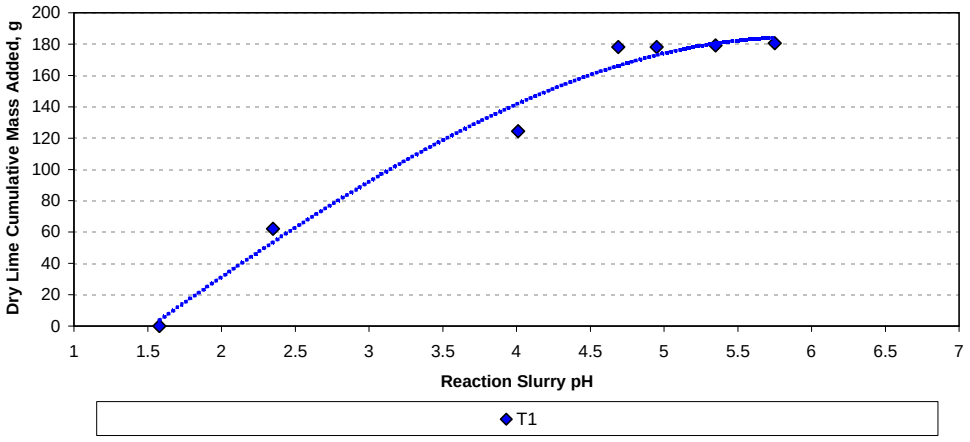


Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech Lithium **Date of Test:** 27-Jun-11
Test No.: **RT COMP SIR T1**

Feed **Bulk COMP** LIMS No.: ***CA03335-JUN11***

Reagents 2 M NaOH
50 wt % Na_2CO_3 Solution
Rocky Ridge Water

Apparatus 10 L reactor flask with lid, overhead mixer and impeller, pH/ORP probes & meter

Test Summary: Secondary Impurity Removal - This test is designed to remove the remaining minor impurites from the Li-solution. Stoichiometric addition of Na_2CO_3

Procedure

Reagent Prep.

Prepare the Na_2CO_3 solution as defined.

Stoichiometric Excess	**1.1**
25	wt% Na_2CO_3(aq) Sol'n
mass Na_2CO_3, g	23.3
mass di H_2O, g	69.8

Test Procedure

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solution therefore ensure that everything that is in contact with the solution is clean.
2 Measure & record the tare of the reactor.
3 Obtain the Feed solution. Add **17381** mL of the solution to the reactor.
4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH, ORP & T of the stirred solution.
6 Modify the pH of the solution to pH = 10 using 2 M NaOH. Add slowly so the pH target is not overshot.
7 Record the mass of NaOH used after each addition. Record the pH, ORP, T and elapsed time.
8 Once the pH is stable at 10 add **30.75 g** of the 25 wt % Na2CO3(aq) solution.
9 Stir the resulting solution for 60 minutes. Measure & record the pH, ORP & T every 5 minutes.
10 After 60 minutes the reaction is complete.
11 Measure & record the final mass of the reactor and reaction slurry.
12 Vacuum filter the reaction slurry. Recover the filtrate. Displacement wash the filter cake (3 X 50 mL RRW). **NOTE:** Adjust Wash fraction volume based on recovered solids. Combine the wash fractions.
13 Recover the filter cake - record wet mass and dry O/N in an oven. When dry record the dry mass.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
Bulk COMP	**RT COMP SIR T1 Feed**	**Li-AAS, ICP, S-ICP**	
Filtrate	**RT COMP SIR T1 FIL**	**Li-AAS, ICP, S-ICP**	
Wash	**RT COMP SIR T1 WSH**	**Li-AAS**	
Residue	**RT COMP SIR T1 RES**	**No**	

Project No.: CALR-12607-002
Client: Rock Tech Lithium
Test No.: **RT COMP SIR T1**

Project Technologist: A.Ferguson
Date of Test: 27-Jun-11

Data

Bulk COMP

Weight (g):	19130
SG (g/mL):	1.1032
Volume (mL):	17340.5

	Mass, kg	
Reactor	**Gross**	**Net**
tare	**0.98**	
Initial	**20.11**	19.13
Final	**20.23**	19.25

2 M NaOH	S.G., g/mL	1.081

Reaction Time, min	Sol'n T, °C	Sol'n pH	Sol'n emf, mV	Mass of 2 M NaOH added, g	Comment/Observations
0:00	21.0	5.38	527	0	**Impeller Type:** pitch 4 blade
0:03	21.0	9.51	169	8.50	**Impeller Daimeter, in.:** 5
0:06	21.0	9.72	47	11.10	rpm: 200
0:13	21.0	9.95	17	10.60	
0:19	21.0	9.99	26	3.10	
0:25	21.0	10.00	33	1.40	
0:30	21.0	10.04	30	1.80	
0:40	21.0	10.00	58	1.80	
0:45	21.0	10.37	25	0	
0:50	21.1	10.34	43	0	
0:55	21.2	10.32	52	0	
1:00	21.2	10.30	63	0	
1:05	21.2	10.29	74	0	
1:10	21.2	10.28	84	0	
1:15	21.2	10.27	89	0	
1:20	21.2	10.26	95	0	
1:25	21.2	10.25	99	0	
1:30	21.2	10.24	103	0	
1:35	21.2	10.23	105	0	
1:40	21.3	10.22	110	0	
1:45	21.3	10.21	115	0	
Average/*Total*	21.12	9.91		38.30	

Final reaction Slurry

	Solution			Solid				
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Net Dry	
Filtrate, kg	19.09	1.1035	17300					
Wash, g	330.00	1.0108	326					min
Residue				529.3	553.5	533.8	4.5	mL

Rxn Slurry % Solids, % 0.023

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	0.45 um	
Filtration time:	>120	min
Clarity of filtrate:	clear	
Colour of filtrate:	colourless	
Cake thickness:	~5	mm

Washing time:	>60
Clarity of wash:	clear
Volume of wash:	250
Colour of wash:	colourless

