The Raleigh Lake Project

NI 43-101 Technical Report - PEA

January 18, 2024

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The business of sustainability

ERM

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ERM Consultants Canada Ltd.

120 Adelaide Street West, Suite 2010
Toronto, ON
Canada, M5H 1T1

T: +1 416 646 3608
F: +1 416 642 1269

CONTENTS

List of Tables

List of Figures

1. SUMMARY

Environmental Resource Management ("ERM") was retained by International Lithium Corp. ("ILC" or the "Company") to prepare a Preliminary Economic Assessment ("PEA") in accordance with National Instrument 43 101 (NI 43-101) for the Raleigh Lake Project (the "Project") located approximately 25 km western Ignace, Ontario, Canada.

The report was prepared by the independent consultant of Nordmin Engineering Ltd., Christian Ballard P.Geo., and consultants of ERM: Garth Liukko, Nigel Fung, Richard Wagner, Carlos Tapia Cardenillas, Efrain Ugarte, Pim Van Geffen, Kayvan Samadani, Rolf Schmitt, Mark Welsh, Nikolett Kovacs, and Associate Consultant of ERM Georgi Doundarov.

All measurement units used in this Technical Report are metric unless otherwise noted.

The MRE for the Project conforms to industry best practices and is reported using the 2014 CIM Definition Standard for Mineral Resources and Mineral Reserves and 2019 CIM Best Practice Guidelines. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

Mineral Resources were classified into Indicated and Inferred categories based on geological and grade continuity, in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, kriging variance, and search volume block estimation assignment.

1.1 Property Description and Location

The Raleigh Lake Project, known as the "Project," is roughly 25 kilometres west of Ignace and 235 kilometres west of Thunder Bay in the northwestern part of Ontario within the Kenora Mining District. UTM defines the Project's central point coordinates 576550mE/5473800mN (EPSG: 26915: UTM NAD83 Zone 15N).

Initially, ILC acquired fifty-five single-cell mineral claims ("SCMC"), covering around 1,976 hectares of mineral-rich land in the Project area. In October 2018, ILC continued to expand its claims through online staking and obtained an additional fifty SCMCs from Perry English. These newly acquired claims were adjacent to the southwest of the existing claim group, resulting in a total claim area of 3,025 hectares. ILC pursued an aggressive land acquisition strategy following the success of their initial drilling program in the 3rd and 4th quarters of 2021 and into 2022. This strategy led to a significant expansion of their land holdings through several phases. ILC increased their total tenure from 3,025 to 48,500 hectares, consisting of 2,308 unencumbered SCMCs. The removal of three SCMCs from one claim block due to surface rights ownership reduced the total to 2,305 cells. These claims are divided into two separate, non-contiguous blocks known as the Raleigh Lake/White Otter and Owl Lake blocks.

The company is not aware of any specific risks or factors that could impact its claim ownership, private property rights, or the permitting process for the Project.

1.2 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

Current access to The Project site is convenient via watercraft from Raleigh Lake or through well-maintained logging roads south of Highway 17 (Trans-Canada Highway) along Doreen Lake Road.

Water access consists of driving 25 kilometres west of Ignace on Highway 17 to Raleigh Lake Road, which leads one kilometre south to the Raleigh Lake shoreline and Raleigh Lake Outpost and Resort. From there, watercraft can navigate to the northern and eastern parts of the claim group, a helpful option for frequent access to the northernmost property, especially when supported by nearby lodges or cabins during the summer months.

Vehicular access is obtained by driving 3.8 kilometres west on Raleigh Lake Road and then south from Highway 17 onto Doreen Lake Road. Travel 8.7 kilometres south to the George Lake junction, continue left for 3.1 kilometres, and then turn left again (east) onto Trent Road (logging road 46-02). Approximately 1.5 kilometres from this junction, an old logging road branches off Trent Road to the left (north) and provides easy access to the Project's primary pegmatite occurrences on claim cell 158259. In addition, Highway 17 and the CP rail line serve as key transportation routes for both truck and trailer traffic, facilitating train services with links to eastern and western Canada, as well as southward to the USA.

The property is conveniently situated 25 km west of Ignace, a small town in the Kenora District of Northwestern Ontario with a population of 1,202 (2016 Census). Ignace lacks sufficient support services and skilled labour for mining and specialized exploration, making it more practical to source such services and equipment from larger nearby towns like Dryden or even bigger centers such as Thunder Bay, Ontario (250 km to the east) and Winnipeg, Manitoba (425 km to the west). The nearest airport is 100 km west in Dryden, providing connecting flights to major Canadian cities and serving as a gateway to international destinations.

The property is located within the Southern Boreal Shield climate of Canada, so the climate is characterized by long, cold winters and short, warm summers. This temperate, mid-latitude continental environment allows for year-round field operations without access restrictions. Mid-winter temperatures average around -15°C, while mid-summer temperatures hover at approximately 17°C.

The topography conforms to a Canadian Shield paleo-glacial terrain, ranging from generally flat low-lying swamps to slightly undulating areas with notable hills. The project's topographic relief is roughly 50 metres, with elevations ranging from about 450 metres along the lakeshore to crests and ridges as high as 500 metres in select property areas.

The region's characteristic vegetation includes trembling aspen, paper birch, white and black spruce, and balsam fir. Colder and wetter areas support the growth of black spruce and tamarack. Notable wildlife species in the area include moose, black bear, wolf, lynx, snowshoe hare, and woodchuck. Bird species encompass ruffed grouse, woodpecker, bald eagle, herring gull, and waterfowl. Forestry, recreation, fishing, and hunting are the primary land uses in this region.

1.3 Project History

Historical exploration work in the Raleigh Lake area has primarily focused on greenstone-hosted gold and base metal mineralization. Exploration by Avalon Ventures in the early 2000s concentrated solely on the tantalum potential of the property, neglecting its lithium potential. The first spodumene-bearing pegmatite, known as the Johnson Pegmatite, was discovered in 1966 by Stan Johnson. The Ontario Geological Survey (OGS) conducted further exploration in the Raleigh Lake pegmatite field, as part of a broader investigation into granite-related mineralization in the Superior Province. Avalon Ventures Ltd. and later Abaddon Resources Ltd. continued delineating the main pegmatites on the property for tantalum exploration, using magnetometer surveys and drilling programs in the 1990s and early 2000s.

International Lithium Corp. acquired the Project in 2016 from Robert Fairservice and has since worked to identify the Raleigh Lake pegmatite field's lithium potential. On April 13, 2023, a maiden Technical Report supporting the disclosure of Mineral Resources was prepared by Nordmin Engineering Ltd. entitled “NI 43-101 Technical Report and Mineral Resource Estimate” at The Raleigh Lake Lithium Project, Ontario, Canada.

1.4 Geology and Mineralization

The Raleigh Lake property can be found in the Wabigoon Subprovince of the Canadian Shield's Archean Superior Province. Within this region, a greenstone belt primarily consisting of mafic metavolcanic rocks can be encountered. There are also some intermediate to felsic metavolcanic and volcaniclastic rocks, gabbros, and their corresponding metasedimentary equivalents, although they are less prevalent.
The property's geological composition is mainly composed of Archean supracrustal rocks, including mafic metavolcanics and their metasedimentary counterparts. These rocks both overlay and are intruded by various granitic plutons and batholiths of different ages. One notable feature is the peraluminous Revell Lake Batholith, which is of the S-type variety.

The Raleigh Lake Greenstone Belt, consisting of supracrustal volcanic rocks, stretches southeastward for over 50 km into the central Wabigoon Subprovince. Its eastern boundary is defined by the granitic White Otter and Indian Batholiths. To the southwest of the Raleigh Lake Deposit, you'll find the elongated Revell Batholith, which trends northwest. This batholith's composition changes from tonalite to a muscovite-biotite "two-mica" granite along its southeastern edge. The older two-mica granite phase predates the formation of the rare element pegmatites in the Raleigh Lake pegmatite field.

These geological features suggest the presence of dome and basin fold structures, especially in the Raleigh Lake vicinity. These structures likely result from shallow-dipping layers and sills, and they create topographic variations in the area. Most of the pegmatite occurrences in this region trend north-northeast and have moderate easterly dips, ranging from 25° to 40° (Barclay, 2001).

The rare metal-bearing pegmatite dykes on the Raleigh Lake property occur in a south-southeast striking zone approximately 1.5 kilometres wide and at least 4 kilometres long, with a trend of tantalum mineralization in albitic dykes occurring south of Raleigh Lake. The main pegmatite trend (which includes Pegmatite 1 through 3) and the Johnson Pegmatite belong to the albite-spodumene sub-type of rare metal pegmatites. These pegmatites are at least partially hosted within a sheared, coarse-grained gabbro. Most pegmatites trend north-northeast and dip moderately or shallowly to the east. The dykes are rich in K-feldspar-albite and include secondary cleavelandite, quartz, and spodumene. Accessory minerals identified in the field and drilling include microlite, tantalite, and bismuthinite.

The known pegmatites form shallowly to moderately dipping, north-northeast trending undeformed sheets with extensive lateral continuity. Strong fractionation of contained minerals and weak zonation within the pegmatites suggest that strongly enriched rare-metal zonation may exist within other domains of the pegmatites. The pegmatites are hosted in both mafic and intermediate volcanic rocks. Crude zoning is evident in the wider pegmatites with albitic “wall” zones and “core” zones of albite-quartz-muscovite or spodumene-K‐feldspar-albite. Average lithium grades range from 1.0% to 2.7% locally across the true width of the pegmatite veins.

The Project falls under the classification of rare-metal pegmatites, further categorized as the albite-spodumene sub-type. Pegmatites are a common plutonic rock that contains abundant crystals with skeletal, graphic, or strongly directional growth habits or anisotropic layered mineral fabrics. Giant or megacrystic crystals can also be present, and the rare-element pegmatites contain anomalous and elevated Be, Li, Ta, Sn, and Cs. The Project contains lithium-caesium-tantalum (“LCT”) pegmatites enriched in Li, Cs, Ta, Be, B, F, P, Mn, Ga, Rb, Nb, Sn, and Hf. Other examples of LCT pegmatite deposits include Tin Mountain in the United States, Tanco in Canada and Wodgina and Pilgangoora in Western Australia. They are formed by fractional crystallization of an incompatible element-enriched granitic melt.

LCT pegmatites most likely form in orogenic hinterlands related to plate convergence and are consequently hosted in metamorphosed supracrustal rocks such as greenstone belts. The Raleigh Lake Project is hosted within the western Wabigoon subprovince and emplaced upon the Winnipeg River and Marmion greenstone terranes.

1.5 Exploration

In 2016, the company initiated exploration efforts by combining regional and property-scale lithogeochemical bedrock sampling with an unmanned airborne magnetometer survey. These activities in 2016 successfully verified the presence of lithium and rare-element enriched pegmatites across the property. Building on the achievements of the 2016-2018 initiatives, ILC carried out additional lithogeochemical sampling, biogeochemical sampling of tree bark, a prospecting and mapping program, and more outcrop channel sampling during the 2021 season. This work led to the confirmation of extensions of pegmatite deposits and lithogeochemical anomalies. The drilling targets for 2021 and the three-phase 2022 program were determined based on historical exploration findings and initial investigations conducted by the company.

Between 1999 and 2010, a total of 16 diamond drill holes were drilled on the property, covering a distance of 2,817 metres. In 2021 and 2022, ILC completed diamond drilling in 65 drill holes, with a combined length of 11,003 metres, as part of their drilling programs. Follow-up activities for 2023 are currently in the planning stages, and they will primarily involve infill and expansion drilling on the main pegmatite deposits spread throughout the property.

1.6 Data Verification, Sampling Preparation, Analysis, and Security

Drill holes conducted in programs from 1999 to 2010 have been incorporated into the current Mineral Resource Estimate (MRE) database. The sampling methods employed in these programs generally involved sampling the visually identified pegmatite veins corresponding to significant geological units and examining various adjacent materials.

Surface diamond drilling programs carried out by International Lithium Corp. in 2021 and 2022 were executed by Rodren Drilling. Sampling was aligned with major lithological units and was selectively based on the contacts between Lithium-Cesium-Tantalum (LCT) pegmatites, which is appropriate for this type of deposit as these contacts are distinct, and the surrounding host rock contains negligible amounts of rare elements. For quality control, one blank, one standard, and one duplicate sample were inserted in every batch of thirty samples. All samples were sent to Actlabs in Dryden, ON, contained in rice bags and batched with 5 to 15 samples. Lithium and other elements were analyzed using Peroxide Fusion ICP/MS (inductively coupled plasma mass spectrometry), referred to as "ultratrace-7" at Actlabs. In the event of any failures, sample batches were subjected to re-analysis.

To maintain the integrity of the samples, sample bags were securely sealed with zip ties and shipped for analytical analysis. The drill core is stored either in outdoor core racks or cross-piled at the project site. The Qualified Person (QP) believes that all parties' sample preparation, security measures, and analytical procedures conform to standard industry practices and that the data is suitable for the 2023 MRE.

Records of core samples, lithologic logs, laboratory reports, and relevant drill hole details for all drilling programs conducted between 1999 and 2022 were digitally compiled for use in Leapfrog Geo® and Datamine Studio RM® to create a geological model and MRE. Both historical and current drilling program information underwent a comprehensive review.

Drill hole data from 2021 to the present has been compiled into Geospark® and exported directly into comma-separated values tables. Nordmin reviewed the database using Excel®, Datamine Studio RM®, and QGIS® software and did not identify any errors or discrepancies within the drill hole database.

Regarding quality assurance, the QP conducted a spot check verification on the Project, covering two drill holes that included all main lithologies and 8% of the assays.

1.7 Mineral Processing and Metallurgical Testing

The Raleigh Lake Orebody contains two metallurgical domains, the lithium spodumene domain and the rubidium microcline domain. These two separate domains represent zones in the Raleigh Lake orebody that require customized process flowsheets to be developed for each zone. For the lithium domain, the objective is the recovery of spodumene to 6% Li_2O concentrate grade, while the rubidium bearing microcline domain objective is to develop a flowsheet for extraction of the rubidium from the microcline.

The current focus was to perform mineralogy and mineral processing testing to develop the flowsheet for the lithium zone (Li-Head) and do a literature review to begin to investigate the flowsheet development of the rubidium zone (Rb Head). Samples of the lithium and rubidium domains were sent to SGS Canada in August of 2023, to perform phase one mineralogy tests with follow-up mineral processing testing and literature review. The Li Head and Rb Head were collected from the Raleigh Lake Deposit and were received by the SGS Lakefield Canada Advanced Mineralogy Facility for mineralogy. Mineralogy was conducted to determine liberation, mineral assemblages which would help to support and guide the metallurgical testwork.

Spodumene is the main lithium mineral in the sample. Spodumene is well liberated (94%) at this grind target (P_{80} of ca. 400 µm).

- Gangue silicates are also well liberated.
- Therefore, spodumene may be recovered by gravity or flotation at relatively coarse-grained particle sizes.
- Based on mineralogy, theoretically, a high purity lithium concentrate is achievable.
- Spodumene hosts greater than 98% of the total lithium in the sample.
- A pure spodumene concentrate is expected to have approximately 3.5% Li.

The Mineralogy Summary of the Microcline Domain - Rb-head:

- K-feldspars host an average of 1.42% Rb_2O and 0.07% Cs_2O. A pure feldspar concentrate is expected to have a chemical composition as the average chemistry of the feldspar from the EPMA.
- Muscovite averages 1.90% Rb_2O 0.19% Cs_2O.
- K-feldspars hosts ca. 98% of the total Rb and 95% of the total Cs in the sample.
- K-feldspars are well liberated (92%), while middling with plagioclase are minor (7%). Therefore, theoretically, a high purity K-feldspar concentrate is achievable (for the P_{80} of 700 µm).

The lithium Li head sample assayed 1.59% Li_2O and 0.56% Fe_2O_3, while the rubidium head sample graded 6,580 g/t Rb (equivalent to 0.72% Rb_2O) with 0.12% Li_2O and 0.24% Fe_2O_3.

The main objective of the phase one scoping level mineral processing test investigation was to provide a preliminary indication of the lithium beneficiation of the Li head by heavy liquid separation (HLS). The metallurgical target was the preparation of spodumene concentrate grading >6.0% Li_2O while maximizing lithium recovery.

The pegmatite Li Head sample was initially stage-crushed to 100% passing 12.7 mm, homogenized, and split into 10 kg test charges. One of the 10 kg charges was sub-sampled 500 g for head assays and the remaining was screened at 16 mesh to remove the -1 mm fraction for mineralogy.

From the 10kg charges, the minus 12.7 mm +1 mm fraction was further screened at ¼" (6.3 mm) to generate two fractions of -12.7 mm +6.3 mm and -6.3 mm +1 mm. The two coarse fractions, -12.7 mm +6.3 mm and -6.3 mm +1 mm, were submitted for Heavy Liquid Separation (HLS) testing.

The HLS testing results at SG 2.85, 6.0% Li_2O concentrate grade of 14.9 weight % with global lithium recovery of 53.0% was obtained in the fraction of -12.7 mm/+6.3 mm.

The HLS Testing results interpolated to SG 2.83, a 6.0% Li_2O concentrate grade of 8.0 wt% with a global lithium recovery of 28.5 % was obtained in the -6.3 mm/+1 mm fraction.

Combining the 6% Li_2O concentrates from the two fractions of -12.7 mm/+6.3 mm and 6.3 mm/+1 mm (highlighted in cyan in Table 1-1) generated a combined global lithium recovery of 81.5%.

Table 1-1: Summary of HLS Global Mass Balance (Interpolated @ 6.0% Li_2O)

Product		HLSG	Weight	Assays (%)			Distribution (%)	
		g/cm^3	%	Li	Li_2O	Fe_2O_3	Li	Fe_2O_3
-12.7 + 6.3 mm	HLS Concentrate	2.85	14.9	2.79	**6.00**	**0.91**	**53.0**	26.4
	HLS Middling	-2.85 +2.70	8.21	0.78	1.69	0.66	8.21	10.6
	HLS Tailings	2.70	32.1	0.042	0.089	0.28	1.70	17.6
-6.3 +1 mm	HLS Concentrate	2.83	8.01	2.79	**6.00**	**1.06**	**28.5**	16.5
	HLS Middling	-2.83 +2.70	3.07	0.59	1.27	0.99	2.32	5.90
	HLS Tailings	2.70	22.1	0.022	0.047	0.31	0.61	13.3
Fines Fraction (-1 mm)			11.6	0.38	0.82	0.43	5.65	9.72
Head (calc.)			**100**	**0.78**	**1.69**	**0.52**	**100**	**100**
Head (dir.)				**0.74**	**1.59**	**0.56**		
Flotation Feed			**22.9**	**0.55**	**1.19**	**0.59**	**16.2**	**26.2**

Above a tailings SG-cut point of 2.70, the HLS middling from each sample contained between 1.27 – 1.69% Li_2O with 2.3 – 8.2% of the global lithium distribution. Therefore, the HLS middling can potentially be stage crushed then mixed with the minus 1 mm fines fraction to produce a flotation feed (or gravity feed) grading 1.19 % Li_2O and 0.59% Fe_2O-$_3$.

The combined HLS middling and fines fraction contained 16.2% of the lithium distribution graded 1.19% Li_2O. This is potential feed for a rolls crusher and screening, for flotation feed, or a ultrafines DMS gravity circuit to increase the lithium recovery.

The grade of iron in the spodumene concentrate was ~1% Fe_2O_3, which is acceptable, however, this would likely be reduced by treating the concentrate by magnetic separation.

1.8 Mineral Resource Estimates

The maiden MRE for the Project as presented in this report has been prepared in accordance with NI 43-101 standards. The Mineral Resource Estimate was estimated from the main drill hole database comprised of 13,821 m of diamond drilling from 81 drill holes completed between 1999 and the end of 2022.
Guided implicit wireframing was completed using Leapfrog Geo™ for the purpose of modelling the Lithium Pegmatite and Rubidium Domains. A Low-Grade Background Domain wireframe was created to encapsulate all other Domains and Zones. Block model was defined with parent blocks at 5.0 m x 5.0 m x 5.0 m (Northing x Easting x Elevation).

All wireframe volumes were filled with blocks from the prototype (which used the parameters in Table 14-7). Block volumes were compared to all wireframes and were found to be within reasonable tolerance limits. Sub-blocking was allowed to maintain the geological interpretation and to accommodate the Domain and Zone wireframes, the SG, and the category application. The block model parent block size was defined as 5.0 m x 5.0 m x 5.0 m blocks, which are sub-blocked to a minimum size of 0.625 m x 0.625 m in the N-S and E-W directions with a variable elevation calculated based on the other sizes.

The MRE was conducted using Datamine Studio RM™ version 11. 3.2020 within NAD83 UTM Zone 15N.

Three models were independently estimated, one each for the Lithium Pegmatite, Rubidium, and Low-Grade Background Domains. These block models were combined into one overall resource block model.

The Mineral Resources were classified using the 2014 CIM Definition Standards and the 2019 CIM Best Practice Guidelines and have an effective date of February 16, 2023. The Project hosts:

The combined Mineral Resources for lithium, considering both open pit (with a 650 ppm lithium cut-off) and underground (with a 2,000 ppm lithium cut-off) sources, consist of 2,293 thousand tonnes categorized as Measured and Indicated Mineral Resources, with an average lithium grade of 2,976 ppm. Additionally, there are 3,902 thousand tonnes classified as Inferred Resources, with an average lithium grade of 2,691 ppm.

Open Pit Lithium Mineral Resources include 2,101 thousand tonnes of Measured and Indicated Mineral Resources grading 2,956 ppm lithium, and 3,247 thousand tonnes of Inferred Resources grading 2,595 ppm lithium.

Underground Lithium Mineral Resources include 192 thousand tonnes of Measured and Indicated Mineral Resources grading 3,192 ppm lithium, and 655 thousand tonnes of Inferred Resources grading 3,162 ppm lithium.

Total Rubidium Open Pit (at a 4,000 ppm rubidium cut-off) and Underground (at a 4,000 ppm rubidium cut-off) Mineral Resources include 133 thousand tonnes of Measured and Indicated Mineral Resources grading 6,163 ppm rubidium, and 123 thousand tonnes of Inferred Resources grading 4,224 ppm rubidium.

Open Pit Rubidium Mineral Resources include 95 thousand tonnes of Measured and Indicated Mineral Resources grading 6,036 ppm rubidium, and 18 thousand tonnes of Inferred Resources grading 3,005 ppm rubidium.

Underground Rubidium Mineral Resources include 38 thousand tonnes of Measured and Indicated Mineral Resources grading 6,484 ppm rubidium, and 106 thousand tonnes of Inferred Resources grading 4,427 ppm rubidium.

Table 1-2 and Table 1-3 shows the Mineral resources based on validated results of 81 surface diamond drill holes, for a total of 13,821 m of an effective date of February 16, 2023.

Table 1-2: Mineral Resource Estimate, Open Pit (650 g/t Li Cut-off) and Underground (2000 g/t Li Cut-off) (Source: Nordmin, 2023)

Area	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
Open Pit *650ppm Li Cut-off*	Measured	80	3,887	0.84%	313
	Indicated	2,021	2,919	0.63%	5,897
	Measured + Indicated	2,101	2,956	0.64%	6,210
	Inferred	3,247	2,595	0.56%	8,427
Underground *2,000ppm Li Cut-off*	Measured	3	2,560	0.55%	8
	Indicated	189	3,203	0.69%	606
	Measured + Indicated	192	3,192	0.69%	614
	Inferred	655	3,162	0.68%	2,073
Total	**Measured + Indicated**	**2,293**	**2,976**	**0.64%**	**6,824**
	Inferred	**3,902**	**2,691**	**0.58%**	**10,499**

Mineral Resource Estimate Notes:

Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The MRE is developed with data from diamond drill holes totaling 13,821 m.

The pit constrained mineral resources were defined using a parented block model, within an optimized pit shell with average pit slope angles of 45° in rock and 30° in overburden, a 9.8 strip ratio (waste material: mineralized material) and a revenue factor of 1.0. The pit optimization shells were created using Deswik.AdvOPM software.

The lithium resource pit optimization parameters include: 5.5% Li_2O spodumene concentrate; US$1,800 Li_2O spodumene concentrate price; exchange rate of C$1.3/US$1; concentrate transportation and offsite charges of C$175/t, mining cost of C$6/t, processing plus general and administration cost of C$41/t; and a process recovery of 75%. Only lithium value was used to generate the resource optimized pit shell.

Underground constrained mineral resources were defined within 5.0 x 5.0 x 5.0 m minable shape optimization wireframes. The mineable shape optimization constraining wireframes were created using Deswik.SO software.

The lithium resource underground minable shape optimization parameters include: 5.5% Li_2O spodumene concentrate; US$1,800 Li_2O spodumene concentrate price; exchange rate of C$1.3/US$1; concentrate transportation and offsite charges of C$175/t, mining cost of C$80/t, processing plus general and administration cost of C$50/t; and a process recovery of 75%.

A default density of 2.668 g/cm^3 was used for the mineralized zones.

All figures are rounded to reflect the relative accuracy of the estimates; totals may not add correctly.

Table 1-3: Rubidium Open Pit and Underground Mineral Resource Estimate (Source: Nordmin 2023)

Area	Resource Category	Mass (kt)	Grade		Contained Rb (t)
			Rb (ppm)	Rb_2O (%)	
Open Pit *4,000ppm Rb Cut-off*	Measured	5	5,412	0.59%	29
	Indicated	90	6,073	0.66%	547
	Measured + Indicated	95	6,036	0.66%	576
	Inferred	18	3,005	0.33%	53
Underground *4,000ppm Rb Cut-off*	Measured	5	6,547	0.72%	35
	Indicated	33	6,474	0.71%	211
	Measured + Indicated	38	6,484	0.71%	246
	Inferred	106	4,427	0.48%	468
Total	**Measured + Indicated**	**133**	**6,163**	**0.67%**	**822**
	Inferred	**123**	**4,224**	**0.46%**	**521**

Mineral Resource Estimate Notes

Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The MRE is developed with data from diamond drill holes totaling 13,821 m.

The pit constrained mineral resources were defined using a parented block model, within an optimized pit shell with average pit slope angles of 45° in rock and 30° in overburden, and a revenue factor of 1.0. The constraining optimized pit shell is based on lithium value only. The pit optimization shells were created using Deswik.AdvOPM software.

Underground constrained mineral resources were defined within 5.0 x 5.0 x 5.0 m minable shape optimization wireframes. The mineable shape optimization constraining wireframes were created using Deswik.SO software.

The rubidium open pit and underground resource estimate was constrained above market value due to the current limited world market. A 4,000 ppm rubidium cut-off grade was selected. The rubidium resource was excluded from (i.e., neither taken into account nor used as a credit for) the underground and open pit lithium resource.

A default density of 2.668 g/cm^3 was used for the mineralized zones.

All figures are rounded to reflect the relative accuracy of the estimates; totals may not add correctly.

1.9 Mining Methods

The mining method selected for this project will use traditional open pit drilling and blasting followed by load and haul. The primary mining production will be executed using hydraulic excavators, front shovels, and/or wheel loaders as appropriate to the terrain and depending on the major production equipment available for the project. The material will be hauled from the bench to the crusher, ROM stockpiles or waste dump depending on the material type. Furthermore, ancillary equipment, such as bulldozers, graders, and a range of vehicles, is employed to perform functions related to maintenance, support, services, and utilities.

The proposed PEA level mine plan is based around work at a proposed plant feed production rate of 540,000 tpy.

This LoM mine plan is proposed to mine 57Mt of material over the mine life, which will be comprised of 4Mt of mill feed and 53Mt of waste with an average strip ratio of 13.2:1.

1.10 Recovery Methods

The lithium zone flowsheet development testwork showed a viable flowsheet to crush to minus 12.7 mm and screen at 1 mm, followed by screening again at 6.3 mm to make two streams (-12.7 mm plus 6.3 mm and –6.3 mm plus 1mm) for HLS (plant DMS). The minus 1 mm fines and HLS middlings (DMS plant middlings) would be stored for future processing and recovery of additional lithium, and other minerals of interest. HLS floats (DMS floats) may also be considered for road construction projects.

Work remains to be done on the market size and demand for rubidium notwithstanding its current high market price of USD1,140 per kilogram (see https://www.metal.com/Other-Minor-Metals/202012250004). Consequently, planning work has yet to be done on the processing of the rubidium zone, and therefore will not be mentioned in this section.

1.10.1 Process Design Criteria

The base case process plant is designed to crush 1,500 tpd and process 1,500 tpd in a DMS plant to produce a nominal 56,000 tpy of 6% Li_2O at 81% recovery (Table 1-4).

Table 1-4: Design Operating Parameters

Parameters	Nominal	Designed	Units
General Plant			
Operating Schedule			
Operating hours per day	24	24	h
Annual operating days	365	365	days/y
Shifts/day (Crushing and Wet Plant)	3 x 8	3 x 8	shifts x hrs
Equipment utilization – Crushing Plant	68	68	%
Equipment utilization – Wet (DMS) Plant	85	85	%
Material Characteristics			
Plant Feed			
Feed – Crushing Plant	1,500	1,500	tpd
Feed – Wet (DMS) Plant	1,500	1,500	tpd
Solids %	97	97	%
Plant Product			
Concentrate Production	56,000	56,000	tpy
Concentrate Recovery	81	81	%
Concentrate Grade (Li_2O)	6	6	%

Engineering and design were developed to a scoping level based on the results of the SGS laboratory testing. The SGS lab tests obtained 22.9 weight percentages of 6% Lithium Concentrate and estimated 81% lithium recovery.

A design factor of 10 % is applied on nominal requirements to ensure that the process equipment has enough capacity to take care of the expected feed variation.

1.10.2 Process Description

The -12.7 mm / +1.0 mm material will report to the DMS coarse sizing screen where it will be screened at 6.3 mm to produce:

- A coarse fraction (-12.7 mm / +6.3 mm) which reports to the primary coarse DMS.
- A fines fraction (-6.3 mm / +1.0 mm) which reports to the primary fines DMS via a REFLUX™ classifier.

The coarse and fine DMS circuits will consist of primary DMS cyclones (SG 2.65 to 2.7) and secondary DMS cyclones (SG 2.85) to efficiently separate spodumene from the gangue material to produce a 6.0% Li_2O or higher concentrate grade. Mica is proposed to be removed from the fines stream by a REFLUX™ classifier, prior to feeding the DMS fines preparation screen.

Prior to feeding the primary DMS cyclones, each ore stream (coarse and fine) will be mixed with ferrosilicon slurry and pumped to the respective coarse and fine primary DMS cyclones. The ferrosilicon slurry density will be carefully controlled to enable the gravity separation of spodumene from minerals with a lower sg. Spodumene has a higher SG than most gangue minerals and consequently the spodumene will report to the DMS cyclone underflow (sinks), with the gangue material reporting to the cyclone overflow (floats).

The floats from the primary coarse and fines DMS cyclones, the secondary fines DMS cyclone, and the minus 1mm fines, as well as the underflow from the screw classifier (mica and floats) will be screened and stored separately or co-disposed with mine waste in a waste pile.

The process plant buildings include:

- Crusher building
- Multi-stage DMS plant building
- Solid-liquid separation building with tailings screw classifier area
- Main control room and Motor Control Center (MCC) building
- Assay laboratory building
- Plant offices and dry
- Minor spares, materials, and consumables storage buildings.

1.10.3 Concentrate Storage

This material is mostly coarse spodumene concentrate product that would be stored outside, or in a covered area, until ready for concentrate haulage.

1.11 Project Infrastructure

The Raleigh Lake project has favourable access to major roads and robust power supply which can be connected to with relative ease.

1.11.1 Electrical Power and Fuel Supply

Site power will be obtained from the existing Hydro One 235 kV main power transmission line located directly adjacent to Highway 17. Power demand for the Project is estimated to be 4660 KVA.

The close proximity of the project to adequate power infrastructure provides for a relatively low-cost source of power for the project with minimal capital cost and minimal construction and implementation time.

1.11.2 Waste Rock / Tailings Co-disposal Storage Facilities

The 53Mt (approximately 30M cubic metres) of waste rock and 4Mt (approximately 3M cubic metres) of tailings produced is classified as low risk, meaning that it has low acid drainage and leachability potential. It is therefore proposed that a waste rock and dry tailings co-disposal method be utilized.

1.11.3 Waste Rock Storage Facilities

It is proposed that separate storage facilities be constructed for waste rock that will be used for site construction and/or potential sale to external buyers.

1.11.4 Water Storage and Management Facilities

Sedimentation ponds will be constructed to contain all open pit mining contact water and runoff from the co-disposal sites.

Storm water storage ponds will be constructed where required and stormwater management strategy will be implemented to ensure contact water released to nature will meet all applicable quality standards before release.

1.11.5 Water Treatment Plant

Potable and process water for the mine site will be drawn from Raleigh Lake and treated in on site facilities as required.

1.11.6 Fuel Supply

A fuel farm will be constructed on site which to store diesel and gasoline.

1.11.7 Natural Gas

Natural gas will be supplied to site from the existing natural gas pipeline also located adjacent to Highway 17. Natural gas distribution infrastructure may be provided by the supplier with no up front costs.

1.11.8 Diesel Storage

Diesel fuel required for the proposed mobile mining equipment fleet will be stored in the fuel farm facility constructed on the Project site.

The quantity of diesel fuel to be stored at the fuel farms is estimated to be 20,000 to 40,000 litres.

1.11.9 Gasoline

Gasoline required for light vehicles will be stored in the fuel farm facility constructed on the Project site.

The quantity of gasoline to be stored at the fuel farms is estimated to be 3,000 to 5,000 litres.

1.11.10 Heat Recovery for HVAC

A heat recovery system and other energy saving infrastructure should be investigated at the PFS level of study but it is highly conceivable that a heat recovery facility would be technically economically viable.

1.11.11 Plant Control System (PCS)

A SCADA system will be installed to permit automatic control of various components of the processing plant and will have the ability to control and monitor all equipment connected to the remote process control system.

1.11.12 Primary Access Road

Primary access to the site is via a well maintained 9.5 km long gravel surfaced access road from Highway 17, which will be upgraded and widened where necessary to accommodate haul truck traffic.

The existing road is on favourable terrain which will require modest capital investment to achieve the width and bearing capacity required to accommodate haul truck traffic and the delivery of supplies and large equipment to site.

1.11.13 Mining Infrastructure

The mining operations related infrastructure will be comprised of the i) explosives and cap magazines, ii) various pit dewatering and mine water management systems comprised of pumps and pipe systems, and iii) the maintenance and parts warehouse facilities which will include:

- A fully equipped garage for preventative maintenance and overhaul/rebuild tasks for mobile mining equipment
- Electrical shop
- Supply Warehouse

1.11.14 Communications

A fibre optic line will be extended from the existing line located adjacent to Highway 17 to provide internet and telecommunication services to site. On site communications services will include video, internet, VOIP telephone and private radio systems as well as a telecom repeater in order to facilitate mobile telephone communication across the site.

1.11.15 Port Facilities

It is currently assumed that concentrate will be transported by truck and/or rail to its point of sale. However, should it be determined in the future that shipping by boat or barge would be more economic or viable then Thunder Bay has deep water port facilities which might potentially be used for this purpose.

1.11.16 Camp

The plan is to not include a camp on site whereas the workforce will come from and / or be accommodated in Ignace (20 km from the project).

1.11.17 Concentrate Haulage

Concentrate will be hauled by truck to Winnipeg or Thunder Bay and will require a load/unload facility at either terminus. The loading facility on site will be included in the plant infrastructure and will consist of a hopper and chute.

1.12 Market Studies and Contracts

There are currently no offtake contracts in place for the Raleigh Lake project.

A specific quality of spodumene of concentrate referred to as spodumene concentrate 6 or SC6, is of particular economic significance whereas it is a relatively difficult to achieve high-purity lithium ore with approximately 6 percent lithium content. It typically achieves a premium price in comparison to SC5 and SC5.5 which have 5% and 5.5% lithium content respectively.

SC6 is widely being produced as a raw material from mine concentrators for the subsequent production of lithium-ion batteries for electric vehicles.

It is plausible that the Raleigh Lake operation will produce SC6 concentrate as its saleable product.

The standard Product Code for Spodumene concentrate min 6% Li_2O Asia, $/tonne is MB-LI-0012.

Future consensus forecasts for other Lithium products show a decline but there were no future consensus forecast for SC6 publicly available (trailing average prices for SC6 are high).

The Spodumene Concentrate >6% (SC6) price selected for this study was USD2,325 / tonne, which is significantly lower than that used in some studies issued by peers in the recent past.

1.13 Environmental Studies, Permitting, and Social or Community Impact

The preliminary analysis of the Project indicates it will be subject to multiple Class Environmental Assessments under the Ontario provincial *Environmental Assessment Act*. The Project is not anticipated to trigger a federal impact assessment under the *Impact Assessment Act*. Several other permits, approvals or authorizations will be required to continue Project development beyond early exploration, including advanced exploration through closure.

At the time of filing, environmental and socio-economic studies have not been initiated for the Project though will be necessary to support and inform environmental assessment(s) and permitting applications. These studies are required to characterize the existing environmental setting of the Project and to inform design and/or process considerations. Particular attention should be given to a consistent surface and groundwater baseline, characterization of fish and fish habitat, species at risk, and areas of archaeological potential. The environmental characterization will assist to inform project design, waste management and mine closure planning.

Once Project approvals are secured, ILC will be required to comply with any terms and conditions associated with Project-specific authorizations issued by provincial or federal authorities, as well as relevant environmental law and regulation.

First Nations and Metis communities are situated near the Raleigh Lake property and consider the area part of their traditional territory. ILC has identified preliminary Indigenous groups that may have an interest in the Project and has initiated engagement. The lands and community of the Wabigoon Lake Ojibway Nation (WLON) are the closest to the Project, WLON has been the foremost community for communication and involvement by Company representatives.

1.14 Capital and Operating Costs

1.14.1 Capital Costs

The capital and operating costs in this report are based on the design criteria and engineering work performed by the individual QPs under their areas of responsibility and expertise.

Sources used for the estimates include vendor quotations historical data, benchmark costs at similar operations, databases, empirical factors and first principle calculations where and as appropriate.

Total pre-production capital costs will be CAD$ 111.9 million (inclusive of contingency) as shown in Table 1-5 below, which include capitalized operating costs incurred before the open pit mine moves into the production phase.

Table 1-5: Pre-production Capital Costs

Cost Centre	Description	Cost (CAD$ million)
Direct CAPEX Costs		
1000	Open Pit Mining	18.2
3000	Mineral Processing	38.5
4000	Power, Electrical and Instrumentation	3.8
5000	Site Infrastructure and Support Services	12.0
6000	Water Management Systems	2.0
7000	Tailings and Mine Waste Management Facilities	1.8
Total Direct CAPEX Costs		*75.8*
Owner and Indirect CAPEX Cost Summary		
8000	Reclamation and Closure	10.0
9000	Indirect and Owner Costs	18.6
Total Indirect CAPEX Costs		*28.6*
9600	Contingency	7.6
Total Pre-Production CAPEX Costs		*111.9*

Total sustaining capital costs incurred over the production phase of the mine will be CAD$ 17.5 million (inclusive of contingency) as shown in Table 1-6 below.

Table 1-6: Sustaining Capital Costs

Cost Centre	Description	Cost (CAD$ million)
Direct SUSEX Costs		
1000	Open Pit Mining	9.1
6000	Water Management Systems	0.2
7000	Tailings and Mine Waste Management Facilities	3.8
Total Direct SUSEX Costs		*13.0*
Owner and Indirect SUSEX Cost Summary		
9000	Indirect and Owner Costs	3.2
Total Indirect Costs		*3.2*
9600	Contingency	1.3
Total SUSEX		*17.5*

1.14.2 Operating Costs

Total operating costs for the operation will primarily be those for mining and processing.

The total LoM operating cost is estimated to be CAD$ 381 million (CAD$ 413 million including concentrate transport).

The total LoM operating cost of mining is estimated to be CAD$ 179 million.

The total LoM operating cost of processing is estimated to be CAD$ 125 million.

The total LoM operating cost of G&A is estimated to be CAD$ 78 million.

The total LoM cost of concentrate transport is estimated to be CAD$ 31million.

The unit costs of mining, processing and G&A per tonne, and the total operating costs are presented in Table 1-7 below.

Table 1-7: Unit and Total Operating Costs

Parameter	Value	Unit
Unit Operating Costs		
Mining	3.55	$/t mined
Mining including Waste (S.R. =13.2:1)	47.18	$/t mill feed
Milling	28.53	$/t mill feed
G & A	17.74	$/t mill feed
Concentrate transportation	7.13	$/t mill feed
Total	*104.12*	*$/t mill feed*

Parameter	Value	Unit
Overall Project Costs		
Total Mining Cost	179	CAD$ million
Total Milling Costs	125	CAD$ million
Total G&A Costs	*78*	*CAD$ million*
Total Operating Costs	381	CAD$ million
Total Concentrate Transport Costs	31	CAD$ million
Total Operating Costs plus Transport	*413*	*CAD$ million*

1.15 Economic Analysis

The pre-tax economic indicators of the project include a total cash flow CAD$ 709.5 million, a pre-tax net present value (NPV) of CAD$ 385.1 million with an assumed discounting rate of 8% and an internal rate of return (IRR) of 46.5%.

The post-tax economic indicators of the project include a total cash flow CAD$ 634.0 million, a post-tax net present value (NPV) of CAD$ 342.9 million with an assumed discounting rate of 8% and an internal rate of return (IRR) of 44.3%.

There is a payback period of 4 years after construction begins, which equates to two (2) years after the start of the production phase of the project.

1.16 Conclusions

ERM concludes that the Raleigh Lake project demonstrates sufficient favourable attributes such that it would be worthy of further technical investigation to bring it to a PFS level of study.

Based on the data acquired from previous operators and the three-phase drilling program by International Lithium (ILC), the Mineral Resource Estimate (MRE) findings in this technical report demonstrate the Project's technical value. International Lithium plans to continue its drilling operations on the site, with the aim of upgrading the current mineral resource category and identifying additional economic resources.

1.16.1 Risks

1.16.1.1 Mineral Resources Risk

At the time of the writing of this PEA report there are a few elements of uncertainty that could have a substantial impact on the Mineral Resource Estimate (MRE) upon which the PEA is based.
These elements include:

- Long-term metal price assumptions for Spodumene Concentrate SC6 where the market has seen wide variation in the price of various Li related products including Li_2O and Lithium Carbonate.
- Modifications to the input parameters for mining costs processing costs, and general administrative costs to restrict the estimation.
- Revisions in the geological interpretations of the shape of mineralization and the continuity of mineralized zones.
- Adjustments to the density values attributed to the mineralized zones.

1.16.1.2 Regulatory Risk

There is material risk to the Project's realization and/or schedule due to potential regulatory delays securing the necessary permits or approvals.

1.17 Recommendations

The Company should continue exploration activities designed to increase diamond drilling density, particularly targeting areas within and below the optimized open pit shell.

Efforts to de-risk cost uncertainty through pursuit of budget and contractually enforceable quotes for significant equipment and contracts will reduce cost uncertainty and improve the cashflow model reliability.

The Company should continue metallurgical optimization studies with additional variability and mini-bulk samples.

It is recommended that the environmental, social and heritage assessment requirements, including permitting to meet Ontario provincial and Canadian federal regulations, be confirmed with relevant agencies and be completed.

2. INTRODUCTION

Environmental Resource Management ("ERM"), in conjunction with Nordmin Engineering Ltd, has prepared this technical report on the project at the request of International Lithium Corp. ("ILC" or the "Company").

The purpose is to provide a technical report of a Preliminary Economic Assessment ("PEA") in accordance with the disclosure and reporting requirements outlined in the Canadian Securities Administration's National Instrument 43-101 "Standards of Disclosure for Mineral Projects" ("NI 43-101") for International Lithium Corp. by Environmental Resource Management ("ERM") and Nordmin Engineering Inc. on Raleigh Lake Project.

2.1 Terms of Reference

The aim of this report is to present a technical document regarding the Preliminary Economic Assessment (PEA) for the Raleigh Lake Project of International Lithium Corp. Currently, the company's primary strategic focus is on the lithium and rubidium project at Raleigh Lake in Canada, while also seeking additional properties in Canada and Zimbabwe.

Dated November 30, 2023, this Technical Report takes precedence over all previous technical reports related to the project. The contents of this Technical Report are summarized as follows:

- Details about land ownership, exploration history, and drilling activities.
- Mineral resource estimates for the Raleigh Lake deposit.
- A conceptual mine plan, developed to support the PEA.
- A discounted cashflow model derived from the conceptual mine plan.

International Lithium Corp. is a publicly traded company listed on the TSX Venture Exchange, with its corporate office situated at 1120-789 West Pender Street, Vancouver, BC, V6C 1H2.

2.2 Principal Sources of Information

This Technical Report has been prepared by independent consultants who are Qualified Persons (QPs) under NI 43-101. They have prepared the report in accordance with the guidelines laid out in NI 43-101, Form 43-101F1, and Companion Policy 43-101CP. Within the limitations specified here, the independent consultants have confidence in the qualifications, assumptions, and information they have utilized, and reasonable efforts have been made to validate their reliability.

This Technical Report is completed based on a previous MRE Technical Report prepared by Nordmin, and partially based on internal technical reports and maps from the company, publicly available government reports, company correspondence, and publicly accessible information listed in Section 27. Certain sections of reports authored by other consultants have been directly quoted or summarized in this Technical Report and are appropriately credited. Nevertheless, the QPs have made reasonable attempts to authenticate such data and do not disclaim any responsibility for its utilization.

The authors of this report have exercised their professional judgment to meticulously verify and confirm the accuracy of the information contained herein. Excluding the matters outlined in Section 3, they do not disclaim any responsibility for the content of this Technical Report.

2.3 Qualified Person Section Responsibility

The Consultants preparing this technical report possess expertise in various fields such as geology, exploration, mineral resource assessment, open pit mining, geotechnical analysis, environmental considerations, permitting, metallurgical testing, mineral processing, process design, civil and mechanical engineering, electrical systems, and cost estimation. None of them or their associates hold any vested interest in International Lithium Corp. These consultants have no associate, or affiliate status with the Company, and their report's findings are unbiased, devoid of prior agreements, or undisclosed future business arrangements. Their compensation adheres to standard professional consulting practices.

The individuals listed here, by virtue of their education, practical expertise, and affiliation with relevant professional organizations, are recognized as Qualified Persons (QPs) in accordance with the NI 43-101 standard for this report. Furthermore, they maintain active memberships in the appropriate professional associations.

The preceding Qualified Persons contributed to the writing of this report and the responsibilities for each section are indicated in Table 2-1.

Table 2-1: Qualified Persons

Section and Title	QP	Company
1: Summary	All	ERM
2: Introduction	Nigel Fung, P.Eng.	ERM
3: Reliance on Other Experts	Nigel Fung, P.Eng.	ERM
4: Property Description and Location	Nigel Fung, P.Eng.	ERM
5: Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Patrick McLaughlin, P.Geo.	International Lithium
6: History	Patrick McLaughlin, P.Geo.	International Lithium
7: Geological Setting and Mineralization	Christian Ballard, P.Geo.	Nordmin
8: Deposit Types	Christian Ballard, P.Geo.	Nordmin
9: Exploration	Christian Ballard, P.Geo.	Nordmin
10: Drilling	Christian Ballard, P.Geo.	Nordmin
11: Sample Preparation, Analyses, and Security	Christian Ballard, P.Geo.	Nordmin
12: Data Verification	Christian Ballard, P.Geo.	Nordmin
13: Mineral Processing and Metallurgical Testing	Richard Wagner, P.Eng.	ERM
14: Mineral Resource Estimate	Christian Ballard, P.Geo.	Nordmin
15: Mineral Reserve Estimate	NA	NA
16: Mining Methods	Garth Liukko, P.Eng.	ERM
17: Recovery Methods	Georgi Doundarov, P.Eng.	ERM
18: Project Infrastructure	Nigel Fung, P.Eng.	ERM
19: Market Studies and Contracts	Nigel Fung, P.Eng.	ERM

Section and Title	QP	Company
20: Environmental Studies, Permitting, and Social, or Community Impact	Rolf Schmitt, P.Geo.	ERM
21: Capital and Operating Costs	Garth Liukko, P.Eng.	ERM
22: Economic Analysis	Nigel Fung, P.Eng.	ERM
23: Adjacent Properties	Garth Liukko, P.Eng.	ERM
24: Other Relevant Data and Information	Nigel Fung, P.Eng.	ERM
25: Interpretation and Conclusions	Garth Liukko, P.Eng. Nigel Fung, P.Eng.	ERM
26: Recommendations	All	ERM
27: References	All	ERM
28: Glossary	All	ERM

Certificate of the Authors are provided in Appendix A of this Technical Report.

2.4 Qualified Person Site Inspections

The following list provides information about the Qualified Persons who conducted site visits, including the visit dates and the purpose of each visit:

Mr. Christian Ballard, P. Geo., from Nordmin, visited the site on October 18, 2022, to assess the geological conditions, evaluate the Property, examine the diamond drill core, verify drill collar positions, and confirm the technical and geological data presented here.

Mr. Garth Liukko, P.Eng., from ERM, visited the site on September 13, 2023, to analyze the Project's context and examine the potential location of the pit and facilities.

ERM affirms that these site visits comply with the current requirements of NI 43-101CP, Section 6.2.

2.5 Effective Date

This report is dated January 18, 2024. Raleigh Lake Project Integrated PEA became effective on January 18, 2024. As of this report's effective date, the authors have no knowledge of any significant information or changes related to the subject matter of this technical report that have not been included herein, and the omission of which could result in this report being misleading.

3. RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) have meticulously examined data and reports supplied by International Lithium Corp., along with publicly accessible information, and have drawn their conclusions, augmented by direct on-site inspections.

The QPs responsible for this report have placed trust in information provided by experts who are not QPs. This trust is grounded in the belief that these experts possess the requisite education, professional credentials, and pertinent experience about matters addressed in the technical report.

The QPs have taken for granted that all the information and technical documents listed in Section 27, References, of this report are accurate and comprehensive in all material respects. While the QPs have thoroughly reviewed all accessible information, they cannot ensure its absolute accuracy and completeness. It is important to note that the QPs retain the right, though not the obligation, to revise the report and its conclusions should additional information come to light after the report's date.

3.1 Mineral Tenure, Surface Rights, Property Agreements, and Royalties

Copies of the tenure documents, operating licenses, permits, and work contract were reviewed by Nordmin. Independent verification of land title and tenure reported in Section 4 was not performed. Nordmin did not verify the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has instead relied on the Company to have conducted the proper legal due diligence. Information for Section 4 regarding Mineral Tenure, Surface Rights, Property Agreements, and Royalties was obtained from Project Geologist, Patrick McLaughlin, of Coast Mountain Geological Ltd., who has been working on the project with ILC since their ownership in 2016.

3.2 Environmental, Permitting, and Liability Issues

The QP has relied upon the Company via written statement via electronic mail provided to Nordmin on April 10th, 2022, concerning the Project environmental, socio-economic, and permitting matters relevant to the Technical Report.

The QP has relied upon publicly available information on biophysical resources, Ontario and Canadian permitting requirement information on government websites, and ILC for summaries of Indigenous engagement activities, and statement on known site environmental liabilities.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location of Property

The Raleigh Lake Property, referred to as the "Project," is situated about 25 kilometres to the west of Ignace and approximately 235 kilometres to the west of Thunder Bay, located in northwestern Ontario, specifically within the Kenora Mining District. It falls within the geographical region represented on the National Topographic Systems (NTS) map sheet 52G/05, as illustrated in Figure 4-1. The Project is centred around specific UTM coordinates, precisely at 576550 metres East and 5473800 metres North, using the EPSG coordinate reference system 26915 (UTM NAD83 Zone 15N).

Figure 4-1: Map Depicting the Location of the Raleigh Lake Project Situated in the Northwestern Region of Ontario (Source: McLaughlin, 2019)

4.2 Mineral Tenure and Surface Rights

In March 2016, ILC agreed with Robert Fairservice, on acquiring a 100% ownership interest in six legacy mining claims (K4218370, K4218371, K4242501, K4242502, K4242505, K4245250). Subsequently, in July of the same year, they staked an additional eight claims (K4274924, K4274925, K4274926, K4274927, K4279997, K4279998, K4279999, K4280000) contiguous to this initial group.
These fourteen contiguous unpatented 'legacy' mineral claims were transformed into forty-eight single-cell claims and nine boundary-cell claims as part of the MNDMNRF's efforts to modernize the mining act, transitioning from paper-based to online staking and claim management.

In September 2018, ILC expanded its holdings by purchasing the claims south and southwest of the project area, owned by Perry English. This acquisition added fifty-five single-cell mineral claims ("SCMC") covering 1,976 hectares of unencumbered mineral tiles to the Raleigh project. The following month, in October 2018, ILC further extended their claim area by staking an additional fifty single-cell mining claims contiguous to the southwest of the existing claim group, bringing the total claim area to 3,025 hectares.

Following a successful inaugural drilling program in the 3rd and 4th quarters of 2021 and into 2022, ILC pursued an aggressive land acquisition strategy, significantly expanding their land tenure through multiple stages. They increased their tenure from 3,025 hectares to 48,500 hectares, comprising 2,308 unencumbered SCMCs. Three SCMCs within the White Otter block, which had surface rights ownership, were subsequently relinquished, reducing the total to 2,305 claims. These claims are divided into two non-contiguous blocks known as the Raleigh Lake/White Otter and Owl Lake, as depicted in Figure 4-2.

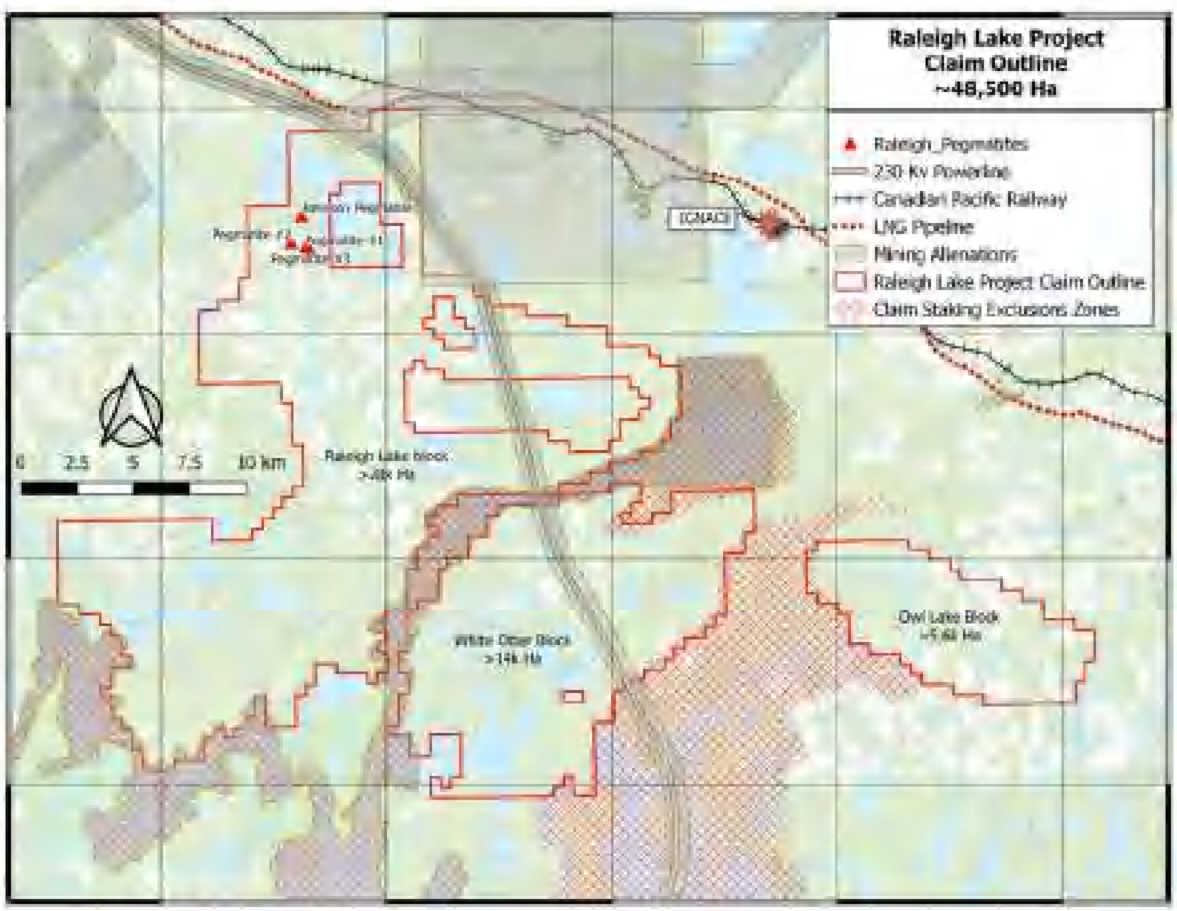


Figure 4-2: Outline of Claim Cells for the Raleigh Lake Project (Source: ILC, 2023)

4.3 Underlying Agreements

The non-contiguous distribution of land ownership is a result of exclusion zones and spatial restrictions mandated by the Campus Lake Conservation Reserve (C2299), which intersect the White Otter and Owl Lake blocks.

In the Province of Ontario, work commitments for mineral claims amount to $400 for single cell mineral claims and $200 for boundary cell mineral claims ("BCMC"). These work commitments must be fulfilled within the anniversary year of staking a claim, which then advances the anniversary date by one year. A maximum of five years' worth of work credits can be applied to a mineral claim at any given time.

ILC holds 100% ownership of all mineral claims, and to the best of our/ILC’s knowledge there are no royalties, back-in rights information available regarding royalties, back-in rights, payments, or any other encumbrances affecting the Raleigh Lake Project. Additionally, there are no legal obstacles or other factors limiting access, title, or the ability to conduct work on the property. Moreover, there are no known environmental liabilities associated with the property, and it primarily covers Crown Land, eliminating any competing surface rights or obstructions for legal access.

4.4 Environmental Liabilities

The Project site is generally undeveloped apart from recent logging activity, existing forestry roads, and minor on-site infrastructure described elsewhere in this report. ERM is unaware of any historic or current environmental liabilities to which the Project and property is subject.

4.5 Permitting

ILC presently possesses two exploration permits and authorizations for conducting mechanized drilling, denoted as PR-20-000001 and PR22-000057. Specifically, PR-20-000001 pertains to diamond drilling within a set of 13 SMCM within the claim group, and it remained in good standing until February 12th, 2023. PR-22-000057 pertains to diamond drilling within a larger area of 195 SCMC within the claim group and maintains its good standing status until September 15th, 2025.

Exploration plans and permits can be sought through MLAS (Mineral Land Administration System) by a registered claim owner or a designated agent. The annual work commitments for the entire 2,305 SCMC tenure area covered in this report amount to CAD$ 922,000.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is situated approximately 230 kilometres to the northwest of Thunder Bay, Ontario. It is accessible via a paved highway, Ontario Highway 17, leading to Doreen South Road, followed by a 10-kilometre journey on a maintained gravel forest road. It's worth noting that there are annual spring weight road restrictions typically lasting for about two months, which primarily affect larger vehicles like buses and transport trucks. The Ontario Ministry of Transportation posts the specific weight restrictions and their effective dates. However, the forest roads offer reliable access to all areas of the Project, as depicted in Figure 5-1.

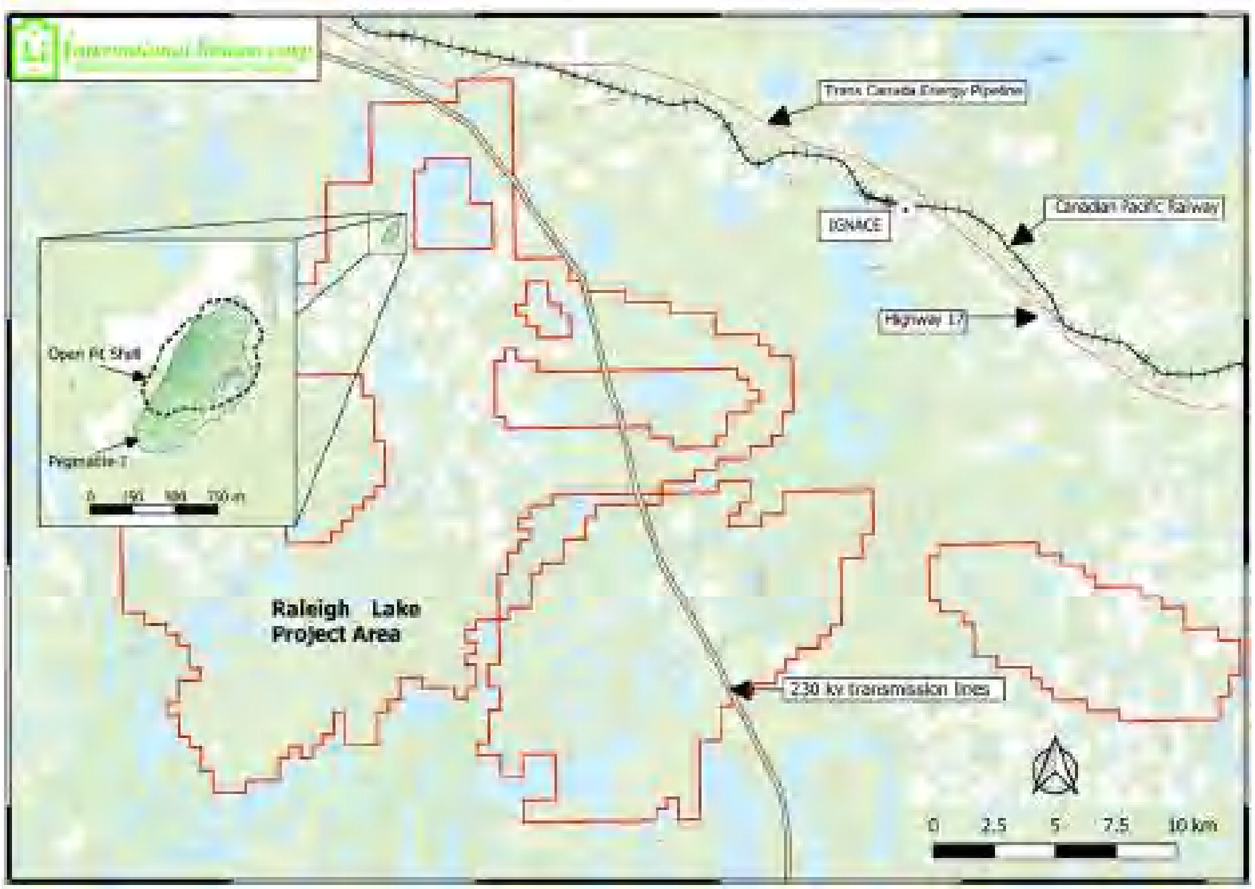


Figure 5-1: Major Public Infrastructure in relation to the Project (Source: ILC, 2023)

5.2 Climate and Physiography

The climate in the region is suitable for year-round drilling and site activities, while fieldwork like mapping and soil sampling can be conducted during spring, summer, and autumn. Ignace experiences a humid continental climate, characterized by warm summers and relatively mild winters. Meteorological data from 1981 to 2010 reveals that the coldest month is January, with an average temperature of -16.8 °C, while the warmest month is July, with an average temperature of 18.9 °C. The average annual precipitation during this period was 719.7 mm, which includes an average annual snowfall of 174.7 cm.

The climate in the area is typical of a mid-latitude continental climate. Field operations can occur throughout the year, and there are no access restrictions on logging roads connected to the Trans Canada Highway.

The Project is situated in a moderately elevated region with a mean elevation of 490 metres above sea level (MSL). Bedrock is exposed on most ridges, while gullies, draws, and significant drainage areas are covered with drift material. The predominant tree species in the area include poplar, balsam, spruce, pine, and birch. Furthermore, the region has been subjected to logging activities within the last decade.

5.3 Local Resources and Infrastructure

The Project lacks connections to any pre-existing utilities. Most necessary supplies and services can be sourced locally within the Ignace or Dryden vicinity, with Thunder Bay serving as an alternative procurement hub when needed.

6. HISTORY

6.1 Project and Exploration History

Research in the vicinity of Raleigh Lake has historically concentrated on investigating the presence of gold and base metal deposits within greenstone formations. However, in more recent times, Avalon Ventures Ltd. ("Avalon") has primarily directed their exploration efforts towards assessing the tantalum potential of the property, while paying limited attention to exploring and comprehending the lithium potential of the Project. The exploration history of the Raleigh Lake area is described below.

6.1.1 1966

Stan Johnson identified a spodumene-rich pegmatite formation in the Raleigh Lake region, which he later named the Johnson Pegmatite. However, this discovery was not publicly revealed until the early 1990s, despite Johnson holding the rights to the land during the 1970s and 1980s.

6.1.2 1993-1998

The Raleigh Lake pegmatite field was investigated by the Ontario Geological Survey as part of a significant project focused on different mineralization events associated with granite formations in the Superior Province. In Breaks' 1993 report, there were descriptions provided for the Johnson Pegmatite, along with comprehensive information about several newly identified pegmatite occurrences, such as Pegmatite 1, 2, and 3, which were identified through boulder mapping (see Figure 6-1).

6.1.3 1996-1998

The Ontario Geological Survey conducted field mapping and geological compilation activities in the Ignace area, including Raleigh Lake, as documented in Stone's 1999 report. This work involved the identification and mapping of the two-mica granite outcrops, which were thought to be the original source of the Raleigh Lake pegmatite formations.

6.1.4 1997-1998

In 1998, R. Fairservice, S. Johnson, and J. Bond secured the properties through staking and later granted Avalon an option on the properties.

6.1.5 1998-2000

After an initial property visit and brief compilation period, Avalon conducted a comprehensive prospecting program, confirming the presence of pegmatite bodies in July. They conducted a lithogeochemical program in the same month, collecting 29 samples across four widely spaced East-West traverses to assess lithophile enrichment within the mafic metavolcanic host rocks (Pederson, 1999a).

In 1999, Avalon staked additional claims, and in September, they established a small grid over the primary pegmatite occurrence encompassing Pegmatites 1 and 3 to guide diamond drilling. Subsequently, in October 1999, a 602-metre diamond drilling program consisting of five holes was carried out to define the tantalum potential around Pegmatites 1 and 3 (Pedersen, 1999b). The drilling confirmed the presence of stacked pegmatite bodies with significant lateral and subsurface continuity, extending up to 450 metres down dip from surface exposures (see Figure 6-2). Notably, Pegmatite 3 exhibited elevated tantalum levels and higher tantalum-to-niobium ratios compared to Pegmatite 1, leading Avalon to interpret a fractionation pattern trending towards the southeast (Pederson, 2000).

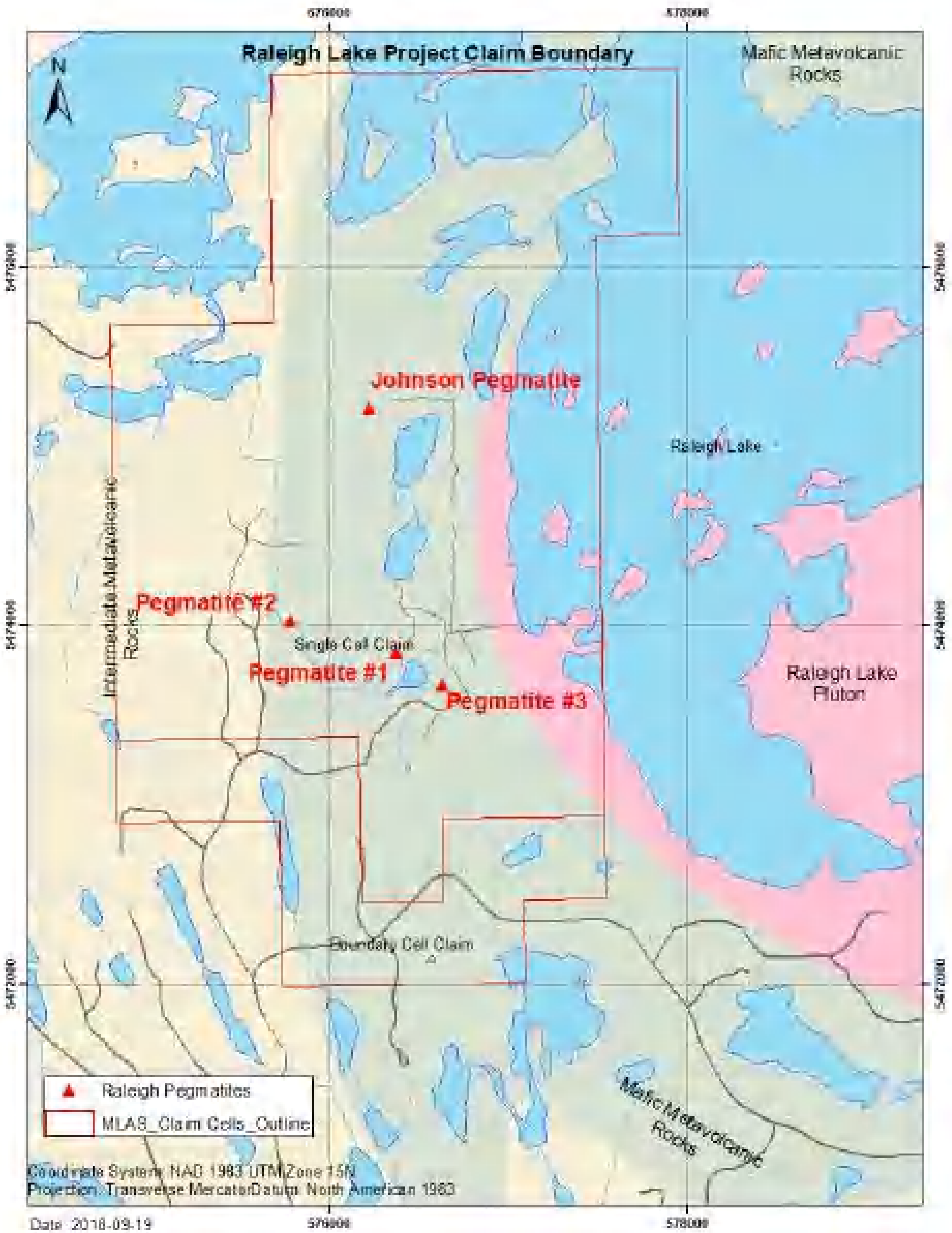


Figure 6-1: Illustration that Emphasizes the Primary Pegmatite Occurrences on the Property (Source: McLaughlin, 2019)

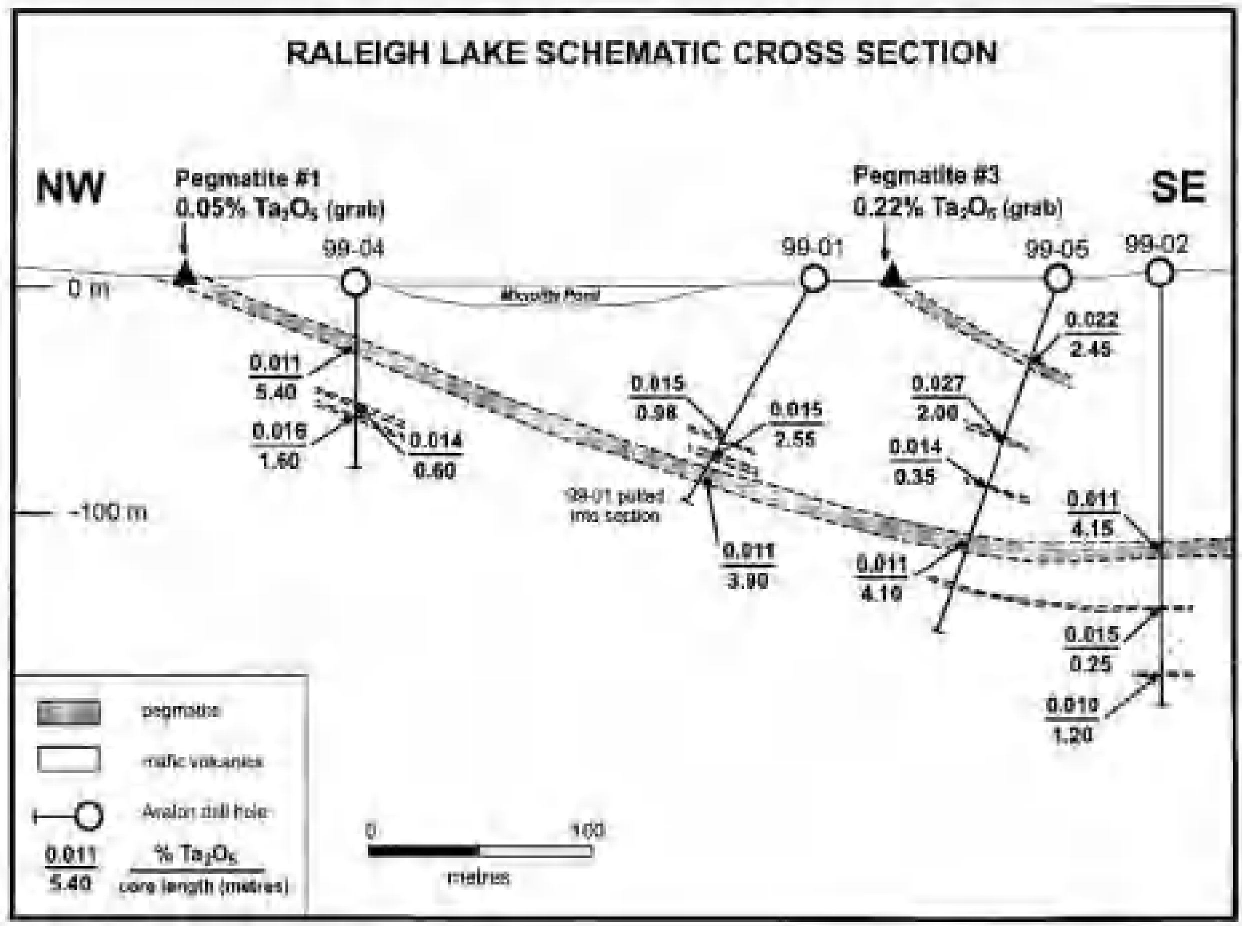


Figure 6-2: A Diagram Illustrating the Vertical Arrangement and Continuous Extension below the Surface of Pegmatite 1 (Source: Pedersen, 2000)

In the autumn of 2000, Global Canada Company ("Global") financed a surface exploration initiative valued at $120,000. This program involved activities such as line-cutting, lithogeochemical sampling, and the assessment of pegmatite deposits, as documented by Pedersen in 2000. During this endeavour, around 966 bedrock samples were collected, leading to the discovery of three noteworthy rare-metal geochemical trends labelled as Trends 1 through 3, alongside several lesser sub-parallel trends designated as a, b, c, and so on (as illustrated in Figure 6-3).

6.1.6 2001

In 2001, Avalon conducted various exploration activities, which included lithogeochemical sampling, trenching, structural investigations, and diamond drilling. During this period, a combined total of 398 bedrock samples were gathered across three distinct zones situated to the south of the primary survey region. Additionally, a fourth single-line survey traverse was conducted on the southern side of the claim group, as depicted in Figure 6-3.

According to Campbell's findings in 2001, the bedrock sampling efforts did not reveal any novel or noteworthy anomalies, even though the highest recorded bedrock chip sample assay was 92 ppm Li, which was not deemed unusual or remarkable.

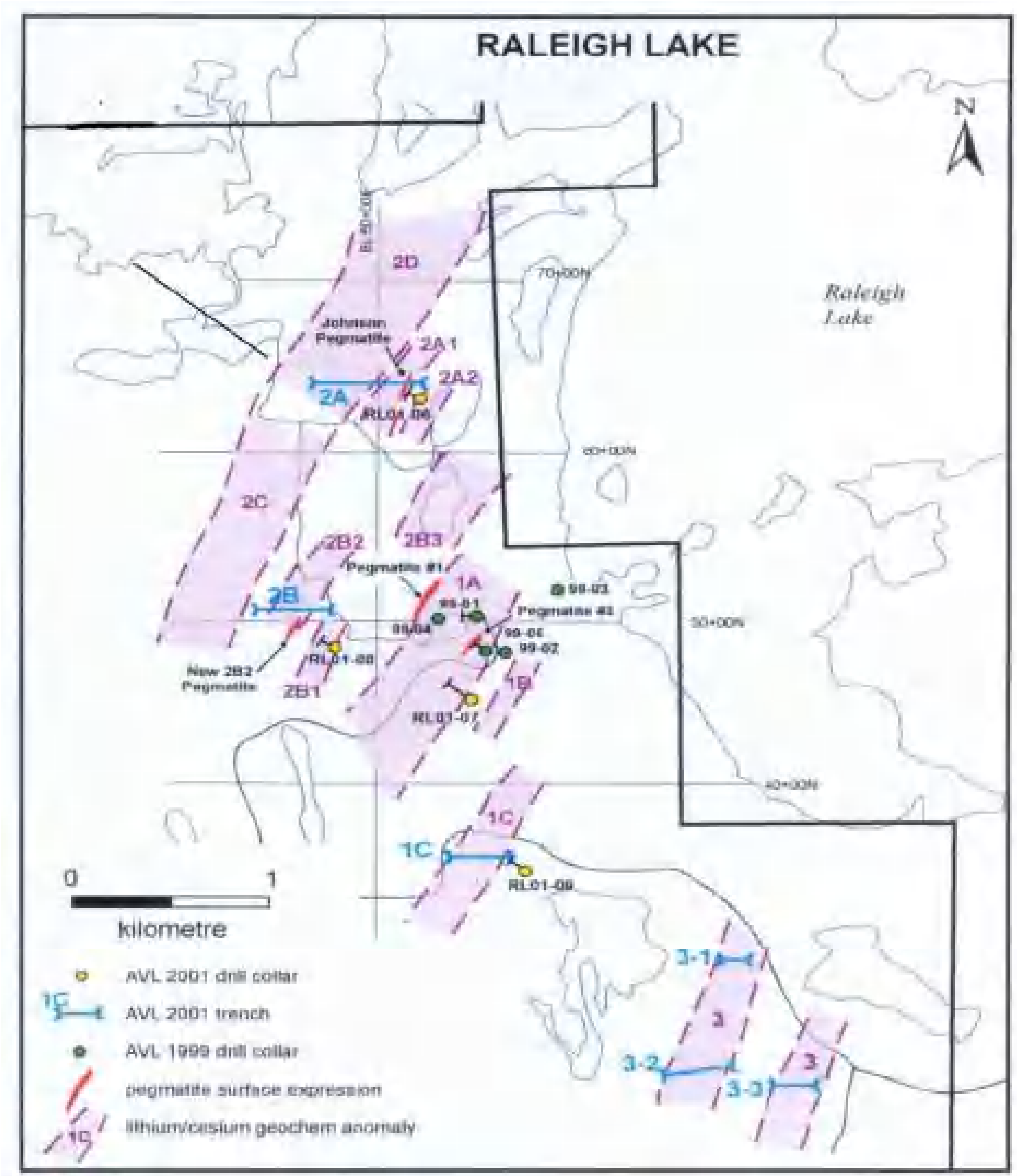


Figure 6-3: Interpretation from Avalon Ventures Ltd Highlighting the Main Geochemical Trends, Pegmatite Occurrences, Trench Locations and Diamond Drilling (Source: Pederson, 2001)

Six trenches were excavated, covering a total linear distance of 1500 metres, and these trenches were strategically positioned across various lithogeochemical trends. They were named A1, 2A, 2B, 3-1, 3-2, and 3-3, each identified by the specific trend they intersected, as indicated in Figure 6 3. During the trenching process, significant pegmatite dikelets were discovered in certain areas, providing a potential explanation for the lithogeochemical patterns observed in trend targets 2A (known as the Johnson Pegmatite) and 2B (referred to as Pegmatite 2). However, in the case of the remaining trenches, no geologically significant findings were made, except for the observation that some sample results from the end of these trenches hinted at the presence of concealed pegmatite formations beyond the trench boundaries, as detailed by Campbell in 2001.

During the field season, J. Willoughby conducted a B.Sc. (Honours) thesis at the University of Windsor in Ontario, focusing on petrological and geochemical investigations associated with Archean granitoids linked to the Raleigh Lake pegmatites (as documented by Willoughby in 1999). In this research, Willoughby categorized the granitoid rocks in the study region into three primary suites and proposed a consistent fractionation pattern that extended toward the southeast.

Structural studies conducted during the bedrock sampling program concluded that major pegmatite bodies have not been significantly modified from interpreted regional deformation patterns and they should have extensive lateral continuity. It was also noted that more evolved dykes and bodies emplaced at higher elevations that the current known pegmatite bodies may occur to the southeast.

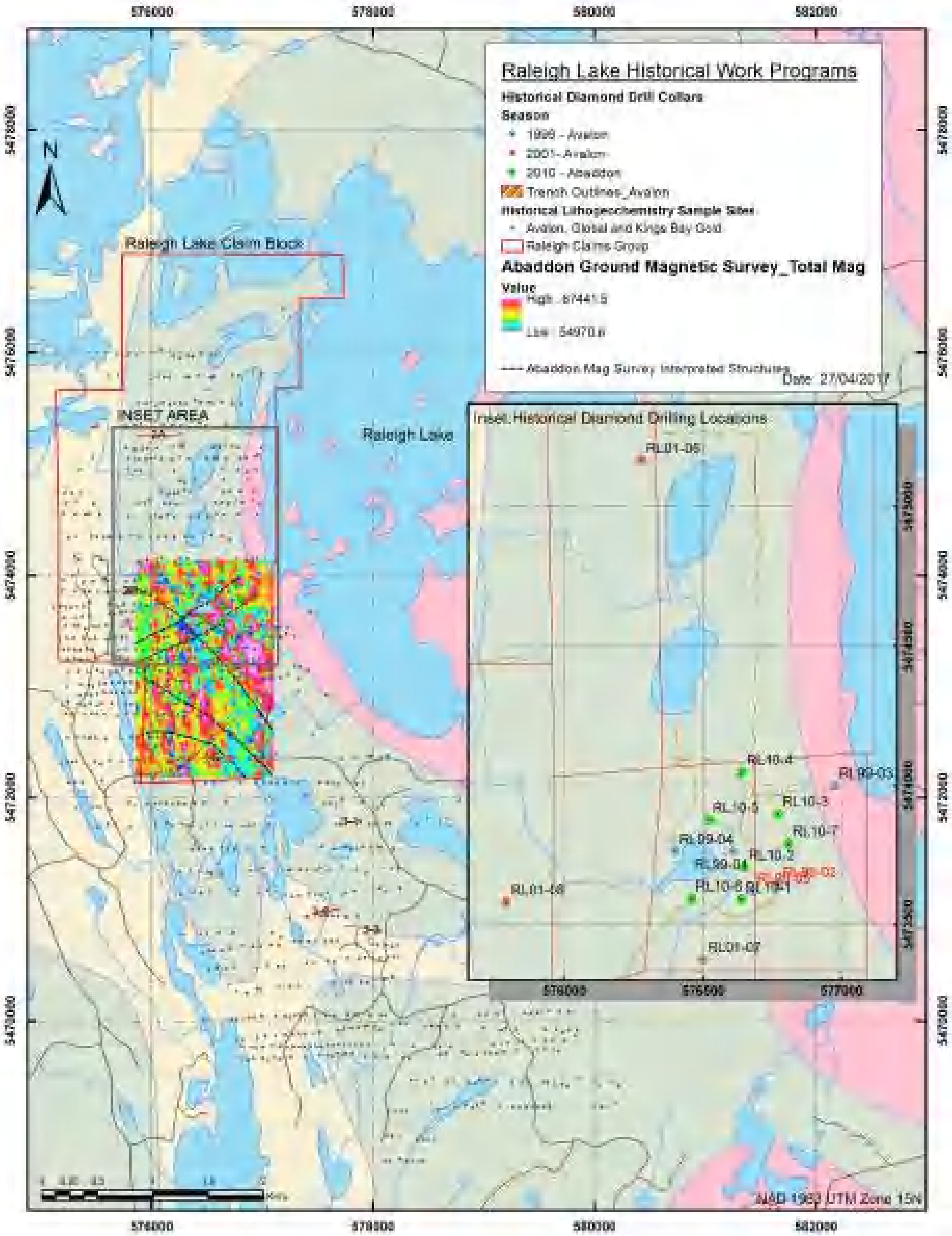
Between July and August of 2001, a diamond drilling program consisting of four holes was carried out, covering a distance of 752 metres. This program served as a continuation of previous efforts and aimed to further investigate the primary pegmatite occurrences at greater depths. Additionally, it included the examination of several holes to assess lithogeochemical irregularities. Holes RL01-06 through RL01-08 intersected multiple shallowly inclined pegmatite dikes and smaller dikelets, while RL01-09 was the sole drilling hole designed to explore a surface lithogeochemical anomaly but failed to intersect any pegmatite veins, remaining within a felsic dike or the Raleigh Lake Pluton granite.

In November of the same year, Kings Bay Gold Corp conducted a soil sampling program, during which 520 soil samples were collected from the property. This initiative was primarily focused on investigating the potential for gold (Au) within the mafic metavolcanic host rocks, particularly in the southwest region of the primary pegmatite occurrences and claim group.

6.1.7 2010

Consolidated Abaddon Resources Inc. ("Abaddon") initiated a ground-based magnetometer survey spanning 50 linear kilometres in October 2009. This survey was followed by the drilling of seven diamond drill holes, totaling 1463.5 metres, conducted in February and March 2010. The objective of these activities was to further assess the tantalum potential of the property. The results from the diamond drilling unequivocally confirmed the existence of multiple stacked and shallowly inclined pegmatite bodies containing spodumene.

The ground-based magnetometer survey was executed with measurement stations positioned at 12.5-metre intervals. It primarily covered the main pegmatite occurrences near pegmatites 1 and 3. The outcomes of the magnetometer survey were primarily used to characterize and emphasize any notable structural elements related to the emplacement of pegmatites. An examination of the newly acquired magnetic data in conjunction with lithogeochemical surveys revealed several potential subsurface structural trends that warranted further investigation, as depicted in Figure 6-4.



Note: that Claim Boundaries Reflect Legacy Claim Shapes

Figure 6-4: Outline of Historical Exploration Activity on the Raleigh Lake Property (Source: Mclaughlin, 2019)

In February 2010, Abaddon engaged P. Vanstone of Vanstone Geological Services, a former Chief Geologist at Cabot Corporation's Tanco Mine, to conduct a comprehensive property and diamond drill core evaluation. Key observations from his report corroborated that the pegmatite fractionation trends extend towards the southeast. Furthermore, it highlighted the presence of a prominent gabbroic host body enclosing pegmatite 1. This information, along with sheared contact relationships caused by reactivated structural features and the presence of multiple pegmatite phases, suggests a complex history of pegmatite emplacement. These findings significantly enhance the Project's value and its potential to host a substantial pegmatite body.

6.1.8 2016

In early 2016, ILC acquired the property from Robert Fairservice. Following this, Pioneer Resources Corp. (renamed Essential Metals) entered into an option agreement with ILC for their Mavis Lake and Raleigh Lake projects in the same year. They initiated an 8-day bedrock sampling program, comprising 310 samples for lithogeochemistry, from late September to October 2016 on the Project. The primary aim of this program was to re-establish precise positioning of the known pegmatite occurrences and to refine the existing anomalous geochemical patterns identified in earlier studies. This refinement was crucial in determining drilling targets for a subsequent drill program. The results of this work successfully delineated more specific geochemical trends within the broader geochemical zones.