Test No.:	RT COMP SIR T1													
Test Parameters		**T, °C**	**Final pH**											
		21	10.2											
							Assays, % or mg/L							
	mass or vol, g or mL	**Li**	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn
RT COMP SIR T1 FEED	17340	16200	<1	4.5	461	<0	220	50	84.7	1620	<5	0.02	<0	<1
Na2CO3	38									43.38				
RT COMP SIR T1 FIL	17300	16500	<1	0.2	280	<0	224	42.4	<0	2300	<5	0.02	<0	<1
RT COMP SIR T1 WSH	326	1680												
RT COMP SIR T1 RES	5	0.41	0.264	1.38	14.6	0.0147	0.0478	1.69	22.4	0.189	0.02	0.00098	0.0004	0.0944
							IN - Elemental Masses, g							
RT COMP SIR T1 FEED		280.92	0.01	0.08	7.99	0.00	3.81	0.87	0.00	28.09	0.00	0.00	0.00	0.00
Na2CO3		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	16.62	0.00	0.00	0.00	0.00
Total IN, g		**280.92**	**0.01**	**0.08**	**7.99**	**0.00**	**3.81**	**0.87**	**0.00**	**44.71**	**0.00**	**0.00**	**0.00**	**0.00**
							OUT - Elemental Masses, g							
RT COMP SIR T1 FIL		285.44	0.01	0.00	4.84	0.00	3.88	0.73	0.00	39.79	0.00	0.00	0.00	0.00
RT COMP SIR T1 WSH		0.55												
RT COMP SIR T1 RES		0.018	0.012	0.062	0.657	0.001	0.000	0.076	1.008	0.009	0.001	0.000	0.000	0.004
Total OUT, g		**286.01**	**0.03**	**0.07**	**5.50**	**0.00**	**3.88**	**0.81**	**1.01**	**39.80**	**0.00**	**0.00**	**0.00**	**0.00**
Relative Distribution														
RT COMP SIR T1 FIL		100.0	53.8	5.3	88.1	0.0	100.0	90.6	0.0	100.0	0.0	0.0	0.0	0.0
Residue		0.0	46.2	94.7	11.9	100.0	0.0	9.4	100.0	0.0	100.0	100.0	100.0	100.0
Total, %		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
%OUT/IN		101.8	185.4	84.0	68.8		101.6	93.4		89.0				
% Precipitation, %		0.01	46.2	94.7	11.9	100.0	0.0	9.4	100.0	0.0	100.0	100.0	100.0	100.0
Cal'd Head		16494	1	4	317	0	223	47	58	2295	0	0	0	0

Project No.: CALR-12607-002
Client: Rock Tech Lithium

Project Technologist: A. Ferguson (2382)
D. Lustig
Date of Test: 28-Jun-11

Test: RTL Bulk IX
Test No.: RTL Bulk IX
Feed: SIR FIL

IX Procedures:

Review MSDS for resin and every chemical reagent requied in this test.

IX Column

Use two 60 cm x 2.5 cm columns set up in a Lead-Lag configuration.
Use ~200 mL of regenerated Purolite S950 resin/column.
Pack column using resin in Rocky Ridge water.
Back wash the column with Rocky Ridge water to remove trapped air bubbles and to properly distribute the resin.

Loading:

1 Use the SIR FIL as the feed for this test.
2 The feed is added to the column in a down flow direction at a flow rate of 10 BV/h (~42 mL/min).
3 Collect the following product BV fractions at 5, 7.5, 10, 20, 30, 45, 60 and 70 BV.
4 The Loading cycle is complete once all of the Feed solution has been added.
5 Switch the feed line from the SIR container to Rocky Ridge Water - maintain the 10 BV/h flow rate. (**Wash-1**)
6 Continue to collect the next fraction for 5 BV. Stop operation after 5 BV's of water has been added to the column.

NOTE: The Stripping and Regeneration steps are only conducted on the Lead column. The Lag column is to be stripped and regenerated in the same way with the exception that no sampling is to be conducted.

Stripping:

Strip the column with 1 M HCl at a flow rate of 4 BV/h for 1 h.
Collect the strip effluent as six separate fractions (1 fraction every 10 minutes).
After 1 h switch the feed line from the HCl to distilled water. (**Wash-2**)
Run the distilled water at the same flow rate for 2 BV's (30 minutes). Collect 6 separate wash fractions (1 fraction every 5 minutes).
Measure & record the pH of each fraction.

Regeneration:

Regenerate the resin using 1 M NaOH added to the column at a rate of 4 BV/h.
Run ~ 10 BV's of 1 M NaOH through the column. Collect the product as one fraction.

Regeneration Wash

After the Regeneration step run distilled water through the column @ 10 BV/h for 30 minutes. Collect the Regen Wash eluant as six separate fractions (1 fraction every 5 minutes).

Sampling Schedule:

Samples		Sample ID	Assays	Sample I.D.
Kinetic Samples	Loading			
		L-X (X=1, 2, 3, …)	Li, Ca, Mg	
		(fractions collected 5, 7.5, 10, 20, 30, 45, 60, 70 BV)		
	Strip	Strip - 1 through 6	Li	

Assay Sample ID

Sample	Sample ID(s)	Analysis
S1 Feed COMP	RTL Bulk IX FEED	Li-AAS, ICP-OES
L-1	RTL Bulk IX L-1	Li, Mg-AAS, Ca-ICP
L-2	RTL Bulk IX L-2	Li, Mg-AAS, Ca-ICP
L-3	RTL Bulk IX L-3	Li, Mg-AAS, Ca-ICP
L-4	RTL Bulk IX L-4	Li, Mg-AAS, Ca-ICP
L-5	RTL Bulk IX L-5	Li, Mg-AAS, Ca-ICP
L-6	RTL Bulk IX L-6	Li, Mg-AAS, Ca-ICP
L-7	RTL Bulk IX L-7	Li, Mg-AAS, Ca-ICP
L-8	RTL Bulk IX L-8	Li, Mg-AAS, Ca-ICP
Strip-1	RTL Bulk IX Strip-1	Li-AAS
Strip-2	RTL Bulk IX Strip-2	Li-AAS
Strip-3	RTL Bulk IX Strip-3	Li-AAS
Strip-4	RTL Bulk IX Strip-4	Li-AAS
Strip-5	RTL Bulk IX Strip-5	Li-AAS
Strip-6	RTL Bulk IX Strip-6	Li-AAS
Product	RTL Bulk IX Product	Li-AAS, ICP-OES

Detailed Data:

Total Volume of SIR FIL, L	17.3	Rxn Time, h	6.92	
Mass of SIR FIL, kg		Feed Soln BV Equiv, BV		70
Soln Den. Of SIR FIL, g/mL				