Additionally, Pioneer Aerial Surveys Ltd. conducted a UAV magnetometer survey, covering a total distance of 189.8 line kilometres with continuous profiling at a 40-metre line-spacing.

7. GEOLOGICAL SETTING

7.1 Regional Geology

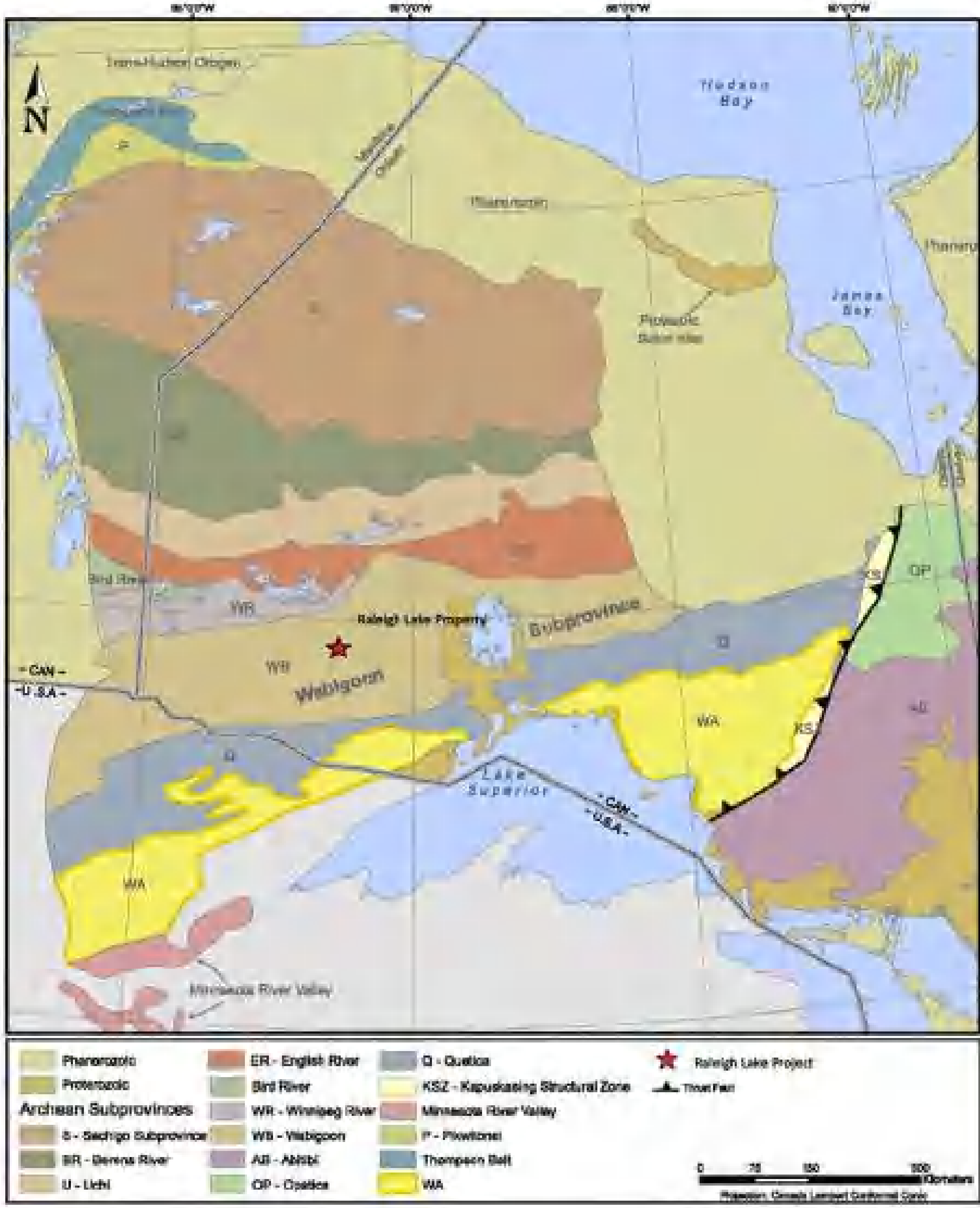
The Raleigh Lake property is located within the Wabigoon Subprovince of the Superior Province of the Canadian Shield, occurring in the western portion of the central Wabigoon region. This area is characterized by ovoid gneissic domes and elliptical batholiths with screens and small belts of supracrustal rocks. Older foliated and gneissic tonalitic bodies are cut and surrounded by younger massive and foliated granitic bodies forming large-scale dome and basin structures. The supracrustal greenstone belt is composed of mafic meta-volcanic rocks with lesser portions of intermediate to felsic metavolcanic and volcaniclastic rocks, gabbros, and their derived metasedimentary equivalents. Greenstone sequences of the western Wabigoon terrane are interpreted to have developed in a somatic environment about 2745 to 2712 Ma and tectonically emplaced onto the Winnipeg River and Marmion terranes at 2703 to 2695 Ma. Metamorphism in the region is commonly low pressure greenschist facies grading to lower amphibolite facies in selvedges around pre and post tectonic plutons. The timing of metamorphism is constrained to 2680-2800 Ma (Easton, 2004).

7.2 Property Geology

The Raleigh Lake rare-element pegmatite field is located immediately west of Raleigh Lake, confined to the mafic volcanic portion of the mafic-intermediate volcanic package of the supracrustal greenstone belt. The Project area is extensively covered by layers of glacial till and sandy soil, and outcrop exposure is generally poor. It is situated between the Revell Batholith to the west and the Raleigh Lake Pluton to the east (Figure 7-1). The property is predominantly underlain by Archean supracrustal rocks of mafic metavolcanic composition and their derived metasedimentary equivalents. These are both underlain and intruded by granitic plutons and batholiths of various ages including the peraluminous (S-type) Revell Lake Batholith. The rare-metal pegmatites are both spatially and genetically related to the Revell Lake Batholith. The extreme southeastern end of the batholith is in contact with the pegmatite hosting metavolcanic rocks. The supracrustal volcanic rocks of the Raleigh Lake Greenstone Belt extend southeasterly over 50 km into the central Wabigoon Subprovince and are truncated in the east by the granitoid White Otter and Indian Batholiths (Figure 7-2). Discontinuous gneissic horizons are developed within the mafic greenstone lithology, and the elongated, northwesterly-trending Revell Batholith transitions in the northwest from a foliated tonalite to a granodiorite, with muscovite and biotite granite in the southeastern margin of the batholith. West-northwesterly and northerly trending dolerite dyke sets can be seen cross-cutting Archean granites on aeromagnetic data, and these are considered related to midcontinent rift magmatism. The youngest granitic phase is the two-mica granite phase that is believed to be parental to the rare element pegmatites of the Raleigh Lake pegmatite field.

A preliminary structural study by Barclay (2001) suggests primary bedding and cleavage foliation trends are generally north and northwest of Raleigh Lake ranging 160°-220° and dipping moderately to the east and swinging around to an easterly trend south of Raleigh Lake. These structural features imply dome and basin fold features, particularly in the vicinity of Raleigh Lake that could be topographic expressions of shallow-dipping, gently undulating layers and sills. Most pegmatite occurrences trend north-northeast with moderate easterly dips ranging from 25°-40°.

The Raleigh Lake area is extensively covered by thin to moderate layers of glacial till and sandy soil. Outcrop exposure is generally poor, even along the shorelines of numerous lakes examined in the area, including Raleigh Lake. Surface exposure of what are termed Pegmatite 1 and Pegmatite 3 are visible on the property and demonstrate large greenish white spodumene laths.



Note: Modified from Map

Figure 7-1: Geological Setting of the Raleigh Lake Property within the Wabigoon Subprovince of the Superior Province of the Canadian Shield

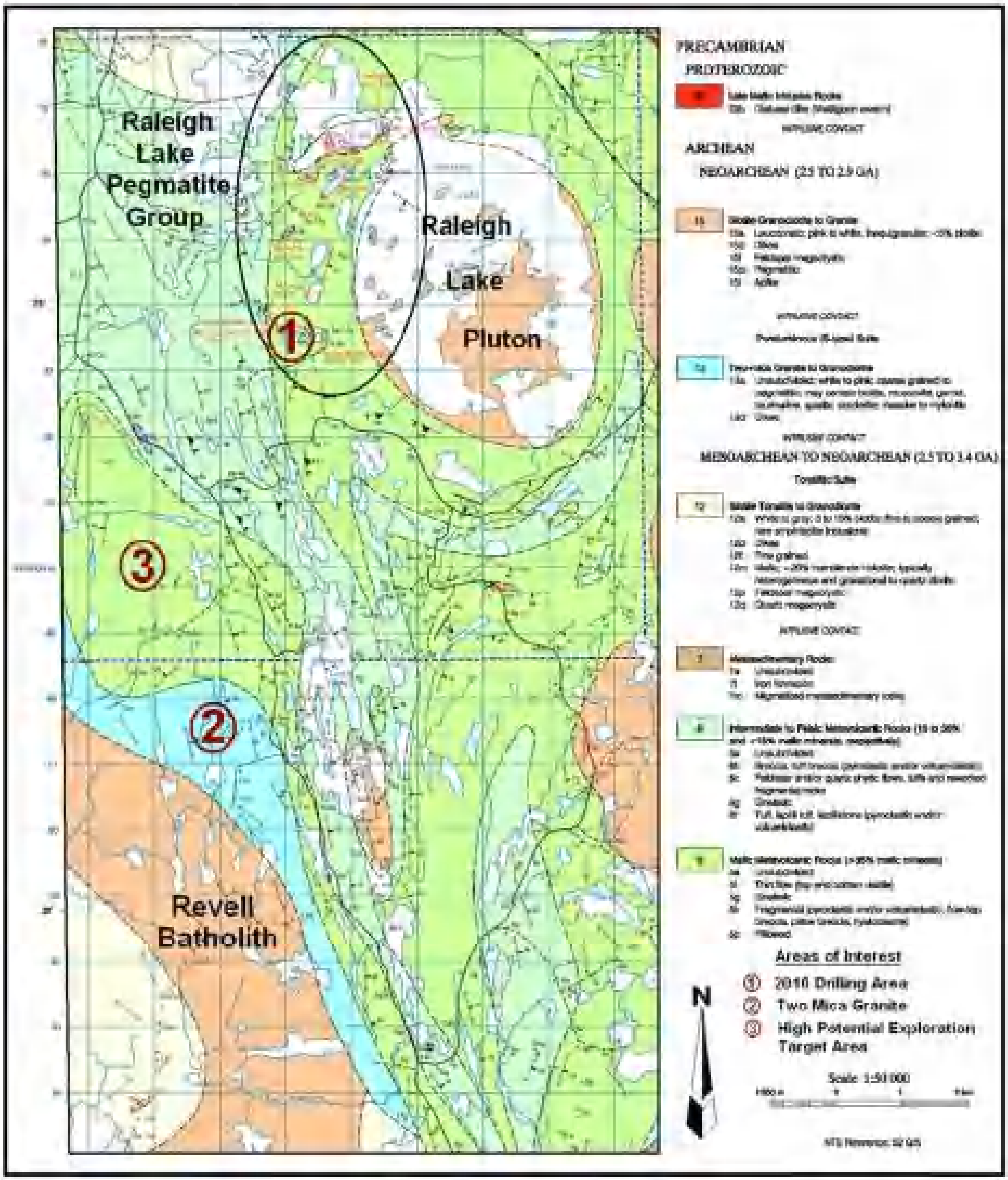


Figure 7-2: Geological Setting of the Raleigh Lake Rare-element Pegmatites (Vanstone Geological Services, 2010)

7.2.1 Mafic To Intermediate Meta-Volcanic Rocks

The metavolcanics in the Raleigh Lake area are comprised predominantly of meta-basalts (which are likely flows) and interbedded pillowed horizons. Where observed, pillows range from exceptionally preserved and undeformed, to highly flattened and recrystallized. These mafic units are intercalated with lighter coloured, more siliceous, volcanic rocks that range in composition from calc-alkalic basalts to rhyolites. All varieties are generally fine-grained, semi-massive, with moderate foliation, and are dark green-grey to yellow-grey in colour. Chloritization ranges from absent to strong and is present in zones of intense silica flooding and remobilization, in large part due to metamorphic recrystallization. In sections of silica flooding breccia textures are common, along with the hematization of fine-grained, disseminated sulphides, and can be observed in drill core from the 1998 and 2001 drill programs. Quartz veins commonly contain epidote and ankerite. Mafic units are locally moderately to strongly magnetic in the presence of disseminated pyrrhotite. Calcareous horizons are also common and occur with silica flooding. These horizons contain distorted nodules and bands of quartz-epidote-calcite-diopside-grossular garnet. The grossular garnets are commonly very coarse and range in size up to several centimetres.

7.2.2 Felsic To Intermediate Intrusive Rocks

Feldspar porphyries have been noted mainly in the vicinity of the 1998 drilling program near Microlite Pond, but also occur randomly as narrow cross-cutting dykes distributed throughout the property. They are massive, medium-grained, and medium to dark grey in colour, with 1 to 2 mm subhedral feldspar phenocrysts. The matrix is aphanitic to fine-grained, commonly with fine-grained biotite and locally disseminated sulphides. They are generally unaltered and are associated with local zones of silica flooding and brecciation. Trace to minor pyrite, pyrrhotite, and chalcopyrite occur in the siliceous zones. The porphyries are granodioritic to dioritic.

7.2.3 Pegmatites

The rare-metal-bearing pegmatite dykes on the Property occur in a south¬-southeast striking zone approximately 1.5 kilometres wide and at least 4 kilometres long with a trend of tantalum-mineralized albitic dykes occurring south of Raleigh Lake. The main pegmatite trend, which includes Pegmatite 1 through 3 and the Johnson Pegmatite, belong to the albite-spodumene sub-type of rare metal pegmatites. These pegmatites are at least partially hosted within a sheared, coarse grained gabbro which is a common feature of rare-element pegmatite fields. Most of the pegmatites trend north-northeaster and dip to the east. The dyke mineralogy consists of K-feldspar + albite, including secondary cleavelandite, quartz and spodumene. Accessory minerals identified include microlite, tantalite, and bismuthinite.

The known pegmatites form shallowly to moderately dipping, north-northeast trending, undeformed sheets with a significant potential for extensive lateral continuity. Strong fractionation of contained minerals and weak zonation within the pegmatites suggests that strongly enriched rare-metal zonation may exist within other domains of the pegmatites. The pegmatites are hosted in both mafic and intermediate volcanic rocks. Crude zonation is evident in the wider pegmatites, with albitic “wall” zones and “core” intermediate zones of albite–quartz-muscovite or spodumene-K-feldspar-albite. The pegmatite zonation can be broken down into six crudely defined zones (Table 7-1). This includes the upper wall zone, the upper intermediate zone, quartz + spodumene + k-feldspar core, lower intermediate zone, lower wall zone, and late-stage albite zone. These zones are not as distinct as in some other pegmatite occurrences, and spodumene distribution within the Raleigh Lake pegmatites is homogenous. Spodumene crystals are generally green to pale green in colour, exhibiting tan colours locally in the presence of albite. Grains typically range in size from < 1 cm to > 8 cm, and display ragged, corroded grain boundaries that have undergone complete replacement by a dark green, aphanitic, serpentine-like assemblage.

Table 7-1: Pegmatite Zone Mineralogy

Zone	Description
Upper Wall Zone	~0.5 m thick consisting of brick-red to pink k-feldspar with alteration of low temperature, late-stage albite, grey quartz, white to slightly pink albite and coarse (up to 1.5 cm) micas.
Upper Intermediate Zone	Characterized by a mineral assemblage of spodumene + k-feldspar+albite with minor albite and accessory mica, quartz content variable, spodumene laths mid-green in colour and indicative of a higher iron content, laths randomly oriented, albite white and mica coarse grained and brown.
Qtz+Spod+K-Feld. Core	Qtz with coarse grained, light green euhedral spodumene laths and coarse grained, euhedral buff k-feldspar
Lower Intermediate Zone	Finger grain size than previous zones, spodumene present but not as common as the core and upper intermediate zones. Pink k-feldspar, fine grained, white albite, green spodumene, greenish mica, and fine-grained, dark green to black tourmaline.
Lower Wall Zone	General absence of spodumene, and albite is more dominant than k-feldspar. Narrow sections of fine-grained, sodic aplite +/- fine grained tourmaline are noted in the core.
Late-stage Albite Zone	Vuggy, off-white to cream coloured, pristine cleavelandite albite + fine grained tourmaline, a result of late-stage in-fill cavities within lower portions of the pegmatite.

Several “main” pegmatites have been identified within the property. These include Pegmatite 1, 2, 3 and the Johnson Pegmatite. Pegmatite 1 has a minimum surface exposure of 200 m, an average width ranging from 3.90 to 8.00 m, and has been traced along strike for 300 m and down dip for over 400 m. Average Li_2O% are ~1.0% and up to 2.7% locally. It is crudely zoned with local strong albitization with heterogeneous intermediate zones consisting of light green to tan spodumene and K-feldspar in an albitic matrix with local muscovite. These zones are bounded by albitic "wall" zones. Pegmatite 1 is the widest and most laterally continuous pegmatite intercepted to date and forms a train of outcrops up to 10 m wide that extend for 200 m. Drilling has shown that Pegmatite 1 flattens down - dip from 15 - 20° easterly to a horizontal position. The country rock is albitized and contains exomorphic minerals of holmquistite and biotite adjacent to the dyke. This pegmatite, along with Pegmatite 3, is characterized by a strong crescumulate texture defined by elongate spodumene crystals up to 1.5 by 75 cm oriented normal to pegmatite contacts.

Pegmatite 2 was discovered and subsequently mapped by Breaks (1993) owing to the presence of several bright blue, holmquistite, bearing boulders nearby. The pegmatite is located approximately 800 m west of Microlite Lake and specifically noted to have a lithium dispersion halo greater than the Johnson Pegmatite at two metres. Lithogeochemical analyses show Pegmatite 2 to have the highest lithium assay of 2290 ppm to date.

Pegmatite 3, located SE of Microlite Lake, is exposed for ~50 m and is at minimum four metres thick at surface. It is crudely zoned with feldspathic wall zones and heterogeneous intermediate and "core" zones comprised of albite-quartz-muscovite, and spodumene-K-feldspar-albite. Diamond drilling has shown that Pegmatite 3 ranges up to 1.20 metres in thickness at depth but does show strong lateral continuity having been identified over approximately 300 m of strike length.

The Johnson Pegmatite, located 1400 metres north of Pegmatites 1, is exposed on surface for 83 metres along strike, with an apparent width of 3 to 4 metres. It consists predominantly of coarse white to pink K-feldspar and accessory muscovite and trace tantalum oxides. Diamond drilling of hole RL01-06 produced an average grade of 0.017% Ta205 over a core length of 2.65 metres.

8. DEPOSIT TYPES

8.1 Deposit Type

The Raleigh Lake project falls under the classification of rare-metal pegmatites. The Project can be further sub-categorized as albite-spodumene type pegmatites. Pegmatites are a common plutonic rock that contains abundant crystals with skeletal, graphic, or strongly directional growth-habits, or anisotropic layered mineral fabrics. Giant or megacrystic crystals can also be present, and the rare-element pegmatites contain anomalous and elevated Be, Li, Ta, Sn, and Cs. The Raleigh Lake Project contains lithium-caesium-tantalum ("LCT") pegmatites which are enriched in Li, Cs, Ta, Be, B, F, P, Mn, Ga, Rb, Nb, Sn, and Hf. Other examples of LCT pegmatite deposits include the Tin Mountain pegmatite in the United States, Tanco pegmatite in Canada and the Wodgina and Pilgangoora pegmatites in Western Australia. They are formed by fractional crystallization of an incompatible element-enriched granitic melt. Several factors control if barren granite will fractionate to produce a fertile granite melt (Černý, 1991; Breaks, 2003):

- Presence of trapped volatiles: fertile granites crystallize from a volatile-rich melt.
- Composition of melt: fertile granites are derived from an aluminum-rich melt.
- Source of magma: barren granites are usually derived from the partial melting of an igneous source (I-type), whereas fertile granites are derived from partial melting of a peraluminous sedimentary source (S-type).
- Degree of partial melting: fertile granites require a high degree of partial melting of the source rock that produced the magma.

Initially, fractional crystallization of a granitic melt will form barren granite consisting of common rock forming minerals such as quartz, potassium feldspar, plagioclase, and mica. Incompatible rare elements, such as Be, Li, Nb, Ta, Cs, and B do not easily fit into the crystal structure of common rock-forming minerals, and thus become increasingly concentrated in the granitic melt as common rock-forming minerals continue to crystallize and separate from the melt. At this point if the granitic melt is of a volatile-rich modestly peraluminous composition, further fractional crystallization will lead to fertile granite melt enriched in incompatible rare-elements/metals. The rare metals will remain in the melt until the last possible moment when they will crystallize as pegmatitic minerals such as spodumene, petalite, tantalite, columbite, etc.

After most of the fertile granite pluton has crystallized, the residual fractionated granitic melt that remains (as concentrates at the top of the pluton), can then intrude along rheological contacts, fractures, and faults into the host rocks to form pegmatite dikes. The forms of rare metal granitic pegmatite are greatly variable and are controlled mainly by the competency of the enclosing rocks, the depth of emplacement, and the tectonic and metamorphic regime at the time of emplacement.

LCT pegmatites most likely form in orogenic hinterlands related to plate convergence. Consequently, LCT pegmatites are hosted in metamorphosed supracrustal rocks (greenstone belts). The rare-metal pegmatites are regionally scattered throughout the boundary zone between the granitoid-dominant Winnipeg River Subprovince to the north and the greenstone-granite Wabigoon Subprovince to the south. This terrane is characterized by (Figure 8-2):

- Inverted stratigraphy and out-of-sequence thrust stacking of allochthonous terrane.
- Metavolcanic and metasedimentary assemblages, ranging in age from 2733±1 to 2703±2 Ma.
- Broad scale low grade metamorphism with locally moderate grade due to contact with granitoid intrusions.
- Zones of metasedimentary migmatite.
- Two-mica, peraluminous granite plutons distributed over 150 km.
- A distinctive metallogeny relative the adjacent Wabigoon Subprovince and Winnipeg River featured by widespread lithophile metal enrichment, which is in addition to rare-metal pegmatites.

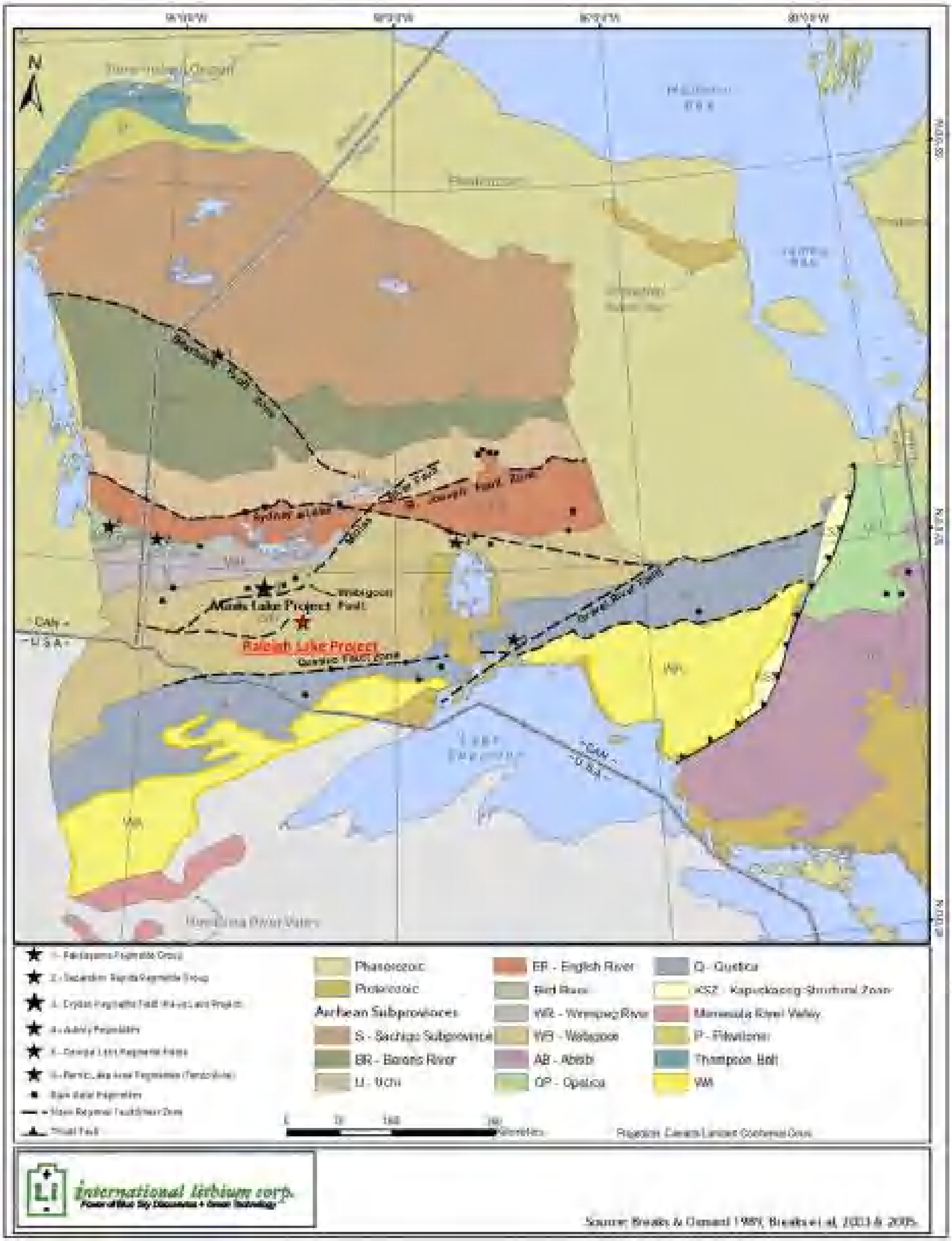


Figure 8-1: Location of the Raleigh Lake Project within the Superior Province

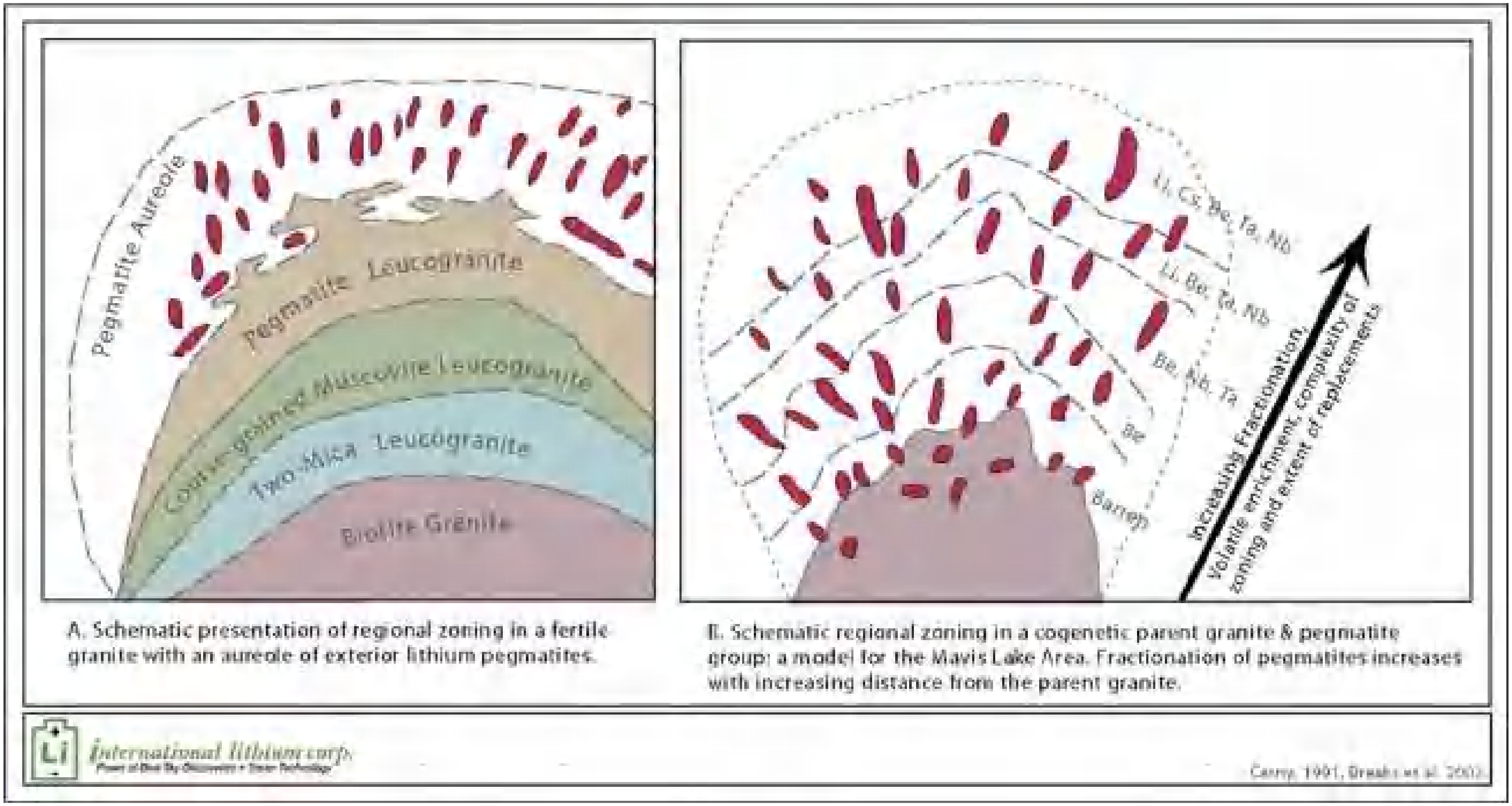


Figure 8-2: A) Schematic Representation of Regional Zoning of Fertile Granites, B) Schematic Regional Zoning of Cogenetic Parent Granite and Pegmatite Group

9. EXPLORATION

9.1 Historic Exploration

9.1.1 Avalon Ventures Ltd.

Avalon conducted an initial exploration program on the Property in 1998 which consisted of a lithogeochemical sampling program. The main objectives of this program were to sample granite plutons in the vicinity of Raleigh Lake for major oxide analysis, to sample mafic volcanics for analysis to determine fractionation trends, and to prospect for new rare-metal pegmatites in the vicinity of Raleigh Lake.

This regional reconnaissance program confirmed the presence of pegmatite bodies and the potential for discovering new pegmatites with economic concentrations of rare metals. After positive results of this initial field program, further follow up work was recommended including detailed lithogeochemical sampling, data compilation and interpretation, and diamond drilling.

Following the successful reconnaissance program in 1998, Avalon Ventures Ltd. carried out further work in September-October 1999 and consisted of line-cutting. Results of the program found anomalous tantalum values combined with very high rubidium assays and cesium assays and lead to a detailed exploration program funded by Global Canada Company.

In the fall of 2000 further detailed lithogeochemical surveying was conducted. This program contained 996 bedrock samples which identified three extensive north-south bedrock trends, which were thought to be unexposed pegmatite bodies, including Pegmatites 1 and 3. Sampling suggested a potential strike length extending at least 800 metres further north than originally suspected. There were also several Ta-enriched albitic dykes discovered that had values up to 0.021% Ta_2O_5. As a follow up, it was recommended that a stripping and trenching program should be carried out over the most predominant lithogeochemical anomalies to define any potential sub-cropping pegmatites. A summary map of work conducted by Avalon Ventures Ltd. is present in Figure 9-1.

9.1.2 Abaddon Resources inc.

In 2010, work on the Property was completed by Consolidated Abaddon Resources Inc. and included a magnetometer survey of the entire Property. This survey was carried out over the equivalent of 50 line kilometres, with readings at 12.5 metre intervals along lines spaced 50 metres apart. A map of total survey data is available in Figure 9-2. The magnetometer and base station were GSM 19T models manufactured by GEM Systems Inc.

The magnetometer survey was intended to identify structural elements that might have caused localization of more highly evolved magmatic fluids. A pronounced northwest trend was identified and combined with rock chips from Avalon lithogeochemical surveys, this trend aligned with elevated Li, Cs, and Rb. Two northeast trending faults were identified to contain the area wherein the known lithium anomalies lied. This confirmed the presence of a shallow dipping spodumene bearing pegmatite, along with drillhole data from the same exploration program. Based on these results, it was recommended that further drilling be performed on both the northwest trending structure in the northwest quarter of the claim bedrock and around drill hole RL10-7, which contained elevated tantalum values.

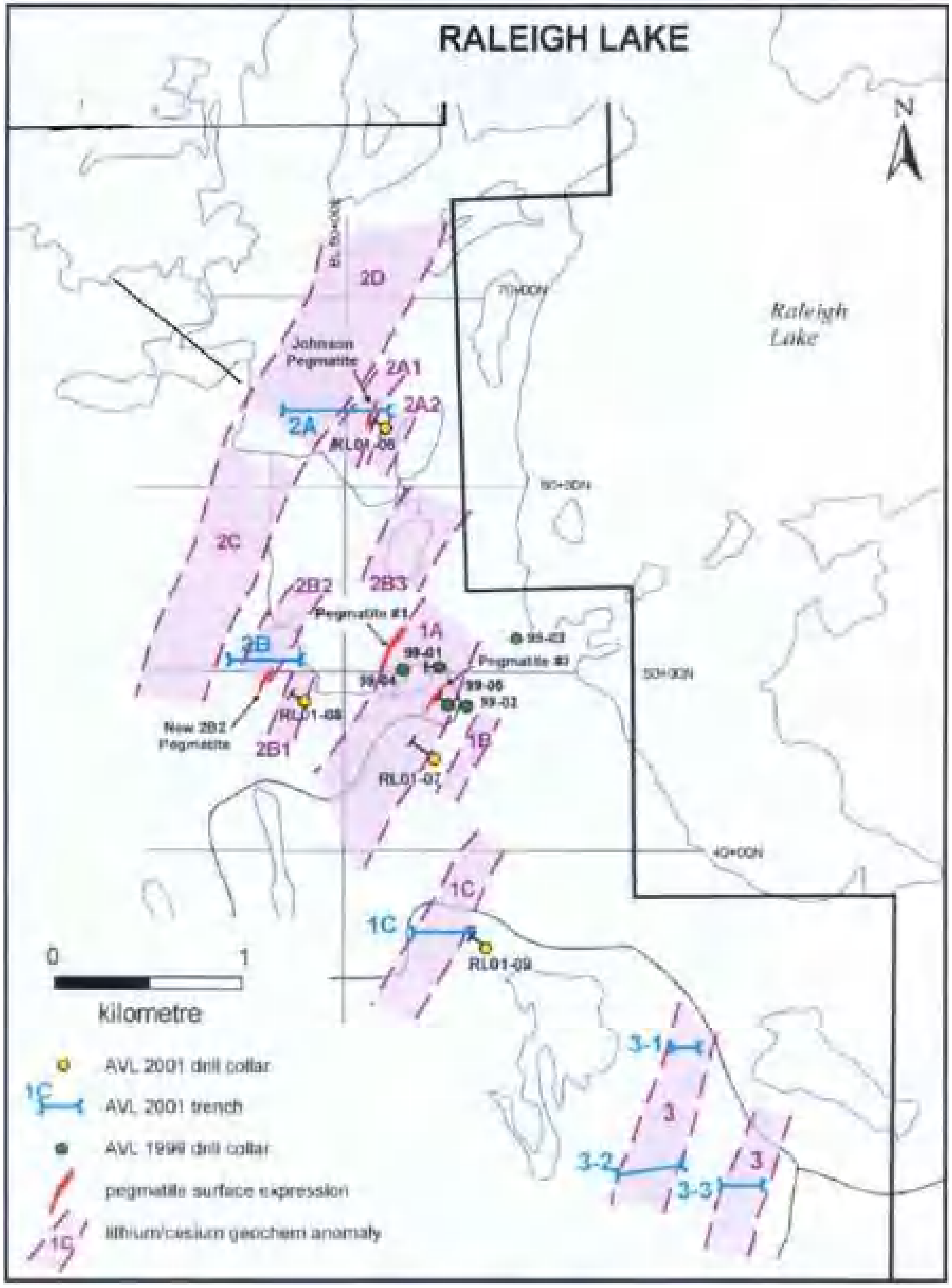


Figure 9-1: Avalon Ventures Ltd. Highlighting Main Geochemical Trends, Pegmatite Occurrences, Trench Locations and Diamond Drilling (Pederson, 2001)

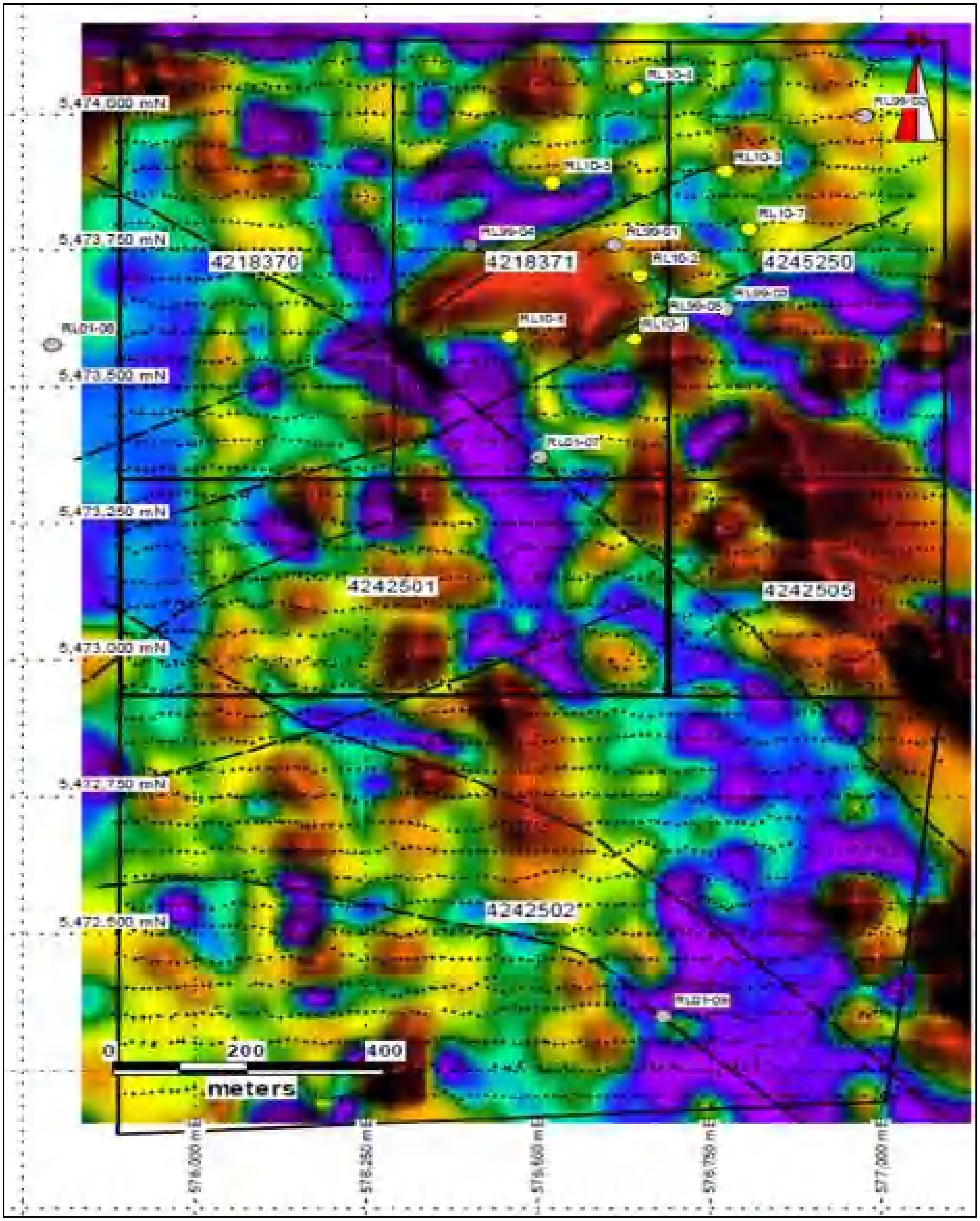


Figure 9-2: Magnetometer Total Field Survey Information Collected by Abaddon Resources Inc.

9.2 2016 International Lithium Corp.

ILC acquired the Project from Robert Fairservice in 2016, and after a quick reconnaissance visit with Pioneer Resources Corp. (renamed Essential Metals), an option agreement partner, and pegmatite specialist F. Breaks, ILC conducted an initial exploration program. The program consisted of an unmanned airborne ("UAV") magnetometer survey. The primary objectives of this initial program were to 1) confirm the presence of previously reported rare-element pegmatite bodies, 2) test metavolcanic host bedrock to delineate further geochemical trends and 3) conduct a low-cost, UAV magnetometer survey to conclude whether the technique could adequately differentiate between the highly contrasting geophysical properties of mineralized pegmatite and mafic metavolcanic host rocks.

ILC commissioned Pioneer Aerial Surveys Ltd. (formerly Pioneer Exploration Ltd.) to conduct a UAV magnetometer survey consisting of a total of 189.8 line kilometres of continuous profiling at 40 metre line spacing. ILC also commissioned another airborne UAV magnetometer survey to cover additional ground added to the Project in 2018, conducting an additional 560.62 line-km UAV potassium magnetometer survey in the spring of 2019. The total magnetic field results from this survey combined with the 2016 results are detailed in Figure 9-3.

The 2018 reconnaissance visit and exploration programs successfully confirmed the presence of lithium and rare-element enriched pegmatites across the property. The airborne magnetic data reinforced the significance of the Projects mineral potential. The following proposal was designed as a follow up:

- It had been previously determined from a property visit in 2010 that pegmatite 1 is hosted in a coarse-grained, sheared gabbro. Pegmatite 1 (and potentially other pegmatites) are emplaced within a reactivated dilational environment. This added additional merit to the evaluation of the extent of mafic intrusions within the Property.
- Although the historical drillhole RL01-09 previously drill-tested the Li-Cs geochemical anomaly and trench 1C, further ground-based work was required to evaluate the target or reposition and core another hole near the anomalous bedrock samples.
- It was determined that a further evaluation of the magnetic line profile data from Pioneer/ILC, particularly in the immediate vicinity of known pegmatite occurrences, was warranted.
- Further lithogeochemical sampling to extend the southwest corridor of geochemical trend 2C was required along with mapping and prospecting.
- A diamond drilling program of up to 1200 metres in 8 to 10 holes was recommended to effectively drill test the lateral extension of the Projects main pegmatite bodies showing anomalous lithogeochemistry.
- It was recommended that diamond drilling similarly drill test the most significant coincidental structural features that are associated with anomalous lithogeochemistry.
- Although geochemical trends and pegmatites occurrences are the most prevalent in the central and northwestern sections of the Project area, fractionation trends are to the southeast and efforts should continue to explore for stratigraphically higher pegmatites in this area.

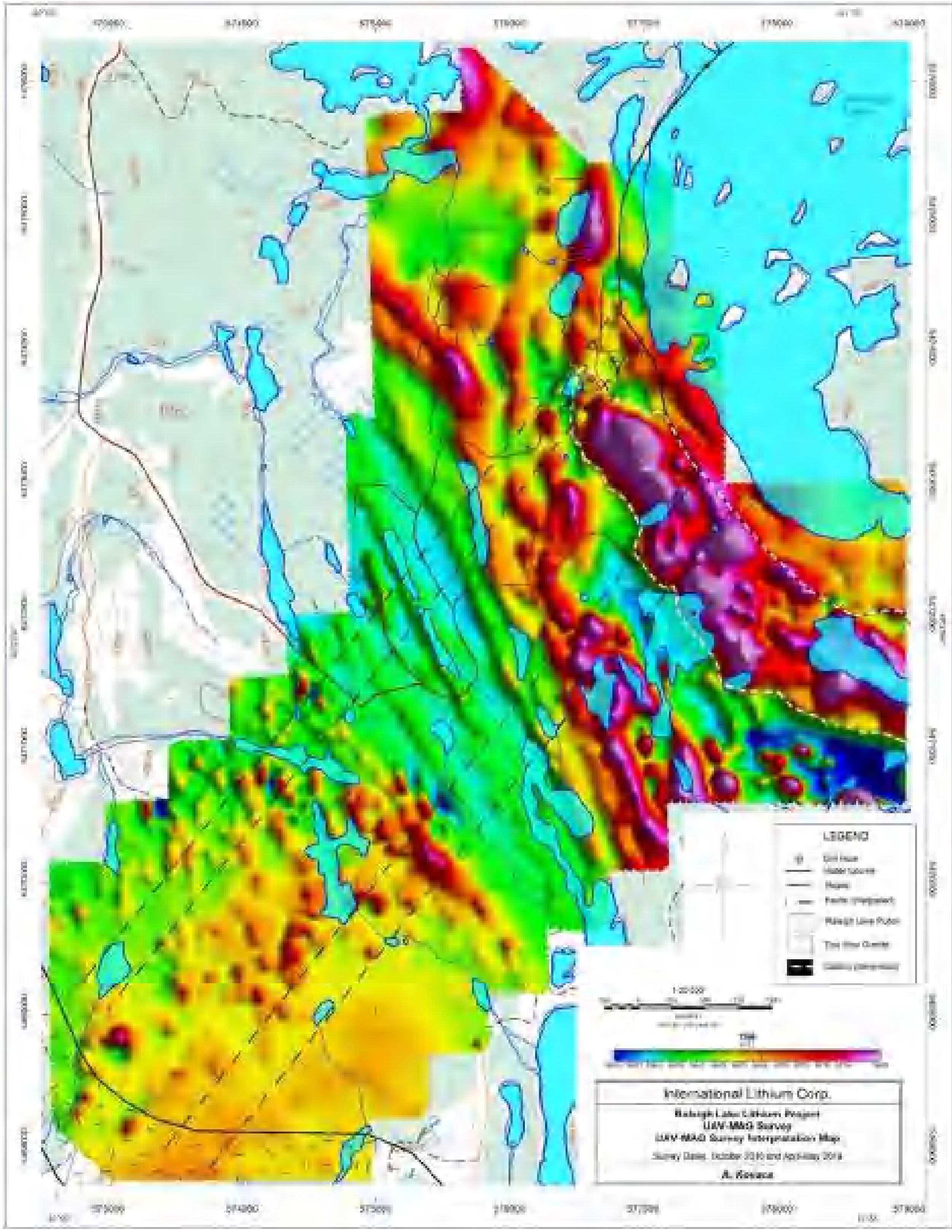


Figure 9-3: Total Magnetic Intensity (TMI) Map with Combined Results from the 2016 and 2019 Airborne Surveys (McLaughlin, 2019)

9.3 2021 International Lithium Corp.

ILC commissioned Coast Mountain Geological Ltd. ("CMG") to facilitate several ground surveys across the Property after ILC significantly expanded their land holdings during 2021. The collection of surveys within the proposed works consisted of:

- Regional lithogeochemical sampling component within the Raleigh Lake Block adjacent to and complimenting the historical bedrock sampling data working outwards from the Raleigh Lake Pegmatite Field ("RLPF").
- Biogeochemical orientation survey around and over Pegmatites 1 and 3 as well as Zone 1 and Microlite Lake exploration areas.
- Regional mapping, prospecting, and channel sampling of cut blocks and near mineral occurrences within the claim group designed to evaluate both the base and rare-metal potential.

Field crews of geologists and geotechnicians from Coast Mountain Geological Ltd. conducted work between September 28th to October 31st of 2021. The details of each survey are described below.

9.3.1 Lithogeochemistry Program

A total of 1,089 lithogeochemistry samples were collected between September 29th, 2021, and October 30th, 2021, with a primary goal to test for exomorphic dispersion of lithium and other rare-metals into the volcanic, volcaniclastic, and subvolcanic intrusions of the host Raleigh Lake greenstone belt. The bedrock sampling program also ensured a systematic examination of outcrops are completed while sampling and conducting traverses. The general location of sampling with respect to the claim area and historical sampling grids is identified in Figure 9-4.

All samples were directly delivered by company personnel to Activation Labs in Dryden, ON. Samples were processed by an RX-1 preparation followed by aggressive digestion by sodium peroxide fusion with an ICP-OES and ICP-MS finish with 55 elements (Code UT-7, up to 5% Li).

Batch were delivered across the month of October. Sampling lines were designed to seamlessly mesh into the grids of historical work programs. The sampling procedure at each site mirrored historical procedures.

Sampling procedures consisted of the collection of up to 2 kg as a composite of bedrock samples within a 1-2 m search radius of the predesigned sampling site. Lines were spaced approximately 200 m apart and station density was every 50 m along the line +/- 25 m but was highly dependant on exposure.

9.3.1.1 2021 Lithogeochemistry Results

Lithological determination from the field teams classified the rocks into 5 primary types for analysis:

- Intermediate to Mafic Metavolcanic rocks: 1034 sites (95%);
- Felsic to Intermediate Metavolcanic rocks: 33 sites (3%);
- Mafic to ultramafic metavolcanic rocks: 12 sites (1%);
- Massive granodiorite to granite: 9 sites and (1%); and
- Metasediments: 1 site (0.01%).

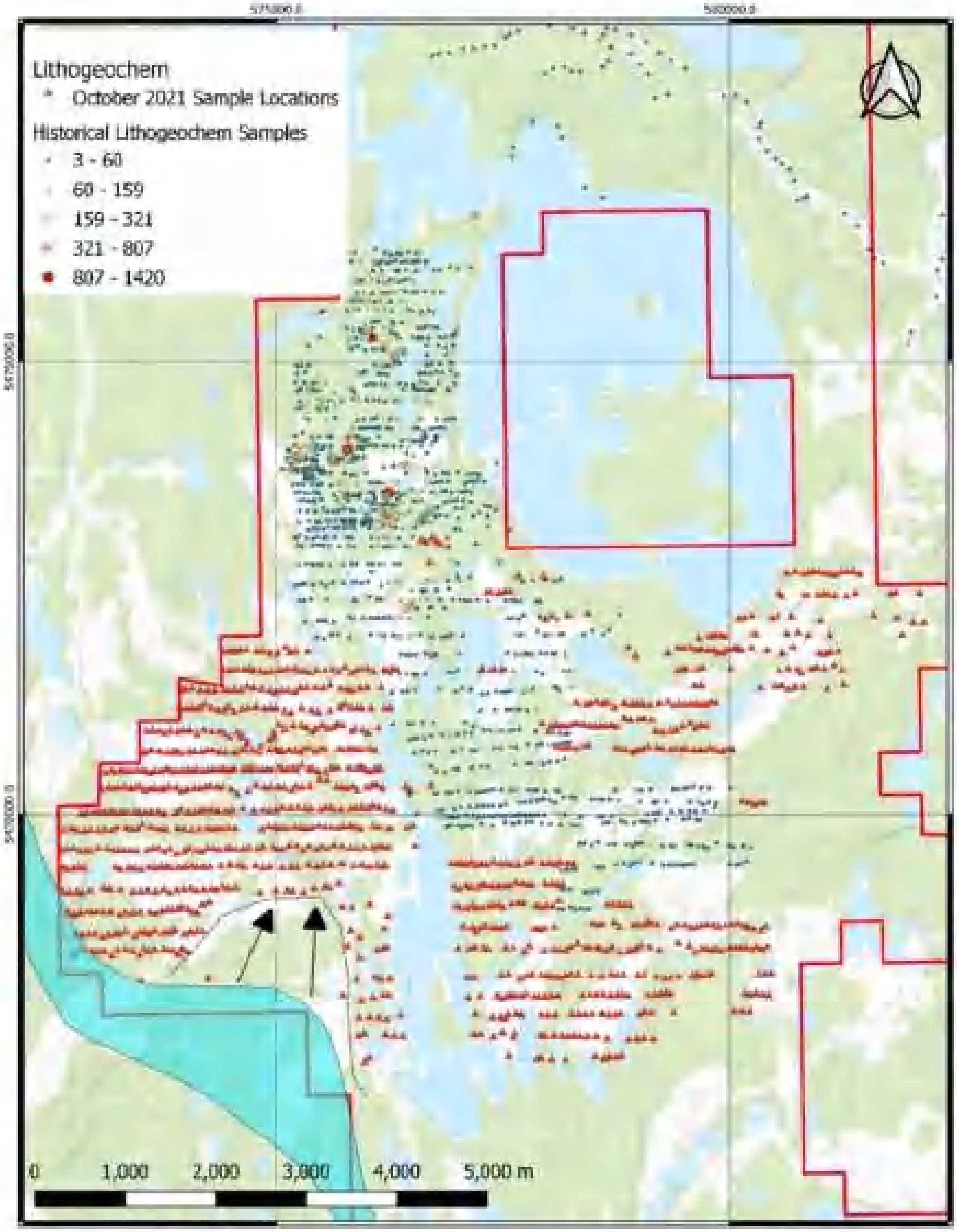


Figure 9-4: Lithogeochemistry Sampling Coverage From 2021 – Note Inflection of the Sampling around the Cupola of the Two-Mica Granite in the West That More Accurately Reflects the Contact of the Two-Mica Granite and the Volcanic Pile Where it is Improperly Mapped in the 1-250K Regional Bedrock Geology Data

The threshold determination for each of rock type was defined in part from Galeshuck (1999) and determined from an analysis of results from an extensive regional program conducted within the Bird River greenstone belt around the Tanco Deposit (Figure 9-5). Over 98% of sampling sites from this program were within volcanic/volcaniclastic terrain, 95% of which of were comprised of intermediate to mafic composition. The “background” and “possibly anomalous” limits were blended in each element group between volcanic/volcaniclastic compositions to simplify threshold classification and anomaly recognition in the “anomalous” and “highly anomalous” categories.

Rare Metal Assay Result Comparison

Lithogeochemical statistically derived anomaly threshold values from the Separation Rapids Project.

Rock Type	Background (1)			Possible Anomalous			Anomalous			Highly Anomalous (?)		
	Li	Cs	Rb	Li	Cs	Rb	Li	Cs	Rb	Li	Cs	Rb
Felsic Volcanic	37	5	78	13-36	5-8	[illegible]	39-160	8-22	160-807	160	22	807
Mafic Intrusive	46	4	30	46-54	4-6	30-38	54-135	6-15	38-134	135	15	133
Mafic Volcanic	38	1	73	34-58	2-6	73-101	60-228	6-40	101-485	2278	40	485
Felsic Intrusive	27	7	102	27-42	7-13	102-483	42-66	13-25	485-1769	66	25	1769
Pegmatite	59	12	236	59-116	12-16	236-321	116-181	16-126	321-2583	583	126	3983

Galeschuk, C.R. 1999. Report on the 1996 and 1997 lithogeochemistry and geological mapping activity Separation Lake, Ontario, summers 1996 and 1997; Kenora District Geologist's office, assessment file 52L07SE Q-3, AFRO 2.19810 Tantalum Mining Corporation of Canada Ltd.

Figure 9-5: Rare-metal Threshold Determination for Lithogeochemistry Exploration Programs (Source: McLaughlin, 2019)

The current results were not merged with the existing historical datasets. There was some concern over a bias in the existing set imposed by the proximity and sampling density to and within the Raleigh Lake pegmatite field which could mask and deprioritize anomalies form the current results from 2021.

Lithogeochemical program results coincided well with the existing exploration models and prospective corridors.

Lithophile enrichment and dispersion profiles are generally restricted to distances of no more than 10 m in LCT pegmatite systems and volcanic greenstone belts. Syngenetic faults and joints sets around pegmatites during emplacement may promote transportation of fluids and increase dispersion profiles, however their dispersive effect would be localized to the transport mechanism and structures. Up to ten anomalies were recognized during the analysis and their general locations are mapped out in Figure 9-6. Out of the list of defined anomalies, approximately 3 areas have emerged as tier 1 exploration targets.

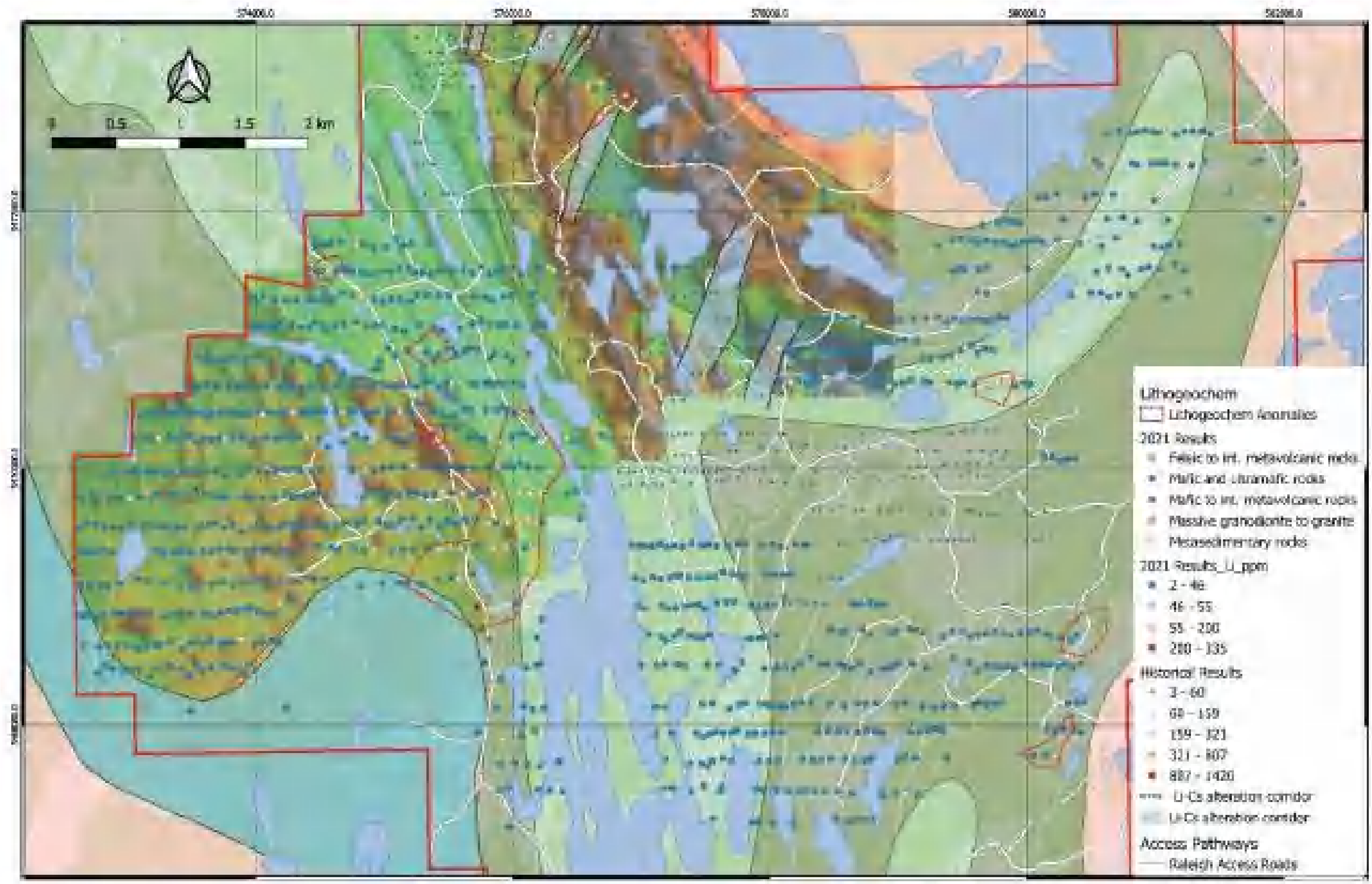


Figure 9-6: Approximate Location of Lithogeochemical Anomalies within the Survey Area; Lithium Results are Mapped in this Figure (Source: Mclaughlin, 2019)

9.3.2 2021 Biogeochemistry Orientation Survey

A four-line, 65 station vegetation orientation survey was conducted on the Raleigh Lake Property between September 29th and October 1st of 2021.

9.3.2.1 Survey Layout

Figure 9-9 demonstrates four survey lines oriented normal to the projected strike of pegmatites. Lines 01 and 02 are believed to transect and potentially extend surface exposures of Pegmatite 1 and 3, while Line 03 was designed to determine if a blind pegmatite can be traced further east, and Line 04 transects several drill-tested, narrow, closely spaced, spodumene-bearing pegmatites.

The line-spacing was set at 100 m with lines 01 through 03 and the station density was set at a maximum of 25 m that was reduced to 12.5 m over the known or surface projections of pegmatite 1 and 3.

9.3.2.2 Sampling Criteria

About 100 g of bark-strip, consisting of predominantly bark with lesser outer cambium, was collected from black spruce trees (*picea mariana*) within a 3 m search radius of the pre-defined station.
Sample preference was given to trees with a larger trunk, longer growing history, and wider root base for anomaly catchment and recognition. Bark was collected from around chest height from each tree location. A standard sample site within the survey area is shown in Figure 9-7 and Figure 9-8.

To compliment the bark samples, leaves and twigs from the outer 30 cm of branches of alders were also collected. The alder samples were stored and can be analyzed if necessary. Black spruce bark was collected from 64 of the 66 sites. Alder samples taken were in lieu of bark for sites Line 02 Stn 07 (sample 277621) and Line 03 Stn 12 (sample 277647). The survey layout is mapped in Figure 9-9

Figure 9-7: Line 02 Site 05 above Pegmatite #1; Fairly Standard Site in Elevated Terrain; a Strong Presence of Both Alder, Spruce Bark to Select from in Addition to Poplar and Birch That Are Not Included in the Sample Medium

Figure 9-8: Black Spruce Bark and Typical Sample Strip; Notice Small Red Spot Pattern within the Outer Cambium Which Was a Consistent Feature on Most Samples

Figure 9-9: Layout of Biogeochemistry Orientation Program (Source: McLaughlin, 2019)

9.3.2.3 Analysis of Results

Element anomalies were set at the 80, 90, 95 and 97th percentiles and results are plotted for K, Rb and Cs. The lithium response within the black spruce bark samples by the selected testing methods is low and ranges from the detection limit to 0.2 ppm, thus were not included in the analysis. The elemental response of Cs is displayed in Figure 9-10 and the resultant map indicate there is a spatial relationship between elevated Cs and the known bedrock occurrences of Pegmatite #1 and the narrow, spodumene-bearing pegmatites identified in Line 4.

Figure 9-10: Cs Results (red circles) across the Survey Area within Zone 1 (Source: Nordmin, 2023)

9.3.3 Rock Sampling and Mapping Prospecting

A total of 57 grab samples were collected during the 2021 prospecting program. Forty-nine were rare-metal specimens and the remaining eight were sent for base-metal analysis. Figure 9 11 shows the spatial location of all the grab samples and areas mapped from this program and from the mapping program along with the ILC claim boundary.

Cursory mapping was conducted, designed to coarsely evaluate the highest priority forestry clear cuts with the primary purpose of identifying rare-metal potential. More than twenty new pegmatite sites were identified within the White Otter exploration area. The most abundant pegmatite type in this area were composed predominantly of megacrystic potassium feldspar, smoky quartz, and dark muscovite pegmatite. Rare molybdenite and rarer purple fluorite were observed which increased in abundance from west to east. All of these identified pegmatites were sampled.

The second style of pegmatite identified were comprised of albite-muscovite-quartz-garnet, and primarily occur as mafic volcanic or bedded to laminated volcaniclastics. These are quite different visually from the first style, presenting as bright white and less coarse-grained. At the eastern edge of the claim block, a metre-scale, shallow, north-dipping microcline pegmatite contacts or has a core of the albite-muscovite type. Muscovite at this location is green/bronze, accompanied by orange spessartine garnets.

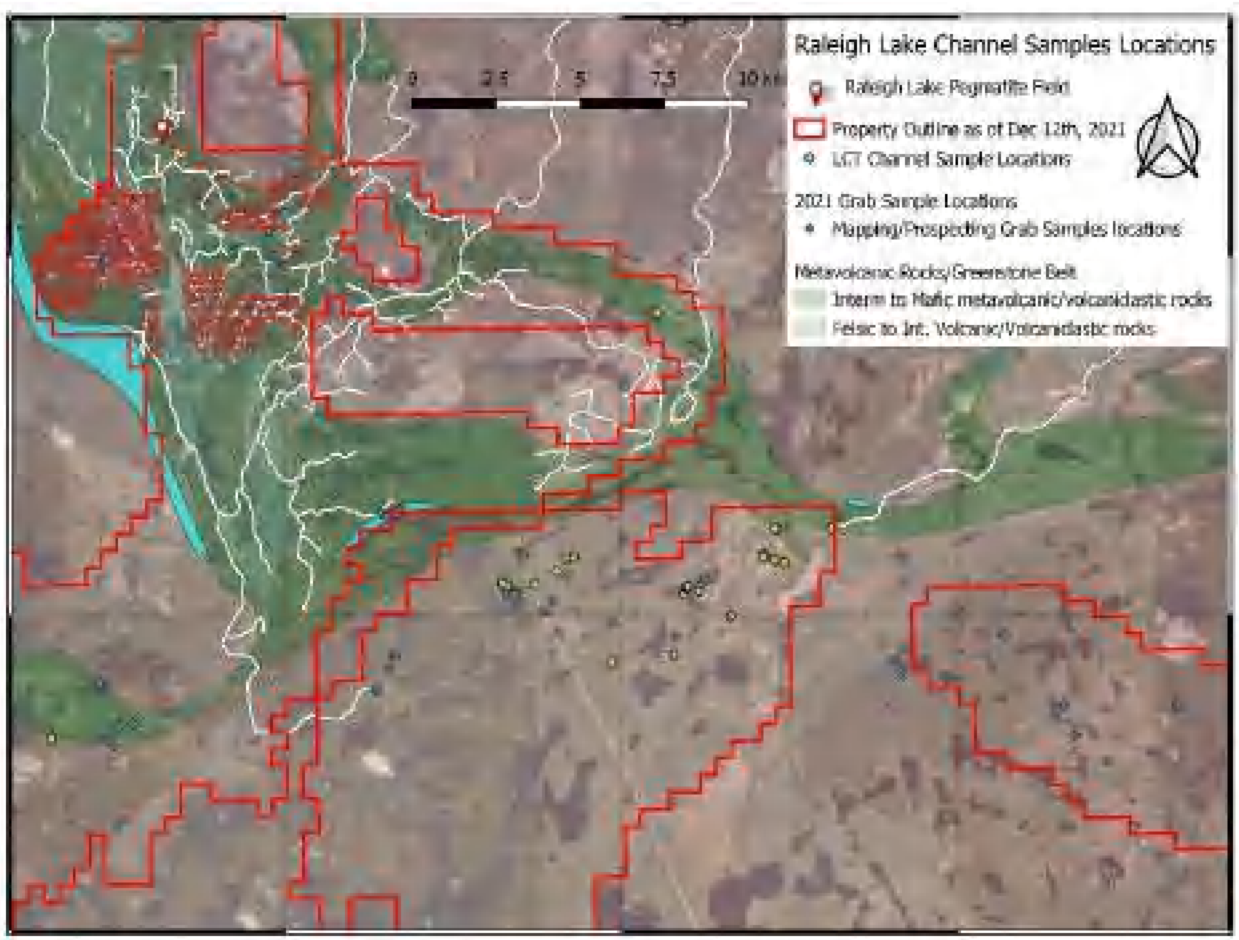


Figure 9-11: Spatial Distribution of Grab and Channel Samples within the Most Recent Raleigh Lake Claim Fabric; the Location of New Pegmatite Discoveries Correspond to Channel Sample Locations within the Image

Twelve new pegmatites of significant size (> 1 m apparent width on surface) were located in the continuous block of claims northwest of the Balmoral River in the Raleigh Lake Block. These new pegmatites were all sampled and at surface level contained no rare-metals of appropriate volume or rare accessory minerals.

9.3.4 Channel Sampling

A total of 40 channel samples were collected across the claim group and included 23 from rare-metal prospects and new pegmatites discoveries identified while mapping this season, as well as 7 from base-metal and gold prospects. The nature and mechanical properties of the sampling sites for channels did not lend themselves to grab sampling however their measured locations can be treated as mapping samples.

10. DRILLING

10.1 Drilling Summary

Pegmatites within the Project property generally strike northeast and dip moderately to the southeast (generally 25°-75°). Drilling is therefore generally directed northwest and at a steep near-vertical dip to obtain the true thickness of mineralization. Three main pegmatites are recognized on the property and drilling has focused on testing their lateral continuity, with Pegmatite 1 shown to be the most laterally continuous from both historic and current drilling (Figure 10-1). The focus of recent programs has been to increase drill density within Pegmatite 1 and to further test continuity and thickness of Pegmatites 2 and 3. A summary of recent and historic drilling on the property is provided in Table 10-1.

Table 10-1: Summary of Drilling on the Raleigh Lake Deposit Property

Year	Operator	Number of Holes	Hole Diameter	Total Metreage
1999	Avalon Ventures Ltd.	5	NQ	602
2001	Avalon Ventures Ltd.	4	NQ	752
2010	Abaddon Resources Inc.	7	NQ	1,463.5
2021	ILC	8	NQ	1,504
2022	ILC	56	NQ	9,496.46

10.1.1 1999 to 2001 Avalon Ventures Ltd.

The 1999 drilling program was based off a successful lithogeochemical exploration program conducted in 1998. This program consisted of five diamond drill holes totalling 602 metres designed to test the extension of known pegmatites on the property, and additionally to delineate new “blind” pegmatites. The drilling confirmed the presence of stacked, flat to gently dipping pegmatites at Raleigh Lake. Pegmatite 1 was intersected in four of the five drillholes confirming lateral and down-dip continuity of surface exposed pegmatites.

In 2001 Avalon Ventures Ltd. conducted a follow up summer drill program consisting of four holes totaling 752 metres. This program was used to test the main pegmatite occurrences at depth, and several lithogeochemical anomalies. Three of the four holes intersected multiple shallow-dipping pegmatites ranging up to 1 m in thickness. No new pegmatites were discovered.

10.1.2 2010 Abaddon Resources Inc.

In addition to a 50-line kilometre ground magnetometer survey, Abaddon Resources completed a seven hole drill program from February to March of 2010; total drilling was 1,463.5 metres. The drilling confirmed the previous intercepts of lithium bearing pegmatites in the area West of Raleigh Lake. All 2010 drillholes intersected Pegmatite 1.

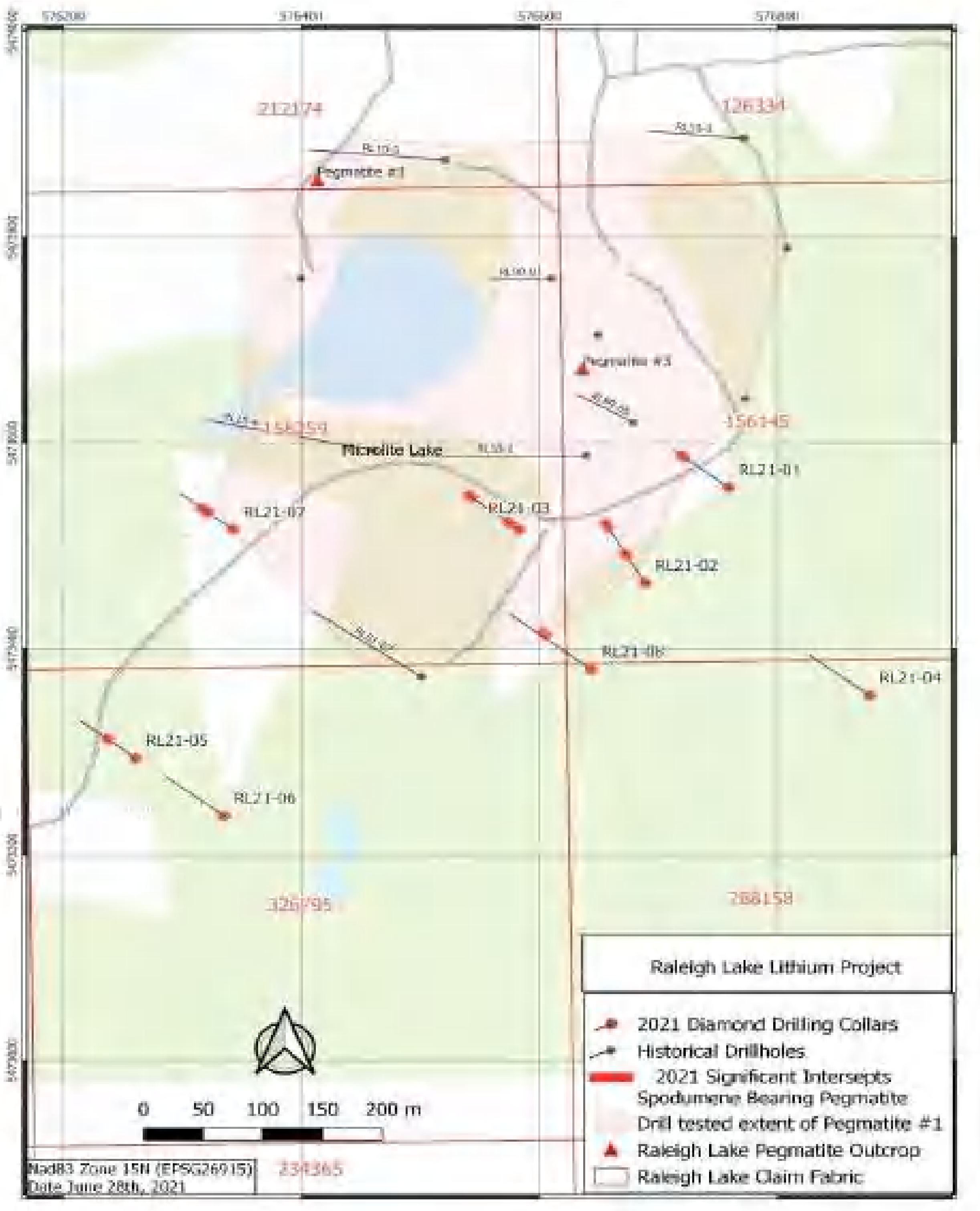


Figure 10-1: Layout of 2021 Diamond Drilling Locations within the Project

10.1.3 2021 International Lithium Corp.

ILC initially acquired the Project in March 2016 and a drill program commenced in March 2021 to aid in the definition of a modern mineral inventory on the project. A total of 1,504 metres of oriented NQ diamond drill core was completed on the Raleigh Lake property between March 19th, 2021, and April 5th, 2021. Geological support crews provided by Coast Mountain Geological Ltd. arrived on site to prepare for the program on March 12th, 2021, and supported the program out of the town of Ignace, Ontario. Drilling was completed by Rodren Drilling from Winnipeg using a skid-mounted Hydracore drill.

The collaring attributes from all holes cored on the property during the program are listed in Table 10-2. The drillhole spacing proposed for the program was designed to facilitate a fast and thorough evaluation of Pegmatite 1 and 3. The work program was completed under the terms laid out in International Lithium Corp.'s Exploration Permit PR-20-000001 and all drilling occurred within the boundaries of single cell mineral claims ID 156145, 158259, 326795 and 288158.

Table 10-2: Drill Hole Collar Attributes Table from the 2021 Drilling Program

Hole	Easting (m)[1]	Northing (m)[1]	Elevation (m)	Azimuth	Dip	Length (m)
RL21-01	576,759	5,473,557	474	308	-70	170.0
RL21-02	576,689	5,473,464	478	330	-70	209.0
RL21-03	576,583	5,473,516	468	308	-70	170.0
RL21-04	576,877	5,473,355	485	308	-70	185.0
RL21-05	576,261	5,473,294	479	308	-70	173.0
RL21-06	576,335	5,473,238	475	308	-70	176.0
RL21-07	576,343	5,473,516	472	308	-70	167.0
RL21-08	576,644	5,473,380	474	308	-70	254.0
Total						**1504.0**

Pegmatite 1 and 3 intersections from this season of drilling occurred somewhat deeper than what was projected by the model based on historic Avalon Ventures Ltd. and Abaddon Resources Inc. drilling, suggesting that Pegmatite 1 and 3 geometries have exceptional laterally continuity, however the geometries steepened slightly beyond the southern limits of historical drilling. This is also supported in the oriented core data on pegmatite contact orientations, as seen in Figure 10-2 which is a stereonet projection of pegmatite contact orientations. There are two primary distributions and a slight arc of data in between. Contact orientations of pegmatites occur in the moderate to shallow-dipping, east-trending domains which is dominant in the Pegmatite 1 and 3 area. Contact orientations of pegmatites from RL21-05, RL21-06, and RL21-07 occur in shallow to moderately-dipping and north-northwest to north-northeast trending domains. Pegmatites bodies in the Pegmatite 1 and 3 area generally occur at a moderate to low angle to bedding/foliation planes as seen in Figure 10-2.

RL21-04 appears to have been cored into a subsided block of highly magnetic metavolcanic rocks. The step-out distance to RL21-04 is nearly 200 m away from RL21-01 and RL21-02.

[1] Collar coordinates are positioned in NAD 83 Zone 15N (EPSG:26915).

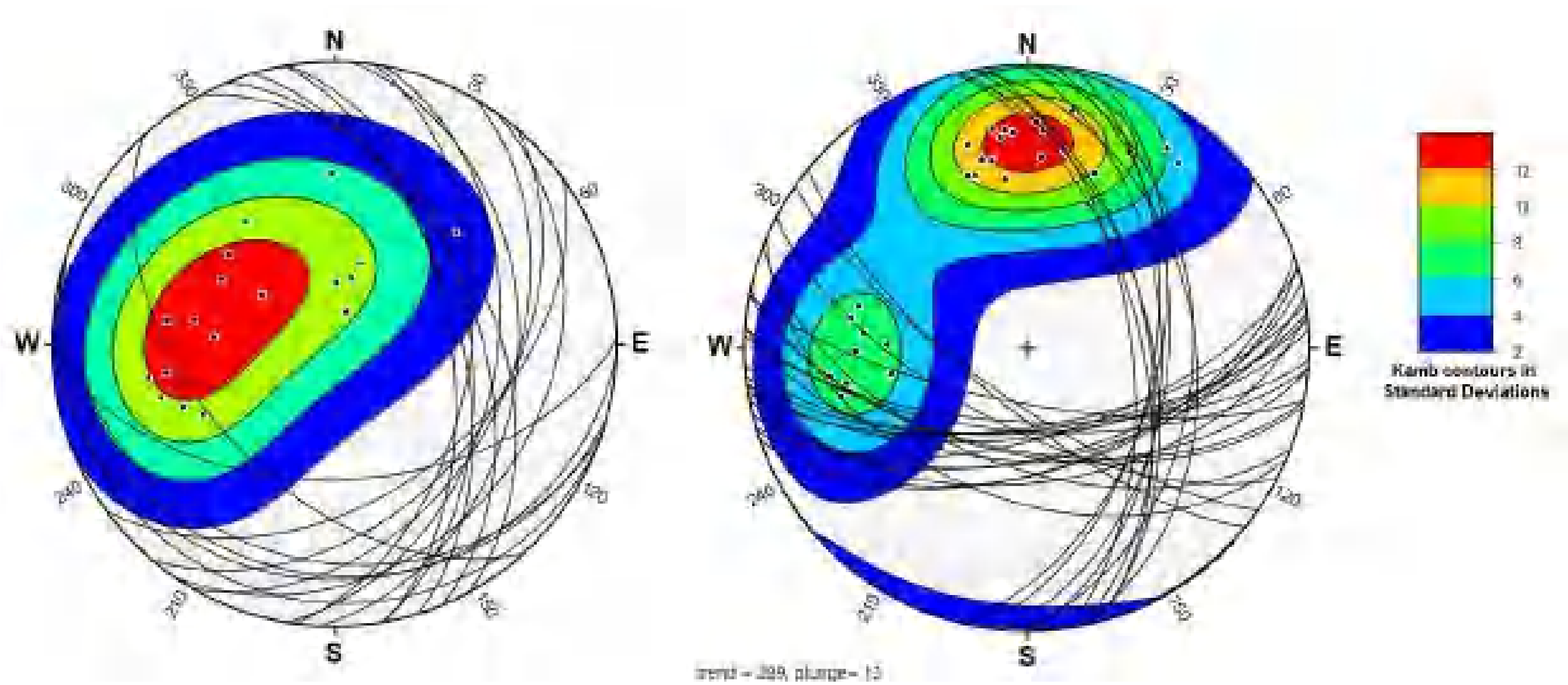


Figure 10-2: Stereonet Projection of Pegmatite Contact Orientations (left) and Metavolcanic Bedding Plane (right) Results from 2021 Drilling

10.1.4 2022 International Lithium Corp.

The ILC 2022 drilling program was divided into 3 phases; while each phase of drilling had a different focus, all phases concentrated on delineating the geological extents of known pegmatites for the Project's Mineral Resource Estimate (Figure 10-3).

The objective of Phase I was to target widely spaced gaps in the geological model of Pegmatite 1 based on historical and 2021 ILC drilling results. As well, Pegmatite 2 was largely untested, and narrow, Li-enriched pegmatite veins were intercepted that corresponded with spodumene-bearing surface occurrences of Pegmatite 2.

Table 10 3 and Table 10 4 outline drillholes from Phases I, II, and III drilling.

Phase II began with drilling distant exploration holes to target geochemical anomalies, including drillholes RL22-19 through RL22-24. Remaining holes strongly focused on the high-grade centre of Pegmatite 1 with higher density drill spacing and a grid of approximately 50m.

Phase III began in the fall of 2022 and focused on delineating the shallower northern/northeastern extents of Pegmatite 1. Structural models and concepts were also tested during this phase of the 2022 drill program.

Table 10-3: ILC 2022 Drill Program Breakdown by Phase

Phase	Program Start Date	Program End Date	No. of Holes	Total Metreage
Phase I	March 12th, 2022	April 15th, 2022	13	1,973.46
Phase II	May 9th, 2022	July 5th, 2022	26	4,198
Phase III	September 27th, 2022	November 9th	20	3,325

Table 10-4: International Lithium Corp. 2022 Summary of Drilling

Drillhole ID	Northing	Easting	Elevation	Azimuth	Dip
RL22-09	576714.9	5473645	477.782	315	-75
RL22-10	576718.4	5473645	477.882	20	-50
RL22-11	576715.4	5473647	477.752	290	-55
RL22-12	576708.1	5473647	477.576	63	-63
RL22-13	576740.9	5473619	475.165	315	-75
RL22-14	576646.5	5473582	474.479	300	-70
RL22-15	576540.9	5473606	471.975	50	-50
RL22-16	576536.9	5473601	472.084	315	-50
RL22-17	575973	5474025	476.193	300	-60
RL22-18	575940	5473889	463.986	305	-50
RL22-19	575391.6	5474665	461.053	315	-50
RL22-20	575577.5	5474468	460.187	290	-50
RL22-21	575529.4	5474531	458.308	315	-50
RL22-22	575480.8	5474545	460.606	315	-50
RL22-23	575293.1	5473901	468.75	280	-50
RL22-24	575318.4	5473994	468.449	285	-50
RL22-25	576827.4	5473597	480.143	315	-75
RL22-26	576791.9	5473573	476.9	315	-75

Drillhole ID	Northing	Easting	Elevation	Azimuth	Dip
RL22-27	576754.3	5473674	481.731	310	-50
RL22-28	576757.3	5473674	481.664	35	-60
RL22-29	576755.1	5473672	481.709	305	-70
RL22-30	576718.5	5473585	477.407	310	-70
RL22-31	576742.7	5473504	479.673	315	-70
RL22-32	576672.8	5473551	477.928	315	-70
RL22-33	576793.1	5473641	480.763	315	-80
RL22-34	576745.5	5473651	478.109	315	-70
RL22-35	576770.3	5473633	478.173	315	-70
RL22-36	576710.1	5473618	474.733	315	-70
RL22-37	576690.2	5473637	474.135	310	-70
RL22-38	576692.5	5473567	478.496	315	-70
RL22-39	576658.7	5473601	474.077	315	-70
RL22-40	576724.1	5473671	480.058	305	-60
RL22-41	576724.5	5473671	480.142	355	-58
RL22-42	576621.3	5473543	474.473	315	-70
RL22-43	576652.8	5473512	477.617	315	-70
RL22-44	576616	5473478	479	315	-70
RL22-45	576753.2	5473739	477.924	310	-70
RL22-46	576674.7	5473744	477.032	310	-70
RL22-47	576644.8	5473767	476.396	310	-70
RL22-48	576698.1	5473785	477.616	310	-71
RL22-49	576676.2	5473810	477.593	310	-70
RL22-50	576603.8	5473813	480.384	310	-70
RL22-51	576603.9	5473813	480.48	262	-50
RL22-52	576750.8	5473812	477.959	310	-70
RL22-53	576713.9	5473842	477.813	300	-70
RL22-54	576648.2	5473842	477.249	310	-70
RL22-55	576608.8	5473890	478.223	310	-70
RL22-56	576655.5	5473927	479.307	310	-70
RL22-57	576807.4	5473949	487.633	310	-70
RL22-58	576813.2	5473452	484.015	310	-75
RL22-59	576732.2	5473424	484.281	310	-70
RL22-60	576511	5473471	479.094	310	-70
RL22-61	576567.7	5473423	474.801	316	-70
RL22-62	576433.4	5473312	475.193	310	-60
RL22-63	576330.7	5473382	479.757	326	-72
RL22-64	576610.1	5473971	481.064	310	-70

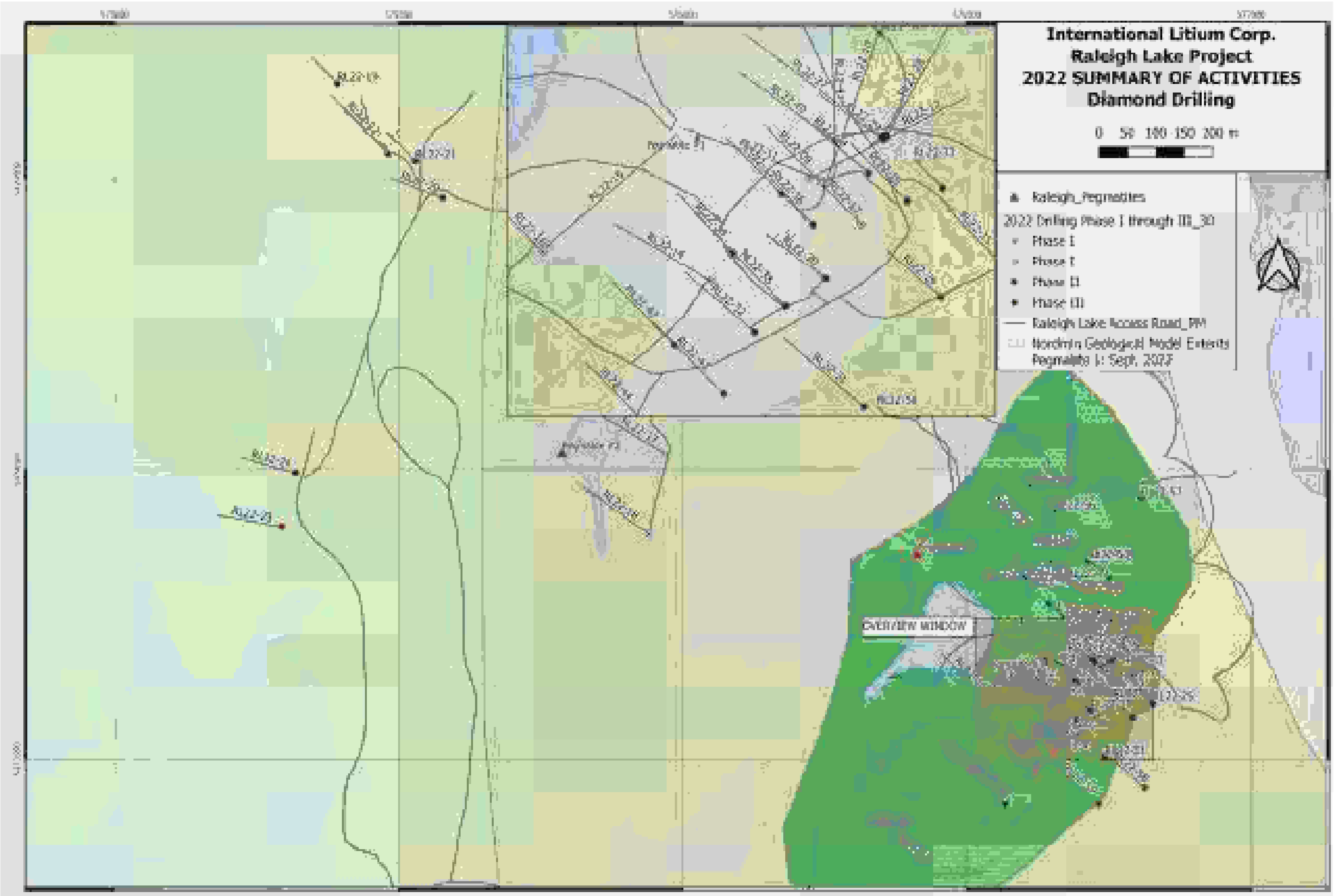


Figure 10-3: International Lithium 2022 Summary of Activities Map

Pegmatites 1 and 3 were intersected in all three phases of drilling in 2022 along with several other thin pegmatite veins. Areas of increased microcline abundance in Phase II and III drilling were intersected in Pegmatite 1 in the thickest portions and along margins and contained elevated rubidium grades. This area was delineated within the Mineral Resource Estimate as an independent rubidium zone, separate from the spodumene-dominant lithium Mineral Resource Estimate.

Approximately1,250 structural measurements were captured during the 2022 drill programs from oriented diamond drill core, with nearly 650 from pegmatites and pegmatite-related structures. The most dominant pegmatite trend is a northeast 030-035º dip. Another smaller population of data has a strong easterly trend of approximately 060º, which is also moderately dipping. This is thought to be contacts and internal fabric in the northeast corner of the geologic model. A stereonet depicting these contacts can be found in Figure 10-4.

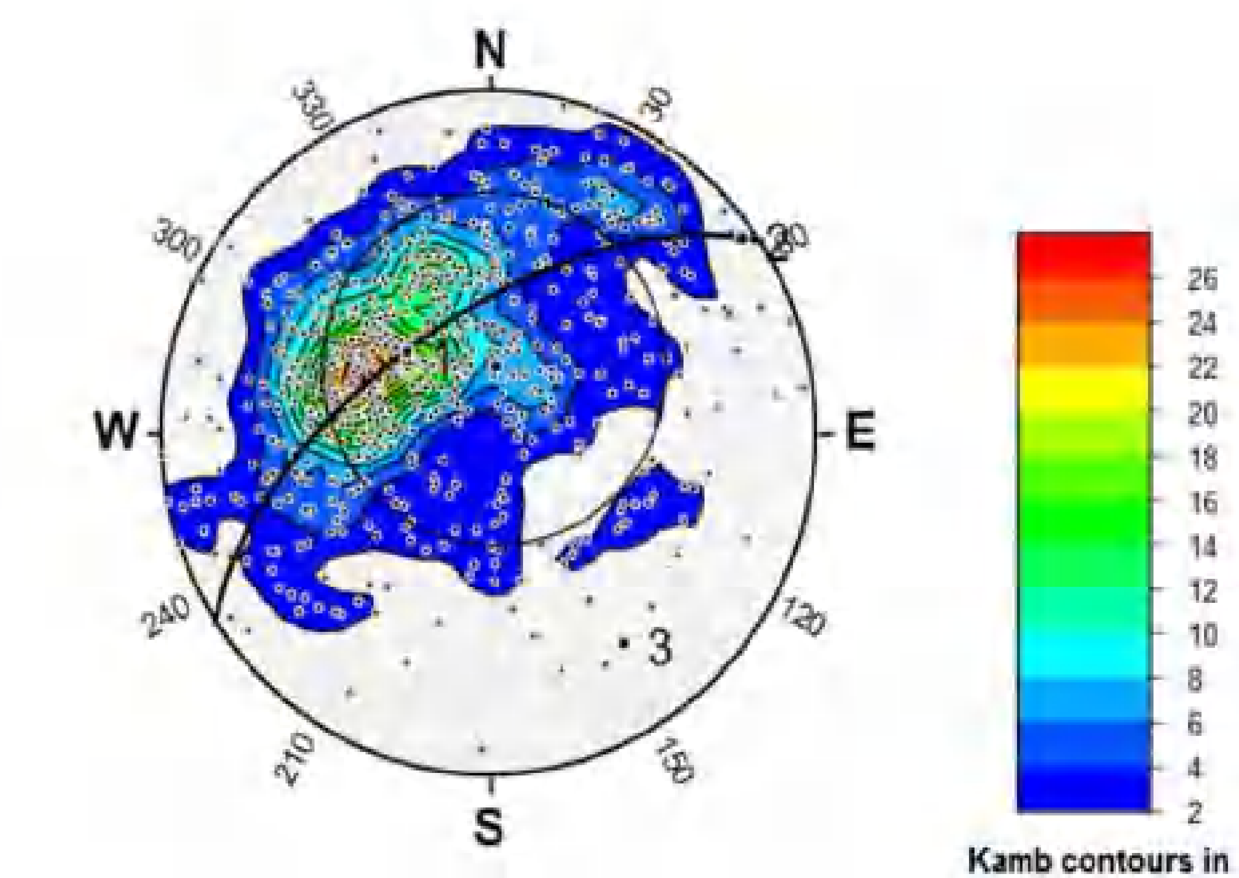


Figure 10-4: Stereonet Plot of All Pegmatite and Pegmatite-related Contacts

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

Quality control ("QC") measures were set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling, assaying, data management, and database integrity. Appropriate documentation of quality control measures and regular analysis of QC data are essential as a safeguard for project data and form the basis for the quality assurance ("QA") program implemented during exploration.

Analytical QC measures involve internal and external laboratory procedures implemented to monitor the precision and accuracy of the sample preparation and assay data. They are also important to identify potential sample sequencing errors and to monitor for contamination of samples.

Sampling and analytical Quality Assurance/Quality Control ("QA/QC") protocols typically involve taking duplicate samples and inserting quality control samples (CRM and blanks) to monitor the reliability of the assay results throughout the drill program. Umpire check assays are typically performed to evaluate the primary lab for bias and involve re-assaying a set proportion of sample rejects and pulps at a secondary umpire laboratory.

11.1 Assay Sample Preparation and Analysis

11.1.1 Historic Programs

11.1.1.1 1998-2000 Avalon

Avalon conducted a 5-hole diamond drill program on September 30, 1999, for a total of 602 m. A total of 45 pegmatite samples were assayed for tantalum (Ta), rubidium (Rb), niobium (Nb), and tin (Sn) by X-ray fluorescence (XRF). Lithium (Li) and cesium (Cs) by atomic absorption (AA). Samples were milled to minus 200 mesh. Avalon completed additional QA/QC analyses to test for a Ta nugget effect on five samples by analysing quarter-split pulps via XRF. The effect was considered minimal. Duplicate analyses were conducted but no standards or blanks were used in standard QA/QC protocol.

11.1.1.2 2001 Avalon

A four-hole drilling program was conducted between July and August 2001. RL01-06, 07, 09 were sampled every 10 metres to define a downhole geochemical profile. All samples were shipped to XRAL laboratories in Toronto, Ontario for analysis. Duplicate, blank and standards were analyzed for QA/QC at a rate of approximately every 20 samples.

11.1.1.3 2010 Abaddon Resources Inc.

Abaddon completed a 7-hole diamond drilling program in 2010. No standards for Li, Cs, Ta or Rb could be obtained; duplicates and four batches of improvised Li-bearing material was used for standards. Standards, blanks, and duplicates were used with standards inserted at a rate of one every thirty samples and one duplicate every 14 samples.

11.1.2 International Lithium Corp. 2021-2022

Drill core sampled by the Company was numbered using consecutive series of sample numbers, with a sample label attached to the core box labelled with the drill hole number and total sampled interval. One blank, one standard, and one duplicate were inserted every thirty samples. Core was sawed lengthwise using a wet saw along a cut line marked by a geologist. One half of the sample was placed in a plastic sample bag, labelled, and sealed with a zip-tie, and the other half returned to the core box for reference. Sample bags were then batched into rice bags of 5 to 15 samples (depending on sample size) for delivery directly from site to the lab in Dryden.

All samples were prepared by Actlabs Laboratories in Dryden, Ontario. The global QA/QC program at Actlabs includes internal and external inter-laboratory test programs and regularly scheduled internal audits that meet all requirements of ISO/IEC 17025 standards. The samples were dried, crushed to 70% passing < 2 mm, split using riffle splitter, and a 1 kg subsample was pulverized to 85% passing 75 microns.

Lithium, and other elements, were measured via Peroxide Fusion ICP/MS (inductively coupled plasma mass spectrometry, coded Ultratrace-7 by Actlabs). Analytical results were analyzed upon receipt and failures were sent for re-assay when they occurred.

11.1.2.1 Standards

The Company submitted 40 CRMs between 2021 and 2022 as part of their QA/QC process. The CRM results are summarized in Table 11-1 and Figure 11-1 through Figure 11-4.

Table 11-1: Summary of Certified Reference Material (CRM) Outcomes (Source: Nordmin, 2023)

Standard	Count	Certified Reference Value Au (g/t)	Mean Assay Value Au (g/t)	Acceptable Standard Deviation	% Bias
OREAS 750	21	2300	2246.90	200	2.36
MF-1	19	8175	8078.95	548	-1.17

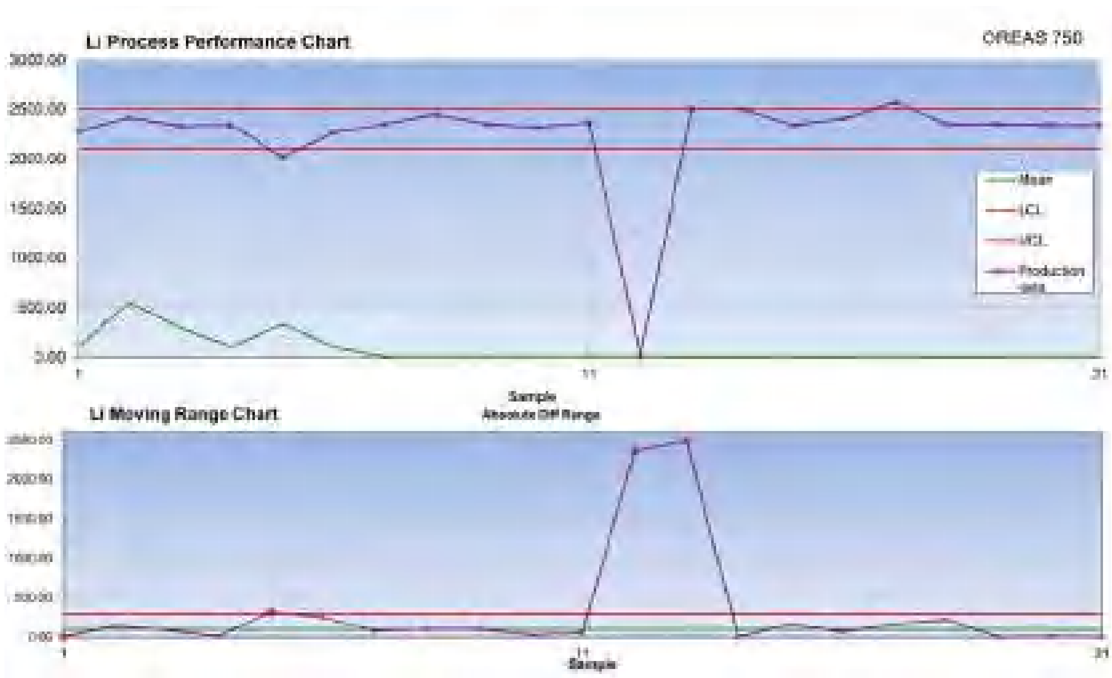


Figure 11-1: Deposit Standard OREAS 750 (Li ppm) (Source: Nordmin, 2023)

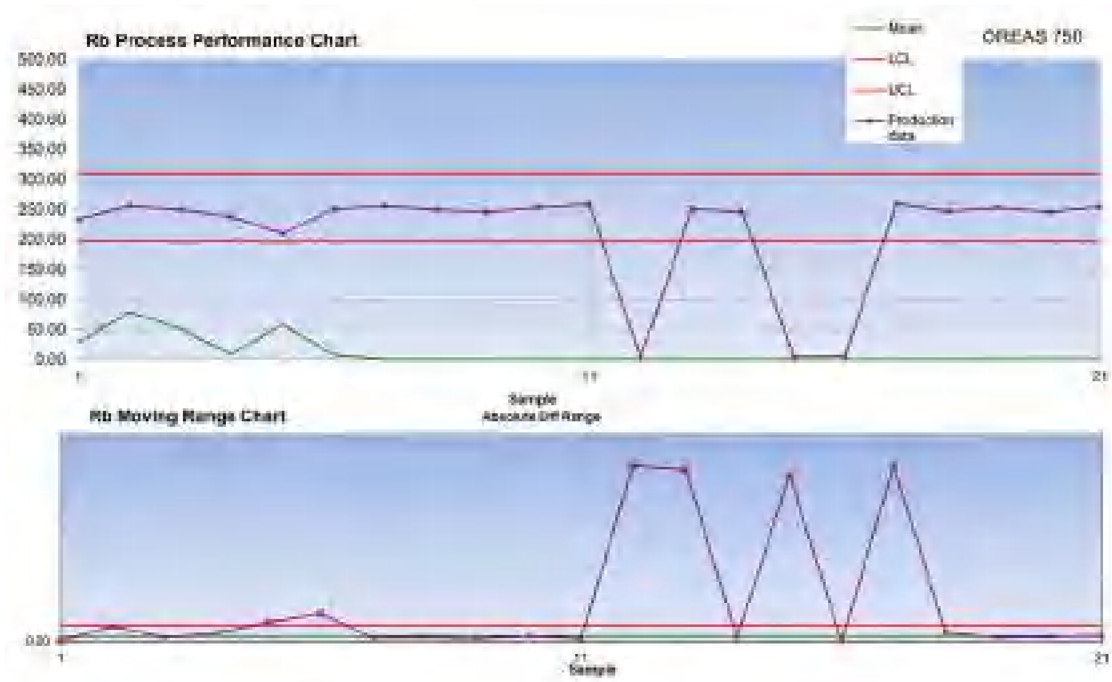


Figure 11-2: Deposit Standard OREAS 750 (Li ppm) (Source: Nordmin, 2023)

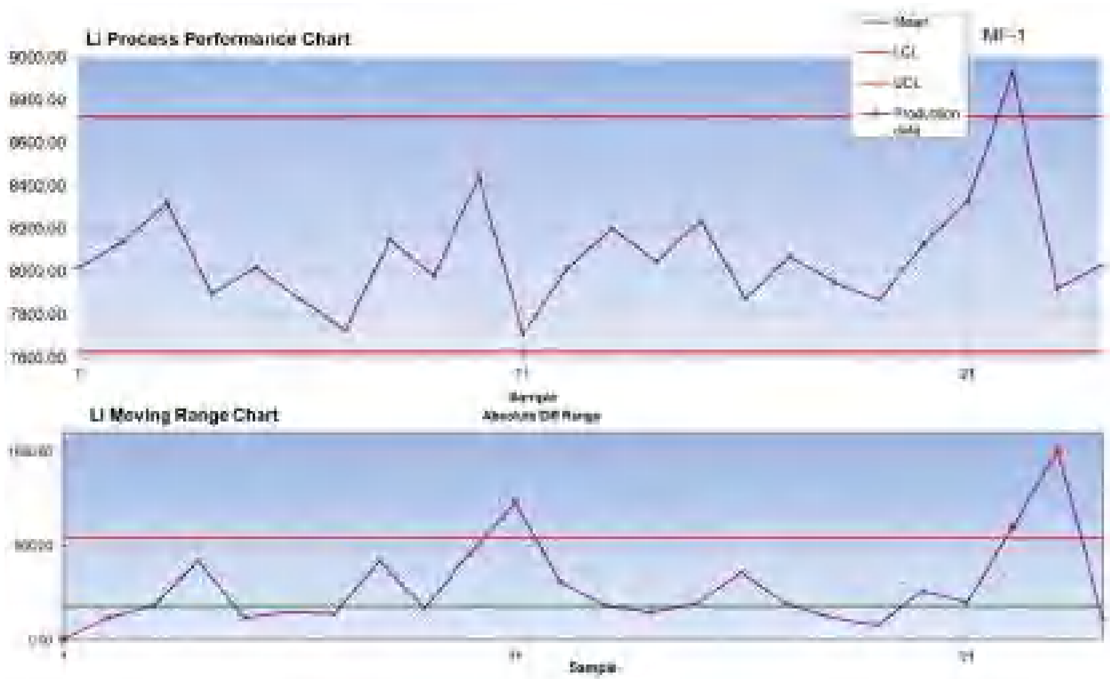


Figure 11-3: Deposit Standard MF-1 (Li ppm) (Source: Nordmin, 2023)

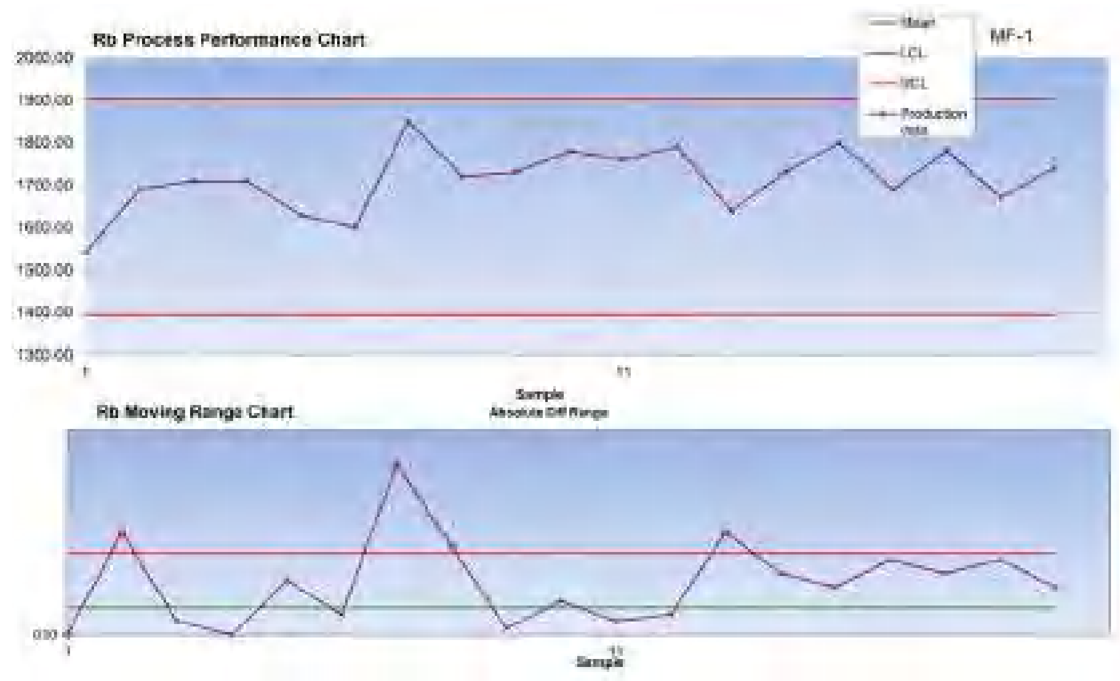


Figure 11-4: Deposit Standard MF-1 (Li ppm) (Source: Nordmin, 2023)

11.1.2.2 International Lithium Corp. Blanks

The Company submitted 31 coarse blanks between 2021 and 2022 as part of its QA/QC process. No significant carryover is evident (Figure 11-5).

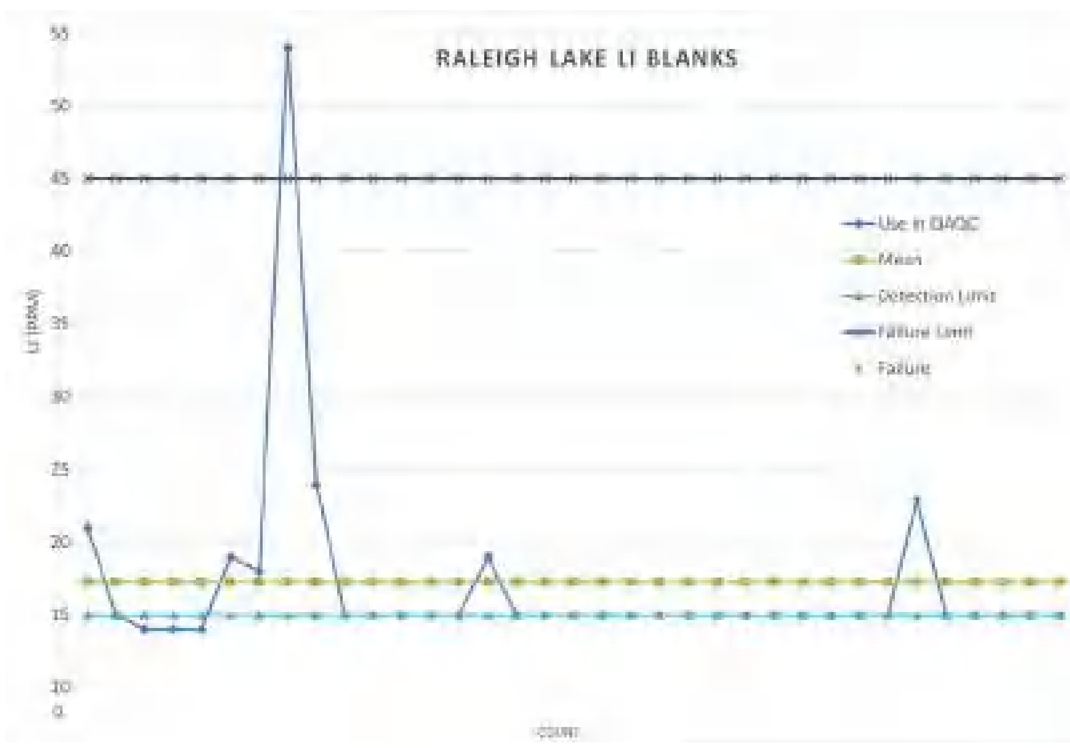


Figure 11-5: Deposit Li (ppm) Blanks (Source: Nordmin 2022)

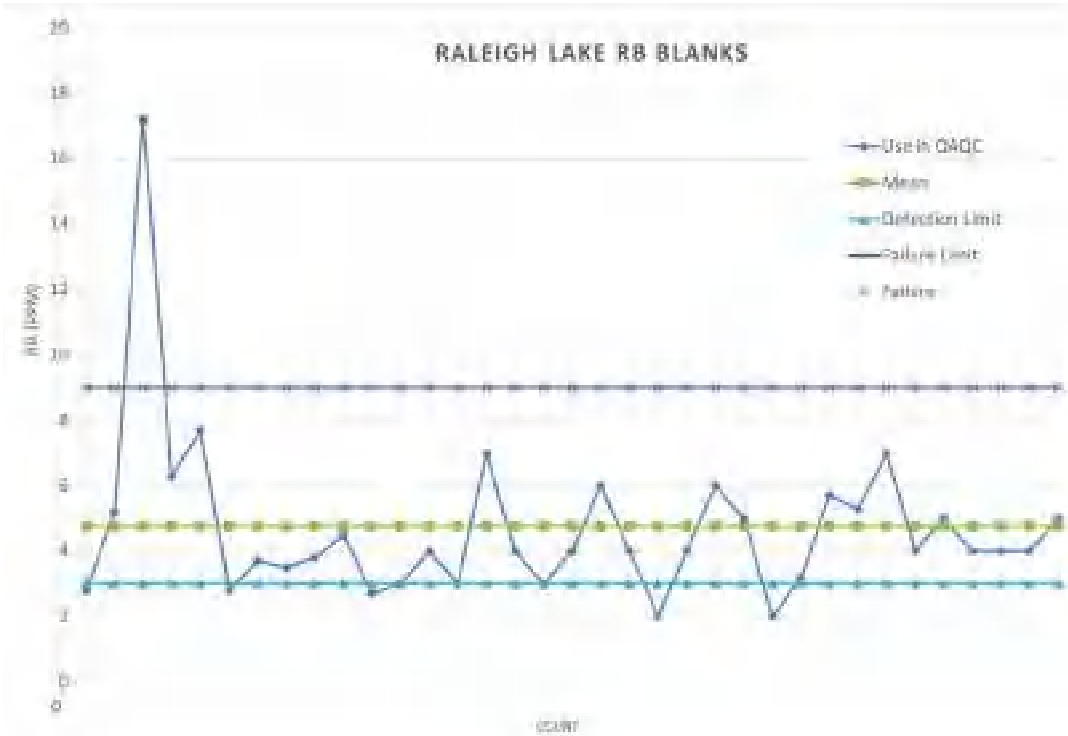


Figure 11-6: Deposit Rb (ppm) Blanks (Source: Nordmin 2022)

11.1.2.3 Field And Laboratory Duplicates

The Company submitted 27 duplicates as a part of the 2022 drilling program, as part of their QA/QC process. No duplicates were submitted during the 2021 drill program, rather, duplicates were submitted as lab duplicates rather than field duplicates involving quarter cutting core. The duplicate pair show good agreement for both lithium (Figure 11-7) and rubidium (Figure 11-8).

11.1.2.4 Umpire Checks

No umpire checks have been completed.

11.1.3 Specific Gravity Sampling

A total of 190 SG measurements were provided from the Company using the Archimedes method which measures the weight of the core in air versus the weight in water by applying the following formula:

$$Specific\ Gravity = \frac{Weight\ in\ Air}{(Weight\ in\ Air - Weight\ in\ Water)}$$

Average SG measurements for lithology type are provided in Section 14.7.

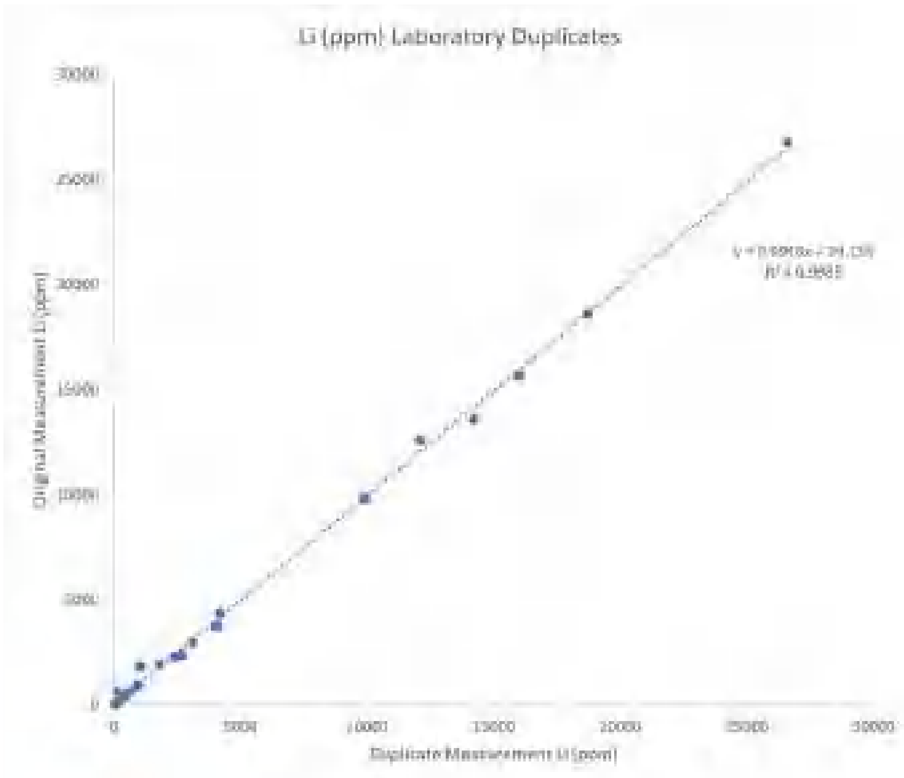


Figure 11-7: Laboratory Duplicates for Li (ppm) (Source: Nordmin, 2022)

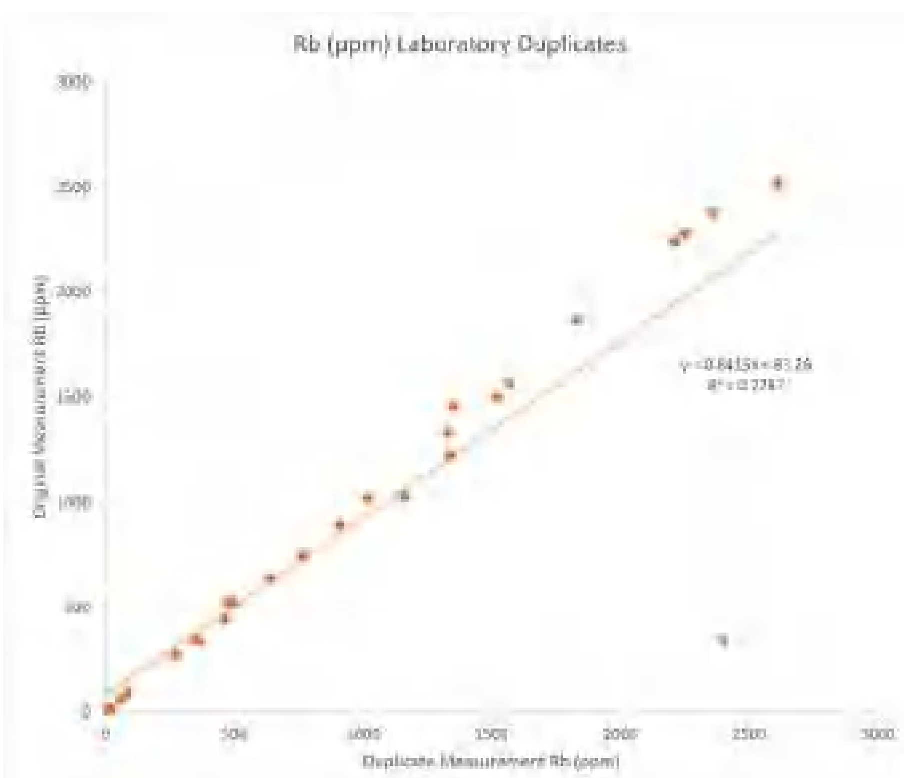


Figure 11-8: Laboratory Duplicates for Rb (ppm) (Source: Nordmin, 2022)

11.2 Sample Security

Sample bags are sealed with zip ties to ensure sample integrity and are securely shipped to Actlabs laboratories in Dryden for analysis. Both historic and current drill core are stored on-site (Figure 11-9 and Figure 11-10), while pulps are stored at a secure location in Ancaster, Ontario and coarse rejects are stored at Actlabs in Dryden, Ontario.

Figure 11-9: Coarse Storage Area; Core Stored in Cross-piles (Source: Nordmin, 2023)

Figure 11-10: Historic Core Storage Area, Cross-piled on Ground (1990, 2010) (Source: Nordmin, 2023)

11.3 QP's Opinion

Nordmin has been supplied with all raw QA/QC data and has reviewed and completed an independent check of the results for all Project sampling programs. It is Nordmin's opinion that the sample preparation, security, and analytical procedures used by all parties are consistent with standard industry practices and that the data is suitable for the 2022 MRE. Nordmin identified several further recommendations to ILC to ensure the continuation of a robust QA/QC program but has noted that there are no material concerns with the geological or analytical procedures used or the quality of the resulting data.

12. DATA VERIFICATION

Nordmin completed several data validation checks throughout the duration of the 2023 MRE.
The verification process included a site visit to the Project in Ignace by the Nordmin QP to review surface geology, drill core geology, geological procedures, chain of custody of drill core, sample pulps, and for the collection of independent samples for metal verification. Data verification included a survey spot check of drill collars, a spot check comparison of Li and Rb assays from the drill hole database against original assay records (lab certificates), spot check of drill core lithologies recorded in the database versus the core located in the core storage shed and a review of QA/QC performance of the drill programs. Nordmin has also completed additional data analysis and validation, as outlined in Section 11.

12.1 Nordmin Site Visit 2022

A site visit to the Project was carried out on October 18th, 2022, by Christian Ballard, P.Geo., QP. Mr. Ballard was accompanied by Patrick McLaughlin, P.Geo of Coast Mountain Geological Consulting, who has been involved with the Project for several years, and Annika Van Kessel, P.Geo, of Nordmin Engineering. Activities during the site visit included:

- Review of the geological and geographical setting of the Project.
- Review and inspection of the site geology, mineralization, and structural controls with respect to Lithium and Rubidium distribution.
- Review of the drilling, logging, sampling, analytical, and QA/QC procedures.
- Review of the chain of custody of samples from the field to the assay lab.
- Review of the drill logs, drill core, storage facilities, and independent assay verification on selected core samples.
- Confirmation of a variety of drill hole collar locations.
- Validation of a portion of the drill hole database.
- Review of mineralization.
- Company geologists completed the core logging and sampling associated with the drill programs. Nordmin used the Company's database to review the core logging procedures, the collection of samples, and the chain of custody associated with the drilling and sampling programs.

12.1.1 Observations

Site visit observations are as follows:

- Core logging is completed using Geospark™ software, where geologists enter lithology, mineralization, structure, alteration, RQD, and recovery data.
- Core logging occurs within the core logging building which also serves as an office (Figure 12-1 and Figure 12-2).
- Core cutting and bagging occurs within a separate facility in Dryden, Ontario.

Figure 12-1: Core Shack Logging Table with Metal Bar Used for Orienting Core (Source: Nordmin, 2023)

Figure 12-2: Core Shack Logging Table Displaying Core Library (Source: Nordmin 2023)

- QA/QC are inserted at a rate one every 10 samples alternating between one blank, one standard, and one pulp duplicate. Assay analyses are carried out by Actlabs Laboratories in Dryden, Ontario, Canada. QA/QC analysis is handled by the on-site geologist who has, thus far, run QA/QC checks on an annual basis based on the completion of seasonal drilling programs.
- Lithium mineralization is associated with mainly spodumene but occasionally with other lithium-bearing amphiboles within the LCT-pegmatite intrusive bodies (Figure 12-3, Figure 12-5, Figure 12-6). There are also zones of dominantly microcline mineralization which contain higher concentrations of rubidium (Figure 12-4).

Figure 12-3: Pegmatite within RL22-39 at 117-124 m Displaying Megacrystic Spodumene (Source: Nordmin, 2023)

Figure 12-4: Interval of Rb Rich Microcline Mineralization within Pegmatite from RL22-12 from 115.61 m to 125.90 m (Source: Nordmin, 2023)

Figure 12-5: 16 cm Spodumene Lath in Outcrop from Pegmatite 1 (Source: Nordmin, 2023)

Figure 12-6: Spodumene Lath Showing Crystal Habit, 5-10 cm in Length, within Outcrop of Pegmatite 1 (Source: Nordmin, 2023)

12.1.2 Core Logging, Sampling, and Storage Facilities

Drill holes were logged, photographed, and sampled on-site at the core logging facility. Most of the core is stored on-site, and the samples, pulps, and coarse rejects are archived in secure storage facilities (Figure 12-7 through Figure 12-12).

Figure 12-7: Core Shack Located off Highway 11/17 in Ignace, Ontario (Source: Nordmin, 2023)

Figure 12-8: Core Storage Facilities Yard (Source: Nordmin, 2023)

Figure 12-9: Historic 1999, 2001, and 2010 Drill Core Storage in the Field (Source: Nordmin, 2023)

Figure 12-10: Core Photography Station at Core Shack in Ignace, Ontario (Source: Nordmin, 2023)

Figure 12-11: SG Weighing Station (Source: Nordmin, 2023)

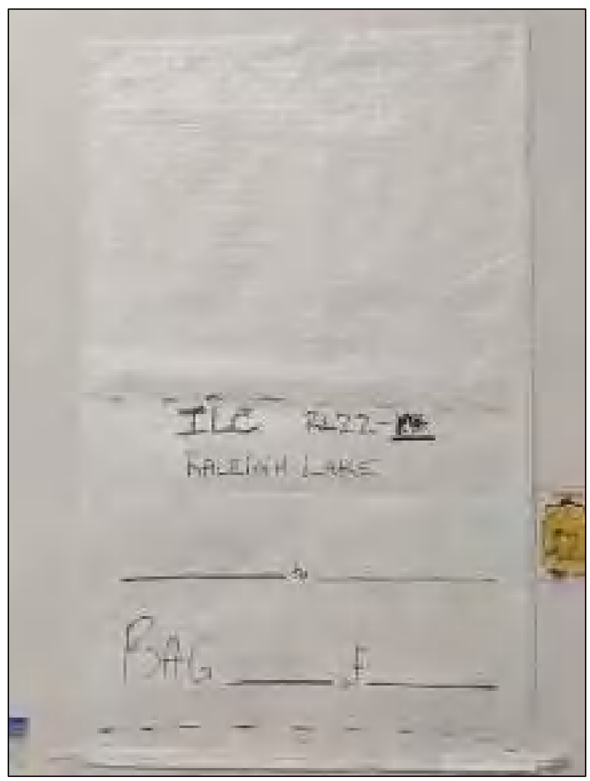


Figure 12-12: Samples Bag Labelling (Source: Nordmin, 2023)

12.1.3 Field Collar Validation

Nordmin was able to locate several drill collars from multiple drilling campaigns within the field during the field visit. A total of 11 drill collar sites were visited during the site visit (Figure 12-13 and Table 12-1). Collar coordinates were measured using a hand-held GPS unit and then compared to the official collar information within the drill hole database. No significant discrepancies were identified. Figure 12-14 is a typical example of what was observed of collars in the field during the site visit.

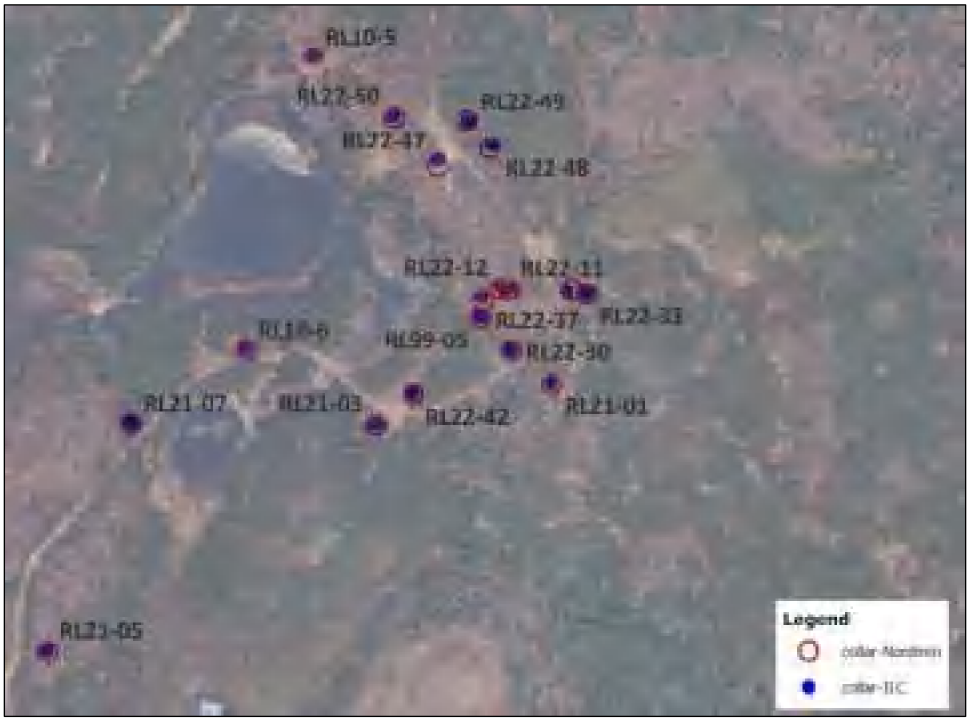


Figure 12-13: Collar Validation, Nordmin GPS Measurements (Red) vs. ILC Collar Measurements (Blue) (Source: Nordmin 2023)

Table 12-1: Drill Collar Coordinate Validation (Source: Nordmin, 2023)

Drill Hole	Nordmin-Surveyed Collars			Database Collars (DGPS Measured)			Difference		
	Easting	Northing	Elevation	Easting	Northing	Elevation	Easting	Northing	Elevation
RL21-03	576,586	5,473,511	479.1	576,585	5,473,513	474.4	-0.3	1.8	-4.7
RL21-05	576,263	5,473,288	488.8	576,262	5,473,291	484.6	-1.0	2.3	-4.2
RL21-07	576,345	5,473,513	479.8	576,345	5,473,513	477.0	-0.2	0.7	-2.8
RL22-09	576,715	5,473,644	483.4	576,715	5,473,645	477.8	-0.3	0.7	-5.6
RL22-11	576,718	5,473,644	481.0	576,715	5,473,647	477.8	-3.0	2.5	-3.3
RL22-12	576,708	5,473,645	482.7	576,708	5,473,647	477.6	0.5	2.3	-5.1
RL22-30	576,719	5,473,584	481.2	576,719	5,473,585	477.4	-0.2	0.5	-3.8
RL22-33	576,793	5,473,641	485.7	576,793	5,473,641	480.8	-0.1	0.8	-5.0

Drill Hole	Nordmin-Surveyed Collars			Database Collars (DGPS Measured)			Difference		
	Easting	Northing	Elevation	Easting	Northing	Elevation	Easting	Northing	Elevation
RL22-37	576,691	5,473,634	476.9	576,690	5,473,637	474.1	-0.9	3.6	-2.7
RL22-42	576,622	5,473,541	477.4	576,621	5,473,543	474.5	-0.7	1.7	-2.9
RL22-47	576,647	5,473,765	481.4	576,647	5,473,772	470.0	0.3	6.7	-11.4
RL22-48	576,698	5,473,783	483.3	576,700	5,473,787	481.0	1.9	4.4	-2.3
RL22-49	576,675	5,473,811	480.7	576,677	5,473,810	478.0	1.6	-0.6	-2.7
RL22-50	576,604	5,473,813	481.2	576,602	5,473,816	480.0	-1.8	3.2	-1.2
RL22-51	576,604	5,473,813	481.2	576,602	5,473,815	480.0	-1.8	2.2	-1.2

Figure 12-14: Typical Field Collar Site (Source: Nordmin, 2023)

12.1.4 Independent Sampling

The Nordmin QP selected a total of 53 samples from 8 diamond drill holes. Coarse rejects stored at the Dryden laboratory facility were selected for re-sampling. Only samples from the 2022 drill program were selected as they were the most readily available. Historic 2010 core was poorly stored (Figure 12-9), and 2021 drill core was cross-piled in the storage yard (Figure 12-8). Nordmin elected to choose a variety of grade ranges from various drill holes (Table 12-2).

Table 12-2: Independent Sampling (Source: Nordmin, 2023)

Hole ID	From	To	Width	Sample No	Lippm	Lippm_Check
RL22-09	93.05	94.15	1.1	151514	16700	16400
RL22-09	94.15	95.1	0.95	151515	17500	18300
RL22-09	97.2	99.3	2.1	151518	22400	23600
RL22-10	66.44	66.94	0.5	151537	399	411
RL22-10	66.94	67.3	0.36	151538	28	31
RL22-10	67.3	67.8	0.5	151539	235	218
RL22-10	96.65	96.95	0.3	151541	121	123
RL22-12	116.5	118.4	1.9	151606	7.5	7.5
RL22-12	118.4	120.4	2	151607	15	7.5
RL22-12	120.4	121.4	1	151608	7.5	7.5
RL22-12	121.4	122.4	1	151609	7.5	7.5
RL22-18	141.5	142	0.5	151759	1290	1010
RL22-18	142	142.85	0.85	151761	148	114
RL22-18	142.85	143.35	0.5	151762	597	611
RL22-18	143.35	144.35	1	151763	651	619
RL22-31	88.65	90.35	1.7	151897	149	155
RL22-31	90.35	92.2	1.85	151898	420	401
RL22-31	92.2	93.68	1.48	151899	58	59
RL22-31	93.68	95.3	1.62	151901	447	524
RL22-31	95.3	96.7	1.4	151902	162	163
RL22-31	96.7	98	1.3	151903	145	140
RL22-31	98	99.94	1.94	151904	225	227
RL22-35	99.64	100.5	0.86	276537	21100	20900
RL22-35	100.5	101.4	0.9	276538	17400	17000
RL22-35	101.4	102.72	1.32	276539	18600	24600
RL22-35	103.23	104.28	1.05	276542	16900	16300
RL22-35	105.33	106.33	1	276544	16600	16200
RL22-37	25	27	2	276588	643	688
RL22-37	27	27.7	0.7	276589	3610	3720
RL22-37	27.7	28.47	0.77	276591	1160	1900

Hole ID	From	To	Width	Sample No	Lippm	Lippm_Check
RL22-37	28.47	29.45	0.98	276592	44	48
RL22-37	87	89	2	276595	368	363
RL22-37	89	91	2	276596	419	429
RL22-37	91	92.25	1.25	276597	839	890
RL22-37	92.75	94.33	1.58	276599	994	1020
RL22-37	94.33	95.95	1.62	276601	1700	1770
RL22-37	95.95	97.7	1.75	276602	4370	4830
RL22-37	97.7	99.36	1.66	276603	3100	3220
RL22-37	99.36	100.9	1.54	276604	630	875
RL22-37	100.9	102.25	1.35	276605	19100	19300
RL22-37	102.25	103.85	1.6	276606	9800	10900
RL22-37	103.85	104.65	0.8	276607	27000	25700
RL22-37	104.65	106	1.35	276608	26800	26100
RL22-37	106	106.5	0.5	276609	9950	10700
RL22-44	201	203	2	276803	513	440
RL22-44	203	205	2	276804	788	779
RL22-44	205	205.5	0.5	276805	646	477
RL22-44	205.5	207.5	2	276806	6000	5760
RL22-44	207.5	209.5	2	276807	646	592
RL22-44	209.5	211.5	2	276808	964	910
RL22-44	214.55	215.55	1	276809	928	891
RL22-44	215.55	216.55	1	276811	105	116
RL22-44	216.55	218	1.45	276812	985	809

Samples selected by Nordmin for verification analysis were taken from coarse rejects located in storage at Actlabs in Dryden, Ontario. Samples were analyzed using the Company's analytical procedures.
The Nordmin assay results were compared to the Company database and summarized in a lithium scatter plot (Figure 12-15). Despite some sample variance, assays compared within reasonable tolerances for the deposit type and no material bias was evident.

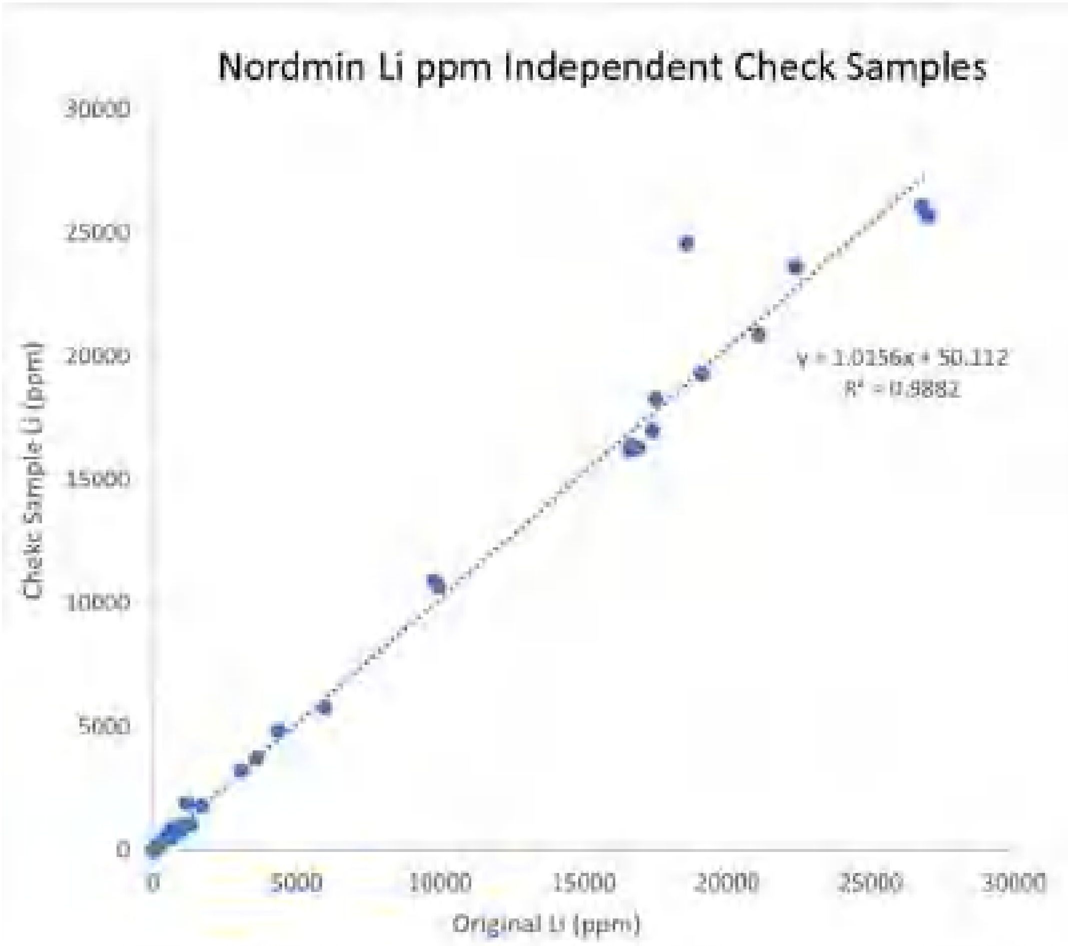


Figure 12-15: Scatter Plot Comparison of Independent Sampling Results (Li g/t) (Source: Nordmin, 2023)

12.2 Database Validation

Core sample records, lithology logs, laboratory reports, and associated drill hole information for all drill programs were digitally compiled for use in Leapfrog Geo® and Datamine Studio RM® software for deposit modelling and resource estimation.

Drill hole data from 2021 to present has been compiled into Geospark® and directly exported into comma-separated values tables. Nordmin reviewed the database using Excel®, Datamine Studio RM®, and QGISTM software. No errors or discrepancies were found within the drill hole database.

The QP completed a spot check verification on the Project for:

- Two drill holes including all main lithologies and 8% of the assays.

The geology was validated by comparing lithologic units from Geospark® with stored half core and are deemed acceptable for use by QP.

12.3 Review of ILC QA/QC

ILC has a QA/QC process in place as defined in Section 11 and 12. Company geologists actively monitor the assay results throughout the drill programs and summarize the QA/QC results in weekly and monthly reports. Most of the CRMs performed as expected within tolerances of two standard deviations of the mean grade. Nordmin is satisfied that the QA/QC process is performing as designed to ensure the quality of the assay data.

12.4 QP's Opinion

Upon completion of the data verification process, it is the Nordmin QP's opinion that the geological data collection and QA/QC procedures used by ILC are consistent with standard industry practices and that the geological database is of suitable quality to support the Mineral Resource. The QP recommended the implementation of umpire checks on a drill program completion basis. It was also recommended that historic core cross-piled in the field be lifted off the ground and covered to better preserve the boxes for future reference. As well, it was recommended that additional security such as a locked fence surrounding the current core storage should be implemented for increased security.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The geological work determined two domains, one for lithium and one for rubidium, which were of interest for mineral processing and metallurgical extraction testing. Representative samples of the lithium domain and rubidium domain (no dilution) were submitted for scoping level mineralogical study and metallurgical flowsheet development.

The mineralogy revealed albite and quartz in both the Li and Rb mineral domains. Most of the lithium was hosted in spodumene in the Li domain. In the Rb domain the rubidium was mostly associated with microcline (K-feldspar).

The summary findings of the phase one scoping level investigations performed are reported in this section. The detailed initial testing investigations were reported in three final SGS Canada reports:

1. An Investigation into the Mineralogical Characterization of Rubidium and Lithium Samples from the Raleigh Lake Ores, Ontario, prepared for International Lithium Corp., Project 19977-01 – MI5018-SEP23 – Mineralogy Report October 30, 2023.
2. An Investigation into Mineral Processing Testwork on Samples from the Raleigh Lake Ores Deposit – Phase 1 prepared for International Lithium, Project 19977-01 – Report 1– October 27, 2023.
3. Literature Review – Rubidium, Cesium and Potassium Extraction from LCT Pegmatite Deposit, prepared for International Lithium, Project 19977-01- Literature Review October 26, 2023.

Additional phase two metallurgical investigations are in progress to evaluate the effects of additional crushing and magnetic separation on spodumene concentrate grade and recovery, and how chlorination roasting affects rubidium leachability. Phase two results are pending.

13.2 Orebody and Metallurgical Domains

The Raleigh Lake Orebody contains two separate metallurgical domains, the lithium spodumene domain and the rubidium microcline domain, that require customized process flowsheets to be developed for each domain.

For the lithium domain, the objective is the recovery of spodumene to 6% Li_2O concentrate grade, while for the rubidium bearing microcline domain, the objective would be to develop a flowsheet for extraction of the rubidium from the microcline. To develop the flowsheet for each orebody domain, individual split core metallurgical samples were collected from each of the metallurgical domains by International Lithium, Patrick McLaughlin and transported to the SGS Canada laboratory, in Lakefield, Ontario, attention Tassos Grammatikopoulos. Below are two images, Figures 13-1 and 13-2, of the shipment to SGS Canada. The first image is the shipment bill of lading from Manitoulin Transport, with the tracking number circled in red.

The second image is the blue totes containing representative metallurgical samples, in each bin, respectively.

The packing list for each tote and representative metallurgical sample were also located in each blue bin respectively. Weights of each representative deposit sample were estimated as 40.3 Kg for Spodumene pegmatite lithium zone and approximately 16kg for the rubidium-bearing microcline zone.

A shipment consisting of one skid with two blue-lidded totes was received at the SGS Lakefield site on August 11, 2023, and assigned an internal receipt number of 0122-AUG23. Upon receipt, all samples were inventoried and weighed. After samples were inventoried at the SGS Canada lab, the individual samples were prepared by size reduction, compositing and riffle methods to provide representative subsamples for mineralogical, mineral processing and hydrometallurgical investigations.

Figure 13-1: Bill of Lading for Metallurgical Sample Shipment

Figure 13-2: Blue Totes with Metallurgical Samples for Shipment to SGS Lab

13.3 Mineralogical and Metallurgical Investigations

The objective of the investigations was to determine the amenability of known metallurgical processes to recover lithium as 6% Li_2O spodumene concentrate from the lithium domain (Li Head); and determine the leachability of rubidium from the microcline domain (Rb Head).

The initial mineralogy on the two metallurgical samples (Li Head and the Rb Head) were intended to determine the mineral deportment and be used to guide the metallurgical investigations.

13.3.1 Mineralogy of Rb Head and Li Head Samples (from SGS report)

Mineralogy was conducted on the two received samples to determine liberation, mineral assemblages which would help to support and guide the metallurgical testwork. The project number 19977-01 and LIMS number MI5018-SEP23 were assigned to track the mineralogy work.

Sample preparation and homogenization was conducted (as described in the metallurgical section 13.3.2), with subsample charges of the Rb Head and the Li Head prepared, crushed, and sized for mineralogy. A subsample of the Rb Head of about 1 kg was stage crushed to a P_{80} of circa 600-700 µm. The Rb-head was analyzed as a single size fraction. The Li Head subsample represents the -1 mm material from the initial head sample. For the mineralogy and liberation study, the Li-head was further stage crushed into 425 µm, screen and recombined into four size fractions including +425 µm, -425/+300 µm, -300/+150 µm and -150 µm.

The mineralogical work was conducted with TIMA-X (Tescan Integrated Mineral Analyzer), Electron Probe Micro-Analysis (EPMA), Laser Ablation by Inductively Coupled Plasma Mass Spectrometry (LA by ICP-MS), X-ray diffraction analysis (XRD), and chemical assays. The purpose of the mineralogy was to determine the overall mineral assemblage of the sample and define the liberation and association attributes of rubidium minerals in the Rb Head and lithium minerals in the Li Head.

X-Ray Diffraction (XRD) Analysis

XRD analysis indicates that the two samples consist of a different mineral assemblage. The Rb-head consists mainly of microcline (46.6%), albite (33.2%), minor quartz (13.2%), spodumene (2.5%), muscovite (2.0%) and diopside (2.6%). The Li-head consists of albite (58.1%), quartz (21.4%), minor spodumene (8.0%), minor microcline (8.4%), muscovite (3.0%) and diopside (1.1%) (Table 13-1).

Table 13-1: XRD Results for the Rb-head and Li-head Samples

Mineral/Compound	Rb-head	LI-head
Quartz	13.2	21.4
Albite	33.2	58.1
Spodumene	2.5	8.0
Muscovite	2.0	3.0
Microcline	46.6	8.4
Diopside	2.6	1.1
Total	**100**	**100**

The Mineralogy Summary of the Spodumene Domain - Li-head

a. Spodumene is the main lithium mineral in the sample. Spodumene is well liberated (94%) at this grind target (P_{80} of ca. 400 µm). Liberation of the spodumene is similar across the size fractions examined (+425 to -150 µm). The pure spodumene particles increase only below the 1 to 150 µm fraction.

b. Gangue silicates are also well liberated.

c. Therefore, spodumene may be recovered by gravity or flotation at relatively coarse-grained particle sizes. Theoretically, a high purity lithium concentrate is achievable.

d. Spodumene hosts greater than 98% of the total lithium in the sample.

e. A pure spodumene concentrate is expected to have approximately 3.5% Li.

The Mineralogy Summary of the Microcline Domain - Rb-head

a. K-feldspars host an average of 1.42% Rb_2O and 0.07% Cs_2O. A pure feldspar concentrate is expected to have a chemical composition as the average chemistry of the feldspar from the EPMA.

b. Muscovite averages 1.90% Rb_2O 0.19% Cs_2O.

c. K-feldspars hosts ca. 98% of the total Rb and 95% of the total Cs in the sample.

d. K-feldspars are well liberated (92%), while middling with plagioclase are minor (7%). Therefore, theoretically, a high purity K-feldspar concentrate is achievable (for the P_{80} of 700 µm).

13.3.2 Mineral Processing - Sample Preparation, Heavy Liquid Separation (HLS) of Lithium Head Ore Sample (from SGS report)

The main objective of the phase one scoping level mineral processing test investigation was to provide a preliminary indication of the lithium beneficiation of the Li head by heavy liquid separation (HLS). The metallurgical target was the preparation of spodumene concentrate grading >6.0% Li_2O while maximizing lithium recovery.

Two samples were submitted for PEA scoping level testwork sample preparation: Li Head representing the Li head LCT pegmatite ore and Rb Head representing the microcline-rubidium ore.

The pegmatite Li Head sample was initially stage-crushed to 100% passing 12.7 mm, homogenized, and split into 10 kg test charges. One of the 10 kg charges was sub-sampled 500 g for head assays and the remaining was screened at 16 mesh to remove the -1 mm fraction. The +1 mm fraction was further screened at ¼" to generate two fractions of -12.7 mm +6.3 mm and -6.3 mm +1 mm. Two subsamples from the fine fraction, -1 mm, were taken for mineralogical analysis, and assays on lithium and Whole Rock Analysis (WRA). The two coarse fractions, -12.7 mm +6.3 mm and -6.3 mm +1 mm, were submitted for Heavy Liquid Separation (HLS) testing.

The microcline Rb Head sample was prepared by stage-crushing it to 100% passing -10 mesh, homogenizing, and split into 1 kg test charges. One of the charges was sub-sampled 100 g for head assays and the remaining was submitted for mineralogical analysis. The remaining Rb Head charges were stored for future hydrometallurgical roasting and leach testing for rubidium extraction.

The key head assays of the lithium and rubidium head samples are presented in Table 13-2. The lithium Li head sample assayed 1.59% Li_2O and 0.56% Fe_2O_3, while the rubidium head sample graded 6,580 g/t Rb (equivalent to 0.72% Rb_2O) with 0.12% Li_2O and 0.24% Fe_2O_3. Low levels of 20 g/t Be to 80 g/t Be are seen in both head samples. In future testing, the Be will also be tracked in final spodumene concentrate analyses as this would be undesirable impurity if reported to spodumene concentrate.

Table 13-2: Summary of the Head Assay Results

Sample ID	Assays											
	Li (%)	Li_2O (%)	Rb (g/t)	SiO_2 (%)	Al_2O_3 (%)	Fe_2O_3 (%)	MgO (%)	CaO (%)	Na_2O (%)	K_2O (%)	S (%)	Be (g/t)
Lithium Head	0.74	1.59	-	73.2	16.8	0.56	0.05	0.22	4.73	1.94	<0.01	79.9
Rubidium Head	0.056	0.12	6,580	70.0	16.8	0.24	0.05	0.16	4.05	7.92	0.02	20.6

Heavy Liquid Separation (HLS) tests were conducted to assess the amenability of the samples to the Dense Media Separation (DMS) for the beneficiation of spodumene. Heavy liquid separation (HLS) tests were performed on the lithium sample stage-crushed to 100% passing -12.7 mm and screened to -12.7 mm/+6.3 mm and -6.3 mm/+1 mm fractions.

Each of the two coarse fractions submitted for HLS testwork, was immersed in a heavy liquid comprised of methylene iodide diluted with acetone to target liquid-specific gravities (SG) of 2.90, 2.85, 2.80, 2.75, 2.70, and 2.65. The first pass was conducted using a heavy liquid with the highest specific gravity (SG 2.90), with each subsequent pass on the float fraction using a heavy liquid with a lower specific gravity. The six (6) resulting HLS sink products, as well as the final (SG 2.65) HLS float product and -1 mm fine fraction, were submitted for lithium and whole rock analysis (WRA).

The HLS results of all samples provided a strong indication of the amenability to DMS for lithium beneficiation (Table 13-3) and Figures 13-3 and 13-4. The green colour in the HLS sink in the figures shows the green spodumene mineralization. The interpolated HLS results indicated that the ideal SG cut points for producing a 6.0% Li_2O concentrate were 2.83 and 2.85, for -12.7 mm/+6.3 mm and -6.3 mm/+1 mm fractions, respectively. The ideal SG cut-point increased as the fraction size increased.

Figure 13-3: HLS Sink Products of 2.80, 2.85, and 2.90 SG from -12.7 mm +6.3 mm Fraction

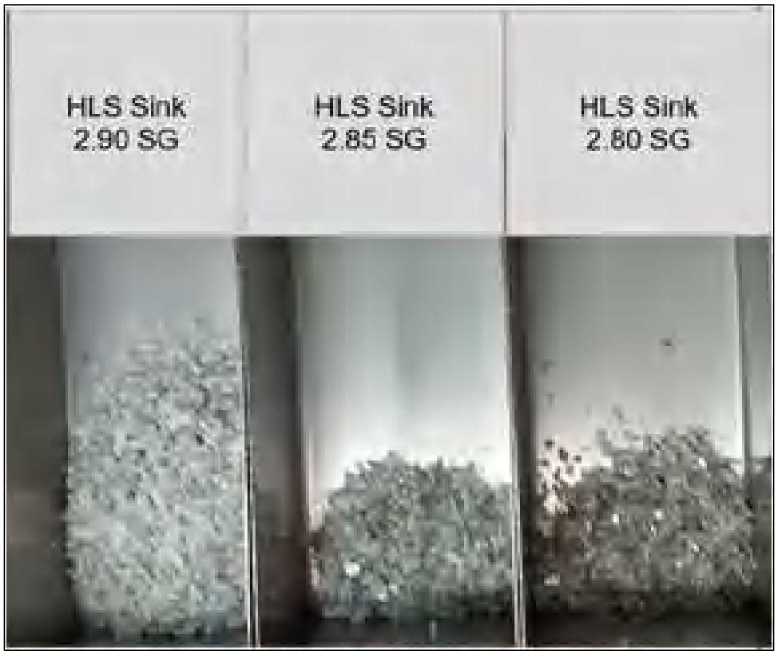


Figure 13-4: HLS Sink Products of 2.80, 2.85, and 2.90 SG from -6.3 mm +1 mm Fraction

At SG 2.85, 6.0% Li_2O concentrate grade of 14.9 weight % with global lithium recovery of 53.0% was obtained in the fraction of -12.7 mm/+6.3 mm.

At SG 2.83, a 6.0% Li_2O concentrate grade of 8.0 wt% with a global lithium recovery of 28.5 % was obtained in the -6.3 mm/+1 mm fraction

Combining the 6% Li_2O concentrates from the two fractions of -12.7 mm/+6.3 mm and 6.3 mm/+1 mm (highlighted in cyan in Table 13-3), generated a combined global lithium recovery of 53.0% plus 28.5% equals 81.5%. This means 81.5% lithium recovery was achieved with 22.9 wt% (14.9 wt% plus 8.0 wt%). In summary, HLS results of the combined -12.7 mm +1mm fraction, (Combining two fractions highlighted in color) show a global recovery of 81.5% lithium in 22.9 weight% of 6% Li_2O spodumene concentrate.

Table 13-3: Summary of HLS Global Mass Balance (Interpolated @ 6.0% Li_2O)

Product		HLSG	Weight	Assays (%)			Distribution (%)	
		g/cm^3	%	Li	Li_2O	Fe_2O_3	Li	Fe_2O_3
-12.7 + 6.3 mm	HLS Concentrate	2.85	14.9	2.79	**6.00**	**0.91**	**53.0**	26.4
	HLS Middling	-2.85 +2.70	8.21	0.78	1.69	0.66	8.21	10.6
	HLS Tailings	2.70	32.1	0.042	0.089	0.28	1.70	17.6
-6.3 +1 mm	HLS Concentrate	2.83	8.01	2.79	**6.00**	**1.06**	**28.5**	16.5
	HLS Middling	-2.83 +2.70	3.07	0.59	1.27	0.99	2.32	5.90
	HLS Tailings	2.70	22.1	0.022	0.047	0.31	0.61	13.3
Fines Fraction (-1 mm)			11.6	0.38	0.82	0.43	5.65	9.72
Head (calc.)			**100**	**0.78**	**1.69**	**0.52**	**100**	**100**
Head (dir.)				**0.74**	**1.59**	**0.56**		
Flotation Feed			**22.9**	**0.55**	**1.19**	**0.59**	**16.2**	**26.2**

Above a tailings SG-cut point of 2.70, the HLS middling from each sample contained between 1.27 – 1.69% Li_2O with 2.3 – 8.2% of the global lithium distribution. Therefore, the HLS middling can potentially be stage crushed then mixed with the minus 1 mm fines fraction to produce another process feed grading 1.19 % Li_2O and 0.59% Fe_2O_{-3}. The combined HLS middling and fines fraction contained 16.2% of the lithium distribution graded 1.19% Li_2O, which could become a potential flotation feed for a rolls crush and screening, and ultrafines DMS gravity circuit to improve lithium recovery.

The grade of iron in the spodumene concentrate was ~1% Fe_2O_3, which could likely be reduced by treating the concentrate by magnetic separation.

The HLS results in the Figure 13-5 indicated that the spodumene concentrate grades and lithium recoveries were higher at the -6.3 mm +1 mm fraction, most likely due to improved spodumene liberation in the finer fraction.

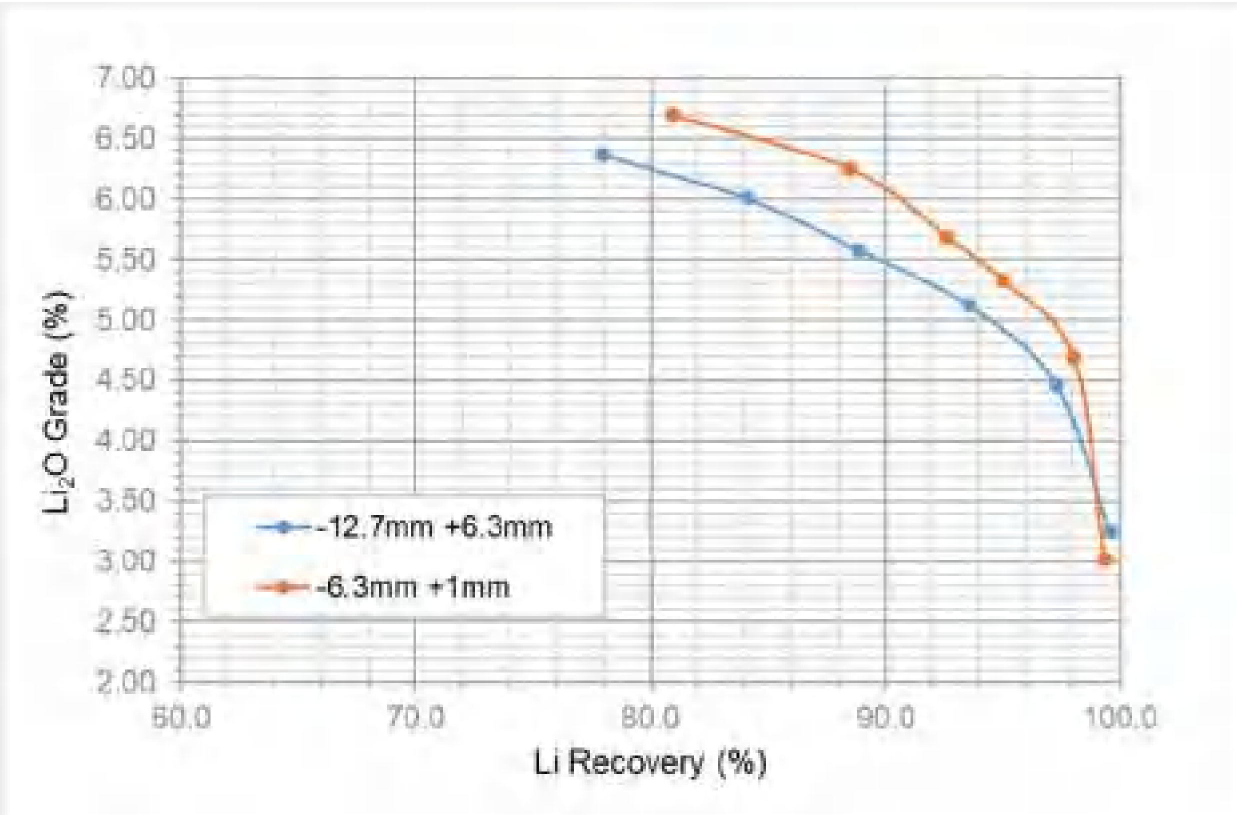


Figure 13-5: Stage Cumulative Lithium Grade / Recovery Plot

In future, it is also recommended to optimize the crushing size by performing additional HLS testing at different crushing sizes (between -15 mm to -6.5 mm). The optimum crush size and HLS conditions would be selected for use as a DMS plant feed parameter to perform a future DMS pilot campaign.

Additional phase two testing is in progress now at the SGS Canada lab to determine the effect of a minus 9.5 mm crush size on HLS grade and recovery.

Once the beneficiation process flowsheet is finalized it is recommended to perform confirmatory HLS and pilot scale DMS beneficiation testwork on variability samples to confirm the amenability of various samples to the developed flowsheet, and the effect of dilution. The effect of a lower lithium head grade is recommended to be evaluated by HLS, to determine the effect on the spodumene concentrate grade and recovery.

The lithium head samples have a variety of colors of minerals and densities. Therefore, the lithium head is recommended be evaluated for upgrading potential by ore sorting methods of colour and density (XRT).

13.4 Comminution

Comminution testing was not performed on the Li head or Rb head samples. For design purposes for the Li crushing parameters, ERM assumed a typical hard, abrasive lithium ore as described in Canadian Hardrock deposit Feasibility Study of July 26, 2022, report by WSP for the Rose Li-Ta Project, PQ.

A recommendation is to perform comminution testing of: Crusher Work Index by Impact Testing (CWI), Abrasion Index testing (Ai), and Bond Ball Mill Grindability (BWI) to determine crushability, abrasiveness, and grindability power parameters of the Raleigh Lake ores.

13.5 Hydrometallurgical Literature Review for Rb Head Ore – Rb, Cs, K Extraction

The following Section is taken from the SGS Report (2023).

SGS Canada Inc. (SGS) conducted a literature review of possible extraction methods of rubidium, potassium, and cesium from the Rb head. This literature review only includes extraction of rubidium, potassium, and cesium from the ore, and did not include downstream separation and purification processes such as precipitation, solvent extraction, ion exchange and so on.

Possible extraction methods for rubidium, potassium and cesium from a similar deposit as Raleigh Lake Project is discussed in this section. Routes for possible potassium and cesium extraction were reviewed because potassium and cesium are known to carry levels of radioactivity. This literature review will guide in choice of the metallurgical process conditions for extraction of Rb, and radioactive elements of Cs and K from the rubidium head ore.

According to Pekoy and Kononkoya (2010), among the minerals in LCT pegmatites, the highest rubidium concentrations occur in micas and alkaline feldspar. Before any data on natural rubidium-dominant feldspars were published in 1997, among feldspars, the highest rubidium concentrations were found in a microcline from Red Cross Lake in Canada (Pekov & Kononkova, 2010).

At present, rubidium is primarily extracted from brines, and rarely from these rubidium-containing ores (Liu et al., 2023). Liu et al. (2023) summarizes some of the commonly used methods in extracting rubidium from different minerals. These are: acid decomposition method, alkali leaching method, sulfate roasting method, carbonate roasting method, and chlorination roasting method. One of the goals is to break the main constituent silicate minerals and transform them, usually at high temperatures, to a more reactive and easily dissociated phase. Direct sulfuric acid digestion and sulfuric acid roasting method, which are usually employed in industrial setting, are primarily used for extraction of rubidium and cesium from spodumene, lepidolite, pollucite and cesium garnet (Liu et al., 2023; Xing et al., 2021). These methods take advantage of the fact that these certain minerals are easily attacked by acid. Unfortunately, these methods do not work very well with K-feldspars since K-feldspar structure is acid resistant and is hard to destroy with these methods (Liu et al., 2023; Xing, Wang, Ma, et al., 2018).

Cesium is not always associated with rubidium. It can be found in a variety of other pegmatite minerals. Rubidium and cesium can also occur within the aluminosilicate framework. Hence, to recover cesium, it is required to break those frameworks too. Interestingly, one of the most popular cesium extraction methods are decomposition methods. In this method, different additives such sulfate-base, carbonate-based or chloride-based are used. Among the most effective methods, the highest potassium recovery involved chlorination roasting which efficiently breaks the aluminosilicate framework, releases potassium for easier extraction. In chlorination roasting, the ore is added with chloride salts such as $CaCl_2$, NaCl, NH_4Cl or in the presence of HCl or chlorine gas (Zhang et al., 2020). Mixture of additives are also something that was explored such as $CaCO_3$-$CaCl_2$ mix with roasting at 800 - 900 °C or a Na_2CO_3-NaCl roasting at 600 to 800 °C followed by a water leach (Lu et al., 2022). Most of these processes track individual metal recovery such as lithium only, cesium only, rubidium only, or potassium only. There are limited studies that consider the co-extraction of rubidium, cesium, and potassium. The work of Zhang et al. (2020) stood out due to the simultaneous co-extraction of the critical metals.

In Table 13-4 below, a comparison between single additive ($CaCl_2$ or NaCl) and mixture ($CaCl_2$-NaCl) is presented. When $CaCl_2$ was added with a ratio to ore of 0.3:1, 70% of lithium, 91% of rubidium, 97% of cesium, and 71% of potassium were recovered. When it was added with NaCl, different concentrations did not yield big variations in metal recoveries. The recoveries were 30-39% lithium, 66-71% rubidium, 59-71% cesium, and 40-49% potassium. This indicates that NaCl is less effective in metal extractions than $CaCl_2$. When the mixture of the two additives were considered, the metal recoveries increased significantly to 92% lithium, 98% rubidium, 98% cesium, and 93% potassium.

Table 13-4: Comparison between Single Additive ($CaCl_2$ or NaCl) and Mixture ($CaCl_2$-NaCl)

Process	Sample	Ore Content	Roasting Conditions	Leaching	K Recovery	Source
Chlorination roasting ($CaCl_2$ or NaCl)	Lepidolite, albite, quartz	6.25% K_2O, 1.08% Rb_2O, 0.22% Cs_2O, 3.34% Li_2O	0.3:1 (agent:ore), 850 C, 60 min	Water leaching, 60°C, 60 min, 4:1 liquid-solid ratio	70.33% Li, 91.43% Rb, 97.20% Cs, 71.45% K	Zhang et al 2020
Chlorination roasting ($CaCl_2$ or NaCl)	Lepidolite, albite, quartz	6.25% K_2O, 1.08% Rb_2O, 0.22% Cs_2O, 3.34% Li_2O	0.3:0.2:1 ($CaCl_2$:NaCl:ore)	Water leaching, 25°C, 60 min, 3:1 liquid-solid ratio	92.49% Li, 98.04% Rb, 98.33% Cs, 92.90% K	Zhang et al 2020

In the Raleigh Lake deposit, based on mineralogy, rubidium is chemically bound to microcline (K-feldspar), while cesium is found in muscovite. Based on the literature review and the limited mineralogical study of the Raleigh Lake deposit, the following methods are recommended to extract these target metals.

Article I. Rubidium recovery in K-feldspar

Section 1.01 Alkaline leaching method in an autoclave with the following conditions: 200-250 g/L sodium hydroxide, 10-15:1 liquid-solid ratio, leaching at 150-230 °C for 60 minutes (Xing, Wang, Ma, et al., 2018; Xing, Wang, Wang, et al., 2018).

Article II. Rubidium and potassium recovery in K-feldspar

Section 2.01 Thermal activation with CaO in a muffle furnace at 1300 °C for 60 minutes then leaching with 120 g/L sulfuric acid at 50 °C for 90 minutes (Liu et al., 2023).

Article III. Potassium recovery from K-feldspar

Section 3.01 Chlorination roasting using $CaCl_2$ or $CaCl_2 \cdot 2H_2O$ at 850-950 °C for 30-60 minutes, then water leaching at room temperature to 70 °C for 30-120 minutes (Samantray et al., 2020; Serdengeçti et al., 2019; Tanvar & Dhawan, 2020; Türk et al., 2021; Yuan et al., 2015).

Article IV. Co-extraction of rubidium, potassium and cesium

Section 4.01 Chlorination roasting using mixed $CaCl_2$-NaCl at >750 °C for 45 min followed by water leaching at ambient temperature for 60 minutes.

Phase two hydrometallurgical lab testing has been initiated at SGS Canada for the Co-extraction of rubidium, potassium and cesium and using the chlorination roasting using mixed $CaCl_2$-NaCl at >750 °C for 45 min followed by water leaching at ambient temperature for 60 minutes. The process initiated and chosen for phase two was based on expected high metal recoveries. The optimum conditions may only be obtained based on exhaustive variability testing. In this regard, it must be highlighted that employing the same process and same conditions as in the literature may not yield similar results. Typically, conditions are optimized to each specific ore. The Raleigh Lake ore may not have the exact composition, the same associated gangue minerals, and impurities. Phase two hydrometallurgical results are not available yet.

13.6 Flotation

Although flotation was considered, flotation testing of fines was not evaluated to defer capital costs of a flotation plant and avoid the addition of flotation reagents, which are chemicals which may delay environmental permit approvals.

Flotation feed investigations are recommended for future trade-off studies.

13.7 Dewatering

Dewatering and solid liquid separation of the process fines have not yet been tested in the laboratory.

For dewatering, assume up to 10g/t flocculant dosage will be required to settle and filter the process fines. A 10 g/t flocculant dosage was applied to settle fines in the June 2023, 43-101 feasibility study on the Sigma XUXA lithium ore.

It is recommended that dewatering (solid liquid separation) studies be included in future investigations.

13.8 Future Investigations

It is recommended to perform HLS testing between crushing sizes to -15 mm, and -6.5 mm to determine the optimum crush size.

The lithium head samples have a variety of colors of minerals and densities. Therefore, the lithium head is recommended be evaluated for upgrading potential by ore sorting methods of colour and density (XRT).

Once the beneficiation process flowsheet and crush size are optimized it is recommended to perform confirmatory HLS and pilot scale DMS beneficiation testwork on variability samples. The variability samples would confirm the amenability of various samples, the effects of dilution, and lithium recoveries on lower lithium head grades.

It is recommended to perform Crusher Work Index (CWi) Abrasion Index testing (Ai) and Bond Ball Mill Grindability Work Index (BWi) to determine the comminution parameters.

Solid liquid separation testing of the lithium fines fractions would be recommended be included with additions of flocculant of up to 10g/t (or more).

Modified acid base accounting testing is recommended to determine if waste rock and HLS floats (DMS floats) are suitable for use in highway construction or mine road construction.

Although flotation feed was not evaluated for the purposes of the current Preliminary Economic Assessment testing and design work, basic flotation test program is recommended to conduct a trade-off study – "No flotation" (to defer capital costs of a flotation plant and avoid the addition of flotation reagents, which are chemicals which may delay environmental permit approvals) vs "Flotation" (to potentially enhance the overall process performance parameters and achieve overall superior recoveries and stable final product grades.

It is recommended to continue the monitoring and removal of deleterious of Be and Fe on the spodumene concentrate. To be marketable as spodumene concentrates the Fe specification is less than 1% and Be is less than 20 g/t (ppm). The grade of iron and beryllium in the spodumene concentrate which could likely be reduced by treating the concentrate by magnetic separation, which is recommended for further investigation.

The weight percent removal of muscovite (mica) minerals from the minus 6.3mm plus 1.0mm should be evaluated and confirmed with REFLUX™ classifier testing.

Since the combined HLS middling rejects and - 1mm fines fraction contained 16.2% of the lithium distribution and graded 1.19% Li_2O, this is recommended to be tested as potential flotation feed and as ultrafines DMS gravity circuit feed to increase lithium recovery.

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource Estimate for the Project is based on a drill hole database comprised of 13,821 m of diamond drilling from 81 holes completed between 1999 and the end of 2022, following a three-phase drilling program by ILC consisting of 9,496 m of diamond drilling completed during 2022. Drilling completed during this period assisted in confirming and refining the mineralogical, lithological, and structural controls of the Project. This program is described in detail in Section 10.

The deposit style of the Project is that of rare-element LCT-Pegmatites. The Project is hosted within the Wabigoon Subprovince of Archean Superior Province at the junction of the western Wabigoon Subprovince, the Winnipeg River greenstone terrane, and the Marmion greenstone terrane. Terrane boundaries are recognized as important geologic settings for Li and rare-metal pegmatites. The property is underlain by Archean supracrustal rocks comprised largely of mafic metavolcanics and metasediments. These units overlie and are intruded by granitic plutons and batholiths of various ages and compositions. Pegmatite occurrences on the property strike north-northeast with a moderate south-easterly dip and are hosted within the metavolcanics and metasediments of the western Wabigoon terrane. Pegmatite dykes and high-grade rubidium microcline were modelled along with a low-grade background wireframe which encapsulated all wireframes. Assays were manually flagged to wireframes, composites were calculated, and estimation and classification was performed followed by open pit shell optimization and underground MSO constraints.

14.2 Drill Hole Database

The Mineral Resource Estimate was estimated from the main drill hole database comprised of 13,821 m of diamond drilling from 81 drill holes completed between 1999 and the end of 2022. Figure 14-1 shows drilling throughout the Project in relation to the overburden surface and the pit shell. Table 14-1 presents a summary of drill hole sampling throughout the Project. ICP data exists for 1,698 samples from 81 diamond drill holes.

Table 14-1: Summary of Diamond Drilling (Source: Nordmin, 2023)

Year	Diamond Drilling	
	Count	Length (m)
1999	5	602
2001	4	752
2010	7	1,464
2021-2022	64	11,003

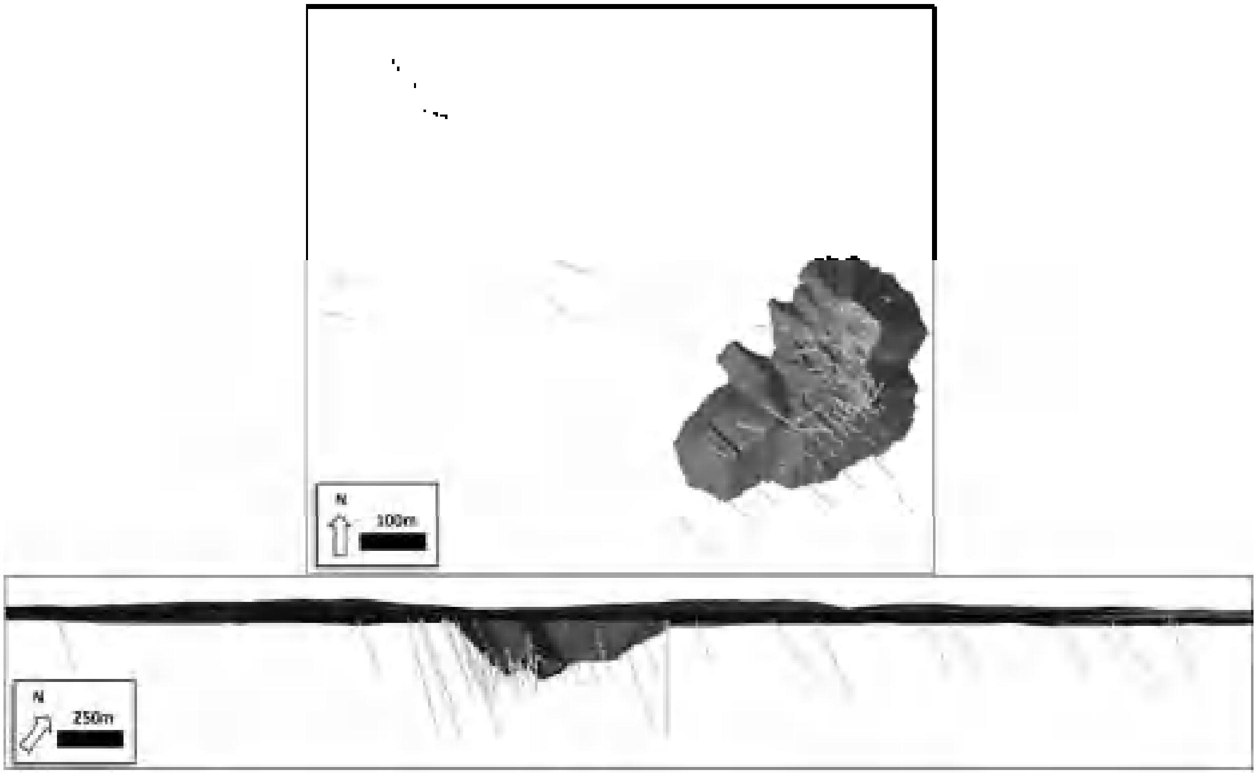


Figure 14-1: Deposit Drilling Plan View (top) and Looking Northwest (bottom) with the Pit Shell (Source: Nordmin, 2023)

14.3 Geological Domaining

The Project lies within the western Wabigoon Terrane, which is located within the Archean Superior Province. Specifically, the Project sits within the western portion of the central Wabigoon Region and is characterized by bifurcated and anastomosed supracrustal greenstone belts separated by large ovoid gneissic domes and elliptical batholiths.