Operating Conditions:

				Approx. flow	
	Bed Volume:	250 mL		L/h	mL/min
Loading Feed:	SIR FIL	Loading Flowrate:	10 BV/h	2.5	41.7
Wash-1:	Rocky Ridge	Wash Flowrate:	10 BV/h	2.5	41.7
Strip Sol'n:	2 M HCl	Stripping Flowrate:	5 BV/h	1.3	20.8
Wash-2:	Rocky Ridge	Wash Flowrate:	5 BV/h	1.3	20.8
Regen Sol'n:	2 M NaOH	Regen Flowrate:	5 BV/h	1.3	20.8
Final Wash:	Rocky Ridge	Wash Flowrate:	10 BV/h	2.5	41.7

Kinetic Samples:

Loading (10 BV/h)			
Time	**Sample**	**Bed Vol.**	**Total V, L**
h	ID	BV	
0	Start	0	
0.50	L-1	5	1.25
0.75	L-2	7.5	1.88
1.00	L-3	10	2.50
2.00	L-4	20	5.00
3.00	L-5	30	7.50
4.50	L-6	45	11.25
6.00	L-7	60	15.00
7.00	L-8	70	17.50

Sampling:
Collect the sample from the discharge line of the **Lead column**. Sample collection is based on elapsed time. At the sampling time collect ~80 mL of the product solution from the Lead column in a tared Sample cup.
Sample Processing:
Measure & record NET mass of sample.
Cut ~30 mL of the sample for ANA purposes.
Measure & record solution density of sample.

DATA:

Loading

Sample ID	Net Mass, g	Solution Den., g/mL	Cal'd Vol, mL
L-1	35.3	1.1021	32.0
L-2	36.0	1.1020	32.7
L-3	36.7	1.1020	33.3
L-4	30.6	1.1020	27.8
L-5	39.5	1.1023	35.8
L-6	30.3	1.1021	27.5
L-7	36.7	1.1019	33.3
L-8	34.7	1.1022	31.5
Avg/Sum	279.8	1.1021	253.88

Sample ID	Mass, g	Solution Den., g/mL	Cal'd Vol, mL
Feed Wash-1	1212.8	1.0023	1210.02

Stripping

Sample ID	Mass, g	Solution Den., g/mL	Cal'd Vol, mL
Strip-1	27.9	1.0195	27.4
Strip-2	43.8	1.0493	41.7
Strip-3	55.4	1.0367	53.4
Strip-4	44.3	1.0332	42.9
Strip-5	40.0	1.0326	38.7
Strip-6	41.0	1.0311	39.8
Avg/Sum	252.4	1.0337	243.92

Strip Wash

Sample ID	Mass, g	Solution Den., g/mL	Cal'd Vol, mL	pH
SW-1	49.0	1.0323	47.5	0.49
SW-2	46.3	1.0420	44.4	0.88
SW-3	22.5	0.9986	22.5	1.75
SW-4	21.9	0.9982	21.9	2.04
SW-5	18.2	0.9983	18.2	2.25
SW-6				
Avg/Sum			154.6	

Notes	start weight, g	End Weight, g	Total	Soln. Den. g/mL	Actual Flowrate mL/Min
Feed	18960	18620		1.1016	39.6
Feed WSH		1212.8	1212.8		40.4
Strip		1176.50	1176.50	1.0297	23.1
Strip WSH		414.8	414.8	1.0031	18.9

Further IX Polishing:

	Feed		
	Mass, kg	Soln Den., g/mL	Cal'd V, L
Initial	18.62	1.1016	16.90
Final	18.43	1.1001	16.75
tare			

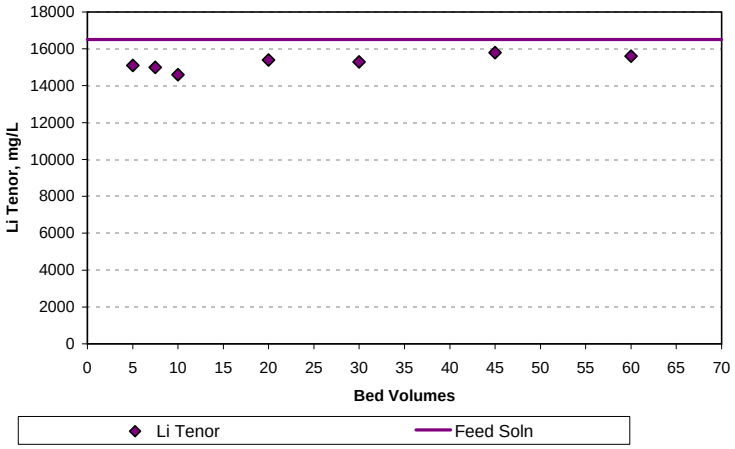
Volume of Resin, mL	510

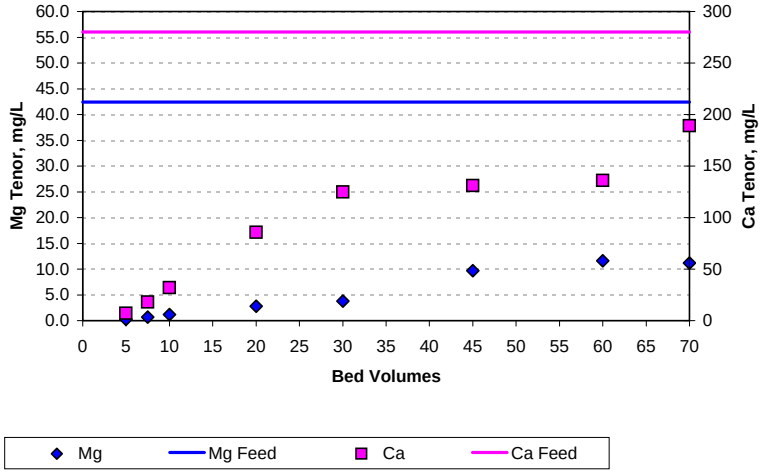
Procedure

Measure out ~500 mL of resin in a graduated cylinder. Record volume.
Rinse the resin with deionized water.
Collect a 300 mL sample of the feed solution. Label and set aside.
Measure & record the mass of the feed solution.
Add the resin to the feed solution. Gently stir the resulting mixture with a clean teflon impeller. Stir for 60 min.
After 60 min. Stop the agitation and separate the resin from the solution.
Measure & record the mass and solution density of the treated solution.
Store the recovered resin for regenreation at a later date. Label accordingly.
Grab a sample of the treated solution. **ANA - Li, ICP**

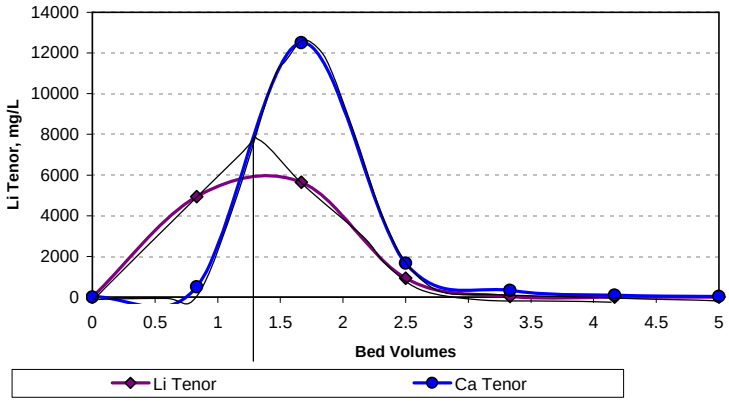
		Assays, mg/L
Process Stream	V, L	Li
Feed	17.3	16500
Strip	1.1	1727
Product Solution	17.2	14400
		IN (Elemental Masses, g)
Process Stream		Li
Feed		285
Total IN, g		**285**
		OUT (Elemental Masses, g)
Process Stream		Li
Strip		2
Product Solution		247
Total OUT, g		**249**
		Distribution, %
Strip		**0.8**
Product Solution		**99.2**
% Total, %		**100**
OUT/IN, %		**87.2**
Cal'd Head		**14395**
Li %Loss, %		**0.8**
%M Removed, %		

Loading Profiles





Stripping Profiles



Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson J. Condon
Client: Rock Tech **Date of Test:** 12-Jul-11

Li_2CO_3 Precipitation
Test No. Bulk Li2CO3 PPT T1
Objective: To conduct Li_2CO_3 PPT using the bulk IX RTL Product solution.

Feed Polished IX Product Soln

Reagents	Li_2CO_3 (99.3%$^+$)	Rocky Ridge Water
	25 wt. % Na_2CO_3(aq)	(This solution was preheated to 45 °C)

Apparatus One 4 L reactor with lid, overhead mixer and impeller, with pH and ATC probes

Test Summary:

To produce Li2CO3 from a Spodumene sample.

Stoichiometric Excess of Na_2CO_3	**1.04**

	Required 25	wt% Na_2CO_3(aq) Sol'n
mass Na_2CO_3, g		954.9
mass RR water H_2O, g		2864.8
Total Mass of 25% Na_2CO_3(aq), g		**3819.7**

Procedure

Reagent Prep.

Prepare the Na_2CO_3 solution as defined.

Mass, g	9200

Li2CO3 Precipitation Test

1 Weigh out the required amount of feed solution.
2 Add solution to the 15 L reactor. Pre heat the feed solution to ~90 - 95°C.
3 Prepare the Na_2CO_3 solution. Pre-heat to 45 °C.
4 Measure & record the pH & temp of the feed solution.
5 Start the addition of Na_2CO_3(aq). Add total volume of Na_2CO_3(aq) over 10 minutes. Note the time when all of the solution has been added. This is T = 0 min.
6 Stir vigourously for 20 minutes.
7 After 20 minutes record final pH. Record mass of reaction slurry.
8 Vacuum filter the reaction slurry.

NOTE: Theoretically 640.1 g of Li_2CO_3 will be produced.

Product

Recover the Product Filtrate. Measure & record the mass and solution density.
Wash the Product solids with hot d.i. water (T≥ 95 °C) (Adjust volume of Wash fraction based on the amount of solids present.)).
Combine Wash fractions. Measure & Record mass and solution density of combined wash fraction.
Dry the filter cake.
Record the dry mass of the product filter cake.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
Feed	Bulk Li2CO3 PPT T1 Feed	Li-AAS, ICP-OES	CA02475-JUL11
Product Filtrate	Bulk Li2CO3 PPT T1 FIL	Li-AAS, ICP-OES	CA02475-JUL11
Product Wash	Bulk Li2CO3 PPT T1 WSH	Li-AAS, ICP-OES	CA02475-JUL11
PROD	Bulk Li2CO3 PPT T1 PROD	Li-AAS, ICP-OES	CA02504-JUL11

Test No. **Bulk Li2CO3 PPT T1**

Project Technologist: A.Ferguson J. Condon
Date of Test: **12-Jul-11**

Initial Data

Feed Solution

Mass, g	**9200.0**
Soln Den., g/mL	1.1001
Cal'd V, mL	8362.9

Reaction Time, min	Precipitation Test Data			
	Sol'n T, °C	Sol'n pH	Mass of Na2CO3(aq) Added, kg	Comment
11:25	21.2	10.70		Heat On
12:00	95.0			adding Na2CO3
12:05	90.5		3.8	All Na2CO3 in. T=0
12:37	96.1	10.96		test complete. Filtering
Average:	93.9	10.96		

Mass of Na_2CO_3(aq) Added,kg	3.81

Final Reaction Slurry

	Solution			**Solids**	**Gross Mass, g**		**Net Mass, g**	
	Mass,k g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	11.76	1.1117	10578.4					
Wash	0.99	1.0485	944.2					
Product				1630.0	2332.0	2052.7	702.0	422.7

Final Filtration:

Diameter of filtration paper:	1	mm
type of paper (Whatman ##):	185	
Filtration time:	<10	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	50.0	mm

Washing time:	<5	min
Clarity of wash:	cc	
Volume of wash:		mL
Colour of wash:	ww	

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson J. Condon
Client: Rock Tech **Date of Test:** 12-Jul-11

Li_2CO_3 Precipitation
Test No. Bulk Li2CO3 PPT T1
Objective: To conduct Li_2CO_3 PPT using the bulk IX RTL Product solution.