Mineralization at the Project is hosted within several stacked and continuous LCT pegmatites; there are two significant, continuous, and well-defined pegmatites historically identified as Pegmatites 1 and 3. Five additional modelled mineralized pegmatites also exist between these two dominant pegmatites.

Implicit mineralization wireframes (“Zones”) were created for each of seven pegmatite intrusions and an elevated-Rb area (“Lithium Pegmatite and Rubidium Domains”, “Domains”). Individual Lithium Pegmatite Zones were modelled based on a cut-off grade of 500 ppm Li (Figure 14-2 and Figure 14-3).

One Rubidium Domain wireframe was modelled based on a cut-off grade of 5,000 ppm and was defined based on microcline abundance. Within this rare-element Rubidium Domain, the microcline becomes the dominant feldspar species, and spodumene is in very low modal abundance (see Figure 14-4 and Figure 14-5).

Pegmatites were built to follow the north-northeast trend with moderate dips ranging from 25°-40° in a south-south-easterly direction. The Rubidium Domain wireframe follows the same trend that is present in the thickest part of the Lithium Pegmatite Zone 1.

A Low Grade Background Domain wireframe was created to encapsulate all other Domains and Zones.

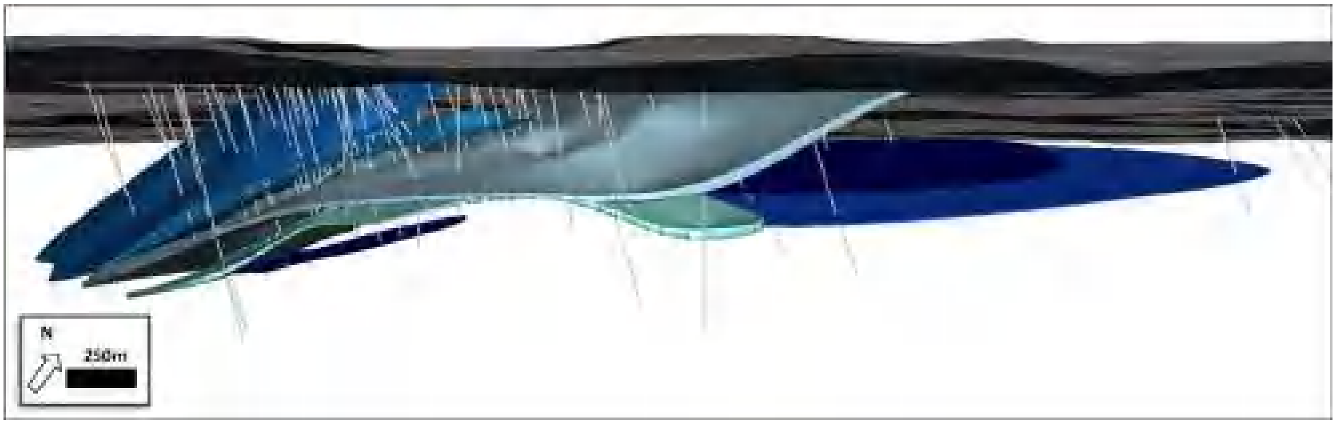


Figure 14-2: Raleigh Lake Pegmatite Wireframes Looking Northwest (Source: Nordmin, 2023)

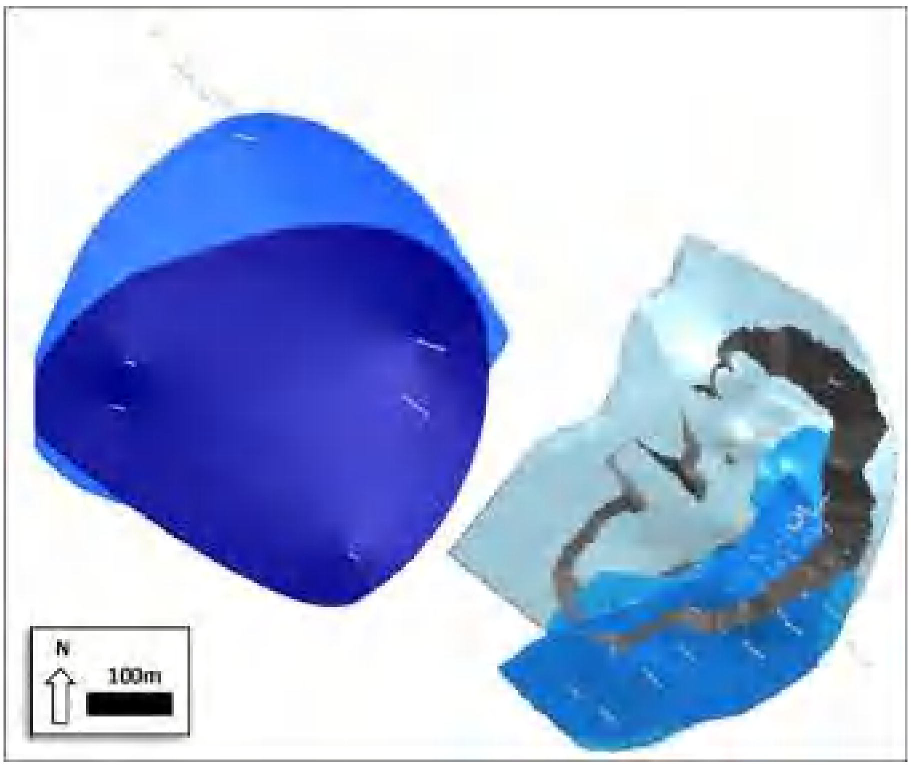


Figure 14-3: Raleigh Lake Pegmatite Wireframes, Plan View, with Pit Shell Displayed (Source: Nordmin, 2023)

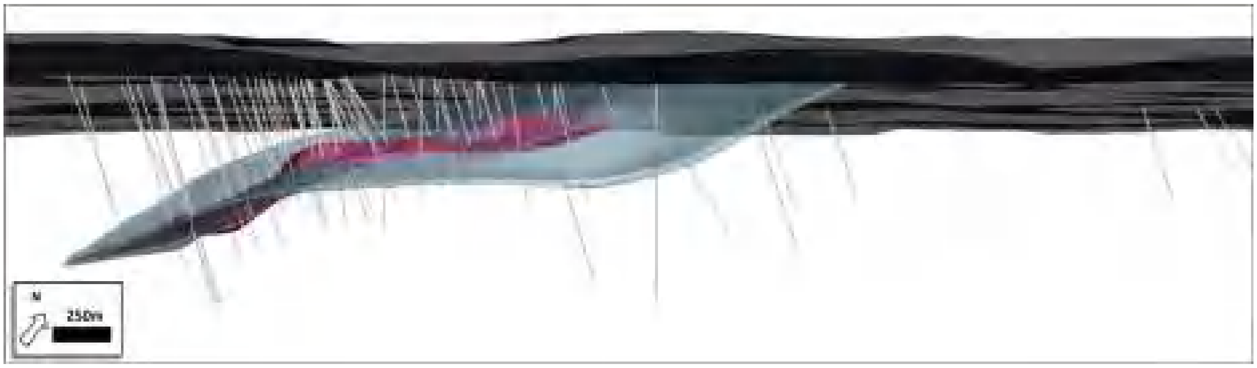


Figure 14-4: Rb Wireframe (pink) within Pegmatite 1 (blue), Looking Northwest (Source: Nordmin, 2023)

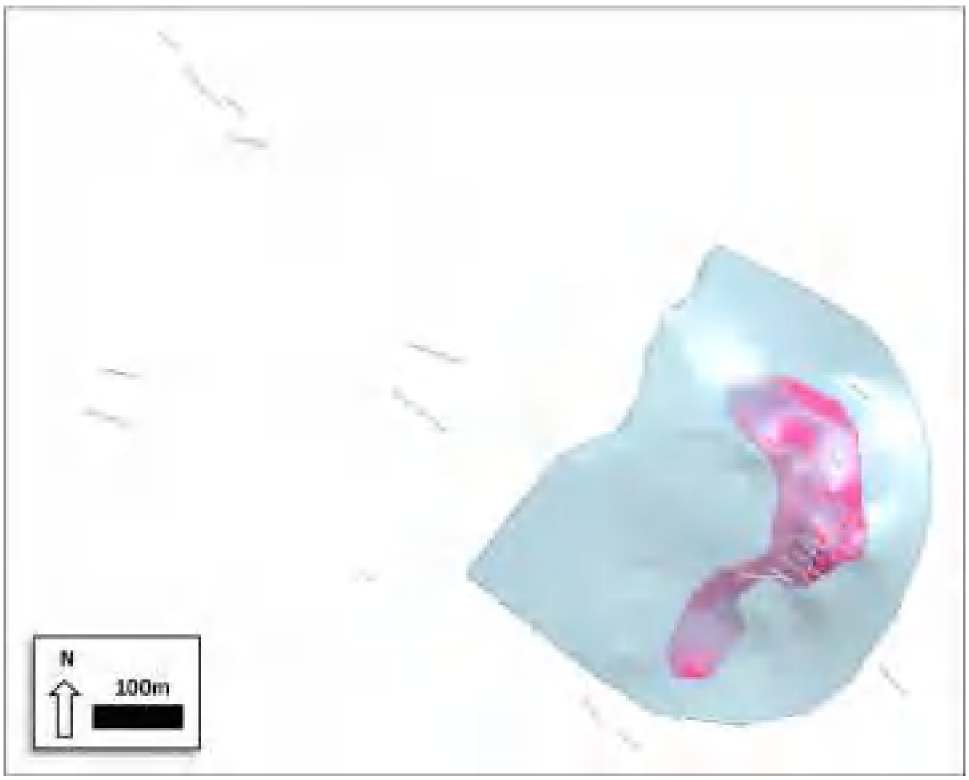


Figure 14-5: Rb Wireframe (pink) within Pegmatite 1 (blue), Plan View (Source: Nordmin, 2023)

Wireframe overlapping occurs between the Rubidium and Lithium Pegmatite Domains and with the Low Grade Background Domain. No overlapping occurs within any given Domain.

All efforts were made to terminate wireframes at the halfway point between drillholes where appropriate or 50 m past the last mineralized drillhole. Section and surface maps from previous exploration drilling and lithogeochemical sampling work were referenced in the pegmatite interpretation; they were georeferenced and imported into Leapfrog Geo™ 2021.2. See Figure 14-2 and Figure 14-3.
All wireframes were clipped to the overburden surface.

It is the QP's opinion that the implicit modelling approach was the most appropriate method for wireframe interpretation due to the continuous nature of the Project lithologies.

14.4 Exploratory Data Analysis

The exploratory data analysis was conducted on raw drill hole data to determine the nature of Lithium and Rubidium distribution, correlation of grades within individual rock units, and the identification of high-grade outlier samples. Nordmin used a geostatistical software package (X10 Geo™) to complete various descriptive statistics, histograms, probability plots, and XY scatter plots to analyze the grade population data. The findings of the exploratory data analysis were used to help define modelling procedures and parameters used in the MRE. Table 14-2 gives sample statistics for Lithium and Rubidium within their respective domains. Figure 14-6 through Figure 14-8 present histograms and other statistical plots used to examine Li distribution within each Domain. Descriptive statistics were used to analyze the grade distribution of each sample population, determine the presence of outliers, and identify correlations between grade and rock types for each mineralized wireframe.

Table 14-2: Drillhole and Sample Statistics for Each Zone (Source: Nordmin, 2023)

Lithium Domain		
Zone	**Sample Count**	**Drillhole Count**
101	843	65
102	20	6
103	195	34
104	94	25
105	138	31
106	24	5
107	22	5
Rubidium Domain		
Zone	**Sample Count**	**Drillhole Count**
201	**165**	**40**

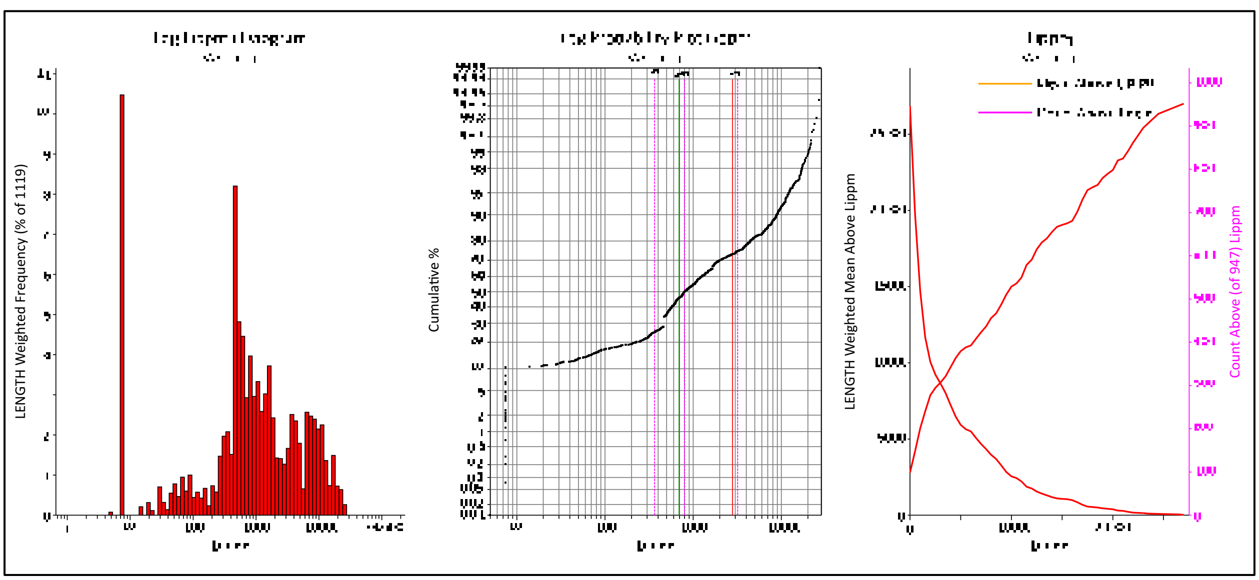


Figure 14-6: Li Distribution for Samples within the Lithium Pegmatite Domain, Zone 1 (Source: Nordmin, 2023)

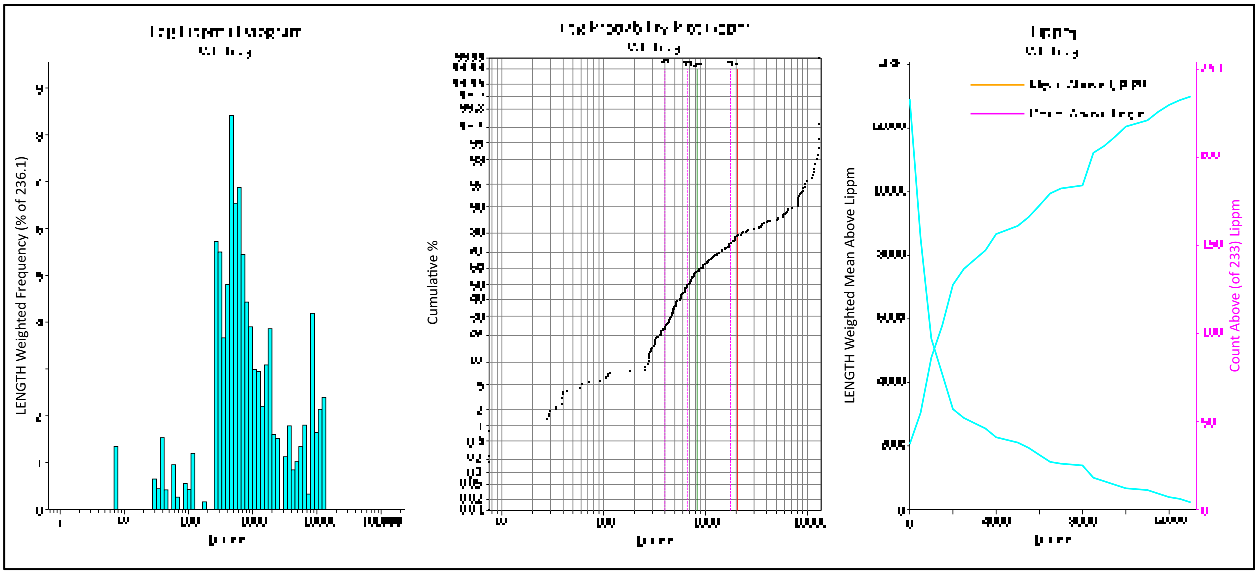


Figure 14-7: Li Distribution for Samples within the Lithium Pegmatite Domain, Zone 3 (Source: Nordmin, 2023)

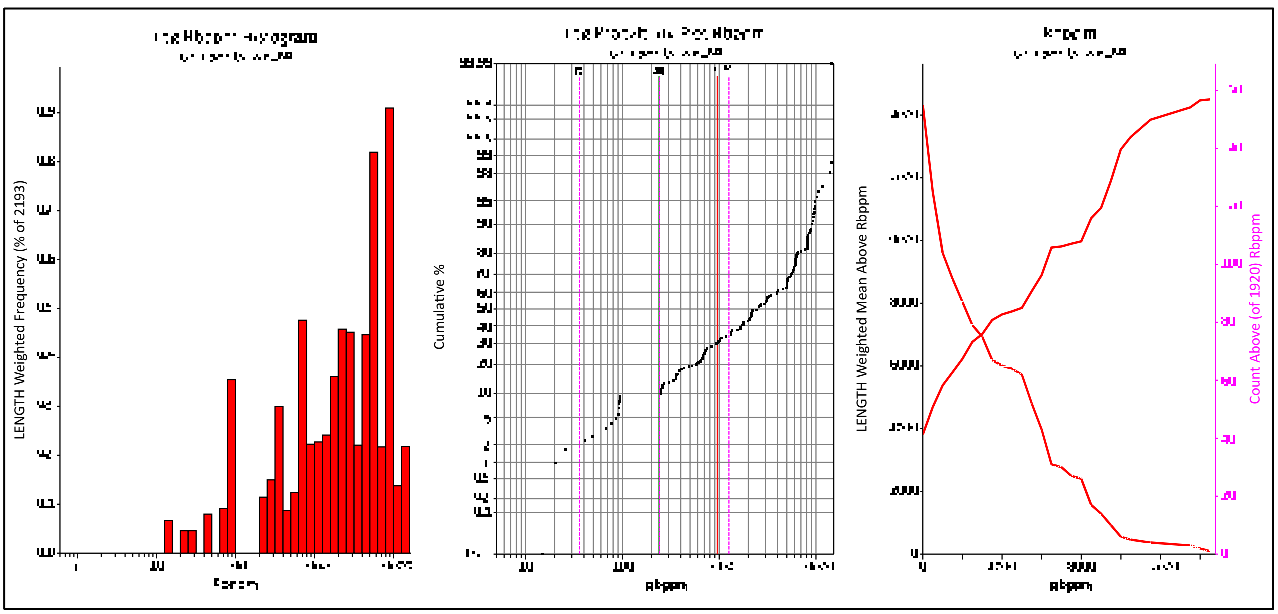


Figure 14-8: Rb Distribution for Samples within Rubidium Domain (Source: Nordmin, 2023)

14.5 Data Preparation

Prior to grade estimation, the data was prepared in the following manner for each of the Domains:

- Unsampled intervals were assigned a half-minimum detection limit value (7.5 ppm lithium, and 0.2 ppm rubidium).
- The raw assay data was independently "flagged" for each Domain and Zone by the assignment of integer codes.
- Flagged assays for each Domain and Zone were statistically analyzed to define appropriate capping, modelling procedures, and parameters.
- High grade outlier samples in each Zone were individually examined, and the distribution of grade within the domains was found to be acceptable and no capping was required.

14.5.1 Non-sampled Intervals and Minimum Detection Limits

Table 14-3 summarizes the drill hole assays at minimum detection used in the resource model. The assay table received by Nordmin contained half-minimum detection limit Li and Rb values substituted for assays below minimum detection. When non-assayed intervals exist for payable and non-payable fields, half-minimum detection values were substituted by Nordmin to remove bias from the block model. All samples were measured via Sodium Peroxide Fusion ICPOES + ICPMS and have a minimum detection limit of 15 ppm Li, and a minimum detection limit of 0.4 ppm Rb. Overlimit Rb was re-analyzed with code 8-Peroxide ICPMS/ICP with Actlabs in Dryden, ON, which has a minimum detection limit of 0.001% or 10 ppm.

Table 14-3: Assays at Minimum Detection (Source: Nordmin, 2023)

Field	Count	Minimum Detection Limit	Count at Minimum Detection	% at Minimum Detection
Li (g/t)	1699	15	115	6.77
Rb (g/t)	1699	0.4	84	4.94

14.6 Outlier Analysis and Capping

Grade outliers are high grade assay values that are much higher than the general population of samples and have the potential to bias (inflate) the quantity of metal estimated in a block model. Geostatistical analysis using XY scatter plots, cumulative probability plots, and decile analysis was used by Nordmin to analyze the raw drill hole assay data for each Domain to determine appropriate grade capping. Statistical analysis was performed by X10 Geo software and no significant outliers were noted, and no capping was applied. This is due to the homogenous nature of the spodumene distribution within the rare-metal pegmatite intrusions.

14.6.1 Compositing

Compositing of samples is a technique used to give each sample near-equal lengths to reduce the potential for bias due to uneven sample lengths; it prevents the potential loss of sample data and reduces the potential for grade bias due to the possible creation of short and potentially high-grade composites that are generally formed along the zone contacts when using a fixed length.

The raw sample data was found to have a moderately consistent range of sample lengths. Samples captured within all Zones were composited to 1.0 m regular intervals based on the observed

modal distribution of sample lengths, which supports a 5.0 x 5.0 x 5.0 m block model (Northing x Easting x Elevation) with three sub-blocking levels (a minimum size of Northing = 0.625 m x Easting = 0.625 m x Variable Elevation). An option to use a slightly variable composite length was chosen to allow for the backstitching of shorter composites that are located along the edges of the composited interval.
All composite samples were generated within each Zone, and there are no overlaps along boundaries. The composite samples were statistically validated to ensure no material loss of data or change to each sample population's mean grade. Table 14-4 summarizes composite counts by Domain and Zone.

Table 14-4: Composite Counts by Zone for Each Domain (Source: Nordmin, 2023)

Domain	Zone	# of Composites
Lithium Pegmatites	101	1,012
	102	30
	103	192
	104	77
	105	197
	106	10
	107	16
Rubidium	201	192
Low Grade Background	99	10,660

14.7 Specific Gravity

A total of 191 SG measurements existed within the Project, provided from measurements made by Company personnel. Measurements were taken from DDH samples using the weight of the core in air versus the weight in water method (Archimedes method) by applying the following formula:

$$Specific\ Gravity = \frac{Weight\ in\ Air}{(Weight\ in\ Air - Weight\ in\ Water)}$$

SG measurements were taken across lithologies as seen in Table 14-5.

Table 14-5: Specific Gravity by Lithology (Source: Nordmin, 2023)

Lithology Abbreviation	Lithology Details	Count
GAB (Gabbro)	Gabbro	37
IV	Intermediate Volcanic	5
MV	Mafic Volcanic	57
PEG	Pegmatite	90
QTZVN	Quartz Vein	1

Nordmin determined that there were insufficient SG measurements for direct estimation, and that the most appropriate SG application was to employ and assign a weighted average for each Domain, including the Lithium Pegmatites and Rubidium Domain wireframes and the Low Grade Background Domain wireframes. SG values used can be seen in Table 14-6.

Table 14-6: Specific Gravity Assignment (Source: Nordmin, 2023)

Domain/Zone	SG Assigned
Pegmatite and Rubidium Wireframes	2.668
Background	2.973

14.8 Block Model Mineral Resource Estimation

14.8.1 Block Model Strategy and Analysis

A series of upfront test modelling was completed to define an estimation methodology to meet the following criteria:

- Representative of the Project geology, structural models, and geological controls on mineralization.
- Accounts for the variability of grade, orientation, and continuity of mineralization.
- Controls the smoothing (grade spreading) of grades and the influence of outliers.
- Accounts for most of the mineralization.
- Is robust and repeatable within domains.
- Supports multiple high grade and low-grade zones.

Multiple test scenarios were evaluated to determine the optimum processes and parameters to use to achieve the stated criteria. Each scenario was based on NN, ID2, ID3, and OK interpolation methods.

All test scenarios were evaluated based on global statistical comparisons, visual comparisons of composite samples versus block grades, and the assessment of overall smoothing. Based on the results of the testing, it was determined that all scenarios would constrain the mineralization by using hard wireframe boundaries to control the spread of high grade and low-grade mineralization. OK was selected as the most representative interpolation method for the estimation.

14.8.2 Block Model Definition

Block model shape and size is typically a function of the geometry of the Project, the density of sample data, drill hole spacing, and the selected mining unit. Block models were defined with parent blocks at 5.0 x 5.0 x 5.0 m (Northing x Easting x Elevation). Block model parameters are defined in Table 14-7.

All wireframe volumes were filled with blocks from the prototype (which used the parameters in Table 14-7). Block volumes were compared to the wireframe volumes to confirm there were no significant differences, and block volumes for all wireframes were found to be well within reasonable tolerance limits. Sub-blocking was allowed to maintain the geological interpretation and to accommodate the domaining and category application. Sub-blocking has been allowed to the following: 5.0 x 5.0 x 5.0 m blocks are sub-blocked three-fold to a minimum extreme of 0.625 x 0.625 m in the N-S and E-W directions with a variable elevation calculated based on the other two block dimensions.

Table 14-7: Block Model Definition

Item	Block Origin	Block Maximum	Block Extent (m)	Block Dimension (m)	Number of Blocks	Minimum Sub-Block (m)
Easting	574,500	577,400	549,800	5	580	0.625
Northing	5,472,700	5,475,900	4,981,000	5	640	0.625
Elevation	-50	700	300	5	150	Variable

Block models were not rotated, were clipped to topography, and an overburden layer was coded in the block model. No estimated grade exists above the bottom contact of the overburden. The MRE was conducted using Datamine Studio RMTM version 1.13.202.0 within the NAD83 UTM Zone 15N datum.

Each Domain was independently estimated, resulting in three independent block models which were subsequently and appropriately combined into one overall resource block model.

14.8.3 Interpolation Method

The Project block models were estimated using NN, ID2, ID3, and OK interpolation methods for global comparisons and validation purposes. The OK method was selected for the MRE, and was selected over NN, ID2, and ID3 as the method best controlling estimation and smoothing of grades and was the most representative of all domains in the Project and was well-supported by variography.

14.8.4 Search Strategy

Zonal controls were used to constrain the grade estimates to each wireframe. These controls prevented the samples from individual wireframes from influencing the block grades of others, acting as a “hard boundary” between the wireframes.

The search orientation strategy determined to be most representative of the mineralization for the Project was to use an overall search ellipsoid.

Estimation passes were defined with carefully selected search distances. The three passes of increasing distance were as follows (major axis x semi-major axis x minor axis). Overall search parameters can be found in Table 14-8.

- Lithium and Rb Domain:
  - First Pass: 50 x 20 x 25 m
  - Second Pass: 87.5 x 35 x 43.75 m
  - Third Pass: 125 x 50 x 37.5 m
- Low Grade Background Domain:
  - First Pass: 25 x 35 x 30 m
  - Second Pass: 37.5 x 52.5 x 45 m
  - Third Pass: 75 x 105 x 90 m.

Table 14-8: Search Parameters (Source: Nordmin, 2023)

Domain	Ellipsoid Rotation Angles			Ranges, Search Pass 1 (m)			Ranges, Search Pass 2 (m)			Ranges, Search Pass 3 (m)			Composites, Pass 1		Composites, Pass 2		Composites, Pass 3		Max Composites
	1 (X)	2 (Z)	3 (Y)	1	2	3	1	2	3	1	2	3	Min	Max	Min	Max	Min	Max	Per Hole
Li and Rb Domains	47	-10	-18	50	20	25	87.5	35	30	125	50	37.5	3	12	3	12	2	8	3
Background Domain	47	-10	-18	25	35	30	37.5	52.5	45	75	105	90	3	12	3	12	2	8	3

14.8.5 Assessment of Spatial Grade Continuity

Datamine™, X10 Geo™, and Leapfrog Edge™ 2021.2 was used to determine the geostatistical relationships of the Project Independent variography was performed on composite data for Li, Rb, and rare-elements of interest which includes Ta, Cs, Be, and Nb. Experimental grade variograms were calculated from the capped and composited sample Li and Rb data to determine the approximate search ellipsoid dimensions and orientations. Variography parameters can be found in Table 14-9. Semivariograms can be found in Figure 14-9 to Figure 14-16.

The analyses considered the following:

- Downhole variograms were created and modelled to define the nugget effect.
- Experimental pairwise-relative correlogram variograms were calculated to determine directional variograms for the strike and down-dip orientations.
- Variograms were modelled using a spherical fit.
- Directional variograms were modelled using the nugget defined in the downhole variography, and the ranges for strike, perpendicular to strike, and down dip directions.
- Variograms outputs were re-oriented to reflect the orientation of the mineralization.
- Individual variograms were created for each Domain and applicable grade.

Table 14-9: Variography Parameters (Source: Nordmin, 2023)

Domain	Element	Rotation Angles				Structure 1				Structure 2				Nugget
		1	2	3	Axes	Range 1	Range 2	Range 3	C1	Range 1	Range 2	Range 3	C2	
Pegmatite 1	Li	200	45	60	Z-X-Z	22.4	29.5	43.0	0.083	120.9	173.4	51.6	0.828	0.093
Pegmatite 1	Rb	185	35	80	Z-X-Z	49.0	17.0	43.0	0.529	126.0	129.0	52.0	0.040	0.431
Li, Rb, BG	Be	48	0	85	Z-X-Z	50.14	130.0	43.0	0.420	134.8	156.0	51.6	0.467	0.114
Li, Rb, BG	Cs	180	35	85	Z-X-Z	57.8	130.0	11.3	0.592	184.1	156.0	52.8	0.246	0.162
Li, Rb, BG	Nb	48	0	90	Z-X-Z	92.0	89.5	9.7	0.382	253.2	156.0	52.8	0.419	0.190
Li, Rb, BG	Ta	0	0	90	Z-X-Z	73.5	38.4	44.0	0.027	246.3	87.2	52.8	0.699	0.279
Background	Li	285	0	85	Z-X-Z	73.5	180.0	60.0	0.003	373.3	365.5	72.0	0.878	0.119
Background	Rb	30	5	90	Z-X-Z	120.0	35.4	22.3	0.446	443.6	293.5	72.0	0.350	0.204

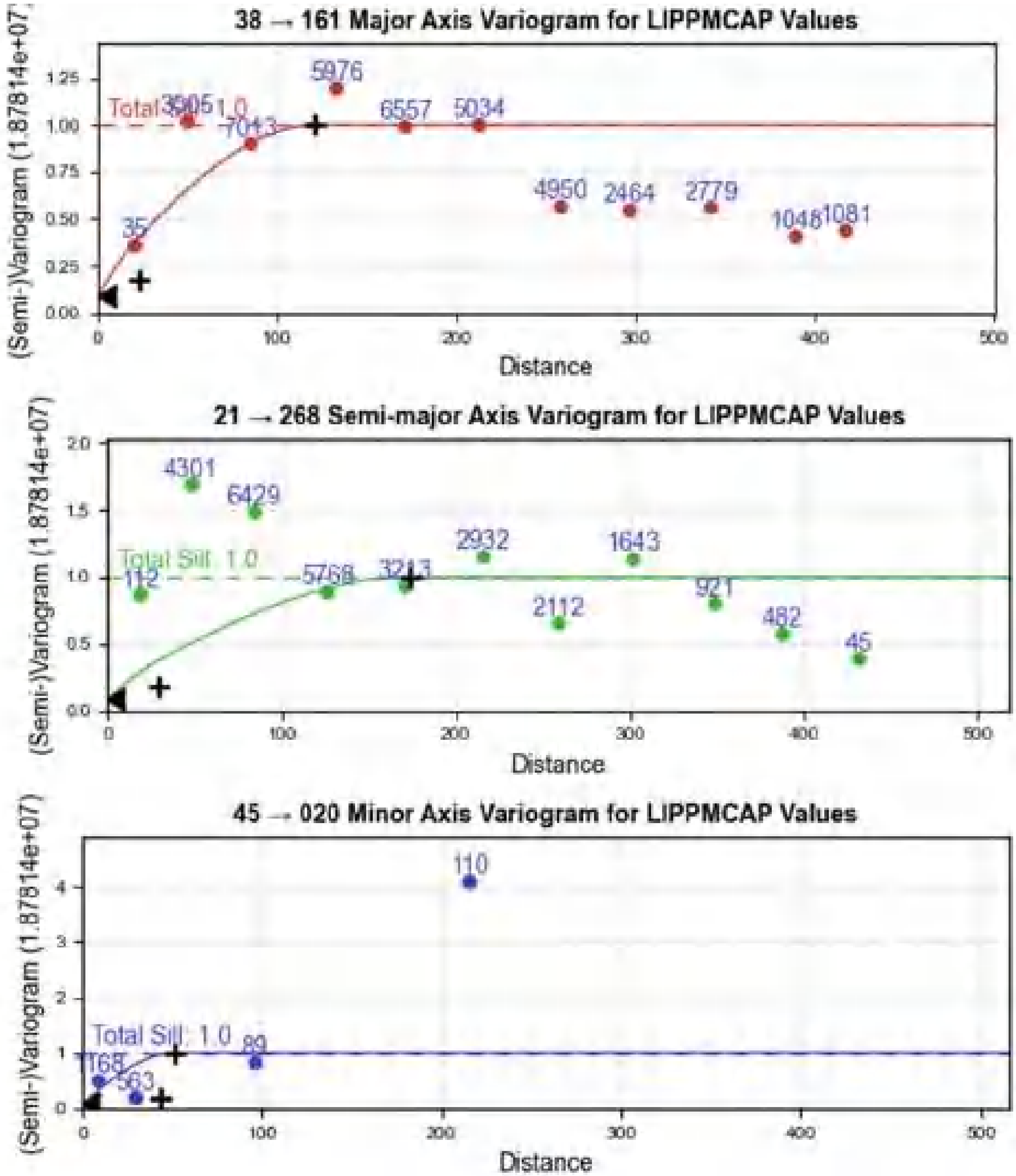


Figure 14-9: Semivariograms for Li within the Lithium Pegmatite Domain, Zone 1 (Source: Nordmin, 2023)

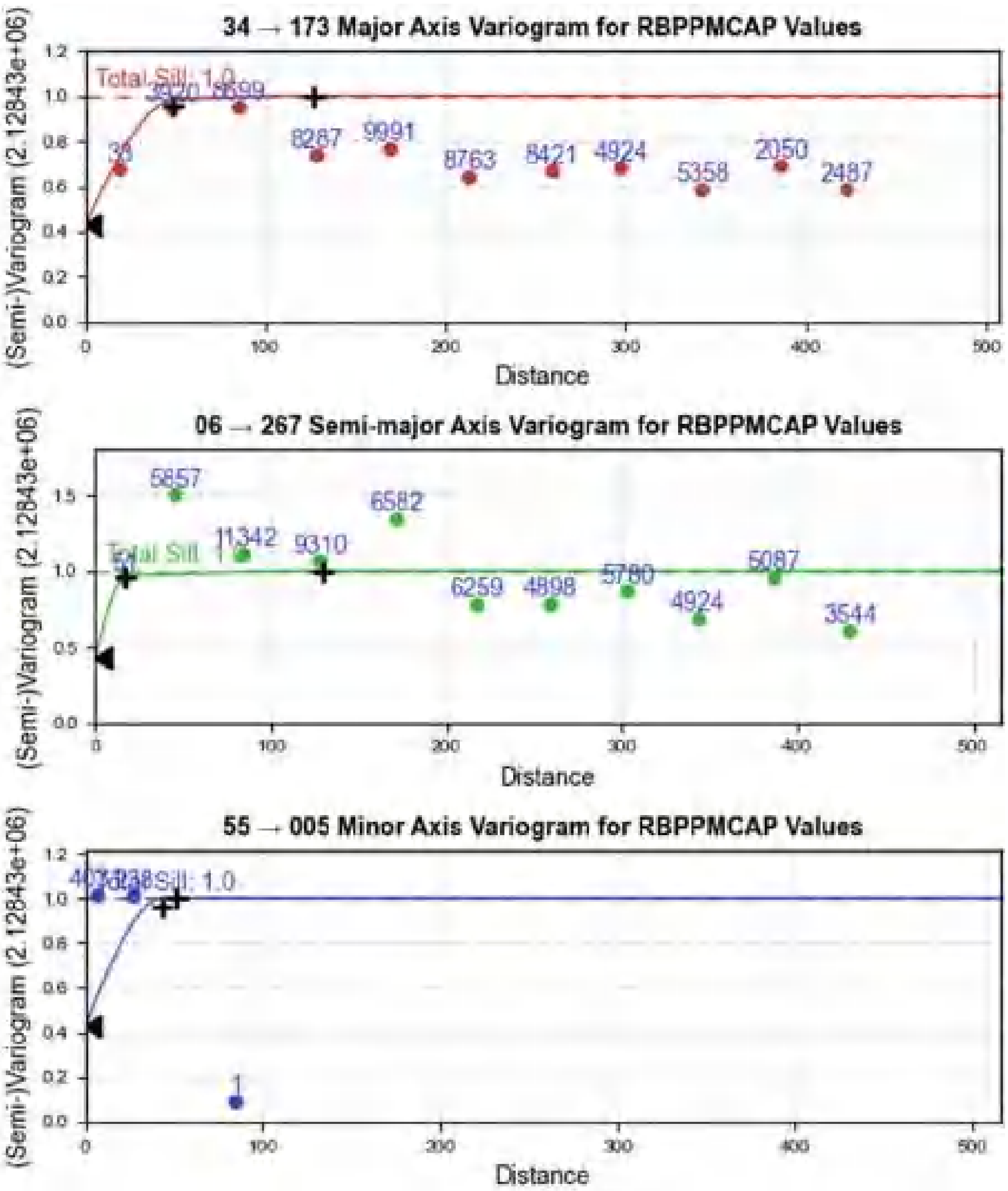


Figure 14-10: Semivariograms for Rb within the Lithium Pegmatite Domain, Zone 1 (Source: Nordmin, 2023)

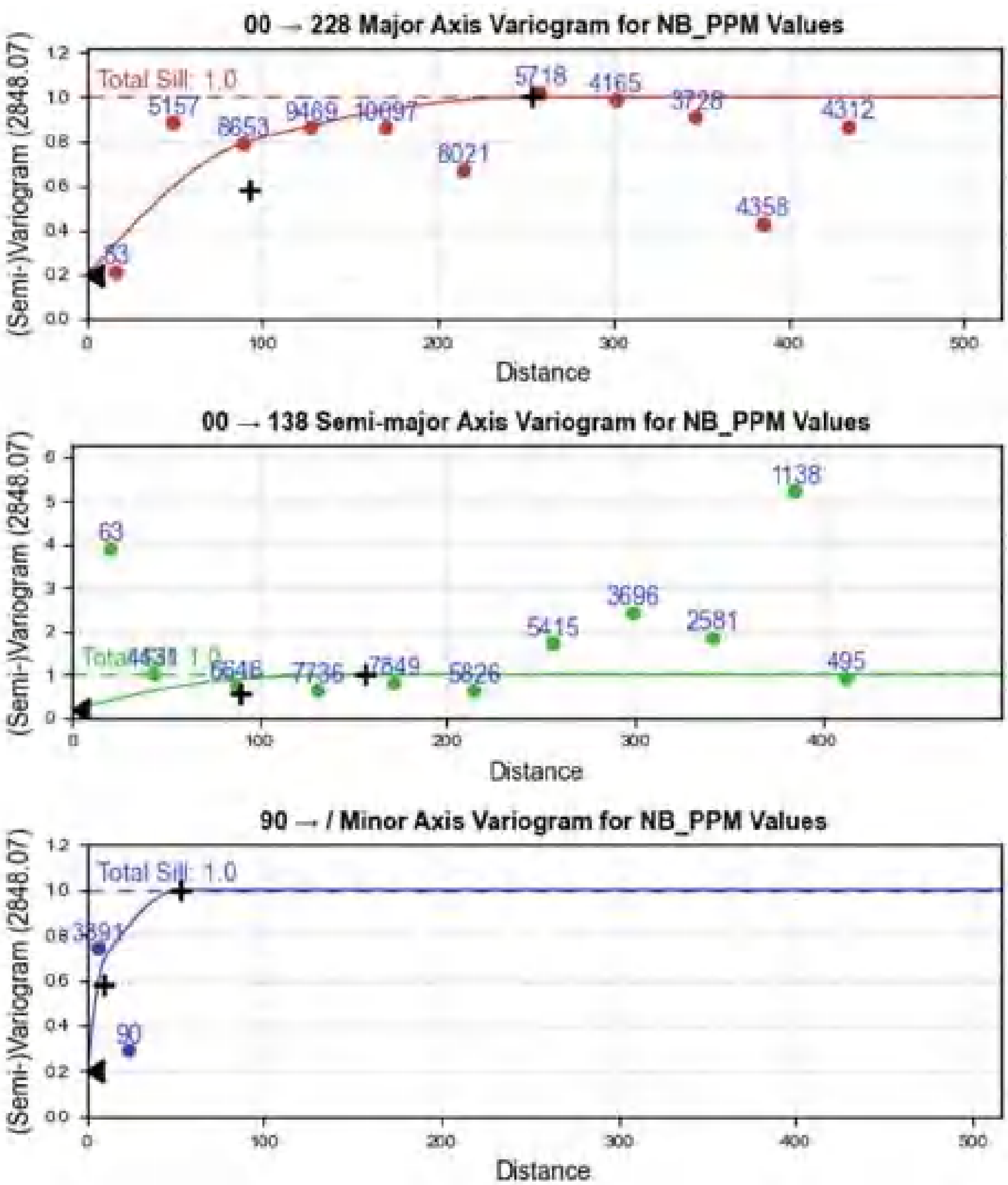


Figure 14-11: Nb Semivariograms within the Li Pegmatite Domain (Source: Nordmin, 2023)

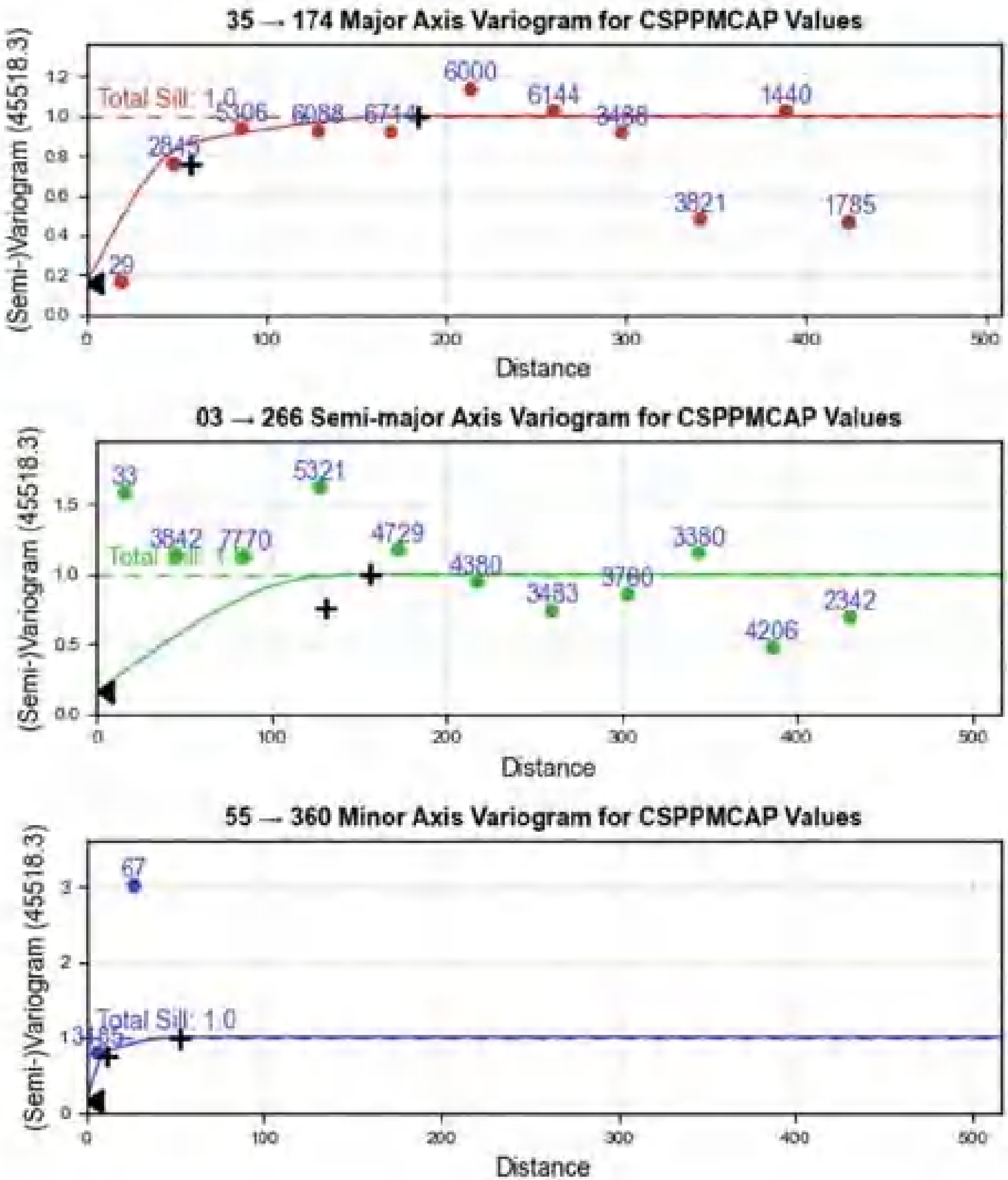


Figure 14-12: Cs Semivariograms within the Li Pegmatite Domain (Source: Nordmin, 2023)

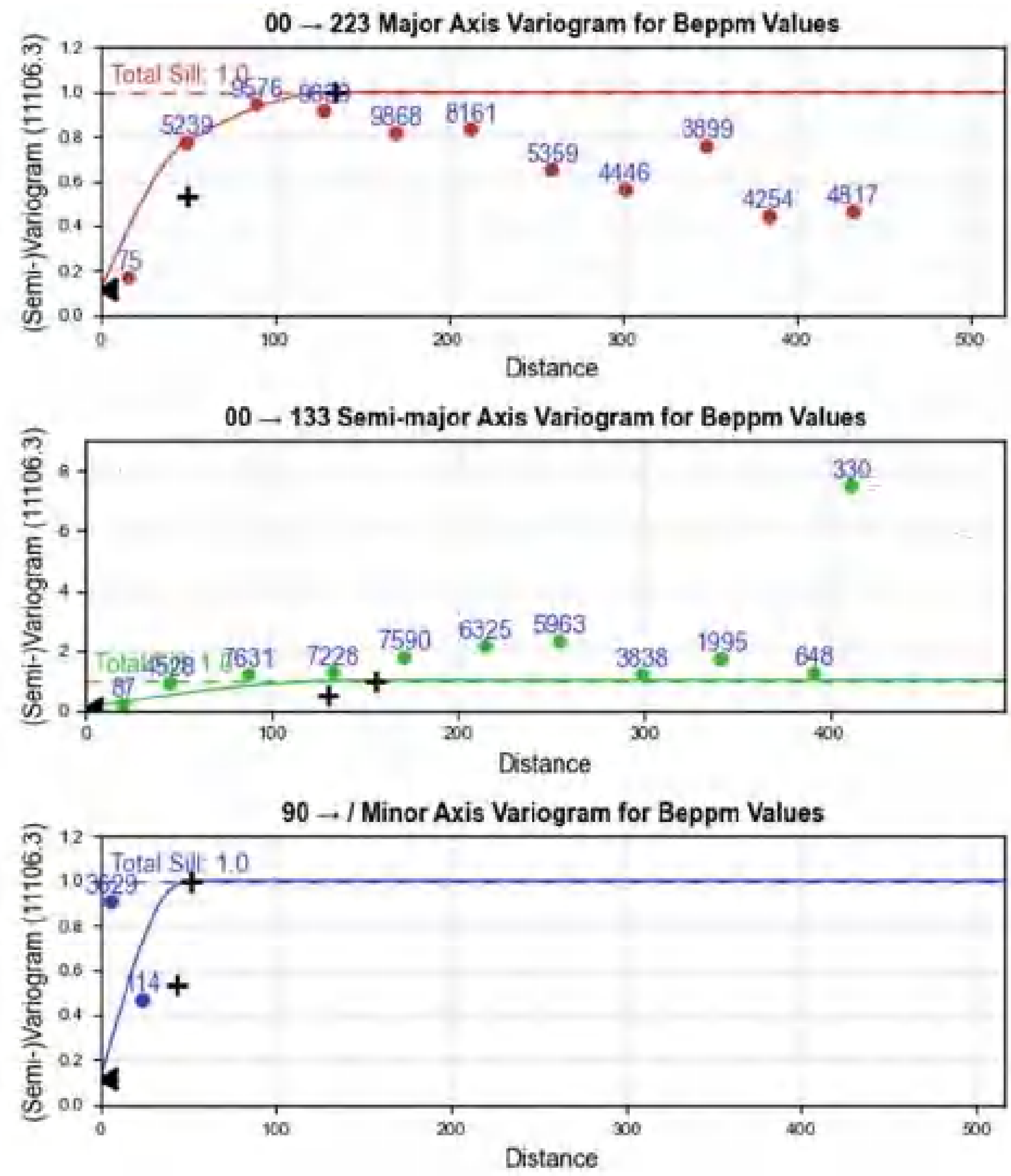


Figure 14-13: Be Semivariograms within the Li Pegmatite Domain (Source: Nordmin, 2023)

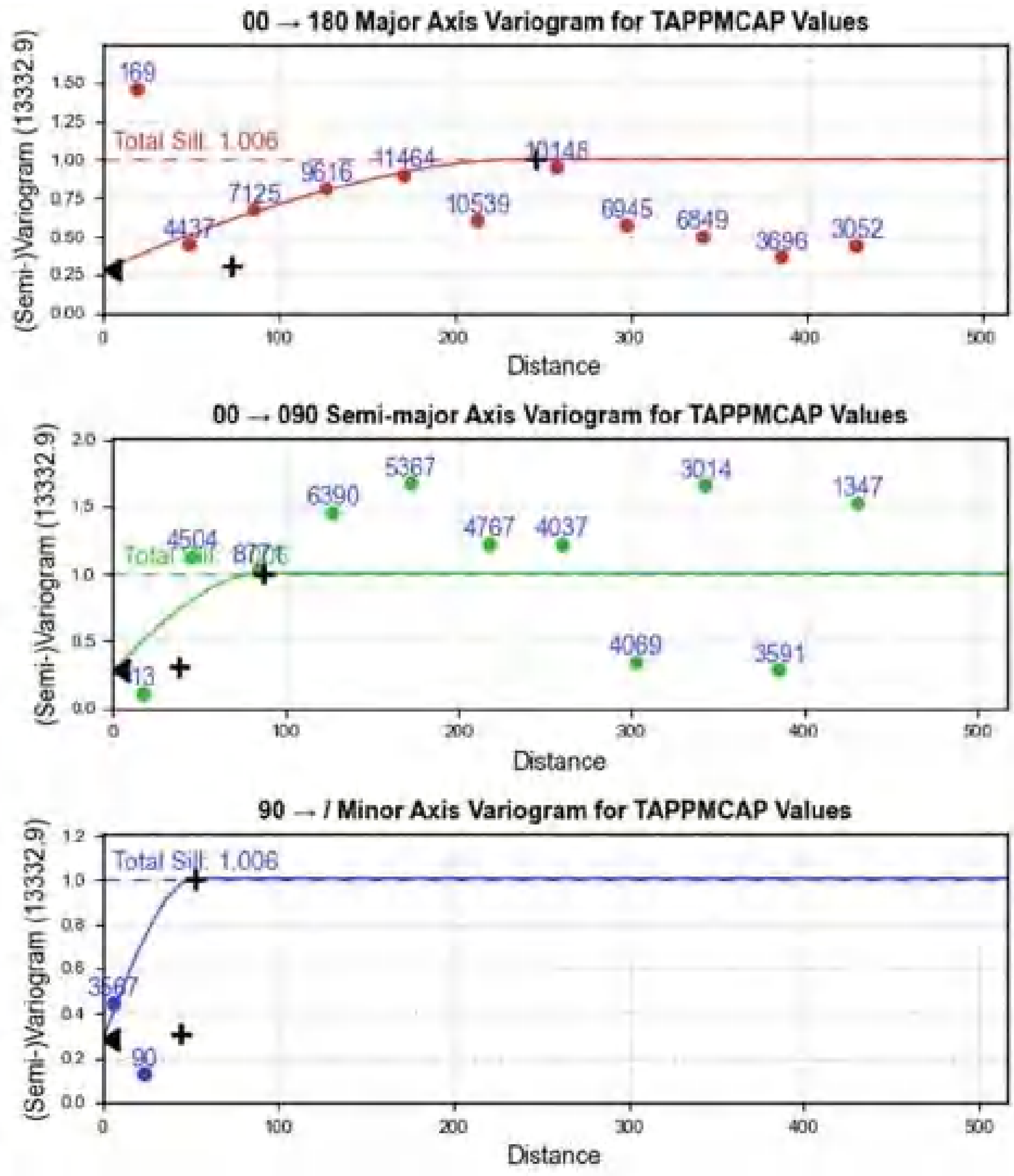


Figure 14-14: Ta Semivariograms Using Values within the Li Pegmatite Domain (Source: Nordmin, 2023)

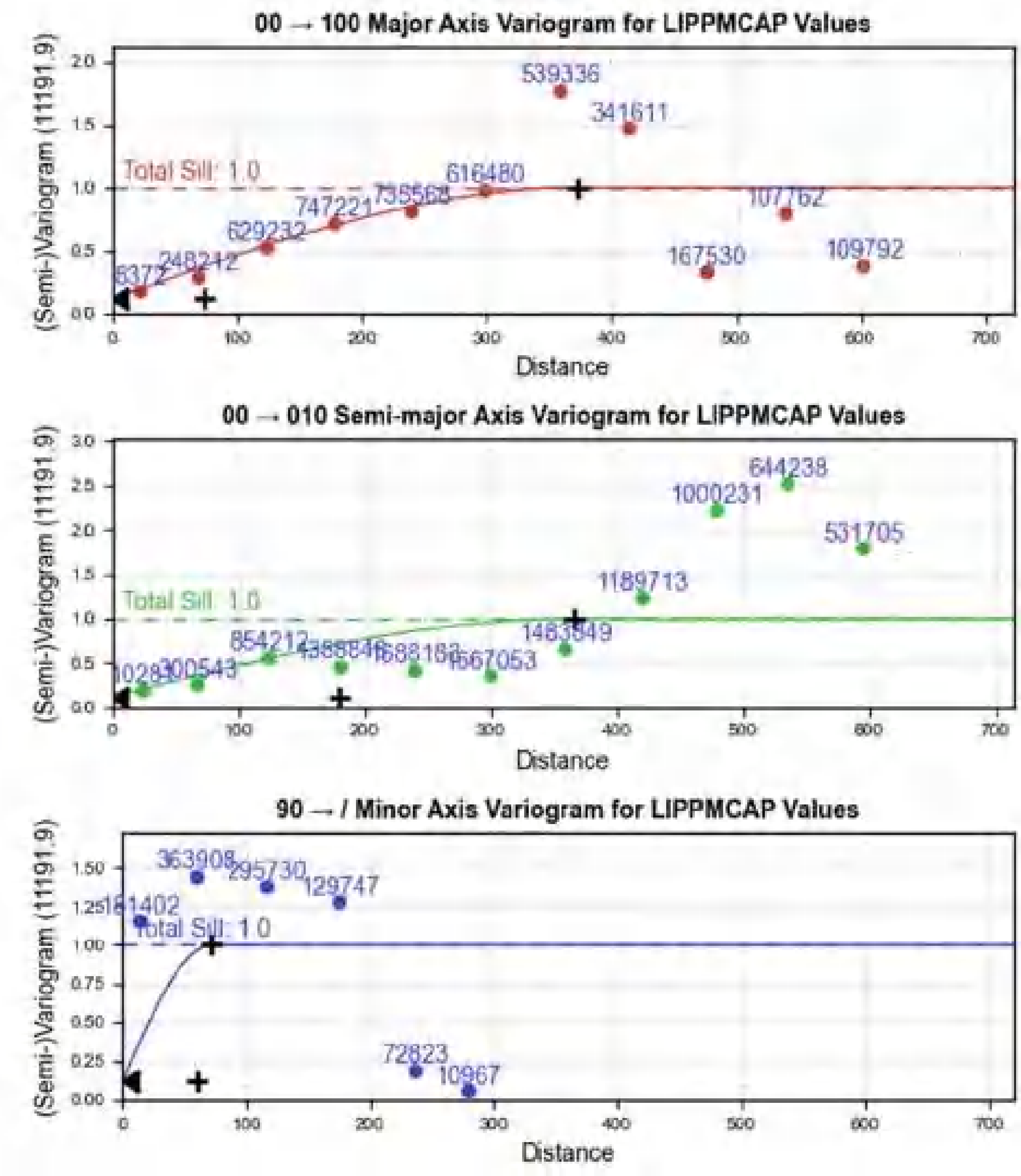


Figure 14-15: Semivariograms for Li within the Low Grade Background Domain (Source: Nordmin, 2023)

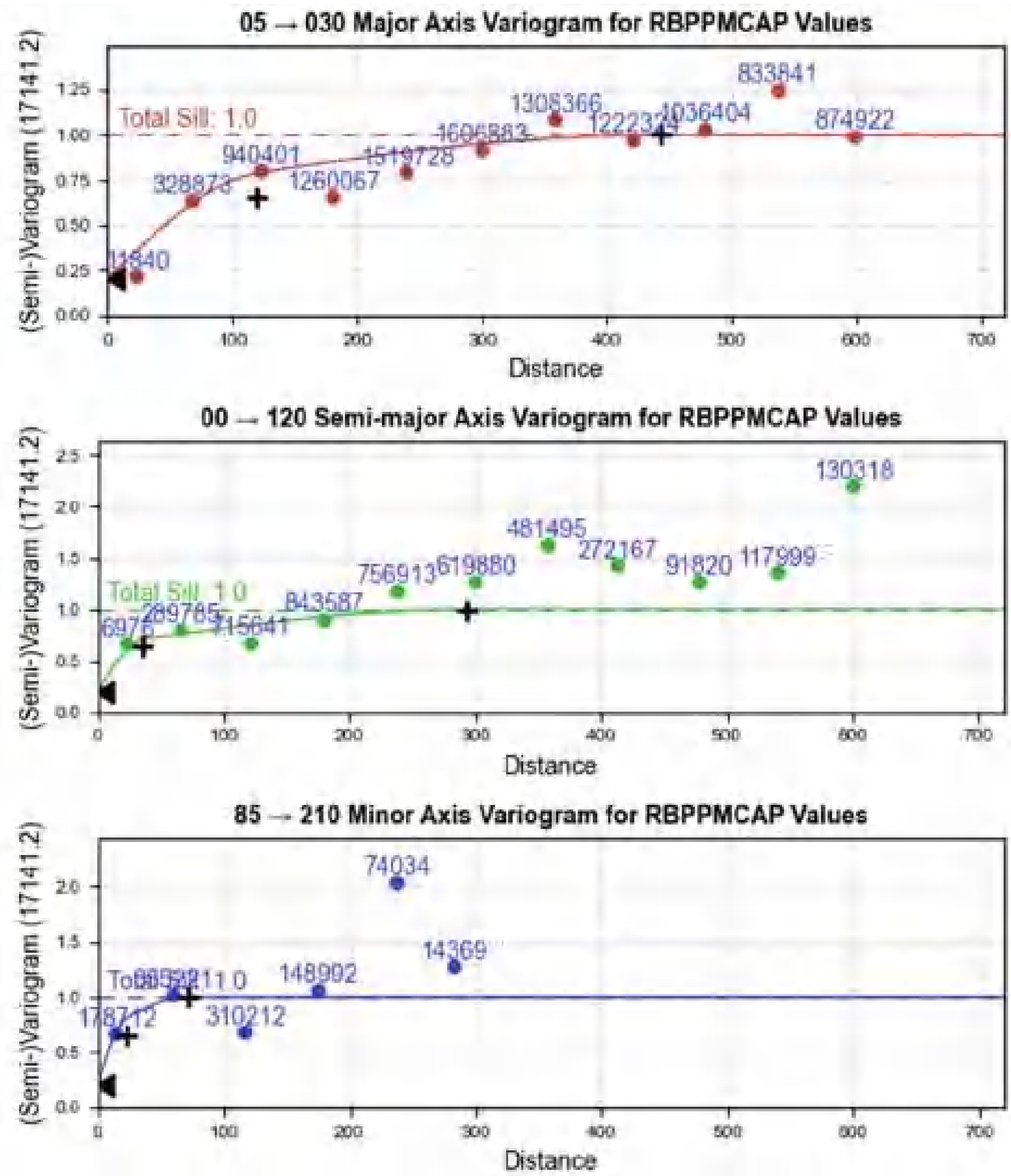


Figure 14-16: Semivariograms for Rb within the Low Grade Background Domain (Source: Nordmin, 2023)

14.9 Block Model Validation

The block model validation process included visual comparisons between block estimates and composite grades in plan and section, local versus global estimates for nearest neighbour ("NN"), inverse distance squared ("ID2"), inverse distance cubed ("ID3"), and ordinary kriging ("OK"), as well as swath plots. Block estimates were visually compared to the drill hole composite data in all domains and corresponding zones to ensure agreement. No material grade bias issues were identified, and the block model grades compared well to the composite data.

14.9.1 Visual Block Model Validation

The validation of the interpolated block model was assessed by using visual assessments and validation plots of block grades versus capped assay composites. The review demonstrated a good comparison between local block estimates and nearby assays and composites without excessive smoothing in the block model. Figure 14-17 through Figure 14-24 provide the visual comparisons, displaying lithium composite grades versus block model grades.

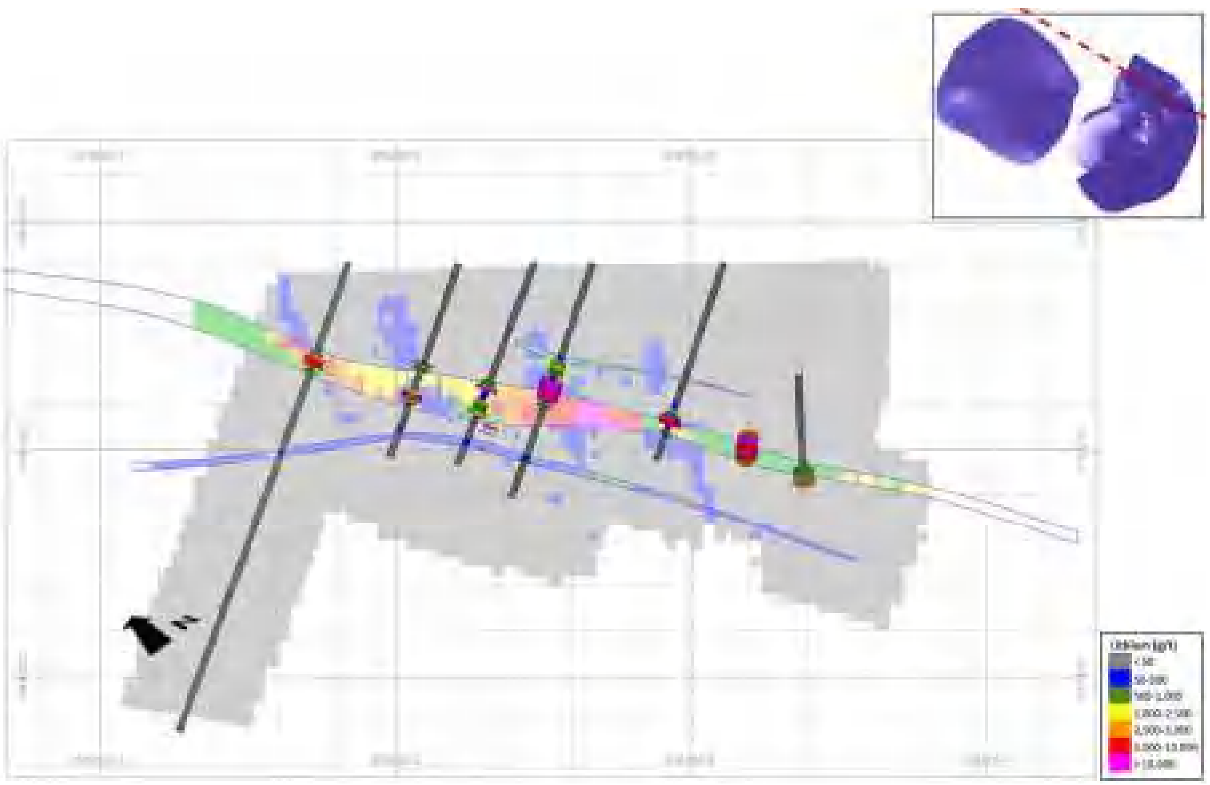


Figure 14-17: Cross-section, Block Model Validation Displaying Block Model and Capped Li Composites (Source: Nordmin, 2023)

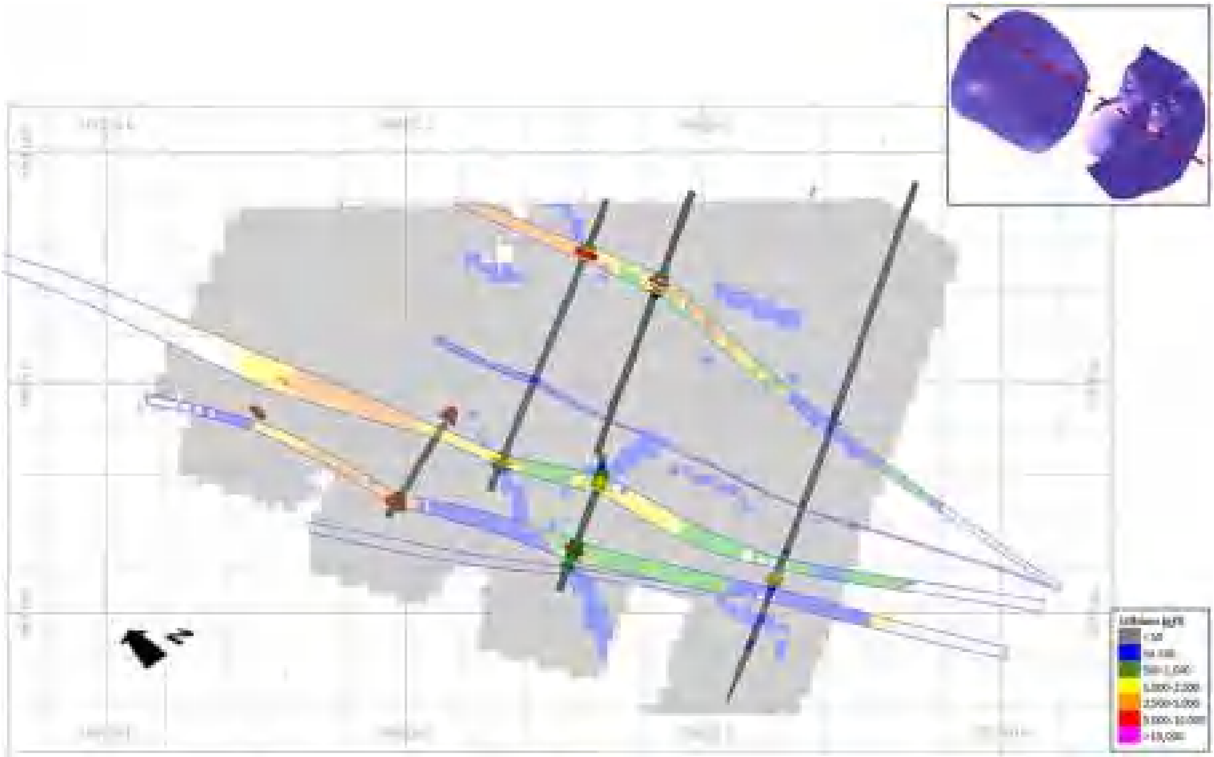


Figure 14-18: Cross-section, Block Model Validation Displaying Block Model and Capped Li Composites (Source: Nordmin, 2023)

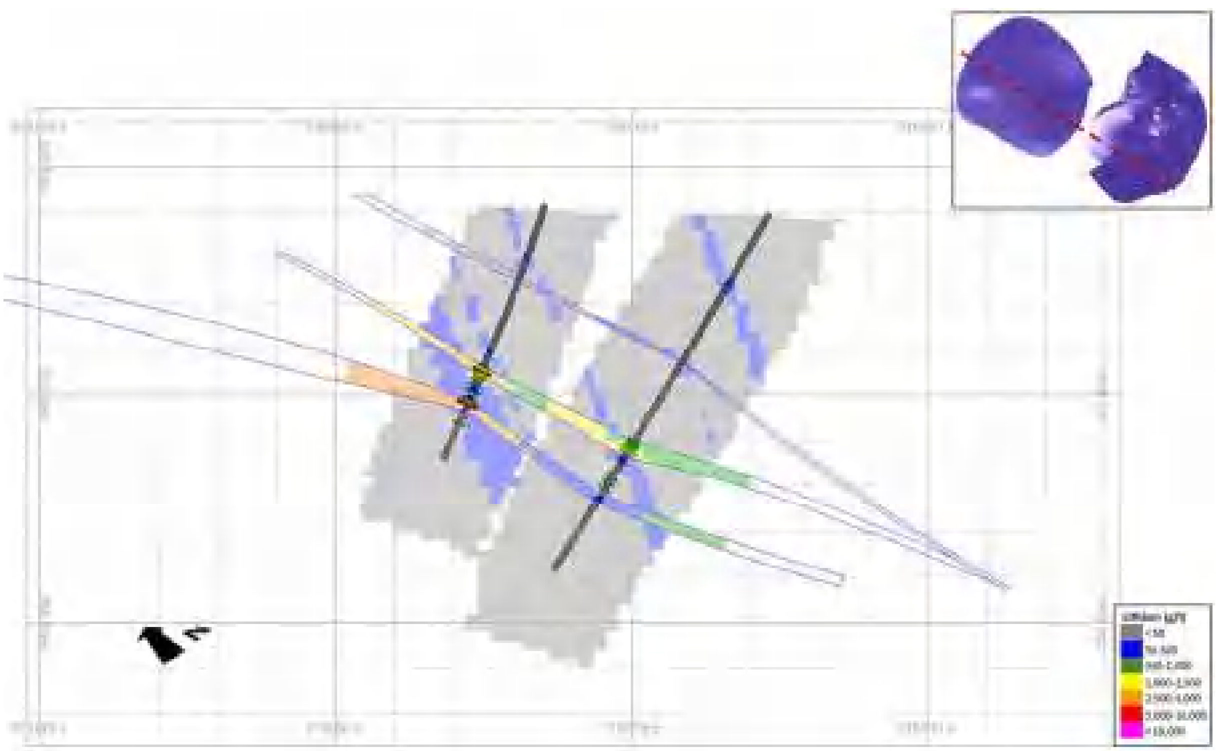


Figure 14-19: Cross-section, Block Model Validation Displaying Block Model and Capped Li Composites (Source: Nordmin, 2023)

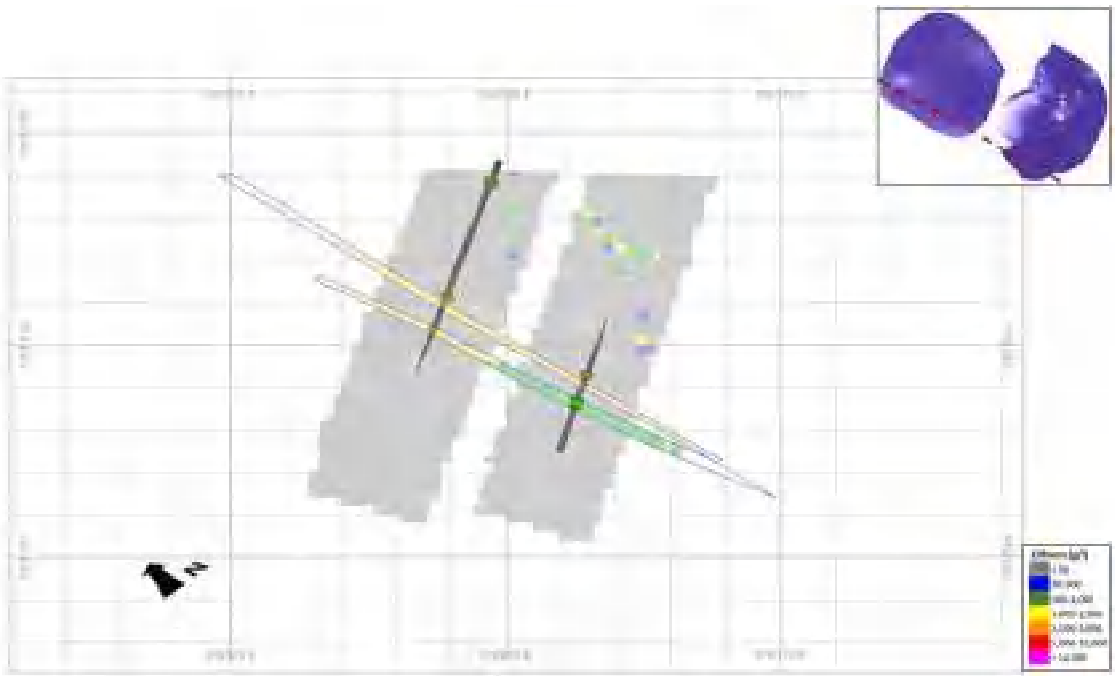


Figure 14-20: Cross-section, Block Model Validation Displaying Block Model and Capped Li Composites (Source: Nordmin, 2023)

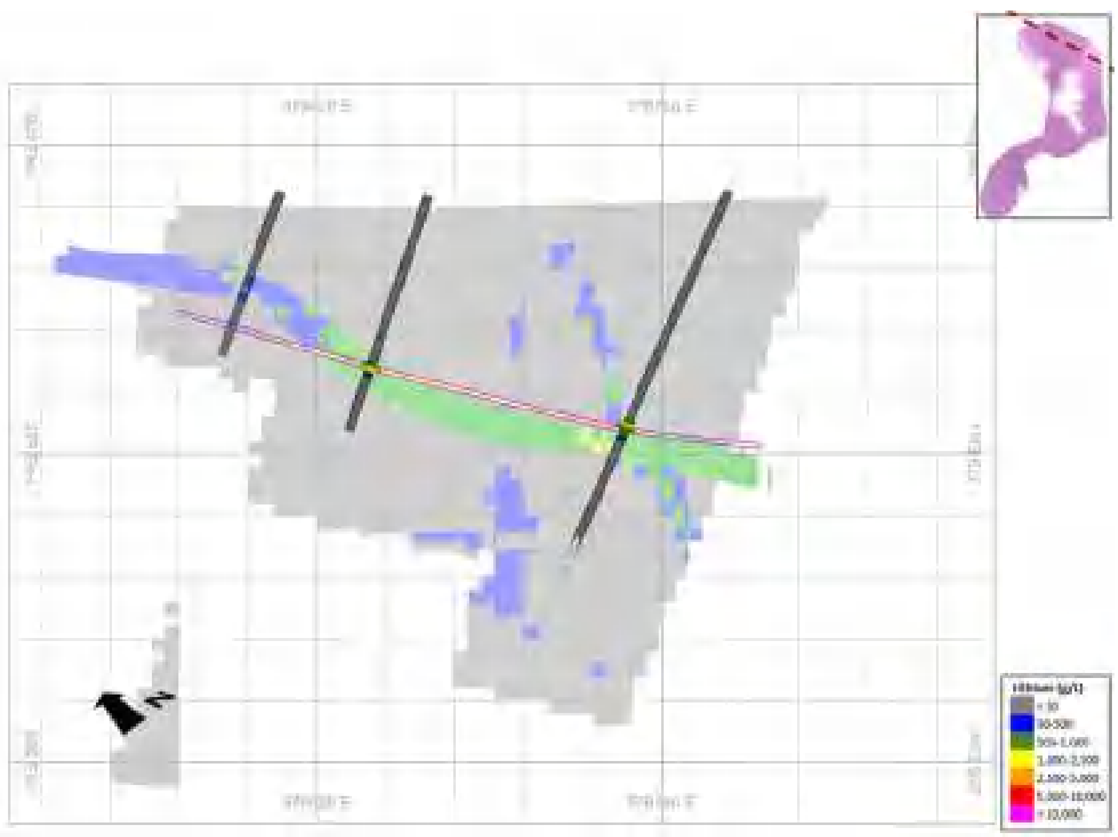


Figure 14-21: Cross-section, Block Model Validation Displaying Block Model and Capped Rb Composites (Source: Nordmin, 2023)

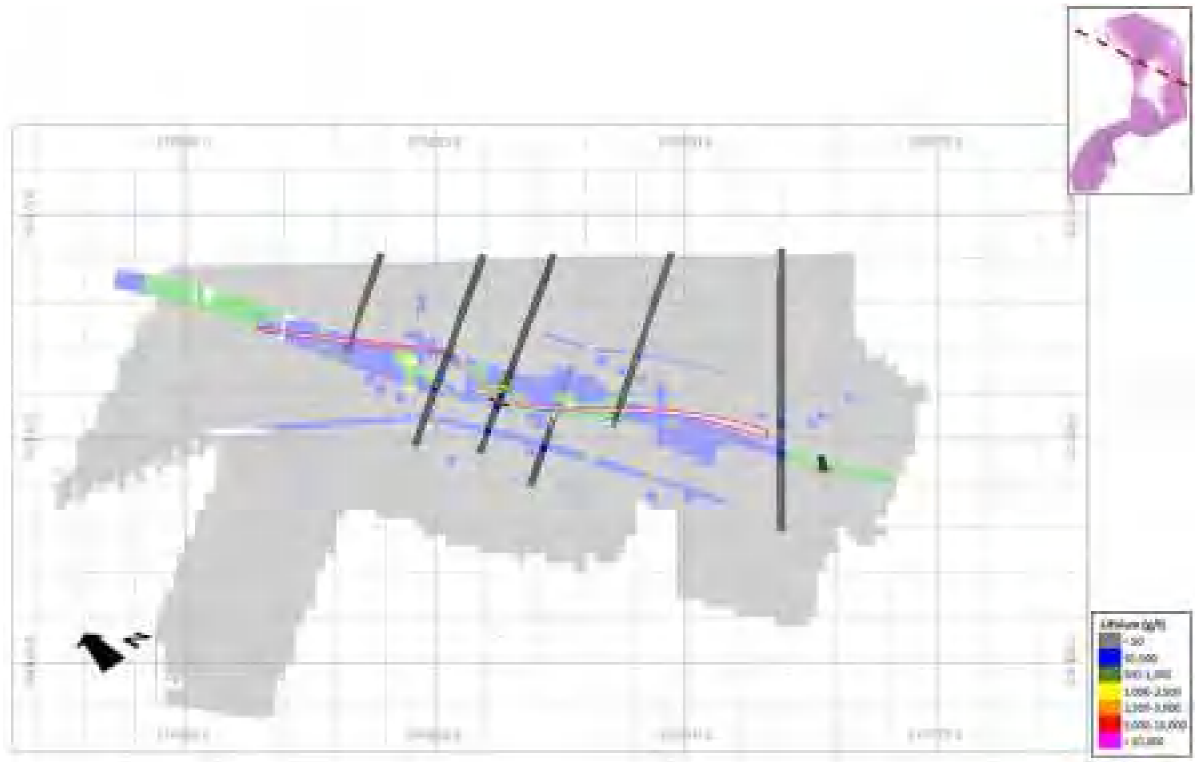


Figure 14-22: Cross-section, Block Model Validation Displaying Block Model and Capped Rb Composites (Source: Nordmin, 2023)

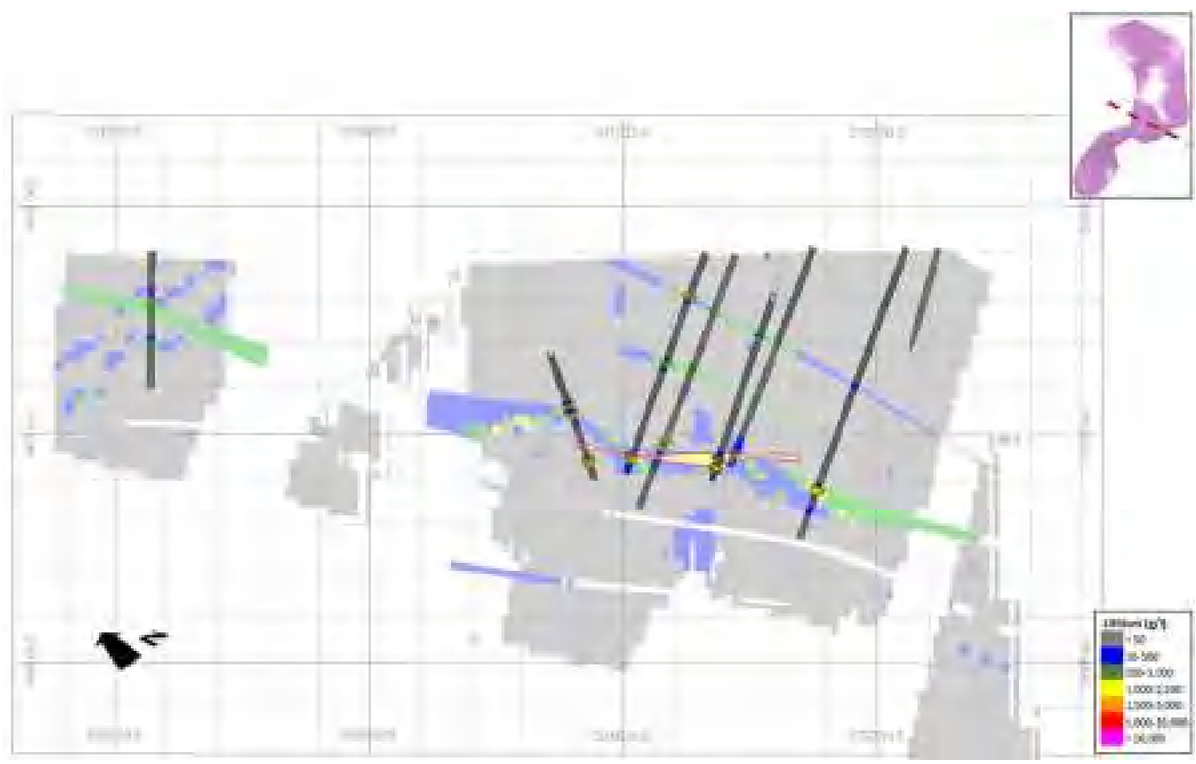


Figure 14-23: Cross-section, Block Model Validation Displaying Block Model and Capped Rb Composites (Source: Nordmin, 2023)

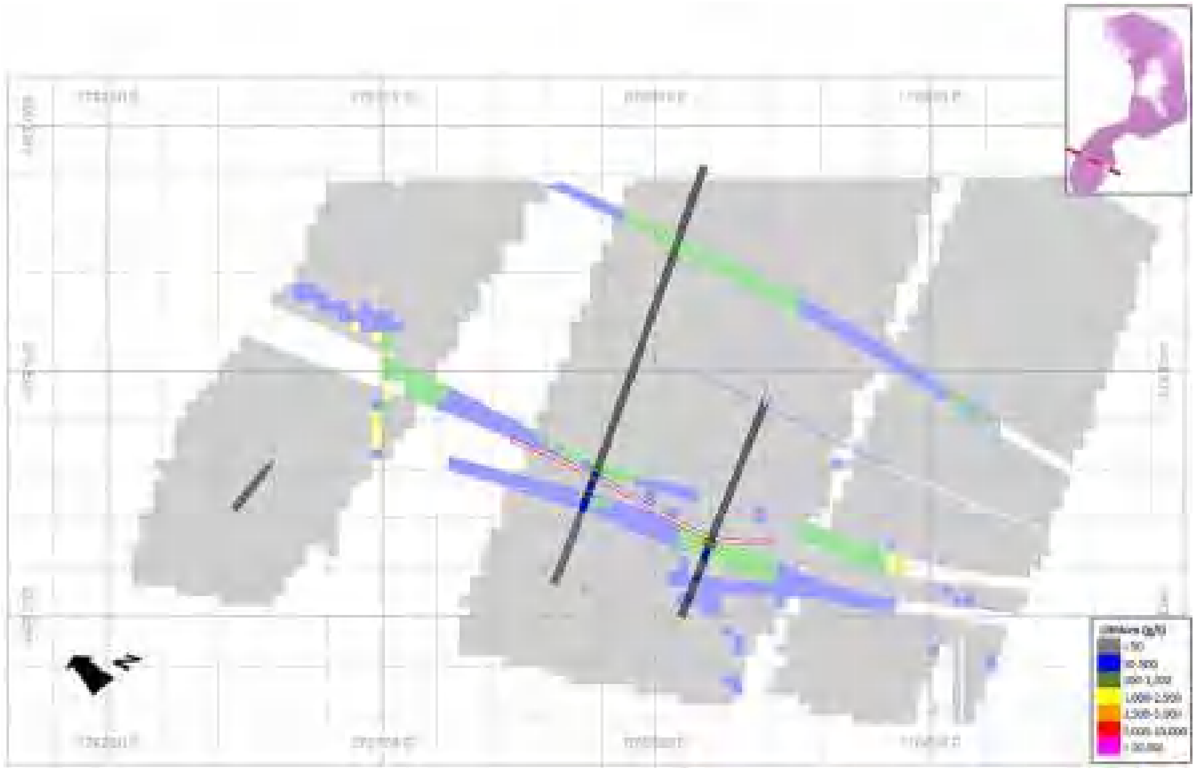


Figure 14-24: Cross-section, Block Model Validation Displaying Block Model and Capped Rb Composites (Source: Nordmin, 2023)

14.9.2 Swath Plots

A swath plot is a graphical representation of grade distribution derived by a series of sectional “swaths”, or slices, throughout the Project. Swath plots were generated for Li and Rb for each of the three Domains, including the Lithium Pegmatites, Rubidium, and Low Grade Background Domains from slices throughout the Deposit. They compare the block model grades for NN, ID2, ID3, and OK to the drill hole composite grades to evaluate any potential local grade bias. Review of the swath plots did not identify any model bias that is material to the 2021 MRE, as there was a strong overall correlation between the block model grades (across all interpolations) and the capped composites used in the 2021 MRE. The swath plots for easting, northing, and elevation respectively are found in Figure 14-25, Figure 14-26, and Figure 14-27. For these figures, the composite grades (S_LICAP and S_RBCAP) is compared across swaths with the four different interpolation estimation grades for each of Li and Rb from the block model.