Feed Polished IX Product Soln

Reagents	Li_2CO_3 (99.3%$^+$)	Rocky Ridge Water
	25 wt. % Na_2CO_3(aq)	(This solution was preheated to 45 °C)

Apparatus One 4 L reactor with lid, overhead mixer and impeller, with pH and ATC probes

Test Summary:

To produce Li2CO3 from a Spodumene sample.

Stoichiometric Excess of Na_2CO_3	**1.04**

	Required 25	wt% Na_2CO_3(aq) Sol'n
mass Na_2CO_3, g		954.9
mass RR water H_2O, g		2864.8
Total Mass of 25% Na_2CO_3(aq), g		**3819.7**

Procedure

Reagent Prep.

Prepare the Na_2CO_3 solution as defined.

Mass, g	9200

Li2CO3 Precipitation Test

1 Weigh out the required amount of feed solution.
2 Add solution to the 15 L reactor. Pre heat the feed solution to ~90 - 95°C.
3 Prepare the Na_2CO_3 solution. Pre-heat to 45 °C.
4 Measure & record the pH & temp of the feed solution.
5 Start the addition of Na_2CO_3(aq). Add total volume of Na_2CO_3(aq) over 10 minutes. Note the time when all of the solution has been added. This is T = 0 min.
6 Stir vigourously for 20 minutes.
7 After 20 minutes record final pH. Record mass of reaction slurry.
8 Vacuum filter the reaction slurry.

NOTE: Theoretically 640.1 g of Li_2CO_3 will be produced.

Product

Recover the Product Filtrate. Measure & record the mass and solution density.
Wash the Product solids with hot d.i. water (T≥ 95 °C) (Adjust volume of Wash fraction based on the amount of solids present.)).
Combine Wash fractions. Measure & Record mass and solution density of combined wash fraction.
Dry the filter cake.
Record the dry mass of the product filter cake.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
Feed	Bulk Li2CO3 PPT T1 Feed	Li-AAS, ICP-OES	CA02475-JUL11
Product Filtrate	Bulk Li2CO3 PPT T1 FIL	Li-AAS, ICP-OES	CA02475-JUL11
Product Wash	Bulk Li2CO3 PPT T1 WSH	Li-AAS, ICP-OES	CA02475-JUL11
PROD	Bulk Li2CO3 PPT T1 PROD	Li-AAS, ICP-OES	CA02504-JUL11

Test No. **Bulk Li2CO3 PPT T1**

Project Technologist: A.Ferguson J. Condon
Date of Test: **12-Jul-11**

Initial Data

Feed Solution

Mass, g	**9200.0**
Soln Den., g/mL	1.1001
Cal'd V, mL	8362.9

	Precipitation Test Data			
Reaction Time, min	Sol'n T, °C	Sol'n pH	Mass of Na2CO3(aq) Added, kg	Comment
13:20	32.6	10.70		Heat on
13:40	95.7			at temp
13:43	95.2			adding Na2CO3
13:47	90.7		3.81	All Na2CO3 in. T=0
14:17	95.4	10.95		Test complete. Filerting
Average:	94.3	10.95		

Mass of Na_2CO_3(aq) Added, kg	3.81

Final Reaction Slurry

	Solution			Solids	Gross Mass, g		Net Mass, g	
	Mass, kg	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	12.0	1.1115	10832.2					
Wash	1.1	1.0433	1092.7					
Product				1626.2	2307.7	2150.6	681.5	524.4

Final Filtration:

Diameter of filtration paper:	1	mm
type of paper (Whatman ##):	185	
Filtration time:	<10	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	50.0	mm

Washing time:	<10	min
Clarity of wash:	cc	
Volume of wash:	1093	mL
Colour of wash:	ww	

Test No.:	Li2CO3 PPT T1, Bulk Li2CO3 PPT T1													
Test Parameters		**T, °C**	**Final pH**											
		94	11.0											
							Assays, % or mg/L							
	mass or vol, g or mL	**Li**	Al	Fe	Ca	Cr	K	Mg	Mn	Na	P	Ti	V	Zn
Bulk Li2CO3 PPT Head	16726	14400	0.8	0.7	16.4	< 0.2	202	1.19	< 0.04	4300	< 5	< 0.02	< 0.2	< 0.7
Na2CO3	7620									43.3815307				
Bulk Li2CO3 PPT T1 FIL	21411	2460	0.8	0.4	2.5	< 0.2	0.4	2.75	< 0.04	46700	< 5	< 0.02	< 0.2	< 0.7
Bulk Li2CO3 PPT T1 WSH	2037	2280												
Bulk Li2C03 PPT T1 PROD	947	19.1	<0	<0	0.026	<0	0.002	0.0035	<0	0.15	<0	0.00009	<0	<0
							IN - Elemental Masses, g							
Bulk Li2CO3 PPT Head		240.85	0.01	0.01	0.27	0.00	3.38	0.02	0.00	71.92	0.00	0.00	0.00	0.00
Na2CO3		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	3305.67	0.00	0.00	0.00	0.00
Total IN, g		**240.85**	**0.01**	**0.01**	**0.27**	**0.00**	**3.38**	**0.02**	**0.00**	**3377.59**	**0.00**	**0.00**	**0.00**	**0.00**
							OUT - Elemental Masses, g							
Bulk Li2CO3 PPT T1 FIL		52.67	0.02	0.01	0.05	0.00	0.01	0.06	0.00	999.88	0.00	0.00	0.00	0.00
Bulk Li2CO3 PPT T1 WSH		4.64												
Bulk Li2C03 PPT T1 PROD		180.896	0.019	0.004	0.246	0.004	0.019	0.033	0.000	1.421	0.028	0.001	0.002	0.002
Total OUT, g		**238.21**	**0.04**	**0.01**	**0.30**	**0.00**	**0.03**	**0.09**	**0.00**	**1001.30**	**0.03**	**0.00**	**0.00**	**0.00**
Relative Distribution														
Bulk Li2CO3 PPT T1 FIL		24.1	47.5	69.3	17.9	0.0	31.1	64.0	0.0	99.9	0.0	0.0	0.0	0.0
Residue		75.9	52.5	30.7	82.1	100.0	68.9	36.0	100.0	0.1	100.0	100.0	100.0	100.0
Total, %		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
%OUT/IN		98.9	269.6	105.5	109.3		0.8	462.4		29.6				
% Precipitation, %		75.94	52.5	30.7	82.1	100.0	68.9	36.0	100.0	0.1	100.0	100.0	100.0	100.0
Cal'd Head		14242	2	1	18	0	2	6	0	59866	2	0	0	0

Project No.: CALR-12607-002 **Project Technologist:** A.Ferguson
Client: Rock Tech **Date of Test:** 14-Jul-11
Test No.: **RT Bulk BCB T1**

Feed **RT Bulk Li2CO3 PPT T1**

Reagents CO_2(g) Rocky Ridge Distilled Water

Apparatus 3 L Reactor, overhead mixer and impeller, pH probe & meter, glass frit sparger, bubbler with frit
CO_2 regulator, small CO_2 Cylinder

Test Summary: A charge of Li_2CO_3 is converted to $LiHCO_3$ using CO_2(g).

$CO2(g) + H2O \longrightarrow H_2CO_3(aq)$

$Li_2CO_3(s) + H_2CO_3(aq) \longrightarrow 2\ LiHCO3(aq)$

Procedure

A) Bicarbonate conversion

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solids therefore ensure that everything that is in contact with the reaction solutions and solids is clean.
2 Measure & record the tare of the reactor.
3 Prepare a 5.0% solids slurry of the Li_2CO_3 product in distilled water.

Solution %Solids, %	**5**	Mass of Li_2CO_3, g	**926.7**
		Mass of Distilled Water, g	**17607**

4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH of the stirred solution.
6 **Measure & record the initial mass of the CO_2 cylinder.**
7 Sparge the solution with CO_2(g) (~2.5 L/min)
8 Monitor pH and note any changes to the appearance of the solution.
9 Once pH stabilizes. Run for an additional 60min
10 Measure & record the final mass of the reactor and reaction slurry.
11 Vacuum filter the reaction slurry. Recover the filtrate.
12 Recover the filter cake - record wet mass and dry O/N in an oven (105 - 110 °C). When dry record the dry mass.
13 **Measure & record the final mass of the CO_2 cylinder.**

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
RT Bulk Li2CO3 PPT T1	RT Bulk BCB T1 HEAD	ANA2010-475	CA02504-JUL11
Filtrate	RT Bulk BCB T1 FIL	Li-AAS	
Residue	RT Bulk BCB T1 RES	Hold	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **RT Bulk BCB T1**

Project Technologist: A.Ferguson
Date of Test: 14-Jul-11

Data

A) Bicarbonate conversion

	Mass
Li_2CO_3 PPT Prod (FEED), g	926.7
Distilled Water, kg	17.61
CO_2 used, kg	1.328

	Mass, g		
Reactor	Gross	Net	CO_2
tare	**n/a**		
Initial	--	**n/a**	n/a
Final	-	**n/a**	**n/a**

Reaction Time, min	Sol'n T, °C	Sol'n pH	Flow CO_2(g), L/min	ORP	Comment/Observations
0		10.99	5	8	introducing CO2
1		10.42	5	52	T=0
5		9.58	5	115	
10		9.18	5	146	
15		8.87	5	169	
20		8.63	5	187	
25		8.40	5	203	
30		8.19	5	217	4mL of solids Visual check
45		7.74	5	186	
60		7.55	5	150	visual check .5mL of solids
90		7.54	5	156	visual check .5mL of solids
120		7.53	5	158	Test complete
Average/*Total*	#DIV/0!		5.0		

Reaction Slurry

	Solution			**Solids**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	18.66	1.0425	17.90					
Wash								
Residue				1630.0	1858.4	1798.1	228.4	168.1

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	milli	
Filtration time:	<10	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	<5	mm
Temperature of Drying Oven, °C	105	

Washing time:		min
Clarity of wash:		
Volume of wash:		mL
Colour of wash:		

Test No.:		BCB KIN T1					
Test Parameters		**T, °C**	**Final pH**	**Final ORP, mV**	**CO_2 Flow, L/min**	**Reactor dia/height ratio**	
		24	7.5	-71	2.5	2.24	
		Assays, % or mg/L					
	mass or vol, g or L	**Li**	Ca	Cu	Fe	K	Mg
RT Bulk BCB T1 HEAD	927	19.1					
RT Bulk BCB T1 FIL	18	7930					
RT Bulk BCB T1 RES	168	***19.8***					
IN (Elemental Masses)							
RT Bulk BCB T1 HEAD		177.0	0.000	0.00	0.00	0.00	0.00
Total IN, g		**177.0**	**0.000**	**0.00**	**0.00**	**0.00**	**0.00**
OUT (Elemental Masses)							
RT Bulk BCB T1 FIL		141.94	0.000	0.00	0.00	0.00	0.00
RT Bulk BCB T1 RES		***33.28***					
Total OUT, g		**175.23**	**0.000**	**0.00**	**0.00**	**0.00**	**0.00**
Relative Distribution							
RT Bulk BCB T1 FIL		81.0					
Residue		19.0	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
Total, %		**100.0**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**
%OUT/IN		99.0	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
% Extraction, %		***80.2***	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
Cal'd Head		18.9	0.000	0.000	0.000	0.000	0.000
% Mass Reduction, %	**81.9**						

Based on solution data and the initial Li IN value

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **RT Bulk BCB DC T1**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Feed **RT Bulk BCB T1 FIL**

Reagents Rocky Ridge Distilled Water

Apparatus 20 L reactor flask with lid, overhead mixer and impeller, heating mantle, YSI controller, T probe

Test Summary: A charge of LiHCO3 is heated to ~95°C to decompose the bicarbonate to the carbonate. The solids are recovered and washed with distilled water at 95 °C.

Procedure

Bicarbonate decomposition - Li_2CO_3 production

1 Add the blance of the bicarbonate solution to the 2 L reactor. Measure the initial pH of the solution.
3 Heat the solution to ~ 95 °C. Note the changes in the solution.
3 At the end of the reaction recover the solids as per procedure below.