Fields include:

- M_TONNES: Block model tonnage
- NRECORDS: Number of records
- S_LICAP: Composite capped lithium grade (ppm)
- S_RBCAP: Composite capped rubidium grade (ppm)
- M_LIID2, M_RBID2: Block model estimated lithium/rubidium grade, ID2 (ppm)
- M_LIID3, M_RBID3: Block model estimated lithium/rubidium grade, ID3 (ppm)
- M_LINN, M_RBNN: Block model estimated lithium/rubidium grade, NN (ppm)
- M_LIOK, M_RBOK: Block model estimated lithium/rubidium grade, OK (ppm)

Figure 14-25: Swath Plots, Lithium Pegmatite Domain Li (Source: Nordmin, 2023)

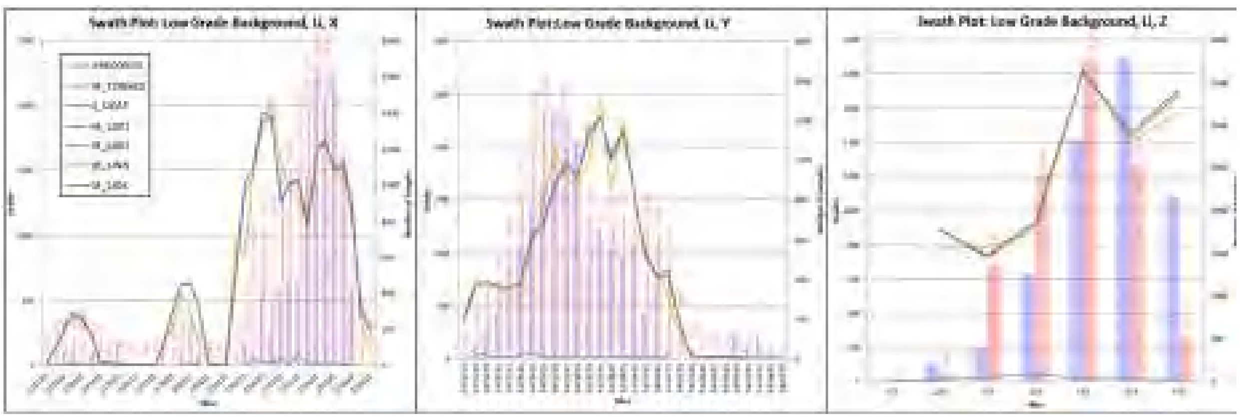


Figure 14-26: Swath Plots, Low Grade Background Domain Li (Source: Nordmin, 2023)

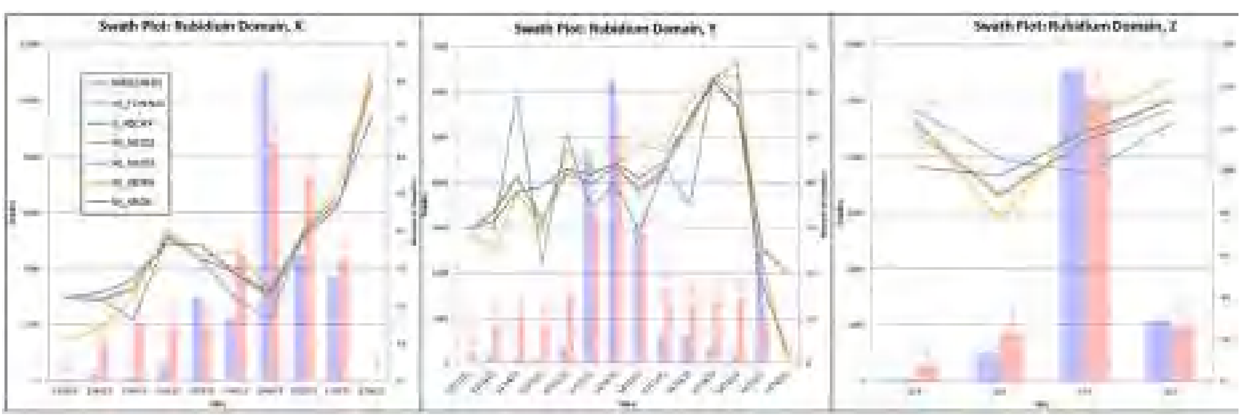


Figure 14-27: Swath Plots, Rubidium Domain Rb (Source: Nordmin, 2023)

14.10 Mineral Resource Classification

The MRE was classified in accordance with the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Mineral Resource classifications were assigned to regions of the block model based on the QPs confidence and judgment related to geological understanding, continuity of mineralization in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, search volume block estimation assignment, and minimum distance to the nearest composite.

Formulas were applied to determine an initial classification for each block, and the block model was subsequently analyzed for further adjustments. Finally, a set of adjustment wireframes were applied to the block model to correct specific areas and to avoid the “spotted dog” effect. The formulas used the following fields, which are per-block:

- Minimum transform distance to the nearest composite (“MINDIS”)
- Kriging Variance (“KVAR”)
- Number of composites used in estimation (“NUMSAMP”)
- Search estimation pass (“SVOL”)

MRE Classification was performed as follows:

- All blocks are initially labelled as non-classified.
- Block is Inferred if: MINDIS < 1.4 or (MINDIS < 1.5 and KVAR < 0.5).
- Block is Indicated if all of the following: block is in the Low-Grade Background Domain, SVOL = 1 or 2, NUMSAMP >= 5, MINDIS <= 1.7 and KVAR < 0.99.
- Block is Measured if all of the following: block is within the Lithium Pegmatite Domain, SVOL = 1, NUMSAMP >= 12, MINDIS < 0.4, and KVAR < 0.2.
- Three wireframe sets were then applied to adjust the above, resulting in the final Classification. These included:
  - Wireframes to downgrade select blocks from Measured to Indicated;
  - Wireframes to downgrade select blocks from Indicated to Inferred; and
  - Wireframes to upgrade select non-classified blocks to Inferred.

Classification can be seen in Figure 14-28 through Figure 14-31.

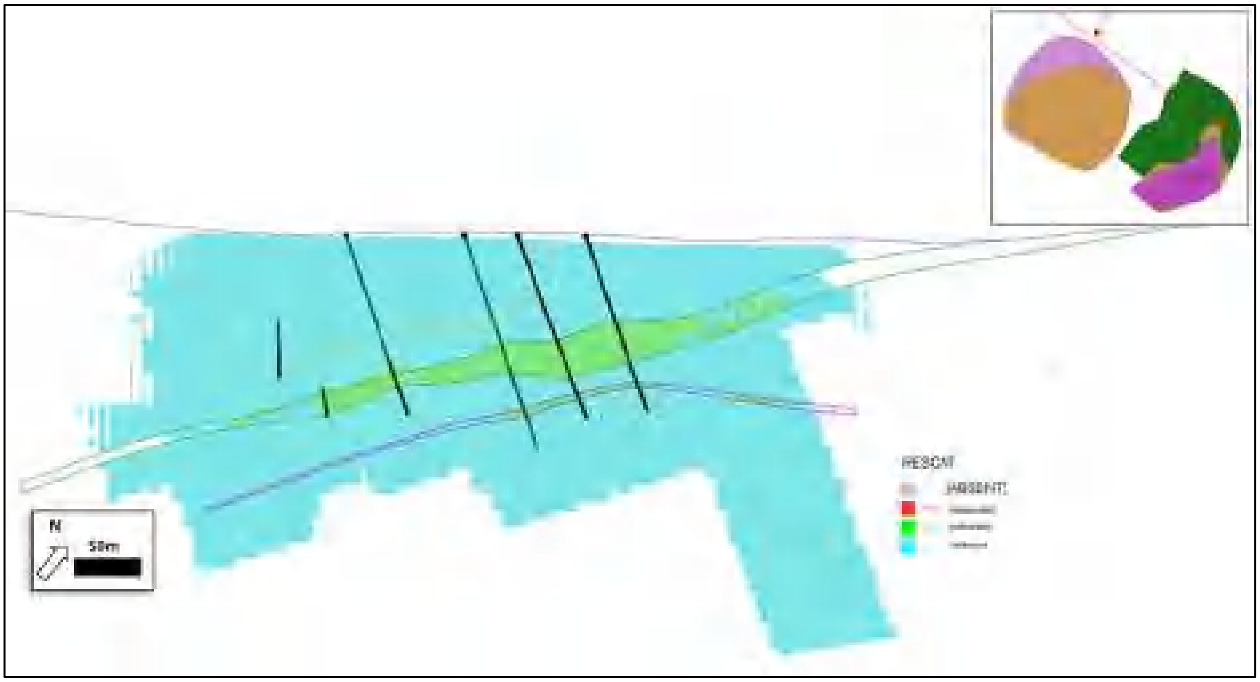


Figure 14-28: Cross-section, Resource Classification, Showing Pegmatite Domains (Source: Nordmin, 2023)

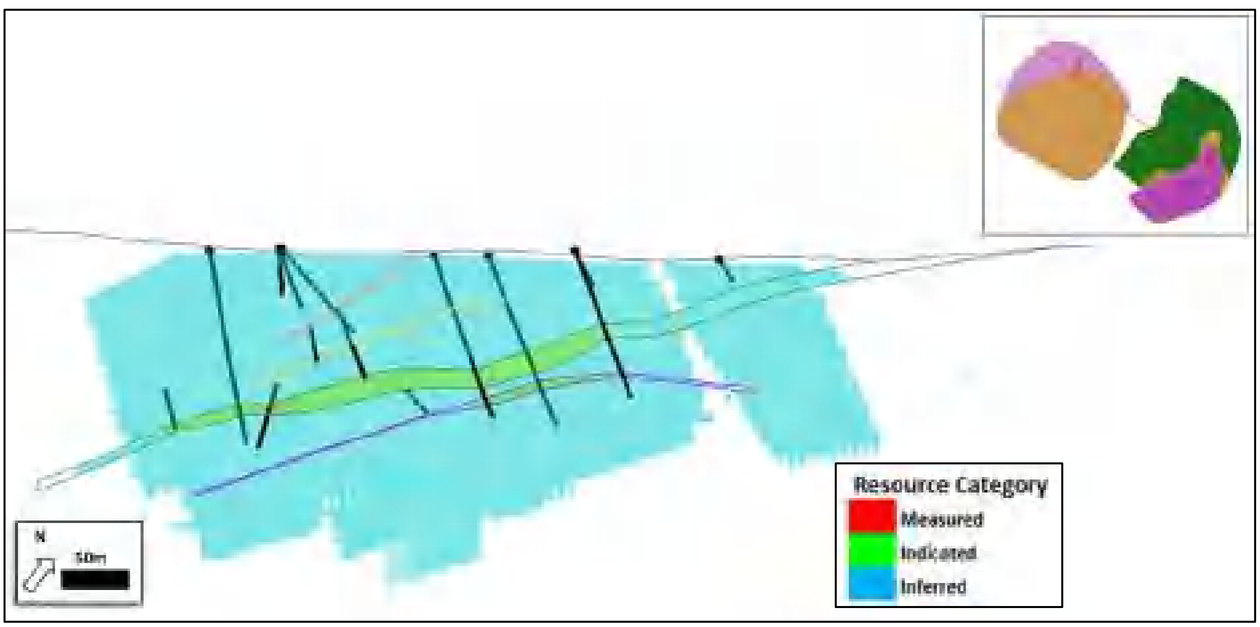


Figure 14-29: Cross-section, Resource Classification, Showing Pegmatite Domains (Source: Nordmin, 2023)

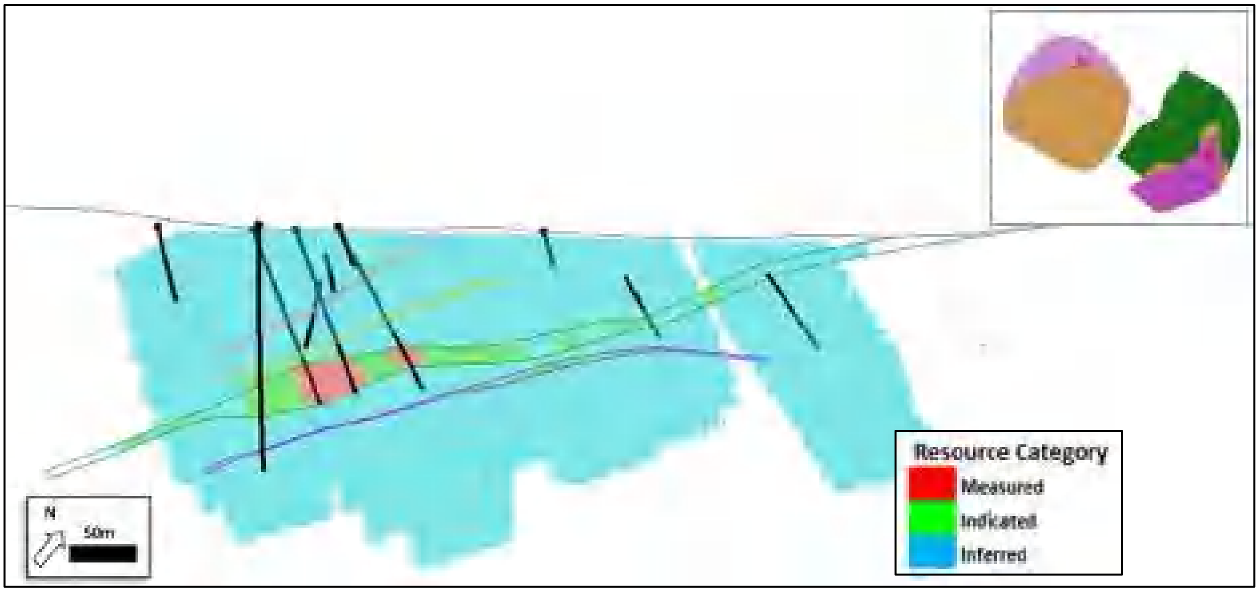


Figure 14-30: Cross-section, Resource Classification, Pegmatite Domains (Source: Nordmin, 2023)

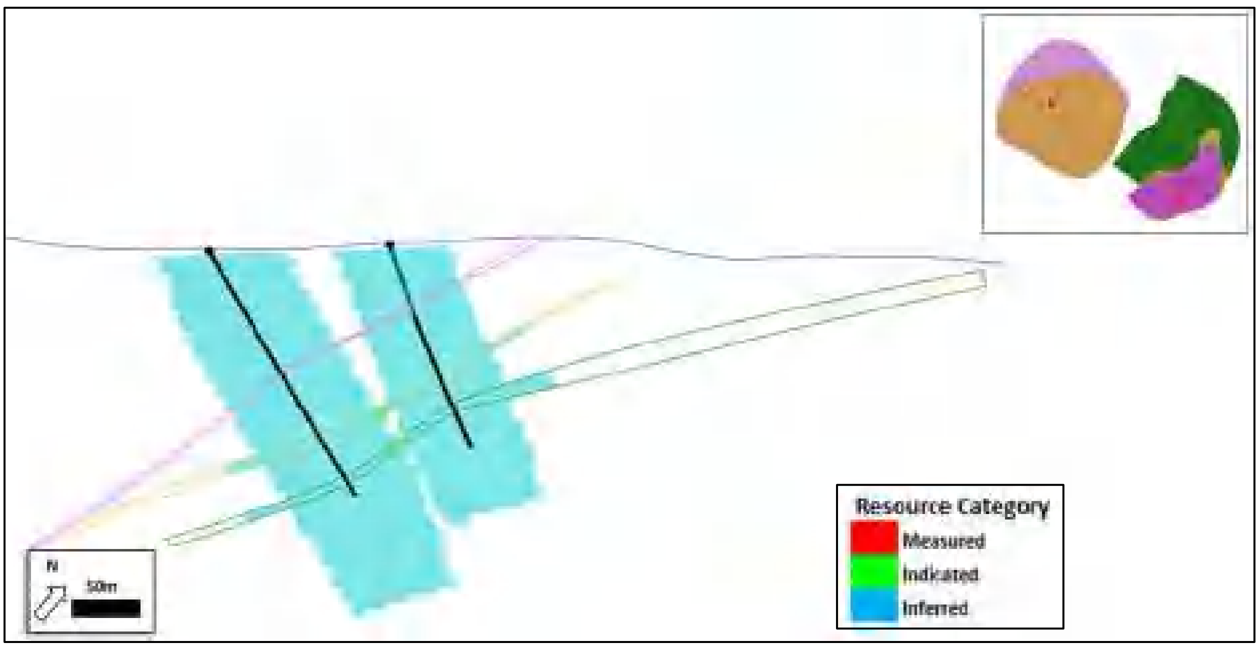


Figure 14-31: Cross-section, Resource Classification, Pegmatite Domains (Source: Nordmin, 2023)

14.11 Interpolation Comparison

Estimation was completed using NN, ID2, ID3, and OK interpolation methods. The Lithium Pegmatite Domain interpolation comparison is presented in Table 14-10 and the Rubidium Domain interpolation comparison is presented in Table 14-11.

Table 14-10: Interpolation Comparison for the Lithium Pegmatite Domain (Source: Nordmin, 2023)

Cut-off (Li ppm)	Resource Category	Li ppm OK	Li ppm NN	Li ppm ID2	Li ppm ID3
450	Measured	3,335	3,807	3,575	3,705
	Indicated	2,358	2,374	2,330	2,347
	Inferred	1,935	1,886	1,908	1,904
550	Measured	3,502	4,006	3,760	3,901
	Indicated	2,475	2,497	2,446	2,465
	Inferred	2,256	2,200	2,223	2,219
650	Measured	3,630	4,158	3,897	4,048
	Indicated	2,592	2,618	2,561	2,583
	Inferred	2,463	2,410	2,425	2,421
750	Measured	3,792	4,361	4,081	4,243
	Indicated	2,723	2,756	2,688	2,715
	Inferred	2,568	2,516	2,525	2,521
850	Measured	3,937	4,537	4,237	4,412
	Indicated	2,856	2,901	2,817	2,848
	Inferred	2,669	2,612	2,623	2,619

Table 14-11: Interpolation Comparison for the Lithium Pegmatite Domain (Source: Nordmin, 2023)

Cut-off (Rb ppm)	Resource Category	Mass (kt)	Rb ppm OK	Rb ppm NN	Rb ppm ID2	Rb ppm ID3
1,600	Measured	55	3,487	3,332	3,482	3,524
	Indicated	911	3,421	3,499	3,382	3,411
	Inferred	2,427	2,571	2,270	2,440	2,294
1,800	Measured	50	3,676	3,521	3,672	3,720
	Indicated	774	3,725	3,788	3,663	3,692
	Inferred	1,796	2,883	2,470	2,733	2,582
2,000	Measured	42	3,973	3,807	3,959	4,014
	Indicated	651	4,074	4,151	3,988	4,017
	Inferred	1,296	3,255	2,909	3,090	2,957

Cut-off (Rb ppm)	Resource Category	Mass (kt)	Rb ppm OK	Rb ppm NN	Rb ppm ID2	Rb ppm ID3
2,200	Measured	38	4,222	3,992	4,186	4,237
	Indicated	570	4,355	4,423	4,251	4,278
	Inferred	1,078	3,494	3,192	3,314	3,173
2,400	Measured	34	4,408	4,132	4,345	4,396
	Indicated	491	4,687	4,771	4,564	4,594
	Inferred	817	3,882	3,407	3,652	3,466

14.12 Reasonable Prospects of Eventual Economic Extraction

14.12.1 Lithium Open Pit

For the Lithium Open Pit Mineral Resource, shown in Table 14-16, a pit limit analysis was undertaken using the Pseudoflow algorithm in Deswik.AdvOPM software to determine physical limits for a pit shell constrained Mineral Resource. The parameters used to generate the pit shell are shown in Table 14-12.

Table 14-12: Lithium Open Pit Limit Analysis Parameters (Source: Nordmin, 2023)

Parameter	Value
Currency Used for Evaluation	CAD$
Block Size	In-Situ model regularized to 5.0 m (x) by 5.0 m (y) by 5.0 m (z)
Overall Stope Angle	Rock: 45° Overburden: 30°
Open Pit Mining Cost	$6.00/$t_{mined}$ Rock 0.8 MCAF for Overburden +$0.01/t per 5 m for depths below pit rim
Process Cost *Includes assumptions for Milling, G&A, sustaining infrastructure, closure*	$41.00/$t_{processed}$
Concentrate Transportation / Insurance	$175.00/$t_{concentrate}$
Spodumene Concentrate Grade	5.5% Li_2O
Spodumene Concentrate Price	$1,800 USD per tonne spodumene concentrate Exchange Rate: 1 USD$=1.30 CAD$ $2,340 CAD per tonne spodumene concentrate
Process Recovery	75.0%
Pit Shell Selection	RF 1.00
Production Rate Assumption	2,000 t/d

The processing CoG is used to classify the material contained within the pit shell limits as open pit resource material. This break-even CoG is calculated to cover the process and selling costs using the parameters listed in Table 14 12. The Open Pit Mineral Resource CoG is estimated to be 650 ppm lithium. For resource cut-off calculation purposes, a mining recovery of 100% and mining dilution of 0% was applied. The Open Pit Mineral Resource Estimate is reported from the model regularized to 5 x 5 x 5 m (X, Y, Z) to include must-take material. The MRE excludes unclassified mineralization located within mined out areas.

14.12.2 Lithium Underground

For the Underground Mineral Resource, shown in Table 14-16, Mineable Shape Optimizer (MSO) wireframes were created with Deswik.SO to determine physical limits for a constrained Mineral Resource. The parameters used to generate the MSO wireframes are shown in Table 14-13.

Table 14-13: Lithium Underground Limit Analysis Parameters (Source: Nordmin, 2023)

Parameter	Value
Currency Used for Evaluation	CAD$
Block Size	In-Situ sub-blocked model 5.0 m (x) by 5.0 m (y) by 5.0 m (z)
Mining Method	Selective shallow dip mining (e.g., cut and fill)
MSO Geometry	5.0 m (x) by 5.0 m (y) by 5.0 m (z) Manual deletion of isolated shapes
Underground Mining Cost	$\$80.00/t_{processed}$
Process Cost *Includes assumptions for Milling, G&A*	$\$50.00/t_{processed}$
Concentrate Transportation / Insurance	$\$175.00/t_{concentrate}$
Spodumene Concentrate Grade	5.5% Li_2O
Spodumene Concentrate Price	$1,800 USD per tonne spodumene concentrate Exchange Rate: 1 USD$=1.30 CAD$ $2,340 CAD per tonne spodumene concentrate
Process Recovery	75%
Production Rate Assumption	1,200 t/d

The Underground Mineral Resource CoG is estimated to be 2,000 ppm lithium. This CoG is calculated to cover the Underground Mining, Process and selling costs using the parameters listed in Table 14-11. For resource cut-off calculation purposes, a mining recovery of 100% and mining dilution of 0% was applied. All material within MSO wireframes has been included in the Underground Mineral Resource Estimate to include must take material.

14.12.3 Rubidium

The rubidium open pit and underground resource estimate was constrained above market value due to the current limited world market. This 4,000 ppm rubidium cut-off grade was selected for both open pit and underground as shown in Table 14-14. The open pit rubidium resource was constrained using the lithium value optimized open pit shell (RF 1.00). The rubidium resource was excluded from (neither taken into account nor used as a credit for) the underground and open pit lithium resource.

Table 14-14: Rubidium Open Pit and Underground Limit Analysis Parameters (Source: Nordmin, 2023)

Parameter	Value
Currency Used for Evaluation	CAD$
Open Pit Block Size	In-Situ model regularized to 5.0 m (x) by 5.0 m (y) by 5.0 m (z)
Open Pit Constraint	Lithium value optimized open pit shell (RF 1.00)
Underground Block Size	In-Situ sub-blocked model 5.0 m (x) by 5.0 m (y) by 5.0 m (z)
Underground Mining Method	Selective shallow dip mining (e.g., cut and fill)
MSO Geometry	5.0 x 5.0 x 5.0 m (X by Y by Z) Manual deletion of isolated shapes
Cut-off Grade *Open Pit & Underground*	4,000 ppm

For reference the market price of rubidium carbonate (Rb_2CO_3≥99%) in February 2023 is approximately USD 1,160 per kg. Using this market price and the assumptions stated in Table 14-15 this would result in rubidium open pit cut-off grade of approximately 160 ppm and rubidium underground cut-off grade of approximately 210 ppm.

Table 14-15: Rubidium Market Value Cut-off Grade Parameters (Not Selected for MRE Cut-off; Source: Nordmin, 2023)

Parameter	Open Pit	Underground
Currency Used for Evaluation	CAD$	CAD$
Mining Cost	Break even COG only including G&A, processing, and selling costs	$80.00/$t_{processed}$
G&A Cost	$16.00/$t_{processed}$	$25.00/$t_{processed}$
Offsite Transportation / Insurance	$175.00/$t_{processed}$	$175.00/$t_{processed}$
Offsite Processing	$100.00/$t_{processed}$	$100.00/$t_{processed}$
Rubidium Carbonate Price	$1,160 USD per kg rubidium carbonate (Rb_2CO_3≥99%) Exchange Rate: 1 USD$=1.30 CAD$ $1,508 CAD per kg rubidium carbonate (Rb_2CO_3≥99%)	$1,160 USD per kg rubidium carbonate (Rb_2CO_3≥99%) Exchange Rate: 1 USD$=1.30 CAD$ $1,508 CAD per kg rubidium carbonate (Rb_2CO_3≥99%)
Process Recovery	90%	90%
Market Value Cut-off Grade *Not selected for MRE cut-off*	160 ppm	210 ppm

14.13 Mineral Resource Estimate

14.13.1 Lithium Mineral Resource Estimate

The Lithium Mineral Resources were classified using the 2014 CIM Definition Standards and the 2019 CIM Best Practice Guidelines and have an effective date of February 16, 2023. The Project hosts:

- Total Lithium Open Pit (at a 650 ppm lithium cut-off) and Underground (at a 2,000 ppm lithium cut-off) Mineral Resources include 2,293 thousand tonnes of Measured and Indicated Mineral Resources grading 2,976 ppm lithium, and 3,902 thousand tonnes of Inferred Resources grading 2,691 ppm lithium. The Lithium Mineral Resources are summarized in Table 14-16 and shown in Figure 14-32.
- Open Pit Lithium Mineral Resources include 2,101 thousand tonnes of Measured and Indicated Mineral Resources grading 2,956 ppm lithium, and 3,247 thousand tonnes of Inferred Resources grading 2,595 ppm lithium. The Open Pit Lithium Mineral Resources are summarized in Table 14-16 and shown in Figure 14-33.
- Underground Lithium Mineral Resources include 192 thousand tonnes of Measured and Indicated Mineral Resources grading 3,192 ppm lithium, and 655 thousand tonnes of Inferred Resources grading 3,162 ppm lithium. The Underground Lithium Mineral Resources are summarized in Table 14-16 and shown in Figure 14-34.

Table 14-16: Lithium Open Pit and Underground Mineral Resource Estimate (Source: Nordmin, 2023)

Area	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
Open Pit *650ppm Li Cut-off*	Measured	80	3,887	0.84%	313
	Indicated	2,021	2,919	0.63%	5,897
	Measured + Indicated	2,101	2,956	0.64%	6,210
	Inferred	3,247	2,595	0.56%	8,427
Underground *2,000ppm Li Cut-off*	Measured	3	2,560	0.55%	8
	Indicated	189	3,203	0.69%	606
	Measured + Indicated	192	3,192	0.69%	614
	Inferred	655	3,162	0.68%	2,073
Total	**Measured + Indicated**	**2,293**	**2,976**	**0.64%**	**6,824**
	Inferred	**3,902**	**2,691**	**0.58%**	**10,499**

Mineral Resource Estimate Notes

- Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
- The MRE is developed with data from diamond drill holes totaling 13,821 m.

- The pit constrained mineral resources were defined using a parented block model, within an optimized pit shell with average pit slope angles of 45° in rock and 30° in overburden, a 9.8 strip ratio (waste material: mineralized material) and a revenue factor of 1.0. The pit optimization shells were created using Deswik.AdvOPM software.
- The lithium resource pit optimization parameters include: 5.5% Li_2O spodumene concentrate; US$1,800 Li_2O spodumene concentrate price; exchange rate of C$1.3/US$1; concentrate transportation and offsite charges of C$175/t, mining cost of C$6/t, processing plus general and administration cost of C$41/t; and a process recovery of 75%. Only lithium value was used to generate the resource optimized pit shell.
- Underground constrained mineral resources were defined within 5 x 5 x 5 m mineable shape optimization wireframes. The mineable shape optimization constraining wireframes were created using Deswik.SO software.
- The lithium resource underground mineable shape optimization parameters include: 5.5% Li_2O spodumene concentrate; US$1,800 Li_2O spodumene concentrate price; exchange rate of C$1.3/US$1; concentrate transportation and offsite charges of C$175/t, mining cost of C$80/t, processing plus general and administration cost of C$50/t; and a process recovery of 75%.
- A default density of 2.668 g/cm^3 was used for the mineralized zones.
- All figures are rounded to reflect the relative accuracy of the estimates; totals may not add correctly.

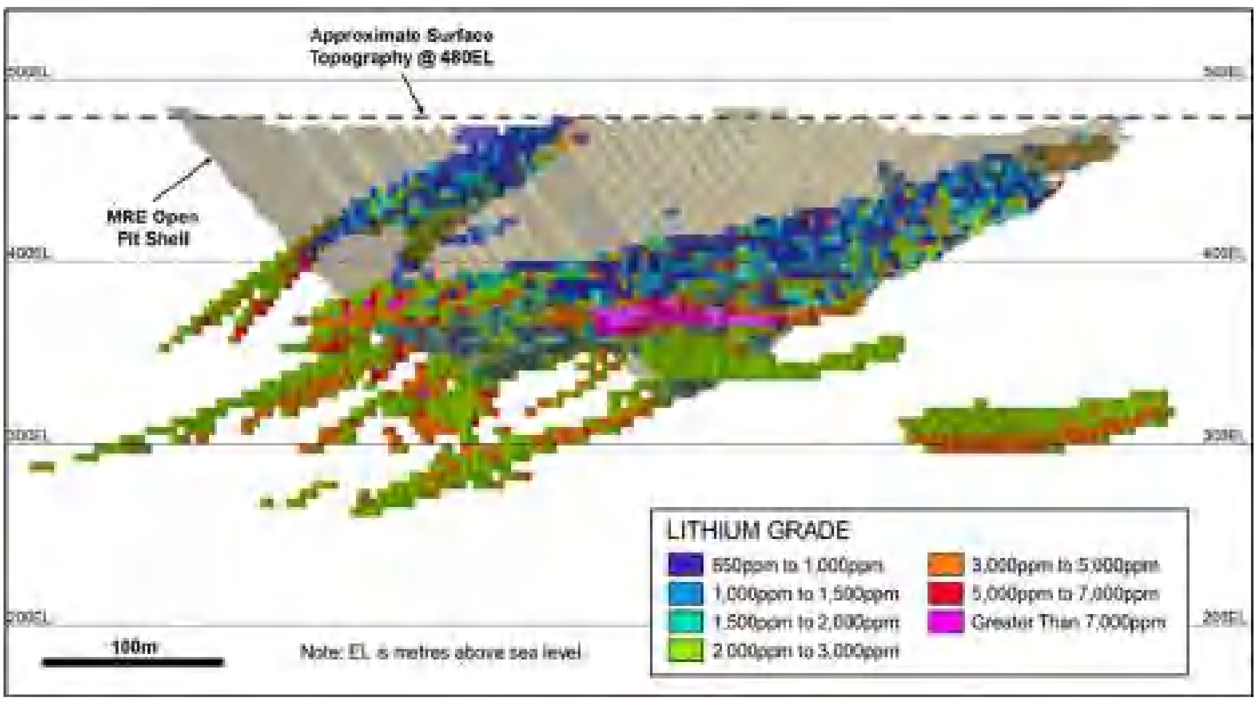


Figure 14-32: Lithium MRE Isometric Section View Looking Southwest with Lithium Grades (Source: Nordmin, 2023)

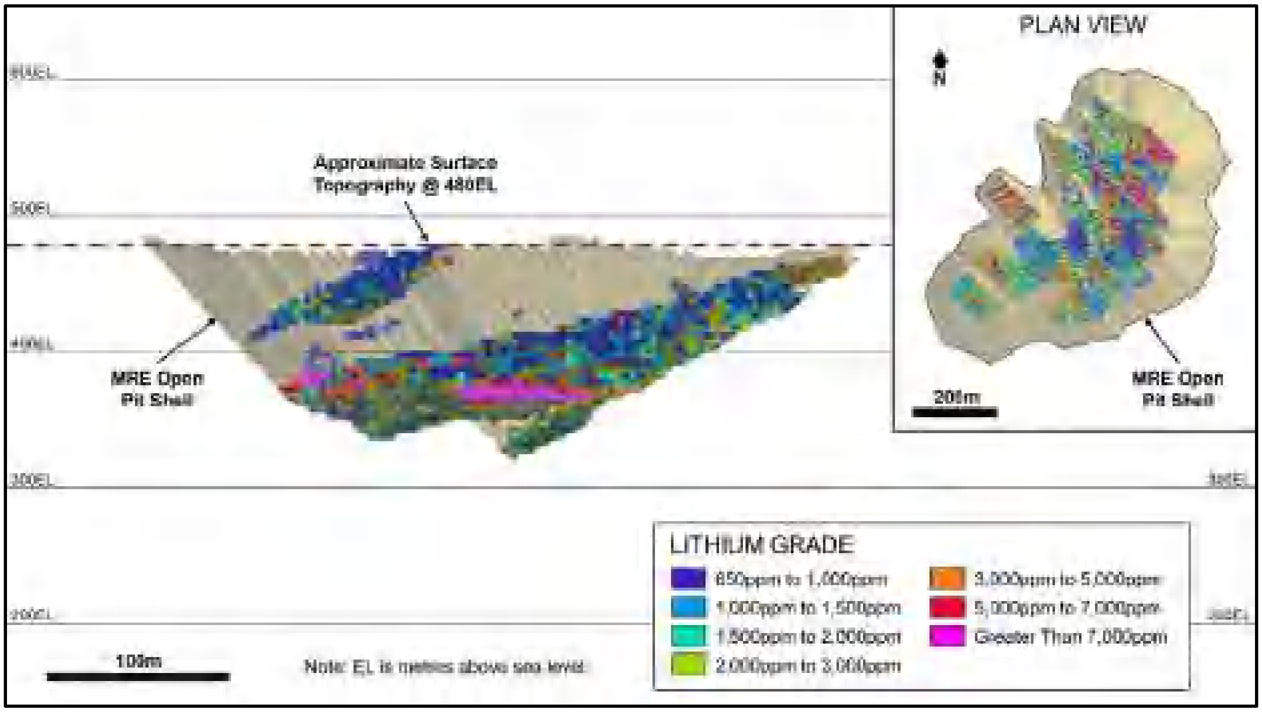


Figure 14-33: Open Pit Lithium MRE Isometric Section View Looking Southwest with Lithium Grades (Source: Nordmin, 2023)

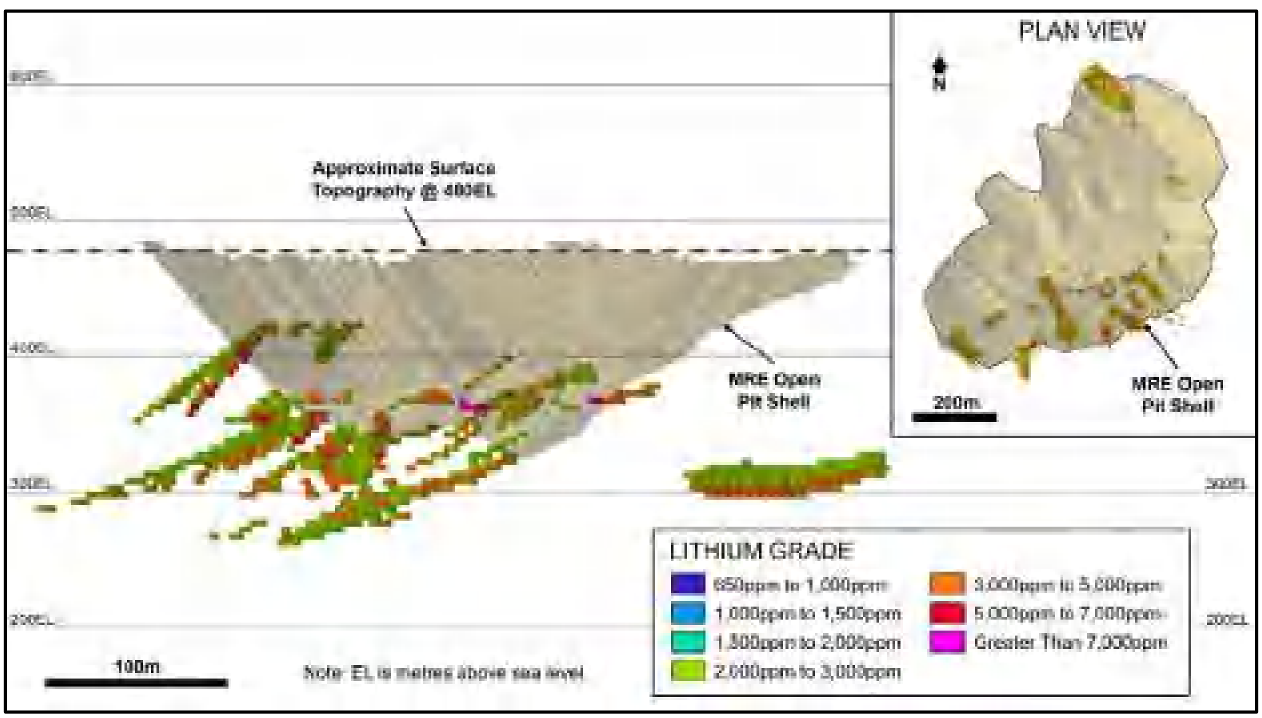


Figure 14-34: Underground Lithium MRE Isometric Section View Looking Southwest with Lithium Grades (Source: Nordmin, 2023)

14.13.2 Rubidium Mineral Resource Estimate

The Rubidium Mineral Resources were classified using the 2014 CIM Definition Standards and the 2019 CIM Best Practice Guidelines and have an effective date of February 16, 2023. The Project hosts:

- Total Rubidium Open Pit (at a 4,000 ppm rubidium cut-off) and Underground (at a 4,000 ppm rubidium cut-off) Mineral Resources include 133 thousand tonnes of Measured and Indicated Mineral Resources grading 6,163 ppm rubidium, and 123 thousand tonnes of Inferred Resources grading 4,224 ppm rubidium. The Rubidium Mineral Resources are summarized in Table 14-17 and shown in Figure 14-35.
- Open Pit Rubidium Mineral Resources include 95 thousand tonnes of Measured and Indicated Mineral Resources grading 6,036 ppm rubidium, and 18 thousand tonnes of Inferred Resources grading 3,005 ppm rubidium. The Open Pit Rubidium Mineral Resources are summarized in Table 14-17 and shown in Figure 14-36.
- Underground Rubidium Mineral Resources include 38 thousand tonnes of Measured and Indicated Mineral Resources grading 6,484 ppm rubidium, and 106 thousand tonnes of Inferred Resources grading 4,427 ppm rubidium. The Underground Rubidium Mineral Resources are summarized in Table 14-17 and shown in Figure 14-37.

Table 14-17: Rubidium Open Pit and Underground Mineral Resource Estimate, (Source: Nordmin, 2023)

Area	Resource Category	Mass (kt)	Grade		Contained Rb (t)
			Rb (ppm)	Rb_2O (%)	
Open Pit *4,000ppm Rb Cut-off*	Measured	5	5,412	0.59%	29
	Indicated	90	6,073	0.66%	547
	Measured + Indicated	95	6,036	0.66%	576
	Inferred	18	3,005	0.33%	53
Underground *4,000ppm Rb Cut-off*	Measured	5	6,547	0.72%	35
	Indicated	33	6,474	0.71%	211
	Measured + Indicated	38	6,484	0.71%	246
	Inferred	106	4,427	0.48%	468
Total	**Measured + Indicated**	**133**	**6,163**	**0.67%**	**822**
	Inferred	**123**	**4,224**	**0.46%**	**521**

Mineral Resource Estimate Notes

- Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
- The MRE is developed with data from diamond drill holes totaling 13,821 m.

- The pit constrained mineral resources were defined using a parented block model, within an optimized pit shell with average pit slope angles of 45° in rock and 30° in overburden, and a revenue factor of 1.0. The constraining optimized pit shell is based on lithium value only. The pit optimization shells were created using Deswik.AdvOPM software.
- Underground constrained mineral resources were defined within 5.0 x 5.0 x 5.0 m mineable shape optimization wireframes. The mineable shape optimization constraining wireframes were created using Deswik.SO software.
- The rubidium open pit and underground resource estimate was constrained above market value due to the current limited world market. A 4,000 ppm rubidium cut-off grade was selected. The rubidium resource was excluded from (neither taken into account nor used as a credit for) the underground and open pit lithium resource.
- A default density of 2.668 g/cm^3 was used for the mineralized zones.
- All figures are rounded to reflect the relative accuracy of the estimates; totals may not add correctly.

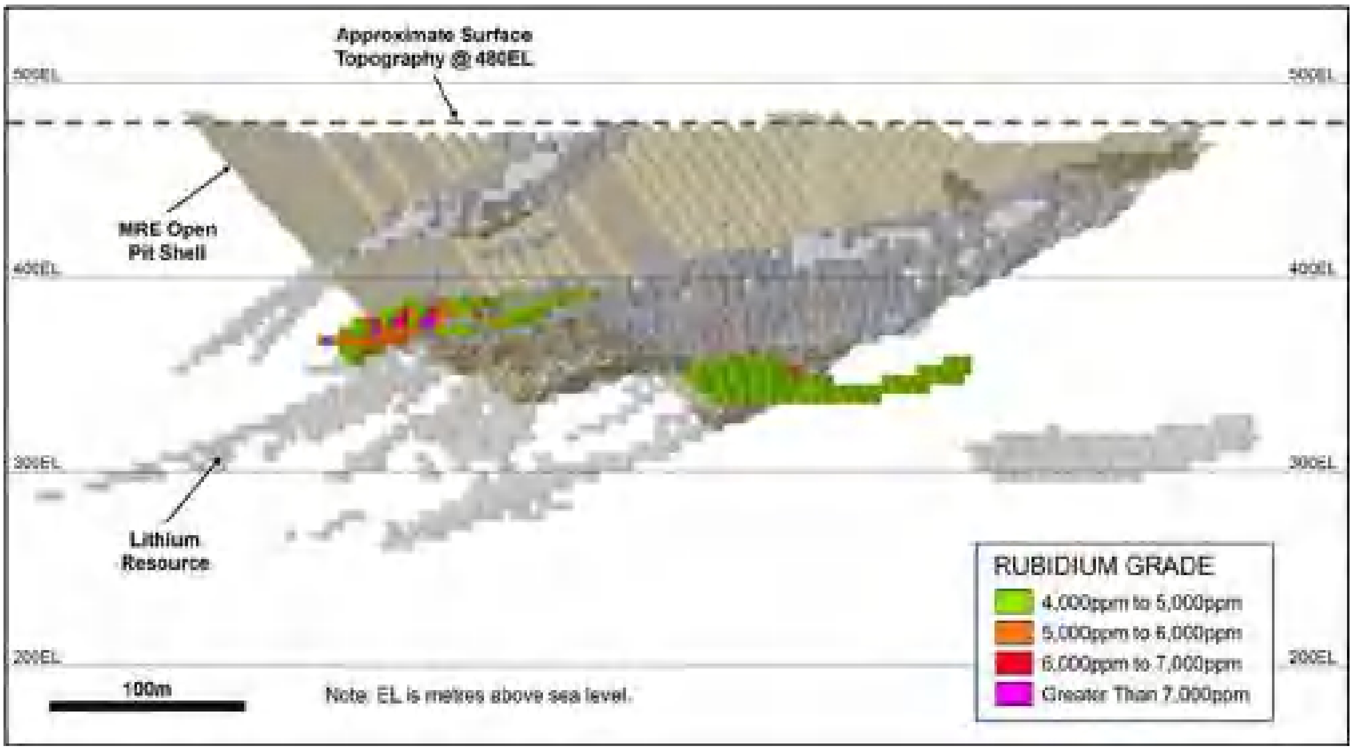


Figure 14-35: Rubidium MRE Isometric Section View Looking Southwest with Rubidium Grades (Source: Nordmin, 2023)

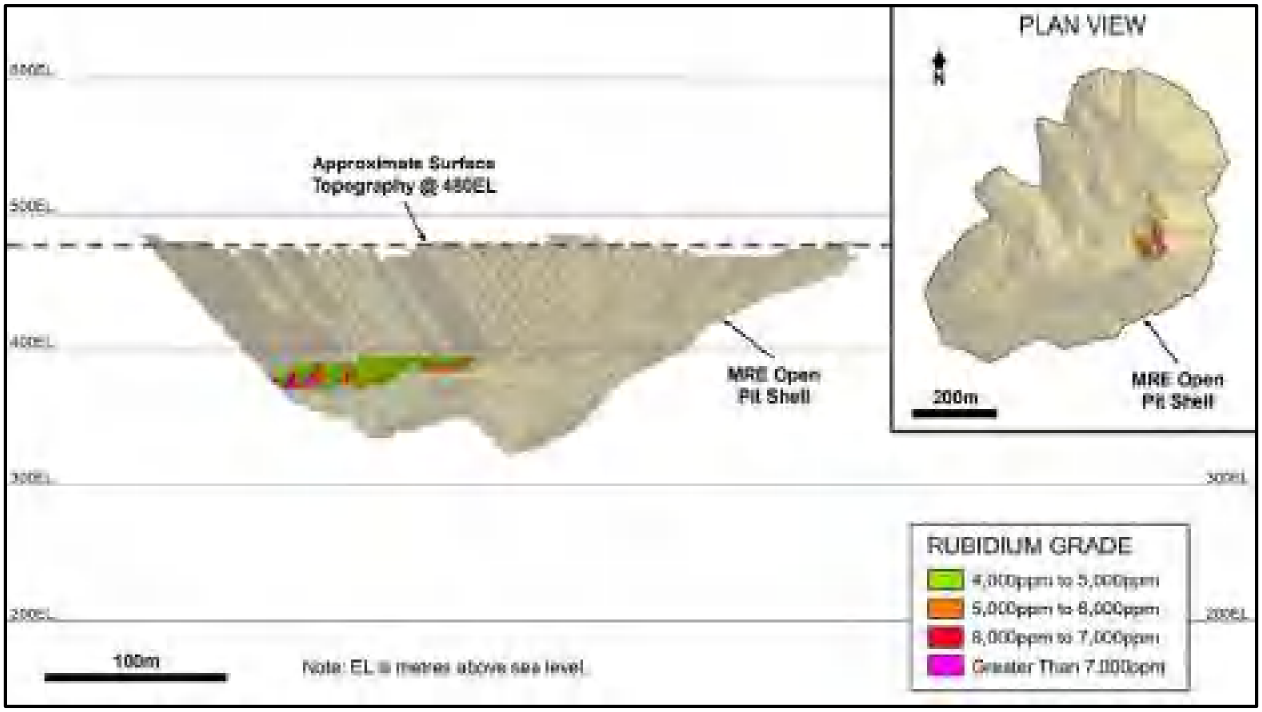


Figure 14-36: Open Pit Rubidium MRE Isometric Section View Looking Southwest with Rubidium Grades (Source: Nordmin, 2023)

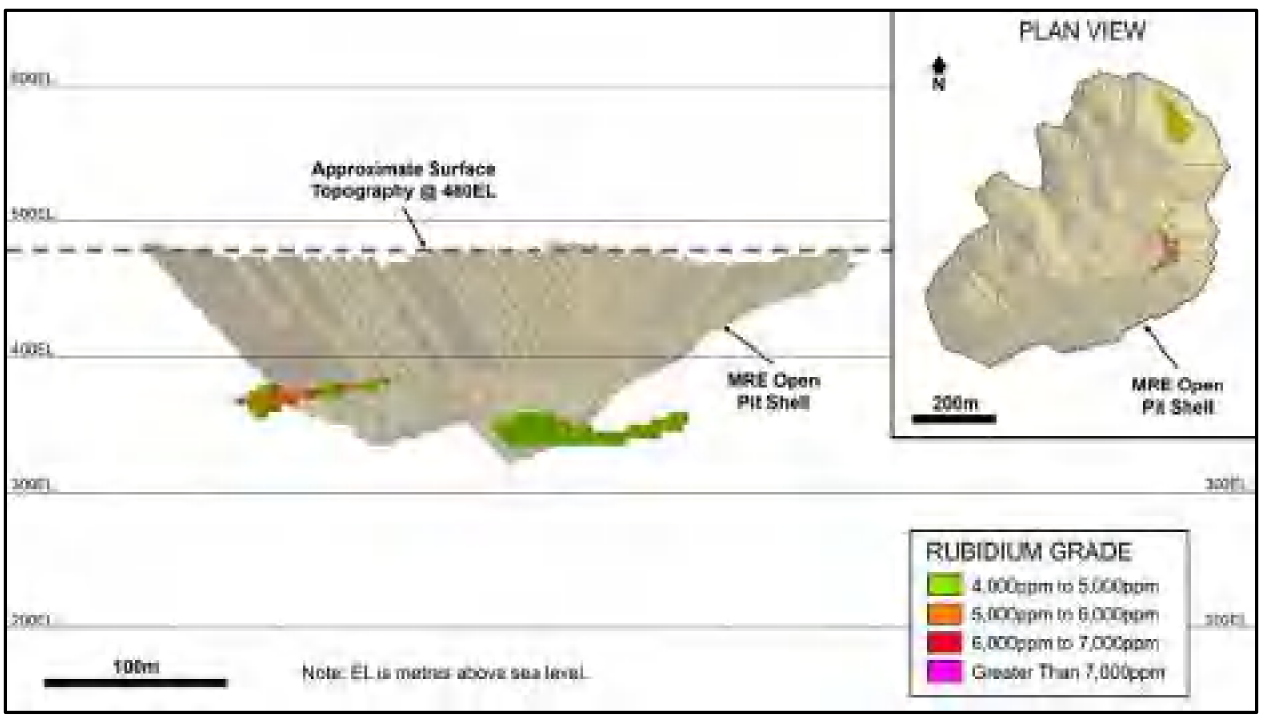


Figure 14-37: Underground Rubidium MRE Isometric Section View Looking Southwest with Rubidium Grades (Source: Nordmin, 2023)

14.13.3 Cautionary Statement Regarding Mineral Resource Estimates

Until mineral deposits are mined and processed, Mineral Resources must be considered as estimates only. Mineral Resource Estimates that are not Mineral Reserves do not have demonstrated economic viability. The estimation of Mineral Resources is inherently uncertain, involves subjective judgment about many relevant factors and may be materially affected by, among other things, environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant risks, uncertainties, contingencies, and other factors described in the foregoing Cautionary Statements. The quantity and grade of reported “Inferred” Mineral Resource Estimates are uncertain in nature and there has been insufficient exploration to define “Inferred” Mineral Resource Estimates as an “Indicated” or “Measured” Mineral Resource and it is uncertain if further exploration will result in upgrading “Inferred” Mineral Resource Estimates to an “Indicated” or “Measured” Mineral Resource category. The accuracy of any Mineral Reserve and Mineral Resource Estimates is a function of the quantity and quality of available data, and of the assumptions made and judgments used in engineering and geological interpretation, which may prove to be unreliable and depend, to a certain extent, upon the analysis of drilling results and statistical inferences that may ultimately prove to be inaccurate. Mineral Reserve and Mineral Resource Estimates may have to be re-estimated based on, among other things: (i) fluctuations in mineral prices; (ii) results of drilling, and development; (iii) results of test stoping and other testing; (iv) metallurgical testing and other studies; (v) results of geological and structural modelling including stope design; (vi) proposed mining operations, including dilution; (vii) the evaluation of mine plans subsequent to the date of any estimates; and (viii) the possible failure to receive required permits, licences, and other approvals. It cannot be assumed that all or any part of an “inferred,” “Indicated” or “Measured” Mineral Resource Estimate will ever be upgraded to a higher category. The Mineral Resource Estimates disclosed in this news release were reported using CIM Definition Standards for Mineral Resources and Mineral Reserves in accordance with National Instrument 43-101 of the Canadian Securities Administrators.

14.14 Mineral Resource Sensitivity to Reporting Cut-off

14.14.1 Lithium Mineral Resource Sensitivity to Reporting Cut-off

The sensitivity of the Lithium MRE to a range of CoGs for each category in the open pit optimization shell can be found in Table 14 18 and for underground in Table 14-19.

Table 14-18: Lithium Mineral Resource Sensitivity to Reporting Cut-off, Open Pit (Source: Nordmin, 2023)

Li Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
500	Measured	85	3,705	0.80%	315
	Indicated	2,090	2,841	0.61%	5937
	Inferred	3,501	2,449	0.53%	8573
550	Measured	83	3,796	0.82%	314
	Indicated	2,066	2,867	0.62%	5925
	Inferred	3,418	2,495	0.54%	8530

Li Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
600	Measured	82	3,832	0.82%	314
	Indicated	2,043	2,894	0.62%	5911
	Inferred	3,325	2,549	0.55%	8476
650	Measured	80	3,887	0.84%	313
	Indicated	2,021	2,919	0.63%	5,897
	Inferred	3,247	2,595	0.56%	8,427
700	Measured	78	3,983	0.86%	311
	Indicated	1,996	2,946	0.63%	5881
	Inferred	3,168	2,643	0.57%	8374
750	Measured	76	4,089	0.88%	309
	Indicated	1,969	2,977	0.64%	5861
	Inferred	3,101	2,685	0.58%	8325
800	Measured	73	4,198	0.90%	307
	Indicated	1,932	3,019	0.65%	5832
	Inferred	3,035	2,726	0.59%	8275
900	Measured	71	4,319	0.93%	305
	Indicated	1,850	3,115	0.67%	5763
	Inferred	2,915	2,804	0.60%	8172
1,000	Measured	66	4,563	0.98%	301
	Indicated	1,783	3,196	0.69%	5699
	Inferred	2,705	2,947	0.63%	7973
1,200	Measured	61	4,845	1.04%	295
	Indicated	1,648	3,368	0.73%	5551
	Inferred	2,404	3,179	0.68%	7643
1,500	Measured	54	5,290	1.14%	286
	Indicated	1,487	3,587	0.77%	5336
	Inferred	2,099	3,442	0.74%	7226
2,000	Measured	47	5,797	1.25%	275
	Indicated	1,216	3,995	0.86%	4858
	Inferred	1,629	3,933	0.85%	6408
2,500	Measured	41	6,312	1.36%	262
	Indicated	952	4,483	0.97%	4269
	Inferred	1,294	4,362	0.94%	5646

Li Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
3,000	Measured	36	6,799	1.46%	248
	Indicated	729	5,016	1.08%	3654
	Inferred	1,058	4,725	1.02%	4998

Table 14-19: Lithium Mineral Resource Sensitivity to Reporting Cut-off, Underground (Source: Nordmin, 2023)

Li Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
500	Measured	4	2,242	0.48%	9
	Indicated	1,055	1,447	0.31%	1527
	Inferred	3,484	1,320	0.28%	4600
750	Measured	4	2,326	0.50%	9
	Indicated	701	1,810	0.39%	1268
	Inferred	2,354	1,675	0.36%	3943
1,000	Measured	4	2,417	0.52%	9
	Indicated	472	2,204	0.47%	1041
	Inferred	1,617	2,062	0.44%	3334
1,500	Measured	3	2,456	0.53%	9
	Indicated	267	2,856	0.61%	764
	Inferred	913	2,754	0.59%	2513
2,000	Measured	3	2,560	0.55%	8
	Indicated	189	3,203	0.69%	606
	Inferred	655	3,162	0.68%	2073
2,500	Measured	1	3,012	0.65%	4
	Indicated	125	3,601	0.78%	449
	Inferred	508	3,461	0.75%	1760
3,000	Measured	1	3,558	0.77%	2
	Indicated	77	3,987	0.86%	307
	Inferred	333	3,846	0.83%	1281
3,500	Measured	-	-	-	-
	Indicated	40	4,571	0.98%	184
	Inferred	155	4,625	1.00%	719

Li Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Li (t)
			Li (ppm)	Li_2O (%)	
4,000	Measured	-	-	-	-
	Indicated	21	5,201	1.12%	110
	Inferred	115	4,973	1.07%	574
5,000	Measured	-	-	-	-
	Indicated	9	6,005	1.29%	53
	Inferred	49	5,683	1.22%	278

14.14.2 Rubidium Mineral Resource Sensitivity to Reporting Cut-off

The sensitivity of the rubidium MRE to a range of CoGs for each category in the open pit optimization shell can be found in Table 14-20 and for underground in Table 14-21. Sensitivity results shown in these tables begins at the rubidium MRE 4,000ppm CoG. Rubidium material below this selected CoG generally exists within the lithium MRE and has been excluded from rubidium sensitivities.

Table 14-20: Rubidium Mineral Resource Sensitivity to Reporting Cut-off, Open Pit (Source: Nordmin, 2023)

Rb Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Rb (t)
			Rb (ppm)	Rb_2O (%)	
4,000	Measured	5	5,412	0.59%	29
	Indicated	90	6,073	0.66%	547
	Inferred	18	3,005	0.33%	53
4,500	Measured	4	5,618	0.61%	23
	Indicated	69	6,508	0.71%	449
	Inferred	13	3,341	0.37%	45
5,000	Measured	2	5,947	0.65%	14
	Indicated	51	6,935	0.76%	356
	Inferred	7	3,449	0.38%	25
6,000	Measured	1	7,100	0.78%	6
	Indicated	33	7,569	0.83%	249
	Inferred	2	2,005	0.22%	4
7,000	Measured	0.4	7,600	0.83%	3
	Indicated	15	8,526	0.93%	127
	Inferred	1	2,195	0.24%	2

Table 14-21: Rubidium Mineral Resource Sensitivity to Reporting Cut-off, Underground (Source: Nordmin, 2023)

Rb Cut-off (ppm)	Resource Category	Mass (kt)	Grade		Contained Rb (t)
			Rb (ppm)	Rb_2O (%)	
4,000	Measured	5	6,547	0.72%	35
	Indicated	33	6,474	0.71%	211
	Inferred	106	4,427	0.48%	468
4,500	Measured	4	7,129	0.78%	29
	Indicated	25	6,980	0.76%	173
	Inferred	84	4,535	0.50%	382
5,000	Measured	4	7,147	0.78%	29
	Indicated	19	7,431	0.81%	142
	Inferred	7	2,603	0.28%	18
6,000	Measured	3	8,286	0.91%	23
	Indicated	9	8,348	0.91%	73
	Inferred	1	22	0.00%	0
7,000	Measured	3	8,286	0.91%	23
	Indicated	6	8,841	0.97%	52
	Inferred	1	26	0.00%	0

14.15 Comparison with the Previous Resource Estimate

No previous Mineral Resource Estimates exist for the Project.

14.16 Factors That May Affect the Mineral Resources

Areas of uncertainty that may materially impact the MRE include:

- Changes to long term metal price assumptions.
- Changes to the input values for mining, processing, and G&A costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social licence assumptions.

14.17 Comments on Section 14

The QP is not aware of any environmental, legal, title, taxation, socio-economic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report.

The QP is of the opinion that Mineral Resources were estimated using industry accepted practices and conform to the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Technical and economic parameters and assumptions applied to the MRE are based on Nordmin's internal calculations and feedback from the Company to determine if they were appropriate.

15. MINERAL RESERVE ESTIMATE

This section is not applicable for this report.

16. MINING METHODS

16.1 Introduction

This section outlines the parameters and steps used to conduct the PEA level mine planning work at a proposed plant feed production rate of 540,000 tpy.

The mining method will use tradition load and haul methods using hydraulic excavators, front shovels, and/or wheel loaders as appropriate to the terrain and depending on the major production equipment available for the project. The material will be hauled from the bench to the crusher, ROM stockpiles or waste dump depending on the material type. Furthermore, ancillary equipment, such as bulldozers, graders, and a range of vehicles, is employed to perform functions related to maintenance, support, services, and utilities.

This mining method is proposed to mine 57Mt of material over the mine life, comprised of 4Mt of mill feed and 53Mt of waste with an average strip ratio of 13.2:1.

16.2 General Arrangement

Figure 16-1 provides a general representation of the mine site layout, with the emphasis on showing the position of the Disposal Storage Facility.

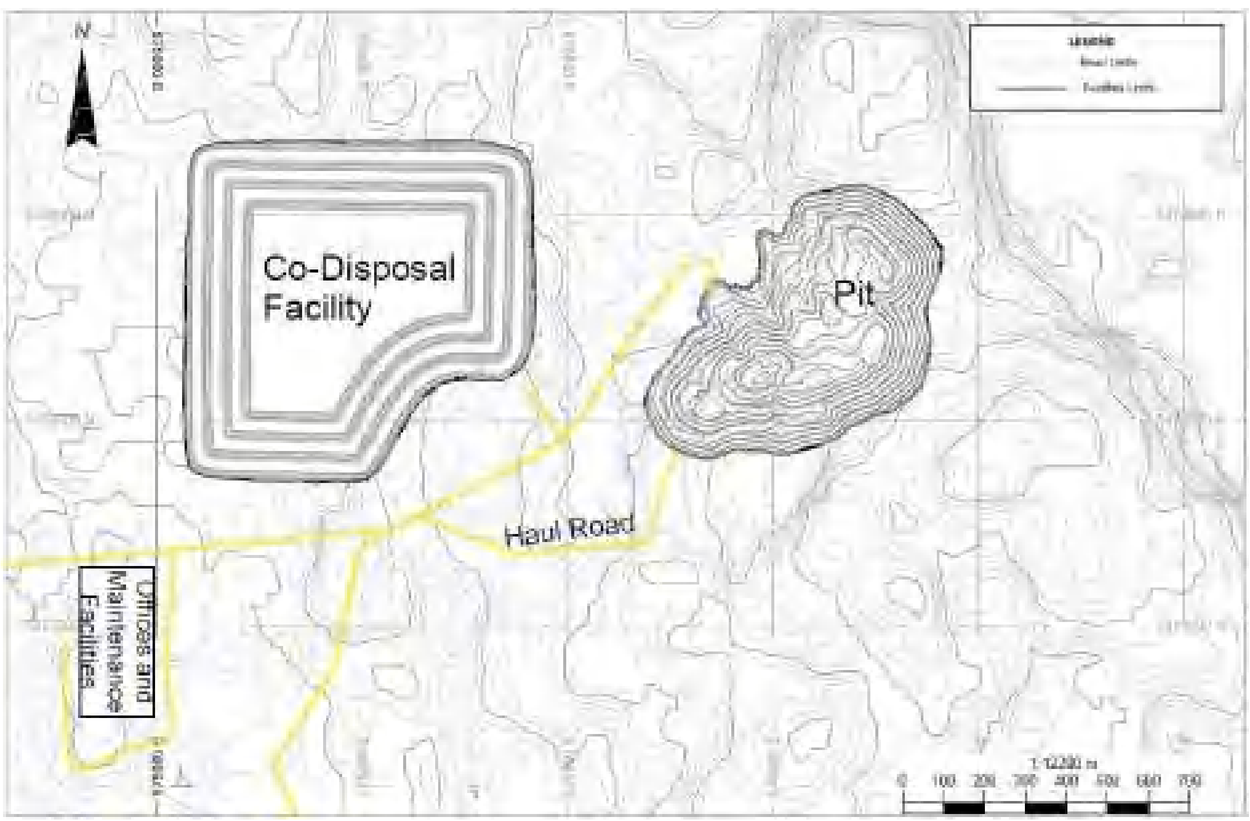


Figure 16-1: General Arrangement

The mine design in the report was conducted using a topographic surface that relies on a 3D Wireframe in Datamine format provided by International Lithium.

The open pit created for the Raleigh Lake deposit covers about 800 metres in length and 450 metres in width at the surface. The pit's lowest point extends to a depth of 330 metres above sea level, while the entrance to the pit is positioned at 475 metres above sea level. The pit incorporates two entrance ramps, with the first granting access to the southern section of the pit and the second facilitating entry to the northern part.

Figure 16-2 provides a plan and sectional view of the pit design.

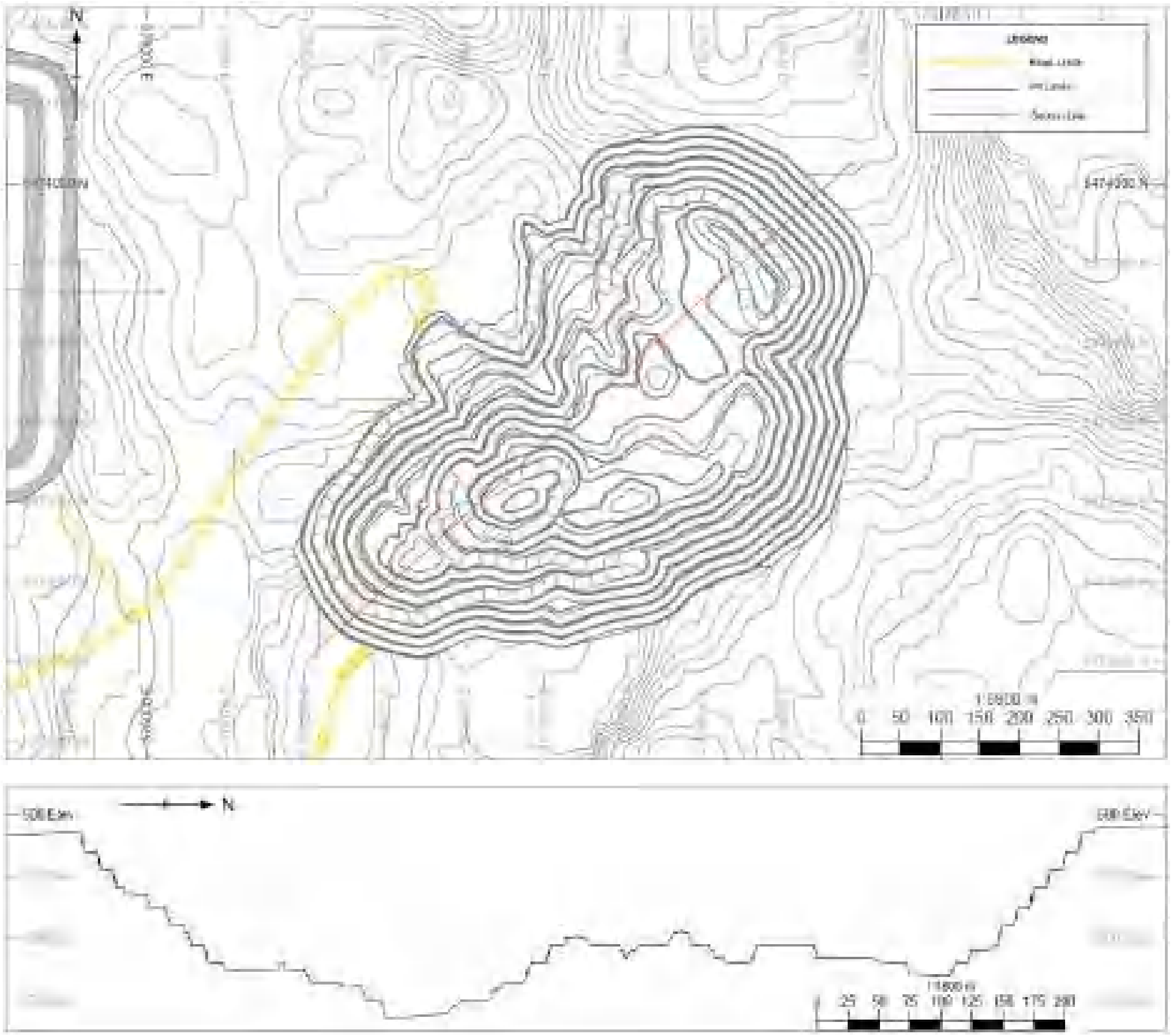


Figure 16-2: Open Pit Design

The approach selected for the storage of tailings generated at the concentrator and the waste rock from the mine will be co-disposal. This co-disposal method involves containing filtered tailings within designated waste rock cells. This approach offers the benefit of enhancing overall stockpile stability and the efficiency of water drainage. The primary goal is to guarantee long-term physical and geochemical stability.

Figure 16-3 illustrates a standard representation of the co-disposal concept's cross-section.
The construction of berms will be overseen to contain the tailings within a surrounding of waste rock. In other words, the deposition will be strategically planned to ensure sufficient space within the cells for accommodating future tailings.

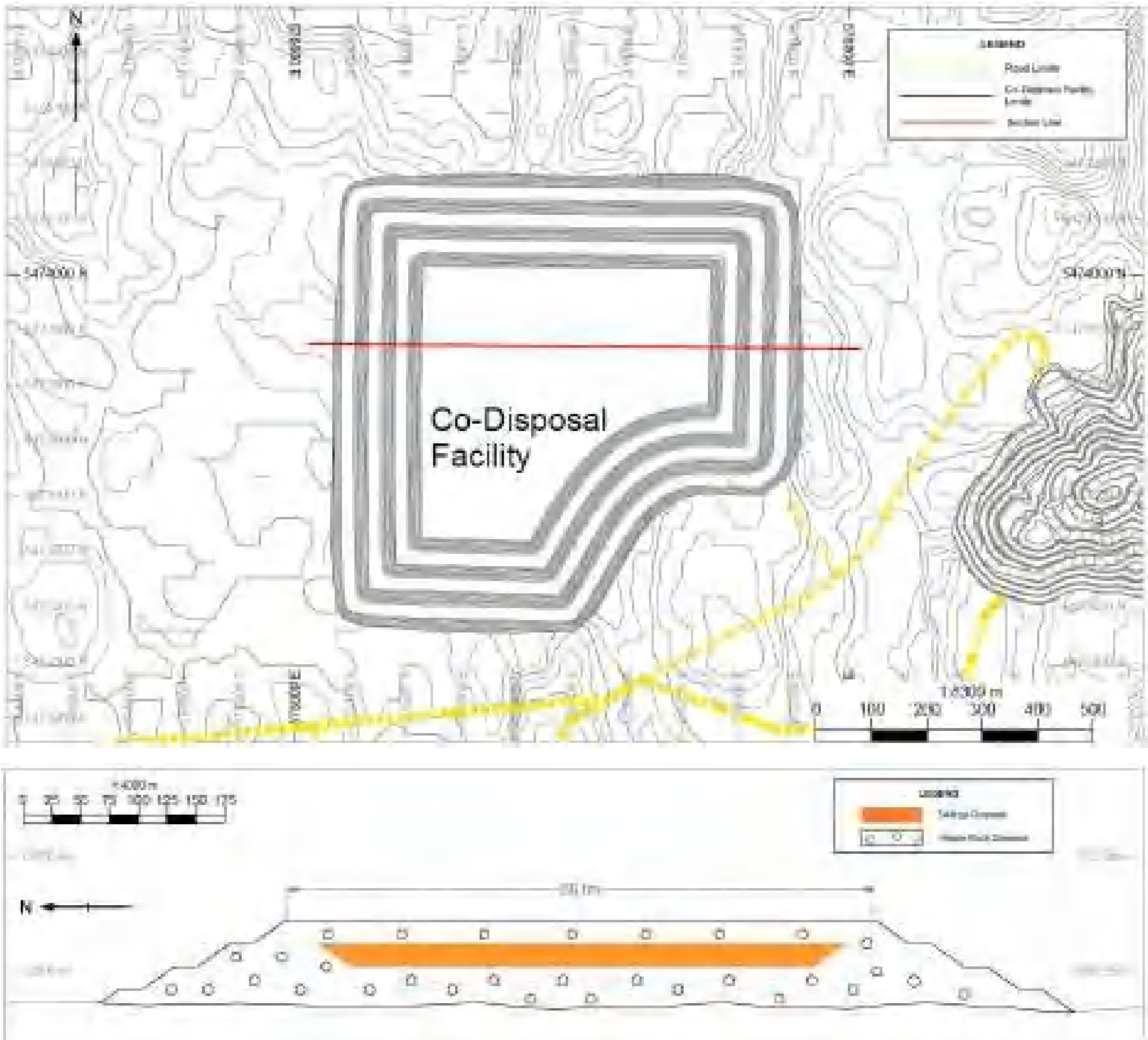


Figure 16-3: Co-disposal Storage Facility

16.3 Open Pit Mine Plan

The Mine Plan is based on the Open Pit Optimization, which utilized the economic parameters detailed in Table 16-1, the Revenue Parameters specified in section 16-4, and various geotechnical and operational assumptions, including dilution and mining recovery.

Table 16-1: Pit Optimization Parameters

Parameters	ERM Value	Unit
Sales Revenue		
Exchange Ratio	1.3	CAD$/USD$
Concentrate Price (Li_2O)	1,500	USD$/t
Concentrate Price (Li_2O)	1,950	CAD$/t
Operating Costs		
Mill Feed Mining Cost	5.00	CAD$/t
Waste Rock Mining Cost	3.50	CAD$/t
Additional Cost per bench (5 m)	0.02	CAD$/t
Total Processing Cost and G&A	46.00	CAD$/t
Manpower	10.16	CAD$/t
Reagents & Maintenance	4.03	CAD$/t
Elec Power	2.75	CAD$/t
G&A Sales	7.16	CAD$/t
Contingency	0.90	CAD$/t
G&A	21.00	CAD$/t
Total Offsite Costs	75.00	CAD$/t concentrate
Truck Loading Costs	5.00	CAD$/t concentrate
Transportation Costs	60.00	CAD$/t concentrate
Misc Offsite	10.00	CAD$/t concentrate
Metallurgy		
Concentration Recovery	81	%
Li_2O Concentrate Grade	6	%
Geotechnical Parameters		
Overall Angle	45	degrees
Material Density		
Mineralized Material	-	Pulled from model
Default Density	2.973	t/m^3
Pit Optimization Parameters		
Dilution	5	%
Mining Recovery	95	%
Mill Throughput Rate	540,000	tpy

The optimization process was conducted using the Datamine Studio NPVS software, employing a mill throughput rate of 540,000 tpy to calculate a reference NPV for each nested pit. A range of revenue factors from 1% to 100% was applied, resulting in the generation of a set of nested pits that will serve as the basis for phase selection (Figure 16-4).

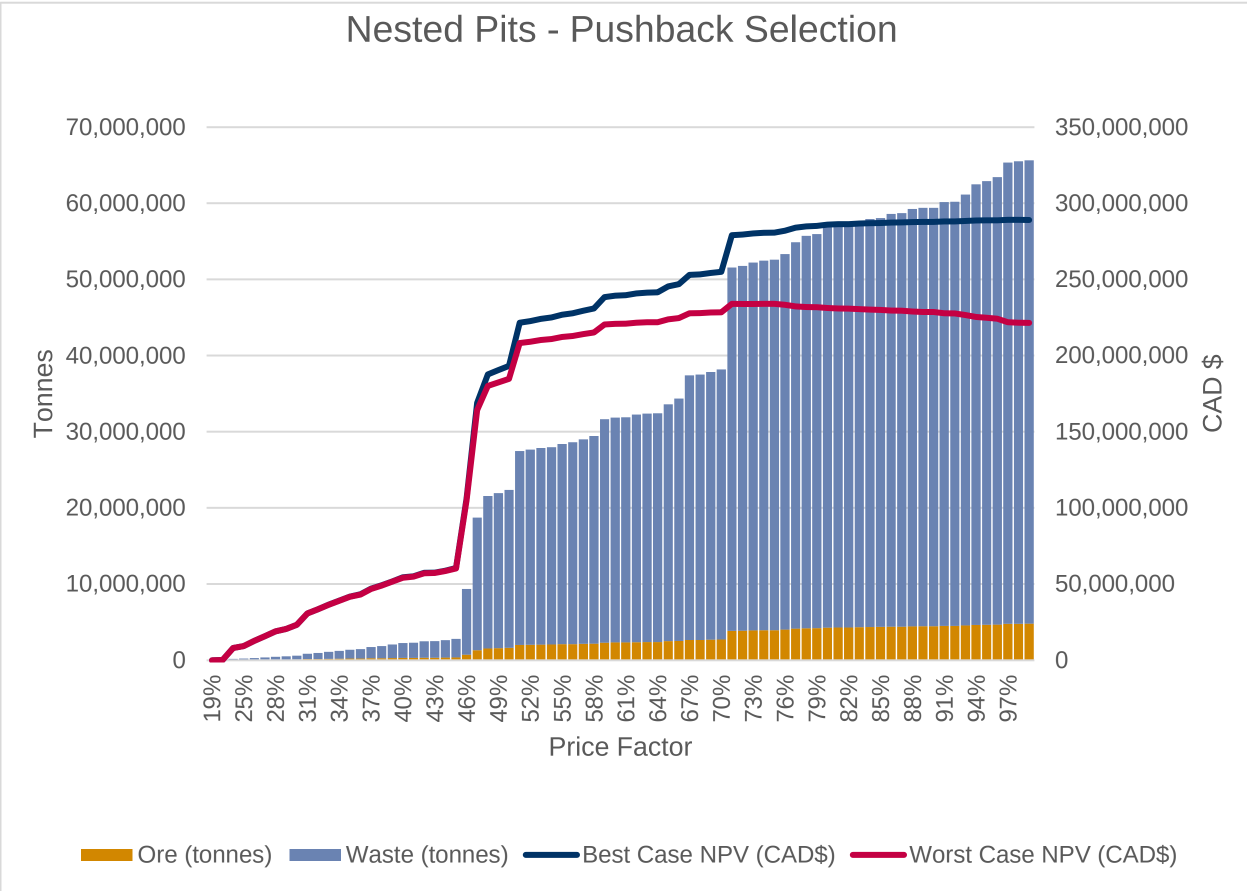


Figure 16-4: Nested Pit Summary

Phases, also referred to as pushbacks, have been chosen to access the ore necessary to meet the annual plant throughput requirements and to manage the sufficient waste removal needed to prevent delays in ore delivery in the coming years. The planning of these phases was guided by the smaller revenue factor pit shells identified in the open pit optimization analysis. The phase layouts adhere to the pit wall configurations discussed in Section 16.7.

The four primary phases begin in the northern section of the ultimate pit, where high-grade ore is located. Subsequent phases progress deeper into the pit and head southward. The ultimate pit was not chosen as the final phase because the Net Present Value (NPV) remains unchanged after reaching a revenue factor of 71%. Therefore, this final phase was implemented as the last pushback. Specific details about these pushbacks are presented in Table 16-2.

Table 16-2: Mineral Resources by Pushback

Push Back	Mill Feed (tonnes)	Waste (tonnes)	Li_2O Grade (%)	Best Case NPV (CAD$)
Incremental				
1	1,410,781	19,033,448	0.88	134,213,967
2	503,055	5,991,488	0.80	38,262,895
3	237,816	3,664,292	0.69	10,317,638
4	2,215,400	24,992,992	0.64	78,196,381
Cumulative				
1	1,410,781	19,033,448	0.88	134,213,967
2	1,913,836	25,024,936	0.86	172,476,862
3	2,151,652	28,689,228	0.84	182,794,500
4	4,367,052	53,682,220	0.74	260,990,882

The selected final pushback is smaller in scale than the ultimate pit, leading to a quicker return on investment and a significant reduction in waste volume. These four pushbacks were utilized to create a mining schedule with an annual mill feed extraction rate of 540,000 tonnes. The initial year is dedicated to waste removal from above the mill feed, and from the second year onward, the plant undergoes a phased ramp-up, reaching its maximum capacity in the third year.

The mine schedule was formulated with an initial focus on maintaining a consistent total rock volume over the first 7 years, followed by a gradual reduction in response to the declining strip ratio. Comprehensive information regarding the mine schedule can be found in section 16.9.

16.4 Revenue Parameters

The projected revenue for this project is anticipated to come from the sale of Lithium Oxide Concentrate, although there is a chance that the excess waste rock could be sold as construction aggregate in the future, and any potential Rubidium mineralised rock might become a byproduct of the concentrate. The current price used in this analysis for the concentrate is $1,500 per tonne (or $1,950 in Canadian dollars).

Table 16-3: Revenue Parameters

Parameters	Value	Unit
Sales Revenue		
Exchange Ratio	1.3	CAD/USD
Concentrate Price (Li_2O)	1500	USD/t
Concentrate Price (Li_2O)	1950	CAD/t

16.5 Geotechnical

In this project, it is important to note that no specific geotechnical studies were conducted. However, to ensure safety and stability, a conservative and standard approach was adopted to determine the overall slope angle of the pit, which has been set at 45 degrees. This approach was employed as a prudent measure to maintain stability and adhere to industry standards.

16.6 Open Pit Mining

The chosen mining approach for the Project is a traditional open-pit operation utilizing truck and shovel techniques, alongside drill and blast procedures. The vegetation, topsoil, and overburden will be removed and stored for future reclamation purposes. The ore and waste rock will be fragmented through drilling and blasting, employing benches that are 15 metres high. Haul trucks will then be loaded with the material using loading equipment on benches that are 5 metres in height.

16.6.1 Overburden Stockpiling

The overburden excavated from the open pit will be deposited in the overburden stockpile, reserved for forthcoming closure and reclamation endeavors. In further studies, the amount of overburden to be extracted will be determine to accurate design a proper stockpile.

16.6.2 Waste Rock and Tailing Co-disposal

The waste rock removed from the open pit will be carried and placed in the co-disposal storage facilities together with the tailings. A dedicated area within the Waste Rock Dump will be specifically planned to contain the tailings. Additional geotechnical investigations will be necessary to ensure the stability of this design.
For additional information on these co-disposal storage facilities, please refer to Section 18 of the Report.

16.6.3 Mine Design

The Raleigh Lake Project pit design employs a triple-bench strategy with 5-metre bench heights, selected to match the mining equipment requirements. Because detailed geotechnical studies were lacking, we used conservative parameters for the conceptual design in this PEA study.

To maintain an overall pit slope angle of 45 degrees, it was recommended to have a minimum berm width of 9 metres with a 70-degree face angle (Table 16-4)

Table 16-4: Design Parameters – Open Pit

Parameters	Value	Unit
Slope Design		
Face Angle	70	Degrees
Bench Height	5.00	Metres
Berm Width	9.50	Metres
Ramp Design		
Ramp Width	15.00	Metres
Max. Ramp Gradient	10	%

The haulage ramps are structured according to the dimensions of the largest haulage truck, accommodating double lane traffic with a buffer space equivalent to half a truck's width. We've incorporated a safety ridge with a height matching a haul truck tire radius and a 2:1 slope for added safety. Additionally, there's a 2-metre-wide ditch to facilitate water drainage and pipe installation.

To ensure safe navigation, all ramp segments have a maximum gradient of 10% on the inner curvature, and we've designed switchbacks with flat rolling surfaces for ease of use.

Based on the optimized pit shell we selected and the geotechnical parameters, we developed a final pit design as shown in Figure 16-5. The process of mine design is iterative and aims to transform the ideal pit shell into a practical open pit mine design. The detailed pit design was generated using Datamine's mining software StudioOP. It incorporates haulage ramp access to all benches, ensuring operational efficiency.

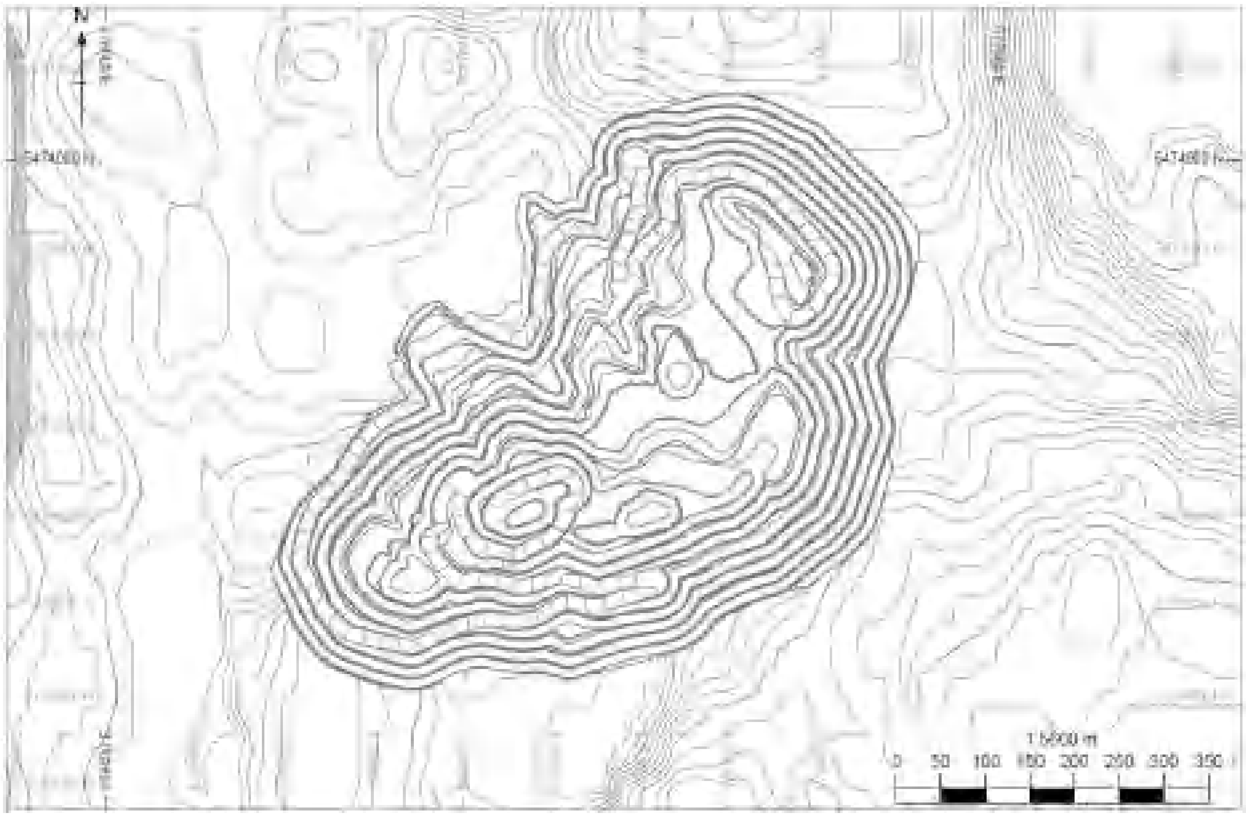


Figure 16-5: Final Pushback Design – Plan View

The design parameters should be reviewed with further geotechnical and hydrogeological studies when more information is available.

16.6.4 Mill Feed Cut-off

The deposit underwent pit optimization following industry best practices. The chosen cut-off for this procedure is determined through engineering and economic analysis, taking into consideration factors and assumptions that can impact the estimation of mineral reserves, including:

- Market prices of the commodity.
- Assumptions regarding operational costs.

- Predicted process plant recovery rates.
- Assumptions concerning mining losses and dilution.

In accordance with the previously presented economic parameters, the mill cut-off employed in the pit optimization has been calculated at 0.20% Li_2O.

16.6.5 Open Pit Mine Schedule

The mine plan was developed using Studio NPVS, and the scheduling was carried out on an annual basis throughout the entire project duration. The primary objective set for the mine planning optimizer was to maximize NPV (Net Present Value).

The preproduction phase is limited to the first two years, with a duration of 16 months. The mining operations will be carried out using their own equipment, ensuring full control over the process. Notably, the plant will operate at only half of its capacity starting from the second year, and in the inaugural year, some preliminary trials will be undertaken using the mill feed extracted during that period. In total, the mining endeavor will yield 4,367,053 tonnes of mill feed, while 53,682,222 tonnes of waste material will be generated.

For the first five years, the annual extraction will remain steady at around 9 million tonnes of rock. However, in the seventh year, the mining ratio will progressively decline, primarily as a result of the extensive removal of waste material during the initial stages of the operation.

The material extraction will commence from the northern part of the site and proceed southward. In the northern region of the pit, the block model reveals zones with high lithium grades. However, it's worthy to note that the highest grade mineralisation is situated at the bottom of the pit, necessitating a significant stripping ratio during the initial years of operation.

Table 16-5 and Figure 16-6 present the mine production schedule. After finalizing the mine plan, it was determined that the mill constraints had been met, and the fleet of mining equipment was considered suitable for the material movement requirements and bench sinking rates.

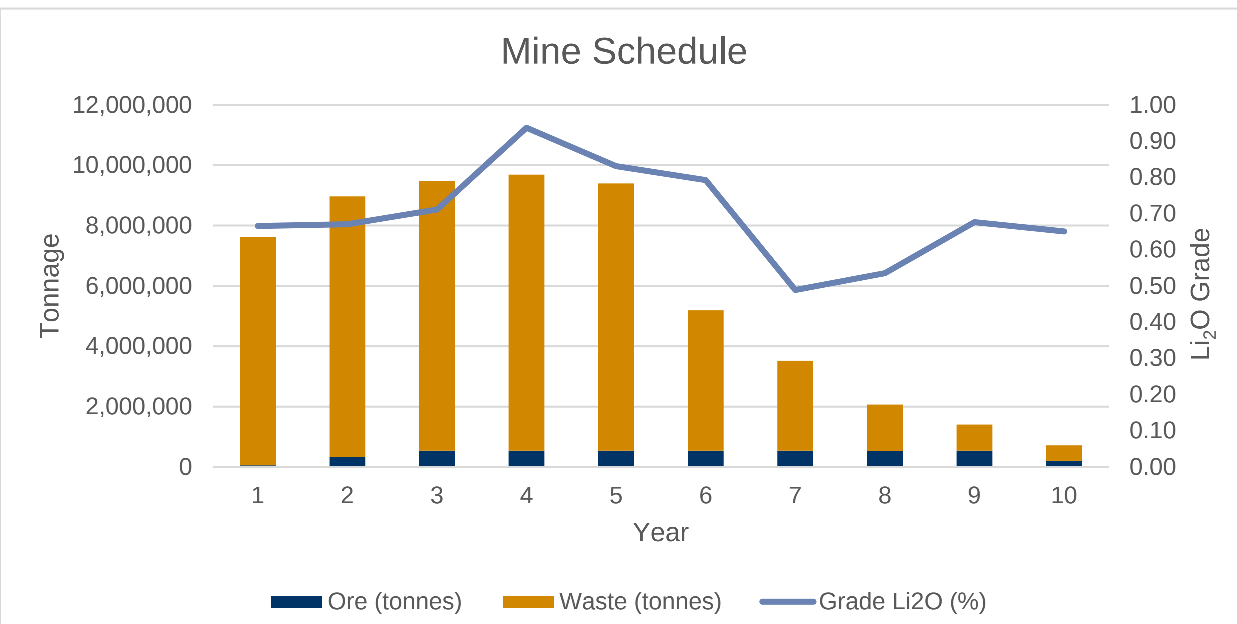


Figure 16-6: Mine Production Schedule

Table 16-5: Mine Production Schedule

Year	-1	1	2	3	4	5	6	7	8	9	Total
Mill feed (tonnes)	**54,037**	**324,183**	**539,881**	**540,107**	**539,899**	**539,895**	**540,305**	**539,713**	**540,312**	**208,721**	**4,367,053**
Measured (tonnes)	0	0	5,566	0	6,338	27,854	0	8,927	23,584	2,725	74,994
Indicated (tonnes)	20,659	112,928	365,731	298,367	265,030	250,939	41,048	152,832	278,257	115,990	1,901,781
Inferred (tonnes)	33,378	211,254	168,584	241,741	268,531	261,103	499,257	377,954	238,471	90,005	2,390,278
Grade Li_2O (%)	**0.67**	**0.67**	**0.71**	**0.94**	**0.83**	**0.79**	**0.49**	**0.54**	**0.68**	**0.65**	**0.70**
Measured (Li_2O%)	0.00	0.00	0.30	0.00	0.63	1.19	0.00	1.13	0.78	0.66	0.92
Indicated (Li_2O%)	0.47	0.51	0.70	0.93	0.67	0.75	0.47	0.63	0.67	0.58	0.70
Inferred (Li_2O%)	0.79	0.75	0.75	0.94	0.99	0.79	0.49	0.48	0.68	0.75	0.70
Waste (tonnes)	**7,572,425**	**8,641,731**	**8,930,422**	**9,147,177**	**8,855,024**	**4,651,476**	**2,979,449**	**1,530,607**	**864,285**	**509,626**	**53,682,222**
Concentrate (tonnes)	0	34,194	51,828	68,321	60,532	57,726	35,668	38,988	49,323	18,324	414,904

16.6.6 Open Pit Mining Equipment

The upcoming section addresses the selection of equipment and the fleet requirements essential for executing the open-pit mining plan. Raleigh Lake will be operated by the mine's own equipment, as detailed in Table 16-6.

Table 16-6: Main Open Pit Equipment

Equipment	Model	Units
Excavator	CAT390	1
Haul Truck	CAT775	3
Rotary Drills	Epiroc	2
Bulldozer	CAT D10	1
Excavator (Spare)	CAT374	1
Haul Truck (Spare)	CAT775	1

The open-pit operations will follow a schedule involving two 12-hour shifts each day, operating seven days a week, and continuing for 50 weeks annually. In the fleet calculations, provision is made for ten days of potential lost mine production attributed to adverse weather conditions.

The Key Performance Indicators (KPIs) estimations employed for the backhoes, trucks, and drills are displayed in Table 16-7.

Table 16-7: Open Pit Equipment KPIs

Equipment	Model	Utilization Factor
Excavator	CAT390	85%
Haul Truck	CAT775	85%
Rotary Drills	Epiroc	85%
Bulldozer	CAT D10	85%
Excavator (Spare)	CAT374	15%
Haul Truck (Spare)	CAT775	15%
Grader	-	50%
Wheel Dozer	-	50%
Front End Loader	-	85%
Boom truck	-	50%
Telehandler	-	50%
Mobile Rock Breaker	-	50%
Mechanics Vehicle	-	50%
Electrician Vehicle	-	50%
Personnel Carrier	-	50%
Supervisor Vehicle	-	50%

Equipment	Model	Utilization Factor
Geo/Eng Vehicle	-	50%
Ambulance	-	50%
Water/Sand Truck	-	50%
Explosive Truck	-	85%

16.6.6.1 Drilling and Blasting

Mill feed and waste rock production drilling will be conducted using down-the-hole track drills powered by diesel, with a hole diameter of 114 mm (4.5 inches). With a penetration rate of 25 metres per hour, each hole is expected to take approximately 15 to 20 minutes to drill, accounting for tasks such as managing drill rods and moving between holes. A contingency redrill ratio was estimated in 10%.

Pre-splitting will be employed for the final pit walls, and its application in temporary walls will depend on the discretion of the on-site geotechnical engineer if deemed necessary.

Bulk emulsion will be the chosen explosive for blasting, and our calculations are based on an explosive density of 1.20 g/cm³. The explosives supplier is responsible for providing and storing the explosives and their accessories, as well as loading the blast holes. Details of the explosive storage facilities will be shown in the section 18. These sites adhere to the minimum distance requirements outlined by Natural Resources Canada Explosives Regulatory Division.

Table 16-8 presents the parameters employed in establishing the drilling and blasting computations.

Table 16-8: Drilling and Blasting Parameters

Description	Units	Mill feed	Waste
Bench Height	m	5.00	5.00
Blasthole Diameter	mm	114	114
Burden	m	3.00	3.00
Spacing	m	4.00	4.00
Sub-Drilling	m	1.20	1.20
Stemming	m	1.50	1.50
Powder Factor	kg/t	0.5	0.3
Re-drilling contingency	%	10	10

16.6.6.2 Haul Trucks

The haul truck selected for the Project is the CAT 775, a rigid frame mining truck with a nominal payload capacity of 76 tonnes and a heaped volume capacity of 42.2 cubic metres. A fleet of five (5) trucks will be required during the first seven (7) years of operation, decreasing the number to three (3) trucks for the final four (4) years. The haul truck fleet will be responsible for transporting mill feed, waste, or overburden. The payload calculations have been adjusted to consider a 2% carry-back for mill feed, waste.

16.6.6.3 Mining Excavators

The main loading equipment chosen for the Project consists of CAT390 mining excavators powered by diesel. These excavators are equipped with 3.9 m^3 buckets and have a maximum payload capacity of 7.0 tonnes.

The calculations derived from the mine plan presented in the Technical Report indicate the need for one (1) excavator at the commencement of operations. Additionally, one (1) spare CAT 374 excavator will be kept on standby in case the primary loading equipment experiences a failure or requires maintenance downtime.

16.6.6.4 Ancillary Equipment

Table 16-9 provides an inventory of ancillary mobile equipment needed to support the Raleigh Lake Mining Project, specifying the equipment categories and the corresponding quantities necessary for project operations.

Table 16-9: Open Pit Ancillary Equipment

Equipment	Units	Utilization Factor
Grader	1	50%
Wheel Dozer	1	50%
Front End Loader	1	85%
Boom truck	1	50%
Telehandler	1	50%
Mobile Rock Breaker	1	50%
Mechanics Vehicle	1	50%
Electrician Vehicle	1	50%
Personnel Carrier	2	50%
Supervisor Vehicle	1	50%
Geo/Eng Vehicle	2	50%
Ambulance	2	50%
Water/Sand Truck	2	50%

16.6.7 Open Pit Mining Labour

The primary workforce for Open Pit Mining will be predominantly composed of operators handling mobile production equipment, as described in Table 16-10. Additionally, there will be a segment of personnel responsible for performing indirect tasks that contribute to supporting the mining production in this project, as indicated in Table 16-11.

Table 16-10: Open Pit Mining Hourly Labour Force

Job	Y - 2	Y - 1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mining											
Excavator Operator	0	4	4	4	4	4	4	4	4	4	4
Haul Truck Driver	0	12	12	12	12	12	12	12	12	12	12
Rotary Drill Operator	0	8	8	8	8	8	8	8	8	8	8
Bulldozer Operator	0	4	4	4	4	4	4	4	4	4	4
Blaster - Lead	0	4	4	4	4	4	4	4	4	4	4
Blaster - Helper	0	4	4	4	4	4	4	4	4	4	4
General Support - Helper	0	4	4	4	4	4	4	4	4	4	4
All Mining Personnel - Company	-	40	40	40	40	40	40	40	40	40	40
Site Support - Company											
Grader Operator	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer Operator	2	2	2	2	2	2	2	2	2	2	2
Front End Loader Operator	4	4	4	4	4	4	4	4	4	4	4
Logistics Crew	4	8	8	8	8	8	8	8	8	8	8
Water/Sand/Snowplow Truck Driver	2	2	2	2	2	2	2	2	2	2	2
All Site Support Personnel - Company	13	17	17	17	17	17	17	17	17	17	17
Mill											
Mill Lead (Spare Supervisor)	0	0	4	4	4	4	4	4	4	4	4
Control Room Operator (L1)	0	0	4	4	4	4	4	4	4	4	4
DMS Operator (L2)	0	0	4	4	4	4	4	4	4	4	4
Product Handling Operator (L2)	0	0	4	4	4	4	4	4	4	4	4
Thickening/Filtration Operator (L3)	0	0	4	4	4	4	4	4	4	4	4
Metallurgical Technician (L3)	0	0	4	4	4	4	4	4	4	4	4
Utility Operator (L4)	0	0	4	4	4	4	4	4	4	4	4
Labourer (L5)	0	0	2	2	2	2	2	2	2	2	2
Crusher Operator (L5)	0	0	4	4	4	4	4	4	4	4	4
All Mill Personnel - Company	0	0	34	34	34	34	34	34	34	34	34
Maintenance											
Tradesman Lead hand	4	4	4	4	4	4	4	4	4	4	4
Electrician Certified	4	4	4	4	4	4	4	4	4	4	4
Electrician Apprentice 4th Yr	2	2	2	2	2	2	2	2	2	2	2
Electrician Apprentice 2nd Yr	0	0	2	2	2	2	2	2	2	2	2
Instrumentation Tech	0	0	2	2	2	2	2	2	2	2	2
Mechanic Certified	8	8	8	8	8	8	8	8	8	8	8

Job	Y - 2	Y - 1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mechanic Apprentice 4th Yr	4	4	4	4	4	4	4	4	4	4	4
Mechanic Apprentice 2nd Yr	0	0	4	4	4	4	4	4	4	4	4
Trades Apprentice 1st Yr	2	2	2	2	2	2	2	2	2	2	2
Millwright Certified	0	0	8	8	8	8	8	8	8	8	8
Welders Certified	4	4	4	4	4	4	4	4	4	4	4
Drill Doctor	2	2	2	2	2	2	2	2	2	2	2
All Maintenance Personnel	30	30	46	46	46	46	46	46	46	46	46
Yard and Warehouse											
Material Controller 1	2	2	2	2	2	2	2	2	2	2	2
Material Controller 2	2	2	2	2	2	2	2	2	2	2	2
Material Expediter	1	1	1	1	1	1	1	1	1	1	1
All Yard and Warehouse Personnel	5	5	5	5	5	5	5	5	5	5	5
Security/First Aid											
Security Officers	4	4	4	4	4	4	4	4	4	4	4
Security/First Aid Officers	4	4	4	4	4	4	4	4	4	4	4
All Security/First Aid Personnel	8	8	8	8	8	8	8	8	8	8	8
Mining Labour Summary (No Supervision)											
Site Support	26	17	17	17	17	17	17	17	17	17	17
Maintenance	30	30	46	46	46	46	46	46	46	46	46
All Hourly Personnel	56	47	63	63	63	63	63	63	63	63	63

Table 16-11: Open Pit Mining Staff Labour Force

Staff Labour	Y - 2	Y - 1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
General Management											
General Manager	1	1	1	1	1	1	1	1	1	1	1
Site Controller	1	1	1	1	1	1	1	1	1	1	1
Administrative Coordinator	1	1	1	1	1	1	1	1	1	1	1
Accountant	1	1	1	1	1	1	1	1	1	1	1
Payroll	1	1	1	1	1	1	1	1	1	1	1
HR Coordinator	1	1	1	1	1	1	1	1	1	1	1
Purchaser	2	2	2	2	2	2	2	2	2	2	2
IT Support	1	1	1	1	1	1	1	1	1	1	1

Staff Labour	Y - 2	Y - 1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine Supervision											
Mine Superintendent		1	1	1	1	1	1	1	1	1	1
Mine General Foreman		2	2	2	2	2	2	2	2	2	2
Shift Boss		8	8	8	8	8	8	8	8	8	8
Mill Supervision / Tech Staff											
Mill Superintendent			1	1	1	1	1	1	1	1	1
Mill General Foreman			2	2	2	2	2	2	2	2	2
Mill Supervisor			4	4	4	4	4	4	4	4	4
Mill Metallurgist			1	1	1	1	1	1	1	1	1
Chief Assayer - Staff			1	1	1	1	1	1	1	1	1
Assayer			2	2	2	2	2	2	2	2	2
Sample Prep			3	3	3	3	3	3	3	3	3
Maintenance Supervision											
Maintenance General Foreman	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	2	2	2	2	2	2	2	2	2	2	2
Electrical Supervisor	2	2	2	2	2	2	2	2	2	2	2
Technical Services											
Manager Technical Services	1	1	1	1	1	1	1	1	1	1	1
Senior Mine Geologist	1	1	1	1	1	1	1	1	1	1	1
Geologist (PGO)	2	2	2	2	2	2	2	2	2	2	2
Regional Geologist	1	1	1	1	1	1	1	1	1	1	1
Geologist	4	4	4	4	4	4	4	4	4	4	4
Core Shack Tech Lead	2	2	2	2	2	2	2	2	2	2	2
Core Shack Tech	2	2	2	2	2	2	2	2	2	2	2
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1
Senior Engineer	1	1	1	1	1	1	1	1	1	1	1
Planning Engineer		2	2	2	2	2	2	2	2	2	2
Mine Technician		2	2	2	2	2	2	2	2	2	2
Surveyor		2	2	2	2	2	2	2	2	2	2
Safety and Training											
Health, Safety and Security Coordinator	1	1	1	1	1	1	1	1	1	1	1
Safety/Training Technician	2	2	2	2	2	2	2	2	2	2	2
Environmental											
Environmental Coordinator	1	1	1	1	1	1	1	1	1	1	1
Environmental Technician	1	1	1	1	1	1	1	1	1	1	1

Table 16-10 provides a comprehensive overview of the labour force allocation for various roles in the Raleigh Lake Mining Project over ten years. It categorizes personnel into Mining, Site Support, Mill, Maintenance, Yard and Warehouse, and Security/First Aid.

Based on the work roster, four (4) operators will be required per piece of primary production mining equipment in operation.

Given that the primary production fleet, in the first year (Y - 2), the Raleigh Lake Mining Project will require a labour hourly force consisting of 47 hourly personnel, with the majority allocated to Mining, Site Support, and Maintenance roles. The Mining sector will employ 17 personnel, Site Support will have 17, and Maintenance will involve 30 personnel. Other sections such as Mill, Yard and Warehouse, and Security/First Aid are not highlighted in this initial year.

Table 16-11 presents a staff labour force breakdown for various roles at the Raleigh Lake Mining Project over a ten-year period. It encompasses General Management, Mine Supervision, Mill Supervision / Tech Staff, Maintenance Supervision, Technical Services, and Safety and Training. Each role is detailed for each year, with the number of personnel required indicated.

In the first year (Y - 2), the labour force at the Raleigh Lake Mining Project totals 57 personnel, with roles primarily concentrated in General Management, Mine Supervision, Mill Supervision / Tech Staff, Maintenance Supervision, Technical Services, and Safety and Training. General Management includes roles like General Manager and Site Controller.

16.7 Underground Mining

16.7.1 Underground Mining Summary

Preliminary analyses indicated that there would not be sufficient mineral resource remaining after considering open pit mining to support underground mining.

The underground mining potential of the current mineral resource was assessed by using MSO (Mineable Stope Optimizer) in Datamine to create rudimentary stope shapes using an appropriate cut-off grade and removing any shapes that fell within a 25 m crown pillar envelope adjacent to the open pit or that did not have sufficient continuity to be economically viable.

If future drilling down dip generates positive results the underground mining scenario should be revisited.

17. RECOVERY METHODS

This section will focus on the recovery methods for spodumene mineral from the lithium mineralisation zone of the Raleigh Lake deposit. The rubidium zone is currently not planned for development and will not be mentioned in this section.

The SGS Lakefield laboratory testing results obtained for the recovery of spodumene from the lithium head mill feed, presented in section 13, provide the basis for the process design and the simplified process flow diagram proposed in this section.

Also presented are the respective design criteria, material balance, equipment selection and sizing for the lithium process plant. The design criteria of the processing plant is presented together with the basic process description. This information provides the basis for the capital and operating cost estimates.

A feasibility study with similar crushing plant design criteria was used as reference1 (ref1):

- Critical Elements Lithium Corporation, Rose Lithium Tantalum Project Feasibility Study Ni-43-101 Technical Report, July 26, 2022, WSP

A feasibility study with similar DMS plant design criteria was used as reference2 (ref2):

- Sigma Lithium, Grota Do Cirlo Lithium Project, Arcuai and Itinga Regions, Minas Gerais, Brazil, Amended and Restated Technical Report, June 12, 2023, SGS, Primero, GE21 Consultoria Mineral.

17.1 Process Design Criteria

The process plant is designed to crush 1,500tpd, then process 1,500tpd in a DMS plant to produce a nominal 56,000 tpy of 6% Li_2O at 81% recovery.

A design factor of 10 % is applied on nominal requirements to ensure that the process equipment has enough capacity to take care of the expected feed variation.

The process plant design is based on testwork performed to date in Section 13, and knowledge acquired in the processing of spodumene mill feeds.

Engineering and design were developed to a scoping level based on the results of the SGS laboratory testing. The SGS lab tests obtained 22.9 weight percentages of 6% lithium concentrate and estimated 81% lithium recovery.

The summary of the proposed process design criteria is presented in the Table 17-1 below:

Table 17-1: Design Operating Parameters

Parameters	Nominal	Designed	Units
General Plant			
Operating Schedule			
Operating hours per day	24	24	h
Annual operating days	365	365	days/y
Shifts/day (Crushing and Wet Plant)	3 x 8	3 x 8	shifts x hrs
Equipment utilization – Crushing Plant	68%	68%	%
Equipment utilization – Wet (DMS) Plant	85%	85%	%

Parameters	Nominal	Designed	Units
Material Characteristics			
Plant Feed			
Feed – Crushing Plant	1,500	1,500	tpd
Feed – Wet (DMS) Plant	1,500	1,500	tpd
Solids %	97%	97%	%
Plant Product			
Concentrate Production	56,000	56,000	tpy
Concentrate Recovery	81%	81%	%
Concentrate Grade (Li_2O)	6%	6%	%

17.2 Process Description

The strong lithium beneficiation performance observed in the HLS testing provided a strong indication that crushing to minus 12.7 mm and screening to 1 mm then using Dense Media Separation (DMS) is a viable process flowsheet for production of 6 % Li_2O spodumene concentrate from the Raleigh Lake Mill feeds with 81% recovery.

17.2.1 General Equipment Selection

The base case for the purposes of the PEA is a standard three stage 1,500 tpd crushing plant and a multi- stage 1,500tpd DMS plant. At 81% recovery, and at the LOM headgrade, it is estimated that the DMS plant will produce approximately 56,000 tpy of 6% Li_2O. Other equipment, to remove mica and magnetic iron has been included to improve the spodumene concentrate quality.

17.2.2 Primary Jaw Crushing (Ref 1)

The primary crushing building houses the mill feed hopper, stationary grizzly, rock breaker, the vibrating grizzly feeder, and the jaw crusher. ROM mill feed will be hauled from the open pit mine. The mine haul trucks dump directly into the mill feed hopper. A stationary grizzly installed on the hopper prevents oversized rocks reporting to the jaw crusher. A rock breaker breaks the oversize boulders. A vibrating grizzly feeder (1.6 m wide x 6.1 m long) extracts the mill feed from the hopper and feeds the oversize to a 224 kW jaw crusher. The undersize, less than (P80) 150 mm in size bypass the jaw crusher. The crushed mill feed and the fines will be conveyed to the secondary crushing building that contains two vibrating screens and the secondary and tertiary cone crushers. A 15-tonne overhead crane installed in the jaw crusher building will be used for maintenance. A dust collector with various pickup points collects dust generated at conveyor discharges and transfer points.

17.3 Secondary and Tertiary Crushing (Ref 1)

The crushed mill feed from the jaw crusher will be screened on the secondary vibrating screen consisting of one 1.8 m wide × 4.80 m long doubledeck screen with top deck screen aperture of 100 mm and the bottom deck screen aperture of 35 mm. The oversize from the two decks will be crushed in the secondary cone crusher of 300 kW producing crushed mill feed, at a P80 of 39 mm. The discharge from the secondary crusher and the screen undersize will be sent to the tertiary doubledeck vibrating screen, 2.4 m wide x 8.5 m long, with top deck screen aperture of 32 mm and bottom deck screen aperture of

19 mm. The tertiary screen will be operated in closed circuit with the tertiary cone crusher. Oversize from the tertiary screen will be crushed in the tertiary cone crusher of 375 kW and will produce crushed mill feed at a P80 of 16 mm. The discharge from this crusher will be sent back to the tertiary double vibrating screen. The undersize from the tertiary vibrating screen will have a P80 of 12.7 mm and will be transported by a conveyor to a crushed mill feed storage dome. The cone crusher building has a 5-tonne overhead crane for maintenance purposes. A dust collector installed in this building collects dust emissions from various conveyor discharge points and transfer tower.

17.4 Crushed Mill Feed Storage Dome (Ref 1)

The crushed mill feed stockpile is covered by a storage dome and is located outside the crushing building, close to the mill. The crushed mill feed storage dome is 42 m diameter x 20 m high and will have a storage capacity of 9,000 tonnes. Variable speed belt feeders are installed under the storage dome. Belt feeders will supply the rated tonnage to the DMS. The feeders discharge the mill feed on the DMS plant feed conveyor. Bin vent type dust collectors control dust emissions from the storage dome and the belt feeders.

Simplified Crushing Plant Flowsheet is presented in Figure 17-1 below:

17.5 DMS Plant

Crushed mill feed in the crushed mill feed storage dome will be conveyed on the DMS plant feed conveyor to a sizing screen to remove the -1.0 mm ultrafines for storage and future processing.

The -12.7 mm / +1.0 mm material will report to the DMS coarse sizing screen where it will be screened at 6.3 mm to produce:

- A coarse fraction (-12.7 mm / +6.3 mm) which reports to the primary coarse DMS.
- A fines fraction (-6.3 mm / +1.0 mm) which reports to the primary fines DMS via a REFLUX™ classifier.

The coarse and fine DMS circuits will consist of primary (SG 2.70) and secondary DMS cyclones (SG 2.85) to efficiently separate spodumene from the gangue material in order to produce a 6.0% Li_2O or higher concentrate grade. Mica will be removed from the fines stream by a REFLUX™ classifier, prior to feeding the DMS fines preparation screen.

Prior to feeding the primary DMS cyclones, each mill feed stream (coarse and fine) will be mixed with ferrosilicon slurry and pumped to the respective coarse and fine primary DMS cyclones. The ferrosilicon slurry density will be carefully controlled to enable the gravity separation of spodumene from minerals with a lower sg. Spodumene has a higher sg than most gangue minerals and consequently the spodumene will report to the DMS cyclone underflow (sinks), with the gangue material reporting to the cyclone overflow (floats).

17.6 Primary DMS Plant Coarse and Fines

The primary DMS circuit will have two sets of DMS cyclones (coarse and fines). They will both share the same SG (2.70) ferrosilicon medium. The floats from the primary coarse DMS cyclones will be sent to tailings, while the underflow streams (sinks) will report to the secondary coarse DMS cyclones.

The primary fines DMS circuit feed will be processed through a REFLUX™ classifier, which aims to remove a portion of the muscovite (mica). This mica stream will be dewatered and report to tailings, while the REFLUX™ classifier underflow will report to the primary fines DMS cyclones. The floats from the primary fines DMS cyclones will be sent to tailings, while the underflow streams (sinks) will report to the secondary fines DMS cyclones.

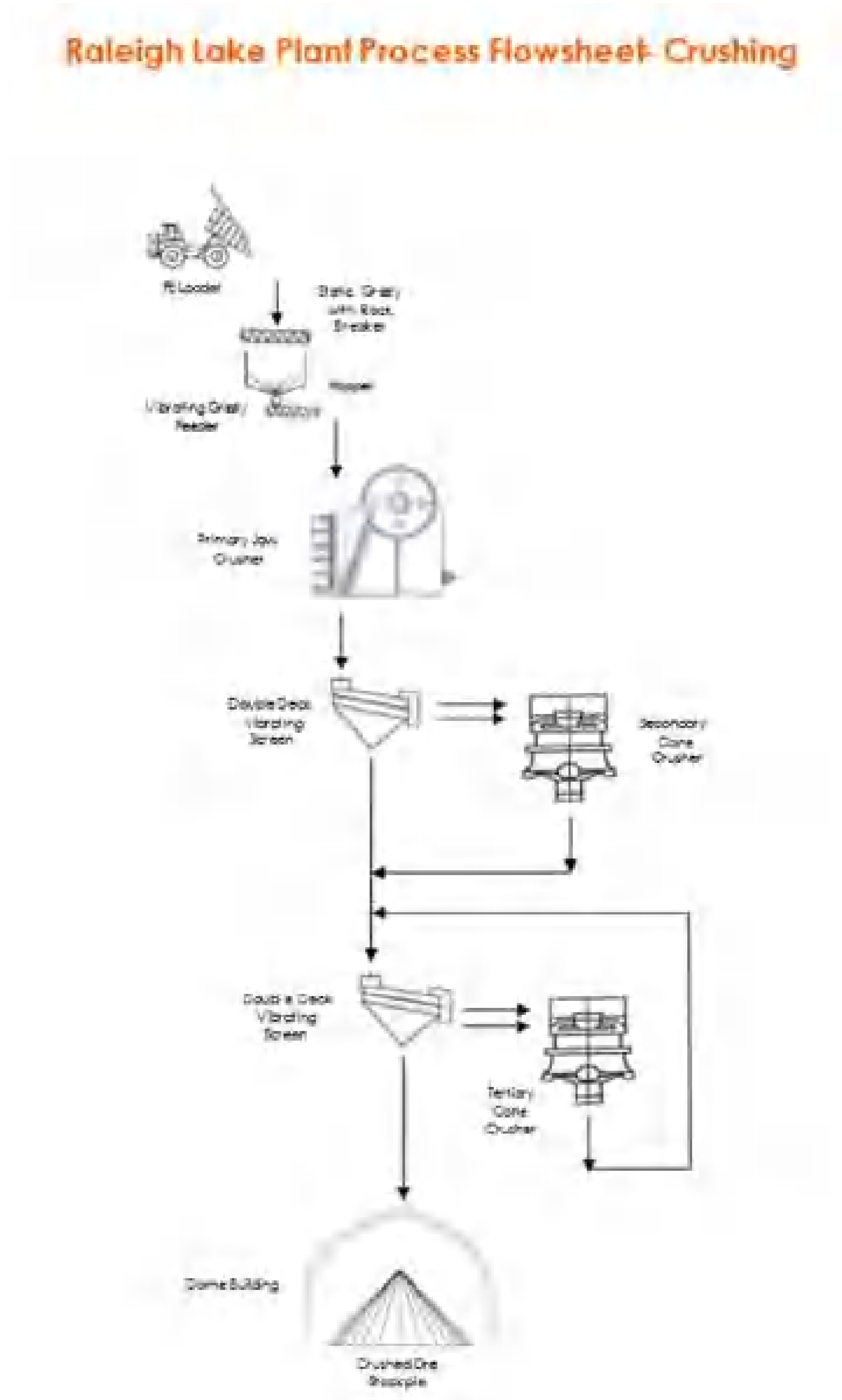


Figure 17-1: Crushing Area Flowsheet

17.7 Secondary DMS Plant Coarse and Fines

The secondary DMS circuit will have two sets of DMS cyclones (coarse and fines DMS cyclones). They will both share the same SG (2.85) ferrosilicon medium.

The floats from the secondary coarse DMS stage will be re-crushed through a rolls crusher and transferred back to the sizing screen. The floats stream from the secondary fines DMS cyclone will report to a waste pile.

The sinks from the secondary coarse DMS cyclones and those from the secondary fines DMS cyclones will be sent to the DMS product stockpile via a magnetic separator for iron removal to meet the product iron content criteria. This will be the final spodumene concentrate product at 6% Li_2O.

Simplified DMS Plant Flowsheet is presented in Figure 17-2 below:

17.8 Ultrafines

The ultrafines (-1.0 mm) from the sizing screen will be stored until a flowsheet is developed to process the ultrafines to obtain additional lithium recovery.

17.9 Tailings Dewater and Store

The tailings will be dewatered and stored. The tailings dewatering equipment may include screw classifiers or cyclones or filtration.

17.10 Plant Utilities

The plant utilities and services area include:

- Fresh water tank that is filled with fresh water for plant use from the nearby lake system.
- Process water tank, that receives the process streams of water for process re-use.
- Plant air compressor and dryer, and instrument air compressor and dryer system for plant air services.
- Oxygen addition system.
- Fire water system.
- Gland water system.

17.11 Plant Buildings

The plant buildings include:

- Crusher building
- Multi-stage DMS plant building
- Solid-liquid tailings separation building (with tailings screw classifier area)
- Main control room and Motor Control Center (MCC) building
- Assay laboratory building
- Plant offices and dry
- Minor spares, materials, and consumables storage buildings.

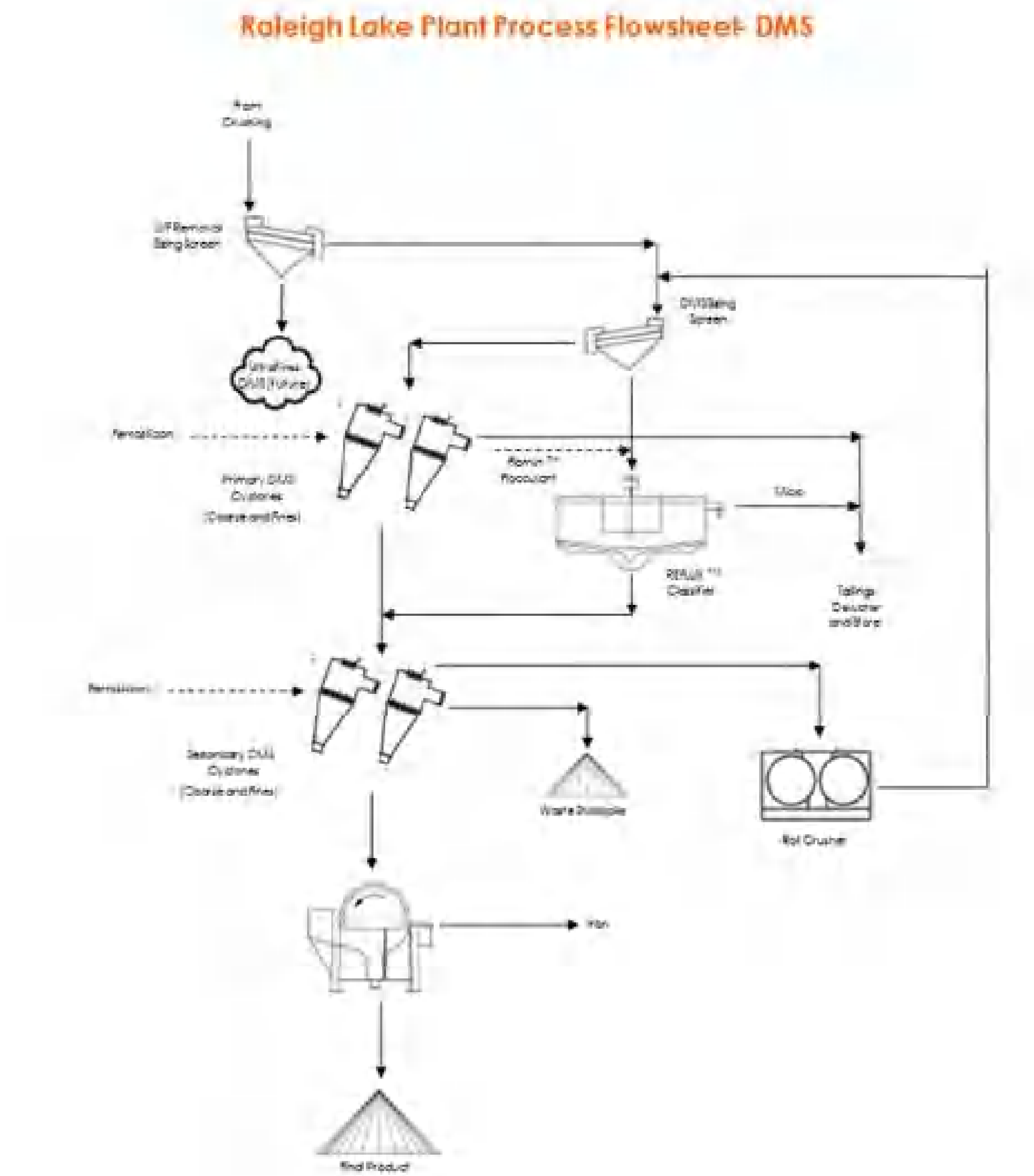


Figure 17-2: DMS Area Flowsheet

17.12 Assay Laboratory

The assay laboratory will be built on site and is expected to utilize approximately 0.1MW power. Depending on discussions and trade-off study to be conducted as part of the future work, the assay laboratory operations can be contracted, or owner operated.

17.13 Process Consumables and Reagents

The process consumables and reagents include:

- Feed and mixing tanks with the respective metering pumps system for Ferrosilicon
- Complete Flocculant system for Flomin flocculant
- Crusher liners
- Spares and consumables for:
  - Cyclones
  - Pumps
  - Pipes
  - Screens
  - Belt filters

18. PROJECT INFRASTRUCTURE

18.1 Power Supply and Electrical Distribution

Site power will be obtained from the existing Hydro One 235 kV main power transmission line located directly adjacent to Highway 17. Power demand for the Project is estimated to be 4660 KVA.

A main substation facility located adjacent to the main Hydro One transmission line will down step the 235 kV voltage down to 13.8 KV and transfer it to the main Project site substation using a power line located in the main access road corridor. Site power will also be transmitted at 13.8 KV which will be converted to 4160/600/240/120 Volts by transformers as required.

The close proximity of the project to adequate power infrastructure provides for a relatively low cost source of power for the project with minimal capital cost and minimal construction and implementation time.

18.2 Waste Storage Facilities

18.2.1 Waste Rock / Tailings Co-disposal Storage Facilities

The waste rock and tailings produced by mining operations is classified as low risk, meaning that it has low acid drainage and leachability potential. It is therefore proposed that a waste rock and dry tailings co-disposal method be utilized.

The total volume of waste rock produced in open pit mining operations is based on the optimized pit shells, which will be equivalent to 53Mt (approximately 30 million cubic metres) of waste rock.

Similarly, the total volume of DMS tailings produced will consist of 4Mt of solids in the tailings which will occupy a volume of approximately 3 million cubic metres.

It is proposed that separate storage facilities be constructed for waste rock that will be used for site construction and/or potential sale to external buyers.

The waste rock storage facilities will be designed such that rubidium containing waste will be segregated for easy recovery should economic factors change such that the rubidium containing waste would become sufficiently attractive economically to process and sell.

18.3 Water Storage and Management Facilities

Sedimentation ponds will be constructed to contain all open pit mining contact water and runoff from the co-disposal sites.

Surface water management system will be designed to minimize contact water and keep surface runoff from entering the pit and other mine disturbed areas to the highest practical degree, with ditches and culverts designed to meet the prevailing maximum storm requirements of local and regional/provincial regulations and legislation.

Storm water storage ponds will be constructed where required and stormwater management strategy will be implemented to ensure contact water released to nature will meet all applicable quality standards before release.

18.4 Water Treatment Plant

Potable and process water for the mine site will be drawn from Raleigh Lake and treated in on site facilities as required.

18.5 Fuel Supply

A fuel farm will be constructed on site which to store diesel and gasoline.

18.5.1 Natural Gas

Natural gas will be supplied to site from the existing natural gas pipeline also located adjacent to Highway 17. Natural gas distribution infrastructure may be provided by the supplier with no up front costs.

18.5.2 Diesel Storage

Diesel fuel required for the proposed mobile mining equipment fleet will be stored in the fuel farm facility constructed on the Project site.

The quantity of diesel fuel to be stored at the fuel farms is estimated to be 20,000 to 40,000 litres.

18.5.3 Gasoline

Gasoline required for light vehicles will be stored in the fuel farm facility constructed on the Project site.

The quantity of gasoline to be stored at the fuel farms is estimated to be 3,000 to 5,000 litres.

18.6 Heat Recovery for HVAC

A heat recovery system and other energy saving infrastructure should be investigated at the PFS level of study, but it is highly conceivable that a heat recovery facility would be technically economically viable.

18.7 Plant Control System (PCS)

A SCADA system will be installed to permit automatic control of various components of the processing plant and will include a data historian system (such as OSI-PI), HMI and Process Control System processor connected to a network of sensors on various equipment and structures.

The system will have the ability to control and monitor all equipment connected to the remote process control system.

18.8 Mining Infrastructure

18.8.1 Maintenance and Warehouse / Storage Facilities

Maintenance facilities will be constructed on site and will include:

- A fully equipped garage for preventative maintenance and overhaul/rebuild tasks for mobile mining equipment
- Electrical shop
- Supply Warehouse

18.8.2 Explosives

Explosives and cap magazines will be constructed on site at allocation to be determined and in compliance with all applicable legislation and safety regulations.

18.8.3 Pit Dewatering

Pit dewatering will be facilitated by a combination of in pit pumps and perimeter pitless well pumps as required following recommended specifications of future hydrology and hydrogeological studies as well as anticipated intensity and frequency of precipitation and snow melt (Frechette) events.

In pit pumps will collect water inflow and transfer it away from the pit into the sedimentation pond via HDPE pipe.

18.9 Communications

A fibre optic line will be extended from the existing line located adjacent to Highway 17 to provide internet and telecommunication services to site. On site communications services will include video, internet, VOIP telephone and private radio systems as well as a telecom repeater in order to facilitate mobile telephone communication across the site.

18.10 Camp

The plan is to not include a camp on site whereas the workforce will come from and / or be accommodated in Ignace (20 km from the project).

18.11 Site Roads

18.11.1 Main Access Road

The currently maintained 9.5 km long gravel surfaced access road from Highway 17 to the mine will be upgraded to accommodate haul truck traffic, which will mainly involve widening it.

The existing road is on favourable terrain which will require modest capital investment to achieve the width and bearing capacity required to accommodate haul truck traffic and the delivery of supplies and large equipment to site.

18.11.2 Site Access Road

A site access road will be constructed from the main access road to the location of the open pit mining operations. Other haulage and site service roads will also be constructed from the open pit to the co-disposal sites and surface facilities.

18.12 Port Facilities

It is currently assumed that concentrate will be transported by truck and/or rail to its point of sale. However, should it be determined in the future that shipping by boat or barge would be more economic or viable then Thunder Bay has deep water port facilities which might potentially be used for this purpose.

18.13 Concentrate Haulage

Li_2O concentrate will be transported by 60 tonne highway haul trucks to a yet to be determined future LiOH conversion plant located in Thunder Bay or Winnipeg.

18.14 Fire Detection and Fire Protection

Fire detectors will be installed in all office, warehouse, and maintenance buildings, in addition to being placed in/and around equipment in the processing plant where fires might be initiated.

A fire water system will be installed with an independent water source that will be separate form other water supplies so as to be ready in case of a fire at any time.

18.15 Security and First Aid

A security gate and office will be constructed at the entrance to the Project site.

A first aid office will also be located in the main office complex.

19. MARKET STUDIES AND CONTRACTS

19.1 Introduction

In 2022, the Canadian Government announced that it would be investing CAD$ 1.2 billion into the extraction and processing of critical minerals which includes Lithium.

The Canadian Critical Minerals Strategy officially issued in November 2022 named lithium as one of the six critical minerals to be prioritized for accelerated development.

The strategy specifies that it includes providing increased financial and administrative support for lithium mining and processing projects across Canada.

19.2 Lithium Product Markets

There are several different types of products that can be produced from a hard rock lithium mine.

The product to be shipped from the Raleigh Lake Project will be Li_2O concentrate which will be sent to an LiOH conversion plant located Thunder Bay or Winnipeg.

The price of various intermediate lithium products often depends on the quality of the product, the consistency of its quality and the dependability of its availability which will often be incorporated into the contract of an offtake agreement.

The various Battery Grades, Prices, Codes are illustrated in Figure 19-1 below. The concentrate product likely to be shipped from Raleigh Lake is highlighted in Red.

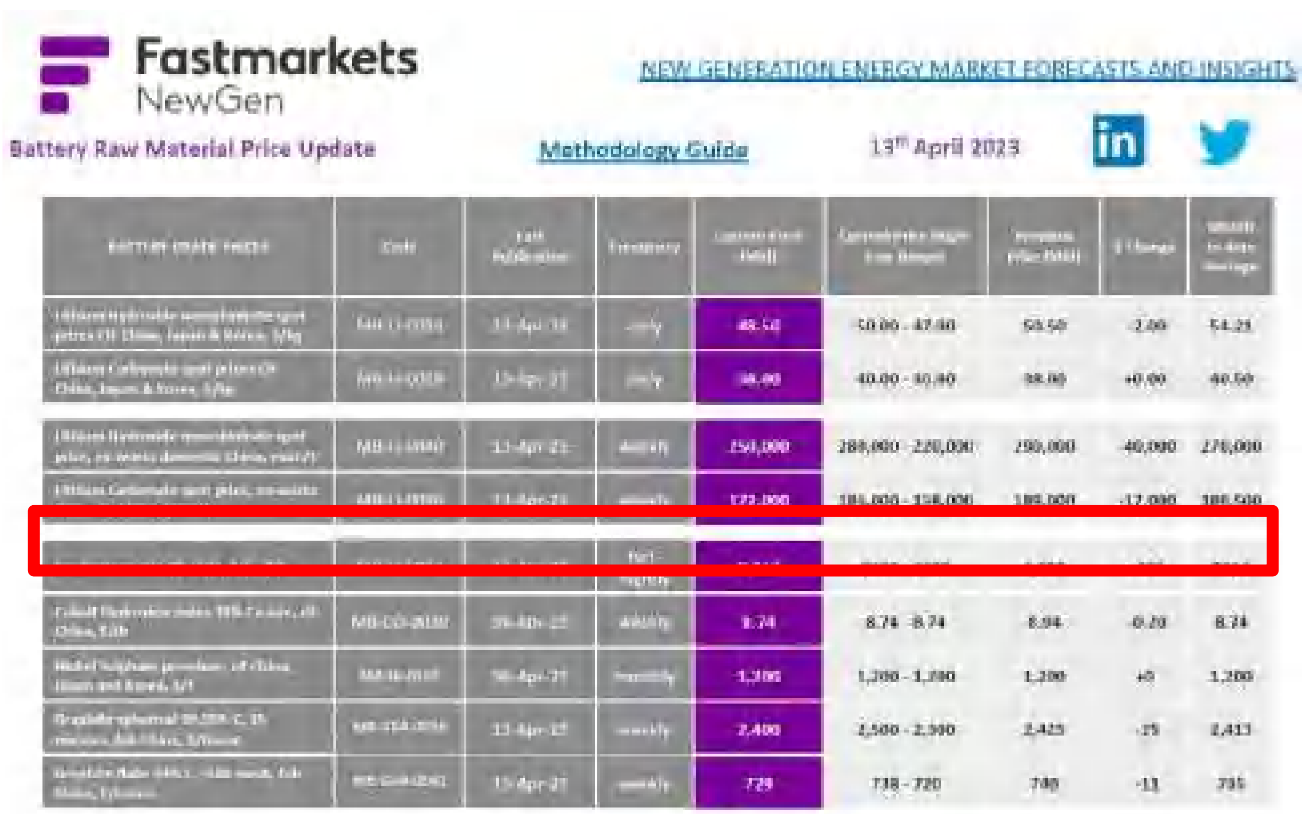


Figure 19-1: Illustration of Various Battery Grades, Prices, and Codes (Source: Fastmarkets, 2023)

19.3 Product Quality

A specific quality of spodumene of concentrate referred to as spodumene concentrate 6 or SC6, is of particular economic significance whereas it is a relatively difficult to achieve high-purity lithium mill feed with approximately 6% lithium content. It typically achieves a premium price in comparison to SC5 and SC5.5 which have 5% and 5.5% lithium content respectively.

SC6 is widely being produced as a raw material from mine concentrators for the subsequent production of lithium-ion batteries for electric vehicles.

It is plausible that the Raleigh Lake operation will produce SC6 concentrate as its saleable product.

The standard Product Code for Spodumene concentrate min 6% Li_2O Asia, $/tonne is MB-LI-0012.

> *CHINA*
>
> *MB-LI-0012 Spodumene min 6% Li_2O, spot price, cif China, $/tonne*
>
> *Quality: A mineral concentrate accepted by buyers for conversion in lithium chemicals used in battery applications (any size will be accepted) and with the following chemical composition: Li_2O 6% min 5.7 Li_2O and max 6.1% Li_2O accepted if it can be normalized to 6%); Fe_2O_3 < 1.3% (max 1.5% Fe_2O_3 accepted if it can be normalized to < 1.3%), H_2O <10%*
>
> *Quantity: 1,000 tonnes; Location: cif China; Timing: 90 days; Unit: USD/tonne*
>
> *Publication: Fortnightly, Thursday, 3-4pm London Time*
>
> *Source: Fastmarkets: https://cdn.fastmarkets.com/9c/1e/04905cbb47ff8e242329211d320e/fm-mb-lithium.pdf*

19.4 Sales Contracts

There are no sales contracts in place at the time of this study.

19.5 Li_2O Concentrate Marketing and Concentrate Terms

As of October 2023, the price of Spodumene concentrate min 6% Li_2O Asia, MB-LI-0012, has dropped significantly since the start of 2023 (Figure 19-2).

Price as of October 31, 2023 is 2,100USD per tonne of concentrate is shown in Figure 19-3.

19.6 LiOH Marketing and Concentrate Terms

The Consensus Forecast for Lithium Hydroxide (min 56.5%, LiOH.H2O Battery Grade cif, China, Japan, Korea forecasts as of October 2023 as per Consensus Economics Inc.' Energy, Metals and Agriculture Consensus Forecasts) is presented in Table 19-1 below.

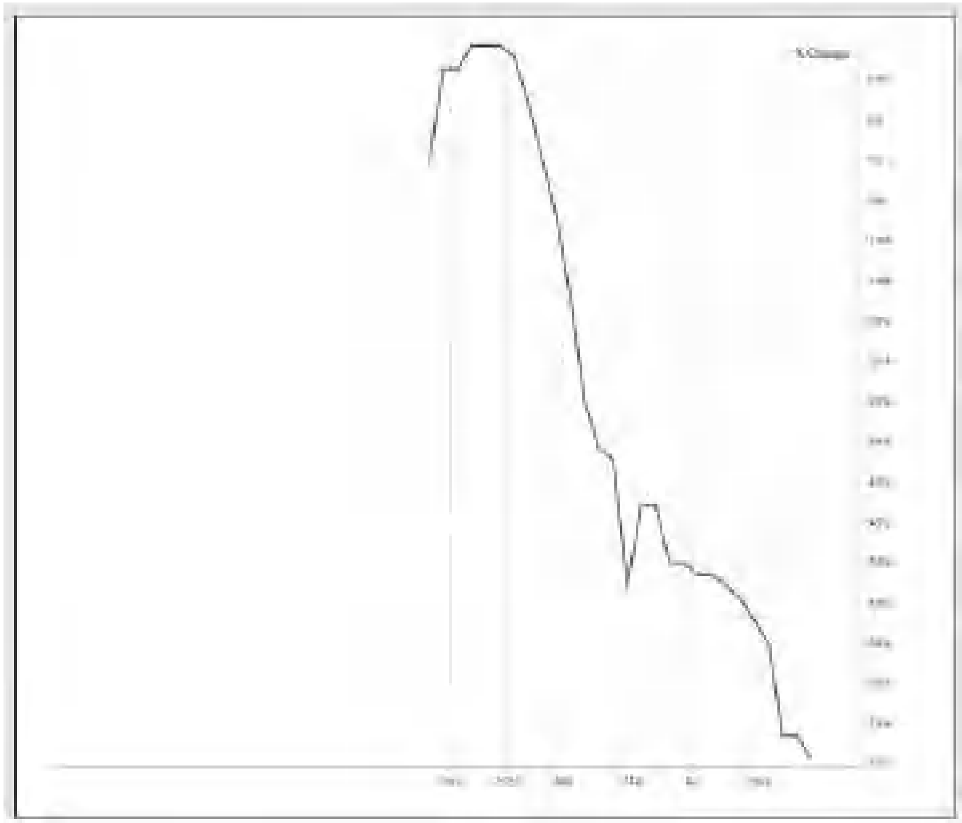


Figure 19-2: Illustration of the Price of Spodumene (Source: Fastmarkets, 2023)

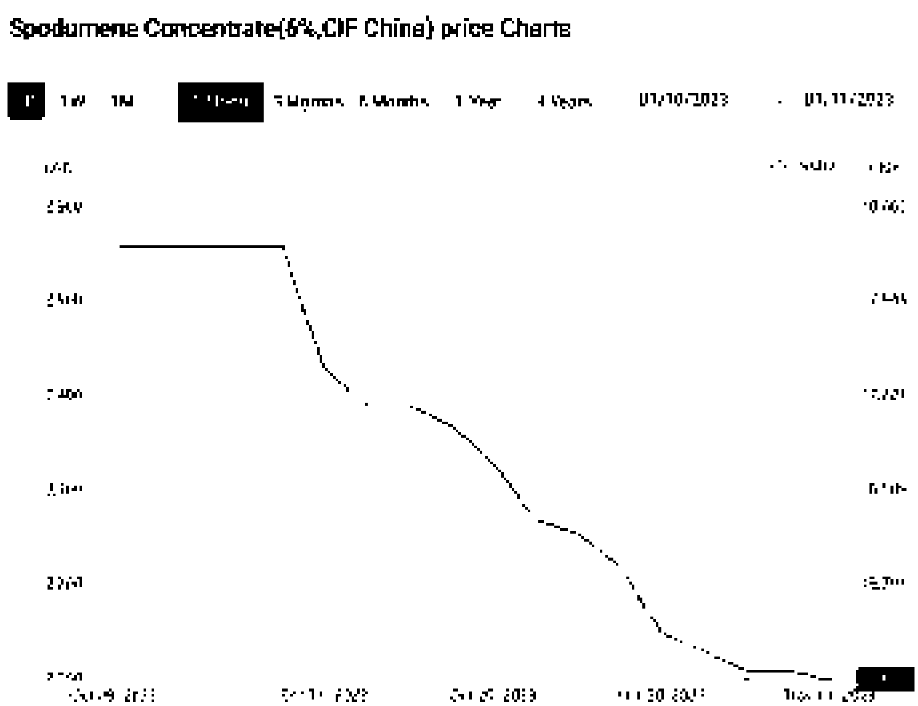


Figure 19-3: Spodumene Concentrate Price (Source: Shanghai Metals Market, 2023)

Table 19-1: Consensus Forecast for Lithium Hydroxide

Lithium Hydroxide (US$/kg)									Long Term	Long Term
October 16, 2023	Dec-23	Mar-24	Jun-24	Sep-24	Dec-24	2025	2026	2027	2028-2032	2028-2032
	(nominal)	(nominal)	(nominal)	(nominal)	(nominal)	(nominal)	(nominal)	(nominal)	(nominal)	(real)
Consensus (Mean)	*30.03*	*29.36*	*29.47*	*27.99*	*27.02*	*26.72*	*24.46*	*23.39*	*20.14*	*16.78*
High	46.21	44.17	42.92	41.67	40.42	37.29	37.00	37.00	25.68	20.70
Low	19.50	19.50	18.50	17.50	16.50	17.50	15.00	15.00	15.17	13.35
Standard Deviation	9.03	9.10	10.14	9.56	8.86	7.51	7.98	8.18	4.90	3.63
Number of Forecasts	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	4.00	4.00
Forecast Sources	Dec-23	Mar-24	Jun-24	Sep-24	Dec-24	2025	2026	2027	2028-2032	2028-2032
Australia Dept of Industry	46.21	44.17	42.92	41.67	40.42	37.29	37.00	37.00		
Liberum Capital	26.60	35.00	39.90	36.55	33.00	32.78	28.18	24.44	22.75	20.70
UBS	28.50	28.50	28.50	28.50	28.50	28.50	26.50	26.50	25.68	19.00
Investec	32.90	27.50	27.50	24.20	24.20	20.90	17.33	15.25	16.98	14.06
Morgan Stanley	26.50	21.50	19.50	19.50	19.50	17.50	15.00	15.00	15.17	13.35
BoA Securities	19.50	19.50	18.50	17.50	16.50	23.38	22.75	22.13		

19.7 Lithium Carbonate

Lithium Carbonate as a product is not considered for this project. For comparison purposes the Long Term Consensus Forecast for Lithium Carbonate as of October 2023 is presented in Table 19-2 below.

Table 19-2: Consensus Forecast for Lithium Carbonate

Survey Date:							Long Term	Long Term
October 16, 2023	2022	2023	2024	2025	2026	2027	2028-2032 (nominal)	2028-2032 (real)
Consensus (Mean) USD/kg	71.70	38.00	24.56	23.72	21.44	20.47	19.73	16.45

Lithium Carbonate Battery grade has the highest electric output per unit weight of any battery material.

Lithium-based batteries are used in applications such as electric vehicles. The largest producers are currently Australia, China, and Argentina.

The short-term consensus forecast and historic pricing for Lithium – Carbonate is presented in Figure 19-4.

LITHIUM – CARBONATE

(Nominal, Quarterly Averages)

US$/kg	Consensus (Mean)	% change from spot	Forecast Range High	Forecast Range Low
Spot price	23.58			
Dec 2023	24.75	4.9%	28.00	20.00
Mar 2024	25.50	8.1%	35.00	18.00
Jun 2024	25.50	8.1%	38.00	17.00
Sep 2024	24.25	2.8%	34.00	16.00
Dec 2024	23.00	-2.5%	30.00	15.00
Mar 2025	24.17	2.5%	30.00	17.50
Jun 2025	23.83	1.1%	29.00	17.50
Sep 2025	23.50	-0.4%	28.00	17.50

Figure 19-4: Short-term Consensus Forecast and Historic Pricing for Lithium – Carbonate (Source: Consensus Economics Inc. – Energy, Metals and Agriculture Consensus Forecasts, October 2023)

Lithium Carbonate (Battery Grade, min 99.5% Li_2CO_3, cif Asia) price since 2017 is presented in Figure 19-5.

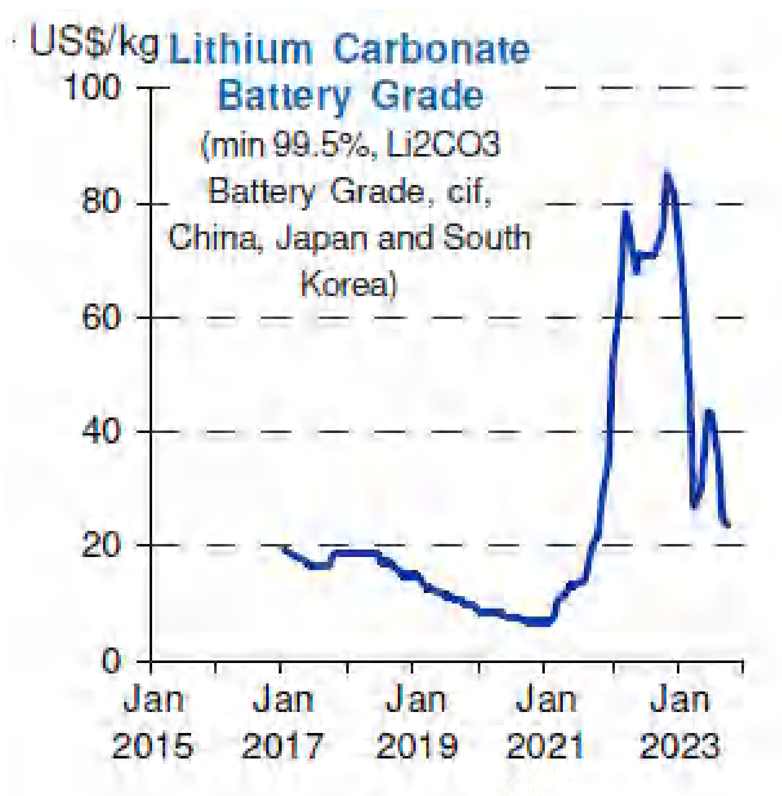


Figure 19-5: Lithium Carbonate Price since 2017 (Source: Consensus Economics Inc. – Energy, Metals and Agriculture Consensus Forecasts, October 2023)

19.8 Lithium Spodumene

The short-term consensus forecast and historic pricing for Lithium – Spodumene is presented in Figure 19-6.

LITHIUM – SPODUMENE
(Nominal, Quarterly Averages)

US$/metric tonne	Consensus (Mean)	% change from spot	Forecast Range High	Forecast Range Low
Spot price	2390			
Dec 2023	2543	6.4%	3178	2000
Mar 2024	2457	2.8%	2867	1950
Jun 2024	2396	0.3%	2667	1850
Sep 2024	2234	-6.5%	2500	1750
Dec 2024	2062	-13.7%	2500	1500
Mar 2025	2063	-13.7%	2250	1835
Jun 2025	2035	-14.9%	2250	1773
Sep 2025	1957	-18.1%	2250	1711

Figure 19-6: The Short-term Consensus Forecast and Historic Pricing for Lithium – Spodumene (Source: Consensus Economics Inc. – Energy, Metals and Agriculture Consensus Forecasts, October 2023)

Spodumene concentrate price since 2017 is presented in Figure 19-7 below.

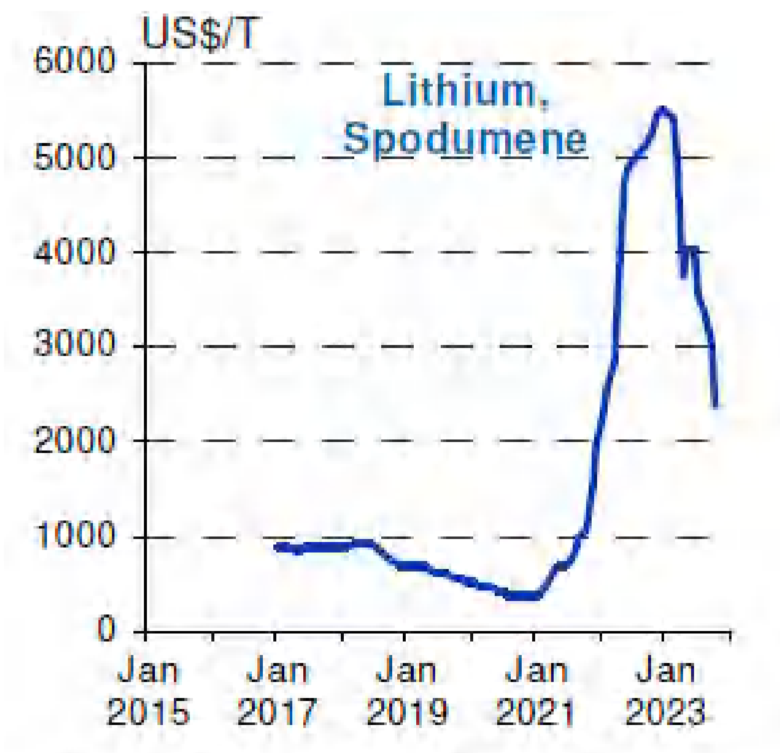


Figure 19-7: Spodumene Concentrate Price since 2017 (Source: Consensus Economics Inc. – Energy, Metals and Agriculture Consensus Forecasts, October 2023)

A slowdown in South /American brine operations has meant that the bulk of new lithium capacity coming online is being supplied from hard rock mines.

Spodumene (Ore min 5-6%, Li_2O, cif China) apyroxene mineral consisting of Lithium aluminum inosilicate, provides the largest proportion of all mineral derived lithium.

19.9 Rubidium

ERM understands that there is significant occurrence of Rubidium in the microcline feldspar at the Raleigh Lake project. The current resource estimate totals 822 tonnes of contained rubidium in the measured and indicated category and a further 521 tonnes in the inferred category. The Company is exploring what potential markets there may be for it, with three key factors being:

1. the current market price for Rubidium is US$1,150 per kilogram or US$ 1,150,000 per tonne, which if even mostly achievable would vastly improve the economic viability of the Raleigh Lake project.
2. there is there is a relative lack of clarity on the true size of the world market for rubidium with estimates ranging from 5 tonnes per annum for rubidium metal up to 30 tonnes per annum of rubidium chemicals.
3. It might be anticipated that world demand for rubidium could expand if there were ever a reliable North American production source for it such as the Raleigh Lake project. Conversely, selling in significant quantities in relation to world demand could adversely affect the market price without a commensurate increase in demand.

Rubidium is on the U.S. Critical Minerals list, and currently known uses for rubidium include the following:

1. Rubidium Carbonate is used in speciality glasses such as fibre optic cables, telecommunications systems including an important role in night vision devices, in vacuum tubes and in radiation monitoring equipment.
2. Rubidium Oxide is used in high voltage electrical conductors.
3. Rubidium atomic clocks which are at the core of satellite navigation/GPS systems and aircraft guidance systems, and military applications such as secure communications, telemetry and navigation. See https://www.worldscientific.com/worldscibooks/10.1142/11249#t=aboutBook
4. Quantum Computing
5. If sodium-ion batteries were to take market share from lithium-ion batteries in future, for example in electric battery storage, small amounts of rubidium and cesium have been shown to improve the performance of sodium-ion batteries.
6. Since Rubidium is easily ionized, it can be used as a propellant in ion engines on spacecraft.
7. Broadly, rubidium can be used where cesium is used, can also be used such as in rubidium formates in drilling fluids. When drilling boreholes to extract oil and gas cesium formate or rubidium formate is used to maintain hydrostatic pressure in the well and protect drilling polymers from thermal degradation while cooling and cleaning the drill bit.
8. Fireworks pyrotechnics for the red colour in fireworks.
9. Rubidium Oxide is added to glazes used in expensive China porcelain dishes.

As a consequence of the very high uncertainties involved in predicting what the near future demand might be or the price point at which very large quantities of rubidium might be sold no estimate has yet been made for it’s effect on any revenues or the economic potential of producing and selling rubidium mined at Raleigh Lake. Consequently, this PEA deals with the lithium resource only.

The assumption in this PEA is that all mined material containing rubidium would be segregated and stored pending future potential offtake opportunities.

19.10 Commodity Prices and Foreign Exchange Rate

There is currently no plan in place to hedge the commodity prices of spodumene concentrate nor LiOH against foreign exchange rates (i.e., CAD/USD).

19.11 Additional Contracts Required

At the time of the writing of this report, ERM was not aware of any contracts material to the issuer that are required for property development, mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts, nor any other arrangements.

19.12 Contracts under Negotiation

At the time of the writing of this report, ERM was not aware of any contracts in place, and which are still under negotiation.

19.13 Contracts in Place

At the time of the writing of this report, ERM was not aware of any contracts that are in place.

20. ENVIRONMENTAL STUDIES, PERMITTING AND COMMUNITY IMPACT

20.1 Environmental Studies

20.1.1 Climate

Climate in the region is moderate, characterized by long, cold winters and relatively short, cool summers. Winter temperatures can fall as low as -40°C and accumulations of snow may reach over 2 metres in depth. Snow-free months typically extend from May to September. Summer temperatures can reach +30°C. Average annual precipitation at the Atikokan Automated Reporting Station (approximately 74 km south of Ignace) is reported to be 535 mm per year.

As the Project location is relatively remote, and there are no large industrial air emissions sources nearby, concentrations of existing air contaminants are expected to be minimal. Air quality baseline studies and dispersion will be required to support future air discharge permitting.

20.1.2 Terrain

The Project is located within the Wabigoon Subprovince of the Canadian Shield's Archean Superior Province. Locally, Project infrastructure will be situated in an area of moderate relief (+/- 10 metres) with a mean elevation of 490 metres above sea level.

20.1.3 Surface and Ground Water

The Project site is located on the height of land between the Nelson River and Winnipeg River watersheds, which ultimately report to Hudson Bay. The hydrological regime in the area is generally snowmelt dominated and characterized by high flows in the late spring and low flows during the winter months. The property has many lakes and watercourses, the most significant waterbody being Raleigh Lake located east of the Project infrastructure.

Project-specific water quality or quantity studies have not been completed to date. Baseline studies, and potentially modelling of both ground and surface waters, will be required to support the EA and various permit applications.

20.1.4 Vegetation and Soils

The Project lies within the Boreal Forest Region and the Lake Wabigoon Ecoregion (Ecoregion 4S). Jack pine and spruce are typical coniferous trees while trembling aspen and white birch are found in association. Bog species are also common including sphagnum moss, Labrador tea, herbaceous species, grasses and sedges. The Project area has recently been disturbed by logging, including a network of primary and secondary logging roads.

Occurrences of provincially listed plants have been documented in the Project area along Highway 17. Further studies on ecosite, vegetation and soils will be undertaken during the environmental assessment. A limited exploration biogeochemistry survey was conducted on site in 2021 (Section 9.3.2).

20.1.5 Fish and Fish Habitat

Based on a review of publicly available information, no federal or provincially listed aquatic species at risk have been identified in the Project area. Further site-specific studies on fish and fish habitat will be undertaken to support the environmental assessment and various permit applications. The further studies will seek to understand the project interactions with aquatic values and inform mitigation design and plans required.

20.1.6 Wildlife

Wildlife found in the Project area are typical of that found in the Boreal Forest, including gray wolf, ermine, fisher, American mink, moose, snowshoe hare, bald eagle, merlin, ruffed grouse, gray jay, common raven, hermit thrush, yellow-rumped warbler, blue-spotted salamander, boreal chorus frog, green frog, western painted turtle, and red-sided garter snake.

Based on a review of publicly available information, both provincially- and federally listed wildlife species at risk have been documented in the Project area, primarily along the Highway 17. Further studies will determine if they have habitat in the Project area, the locations of that habitat and management requirements.

20.1.7 Archaeology and Heritage Resources

The Project will be subject to Provincial archaeological and heritage resource regulation which includes a phased assessment regime, and if necessary, documentation and/or recovery. Archaeological field-based assessment has not been initiated to-date, therefore potential Project interaction with these resources is unknown at this time.

20.2 Permitting

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. All Project phases will be subject to federal, provincial, and potentially municipal environmental regulation. These regulations mandate, among other things, maintenance of air, water, and soil quality standards and govern emissions to the receiving environment. A breach of such regulations can result in provision of fines or penalties.

The EA processes provide a mechanism for reviewing projects to assess potential impacts to the environment. A comprehensive permitting process is completed to allow various Project aspects to proceed and are regulated through all phases by both federal and provincial government agencies.

This section summarizes EA and permitting requirements for the Project.

20.2.1 Federal Environmental Assessment

Production rates for the mine and Process Plant are below the thresholds in Chapter 18 of the Physical Activity Regulations (SOR/2019-285) under the *Impact Assessment Act* (2019), as follows:

- Maximum ore production capacity of the mine is less than 5,000 tonnes per day; and
- Maximum ore input capacity of the Process Plant is less than 5,000 tonnes per day.

As a result, the Project is not anticipated to trigger a federal impact assessment. However, in unique cases the Minister of Environment and Climate Change Canada may designate a physical activity that is not prescribed by the Physical Activities Regulations if, in their opinion, either the carrying out the physical activity may cause adverse direct or indirect effects within federal jurisdiction, or public concerns related to those effects warrant the designation.

20.2.2 Provincial Environmental Assessment

Private sector metal mines in Ontario do not trigger an Individual EA under Ontario's *Environmental Assessment Act* (1990; EAA). However, the Project may be subject to multiple Class EA processes under the EAA associated with specific Project components or activities. The following Class EA processes may apply:

- Minor Transmission Facilities – construction and operation of the substation and/or transmission line; and
- Ministry of Natural Resources Resource Stewardship and Facility Development Projects – occupancy of Crown lands, dams, and access roads.

ILC could enter into a Voluntary Agreement under Section 3.0.1 of the provincial *Environmental Assessment Act*. This regulatory pathway consolidates multiple Class EA processes for specific project components into one single Individual EA for review by the appropriate government review team. The individual EA should be prepared in accordance with project components that are subject to the *Environmental Assessment Act* or that have been otherwise agreed to as part of the Voluntary Agreement. ILC will consult with relevant provincial agencies as planning progresses to confirm the most efficient EA process for the Project.

20.2.3 Federal and Provincial Permits or Authorizations

ILC currently holds an early exploration permit to conduct mechanized drilling identified as PR-22-000057 which covers diamond drilling within 195 SCMC within the claim group and is in good standing until September 15th, 2025. ILC previously held Instrument PR-20-000001 which covered diamond drilling within 13 SMCM within the claim group and was in good standing until February 12, 2023. Advanced exploration will require additional permitting.

Advanced exploration, construction and operation of the Project will require several federal and provincial environmental permits and/or authorizations, depending on activities and infrastructure needs. Potential federal permits or approvals are listed in Table 20-1 and potential provincial permits are listed in Table 20-2. ILC will consult with relevant government agencies as planning progresses to confirm permit requirements.

20.3 Mine Closure

At the completion of mineral extraction, processing and transportation activities, removal of the site infrastructure that supports these activities will be required. All soil cover, vegetation, and surface water features altered during the life of mine must be restored to a quality, quantity, and functionality that existed pre-development. In accordance with the *Mining Act*, ILC will develop a mine Closure Plan that outlines how this will be accomplished, along with associated cost estimates. The Closure Plan will be filed with the Director of Mine Rehabilitation ahead of advanced exploration or construction and may be amended from time to time during the life of mine if material changes are made.

In accordance with Section 145 of the *Mining Act,* a financial guarantee equal to the amount specified in the Closure Plan cost estimate must accompany its filing, to be held in trust by the Ministry of Mines.

Table 20-1: Potential Federal Permits or Approvals

Related Legislation	Permits or Approvals	Agency	Relevant Triggers
EXPLOSIVES ACT, R.S.C., 1985, c. E-17	Factory Licence	Natural Resources Canada	Manufacture and use of explosives.
EXPLOSIVES ACT, R.S.C., 1985, c. E-17	Permit for Use of Explosives	Natural Resources Canada	Manufacture and use of explosives.
EXPLOSIVES ACT, R.S.C., 1985, c. E-17	Explosives Magazine Permit	Natural Resources Canada	Storage of explosives.
FISHERIES ACT, R.S.C. 1985, c. F-14, s. 1.	Authorization	Fisheries and Oceans Canada	Death of fish, harmful, alteration, disruption of fish habitat (HADD), fish collection for scientific purposes.
FISHERIES ACT, R.S.C. 1985, c. F-14, s. 1. ▪ *Metal and Diamond Mining Effluent Regulations, SOR /2002-222*	Schedule 2 Amendment	Fisheries and Oceans Canada	Deposit of deleterious mine waste in waters frequented by fish.
MIGRATORY BIRDS CONVENTION ACT, 1994, S.C. 1994, c. 22	Permit	Environment and Climate Change Canada	Authorizes periods during which migratory birds may be (killed/captured) and their nests or eggs (destroyed).
NAVIGATION PROTECTION ACT, R.S.C., 1985, c. N	Minor Works Order or Approval	Transport Canada	Construction, placement, decommissioning, alteration or rebuild of a "work" on, over, under, through or across any navigable water.
NUCLEAR SAFETY AND CONTROL ACT, S.C. 1997, c. 9	*Nuclear Safety and Control Act* Licence	Canadian Nuclear Safety Commission	Use, possession or importing of a prescribed substance in a device such as analyzers, chromatographs, calibrators, fixed and portable gauges, industrial radiography, logging, detectors, etc.
RADIO COMMUNICATIONS ACT, R.S.C. 1985, c. R-2	Licence	Innovation, Science, and Economic Development Canada	Issuance and operation of designated frequency.
SPECIES AT RISK ACT, S.C. 2002, c. 29	Permit	Environment and Climate Change Canada	Engagement in an activity affecting a listed wildlife species, any part of its critical habitat or the residences of its individuals as per the *Species at Risk Act* (SARA).
TRANSPORTATION OF DANGEROUS GOODS ACT, S.C. 1992, c. 34	Permit	Transport Canada	Transport of dangerous goods.

Table 20-2: Potential Provincial Permits or Approvals

Related Legislation	Permits or Approvals	Agency	Triggers
Crown Forest Sustainability Act, 1994, S.O. 1994, c. 25	Forest Resource Licence	Ministry of Northern Development, Mines, Natural Resources and Forestry	Tree removal on Crown land.
Crown Forest Sustainability Act, 1994, S.O. 1994, c. 25	Fuelwood Permit	Ministry of Northern Development, Mines, Natural Resources and Forestry	Tree removal on Crown land.
Endangered Species Act, 2007, S.O. 2007, c. 6 *Section 17(1)*	Letter of Advice or Overall Benefit Permit	Ministry of the Environment, Conservation and Parks	Activities where adverse effects to endangered or threatened species at risk or their protected habitat cannot be avoided.
Environmental Protection Act, R.S.O. 1990, c. E.19 *O. Reg. 245/11 - Registrations under Part II of the Act - General* *O. Reg. 63/16 - Registrations Under Part II of the Act - Water Taking*	Environmental Activity Sector Registry (EASR) Approval	Ministry of the Environment, Conservation and Parks	Certain routine and lower risk activities must be self-registered online in the EASR. Example: ▪ Specific water taking activities as per O. Reg. 63/16 including water taking for construction site dewatering < 400,000 litres per day.
Environmental Protection Act, R.S.O. 1990, c. E.19 *Ontario Water Resources Act* *s. 53*	Environmental Compliance Approval – Industrial Sewage Works	Ministry of the Environment, Conservation and Parks	Construction of a wastewater collection and treatment facility – sedimentation pond. Construction of tailings facility. Construction of a private sewage disposal system exceeding 10,000 litres per day. Refer to local health unit for < 10,000 litres per day.
Environmental Protection Act, R.S.O. 1990, c. E.19 *s. 9* *O. Reg. 419/05 - Air Pollution - Local Air Quality*	Environmental Compliance Approval – Air and Noise	Ministry of the Environment, Conservation and Parks	Discharge of a contaminant, including noise, into the natural environment. Section 9 requires companies to obtain an approval before construction, alteration, extension or replacement of any equipment or structure that may emit or from which may be emitted a contaminant into the natural environment, other than water.
Environmental Protection Act, R.S.O. 1990, c. E.19 *s. 27*	Environmental Compliance Approval – Waste Disposal and Management	Ministry of the Environment, Conservation and Parks	Construction of a waste disposal site or certain onsite treatment of wastes. Hauling of waste to private landfill. Not required if using licensed hauling companies.
Environmental Protection Act, R.S.O. 1990, c. E.19 *O. Reg. 347/90: General – Waste Management*	Generator Registration	Ministry of the Environment, Conservation and Parks	Storage and transportation of hazardous wastes. Type / amount of waste is registerable or hazardous as defined in Ont. Reg. 347.
Fish and Wildlife Conservation Act, 1997, S.O. 1997, c. 41 *s. 39* *O. Reg. 664/98: Fish Licensing*	Authorization to Collect Fish for Scientific Purposes	Ministry of Northern Development, Mines, Natural Resources and Forestry	Collection of fish for testing (e.g., for environmental baseline studies).
Forest Fire Prevention Act, R.S.O. 1990, c. F.24	Burning Permit	Ministry of Northern Development, Mines, Natural Resources and Forestry	Burning of cleared vegetation.
Health Promotion and Protection Act, R.S.O. 1990, c. H.7 *O. Reg. 554/90: Camps in Unorganized Territory*	Notice of Camp Opening	Ministry of Health and Long Term Care	Opening of a camp housing greater than 5 people in unorganized territory.
Lakes & Rivers Improvement Act, R.S.O. 1990, c. L.3 *O. Reg. 454/96: Construction*	Location Approval Plans and Specification Approval	Ministry of Northern Development, Mines, Natural Resources and Forestry	Construction of dams, dykes or diversions, including tailings dams.
Mining Act, R.S.O. 1990, c. M.14 *O. Reg. 308/12: Exploration Plans and Exploration Permits* *or* *O. Reg. 240/00: Mine Development and Closure*	Permission to Test Material	Ministry of Northern Development, Mines, Natural Resources and Forestry	Bulk sampling to test mineral materials.
Mining Act, R.S.O. 1990, c. M.14 *O. Reg. 308/12: Exploration Plans and Exploration Permits*	Exploration Plan OR Permit	Ministry of Northern Development, Mines, Natural Resources and Forestry	An Exploration Plan submission is required for the following activity thresholds: a. Any geophysical surveys that require the use of a generator to be carried out;

Related Legislation	Permits or Approvals	Agency	Triggers
			b. Mechanized drilling for the purpose of obtaining rock or mineral samples, if the assembled weight of the drill and its associated equipment, excluding drill rods, casings and bits, does not exceed 150 kilograms; c. Line cutting, where the width of the lines does not exceed 1.5 metres; and d. Mechanized surface stripping where: o a single location is to be stripped and the total area to be stripped does not exceed 100 square metres, or o two or more locations are to be stripped and the edges of a location where stripping is to be carried out are within 200 metres of the edges of another location, and the aggregate of the area of the locations to be stripped does not exceed 100 square metres. An Exploration Permit (application) is required for the following activity thresholds: a. Mechanized drilling for the purpose of obtaining rock or mineral samples, if the assembled weight of the drill and associated equipment, excluding drill rods, casings and bits, is greater than 150 kilograms; b. Line cutting, where the width of the lines cut is 1.5 metres or more; c. Mechanized surface stripping where: o a single location is to be stripped and the total area to be stripped exceeds 100 square metres but is less than the threshold for advanced exploration, or o two or more locations are to be stripped and the edges of a location where stripping is to be carried out are within 200 metres of the edges of another location and the aggregate of the total area to be stripped exceeds 100 square metres but is less than the threshold for advanced exploration; and d. Pitting and trenching where: o there is a single pit or trench and the total volume of the pit or trench exceeds three cubic metres but is below the threshold for advanced exploration, or o there are two or more pits or trenches and the edges of a pit or trench are within 200 metres of the edges of another pit or trench and the aggregate of the total volume of the pit or trench exceeds three cubic metres but is below the threshold for advanced exploration.
Mining Act, R.S.O. 1990, c. M.14 *O. Reg. 240/00: Mine Development and Closure*	Closure Plan	Ministry of Northern Development, Mines, Natural Resources and Forestry	Advanced exploration activities require filing of a closure plan.
Ontario Water Resources Act, s. 34 *O. Reg. 387/04: Water Taking*	Permit to Take Water (>50,000 L)	Ministry of the Environment, Conservation and Parks	Taking water (pumping, draining, dewatering) >50,000 litres/day.
Ontario Water Resources Act, R.S.O. 1990 *O. Reg. 170/03: Drinking Water Systems*	Approval	Ministry of the Environment, Conservation and Parks	Installation of a drinking water system.

20.4 Community Engagement

20.4.1 Indigenous Engagement

The existing aboriginal and treaty rights of Indigenous groups are recognized and affirmed under Section 35 of Canada's *Constitution Act.* (1982) The federal and provincial governments share the duty to consult Indigenous communities regarding developments such as this Project as part of EA and approvals processes. In practice, duty to consult is often delegated to the proponent.

It is anticipated that the Ministry of Mines will provide guidance to ILC regarding the consultation that is required for the Project and which aspects of duty to consult may be delegated to ILC. In turn, ILC will prepare a work plan in accordance with the requirements of the *Mining Act* (1990), among others.

First Nation's and Metis communities are situated near the Raleigh Lake property and consider the area part of their traditional territory. ILC has identified and initiated engagement with the following Indigenous groups that may have an interest in the Project:

- Wabigoon Lake Ojibway Nation;
- Eagle Lake First Nation
- Seine River First Nation;
- Nigigoonsiminikaaning First Nation;
- Lac Seul First Nation;
- Metis Nation of Ontario – Northwest Community;
- Lac des Milles Lacs First Nation; and
- Grand Council Treaty #3.

As the lands and community of the WLON are the closest to the Project, WLON has been the foremost community for communication and involvement by Company representatives. There have been regular communications regarding various aspects of the Project. In addition, members of WLON have participated in the 2021 and 2022 drilling programs and Company representatives have participated in WLON community events.

ILC's plans for ongoing engagement that may include meetings with Chiefs and council, tours of the Project site, and community engagement sessions. At present ILC has not entered into any form of joint communications plan, commercial or benefit agreement with Indigenous groups. Completion of an EA for the Project will include demonstration of how Indigenous knowledge is incorporated into effects assessments and environmental mitigations.

20.4.2 Public Engagement

Engagement with local and regional stakeholders has been initiated and will continue as the Project progresses. Engagement efforts will include meetings with local and provincial government agencies, public information sessions and other forms of communications to ensure stakeholders are aware of ILC's plans and activities, and concerns can be heard and resolved.

21. CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating costs in this report chapter are based on the design criteria and engineering work performed by the individual QPs under their areas of responsibility and expertise.

Sources used for the estimates include vendor quotations historical data, benchmark costs at similar operations, CostMine databases, empirical factors and first principle calculations where and as appropriate.

The Raleigh Lake Preliminary Economic Analysis Study (PEA) involves the development of an open pit mine, the construction of on-site processing facilities and all infrastructure required to support those activities.

The main components that form the basis of this study include:

- An open pit mine
- Mine waste rock storage areas and stockpiles
- Run of mine (ROM) stockpiles
- A lithium oxide (Li_2O) concentrator plant with a feed capacity of 540,000 tpa
- A waste rock/tailings co-disposal facility
- Power transmission and distribution
- Main highway access and site access roads
- Site buildings and support infrastructure
- Water supply and distribution systems
- Waste water impoundment and treatment facilities
- Explosives and cap storage facilities
- Mine closure and reclamation

Lithium oxide (Li_2O)) concentrate will be transported to an offsite location located in Thunder Bay or Winnipeg for conversion into market grade lithium products.

The cost estimates for each report chapter have been prepared by the following parties:

- ERM for the open pit capital and operating costs
- ERM for the concentrator plant capital and operating costs
- ERM for the site infrastructure capital and operating costs
- ERM for the mine closure capital costs
- ERM for the indirect capital costs

21.2 Capital Costs

21.2.1 Basis of Estimate

The capital cost estimate for the Raleigh Lake has been prepared to an accuracy of + 30% / - 20% based on a 10% to 40% engineering completion ratio to conform with the requirements for an American Association of Cost Engineers (AACE) Class 3 Estimate.

All capital cost estimates are based on Q4 2023 Canadian Dollars (CAD$) and an assumed US to Canadian Dollar ratio of $1 USD = $1.35 CAD.

21.2.2 Pre-production Period Capital Cost Summary

Total pre-production capital costs will be $111.9M (inclusive of contingency) as shown in Table 21-1 below, which include capitalized operating costs incurred before the open pit mine moves into the production phase.

Table 21-1: Pre-production Capital Costs

Cost Centre	Description	Cost (CAD$ million)
Direct CAPEX Costs		
1000	Open Pit Mining	18.2
3000	Mineral Processing	38.5
4000	Power, Electrical and Instrumentation	3.8
5000	Site Infrastructure and Support Services	12.0
6000	Water Management Systems	2.0
7000	Tailings and Mine Waste Management Facilities	1.8
Total Direct CAPEX Costs		*75.8*
Owner and Indirect CAPEX Cost Summary		
8000	Reclamation and Closure	10.0
9000	Indirect and Owner Costs	18.6
Total Indirect CAPEX Costs		*28.6*
9600	Contingency	7.6
Total Pre-Production CAPEX Costs		***111.9***

21.2.3 Sustaining Capital Cost Summary

Total sustaining capital costs incurred over the production phase of the mine will be $17.5M as shown in Table 21-2 below.