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate and record its mass and solution density. When cool measure final pH.
Collect a sample of the filtrate for analysis .
Conduct 2 displacement washes (~75 mL Rocky H2O /wash @ ambient T).
Collect Wash fraction as one - record mass and solution density.
Remove the filter paper from the wet cake. Dry the solids in a clean glass container.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
PLS Head	RT Bulk BCB DC T1 HEAD		
Filtrate	RT Bulk BCB DC T1 FIL		
CMB WSH	RT Bulk BCB DC T1 CMB WSH		
Solids	RT Bulk BCB DC T1 PROD	ANA2010-0475	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **RT Bulk BCB DC T1**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Data
B) Bicarbonate decomposition - Li_2CO_3 production

Project Technologist: **A.Ferguson**
Date of Test: **26-May-11**

	Mass, kg
RT Bulk BCB T1 FIL	18.59
Solution Den., g/mL	1.0425
Li_2CO_3 Seed, g	--

	Mass, g	
Reactor	Gross	Net
tare		
Initial		
Final		

Reaction Time, min	Sol'n T, °C	Air L/min	Comment/Observations
10:20	75.0		Tried sparging air soln O/F b/c of excess CO2 and air
10:28	90.0	2.00	
11:11	95.0	2.00	
12:37	95.3		air Off tanking foaming
12:55	95.0		Test complete
	93.8		

	Solution			Solid	Gross Mass, g		Net Mass, g	
	Mass, kg	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
FILTRATE	15.32	1.0106	15.2					
CMB Wash	1.08	1.0059	1.1					
Solids				1626.0	2036.2	1928.5	410.2	302.50
Solids in Tank				1624.1		1788.6		164.5

Final Filtration:

Diameter of filtration paper:	185	mm
type of paper (Whatman ##):	1	
Filtration time:	<10	min
Clarity of filtrate:	cc	
Colour of filtrate:	ww	
Cake thickness:	40.0	mm
Temperature of Drying Oven, °C	110	

Washing time:	<1	min
Clarity of wash:	cc	
Volume of wash:	1	mL
Colour of wash:	ww	

Test No.:		RT Bulk BCB DC T1							
Test Parameters		Ave. T, °C							
		94							
		Assays, % or mg/L							
	mass or vol, g or L	Li	Ca	Fe	K	Mg	Na	S	Cl
RT Bulk BCB DC T1 HEAD	17.8	7930							
RT Bulk BCB DC T1 FIL	15.2	2110							
RT Bulk BCB DC T1 CMB WSH	1.1	1490							
RT Bulk BCB DC T1 PROD	467	18.9							
IN (Elemental Masses)									
RT Bulk BCB DC T1 HEAD		141.41	0.00	0.000	0.00	0.0000	0.00	0.00	0.00
Total IN, g		**141.41**	**0.00**	**0.000**	**0.00**	**0.0000**	**0.00**	**0.00**	**0.00**
OUT (Elemental Masses)									
RT Bulk BCB DC T1 FIL		31.99	0.00	0.00	0.00	0.0000	0.00	0.00	0.00
RT Bulk BCB DC T1 CMB WSH		1.60							
RT Bulk BCB DC T1 PROD		88.030	0.000	0.000	0.000		0.000	0.000	0.000
Total OUT, g		**121.62**	**0.00**	**0.0000**	**0.00**	**0.0000**	**0.00**	**0.00**	**0.00**
Relative Distribution									
RT Bulk BCB DC T1 FIL		26.3	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
RT Bulk BCB DC T1 CMB WSH		1.3	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
Residue		72.4	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
Total, %		**100.0**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**	**#DIV/0!**
%OUT/IN		86.0	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!
% Recovery, %		72.4	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!

Cal'd Head		6820							
% Mass Reduction, %									

Project No.: CALR-12607-002 **Project Technologist:** J. Condon
Client: Rock Tech **Date of Test:** 15-Jul-11
Test No.: **RT Bulk BCB T2**

Feed **RT L Fe Li2CO3 PPT T1**

Reagents CO_2(g) Rocky Ridge Distilled Water

Apparatus 2 L Reactor, overhead mixer and impeller, pH probe & meter, glass frit sparger, bubbler with frit
CO_2 regulator, small CO_2 Cylinder

Test Summary: A charge of Li_2CO_3 is converted to $LiHCO_3$ using CO_2(g).

CO2(g) + H2O ⟶ H_2CO_3(aq)
Li_2CO_3(s) + H_2CO_3(aq) ⟶ 2 LiHCO3(aq)

Procedure

A) Bicarbonate conversion

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solids therefore ensure that everything that is in contact with the reaction solutions and solids is clean.
2 Measure & record the tare of the reactor.
3 Prepare a 5.0% solids slurry of the Li_2CO_3 product in distilled water.

Solution %Solids, %	**5**	Mass of Li_2CO_3, g	**168**
		Mass of Distilled Water, g	**3192**

4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH of the stirred solution.
6 **Measure & record the initial mass of the CO_2 cylinder.**
7 Sparge the solution with CO_2(g) (~2.5 L/min)
8 Monitor pH and note any changes to the appearance of the solution.
9 Once pH stabilizes. Run for an additional 60min
10 Measure & record the final mass of the reactor and reaction slurry.
11 Vacuum filter the reaction slurry. Recover the filtrate.
12 Recover the filter cake - record wet mass and dry O/N in an oven (105 - 110 °C). When dry record the dry mass.
13 **Measure & record the final mass of the CO_2 cylinder.**

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
RT L Fe Li2CO3 PPT T1	RT Bulk BCB T2 HEAD		
Filtrate	RT Bulk BCB T2 FIL	Li - AAS	
Residue	RT Bulk BCB T2 RES		

Project No.: CALR-12607-002 **Project Technologist:** J. Condon
Client: Rock Tech **Date of Test:** 15-Jul-11
Test No.: **RT Bulk BCB T2**

Data

A) Bicarbonate conversion

	Mass, g
Stage PPT Prod (FEED)	168.0
Distilled Water	3192
CO_2 used	567

	Mass, g		
Reactor	Gross	Net	CO_2
tare	**3005.40**		
Initial	6363.40	**3358.00**	18.596
Final	6447.90	**3442.50**	**18.029**

Reaction Time, min	Sol'n T, °C	Sol'n pH	Flow CO_2(g), L/min	ORP	Comment/Observations
0	22.1	11.31	0	73	CO2 ON RPM=500
2	22.3	10.53	2.5	70	RPM=600 bubbles
5	22.8	9.48	2.5	100	
10	23.7	8.79	2.5	104	
15	24.4	8.40	2.5	105	
20	24.9	8.15	2.5	107	
30	25.8	7.80	2.5	113	
45	26.2	7.58	2.5	117	
60	26.1	7.50	2.5	115	
90	25.5	7.48	2.5	107	
120	24.9	7.43	2.5	97	
Average/*Total*	24.7		2.5		