Table 21-2: Total Sustaining Capital Costs

Cost Centre	Description	Cost (CAD$ million)
Direct SUSEX Costs		
1000	Open Pit Mining	9.1
6000	Water Management Systems	0.2
7000	Tailings and Mine Waste Management Facilities	3.8
Total Direct SUSEX Costs		*13.0*
Owner and Indirect SUSEX Cost Summary		
9000	Indirect and Owner Costs	3.2
Total Indirect Costs		*3.2*
9600	Contingency	1.3
Total SUSEX		***17.5***

21.2.4 Open Pit Mine Capital Costs

Capital costs for the open pit mine over the pre-production period were estimated to be $18.2M, with an additional $89.1M in sustaining capital costs over Life of Mine (LOM). All costs were estimated by ERM.

The breakdown of open pit mining costs between the pre-production and capital costs are summarized in Table 21-3.

Table 21-3: Breakdown of Open Pit Mining Capital Costs

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Mobile Equipment	17.7	0.0	17.7
Explosives Storage	0.3	0.0	0.3
Fixed Equipment	0.3	0.3	0.6
Capital Replacements	0.0	8.7	8.7
Total	**18.2**	**9.1**	**27.2**

All mobile equipment (including spares) required for open pit mining will be purchased in the pre-production period as summarized in Table 21-4.

Sustaining capital costs were factored to account for major mobile equipment overhauls and rebuilds required over the operating life of the mine.

Table 21-4: Mobile Equipment List

Description	Quantity
Excavator	1
Haul Truck	3
Rotary Drills	2
Bulldozer	1
Explosives Truck	1
Excavator – Spare	1
Haul Truck - Spare	1

21.2.5 Mineral Processing Capital Costs

Capital costs for the on-site mineral processing facilities incurred during the pre-production period were estimated to be $38.5M. All costs were estimated by ERM and Magemi Mining.

The capital estimates include the costs of purchasing the fixed equipment and construction of the crushing and DMS plants as summarized in Table 21-5.

Table 21-5: Summary of Capital Costs for Mineral Processing Facilities

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Crushing Plant	8.8	0.0	8.8
DMS Plant	29.7	0.0	29.7
Total	**38.5**	**0.0**	**38.5**

21.2.6 Power, Electrical, and Instrumentation Capital Costs

Capital costs for on-site power, electrical and instrumentation incurred during the pre-production period were estimated to be $3.8M. All costs were estimated by ERM.

The capital estimate includes the costs of purchasing the equipment and constructing the main power line to site as summarized in Table 21-6.

Table 21-6: Summary of Capital Costs for Power, Electrical, and Instrumentation

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Power Infrastructure	3.8	0.0	3.8
Total	**3.8**	**0.0**	**3.8**

21.2.7 Site Infrastructure and Support Services Capital Costs

Capital costs for site infrastructure and support services incurred during the pre-production period were estimated to be $12M. All costs were estimated by ERM.

The capital estimate includes the costs of purchasing mobile and fixed equipment required for site support services, communications equipment as well as constructing site support buildings as summarized in Table 21-7.

Table 21-7: Summary of Capital Costs for Site Infrastructure and Support Services

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Mobile Equipment	6.2	0.0	6.2
Fixed Equipment	0.1	0.0	0.1
Site Buildings and Roads	4.2	0.0	4.2
Site Communications	1.5	0.0	1.5
Total	**12.0**	**0.0**	**12.0**

21.2.8 Water Management Systems Capital Costs

Capital costs for site water management systems incurred during the pre-production period were estimated to be $2.0M. All costs were estimated by ERM.

The capital estimate includes the costs of constructing a water treatment plant as summarized in Table 21-8.

Table 21-8: Summary of Capital Costs for Water Treatment Plant Construction

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Water Treatment Plant	2.0	0.2	2.2
Total	**2.0**	**0.2**	**2.2**

21.2.9 Tailings and Mine Waste Management Facilities Capital Costs

Capital costs for the mine waste and DMS tailings co-disposal facilities incurred during the pre-production period were estimated to be $1.8M whereas sustaining capital costs will be $3.8M as summarized in Table 21-9. All costs were estimated by ERM.

Table 21-9: Summary of Capital Costs for Tailings and Mine Waste Management Facilities

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Co-disposal Facility	1.8	3.8	5.5
Total	**1.8**	**3.8**	**5.5**

21.2.10 Owner's Costs

Owner's costs incurred during the pre-production period consist mainly of general and administration costs but also includes labor, material and consumables costs for work performed during the pre-production period that would otherwise be categorized as operating costs when the project enters the production phase. Total owner's costs incurred during the pre-production phase were estimated to be $48,2M.

The owner's costs summarized in Table 21-10 were estimated by ERM.

Table 21-10: Summary of Owner's Cost

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
Mine Pre-Production	24.3	0.0	24.3
Processing Pre-Production	0.9	0.0	0.9
Site Pre-Production	11.5	0.0	11.5
Project Team Costs	11.5	0.0	11.5
Total	48.2	0.0	48.2

21.2.11 Indirect Costs

Indirect costs were estimated to be $19.1 million during the pre-production period and $5.4 million during the production phase of the project.

Items covered in this category include external services such as engineering, environmental, procurement and construction management, freight, commissioning/startup and spare parts.

The project indirect costs were estimated by ERM based on first principles or industry standard factors and are summarized in Table 21-11.

Table 21-11: Summary of Project Indirect Cost

Description	Pre-Production (CAD$ million)	Sustaining (CAD$ million)	Total (CAD$ million)
External Services	12.3	4.3	16.5
Freight	3.0	0.5	3.6
Construction Indirect Costs	1.5	0.3	1.8
Spare Parts	2.3	0.4	2.7
Total	**19.1**	**5.4**	**24.5**

21.2.12 Contingency Costs

Contingency costs were factored for capital equipment, structures and goods purchased under each cost centre. A total of $7.6M of contingency will be required in the pre-production period whereas $1.3M will be required during the production period of the mine.

21.2.13 Mine Rehabilitation Bond and Closure Costs

The Raleigh Lake project will employ a co-disposal strategy for permanent storage of the geochemically inert waste material produced by open pit mining activities and DMS tailings material produced by the concentration process. A total of $10M was assumed for the mine rehabilitation bond.

21.2.14 Working Capital

Working capital required during the pre-production was assumed to be equivalent to 10% of the total capitalized expenses incurred during the pre-production phase and is estimated to be $11.2M.

21.2.15 Exclusions

Costs associated with the following list of items are not included in the capital cost estimates:

- Federal and provincial taxes
- Force majeure, labor disputes or major strikes
- Contaminated soil and/or hazardous waste excavation, treatment, disposal or removal
- Significant variations in assumed hourly rates or skill levels of labor cost inputs
- Significant changes in assumed foreign currency exchange rates
- Pre-feasibility, definitive feasibility or value engineering studies
- Capitalized interest payments or financing costs

21.3 Operating Costs

21.3.1 Summary

The summary of operating costs are presented in Table 21-12 below.

Table 21-12: Summary of Operating Costs

Parameter	Value	Unit
Unit Operating Costs		
Mining	$3.55	/t mined
Mining	$47.18	/t mill feed
Milling	$28.53	/t mill feed
G & A	$17.74	/t mill feed
Concentrate transportation	$7.13	/t mill feed
Total	*$104.12*	*/t mill feed*
Overall Project Costs		
Total Mining Cost	179	CAD$ million
Total Milling Costs	125	CAD$ million
Total G&A Costs	78	CAD$ million
Total Operating costs	381	CAD$ million
Total Concentrate Transport Costs	31	CAD$ million
Total Operating costs plus Transport	413	CAD$ million

21.3.2 Basis of Estimate

The operating cost estimate is based on the total amount of labour, materials and consumables that will be required to fully execute the mining and processing plans as described in the previous sections of this report.

The total operating costs incurred over the life of the project are based on sufficient mill feed material being available to begin processing plant operations in Year 1 of the overall project schedule, which will include the processing of 54,037 tonnes of mill feed stockpiled in Year -1 of the pre-production schedule.

21.3.3 Open Pit Mining

Open pit mine operating costs were calculated based on the types and quantities of equipment, labor, materials and consumables that would be required to meet the proposed mining schedule.

All mine operating costs were built up from first principles, with unit costs for labour, materials and consumables being based on historical data at other mines in northwestern Ontario where available and vendor quotations or industry standards and benchmarks were not.

The main activities included in the cost calculations include drilling and blasting, loading, hauling, mining support services, labor and general site support and maintenance.

The total operating expenditures required to mine the quantities of mill feed and waste scheduled over the production period of the open pit mine were estimated to be $190 million, or a unit cost of $3.56 per tonne mined as shown in Table 21-13 below.

Table 21-13: Summary of Open Pit Mining Operating Cost

Description	Total Cost	Unit Cost
	(CAD$ million)	(CAD$/t mined)
Open Pit Mining	$150.5	$2.98
Site Support and Services	$28.5	$0.57
Total	**$179**	**$3.55**

21.3.4 Mineral Processing

Processing plant operating costs were calculated based on the types and quantities of equipment, labor, materials, and consumables that would be required to meet the proposed crushing and DMS processing schedules.

All crushing and processing plant costs were built up from first principles with unit costs for labour, materials and consumables being based on historical data at other mines in northwestern Ontario where available and vendor quotations or industry standards and benchmarks were not.

The total operating expenditures required to crush and process the quantities of mill feed scheduled over the production period of the project were estimated to be $124.6M, or $28.53 per tonne of mill feed processed into the final technical grade spodumene concentrate product as shown in Table 21-14 below.

Table 21-14: Summary of Mineral Processing Operating Cost

Description	Total Cost (CAD$ million)	Unit Cost (CAD$/t milled)
Labor	$44.2	$10.13
Consumables and Materials	$56.7	$12.99
Power	$23.6	$5.40
Total	**$124.6**	**$28.53**

21.3.5 General and Administration

General and administration or G&A costs consists of costs that are not directly related to the open pit mining or processing activities over the production period of the project life.

Total G&A costs were estimated to be $77.5M, or $17.74 per tonne of mill feed mined as shown in Table 21-15.

Table 21-15: Summary of General and Administration Operating Cost

Description	Total Cost (CAD$ million)	Unit Cost (CAD$/t milled)
Labor	$73.0	$16.71
Supplies and Consumables	$2.2	$0.51
External Services	$2.3	$0.53
Total	**$77.5**	**$17.74**

21.3.6 Road Transport of Concentrates

Road transportation costs incurred over the production life of the project assume that the final technical grade concentrate would be transported by truck to a conversion plant located in Winnipeg or Thunder Bay.

A total cost of $75 per tonne of concentrate was assumed based on a conservative truck haulage cost of $60 per tonne and total loading/unloading costs of $15 per tonne of concentrate produced, or $7.13 per tonne of mill feed for a total LoM cost of $31.1M.

21.4 Royalties

There will be no royalties on the Project.

22. ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis of the Raleigh Lake project is based on cost models prepared for each major component of the overall project, which includes an open pit mine, crushing and processing plants, supporting surface infrastructure and a waste rock / tailings co-disposal facility.

The assumed technical grade 6% spodumene concentrate product and cost calculations are all expressed in Canadian dollars unless otherwise noted, with an exchange rate of 1.35 CAD/USD being used for currency conversions.

The calculated internal rate of return (IRR) of the project does not include potential external financing costs and assumes that all required funding will be equity based. The net present value (NPV) calculations assumed a discounting rate of 8%.

The discounted cash flow model includes revenues, costs, taxes and other known factors directly related to the project but excludes indirect factors such as financing costs, sunk costs and corporate obligations.

The pre-tax economic indicators of the project include a total cash flow CAD$ 709.5 million, a pre-tax net present value (NPV) of CAD$ 385.1 million with an assumed discounting rate of 8% and an internal rate of return (IRR) of 46.5%.

The post-tax economic indicators of the project include a total cash flow CAD$ 634 million, a post-tax net present value (NPV) of CAD$ 342.9 million with an assumed discounting rate of 8% and an internal rate of return (IRR) of 44.30%.

A payback period of 4 years after construction begins, which equates to two (2) years after the start of the production phase of the project.

22.2 Cautionary Statement

The project economic analysis and its results are based on forward looking information whose validity and accuracy may vary significantly in the future from what has been assumed in this study based on all currently available information.

Forward looking statements that might significantly affect the project economics include but are not limited to the following items:

- Mineral resource and reserve estimates.
- Variances in project construction and mining schedules due to delays induced by financing, environmental assessment processes or other factors.
- The future availability and costs of skilled labor, equipment, materials, and consumables, including power and fuel costs.
- Variances in processing methods, rates and recoveries.
- Changes in provincial and federal legislative and taxation frameworks.
- Variations in future concentrate prices, selling costs and other offsite costs such as transportation, duties, offtakes, or royalties.
- General business and economic conditions, both globally and domestically.
- Currency rate fluctuations.

22.3 General Assumptions

The assumptions that form the basis of the economic analysis of the Raleigh Lake project are outlined in greater detail in other sections of the report, whereas the general assumptions used for the economic analysis itself are as follows:

- There are no unpredictable extenuating circumstances that would disrupt or delay the development or operation of the project.
- The assumed costs for labor, equipment, materials, and consumables, including power and fuel, are reasonable stable and consistent with the costs used in the analysis.
- The timelines for the completion of time critical tasks such as baseline studies and local stakeholder consultations required for the completion of environmental and impact assessments are reasonably accurate.
- Environmental approvals, permits, licenses and authorizations are obtained from government and local stakeholders are obtained as planned.
- The detailed and typically highly complex taxation structures that will ultimately apply to the project on an operational basis are represented reasonably well by the simplified assumptions used in the discounted cash flow model.
- All assumptions made regarding the mineral resource estimate and the potential economically viable portions thereof are as accurate as they can reasonably be given the level of the currently available information. This includes but is not limited to geological interpretations, commodity pricing and operating costs, mining, and processing rates, and geotechnical, hydrological and hydrogeology characterizations.
- Year 1 of the overall project schedule in the economic analysis assumes that all critical tasks including but not limited to permitting, detailed engineering, financing and procurement will be completed so that construction of the mine can begin in Year 1 and proceed without delays until the start of the mine production phase in Year 3.
- The realization of revenues from concentrate sales fall within the same year as the assumed levels of mill feed are processed.
- Sales contracts or agreements for the technical grade Li_2O concentrate product have been established with one or more parties prior to the production thereof at Raleigh Lake.
- The assumed sales price for the 6.0% chemical grade spodumene concentrate product is $2,325 USD per tonne.
- The Canadian to United States dollar exchange rate remains relatively consistent at around CAD$1.35 per USD over the project life.
- The geochemical characteristics of the mine waste and DMS tailings stored in the co-disposal facilities remain neutral as currently assumed from existing data.
- The mine rehabilitation and closure costs do not vary significantly in scope due to major changes in the requirements assumed for this study.
- The costs of future exploration activities on the property are excluded.
- There are no payable royalties.
- Any project costs incurred prior to Year 1 of the overall project schedule are fully sunk.
- The basic mining, processing, scheduling and economic parameters used for the base case cash flow modelling and project financial analysis are summarized in Table 22-1 below.

Table 22-1: Summary of Base Case Cash Flow Modelling and Project Financial Analysis

Parameter	Value	Unit
Project Schedule		
Overall project life	11	years
Mine life	9	years
Mining, Processing and Economic Parameters		
Total mill feed	4.4	Mt
Average mill feed grade	0.70	% Li_2O
Open pit mining rate	1,500	tpd
Process recovery	81.0	%
Total concentrate produced - 6% TG Li_2O	414,904	t
Commodity price - 6% TG Li_2O	$2,325	USD/t
Exchange Rate	$1.35	CAD/USD

22.4 Taxation and Royalties

22.4.1 Basic Taxation Framework

The taxation structure that will ultimately be applied to the project is highly complex, therefore the tax calculations for the economic analysis have been simplified to approximate what taxes might be paid over the entire project life.

The basic taxation scheme for Raleigh Lake, however, is that the project will be subject to a Canadian federal income tax of 15%, an Ontario provincial income tax of 11.5% and a provincial mining tax rate that varies according to the annual profit margins realized by the project.

The tax calculations in the discounted cash flow model consider federal and provincial income tax after the application of exemptions and allowances for processing, depreciation and other allowable reductions to the net project revenue used to calculate the payable amounts of Ontario and Canada income taxes.

22.4.2 Royalties

There are currently no payable royalties on the project.

22.5 Economic Analysis Results

22.5.1 Base Case

A summary of the base case capital and operating costs calculated and used in the economic analysis exercise is shown in Table 22-2 below. Total costs are based on unit cost rates per tonne mill feed multiplied by the total tonnes of mill feed (4.37Mt).

Table 22-2: Summary of Base Case Capital and Operating Costs

Parameter	Value	Unit
Unit Operating Costs		
Mining	$3.55	/t mined
Mining	$47.18	/t mill feed
Milling	$28.53	/t mill feed
G & A	$17.74	/t mill feed
Concentrate transportation	$7.13	/t mill feed
Total	*$104.12*	*/t mill feed*
Overall Project Costs		
Total Mining Cost	179	CAD$ million
Total Milling Costs	125	CAD$ million
Total G&A Costs	78	CAD$ million
Total Operating costs	381	CAD$ million
Total Concentrate Transport Costs	31	CAD$ million
Total Operating costs plus Transport	413	CAD$ million
Total sustaining capital costs	18	CAD$ million
Total capital costs	163	CAD$ million
Total operating and capital costs	594	CAD$ million

Similarly, a summary of the base case revenues used in the economic analysis exercise is shown in Table 22-3 below.

Table 22-3: Summary of Base Case Revenues

Parameter	Value	Unit
Project Revenue, Profit and Pre- / Post-Tax Cash Flows		
Concentrate sales revenue	$1,302.3	CAD$ million
Concentrate transportation costs	$31.1	CAD$ million
Net operating revenue	$1,271.2	CAD$ million
Operating and sustaining capital costs	$398.6	CAD$ million
EBITDA	$872.6	CAD$ million
Payable taxes	$75.5	CAD$ million
Net profit after taxes (NPAT)	$797.1	CAD$ million
Total pre-production capital costs	$163.1	CAD$ million

Finally, a summary of the pre- and post-tax economic analysis results is shown in Table 22-4 below.

Table 22-4: Summary of Pre- and Post-tax Economic Analysis Results

Parameter	Value	Unit
Economic Analysis Results		
Discount Rate	8	%
Pre-Tax Cashflow	$709.5	CAD$ million
Pre-Tax NPV	$385.1	CAD$ million
Pre-Tax IRR	46.5	%
Post-Tax Cashflow	$634.0	CAD$ million
Post-Tax NPV	$342.9	CAD$ million
Post-Tax IRR	44.3	%

The overall cash flow analysis is shown in Table 22-5.

22.6 Sensitivity Analysis

22.6.1 Pre-tax Basis

Sensitivity analyses were performed on the pre-tax cash flow model to determine the relative influence of fluctuations in lithium metal prices, process plant recoveries, exchange rate, operating costs and capital costs on overall project economics.

The pre-tax Net Present Value (NPV) of the project was found to most sensitive to the lithium concentrate price, process recovery and US to Canadian exchange rate as shown in Figure 22-1.

Similarly, the pre-tax Internal Rate of Return (IRR) of the project was also found to be most sensitive to the lithium concentrate price, process recovery and US to Canadian exchange rate, with slightly less sensitivity to the capital costs as shown in Figure 22-2.

22.6.2 Post-tax Basis

On a post-tax basis, the Raleigh Lake project NPV was found to also be most sensitive to the lithium concentrate price, process recovery and US to Canadian exchange rate as shown in Figure 22-3.

Also on a post-tax basis the Raleigh Lake project IRR was found to be most sensitive to the lithium concentrate price, process recovery, US to Canadian exchange rate and slightly less sensitive to capital costs as shown in Figure 22-4.

Table 22-5: Overall Cashflow Analysis

Item	Units	Value	-2	-1	1	2	3	4	5	6	7	8	9	Total
Production Physicals														
Mineral Resource Tonnes Mined	t		-	54,037	324,183	539,881	540,107	539,899	539,895	540,305	539,713	540,312	208,721	4,367,053
Mineral Resource Grade Mined	% Li_2O		-	0.67	0.67	0.71	0.94	0.83	0.79	0.49	0.54	0.68	0.65	
Mineral Resource Processed	t		-	-	378,220	539,881	540,107	539,899	539,895	540,305	539,713	540,312	208,721	4,367,053
Mineral Resource Processed Grade	% Li_2O				0.67	0.71	0.94	0.83	0.79	0.49	0.54	0.68	0.65	
Open Pit Waste Mined	t		-	7,572,425	8,641,731	8,930,422	9,147,177	8,855,024	4,651,476	2,979,449	1,530,607	864,285	509,626	53,682,222
Li_2O Mill Feed	t	-	-	-	378,220	539,881	540,107	539,899	539,895	540,305	539,713	540,312	208,721	4,367,053
Metal Content In-situ														
Li_2O	t			360	2,173	3,839	5,061	4,484	4,276	2,642	2,888	3,654	1,357	30,734
Grade														
Li_2O grade	%			0.67	0.67	0.71	0.94	0.83	0.79	0.49	0.54	0.68	0.65	0.58
Processing														
Processed Tonnage				-	378,220	539,881	540,107	539,899	539,895	540,305	539,713	540,312	208,721	4,367,053
Process Recovery	%			81%	81%	81%	81%	81%	81%	81%	81%	81%	81%	
Li_2O Concentrate Grade	%			6%	6%	6%	6%	6%	6%	6%	6%	6%	6%	
Tonnes Li_2O Concentrate Produced	t			-	34,194	51,828	68,321	60,532	57,726	35,668	38,988	49,323	18,324	414,904
Revenue (CAD$ million)														
Li_2O Concentrate Price	CAD$/t	$3,139		3,139	3,139	3,139	3,139	3,139	3,139	3,139	3,139	3,139	3,139	
Li_2O Concentrate Revenue	CAD$ million			-	107.3	162.7	214.4	190.0	181.2	112.0	122.4	154.8	57.5	1302.3
Li_2O Concentrate Offsite Costs*	CAD$ million			-	2.6	3.9	5.1	4.5	4.3	2.7	2.9	3.7	1.4	31.1
Net Revenue	CAD$ million			-	104.8	158.8	209.3	185.5	176.9	109.3	119.4	151.1	56.1	1271.2
Unit Net Revenue	CAD$/tonne processed			-	323.2	294.1	387.5	343.5	327.6	202.3	221.3	279.7	269.0	
Operating Cost & Sustaining Capital (CAD$ million)														
Mine Operating Costs	CAD$ million				23.4	23.6	23.8	23.6	21.5	19.6	18.9	16.2	8.4	179.0
Processing Costs	CAD$ million				14.7	14.7	14.7	14.7	14.7	14.7	14.7	14.7	7.3	124.6
G&A Costs	CAD$ million				9.1	9.1	9.1	9.1	9.1	9.1	9.1	9.1	5.0	77.5
Total Operating Costs	CAD$ million				47.1	47.4	47.5	47.3	45.2	43.3	42.6	39.9	20.8	381.0
Total Sustaining Capital Costs	CAD$ million				2.9	5.1	1.5	1.5	1.5	1.5	1.5	1.5	0.6	17.5
Total Opex plus Sustaining Capex	CAD$ million				50.0	52.4	49.0	48.8	46.7	44.8	44.1	41.4	21.4	398.6
Unit Operating Cost	CAD$/tonne processed				132.3	97.1	90.6	90.4	86.5	82.9	81.7	76.6	102.6	91.3

Item	Units	Value	-2	-1	1	2	3	4	5	6	7	8	9	Total
Cashflow Model														
EBITDA (Operating Profit)	CAD$ million				54.7	106.4	160.4	136.7	130.2	64.5	75.4	109.7	34.7	872.6
Ontario Mining Tax - Payable	CAD$ million				0.0	0.0	7.1	8.9	8.4	1.2	2.4	6.5	1.4	35.9
Canada Mining Tax - Payable	CAD$ million				0.0	0.0	3.8	5.7	8.7	1.7	5.5	12.2	2.1	39.7
Net Profit after Tax (NPAT)	CAD$ million		0.0	0.0	54.7	106.4	149.5	122.1	113.1	61.5	67.5	91.1	31.3	797.1
Capital Expenditures	CAD$ million		90.0	73.1										163.1
Pretax Project Cashflow	CAD$ million		-90.0	-73.1	54.7	106.4	160.4	136.7	130.2	64.5	75.4	109.7	34.7	709.5
Cumulative Pre-tax Project Cashflow	CAD$ million		-90.0	-163.1	-108.4	-2.0	158.4	295.0	425.2	489.7	565.1	674.8	709.5	
Post tax Project Cashflow	CAD$ million		-90.0	-73.1	54.7	106.4	149.5	122.1	133.1	61.5	67.5	91.1	31.3	634.0
Cumulative Post-tax Project Cashflow	CAD$ million		-90.0	-163.1	-108.4	-2	147.5	269.5	382.6	444.2	511.6	602.7	634.0	
NPV Calculation														
Year			1	2	3	4	5	6	7	8	9	10	11	
Discount Factor		8%	0.93	0.86	0.79	0.74	0.68	0.63	0.58	0.54	0.50	0.46	0.43	
Pretax Project Discounted Cashflow	CAD$ million	385.1	-83.4	-62.6	43.4	78.2	109.1	86.1	75.9	34.8	37.7	50.8	14.9	385.1
Pretax Project Cashflow %IRR	%	46.5%												
Post tax Project Discounted Cashflow	CAD$ million	342.9	-83.4	-62.6	43.4	78.2	101.7	76.9	66.0	33.3	33.7	42.2	13.4	342.9
Post tax Project Cashflow %IRR	%	44.3%												

**Concentrate transport cost.*

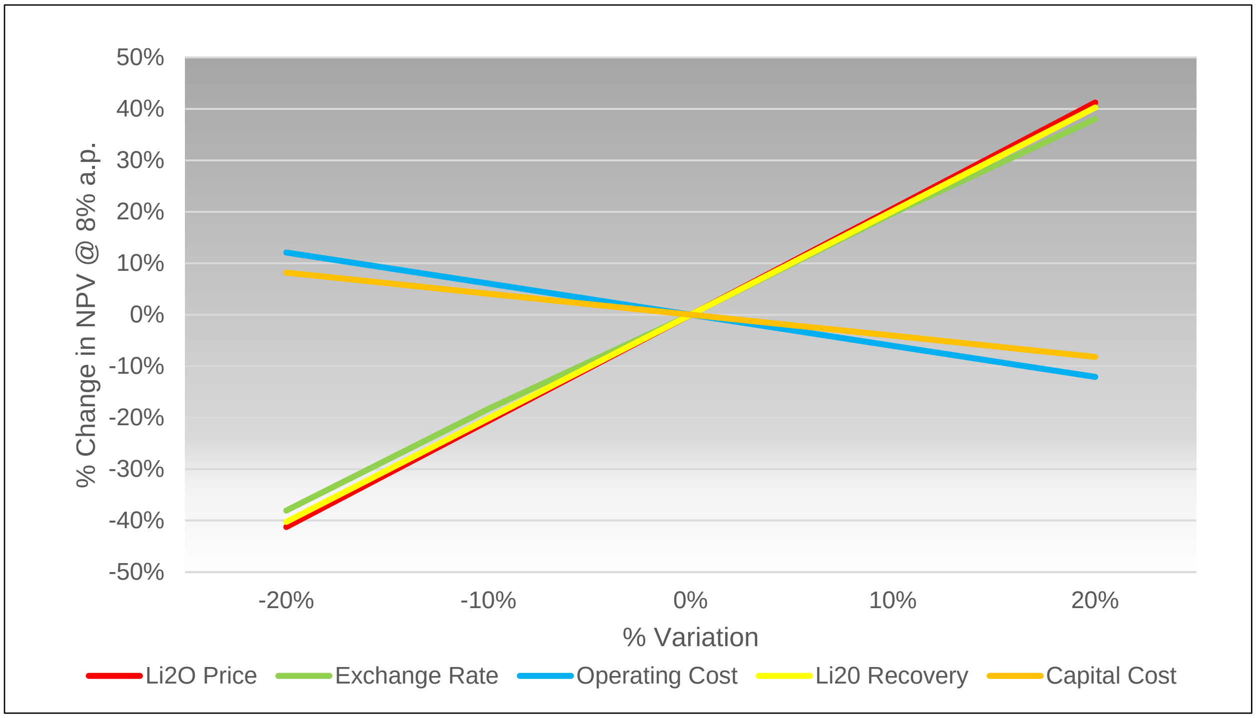


Figure 22-1: Pre-tax NPV Sensitivity – Discount Rate 8% a.p.

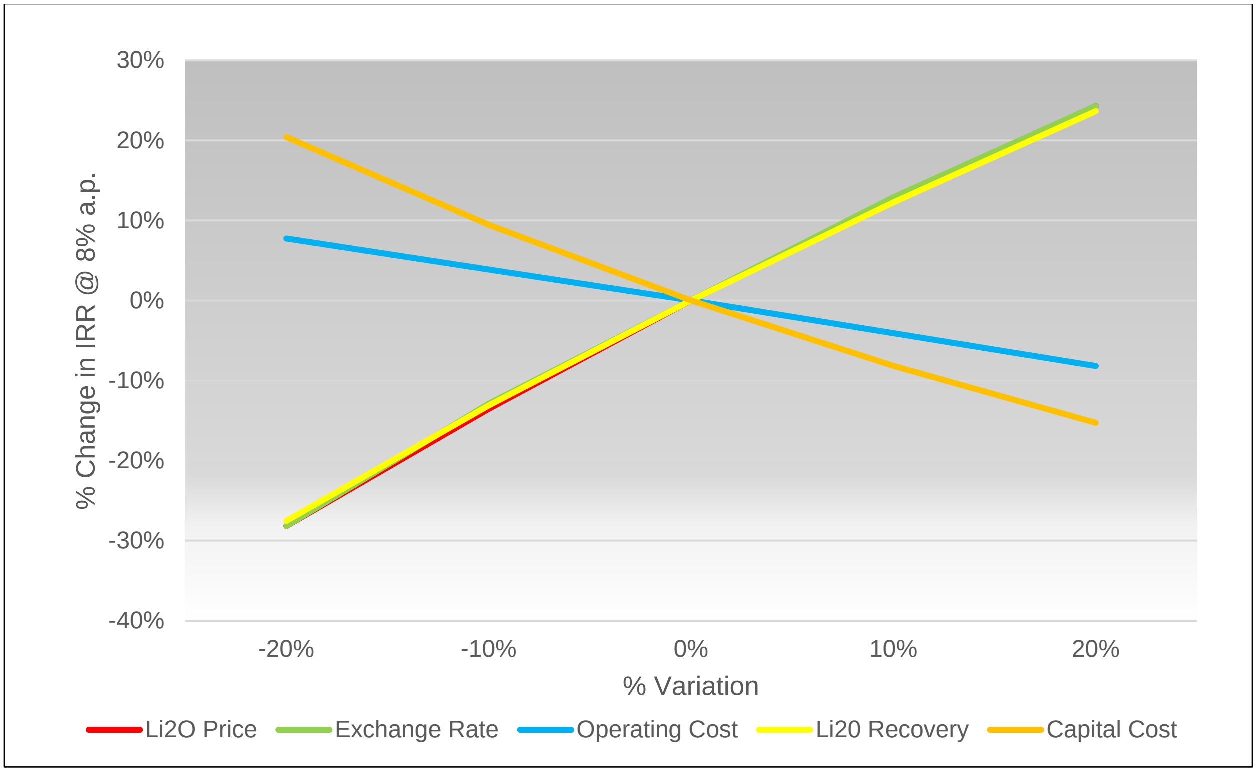


Figure 22-2: Pre-tax IRR Sensitivity

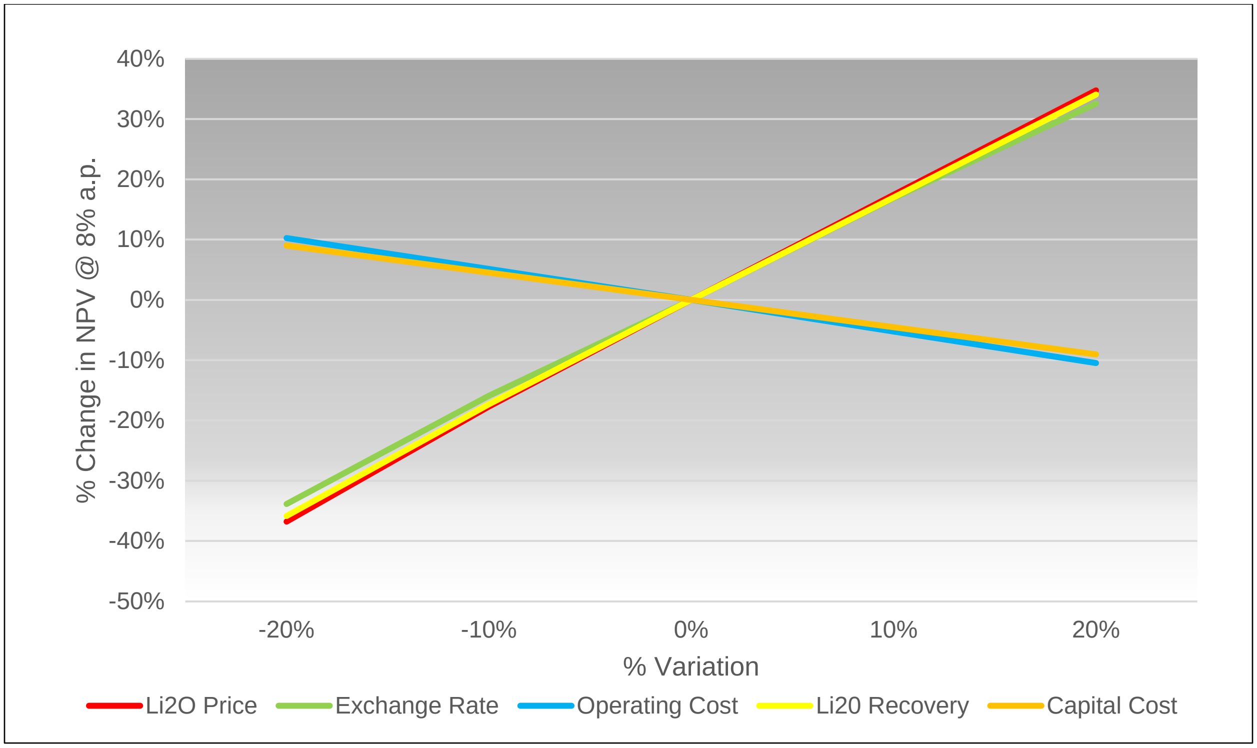


Figure 22-3: Post-tax NPV Sensitivity – Discount Rate 8% a.p.

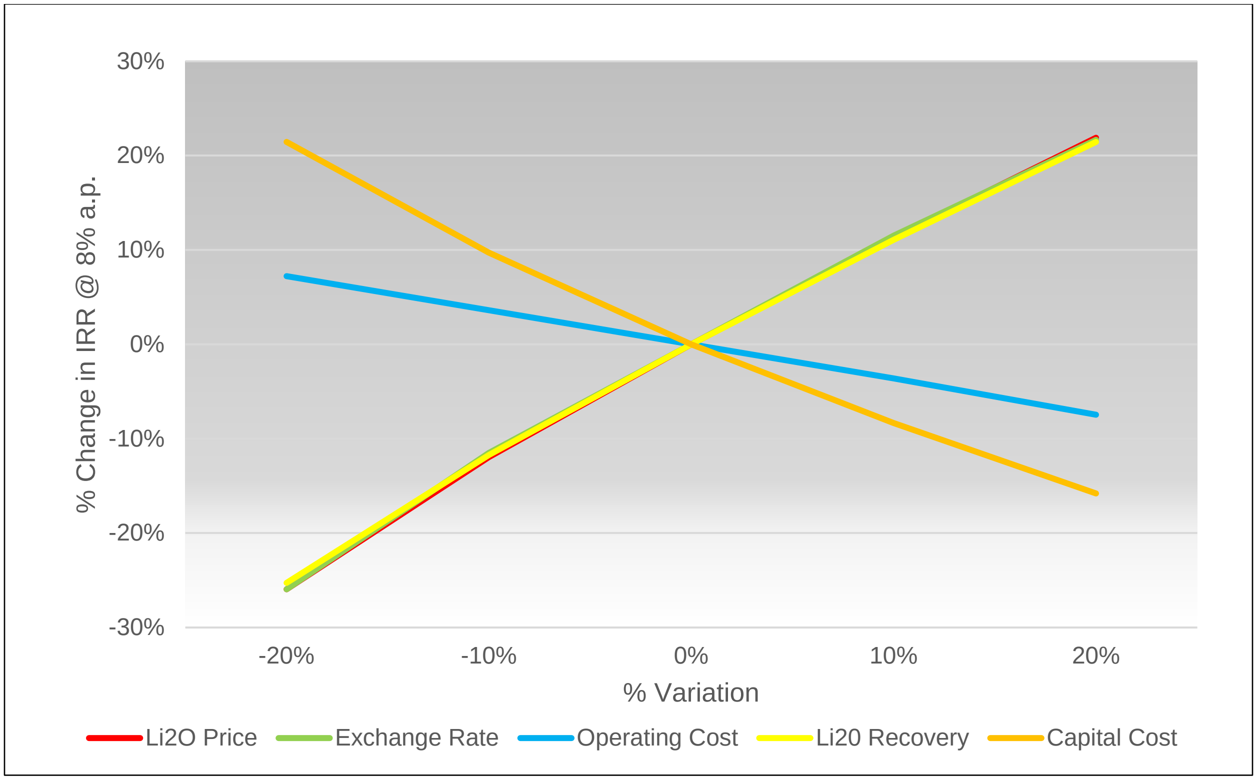


Figure 22-4: Post-tax IRR Sensitivity

22.6.3 Other Sensitivities and Risks

As with all mining projects the projections of NPV, IRR and payback period depend critically on the input assumptions. International Lithium Corp.'s management has a strong background in risk management and has asked that other specific scenarios be examined on an individual basis as presented below.

The key risk factors for the economics of this project are as follows:

- Spodumene concentrate price;
- Lithium spodumene concentrate process recovery factor;
- Overall project capital costs; and
- USD/CAD exchange rate – although the sensitivity analysis indicates that the Raleigh Lake project is sensitive to this key factor it has also stayed in the range of $1.20-$1.40 Canadian to US dollars over the past three years and is therefore not considered to be a high risk factor.

Although it is not possible or useful to model every possible scenario Table 22-6 contains anticipated project NPV and IRR values that would result from specific price points for the sale of the spodumene concentrate produced at Raleigh Lake.

Table 22-6: Project Economic Indicators at Specific Price Points

Spodumene Concentrate 6% Li_2O Price CAD$/tonne	Notes	Pre-tax Economics		Post-tax Economics	
		NPV @ 8% p.a. CAD$ million	IRR (% p.a.)	NPV @ 8% p.a. CAD$ million	IRR (% p.a.)
CAD$ 3,139	USD 2,325/ tonne Base Case Assumption	385.1	46.5%	342.9	44.3%
CAD$ 5,453	USD 4,039/ tonne Price used by Critical Elements for Rose Lithium Project June 2023	970.7	83.2%	781.5	76.5%
CAD$ 4,350	USD 3,210/tonne Market Price on Sep. 1, 2023	691.6	67.3%	691.6	62.1%
CAD$ 2,500	Decline	223.5	33.10%	215.5	32.7%
CAD$ 2,100	Significant Decline	122.3	23.2%	120.1	23.0%
CAD$ 1,700	Serious Decline	21.0	11.0%	21.0	11.0%
CAD$ 1,445	USD 1,070/ tonne no longer profitable	-43.5	0.80%	-43.5	0.80%

23. ADJACENT PROPERTIES

Several companies have staked claims adjacent to the ILC claims, including Ambershaw Metallics Inc., Lion Rock Resources, Portofino Resources Inc., and Grid Metals (Figure 23-1).

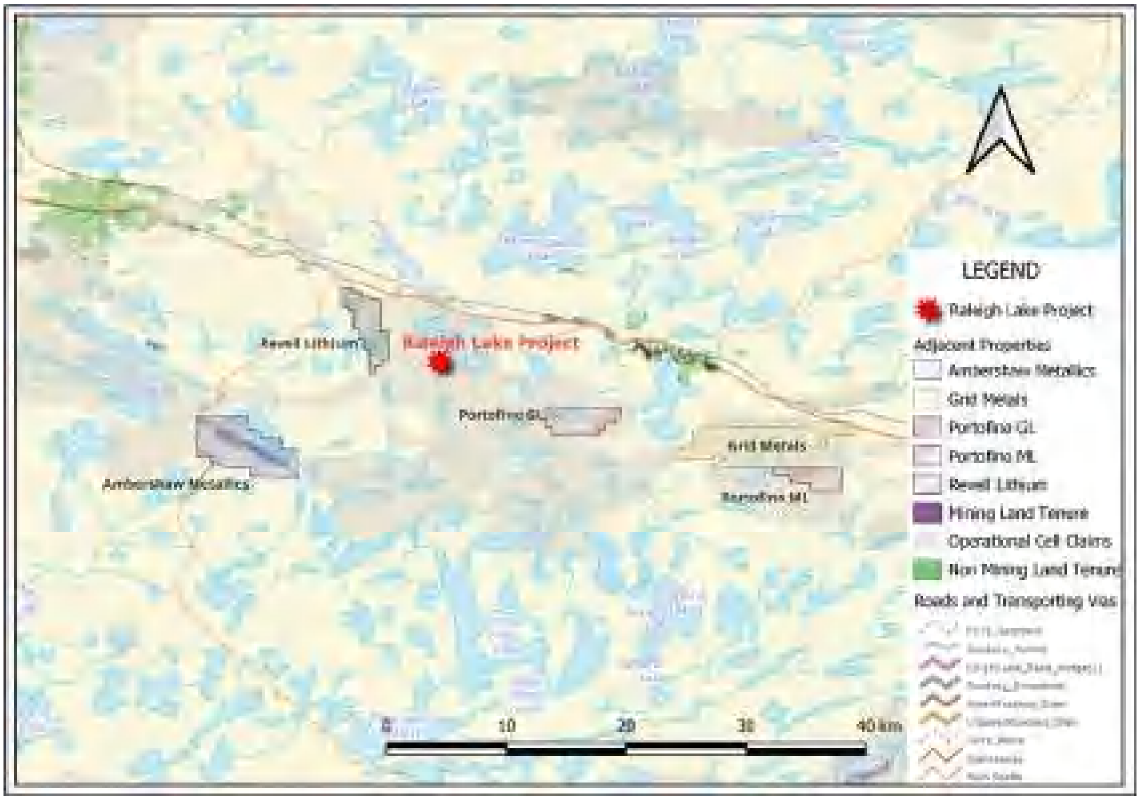


Figure 23-1: Adjacent Properties to the Raleigh Lake Lithium Project

Ambershaw Metallics Inc. (AMI) owns an iron exploration property adjacent and west-southwest of ILC's Raleigh Lake Lithium Project. The Ambershaw Metallics Inc. property includes over 16,000 hectares of mining claims. The AMI property is situated roughly 20 km to the west-southwest of ILC's Raleigh Lake lithium project and approximately 40 km west-southwest of the town of Ignace, Ontario near Bending Lake and can be accessed via Highway 622.

The Revell Lithium Property acquired recently by Lion Rock Resources Inc is located approximately 5 km East to the Raleigh Lake Lithium Project.

Portofino Resources Inc. possesses two adjacent Lithium exploration properties neighboring ILC's Raleigh Lake Lithium Project. In December 2021, Portofino entered into an agreement to acquire the Greenheart Lake Lithium Property and the McNamara Lake Lithium Property. The Greenheart Lake Property comprises 3 claims (60 cells) covering approximately 1,200 hectares, while the McNamara Lake Property includes 3 claims (56 cells) covering around 1,120 hectares. These properties are strategically positioned in the Balmoral and McNamara Lake regions of northwestern Ontario, within 15 km of the town of Ignace. Accessible via established logging roads just off the Trans-Canada highway, the Greenheart Lake Property is located 10 km southeast of International Lithium's Raleigh Lake lithium project and 15 km northwest of Grid Metals Campus Creek lithium project. The McNamara Lake Property adjoins the southern claim boundary of Grid's Campus Creek property. Portofino Resources Inc. has the option to acquire a 100% interest in both the Greenheart Lake and McNamara Lake claim groups.

Grid Metals recently acquired 100% of an exploration property adjacent to ILC's Raleigh Lake Lithium Project. The property hosts at least one highly fractioned and rare metal-enriched pegmatite dyke as evidenced by Grid Metals' preliminary surface grab sampling revealed the presence of premium-quality lithium and notably abnormal cesium, rubidium, and tantalum.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

The project execution plan has not yet been created in detail, the project execution plan will be developed during the pre-feasibility stage of the project after any necessary trade-off studies have been executed in order to consider only a few major project execution options with regards to mine planning and mineral processing.

24.1.1 Health, Safety, and Environment

ILC will be required to undertake desktop and field studies to support the EA and permit applications. The scope of studies could include, but may not be limited to:

- Meteorology and air quality;
- Soils;
- Vegetation and wetlands;
- Wildlife and wildlife habitat;
- Fish and fish habitat;
- Surface and groundwater quality;
- Surface and groundwater quantity;
- Metal leaching and acid rock drainage potential (static and kinetic testing);
- Socio-economics and land use; and
- Archaeological and cultural heritage investigations.

24.1.2 Community and Indigenous Engagement

ILC has commenced engagement activities with communities and Indigenous communities in the vicinity of the Project and will continue to do so throughout Project advancement and future operations. ILC will develop communication plans and Project materials to assist in discussions, identifying and resolving issues, and maintain records of engagement as required by regulatory authorities.

24.1.3 Timeline

After this PEA report is published, it would be assumed that International Lithium would continue with exploration drilling in order to upgrade the resource to indicated or better such that a PFS could be undertaken and a Mineral Reserve Estimate stated. Should the economics of the project appear favourable within the context of the PFS, then a FS study would be undertaken, and all permitting applications and requirements finalized.

Once all governmental authorizations are received, the project would then to detailed engineering, to be ready when the environmental certificates of authorization will be required and in parallel International Lithium would work on the development of construction packages and source the purchase of all major equipment required for the project. Any required detailed engineering not included in the FS would begin immediately after the completion of the FS.

Mine site construction work would proceed once the minimum required detailed engineering is completed and pre-stripping could also begin. The mill start-up would be scheduled for approximately 18-24 months after the site construction is initiated with a mill ramp up over the initial 6-12 months of operation.

24.1.4 Execution Strategy

International Lithium will assemble an owner's team to manage the detailed engineering, procurement, and construction. The owners team will contract consultants to conduct the detailed engineering for each discipline, as required.

24.1.4.1 Engineering

Engineering work performed to date is primarily focussed on mine engineering and mineral processing at a conceptual PEA level.

Mine engineering work has focused on the mine pit optimisation, scheduling, and design.

Mineral processing engineering work has focussed on providing a processing option typical for the mineral type.

Substantial additional and detailed engineering work is required at the next stages of the project and prior finalizing deterioration of project viability.

24.1.4.2 Procurement and Contracts

The cost and revenue assumptions in this study are based on benchmarks and scaling. There are no binding procurement or offtake contracts in place to support the cost and revenue parameter assumptions.

24.1.5 Construction Labour Requirements

The labour requirements will target sourcing the local labour force as much as possible in order to benefit the local community but also to minimize travel, accommodation, and logistics to bring in labour from outside the immediate area, outside of the province, or the least favourable option economically and logistically of brining in persons from outside of the country.

24.1.6 Camp

The plan is to not include a camp whereas the workforce will come from and / or be accommodated in Ignace.

24.1.7 Mine Development / Infrastructure

International Lithium will prioritize site preparation and installation of temporary infrastructure to initiate the mill construction as early as possible.

While the main access road is already in place but requiring some upgrade, where required, temporary roads will be established using exploration roads.

The industrial pads for the process plant and other infrastructure will be cleared and leveled, and excavation of waste material from the starter pit will be initiated early to provide aggregates for the infrastructure.

Permanent roads and other infrastructure not essential from the start will be initiated once mill construction has begun,

24.1.8 Construction Equipment and Equipment Lead Time

Consideration will need to be made for the availability of contractor construction equipment availability and potentially long lead times for critical plant and mobile equipment.

24.1.9 Communication

The project site is located within range of commercial telecommunications providers.

24.1.10 Construction Power

Due to proximity to the grid, the project assumes that power will be connected and ready to use right from the beginning of the construction phase.

24.2 Risk Management

At the PEA level of study there are many technical risks that could affect the technical feasibility and economic outcome of the Project. It is not permitted to suggest economic viability of a project at the PEA level of study as stated in NI 43-101 guidelines for mineral resource disclosure.

Risks common to most mining projects are typically mitigated with engineering, planning, and pro-active management.

External risks beyond the control of the Project include things such as wildfires, political conditions, mineral prices, exchange rate, regulations and government legislation.

At the next stage of the project a Risk Register will be created to outline evaluate all potential foreseeable risks and recommended mitigation if feasible.

Some additional common sources of risk include:

- Permit approval delays
- Inflation of Capex and Opex costs
- Overrun of the project cost / Estimate accuracy
- Supply chain disruptions
- Mineral recovery underperformance
- Mineral Reserve reconciliation issues
- Lack of skilled labour and consequent personnel logistics
- Unpredictable accommodation and flight costs for personnel not local to project
- Risk of a severe injury or fatality / subsequent shut down for investigation
- Uncertainty of Social Licence
- Unforeseen Natural Events such as severe precipitation or draught for example.

25. INTERPRETATION AND CONCLUSIONS

Based on the data acquired from previous operators and the three-phase drilling program by International Lithium (ILC), the Mineral Resource Estimate (MRE) findings in this technical report demonstrate the Project's technical value. International Lithium plans to continue its drilling operations on the site, with the aim of upgrading the current mineral resource category and identifying additional economic resources.

25.1 Economic Conclusions

The economic analysis of the project vis-à-vis the discounted cashflow model generated from the conceptual mine plan produced a pre-tax cashflow of CAD$709.5, post-tax NPV_8 of CAD$342.9 and had a post-tax IRR of 44.3%.

Table 25-1: Summary of Pre- and Post-tax Economic Analysis Results

Parameter	Value	Unit
Economic Analysis Results		
Discount Rate	8	%
Pre-Tax Cashflow	$709.5	CAD$ million
Pre-Tax NPV	$385.1	CAD$ million
Pre-Tax IRR	46.5	%
Post-Tax Cashflow	$634.0	CAD$ million
Post-Tax NPV	$342.9	CAD$ million
Post-Tax IRR	44.3	%

25.2 Risks

25.2.1 Mineral Resources

At the time of the writing of this report there are a few elements of uncertainty that could have a substantial impact on the Mineral Resource Estimate (MRE). These elements include:

- Long-term metal price assumptions for Spodumene Concentrate SC6 where the market has seen wide variation in the price of various Li related products including Li_2O and Lithium Carbonate.
- Modifications to the input parameters for mining costs processing costs, and general administrative costs to restrict the estimation.
- Revisions in the geological interpretations of the shape of mineralization and the continuity of mineralized zones.
- Adjustments to the density values attributed to the mineralized zones.

25.2.2 Regulatory

There is material risk to the Project's realization and/or schedule due to potential regulatory delays securing the necessary permits or approvals.

26. RECOMMENDATIONS

26.1 For Mineral Resources

- The QP advises that to improve the classification of resource evaluations from inferred to indicated and from indicated to measured within the deposit, it is essential to continue with infill drilling and further initiatives directed at enhancing data precision and economic feasibility.
- To increase the mineral resources, it is necessary to continue examining the limits of Pegmatite 1 and evaluating the remote spodumene-bearing pegmatites detected during surface Litho-geochemical sampling programs.

It is recommended that the environmental, social and heritage assessment requirements, including permitting to meet Ontario provincial and Canadian federal regulations, be confirmed with relevant agencies and be completed.

ILC should continue active engagement with the public and Indigenous groups.

26.2 For Metallurgy

It is recommended to collect and perform HLS testing on variability samples and mini-bulk samples to optimize the beneficiation process flowsheet and crush size and perform confirmatory HLS and pilot scale DMS beneficiation testwork. Variability samples would confirm the amenability of the flowsheet to handle the effects of dilution, and predict the lithium recoveries on lower lithium head grades.

The lithium head samples have a variety of colors of minerals and densities. Therefore, the lithium head is recommended be evaluated for upgrading by ore sorting methods of colour and density (XRT).

The weight percent removal of muscovite (mica) minerals from the minus 6.3mm plus 1.0mm should be evaluated and confirmed with REFLUX™ classifier testing.

It is recommended to perform comminution testing of: Crusher Work Index (CWi) Abrasion Index testing (Ai) and Bond Ball Mill Grindability Work Index (BWi).

Solid liquid separation testing of the lithium fines fractions are recommended with additions of flocculant.

Modified acid base accounting testing is recommended to determine if the waste rock and HLS floats (DMS floats) are suitable for use in highway construction and mine road construction.

Basic flotation test program is recommended to conduct a trade-off study to potentially enhance the overall process performance parameters and achieve overall superior recoveries and stable final product grades.

It is recommended to continue the monitoring and removal of deleterious of Be and Fe from SC6 spodumene concentrate by magnetic separation.

The combined HLS middling rejects and - 1mm fines fraction contained 16.2% of the lithium distribution and graded 1.19% Li_2O. The HLS middlings are recommended to be crushed to minus 1mm and blended with the –1 mm fines and tested as potential feed to make additional SC6 concentrate to increase lithium recovery.

26.3 For Mining

ERM recommends further investigation into the possibilities of improvements in the areas of:

1. Mine Design and Scheduling beyond the level of detail in the conceptual design and schedule presented in this PEA Study.
2. Geotechnical review and testing to prove the waste and tailings suitability for co- disposal as proposed in this study/report.
3. Geotechnical review and testing to confirm pit design parameters to be used for an operational mine design and pushback designs.
4. Capex and Opex review using budget quotes or contractually enforceable quotes where available especially for major plant and equipment packages as well as contract miner and construction rates.
5. Hydrology and hydrogeological studies in order to de-risk the possibility of any water related fatal flaws.
6. ERM understands that there are significant quantities of rubidium bearing microcline feldspar material contained within the lithium constrained RPEEE pit shell. None of this material was included in the economic analysis due to uncertainties regarding the achievable selling price of rubidium given its currently limited commercial uses. However, should demand for rubidium ever sufficiently increase it would add significant economic value to the Raleigh Lake project.

The assumption in this PEA is that the rock containing rubidium would be segregated and stored pending future offtake.

As a consequence of the investigation and modelling required to determine the revenue potential and the consequent economics, no estimate has yet been made of any revenues or of the economic potential of the rubidium at Raleigh Lake. Consequently, this PEA deals with the lithium resource only.

ERM recommends monitoring global demand and re-asses regularly if there is the potential for an economical offtake agreement or other arrangement to monetize this asset, which may have a material impact on a revised PEA where Rubidium provides significant revenue.

27. REFERENCES

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28. ABBREVIATIONS AND UNITS OF MEASUREMENT

°	degrees
°C	degrees Celsius
µm	micron
1D, 2D, 3D	one-dimensional, two-dimensional, three-dimensional
Acme	Acme Analytical Labs Ltd
ActLabs	Activation Laboratories Ltd
Ag	Silver
ARD	acid rock drainage
Au	Gold
CAPEX	Capital cost
B	Boron
Be	Beryllium
CDN	CDN Resource Laboratories Ltd
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetres
COA	certificate of analysis
CRM	certified reference material
Cs	Cesium
CSA Global	CSA Global Consultants Canada Limited
Cu	Copper
CV	coefficient of variation
ft	feet (or foot)
g, g/L, g/t	grams, grams per litre, grams per tonne
Ga	Gallium
GPS	global positioning system
ha	hectares
Hf	Hafnium

HLS	Heavy liquid separation
ICP	inductively coupled plasma
ICP-AES	inductively coupled plasma-atomic emission spectroscopy
ICP-MS	inductively coupled plasma-mass spectrometry
ID2	inverse distance squared
IDW2	inverse distance weighting to the power of two
ILC	International Lithium Corporation
IP	induced polarisation
JV	joint venture
kg	kilograms
kHz	kilohertz
km, km^2	kilometres, square kilometres
KNA	kriging neighbourhood analysis
kph	kilometres per hour
kV	Kilovolts
LCT	Lithium-Cesium-Tantalum
LOM	Life of Mine
Li	Lithium
lb	pound(s)
m	metre(s)
M	million(s)
MECP	(Ontario) Ministry of the Environment, Conservation and Parks
Mlb	million pounds
mm	millimetres
MNDMNRF	(Ontario) Ministry of Northern Development, Mines, Natural Resources and Forestry
Mn	Manganese
Moz	million ounces
MRE	Mineral Resource Estimate

Mt	million tonnes
Nb	Niobium
NI 43-101	National Instrument 43-101 – Standards for Disclosure for Mineral Projects
NN	nearest neighbour
NSR	net smelter return
OK	ordinary kriging
OREAS	ORE Research and Exploration of Australia
OPEX	Operating cost
oz	ounce(s)
P	Phosphorus
ppm	parts per million
QAQC	quality assurance and quality control
Q-Q	quantile-quantile
Rb	Rubidium
RPEEE	reasonable prospects for eventual economic extraction
SD	standard deviation(s)
SGS	SGS Laboratories
Sn	Tin
SUSEX	Sustaining Capital Costs
t	tonne(s)
Ta	Tantalum
TSX-V	TSX Venture Exchange
UTM	Universal Transverse Mercator
WLON	Wabigoon Lake Ojibway Nation

29. CERTIFICATES

29.1 Certificate of Author – Nigel Fung, P.Eng.

I, Nigel Fung, do hereby certify that,

- I am employed as a Partner and Principal Mining Engineer by **ERM Consultants Canada Limited, 2010-120 Adelaide Street West, Toronto, Ontario, Canada, M5H 1T1** and carried out this assignment for ERM Consultants Canada Ltd.
- I graduated with a B.Eng. degree in Mining Engineering from McGill University in Montreal, Canada (2001).
- I am a member in good standing of the Professional Engineers of Ontario, License 100173276.
- I have worked as a mining engineer for 12 years and in the mining industry for 25 years (since 1998).
- I have read the definition of "Qualified Person" set out in Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in Regulation 43-101/NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of Regulation 43-101/NI 43-101.
- I have not visited the property.
- I am responsible for sections 1, 2, 3, 16, 25, 26, and 27, of the technical report titled "*The Raleigh Lake Project NI 43-101 Technical Report - PEA"* (following the NI 43-101 Standards of Disclosure and Form 43-101F1) (the "Technical Report"), effective date of January 18, 2024 and signature date dated of January 18, 2024.
- I have not had prior involvement with the property that is the subject of the Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer, applying all of the tests in section 1.5 of Regulation 43-101.
- I have read Regulation 43-101 respecting standards of disclosure for mineral projects, as well as Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: January 18, 2024

Signature Date: January 18, 2024

29.2 Certificate of Author – Garth Matti Liukko, P.Eng.

I, Garth Matti Liukko, do hereby certify that,

- I am employed as a Principal Mining Engineer by **ERM Consultants Canada Limited, 2010-120 Adelaide Street West, Toronto, Ontario, Canada, M5H 1T1** and carried out this assignment for ERM Consultants Canada Ltd.
- I graduated with a B.Eng. degree in Mining Engineering from Laurentian University in Sudbury, Ontario, Canada (1991).
- I am a member in good standing of the Professional Engineers of Ontario, License 90533399.
- I have continuously worked as a mining engineer for 32 years since graduation.
- I have read the definition of "Qualified Person" set out in Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in Regulation 43-101/NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of Regulation 43-101/NI 43-101.
- I have not visited the property.
- I am responsible for the preparation of sections 16, 18, 21, 22 and co-authored 25 and 26 of the technical report titled "*The Raleigh Lake Project NI 43-101 Technical Report - PEA"* (compliant with Regulation 43-101/NI 43-101 and Form 43-101F1) (the "Technical Report"), effective date of January 18, 2024 and signature date dated of January 18, 2024.
- I have not had prior involvement with the property that is the subject of the Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer applying all of the tests in section 1.5 of Regulation 43-101.
- I have read Regulation 43-101 respecting standards of disclosure for mineral projects, as well as Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: January 18, 2024

Signature Date: January 18, 2024

29.3 Certificate of Author – Harold Rolf Schmitt, P.Geo.

I, Rolf Schmitt, do hereby certify that,

- I am employed as a Technical Director - Permitting by **ERM Consultants Canada Limited**, **#10001100 Melville Street, Vancouver, British Columbia, Canada, V6E 4A6** and carried out this assignment for ERM Consultants Canada Ltd.
- I graduated from the University of British Columbia – Bachelor of Science (B.Sc.) Geology (1977), and a Master of Science (M.Sc.) Regional Planning (1985), and University of Ottawa - Master of Science (M.Sc.) Exploration Geochemistry (1993).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License 19824.
- I have practiced my profession for 46 years since graduation; 6 years in mineral exploration, 22 years in government mining regulation and geochemical research, and 20 years as a senior mining and natural resource regulatory consultant (since 2005).
- I have read the definition of "Qualified Person" set out in Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in Regulation 43-101/NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of Regulation 43-101/NI 43-101.
- I have not visited the property.
- I am responsible for section 20 of the technical report titled "*The Raleigh Lake Project NI 43-101 Technical Report - PEA"* (following the NI 43-101 Standards of Disclosure and Form 43-101F1) (the "Technical Report"), effective date of January 18, 2024 and signature date dated of January 18, 2024.
- I have not had prior involvement with the property that is the subject of the Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer, applying all of the tests in section 1.5 of Regulation 43-101.
- I have read Regulation 43-101 respecting standards of disclosure for mineral projects, as well as Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: January 18, 2024

Signature Date: January 18, 2024

29.4 Certificate of Author – Richard Wagner, P.Eng.

I, Richard Wagner, do hereby certify that,

- I am employed as a Principal Metallurgist by **ERM Consultants Canada Limited, 2010-120 Adelaide Street West, Toronto, Ontario, Canada, M5H 1T1** and carried out this assignment for ERM Consultants Canada Ltd.
- I graduated with a B.SC. degree in Mining Engineering, Mineral Processing Option from Queens University in Kingston, Canada (1979).
- I am a member in good standing of the Professional Engineers of Ontario, License 48460505, and have been a P. Eng since 1982.
- I have worked as a mining metallurgical engineer in the mining industry for 43 years (since 1980).
- I have read the definition of "Qualified Person" set out in Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in Regulation 43-101/NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of Regulation 43-101/NI 43-101.
- I have not visited the property.
- I am responsible for section 13, of technical report titled "*The Raleigh Lake Project NI 43-101 Technical Report - PEA"* (following the NI 43-101 Standards of Disclosure and Form 43-101F1) (the "Technical Report"), effective date of January 18, 2024 and signature date dated of January 18, 2024.
- I have not had prior involvement with the property that is the subject of the Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer, applying all of the tests in section 1.5 of Regulation 43-101.
- I have read Regulation 43-101 respecting standards of disclosure for mineral projects, as well as Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: January 18, 2024

Signature Date: January 18, 2024

29.5 Certificate of Author – Georgi Doundarov, P.Eng., PMP, CCP

I, Georgi Doundarov, do hereby certify that,

- I am a CEO of Magemi Mining Inc. and an associate to **ERM Consultants Canada Limited, 2010120 Adelaide Street West, Toronto, Ontario, Canada, M5H 1T1** and carried out this assignment for ERM Consultants Canada Ltd.
- I am a graduate of the University of Mining and Geology, 1996 with a M.Sc. degree in Mineral Processing and Metallurgy as well as a graduate from the Yokohama National University, Yokohama, Japan, 2005 with a M.Sc. degree in Infrastructure Management - Mineral Processing and Metallurgy.
- I am a Member of the Professional Engineers Ontario (PEO) and registered as a Professional Engineer in the province of Ontario with a number 100107167.
- I have worked as a metallurgical engineer and project/study manager for a total of over 30 years since my graduation.
- I have read the definition of “Qualified Person” set out in Regulation 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in Regulation 43-101/NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of Regulation 43-101/NI 43-101.
- I have not visited the property.
- I am responsible for sections 13, 17, and 21, of technical report titled “*The Raleigh Lake Project NI 43-101 Technical Report - PEA”* (following the NI 43-101 Standards of Disclosure and Form 43-101F1) (the “Technical Report”), effective date of January 18, 2024 and signature date dated of January 18, 2024.
- I have not had prior involvement with the property that is the subject of the Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer, applying all of the tests in section 1.5 of Regulation 43-101.
- I have read Regulation 43-101 respecting standards of disclosure for mineral projects, as well as Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: January 18, 2024

Signature Date: January 18, 2024

29.6 Certificate of Author – Patrick McLaughlin, P.Geo.

I, Patrick McLaughlin, do hereby certify that,

- I am employed as a Partner and Sr. Geologist by **Coast Mountain Geological Ltd, 488-625 Howe Street, Vancouver, BC, V6C 2T6**.
- I graduated with a Bachelor of Science (Honours) in Geological Sciences from the University of Manitoba (2005).
- I am a member in good standing of the Professional Geoscientists Ontario (member #3788), and Engineers and Geoscientist British Columbia (member #41479).
- I have worked continuously upon graduation in 2005 and as a Professional Geoscientist since 2015.
- I have read the definition of "Qualified Person" set out in Regulation 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in Regulation 43-101/NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of Regulation 43-101/NI 43-101.
- I have managed all exploration activities work on the Property on behalf of International Lithium Corp. since 2016.
- I am responsible for sections 5 and 6 of technical report titled "*The Raleigh Lake Project NI 43-101 Technical Report - PEA"* (following the NI 43-101 Standards of Disclosure and Form 43-101F1) (the "Technical Report"), effective date of January 18, 2024 and signature date dated of January 18, 2024.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer, applying all of the tests in section 1.5 of Regulation 43-101.
- I have read Regulation 43-101 respecting standards of disclosure for mineral projects, as well as Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: 18 January 2024

Signature Date: 18 January 2024

29.7 Certificate of Author – Christian Ballard, P.Geo.

I, Christian Ballard, P.Geo of Thunder Bay, Ontario, do hereby certify that,

- I am a Senior Geologist with **Nordmin Engineering Ltd.** with a business address at **160 Logan Avenue, Thunder Bay, Ontario, Canada, P7A 6R1**.
- This certificate applies to the technical report titled "The Raleigh Lake Project NI 43-101 Technical Report – PEA" with an Effective Date of January 18, 2024 (the "Technical Report").
- I am a graduate of the University of Saskatchewan, 2002 with a Bachelor of Science in Geology.
- I am a member in good standing of the Association of Professional Geoscientists of Ontario, and am registered as a Professional Geoscientist, license 3025.
- My relevant experience includes 20 years of experience in operations, exploration, and resource estimation. I am a "Qualified Person" for the purposes of Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument").
- I am responsible for Sections 7 through 12 and 14 (the "Relevant Sections") of the Technical Report.
- I have read the 43-101 reporting requirements and the Relevant Sections of the Technical Report, which have been prepared in compliance with the Instrument and Form 43-101F1.
- My most recent personal inspection of the Raleigh Lake Project, located near Ignace, Ontario, Canada, was on October 18, 2022.
- I am independent of all other parties involved as defined in Section 1.5 of the Instrument.
- I prepared a Mineral Resource Estimate and technical report in April 2023 for the Project which is the subject of the Technical Report.
- As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, the Relevant Sections of the Technical Report contain all scientific and technical information relating to the Project that is required to be disclosed to make the Technical Report not misleading.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: January 18, 2024

Signature Date: January 18, 2024

APPENDIX A SGS REPORTS

LITERATURE REVIEW – RUBIDIUM, CESIUM, AND POTASSIUM EXTRACTION FROM LCT PEGMATITE DEPOSIT

prepared for

INTERNATIONAL LITHIUM

Project 19977-01 – Literature Review
October 26, 2023

NOTES

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

Table of Contents

List of Tables

List of Figures

Executive Summary

SGS has conducted a literature review of rubidium, potassium, and cesium extraction from minerals. The report is based on public information provided by International Lithium Corp (ILC). This includes press releases and Technical Report and Mineral Resource Estimate for their Raleigh Lake Project. The main element of interest is rubidium which is found to be chemically associated with microcline (K-feldspar). Secondary elements such as cesium and potassium carry certain levels of radioactivity, which also makes them target elements for extraction.

This report consists of an introduction to the Raleigh Lake deposit, sections on rubidium extraction, potassium extraction, cesium extraction, and a summary and recommendations. Based on the literature review, several processes are available to extract rubidium, potassium, and cesium. However, they are heavily dependent on the mineralogy and elemental associations. A process that works with one deposit may not work with a different deposit.

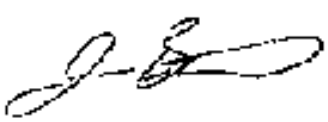
A summary of the findings and recommendations on initial testing for the rubidium-rich Raleigh Lake deposit are presented in this report. These recommendations are based on the limited mineralogical study conducted on the ore. As with other exploratory studies, careful considerations and comprehensive variability tests are necessary to obtain the optimum conditions for a specific process and specific ore deposit.

Introduction

International Lithium Corporation's (ILC's) major project is located at Raleigh Lake near Ignace, Ontario, Canada. The Raleigh Lake deposit is a rare-metal pegmatite, further categorized as the albite-spodumene sub-type. The deposit contains lithium-cesium-tantalum (LCT) pegmatites that are enriched in Li, Cs, Ta, Be, B, F, P, Mn, Ga, Rb, Nb, Sn, and Hf. The drill programs determined two possible mineral resources, one for lithium and one for rubidium. Initial mineralogy revealed significant amounts of albite and quartz in both mineral resources. Lithium is hosted in spodumene while rubidium is associated with microcline (K-feldspar).

SGS Canada Inc. (SGS) was engaged by International Lithium Corporation to conduct a literature review of possible extraction methods for rubidium, potassium, and cesium from a pegmatite deposit. In this report, possible extraction methods for rubidium from deposits that are similar to the Raleigh Lake Project are discussed. International Lithium Corporation was also looking at possible routes for potash recovery from the ore and possible cesium extraction. These two elements can carry certain levels of radioactivity, which is why their extraction may be desirable. This literature review will be a guide in deciding the methods and conditions to be tested for metal extraction from their rubidium ore. This review only includes extraction of rubidium, potassium, and cesium from the ore, and will not include downstream separation and purification processes such as precipitation, solvent extraction, ion exchange, and so on.

Eileen Espiritu, Ph.D.
Metallurgist, Extractive Metallurgy - Hydromet



James Brown, MASc, P.Eng
Manager, Extractive Metallurgy

Literature review by: Eileen Espiritu
Report preparation by: Eileen Espiritu
Reviewed by: Cheryl Mina, Chris Fleming, James Brown

Literature Review

1. The Ore Deposit

1.1. Definition of LCT Pegmatite

Pegmatite is defined as "essentially igneous rock, commonly of granitic composition, that is distinguished from other igneous rocks by its extremely coarse but variable grain-size, or by an abundance of crystals with skeletal, graphic, or other strongly directional growth-habits" (London, 2018).

London (2018) proposed several schemes for granitic pegmatites, but for the purposes of exploration assessment, two distinctions are commonly used. First are "common pegmatites", which have a simpler granite mineralogy, and second are rare-element pegmatites, which are rather more mineralogically complex. LCT pegmatites fall under the rare-element pegmatites (Bradley & McCauley, 2013), a subset of granitic pegmatites that are associated with certain granites such as quartz, feldspar, albite, and muscovite (Bradley & McCauley, 2013). "LCT pegmatites may hosts 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).

1.2. Raleigh Lake Ore Deposit

"The rare-element-bearing pegmatite dykes on the Raleigh Lake property occur in a south-striking zone, with a new trend of tantalum-mineralized albitic dykes occurring south of Raleigh Lake. The main pegmatite trend, which includes Pegmatite #1 through #3, and the Johnson Pegmatite, belong to the albite-spodumene sub-type of rare metal pegmatites" (Mclaughlin & Geo, 2019).

The Mineral Resource Estimates (MRE) for this Project are based on a drill-hole database which include 81 drill holes translated to a total of 13,821 m of diamond drilling completed between 1999 and the end of 2022, followed by a three-phase drilling program by ILC in 2022 consisting of 9,496 m of diamond drilling (Ballard, 2023).

The Project includes MRE's for both lithium and rubidium. Minor amounts of rubidium also occur throughout the lithium-bearing spodumene but are not included in the rubidium MRE. An independent MRE has been calculated for rubidium contained within microcline zones of LCT pegmatites. Rubidium, which reaches 4000 g/t, is attributed to the high modal abundance of microcline (K-feldspar) (Ballard, 2023). There is no official mineralogical study yet for these particular drill-hole samples, but for the purposes of this report, rubidium is assumed to be associated with microcline (K-feldspar) minerals.

2. Rubidium Extraction from Minerals

Rubidium is typically a disseminated element. Up to this date, no single rubidium industrial mineral has been found. Some rubidium is found with potassium-bearing minerals, mainly due to its ability to replace potassium by isomorphism (Pekov & Kononkova, 2010). The possibility that rubidium may be associated with microcline (K-feldspar) is corroborated by literature sources. According to Pekoy and Kononkoya (2010), among the minerals in LCT pegmatites, the highest rubidium concentrations occur in micas and alkaline feldspar. Before any data on natural rubidium-dominant feldspars were published in 1997, the highest rubidium concentrations among feldspars were found in a microcline from Red Cross Lake in Canada (Pekov & Kononkova, 2010).

At present, rubidium is primarily extracted from brines, and rarely from rubidium-containing ores (Liu et al., 2023). Liu et al. (2023) summarizes some of the commonly used methods of extracting rubidium from different minerals. These are the acid decomposition method, the alkali leaching method, the sulphate roasting method, the carbonate roasting method and the chlorination roasting method. One of the goals of these methods is to break the main constituent silicate minerals and transform them, usually at high temperatures, to a more reactive and easily dissociated phase. Direct sulphuric acid digestion and sulphuric acid roasting methods, which are usually employed in industrial settings, are primarily used for extraction of rubidium and cesium from spodumene, lepidolite, pollucite, and cesium garnet (Liu et al., 2023; Xing et al., 2021). These methods take advantage of the fact that these minerals are easily attacked by acid. Unfortunately, they do not work very well with K-feldspars owing to the fact that the K-feldspar structure is acid resistant and is difficult to break down (Liu et al., 2023; Xing, Wang, Ma, et al., 2018).

This section will present the limited literature that discuss rubidium extraction from this type of mineral. Depending on the minerals present in the ore, different processes were proposed. This highlights the importance of proper mineralogical study in deciding which extraction methods to use.

2.1. Direct Leaching of the Ore

Xing, Wang, Ma, et al. (2018) studied the rubidium extraction from a granitic ore. The XRD of the ore (Figure 1) indicates the presence of quartz (SiO_2), kaolinite ($Al_2Si_2O_5(OH)_4$), orthoclase ($KAlSi_3O_8$), muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$), microcline ($KAlSi_3O_8$), chlorite ($(Mg,Al)_6(Si,Al)_4O_{10}(OH)_8$) and albite ($NaSi_3AlO_8$). The major phases were quartz, orthoclase, and micas; while the minor phases were albite, chlorite, and kaolinite. Further investigation through SEM-EDS analysis showed that the micas consisted of biotite ($K(Mg,Fe)_3(AlSi_3O_{10})(OH,F)_2$) and muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$), and that rubidium was scattered in micas and feldspars in the form of isomorphism.

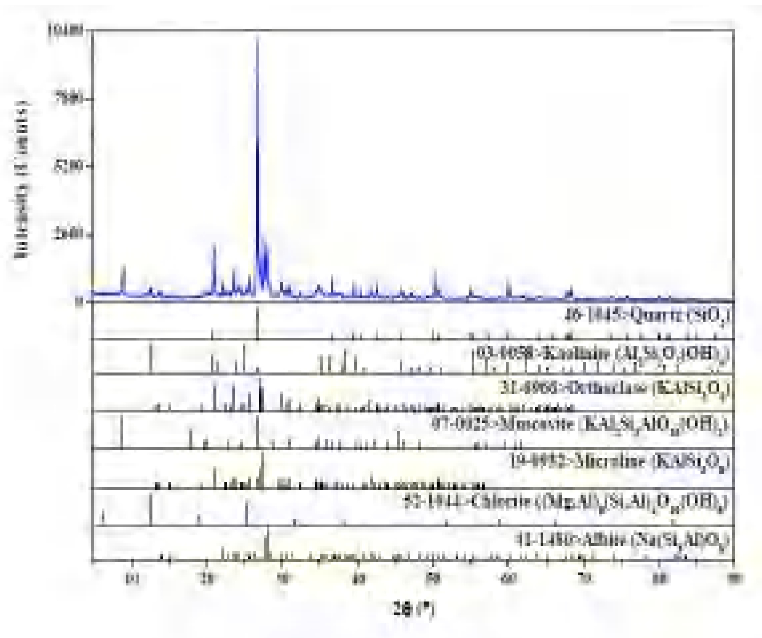


Figure 1: XRD Pattern of the Granitic Rubidium Ore (Xing, Wang, Ma, et al., 2018)

Direct sulphuric acid leaching and direct alkaline leaching of the ore were conducted in a pressure autoclave using the following conditions in Table 1. In both cases, rubidium recovery was poor, indicating that direct leaching is not very effective to extract rubidium. XRD analysis revealed that acid leaching was able to leach micas, chlorite, and kaolinite, while quartz, K-feldspar, and albite remained in the residues. On the other hand, most of the micas did not react to alkali during leaching, while it was possible to attack K-feldspar, transforming it to cancrinite ($Na_6Ca_2Al_6Si_6O_{24}(CO_3)_2{\cdot}2H_2O$) and zeolite ($Na_2Al_2Si_3O_{10}{\cdot}2H_2O$).

Table 1: Direct Leaching Conditions

	Direct acid leaching	Direct alkaline leaching	
Reagent	Sulphuric acid	Sodium hydroxide	Sodium hydroxide
Reagent concentration, g/L	200	300	200 g/L
Liquid/solid ratio	20:1	20:1	10:1
Leaching temperature, °C	200	150	230
Leaching time, hour	1.5	1	1
Rubidium recovery, %	66.3	40.2	95.1
Potassium recovery, %	39.7	60.5	Not indicated
Source	(Xing, Wang, Ma, et al., 2018)	(Xing, Wang, Ma, et al., 2018)	(Xing, Wang, Wang, et al., 2018)

On the contrary, Xing, Wang, Wang, et al. (2018) presented a similar approach (i.e., alkaline leaching in a pressure autoclave) and were able to yield a better rubidium recovery of 95.1%. The ore had 0.09% Rb grade, and consisted of similar primary phases: micas, quartz, and orthoclase. The secondary phases were albite, chlorite, kaolinite, and microcline. The micas were composed of biotite and muscovite, and rubidium was scattered in the micas and K-feldspar in isomorphic form (Xing, Wang, Wang, et al., 2018).

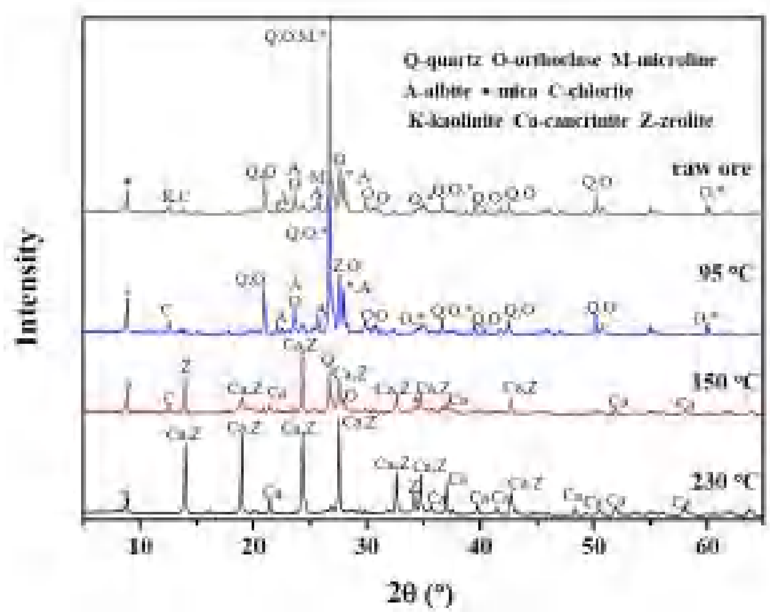


Figure 2: XRD Pattern of the Granitic Rubidium Ore (Xing, Wang, Ma, et al., 2018)

The conditions used for alkaline leaching are presented in Table 1. Between the two methods of alkaline leaching, the latter has a lower reagent concentration, lower liquid/solid ratio, but higher leaching temperature. An investigation on the rubidium extraction as a function of temperature, NaOH concentration and liquid/solid ratio, (Figure 3), showed that:

- Increasing the caustic reagent concentration from 200 g/L to 300 g/L did not affect rubidium extraction.
- Rubidium extraction reached a plateau at liquid/solid ratios from 10 mL/g to 20 mL/g.
- Finally, the leaching temperature seemed to significantly affect the extraction. Increasing the temperature increases the extraction.

The relatively low rubidium recovery was attributed to the unreacted orthoclase, micas, and quartz.

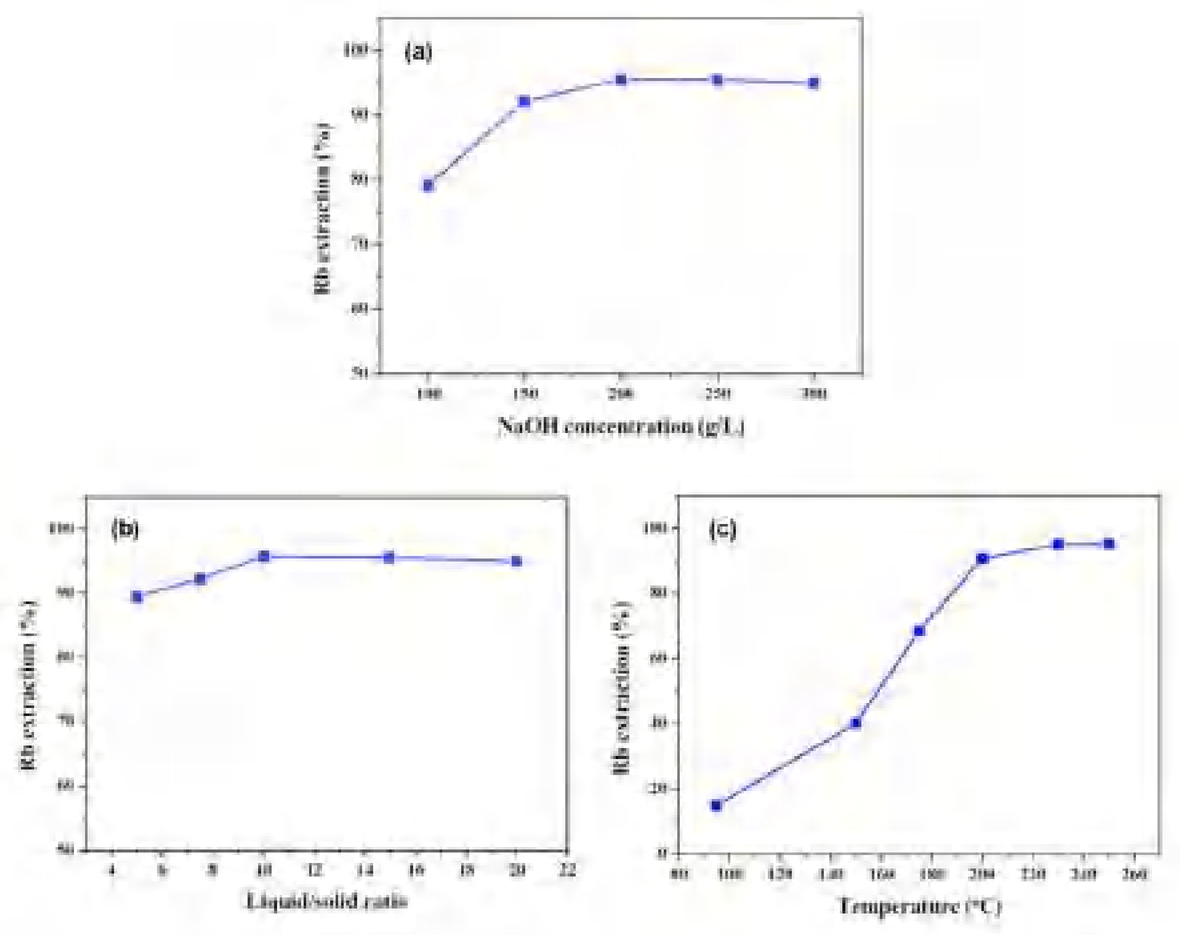


Figure 3: Effects of (a) NaOH Concentration, (b) Liquid/Solid Ratio, and (c) Temperature on Rubidium Extraction (Xing, Wang, Ma, et al., 2018)

2.2. Sulphuric Acid Baking – Decomposition – Alkaline Leaching

Xing et al. (2018) also proposed a multi-step approach (see Figure 4) which considered the results from direct acid and direct alkaline leaching (Section 2.1) and added an extra step to regenerate the sulphuric acid consumed in the acid leaching. In the first step, sulphuric acid baking was supposed to attack the micas under the following optimum conditions: a temperature of 300 °C, mass ratio of sulphuric acid to ore of 55%, and baking time of 20 minutes. The mineralogy study of the baked ore suggests that rubidium-bearing biotite and muscovite formed iron aluminum sulphate ($(Fe,Al)_2(SO_4)_3$), potassium alum ($KAl(SO_4)_2 \cdot 12H_2O$), hydroxylaluminum sulphate ($Al_2(OH)_4(SO_4) \cdot 7H_2O$ and $Al_4(SO_4)(OH)_{10}$), SiO_2 and potassium sulphate (K_2SO_4) after sulphuric acid baking.