Reaction Slurry

	Solution			**Solids**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
Filtrate	3411.20	1.0500	3249					
Wash								
Residue				1629.5	1643.6	1639.8	14.1	10.3

Final Filtration:

Diameter of filtration paper:	millipore	mm
type of paper (Whatman ##):	125	
Filtration time:	<10	min
Clarity of filtrate:	clear	
Colour of filtrate:	ww	
Cake thickness:	<10	mm
Temperature of Drying Oven, °C	110	

Washing time:		min
Clarity of wash:		
Volume of wash:		mL
Colour of wash:		

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **RT Bulk BCB DC T2**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Feed **RT Bulk BCB T1 FIL**

Reagents Rocky Ridge Distilled Water

Apparatus 3 L reactor flask with lid, overhead mixer and impeller, heating mantle, YSI controller, T probe

Test Summary: A charge of LiHCO3 is heated to ~95°C to decompose the bicarbonate to the carbonate. The solids are recovered and washed with distilled water at 95 °C.

Procedure

Bicarbonate decomposition - Li_2CO_3 production

1 Add the blance of the bicarbonate solution to the 2 L reactor. Measure the initial pH of the solution.
3 Heat the solution to ~ 95 °C. Note the changes in the solution.
3 At the end of the reaction recover the solids as per procedure below.

Final Reaction Slurry:

Filter the reaction slurry.
Recover the filtrate and record its mass and solution density. When cool measure final pH.
Collect a sample of the filtrate for analysis .
Conduct 2 displacement washes (~75 mL Rocky H2O /wash @ ambient T).
Collect Wash fraction as one - record mass and solution density.
Remove the filter paper from the wet cake. Dry the solids in a clean glass container.

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
PLS Head	RT Bulk BCB DC T2 HEAD		
Filtrate	RT Bulk BCB DC T2 FIL		
CMB WSH	RT Bulk BCB DC T2 CMB WSH		
Solids	RT Bulk BCB DC T2 PROD	ANA2010-0475	

Project No.: CALR-12607-002
Client: Rock Tech
Test No.: **RT Bulk BCB DC T2**

Project Technologist: A.Ferguson
Date of Test: 26-May-11

Data
B) Bicarbonate decomposition - Li_2CO_3 production

Project Technologist: **A.Ferguson**
Date of Test: **26-May-11**

	Mass, g
RT Bulk BCB T1 FIL	3406.20
Solution Den., g/mL	1.0507
Li_2CO_3 Seed, g	

	Mass, g	
Reactor	Gross	Net
tare	**3063.00**	
Initial	6469.20	**3406.20**
Final	6373.00	**3310.00**

Reaction Time, min	Sol'n T, °C	Sol'n pH	Comment/Observations
0	24.4	7.43	Heat On
30	80.0		
36	90.0		
41	92.0		
T=0	93.0		
10	95.9		
60	95.6		Adding air at 5L/min
100	95.1	9.56	Test complete

	Solution			**Solid**	**Gross Mass, g**		**Net Mass, g**	
	Mass, g	SG, g/mL	Vol, mL	Tare	Wet	Dry	Wet	Dry
FILTRATE	3052.4	1.008	3028					
CMB Wash	381.1	1.0058	379					
Solids				529.2	672.5	649.2	143.3	120.00

Final Filtration:

Diameter of filtration paper:	125	mm
type of paper (Whatman ##):	1	
Filtration time:	<10	min
Clarity of filtrate:	clear	
Colour of filtrate:	ww	
Cake thickness:	<100	mm
Temperature of Drying Oven, °C	110	

Washing time:	<10	min
Clarity of wash:	clear	
Volume of wash:	380	mL
Colour of wash:	ww	

Project No.: CALR-12607-002 **Project Technologist:** A. Ferguson
Client: Rock Tech **Date of Test:** 25-Jul-11
Test No.: **RT Bulk BCB T3**

Feed **RT BCB DC T1 Product and Tank Solids**

Reagents CO_2(g) Rocky Ridge Distilled Water

Apparatus 2 L Reactor, overhead mixer and impeller, pH probe & meter, glass frit sparger, bubbler with frit
CO_2 regulator, small CO_2 Cylinder

Test Summary: A charge of Li_2CO_3 is converted to $LiHCO_3$ using CO_2(g).

$CO2(g) + H2O \longrightarrow H_2CO_3(aq)$

$Li_2CO_3(s) + H_2CO_3(aq) \longrightarrow 2\ LiHCO3(aq)$

Procedure

A) Bicarbonate conversion

1 Ensure that all test equipment (i.e. reactor, impeller, filter flask & funnel, etc.) have been rinsed with a dilute acid solution and then with deionized water. The objective of the test is to purify the solids therefore ensure that everything that is in contact with the reaction solutions and solids is clean.
2 Measure & record the tare of the reactor.
3 Prepare a 5.0% solids slurry of the Li_2CO_3 product in distilled water.

Solution %Solids, %	**5**	Mass of Li_2CO_3, g	**455.5**
		Mass of Solution, g	**8655**

4 Measure & record the mass of the reactor and solution.
5 Measure & record the pH of the stirred solution.
6 **Measure & record the initial mass of the CO_2 cylinder.**
7 Sparge the solution with CO_2(g) (~2.5 - 5 L/min)
8 Monitor pH and note any changes to the appearance of the solution.
9 Once pH stabilizes. Run for an additional 60min
10 Measure & record the final mass of the reactor and reaction slurry.
11 Vacuum filter the reaction slurry. Recover the filtrate.
12 Recover the filter cake - record wet mass and dry O/N in an oven (105 - 110 °C). When dry record the dry mass.
13 **Measure & record the final mass of the CO_2 cylinder.**

Solution	Mass, g
DI H_2O	**6425**
RT BCB DC T1 FIL	**1730**
RT BCB DC T1 WSH	**500**
	8655

Analytical Requirements:

Sample	Sample ID	Assay	LIMS
RT BCB DC T1 Product and Tan	RT Bulk BCB T3 HEAD		
Filtrate	RT Bulk BCB T3 FIL	Li - AAS	
Residue	RT Bulk BCB T3 RES		