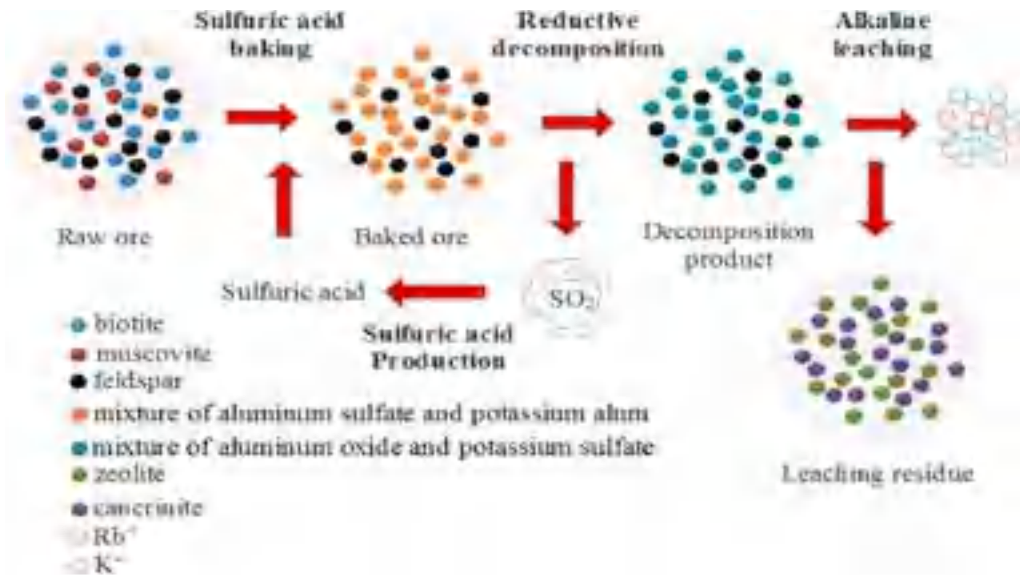


Figure 4: Strategy to Extract Rubidium and Potassium from Granitic Ore (Xing, Wang, Ma, et al., 2018)

After the sulphuric acid baking the sample was ground in a rod mill for 2 minutes. It was then mixed with lignite and placed in a tube furnace for decomposition. The goal of the decomposition is to release SO_2 and recover it for sulphuric acid production. The optimum conditions for decomposition were: a mass ratio of lignite to baked ore of 5%, temperature of 750°C, and decomposition time of 10 minutes. It was proposed that reactions (1) to (3) occurred during decomposition.

$$Al_2(SO_4)_3 + 3\ CO_{(g)} \rightarrow Al_2O_3 + 3\ SO_{2(g)} + 3\ CO_{2(g)} \quad (1)$$

$$Fe_2(SO_4)_3 + 3CO_{(g)} \rightarrow Fe_2O_3 + 3\ SO_{2(g)} + 3\ CO_{2(g)} \quad (2)$$

$$2\ KAl(SO_4)_2 + 3CO_{(g)} \rightarrow K_2SO_4 + Al_2SO_3 + 3\ SO_{2(g)} + 3\ CO_{2(g)} \quad (3)$$

After decomposition, the product was leached with 250 g/L NaOH solution in a pressure autoclave. The optimum conditions for the NaOH leaching are presented in Table 2. As mentioned in Section 0, the interaction of K-feldspar and the lixiviant formed zeolite and some calcium from plagioclase contributed to the formation of cancrinite. These interactions released the potassium and associated rubidium in the feldspar into solution.

Table 2: Sulphuric Acid Baking – Decomposition – Alkaline Leaching Conditions (Xing, Wang, Ma, et al., 2018)

	Sulfuric acid leaching	Decomposition	Alkaline leaching
Reagent	Sulfuric acid	Lignite	250 g/L sodium hydroxide
Reagent ratio	55% acid to ore	5% coal to baked ore	15:1 liquid to solid
Temperature, °C	300	750	150
Residence time, minutes	20	10	60
Rubidium recovery, %			94.7
Potassium recovery, %			92.2

2.3. Thermal Activation – Sulphuric Acid Leaching

Liu (2023) proposed a novel method for extracting rubidium (as well as potassium) from complex rubidium ore. Their ore had a rubidium grade of 0.12% and consisted primarily of quartz, feldspars, and micas. The secondary phases were kaolinite and chlorite. The feldspar was mainly potash feldspar, which included orthoclase and microcline, followed by plagioclase, which was mainly albite.

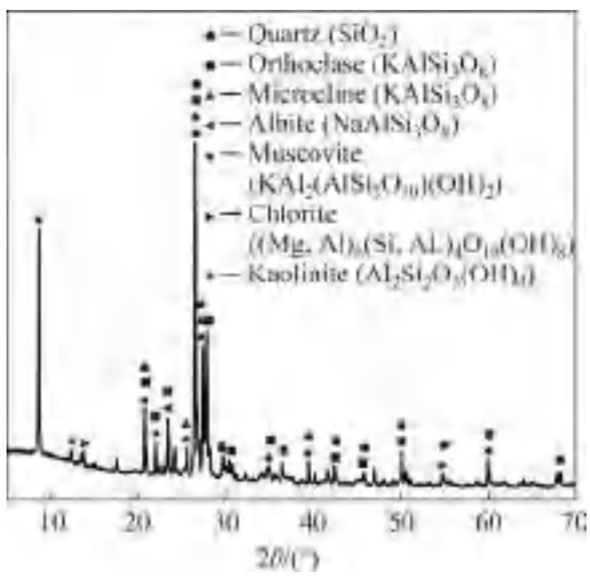


Figure 5: XRD Pattern of the Complex Rubidium Ore (Liu et al., 2023)

Their method consisted of two steps: 1) thermal activation of rubidium ore and water quenching, and 2) leaching with sulphuric acid. In the first step, the ore was mixed with a certain amount of CaO in an alumina crucible and melted in a muffle furnace. The molten slag was poured into water for water-quenching. In the second step, the water-quenched slag was dried in an oven, milled, and subjected to sulphuric acid leaching.

It was proposed that at 1000°C, CaO reacts with feldspar and mica forming Ca_2SiO_4. At 1300°C, where the ore is melted, the XRD scan did not exhibit significant peaks from the main and secondary phases. Scanning electron microscope images revealed that after the thermal activation, the water-quenched slags no longer had the physical phase of the rubidium ore, and all elements were observed to be dispersed homogenously in the sample. The water-quenched slags were said to be an unstable and highly reactive amorphous glass material, which was then readily decomposed by H_2SO_4. After leaching, the residues consisted of calcium sulphate with varying levels of hydration ($CaSO_4•2H_2O$ and $CaSO_4•0.5H_2O$). The optimum conditions for rubidium extraction are presented in Table 3. These conditions yielded 99.2% Rb recovery and 99.0% K extraction.

Table 3: Thermal Activation and Sulphuric Acid Leaching Conditions (Liu et al., 2023)

	Thermal activation	Sulfuric acid leaching
Reagent	CaO	H_2SO_4
Reagent concentration	Not mentioned	120 g/L
Reagent ratio	30% reagent to ore	10 mL/g liquid to solid
Temperature, °C	1300	50
Residence time, minutes	60	90
Rubidium recovery, %		99.2
Potassium recovery, %		99.0

3. Potassium Extraction from K-feldspar

Silicates contain potash values of up to 14% in some rock types (Tanvar & Dhawan, 2020). These rocks are a group of aluminosilicates of potassium, sodium, and calcium (K_2O%: 5-12, Na_2O%: 3-10, SiO_2%: 52-65, Al_2O_3%: 15-20, Fe_2O_3%, 5-7, MgO%: 2-3.5). In general, potassium extraction from silicate rocks has received limited attention due to the challenges in extraction, and the limited availability of economic extraction processes (Tanvar & Dhawan, 2020). The potassium entrapment within the aluminosilicate framework makes physical beneficiation and even chemical processes ineffective. Previous research work on potassium extraction from silicate rocks mainly involve roasting and then leaching. The roasting uses sodium- and calcium-containing additives such as sodium carbonate, sodium sulphate, sodium chloride, calcium carbonate, gypsum, and calcium chloride (Samantray et al., 2019; Yuan et al., 2015). Among the most effective methods mentioned above, the highest potassium recovery involved chlorination roasting which efficiently broke the aluminosilicate framework, releasing potassium for easier extraction (Samantray et al., 2019). In this section, a summary of different chlorination methods found in the literature, mainly using $CaCl_2$, is presented in Table 4. The probable reactions taking place are as follows:

$$CaCl_2 + 3\ KAlSi_3O_8 \rightarrow 2\ KCl + CaAl_2Si_2O_8 + 4\ SiO_2 \quad (4)$$

$$2\ CaCl_2 + 2\ KAlSi_3O_8 + 0.5\ O_2 \rightarrow 2\ KCl + CaAl_2Si_2O_8 + 3\ SiO_2 + CaSiO_3 + Cl_2 \quad (5)$$

In Equation (4), $CaCl_2$ reacts with K-feldspar forming anorthite ($CaAl_2Si_2O_8$), quartz (SiO_2), and sylvite (KCl). Another proposed reaction is the one in Equation 5, where oxygen is suggested to react with the mixture, forming additional by-products such as wollastonite ($CaSiO_3$) and chlorine (Cl_2).

3.1. Methodology Variations

The samples tested in the following literature sources contained primarily microcline and quartz minerals with varying K_2O content from 8.4 to 13.3%. In general, the roasting process involved mixing the chlorination agent with the ore in a certain ratio, placing it in a furnace and heating it to 800-950°C for a specified amount of time. The roasted sample is then leached with a lixiviant for a certain period of time. Slight variations in methodology between each test method were identified from the literature. For example, one method involves using a press to make the mixture (chlorination agent or flux and ore) into a tablet prior to roasting (Yuan et al., 2015). Furthermore, the roasting was done in a dry and damp nitrogen atmosphere to test the effect of moisture (Yuan et al., 2015). The results showed that roasting at dry nitrogen atmosphere is favorable as thermal hydrolysis of $CaCl_2$ in damp atmosphere might occur. The hydrolysis reaction may compete with the conversion reaction with K-feldspar, leading to lower potassium recovery.

In another method, only a thin layer of the mixture (chlorination agent and ore) was spread in a crucible and was placed in a muffle furnace without controlling the atmospheric gas (Türk et al., 2021). This yielded 97.1-98.6% potassium recovery. Several techniques were compared by Tanvar & Dhawan (2020), such as direct leaching (no roasting), roasting – leaching, milling with flux – roasting, and milling – roasting – leaching. Direct leaching and roasting – leaching were the commonly employed methods discussed in the previous sections. Milling with flux – leaching and milling with flux – roasting and leaching was something that was uniquely explored in their work. Directly leaching of the sample after milling-with-flux was ineffective without roasting, in liberating potassium from K-feldspar. On the other hand, milling with flux then roasting released a significant amount of soluble potassium-bearing compound, leading to a satisfactory potassium recovery. Overall, the minor variations from the usual roast-leach procedure did not seem to have a great impact on the potassium recovery.

Table 4: Summary of Potassium Extraction from K-feldspar in the Literature

Process	Sample	K_2O content	Roasting conditions	Leaching	K recovery	Source
$CaCl_2$ roasting	50-75um K-feldspar (microcline), quartz	13.25%	900°C, 40 min, 1:1.15 $CaCl_2$ to ore ratio	70°C, 30min, solid/liquid ratio of 1:50	91%	Yuan et al, 2015
$CaCl_2$ roasting	-106um K-feldspar, albite, muscovite, albite, quartz	9.69%	850°C, 60 min, 1.5:1 $CaCl_2$ to ore ratio	60°C, 120min, 10% solid ratio	99.8%	Serdengecti et al, 2019
Egg shell powder + HCl roasting (1.5 HCl to $CaCO_3$ stoich)	45um microcline, orthoclase, quartz	11.64%	900°C, 30min	Not specified	99%	Samantray et al., 2020
$CaCl_2$ roasting (from Wollastonite-Calcite Ore)	K-feldspar and albite	8.40%	800-950°C, 60 min 1.5:1 $CaCl_2$ to ore ratio	60°C, 120 min, 10% solid ratio	97.1-98.6%	Turk et al., 2021
$CaCl_2{\bullet}2H_2O$ roasting	Microcline, orthoclase, quartz	11.64%	900°C, 60 min, 1.7:1 $CaCl_2$ to ore ratio	Room temperature, 30 min	99.90%	Samantray et al., 2019
$CaCl_2{\bullet}2H_2O$ roasting	Albite, microcline, quartz	8.46%	950°C, 60 min, 1:1 flux to ore ratio,	2% citric acid, room temp, 60 min, 1:10 g/mL solid-liquid ratio	>95%	Tanvar 2019

3.2. Different Sources of $CaCl_2$

A remarkable difference seen between the different methods in the literature (Table 4) is the source of chlorination agent ($CaCl_2$). Most of the experiments used commercial $CaCl_2$ or $CaCl_2 \cdot 2H_2O$, however there are some who used secondary sources such as eggshell powder or wollastonite/calcite minerals.

3.2.1. Eggshell Powder

The XRD scan of the eggshell powder (Figure 6) consists of peaks that match that of $CaCO_3$ (Samantray et al., 2020). To achieve the roasting of the ore, the eggshell powder was added with HCl during the process. It was proposed that eggshell powder converts to $CaCl_2$ when added with HCl, through the following equation:

$$CaCO_3 + 2HCl \rightarrow CaCl_2 + CO_2 + H_2O \quad (6)$$

$CaCl_2$ then reacts with the K-feldspar similar to Equation (5), forming water-soluble sylvite (KCl), and water-insoluble phases such as anorthite ($CaAl_2Si_2O_8$), quartz (SiO_2), and wollastonite ($CaSiO_3$). The optimum conditions for extraction were 1:1.5 eggshell powder to HCl, 1:1.8 feldspar to eggshell ratio, roasting at 900°C for 30 minutes. The roasted product was then leached with water at different leaching parameters. However, it was found that varying the parameters did not significantly change the potassium recovery. The maximum potassium recovery achieved using the process was 99%.

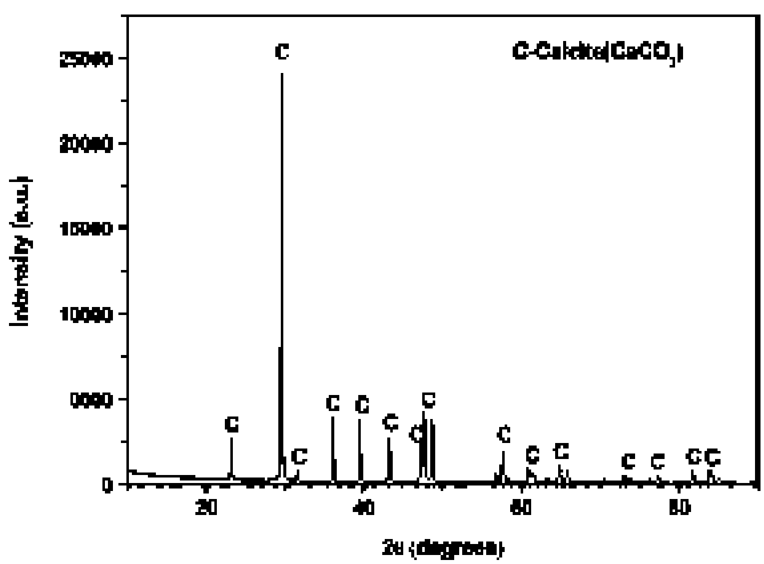


Figure 6: XRD Pattern of the Eggshell Sample (Samantray et al., 2020)

3.2.2. Wollastonite-calcite Ore

Wollastonite and calcite are usually found in the same deposits as K-feldspars. Calcite is a useful alternative source of $CaCl_2$ when treated with HCl. Wollastonite conversion to $CaCl_2$ was not discussed by Türk et al (2021) as it was not typically used for that purpose. However, it was mentioned that acidic digestion of wollastonite can be used as an indirect carbonation route for CO_2 sequestration.

In their work, calcite was separated from the wollastonite-calcite ore via flotation. Flotation was done at pH 6 using potassium oleate as collector. In the process, calcite was floated as the concentrate and wollastonite remained in the slurry as tailings. The calcite concentrate was then treated with HCl, and $CaCl_2$ was recovered via evaporative precipitation. The $CaCl_2$ was then used in the roasting process. The optimum conditions for roasting were 1.5:1 flux to ore ratio at 900°C, roasting for 60 minutes. These conditions yielded 98.6% potassium recovery.

4. Cesium Extraction from Minerals

Cesium can occur in minerals such as pollucites, lepidolites, carnallite, and muscovite (Lu et al., 2022; Mein, 1986; Perel'man, 1961; Zhang et al., 2020). The most important among these commercial sources is the zeolite mineral pollucite ($Cs_2O•Al_2O_3•4SiO_2$). There are different methods for recovering cesium from minerals: 1) direct reduction with metals, 2) acid leaching, 3) autoclave leaching, and 4) thermal decomposition. Depending on the additives used in the decomposition method, the processes can be further categorized into 1) sulphation roasting, 2) carbonation roasting and 3) chlorination roasting.

4.1. Acid Leaching

For minerals that are easily attacked by acid, acid digestion is a good option. Typically, hydrochloric acid and sulphuric acids are used at elevated temperatures. Lu et al., (2022) investigated cesium extraction from lepidolite ore and extracted 95% cesium with 150 g/L sulphuric acid at a leaching temperature of 93°C via a two-stage countercurrent leaching process.

4.2. Autoclave Leaching

Another method involved sparging SO_2 for reductive leaching of the ore. The process involved dissolution of SO_2 into the slurry. This reduced the metals in the ore and generated sulphuric acid (Lu et al., 2022). The process was conducted in an autoclave at 90-110°C, with an initial H_2SO_4 concentration of 50 g/L, pulp density of 100 g/L and a leaching time of 4 hours. The recoveries of cesium and rubidium were 93% and 78%, respectively.

4.3. Direct Reduction

In the direct reduction method, pollucite was reduced with calcium at 950°C, sodium or potassium at 750°C, or aluminum at around 1000°C. This technology is not very popular due to the huge metal consumption, incomplete cesium extraction, and low cesium product quality (Lu et al., 2022).

4.4. Decomposition

In the decomposition method, the ore is roasted with different fluxes, which convert it into a soluble phase. Depending on the flux, it can be called sulphation roasting, carbonation roasting or chlorination roasting. In the sulphation roasting method, various sulphates such as sodium sulphate, potassium sulphate, calcium sulphate, or their mixtures can be used to roast the ore at 850-1100°C (Zhang et al, 2020, 2022). There are drawbacks to this method such as low co-extraction of rubidium and cesium, requirement of higher temperature translating to higher energy input, etc. (Zhang et al., 2022). In carbonation roasting, the ore can be roasted with $CaCO_3$ at 1000°C followed by a water leach at 100°C (Walter & Bichowsky, 1935). In chlorination roasting, the flux is a chloride salt such as $CaCl_2$, NaCl, NH4Cl, or in the presence of HCl or chlorine gas (Zhang et al., 2020). Mixture of additives have also been explored, such as $CaCO_3$-$CaCl_2$ mix with roasting at 800 - 900°C or a Na_2CO_3-NaCl roasting at 600 - 800°C followed by a water leach (Lu et al., 2022). Most of these processes track individual metal recovery such as lithium only, cesium only, rubidium only or potassium only. There are limited studies that consider the co-extraction of rubidium, cesium, and potassium. The work of Zhang et al., (2020) stood out due to the simultaneous co-extraction of the critical metals. In Table 5, a comparison between single additive ($CaCl_2$ or NaCl) and a mixture ($CaCl_2$-NaCl) is presented. When $CaCl_2$ was added with a ratio of flux to ore of 0.3:1, 70% of lithium, 91% rubidium, 97% cesium and 71% potassium were recovered. When it was added with NaCl, recoveries were not as good: 30-39% lithium, 66-71% rubidium, 59-71% cesium and 40-49% potassium. This indicates that NaCl is less effective in metal extractions than $CaCl_2$. When the mixture of the two additives were considered, the metal recoveries increased significantly to 92% lithium, 98% rubidium, 98% cesium, and 93% potassium.

Table 5: Summary of Chlorination Roasting Conducted by Zhang et al., 2020

Process	Sample	Ore content	Roasting conditions	Leaching	K recovery	Source
Chlorination roasting ($CaCl_2$ or NaCl)	Lepidolite, albite, quartz	6.25% K2O, 1.08% Rb2O, 0.22% Cs2O, 3.34% Li2O	0.3:1 (agent: ore), 850 C, 60 min	water leaching, 60°C, 60 min, 4:1 liquid-solid ratio	70.33% Li, 91.43% Rb, 97.20% Cs, 71.45% K	Zhang et al 2020
Chlorination roasting ($CaCl_2$ + NaCl)	Lepidolite, albite, quartz	6.25% K2O, 1.08% Rb2O, 0.22% Cs2O, 3.34% Li2O	0.3:0.2:1 ($CaCl_2$: NaCl: ore), 750 C, 45 min	water leaching, 25°C, 60 min, 3:1 liquid-solid ratio	92.49% Li, 98.04% Rb, 98.33% Cs, 92.90% K	Zhang et al 2020

Summary and Recommendations

Summary

Rubidium in a granitic pegmatite deposit is generally dispersed in micas and potassium feldspar in the form of isomorphism.

Depending on the mineral, different extraction methods have been proposed. Since rubidium is chemically bound with K-feldspars in the Raleigh Lake ore deposit, most of the methods for recovering rubidium discussed here involve first breaking the aluminosilicate framework of the feldspar. Direct acid leaching of ore with sulphuric acid in an autoclave achieves poor rubidium recovery and is not recommended. Initial direct alkaline leaching with strong sodium hydroxide also failed to extract acceptable amounts of rubidium. However, by simply increasing the alkaline leaching temperature, rubidium recovery was improved by about 2.5 times. Other proposed methods include pre-treatment prior to leaching such as thermal activation and acid baking. In the sulphuric acid baking process, some of the minerals are readily attacked by acid and transformed into well-dissociated sulphate phases. However, this treatment does not attack the feldspars, which are acid-resistant minerals. In thermal activation, on the other hand, if the sample is mixed with a flux such as CaO and melted into a highly reactive amorphous glass material, which is then leached with sulphuric acid, rubidium and potassium are efficiently leached, leaving gypsum in the leach residue.

Rubidium extraction methods discussed above should also work for potassium recovery from K-feldspars. However, most of the suggested methods in the literature involve chlorination roasting instead of high-pressure alkaline leaching and other thermal decomposition methods. There is no industrial method for extracting potash from K-feldspar as of yet. Most of the processes that were presented were conducted in a laboratory scale set-up. Typical methods presented included roasting with sodium- and calcium-containing additives such as sodium carbonate, sodium sulphate, sodium chloride, calcium carbonate, gypsum, and calcium chloride. The most effective method, with the highest potassium recovery, involved chlorination roasting to break down the aluminosilicate framework and release potassium for easier extraction.

Cesium is not always associated with rubidium, although it can be found in a variety of other pegmatite minerals. However, similar to rubidium, it can also occur within the aluminosilicate framework. Hence, to recover cesium, it is required to break the aluminosilicate frameworks too. Interestingly, one of the most popular cesium extraction methods is the decomposition method. In this method, different additives such sulphate-based, carbonate-based or chloride-based are used.

Recommendations

Rubidium is chemically bound to microcline (K-feldspar) in the Raleigh Lake deposit, while cesium is found in muscovite. Based on the literature review and the limited mineralogical data available for the Raleigh Lake deposit, the following is recommended to extract the target metals.

- Rubidium recovery in K-feldspar
    - Alkaline leaching in an autoclave under the following conditions: 200-250 g/L sodium hydroxide, 10-15:1 liquid-solid ratio, leaching at 150-230°C for 60 minutes (Xing, Wang, Ma, et al., 2018; Xing, Wang, Wang, et al., 2018).

- Rubidium and potassium recovery in K-feldspar
    - Thermal activation with CaO in a muffle furnace at 1300°C for 60 minutes then leaching with 120 g/L sulphuric acid at 50°C for 90 minutes (Liu et al., 2023).

- Potassium recovery from K-feldspar
    - Chlorination roasting using $CaCl_2$ or $CaCl_2•2H_2O$ at 850-950 °C for 30-60 minutes, then water leaching at room temperature (Samantray et al., 2020; Serdengeçti et al., 2019; Tanvar & Dhawan, 2020; Türk et al., 2021; Yuan et al., 2015.

- Co-extraction of rubidium, potassium, and cesium in lepidolite
    - Chlorination roasting using mixed $CaCl_2$-NaCl at >750°C for 45 minutes followed by water leaching at ambient temperature for 60 minutes (Zhang et al., 2020).

These processes were chosen based on their expected high metal recoveries. The optimum conditions were obtained based on exhaustive variability testing. In this regard, it must be highlighted that employing the same process and same conditions may not yield similar results. This is due to the fact that these conditions were optimized to each specific ore. The Raleigh Lake ore may not have the exact composition, the same associated gangue minerals, and impurities.

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AN INVESTIGATION INTO

MINERAL PROCESSING TESTWORK ON SAMPLES FROM

THE RALEIGH LAKE DEPOSIT – PHASE 1

prepared for

INTERNATIONAL LITHIUM

Project 19977-01 – Report 1
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SGS Canada Inc. | P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0
Tel: (705) 652-2000 Fax: (705) 652-6365 www.met.sgs.com www.ca.sgs.com
Member of the SGS Group (SGS SA)

Table of Contents

List of Tables

List of Figures

Executive Summary

Approximately 40 kg of a lithium ore sample and 16 kg of a rubidium ore sample from the Raleigh Lake deposit were received at SGS Lakefield in August 2023 for PEA scoping study level metallurgical testwork. The scope of work in this Phase 1 mineral processing report included sample preparation, and head assays of the two samples and one Heavy Liquid Separation (HLS) test on the lithium ore sample. Additional testwork including mineralogy, a rubidium extraction literature review and metallurgical testwork will be discussed in separate reports.

The main objective of the mineral processing test program was to provide an indication of the potential for lithium beneficiation by dense media separation (DMS). The metallurgical target was the preparation of a spodumene concentrate grading >6.0% Li_2O while maximizing lithium recovery.

The lithium sample contained 1.59% Li_2O and 0.56% Fe_2O_3, while the rubidium sample graded 6,580 g/t Rb (equivalent to 0.72% Rb_2O), along with 0.12% Li_2O and 0.24% Fe_2O_3. The key assays of the samples are presented in Table I.

Table I: Summary of the Head Assay Results

Sample ID	Assays											
	Li (%)	Li_2O (%)	Rb (g/t)	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	MgO %	CaO %	Na_2O (%)	K_2O (%)	S %	Be g/t
Lithium Head	0.74	1.59	-	73.2	16.8	0.56	0.05	0.22	4.73	1.94	< 0.01	79.9
Rubidium Head	0.056	0.12	6,580	70.0	16.8	0.24	0.05	0.16	4.05	7.92	0.02	20.6

Heavy liquid separation (HLS) tests were performed on the lithium sample after stage-crushing to 100% passing -12.7 mm and screening into -12.7 mm/+6.3 mm and -6.3 mm/+1 mm fractions. The interpolated HLS results indicated that the ideal SG cut points for producing a 6.0% Li_2O concentrate were 2.83 and 2.85 at -12.7 mm/+6.3 mm and -6.3 mm/+1 mm fractions, respectively. The ideal SG cut-point increased as the fraction size increased, and the highest stage lithium recovery of 90.7% was achieved at -6.3 mm/+1 mm fraction. At a tailings SG-cut point of 2.70, the HLS middling from each sample contained between 1.27 – 1.69% Li_2O with 2.3 – 8.2% of the global lithium distribution. The HLS middlings can potentially be mixed with the minus 1 mm fines fraction to produce a flotation feed grading 1.19 % Li_2O and 0.59% Fe_2O_3. The HLS results provided a strong positive indication of the amenability to DMS for lithium beneficiation (Table II).

Based on these HLS results, it is recommended to perform additional HLS testing at different crushing sizes to determine the optimum DMS plant feed parameter and then perform a DMS pilot campaign with the selected feed size and SG cut points from HLS testwork. Additional lithium will be recoverable by flotation of DMS middlings and <1 mm DMS fines at a feed grade at about 1.19% Li_2O. Once the final flowsheet is developed, it is recommended to perform beneficiation testwork on a number of variability samples to confirm the amenability of various samples to the developed flowsheet.

Table II: Summary of HLS Global Mass Balance (Interpolated @ 6.0% Li_2O)

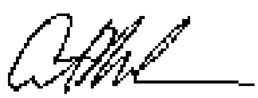
	Product	HL SG	Weight	Assays (%)			Distribution (%)	
		g/cm3	%	Li	Li_2O	Fe_2O_3	Li	Fe_2O_3
-12.7 +6.3 mm	HLS Concentrate	2.85	14.9	2.79	**6.00**	**0.91**	**53.0**	26.4
	HLS Middling	-2.85 +2.70	8.21	0.78	1.69	0.66	8.21	10.6
	HLS Tailings	2.70	32.1	0.042	0.089	0.28	1.70	17.6
-6.3 +1 mm	HLS Concentrate	2.83	8.01	2.79	**6.00**	**1.06**	**28.5**	16.5
	HLS Middling	-2.83 +2.70	3.07	0.59	1.27	0.99	2.32	5.90
	HLS Tailings	2.70	22.1	0.022	0.047	0.31	0.61	13.3
	Fines Fraction (-1 mm)		11.6	0.38	0.82	0.43	5.65	9.72
	Head (calc.)		100	0.78	1.69	0.52	100	100
	Head (dir.)			0.74	1.59	0.56		
	Flotation Feed		22.9	0.55	1.19	0.59	16.2	26.2

Introduction

Mr. Anthony Kovacs from International Lithium contacted SGS Lakefield with a request for PEA scoping study-level metallurgical testwork on samples from the Raleigh Lake deposit. The main objective of the testwork program was to provide HLS results for a preliminary indication of the lithium beneficiation performance by dense media separation (DMS). The metallurgical target was the preparation of spodumene concentrate grading >6.0% Li_2O while maximizing lithium recovery.

The scope of work in this report included the receipt, preparation, and head assays on two samples received in August 2023 and HLS testwork on the pegmatite sample. Additional testwork including mineralogy, rubidium extraction literature review, and testwork will be discussed in separate reports.

Kristina Muntu
Senior Metallurgist, Mineral Processing



Curtis Mohns, M.Sc., P.Eng.
Senior Metallurgist, Mineral Processing

Experimental work by: Kristina Muntu
Report preparation by: Kristina Muntu
Reviewed by: Cheryl Mina, Chris Fleming, Curtis Mohns

Testwork Summary

1. Sample Receipt and Preparation

1.1. Sample Receipt

Two samples were submitted for a PEA scoping-level testwork program by International Lithium: one sample representing Deposit One lithium ore and one sample representing Deposit Two rubidium ore from the Raleigh Lake.

A shipment consisting of one skid with two blue-lidded totes was received at the SGS Lakefield site on August 11, 2023, and assigned an internal receipt number of 0122-AUG23. Upon receipt, all samples were inventoried and weighed. The skid contained approximately 40 kg of a lithium ore sample and 16 kg of a rubidium ore sample.

1.2. Sample Preparation

The pegmatite or lithium samples were initially stage-crushed to 100% passing 12.7 mm, homogenized, and split into 10 kg test charges. One 10 kg charge was sub-sampled at approximately 500 g for head assays and the remainder was screened at 16 mesh to remove the -1 mm fraction. The +1 mm fraction was further screened at 12.7 mm to generate two fractions of -12.7 mm +6.3 mm and -6.3 mm +1 mm. Two -1 mm subsamples were taken from the fine fraction for mineralogical analysis, and assays for lithium and WRA. The two coarse fractions, -12.7 mm +6.3 mm and -6.3 mm +1 mm, were submitted for Heavy Liquid Separation (HLS) testing.

The rubidium sample was prepared by stage-crushing it to 100% passing -10 mesh, homogenizing, and splitting into 1 kg test charges. One 1 kg charge was sub-sampled at approximately 100 g for head assays while the remainder was submitted for mineralogical analysis. The remaining charges were stored for future testing on the rubidium extraction process.

2. Head Assays

Lithium, whole rock analysis (WRA), and an ICP scan were performed on the lithium and rubidium composite samples. The head assay results are presented in Table 1. The lithium sample graded 1.59% Li_2O and 0.56% Fe_2O_3 with 79.9 g/t Be. The rubidium sample graded 6,580 g/t Rb (equivalent to 0.72% Rb_2O) and 0.12% Li_2O with 0.24% Fe_2O_3 and 20.6 g/t Be.

Table 1: Head Assays of the Lithium and Rubidium Composite Samples

Element/Oxide	Unit	Sample ID	
		Li Head	Rb Head
Li	%	0.74	0.056
Li_2O	%	1.59	0.12
Rb	g/t	---	6,580
S	%	< 0.01	0.020
Whole Rock Analysis			
SiO_2	%	73.2	70.0
Al_2O_3	%	16.8	16.8
Fe_2O_3	%	0.56	0.24
MgO	%	0.05	0.05
CaO	%	0.22	0.16
Na_2O	%	4.73	4.05
K_2O	%	1.94	7.92
TiO_2	%	< 0.01	< 0.01
P_2O_5	%	< 0.01	0.02
MnO	%	0.15	0.04
Cr_2O_3	%	0.02	< 0.01
V_2O_5	%	< 0.01	< 0.01
LOI	%	0.71	0.33
Sum	%	98.5	99.7
ICP Scan			
Ag	g/t	< 2	< 2
As	g/t	< 30	< 30
Ba	g/t	7.0	31.0
Be	g/t	79.9	20.6
Bi	g/t	< 20	< 20
Cd	g/t	< 2	< 2
Co	g/t	< 4	< 4
Cu	g/t	6.0	< 5
Mo	g/t	< 5	< 5
Ni	g/t	< 20	< 20
Pb	g/t	< 30	< 30
Sb	g/t	< 10	< 10
Se	g/t	< 30	< 30
Sn	g/t	< 20	< 20
Sr	g/t	15	72
Tl	g/t	< 30	57
Y	g/t	2.4	0.8
Zn	g/t	28	19

3. Heavy Liquid Separation (HLS) Testwork

Heavy Liquid Separation (HLS) tests were conducted to assess the amenability of the lithium sample to Dense Media Separation (DMS) for the beneficiation of spodumene. The two coarse fractions submitted for HLS testwork, as described in section 1.2., were immersed in a heavy liquid comprised of methylene iodide diluted with acetone to target liquid-specific gravities (SG) of 2.90, 2.85, 2.80, 2.75, 2.70, and 2.65. The first pass was conducted using a heavy liquid with the highest specific gravity (SG 2.90), and each subsequent pass on the float fraction used a heavy liquid with a lower specific gravity. The six (6) resulting HLS sink products, as well as the final (SG 2.65) HLS float product and the -1 mm fine fraction, were all submitted for lithium and whole rock analysis (WRA). The HLS results are presented in Table 2 to Table 4 and the test details are provided in Appendix A.

3.1. HLS Testwork on the -12.7 mm +6.3 mm Fraction: Stage

The HLS results with the -12.7 mm +6.3 mm feed indicated that the stage lithium recovery increased gradually from 78.0% to 100% as the SG was reduced from 2.90 to 2.65, while the Li_2O grade decreased from 6.37% Li_2O at SG 2.90 (Figure 1) to 3.4% Li_2O at SG 2.65. A projected spodumene concentrate grade of 6.0% Li_2O would be produced at SG's between 2.80 and 2.85, with cumulative stage lithium recoveries ranging from 89% to 93%. Figure 2 presents photographs of the HLS sink products generated at SG's of 2.80, 2.85, and 2.90.

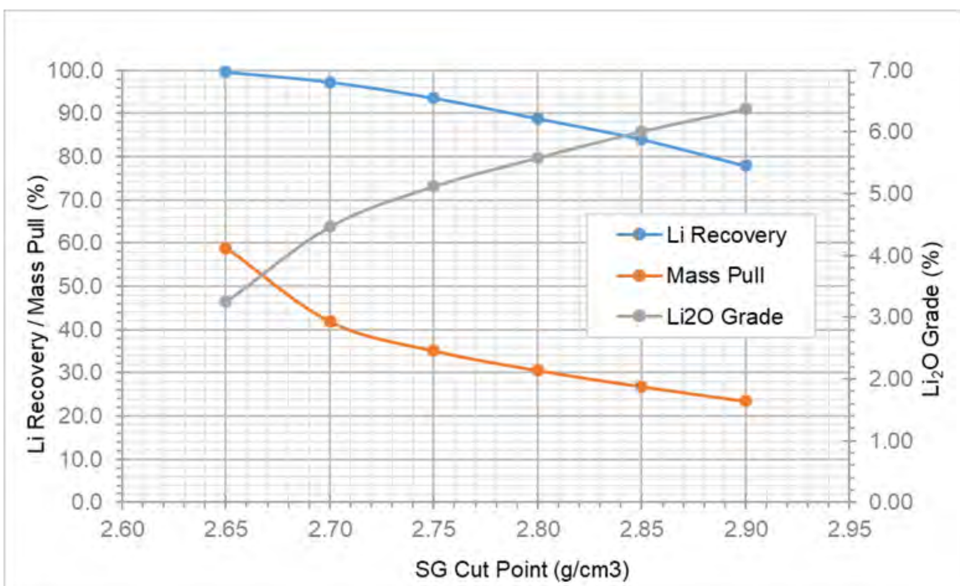


Figure 1: HLS Stage Results of -12.7 mm +6.3 mm Fraction

Figure 2: HLS Sink Products of 2.80, 2.85, and 2.90 SG from -12.7 mm +6.3 mm Fraction

3.2. HLS Testwork on the -6.3 mm +1.0 mm Fraction: Stage

The HLS results with the -6.3 mm +1 mm feed were similar to those at the -12.7 mm +6.3 mm fraction. As shown in Figure 3, the lithium recovery gradually increased and the Li_2O grade gradually decreased as the SG was reduced. A projected spodumene concentrate grade of 6.0% Li_2O would be produced at SG 2.83, with a lithium recovery of ~91%. Stage lithium distribution was >80% at all SG cut points. Figure 4 shows the photos of the HLS sink products generated at 2.80, 2.85, and 2.90 SG.

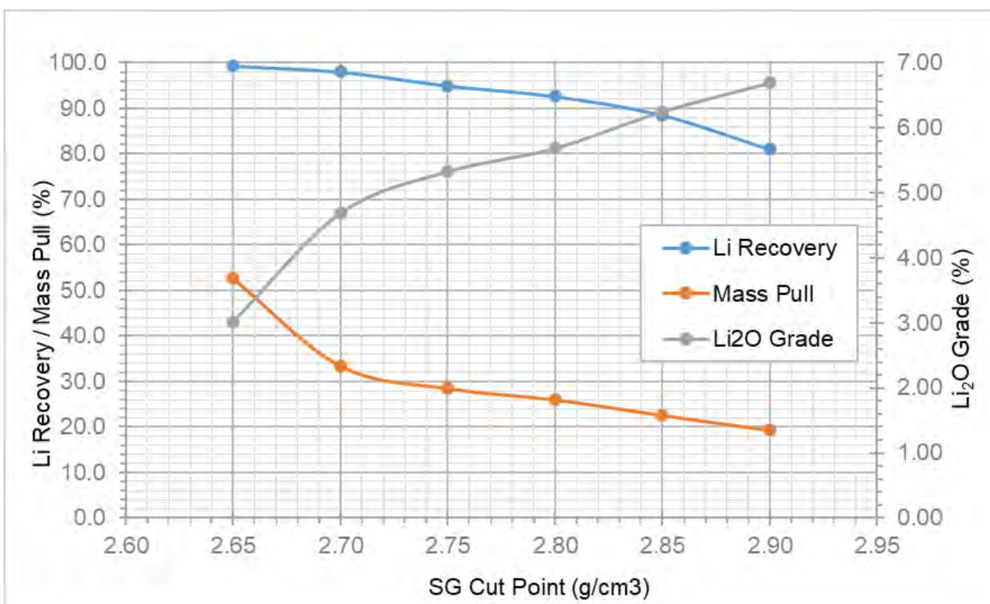


Figure 3: HLS Stage Results of -6.3 mm +1 mm Fraction

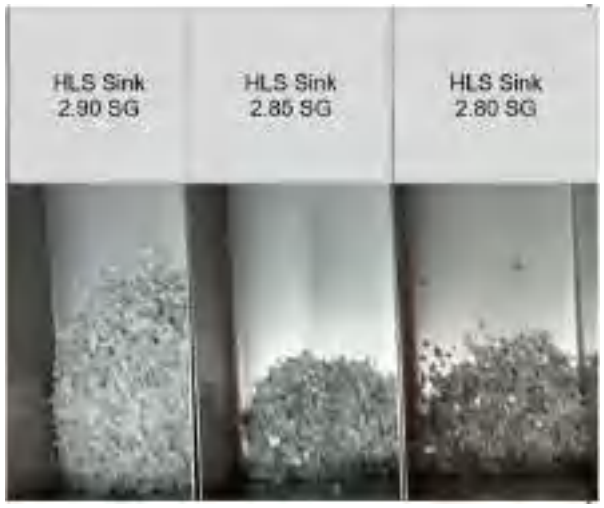


Figure 4: HLS Sink Products of 2.80, 2.85, and 2.90 SG from -6.3 mm +1 mm Fraction

3.3. Evaluation of The HLS Stage Performance at Different Size Fractions

A comparison of the stage performance for the two size fractions is presented in Figure 5. The HLS results indicated that the spodumene concentrate grades and lithium recoveries were higher at the -6.3 mm +1 mm fraction, most likely due to improved spodumene liberation in the finer fraction.

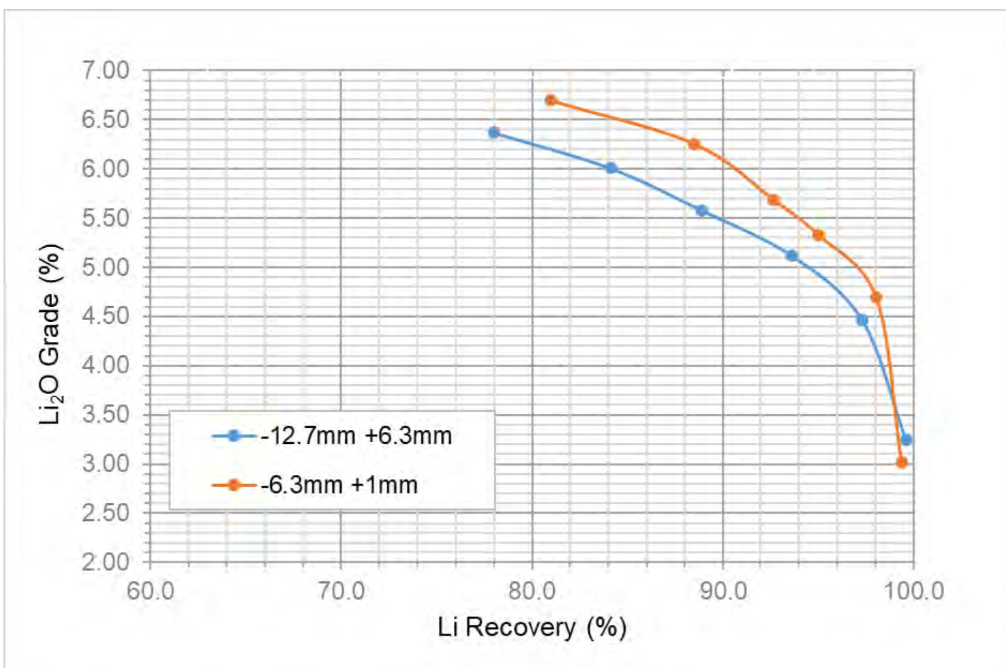


Figure 5: Stage Cumulative Lithium Grade / Recovery Plot

Table 2: HLS Test Results of -12.7 mm +6.3 mm Fraction (@6.0% Li2O) - Stage

Combined HLS Products	HL SG	Weight		Assays (%)									Distribution (%)							
	g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MnO	CaO	Na_2O	K_2O	Li	SiO_2	Al_2O_3	Fe_2O_3	MnO	CaO	Na_2O	K_2O
HLS Sink 2.85 SG	2.85	1,478	26.9	2.79	6.01	67.1	23.6	0.91	0.19	0.10	0.99	0.59	84.1	24.8	36.8	48.2	31.7	13.3	5.87	8.61
HLS Sp Concentrate (interpolated)	**2.85**	**1,483**	**27.0**	**2.79**	**6.00**	**67.1**	**23.6**	**0.91**	**0.19**	**0.10**	**1.00**	**0.59**	**84.2**	**24.9**	**36.9**	**48.3**	**31.8**	**13.4**	**5.91**	**8.72**
HLS Sink 2.80 SG	2.80	1,682	30.6	2.59	5.58	67.5	23.0	0.90	0.18	0.11	1.15	0.82	88.9	28.4	40.9	54.3	35.4	16.1	7.72	13.7
HLS Middling (--2.85 +2.70 SG)	-2.85 +2.70	823	15.0	0.79	1.69	72.7	17.0	0.67	0.23	0.20	3.52	2.35	13.2	15.0	14.8	19.5	21.8	14.1	11.6	19.1
HLS Middlings (interpolated)	-2.85 +2.70	818	14.9	0.78	1.69	72.8	17.0	0.66	0.23	0.20	3.53	2.35	13.0	14.9	14.7	19.4	21.7	14.0	11.6	19.0
HLS Tailings (-2.70 SG)	2.70	3,196	58.1	0.042	0.089	75.1	14.4	0.28	0.13	0.26	6.45	2.28	2.71	60.2	48.4	32.3	46.5	72.6	82.5	72.3
Head (Calc.)		5,497	100	0.89	1.92	72.6	17.2	0.51	0.16	0.21	4.54	1.84	100	100	100	100	100	100	100	100

Table 3: HLS Test Results of -6.3 mm +1 mm Fraction (@6.0% Li2O) - Stage

Combined HLS Products	HL SG	Weight		Assays (%)									Distribution (%)							
	g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MnO	CaO	Na_2O	K_2O	Li	SiO_2	Al_2O_3	Fe_2O_3	MnO	CaO	Na_2O	K_2O
HLS Sink 2.85 SG	2.85	747	22.6	2.91	6.25	64.8	25.1	1.05	0.24	0.12	0.96	0.77	88.5	20.0	35.3	42.8	39.7	13.8	5.04	8.88
HLS Sp Concentrate (interpolated)	**2.83**	**797**	**24.1**	**2.79**	**6.00**	**64.8**	**24.9**	**1.06**	**0.24**	**0.12**	**1.04**	**0.97**	**90.7**	**21.3**	**37.5**	**46.3**	**42.1**	**15.0**	**5.82**	**11.9**
HLS Sink 2.80 SG	2.80	859	26.0	2.64	5.69	64.8	24.7	1.08	0.24	0.13	1.14	1.21	92.6	22.9	40.1	50.7	45.0	16.4	6.86	16.0
HLS Middling (--2.85 +2.70 SG)	-2.85 +2.70	356	10.8	0.66	1.41	68.8	19.5	1.03	0.30	0.22	3.44	3.21	9.53	10.1	13.1	20.0	24.0	12.0	8.60	17.6
HLS Middlings (interpolated)	-2.83 +2.70	306	9.26	0.59	1.27	69.5	18.9	0.99	0.32	0.23	3.64	3.09	7.38	8.75	10.9	16.5	21.6	10.8	7.82	14.6
HLS Tailings (-2.70 SG)	2.70	2,199	66.6	0.022	0.047	77.3	12.4	0.31	0.075	0.22	5.59	2.16	1.96	70.0	51.6	37.2	36.3	74.2	86.4	73.5
Head (Calc.)		3,302	100	0.74	1.60	73.5	16.1	0.56	0.14	0.20	4.31	1.96	100	100	100	100	100	100	100	100

Table 4: HLS Products – Combined Fractions (@6.0% Li2O) - Stage

Product	Weight		Assays (%)									Distribution (%)							
	kg	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MnO	CaO	Na_2O	K_2O	Li	SiO_2	Al_2O_3	Fe_2O_3	MnO	CaO	Na_2O	K_2O
HLS Concentrate	2,281	25.9	2.79	6.00	66.3	24.1	0.97	0.21	0.11	1.01	0.72	86.4	23.6	37.1	47.5	35.3	13.9	5.88	10.0
HLS Middling	1,123	12.8	0.73	1.57	71.9	17.5	0.75	0.26	0.21	3.56	2.55	11.2	12.6	13.3	18.3	21.7	12.9	10.2	17.3
HLS Tailings	5,394	61.3	0.034	0.072	76.0	13.6	0.29	0.11	0.25	6.10	2.23	2.46	63.9	49.6	34.2	43.1	73.2	83.9	72.7
Head (calc.)	**8,798**	**100**	**0.84**	**1.80**	**72.9**	**16.8**	**0.53**	**0.15**	**0.21**	**4.46**	**1.88**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

The interpolated stage results for each fraction needed to generate concentrate with a target lithium grade of 6.0% Li_2O are shown in Table 2 and Table 3. The -12.7 mm +6.3 mm fraction was projected to produce a 6.0% Li_2O concentrate with 84.2% lithium recovery (stage) at an SG cut point of 2.85. Similarly, the SG cut points to generate a 6.0% Li_2O concentrate for -6.3 mm +1 mm fraction was estimated at SG 2.83 but with higher lithium recovery (stage) at 90.7%. The stage results of the combined products from both fractions are presented in Table 4, which indicates the combined stage recovery to produce a 6% Li_2O concentrate to be 86.4% for a -12.7 mm +1 mm feed.

3.4. HLS Test Results - Global

The global metallurgical mass balance for both the -12.7 mm +6.3 mm and the -6.3 mm +1 mm fraction, plus the -1 mm fines fraction, is presented in Table 5. The composition of the combined product resulting from the global HLS tests is provided in Table 6. The global HLS spodumene concentrate recovered 81.5% of the lithium in 22.9% of the mass at a grade of 6.0% Li_2O. The strong lithium beneficiation performance observed in the test provided a strong indication that Dense Media Separation will be a viable unit operation for the production of high-grade spodumene concentrate from the Raleigh Lake.

The combined HLS middling and fines fraction contained 16.2% of the lithium distribution at a grade of 1.19% Li_2O, which would be a potential feed to a flotation circuit to improve recovery. This is shown on the bottom line of Table 6. Flotation testwork is recommended to evaluate the potential for further spodumene concentration from the -1 mm fraction.

Table 5: Global HLS Test Result – Metallurgical Mass Balance

	Product	HL SG	Weight		Assays (%)											Distribution (%)									
		g/cm3	kg	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5	Li	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5
-12.7 +6.3 mm	HLS Concentrate	2.85	1,483	14.9	2.79	**6.00**	67.1	23.6	**0.91**	0.015	0.19	0.10	1.00	0.59	0.010	53.0	13.7	21.0	26.4	7.12	17.7	7.07	3.17	4.81	12.9
	HLS Middling	-2.85 +2.70	818	8.21	0.78	1.69	72.8	17.0	0.66	0.049	0.23	0.20	3.53	2.35	0.010	8.21	8.18	8.35	10.6	12.8	12.1	7.42	6.20	10.5	7.09
	HLS Tailings	2.70	3,196	32.1	0.042	0.089	75.1	14.4	0.28	0.030	0.13	0.26	6.45	2.28	0.010	1.70	33.0	27.6	17.6	30.4	25.9	38.4	44.2	39.9	27.7
-6.3 +1 mm	HLS Concentrate	2.83	797	8.01	2.79	**6.00**	64.8	24.9	**1.06**	0.033	0.24	0.12	1.04	0.97	0.010	28.5	7.11	11.9	16.5	8.32	12.1	4.50	1.78	4.21	6.92
	HLS Middling	-2.83 +2.70	306	3.07	0.59	1.27	69.5	18.9	0.99	0.080	0.32	0.23	3.64	3.09	0.010	2.32	2.92	3.48	5.90	7.71	6.20	3.26	2.39	5.17	2.65
	HLS Tailings	2.70	2,199	22.1	0.022	0.047	77.3	12.4	0.31	0.027	0.075	0.22	5.59	2.16	0.017	0.61	23.4	16.4	13.3	18.9	10.4	22.3	26.4	26.0	32.7
	Fines Fraction (-1 mm)		1,160	11.6	0.38	0.82	73.6	16.1	0.43	0.040	0.21	0.32	6.37	1.49	0.010	5.65	11.7	11.2	9.72	14.7	15.5	17.0	15.9	9.45	10.1
	Head (calc.)		9,958	100	0.78	1.69	73.0	16.7	0.52	0.032	0.16	0.22	4.68	1.84	0.012	100	100	100	100	100	100	100	100	100	100
	Head (dir.)				0.74	1.59	73.2	16.8	0.56	0.050	0.15	0.22	4.73	1.94	< 0.01										

Table 6: Global HLS Test Result – Combined Products

Product	Weight		Assays (%)											Distribution (%)									
	kg	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5	Li	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5
HLS Concentrate	**2,281**	**22.9**	**2.79**	**6.00**	**66.3**	**24.1**	**0.97**	**0.021**	**0.21**	**0.11**	**1.01**	**0.72**	**0.010**	**81.5**	**20.8**	**33.0**	**42.9**	**15.4**	**29.8**	**11.6**	**4.95**	**9.03**	**19.8**
HLS Middling	1,123	11.3	0.73	1.57	71.9	17.5	0.75	0.058	0.26	0.21	3.56	2.55	0.010	10.5	11.1	11.8	16.5	20.5	18.3	10.7	8.59	15.7	9.75
HLS Tailings	5,394	54.2	0.034	0.072	76.0	13.6	0.29	0.029	0.11	0.25	6.10	2.23	0.013	2.32	56.4	44.0	30.9	49.3	36.4	60.7	70.6	65.9	60.4
Fines Fraction (-1 mm)	1,160	11.6	0.38	0.82	73.6	16.1	0.43	0.040	0.21	0.32	6.37	1.49	0.010	5.65	11.7	11.2	9.72	14.7	15.5	17.0	15.9	9.45	10.1
Head (calc.)	9,958	100	0.78	1.69	73.0	16.7	0.52	0.032	0.16	0.22	4.68	1.84	0.012	100	100	100	100	100	100	100	100	100	100
			0.74	1.59	73.2	16.8	0.56	0.050	0.15	0.22	4.73	1.94	< 0.01										
Flotation Feed	2,283	22.9	0.55	1.19	72.7	16.8	0.59	0.049	0.23	0.26	4.99	2.01	0.010	16.2	22.8	23.0	26.2	35.2	33.8	27.7	24.4	25.1	19.8

Conclusions and Recommendations

The following conclusions can be drawn from the testwork completed on the lithium sample from the Raleigh Lake deposit:

- The lithium sample tested was comprised mostly of silicates and alumina silicates (~90%) and contained 1.59% Li_2O and 0.56% Fe_2O_3.
- HLS results on two size fractions of the lithium sample (-12.7 mm +6.3 mm and -6.3 mm +1 mm) indicated excellent potential for spodumene beneficiation of the Raleigh Lake Ores by dense media separation, producing a concentrate grading 6.0% Li_2O at a global lithium recovery of >80%.
- The grade of iron in the spodumene concentrate was ~1% Fe_2O_3, which could likely be reduced by treating the concentrate by magnetic separation.

The following recommendations are made for further testwork:

- An investigation of HLS performance at different crushing sizes to determine the optimum feed size for the DMS plant.
- Treat a HLS concentrate by magnetic separation to determine the potential for lowering iron levels in the concentrate.
- Perform a DMS pilot campaign with the selected size fraction and the SG-cut points indicated from HLS testwork.
- Evaluate the potential for increased lithium recovery with flotation testwork on HLS / DMS middling plus the fine fraction.
- Develop and confirm the final process flowsheet to produce the final spodumene concentrate.
- HLS and flotation testwork should be conducted on a number of variability samples to confirm the amenability of various samples to the developed flowsheet.

Appendix A – Heavy Liquid Separation Testwork

Project	19977-01	**Date**	27-Sep-23
Client	International Lithium		

HLS Test Results

Test Conditions	**HLS1**
Crush size	-12.7 mm
Feed size	-12.7mm +6.3mm
Sample	Lithium

HLS Products	**HL SG**	**Weight**		**Assays (%)**											**Distribution (%)**									
	g/cm^3	**g**	**%**	**Li**	**Li_2O**	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**MgO**	**MnO**	**CaO**	**Na_2O**	**K_2O**	**P_2O_5**	**Li**	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**MgO**	**MnO**	**CaO**	**Na_2O**	**K_2O**	**P_2O_5**
HLS Sink 2.90 SG	2.90	1,293	23.5	2.96	6.37	66.5	24.2	0.92	0.010	0.19	0.10	0.88	0.38	0.010	78.0	21.6	33.0	42.5	8.15	28.1	11.2	4.56	4.87	23.5
HLS Sink 2.85 SG	2.85	185	3.36	1.62	3.49	70.9	19.4	0.86	0.050	0.17	0.13	1.77	2.04	0.010	6.10	3.29	3.79	5.68	5.83	3.59	2.08	1.31	3.74	3.36
HLS Sink 2.80 SG	2.80	204	3.71	1.15	2.48	70.5	19.0	0.84	0.060	0.16	0.16	2.26	2.53	0.010	4.77	3.60	4.09	6.11	7.70	3.72	2.82	1.84	5.11	3.71
HLS Sink 2.75 SG	2.75	249	4.54	0.93	2.00	73.9	16.0	0.59	0.040	0.22	0.24	2.71	2.47	0.010	4.73	4.62	4.21	5.25	6.28	6.27	5.18	2.71	6.10	4.54
HLS Sink 2.70 SG	2.70	369	6.72	0.49	1.05	73.2	16.6	0.62	0.050	0.28	0.19	4.77	2.16	0.010	3.69	6.78	6.47	8.18	11.6	11.8	6.08	7.06	7.91	6.72
HLS Sink 2.65 SG	2.65	938	17.1	0.12	0.26	78.9	12.8	0.41	0.030	0.29	0.22	5.47	1.25	0.010	2.29	18.5	12.7	13.7	17.7	31.1	17.9	20.5	11.6	17.1
HLS Float 2.65 SG	2.65	2,258	41.1	0.009	0.019	73.5	15.0	0.23	0.030	0.060	0.28	6.85	2.71	0.010	0.41	41.6	35.7	18.5	42.7	15.5	54.8	62.0	60.6	41.1
Feed (Calc.)		5,497	100	0.89	1.92	72.6	17.2	0.51	0.029	0.16	0.21	4.54	1.84	0.010	100	100	100	100	100	100	100	100	100	100

Combined Sinks Results

Combined HLS Products	**HL SG**	**Weight**		**Assays (%)**											**Distribution (%)**									
	g/cm^3	**g**	**%**	**Li**	**Li_2O**	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**MgO**	**MnO**	**CaO**	**Na_2O**	**K_2O**	**P_2O_5**	**Li**	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**MgO**	**MnO**	**CaO**	**Na_2O**	**K_2O**	**P_2O_5**
HLS Sink 2.90 SG	2.90	1,293	23.5	2.96	6.37	66.5	24.2	0.92	0.010	0.19	0.10	0.88	0.38	0.010	78.0	21.6	33.0	42.5	8.15	28.1	11.2	4.56	4.87	23.5
HLS Sink 2.85 SG	2.85	1,478	26.9	2.79	6.01	67.1	23.6	0.91	0.015	0.19	0.10	0.99	0.59	0.010	84.1	24.8	36.8	48.2	14.0	31.7	13.3	5.87	8.61	26.9
HLS Sink 2.80 SG	2.80	1,682	30.6	2.59	5.58	67.5	23.0	0.90	0.020	0.18	0.11	1.15	0.82	0.010	88.9	28.4	40.9	54.3	21.7	35.4	16.1	7.72	13.7	30.6
HLS Sink 2.75 SG	2.75	1,931	35.1	2.38	5.12	68.3	22.1	0.86	0.023	0.19	0.13	1.35	1.04	0.010	93.6	33.1	45.1	59.5	28.0	41.6	21.3	10.4	19.8	35.1
HLS Sink 2.70 SG	2.70	2,301	41.9	2.08	4.47	69.1	21.2	0.82	0.027	0.20	0.14	1.90	1.22	0.010	97.3	39.8	51.6	67.7	39.6	53.5	27.4	17.5	27.7	41.9
HLS Sink 2.65 SG	2.65	3,239	58.9	1.51	3.25	71.9	18.8	0.70	0.028	0.23	0.16	2.93	1.23	0.010	99.6	58.4	64.3	81.5	57.3	84.5	45.2	38.0	39.4	58.9

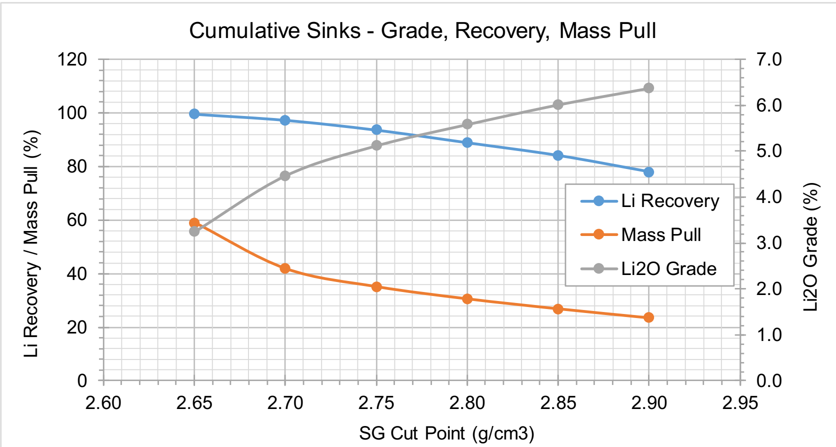
Combined Floats Results

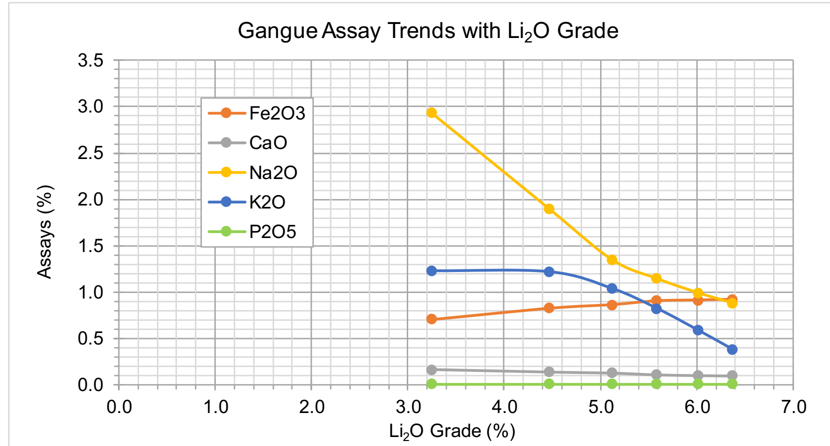
Combined HLS Products	**HL SG**	**Weight**		**Assays (%)**											**Distribution (%)**									
	g/cm^3	**g**	**%**	**Li**	**Li_2O**	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**MgO**	**MnO**	**CaO**	**Na_2O**	**K_2O**	**P_2O_5**	**Li**	**SiO_2**	**Al_2O_3**	**Fe_2O_3**	**MgO**	**MnO**	**CaO**	**Na_2O**	**K_2O**	**P_2O_5**
HLS Float 2.90 SG	2.90	4,203	76.5	0.26	0.55	74.4	15.1	0.38	0.035	0.15	0.24	5.67	2.28	0.010	22.0	78.4	67.0	57.5	91.9	71.9	88.8	95.4	95.1	76.5
HLS Float 2.85 SG	2.85	4,018	73.1	0.19	0.42	74.6	14.9	0.36	0.034	0.15	0.25	5.85	2.29	0.010	15.9	75.2	63.2	51.8	86.0	68.3	86.7	94.1	91.4	73.1
HLS Float 2.80 SG	2.80	3,815	69.4	0.14	0.31	74.8	14.7	0.34	0.033	0.15	0.25	6.04	2.28	0.010	11.1	71.6	59.1	45.7	78.3	64.6	83.9	92.3	86.3	69.4
HLS Float 2.75 SG	2.75	3,565	64.9	0.088	0.19	74.9	14.6	0.32	0.032	0.14	0.25	6.27	2.27	0.010	6.40	66.9	54.9	40.5	72.0	58.4	78.7	89.6	80.2	64.9
HLS Float 2.70 SG	2.70	3,196	58.1	0.042	0.089	75.1	14.4	0.28	0.030	0.13	0.26	6.45	2.28	0.010	2.71	60.2	48.4	32.3	60.4	46.5	72.6	82.5	72.3	58.1
HLS Float 2.65 SG	2.65	2,258	41.1	0.009	0.019	73.5	15.0	0.23	0.030	0.060	0.28	6.85	2.71	0.010	0.41	41.6	35.7	18.5	42.7	15.5	54.8	62.0	60.6	41.1

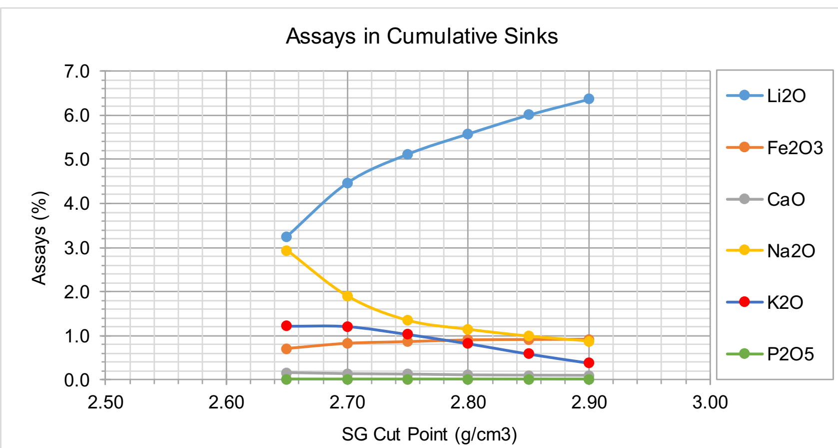
Project	19977-01	**Date**	27-Sep-23
Client	International Lithium		

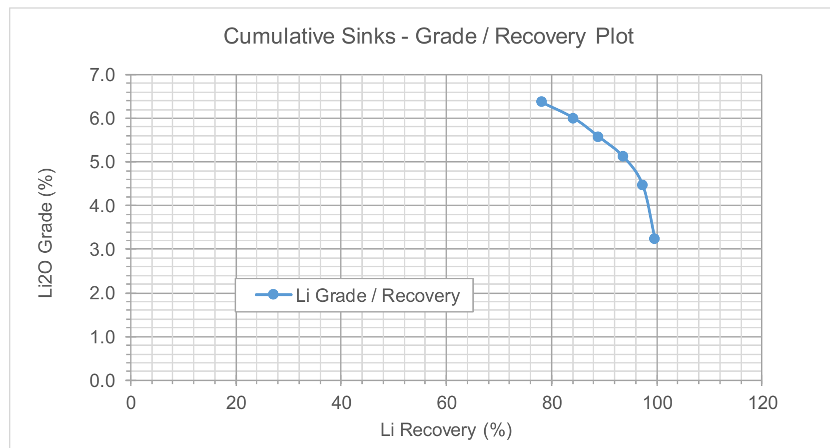
HLS Test Results

Test Conditions	**HLS1**
Crush size	-12.7 mm
Feed size	-12.7mm +6.3mm
Sample	Lithium









Project	19977-01	**Date**	27-Sep-23
Client	International Lithium		

HLS Test Results

Test Conditions	**HLS1**
Crush size	-12.7 mm
Feed size	-6.3mm +1mm
Sample	Lithium

HLS Products	HL SG	Weight		Assays (%)											Distribution (%)									
	g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5	Li	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5
HLS Sink 2.90 SG	2.90	638	19.3	3.11	6.69	64.9	25.3	1.02	0.020	0.24	0.12	0.81	0.42	0.010	81.0	17.1	30.5	35.5	11.6	33.9	11.7	3.63	4.14	13.1
HLS Sink 2.85 SG	2.85	109	3.29	1.70	3.66	64.5	23.6	1.23	0.080	0.24	0.13	1.85	2.82	0.010	7.53	2.89	4.83	7.29	7.88	5.78	2.15	1.41	4.73	2.23
HLS Sink 2.80 SG	2.80	112	3.40	0.90	1.94	64.4	22.7	1.28	0.10	0.21	0.15	2.30	4.13	0.010	4.12	2.98	4.81	7.84	10.2	5.23	2.57	1.81	7.17	2.31
HLS Sink 2.75 SG	2.75	81.2	2.46	0.71	1.53	68.7	19.6	1.16	0.080	0.26	0.20	3.35	3.16	0.010	2.35	2.30	3.00	5.14	5.90	4.68	2.48	1.91	3.97	1.67
HLS Sink 2.70 SG	2.70	163	4.92	0.46	0.99	71.8	17.2	0.79	0.070	0.39	0.28	4.27	2.59	0.010	3.05	4.81	5.27	7.01	10.3	14.0	6.94	4.87	6.51	3.34
HLS Sink 2.65 SG	2.65	633	19.2	0.051	0.11	89.0	6.61	0.36	0.020	0.16	0.15	2.88	0.56	0.010	1.32	23.2	7.89	12.4	11.5	22.4	14.5	12.8	5.48	13.0
HLS Float 2.65 SG	2.65	1,566	47.4	0.010	0.022	72.5	14.8	0.29	0.030	0.040	0.25	6.69	2.81	0.020	0.64	46.8	43.7	24.8	42.6	13.9	59.7	73.6	68.0	64.3
Feed (Calc.)		3,302	100	0.74	1.60	73.5	16.1	0.56	0.033	0.14	0.20	4.31	1.96	0.015	100	100	100	100	100	100	100	100	100	100

Combined Sinks Results

Combined HLS Products	HL SG	Weight		Assays (%)											Distribution (%)									
	g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5	Li	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5
HLS Sink 2.90 SG	2.90	638	19.3	3.11	6.69	64.9	25.3	1.02	0.020	0.24	0.12	0.81	0.42	0.010	81.0	17.1	30.5	35.5	11.6	33.9	11.7	3.63	4.14	13.1
HLS Sink 2.85 SG	2.85	747	22.6	2.91	6.25	64.8	25.1	1.05	0.029	0.24	0.12	0.96	0.77	0.010	88.5	20.0	35.3	42.8	19.5	39.7	13.8	5.04	8.88	15.3
HLS Sink 2.80 SG	2.80	859	26.0	2.64	5.69	64.8	24.7	1.08	0.038	0.24	0.13	1.14	1.21	0.010	92.6	22.9	40.1	50.7	29.7	45.0	16.4	6.86	16.0	17.7
HLS Sink 2.75 SG	2.75	940	28.5	2.48	5.33	65.1	24.3	1.09	0.042	0.24	0.13	1.33	1.38	0.010	95.0	25.2	43.1	55.8	35.6	49.6	18.9	8.77	20.0	19.3
HLS Sink 2.70 SG	2.70	1,103	33.4	2.18	4.69	66.1	23.3	1.04	0.046	0.26	0.15	1.76	1.56	0.010	98.0	30.0	48.4	62.8	45.9	63.7	25.8	13.6	26.5	22.7
HLS Sink 2.65 SG	2.65	1,736	52.6	1.40	3.02	74.5	17.2	0.79	0.036	0.22	0.15	2.17	1.19	0.010	99.4	53.2	56.3	75.2	57.4	86.1	40.3	26.4	32.0	35.7

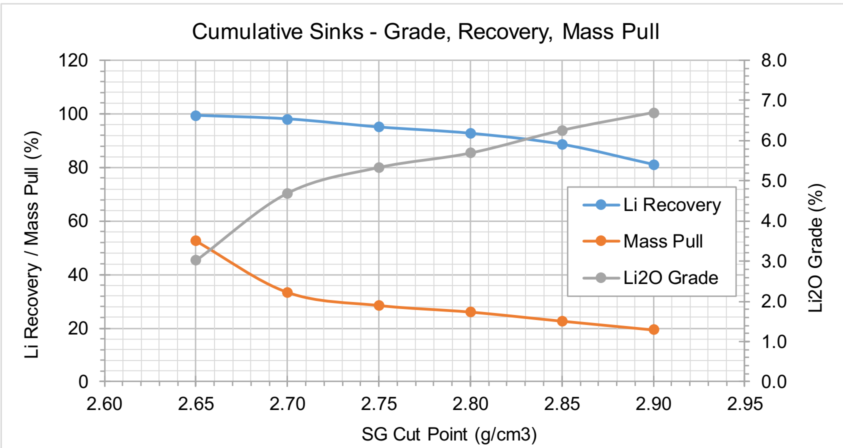
Combined Floats Results

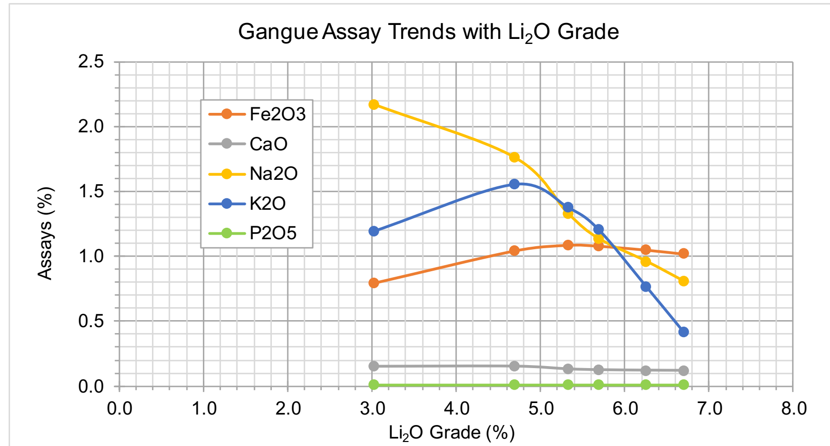
Combined HLS Products	HL SG	Weight		Assays (%)											Distribution (%)									
	g/cm^3	g	%	Li	Li_2O	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5	Li	SiO_2	Al_2O_3	Fe_2O_3	MgO	MnO	CaO	Na_2O	K_2O	P_2O_5
HLS Float 2.90 SG	2.90	2,663	80.7	0.17	0.38	75.6	13.8	0.44	0.037	0.11	0.22	5.15	2.33	0.016	19.0	82.9	69.5	64.5	88.4	66.1	88.3	96.4	95.9	86.9
HLS Float 2.85 SG	2.85	2,555	77.4	0.11	0.24	76.1	13.4	0.41	0.035	0.11	0.22	5.29	2.31	0.016	11.5	80.0	64.7	57.2	80.5	60.3	86.2	95.0	91.1	84.7
HLS Float 2.80 SG	2.80	2,442	74.0	0.074	0.16	76.6	13.0	0.37	0.032	0.10	0.22	5.43	2.22	0.016	7.36	77.1	59.9	49.3	70.3	55.0	83.6	93.1	84.0	82.3
HLS Float 2.75 SG	2.75	2,361	71.5	0.052	0.11	76.9	12.8	0.34	0.030	0.096	0.23	5.50	2.19	0.017	5.01	74.8	56.9	44.2	64.4	50.4	81.1	91.2	80.0	80.7
HLS Float 2.70 SG	2.70	2,199	66.6	0.022	0.047	77.3	12.4	0.31	0.027	0.075	0.22	5.59	2.16	0.017	1.96	70.0	51.6	37.2	54.1	36.3	74.2	86.4	73.5	77.3
HLS Float 2.65 SG	2.65	1,566	47.4	0.010	0.022	72.5	14.8	0.29	0.030	0.040	0.25	6.69	2.81	0.020	0.64	46.8	43.7	24.8	42.6	13.9	59.7	73.6	68.0	64.3

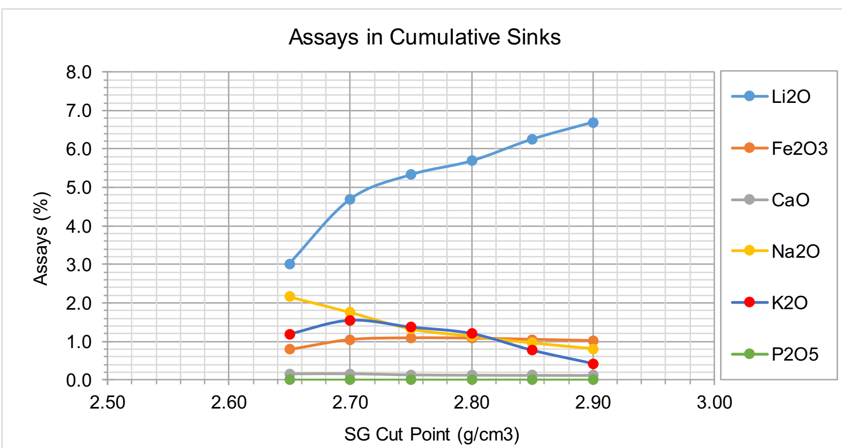
Project	19977-01	**Date**	27-Sep-23
Client	International Lithium		

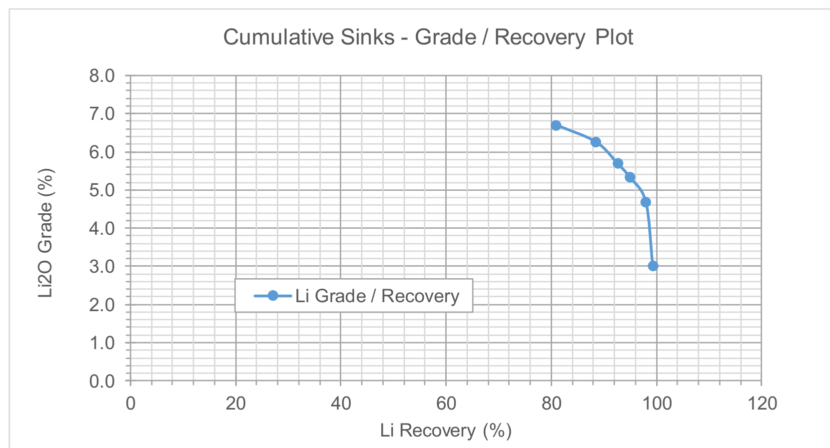
HLS Test Results

Test Conditions	**HLS1**
Crush size	-12.7 mm
Feed size	-6.3mm +1mm
Sample	Lithium









An Investigation into

THE MINERALOGICAL CHARACTERIZATION OF RUBIDIUM AND LITHIUM SAMPLES FROM THE RALEIGH LAKE ORES, ONTARIO

prepared for

INTERNATIONAL LITHIUM CORP.

Project 19977-01 – MI5018-SEP23 – Mineralogy Report
October 30, 2023

NOTES

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

Table of Contents

List of Tables

List of Figures

Executive Summary

Two samples, referred to as Rb Head and Li Head, were received by the SGS Advanced Mineralogy Facility from the International Lithium Corp. for mineralogical examination. The samples are taken from the Raleigh Lake Ores, Ontario. The testwork was conducted to support the metallurgical testwork currently underway. The project number 19977-01 and LIMS number MI5018-SEP23 were assigned to the testwork.

Sample preparation and homogenization was conducted (more details in the metallurgical report).

A sample charge was used from the mineralogy. A subsample from the Rb Head of about 1 kg was stage-crushed to a P_{80} of ca. 600-700 µm. The Rb Head was analyzed as a single size fraction.

The Li Head represents the -1 mm material from the initial head sample. The Li Head was further stage crushed into 425 µm, screen and recombined into four size fractions including +425 µm, -425/+300 µm, -300/+150 µm, and -150 µm.

The mineralogical work was conducted with TIMA-X (Tescan Integrated Mineral Analyzer), Electron Probe Micro-Analysis (EPMA), Laser Ablation by Inductively Coupled Plasma Mass Spectrometry (LA by ICP-MS), X-ray diffraction analysis (XRD), and chemical assays. The purpose of this test program was to determine the overall mineral assemblage of the sample and define the liberation and association attributes of rubidium minerals in the Rb Head and lithium minerals in the Li Head.

X-Ray Diffraction (XRD) Analysis

XRD analysis indicates that the two samples consist of a different crystalline mineral assemblage. The Rb Head consists mainly of microcline (46.6%), albite (33.2%), minor quartz (13.2%), spodumene (2.0%), muscovite (2.0%), and diopside (2.6%). The Li Head consists of albite (58.1%), quartz (21.4%), minor spodumene (8.0%), microcline (8.4%), and diopside (1.1%) (Table 1).

Table 1: XRD Results for the Rb Head and Li Head Samples

Mineral/Compound	Rb-head	Li-head
Quartz	13.2	21.4
Albite	33.2	58.1
Spodumene	2.5	8.0
Muscovite	2.0	3.0
Microcline	46.6	8.4
Diopside	2.6	1.1
Total	100	100

Mineralogical Results from TIMA-X Analysis for the Rb Head

Modal Mineralogy

The mineral abundance is given in wt% for each sample from the TIMA-X analysis and it is graphically illustrated in Figure 1.

The Rb Head consists mainly of K-feldspars (i.e., microcline as defined by XRD) (49.7%), plagioclase (33.1%), and quartz (14.9%), and minor muscovite and spodumene. Note the trace amounts of tourmaline and beryl.

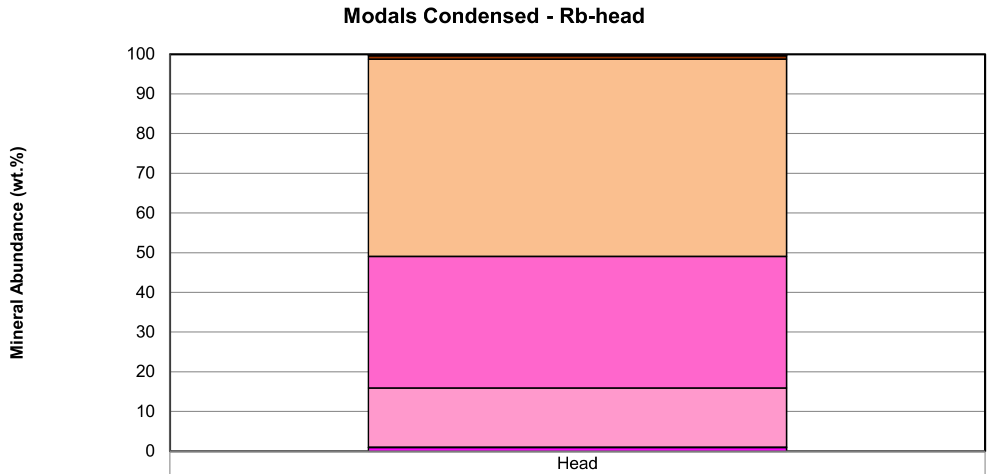
Grain Size Distribution

The grain size report serves to study the distribution of the grain size of a specific phase; within the TIMA software, it is defined as equivalent circle diameter (*d*). It is the diameter of a circle that has the same area (*A*) as the particle (or grain). The diameter is defined in pixels and then multiplied by pixel spacing (*Ps*) to obtain size in micrometres. The precise definition is described in the following formula: $d = 2 \cdot \sqrt{A / \pi} \cdot Ps$.

The P_{80} (Table 2) for the particle is 708 µm, K-feldspars 723 µm, plagioclase 544 µm, quartz 684 µm, and muscovite 221 µm. The term particle refers to both liberated and middling particles, monomineralic, and polymineralic.

Table 2: P_{80} Size of Selected Minerals Calculated for the Rb Head Sample

Sample	**Rb-head**	
Grain Size (µm)	Median	P80
Spodumene	515	810
Petalite/ Cookeite	10	14
Beryl/ Tourmaline	14	84
Quartz	453	684
Plagioclase	335	544
K-Feldspars	422	723
Muscovite	112	221
Other Silicates	475	476
Garnet	304	457
Carbonates	77	161
Phosphates	38	53
Fe-(Ti)-Oxides	6	9
Nb-Ta-Oxides	25	27
Sulphides	232	269
Other	23	42
Particle	412	708



	Head
Other	0.01
Sulphides	0.07
Nb-Ta-Oxides	0.00
Fe-(Ti)-Oxides	0.01
Phosphates	0.00
Carbonates	0.06
Garnet	0.17
Other Silicates	0.11
Muscovite	0.81
K-Feldspars	49.7
Plagioclase	33.1
Quartz	14.9
Beryl/Tourmaline	0.02
Petalite/Cookeite	0.04
Spodumene	0.94

Figure 1: Summary of Modal Mineralogy for the Rb Head

Mineral Chemistry

Electron Probe Micro-Analyses were conducted mainly on K-feldspars and muscovite, while a few analyses of beryl, plagioclase, and chlorite were also performed. The minimum (Min), maximum (Max), and average values from the EPMA are given in Table 3. Back scattered electron (BSE) microscope images of the minerals analyzed are given in Figure 2. The point analyses correspond to the analyses provided in the Appendix.

In the K-feldspars, Rb_2O ranges from 0.01% to 2.39% and averages 1.42%, and Cs_2O ranges from below the detection limit to 0.27% and avg. 0.07%.

In the muscovite, Rb_2O ranges from 0.50% to 2.87% and averages 1.90%, and Cs_2O ranges from below the detection limit to 0.84% and avg. 0.19%.

Note that rubidium is below the detection limit of the instrument in plagioclase. Beryl and chlorite host traces of rubidium. Note that the analyzed beryl hosts an average of 2.25% Cs_2O.

Table 3: Range of Chemistry of K-feldspars, Muscovite, Beryl, Plagioclase and Chlorite from the EPMA for the Rb Head

Oxide/Mineral	K-feldspar (n=52)			Muscovite (n=61)			Beryl (n=3)			Plagioclase (n=4)			Chlorite (n=3)		
Oxide wt%	Min	Max	Avg.	Min	Max	Avg.	Min	Max	Avg.	Min	Max	Avg.	Min	Max	Avg.
SiO2	62.94	66.17	64.94	43.60	53.20	45.22	63.87	64.64	64.29	59.52	68.84	64.79	21.49	33.85	29.27
TiO2	-0.07	0.10	0.00	-0.02	3.08	0.26	-0.02	0.03	0.01	-0.02	0.06	0.03	-0.02	0.04	0.02
ZnO	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
Al2O3	17.97	18.71	18.34	28.06	37.35	32.85	17.07	17.23	17.13	19.44	25.59	22.09	20.02	23.20	21.54
V2O3	-0.04	0.04	0.00	-0.04	0.06	0.02	0.00	0.04	0.02	-0.02	0.02	0.00	0.03	0.04	0.03
Cr2O3	-0.08	0.01	-0.05	-0.11	-0.03	-0.06	-0.06	-0.01	-0.04	-0.05	0.00	-0.02	-0.53	-0.09	-0.25
Sc2O3	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
FeO	-0.03	0.11	0.01	1.00	6.65	3.70	0.13	0.19	0.16	0.01	0.03	0.02	17.90	29.37	22.89
MnO	-0.02	0.03	0.00	0.05	0.39	0.20	-0.01	0.00	-0.01	0.00	0.01	0.01	0.85	7.19	3.10
BeO	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
MgO	-0.02	0.04	0.00	0.00	2.96	0.51	0.01	0.09	0.04	-0.02	0.01	-0.01	3.15	11.55	7.08
CaO	-0.01	0.46	0.02	-0.01	0.24	0.02	0.00	0.02	0.01	0.24	7.31	3.22	0.01	0.26	0.12
SrO	0.00	0.10	0.05	0.00	0.08	0.05	0.00	0.02	0.01	0.02	0.07	0.04	-0.01	0.03	0.01
BaO	-0.13	0.17	0.02	-0.10	0.23	0.02	-0.07	0.06	0.00	-0.08	0.09	0.00	-0.04	0.07	0.01
Na2O	0.08	1.33	0.43	0.03	0.53	0.23	1.17	1.70	1.44	7.52	11.90	10.01	0.00	0.04	0.02
K2O	14.32	16.61	15.47	9.13	10.63	9.98	0.04	0.06	0.05	0.05	0.13	0.08	0.04	2.70	1.53
Rb2O	0.01	2.39	1.42	0.50	2.87	1.90	0.00	0.06	0.02	-0.14	-0.05	-0.09	-0.03	0.16	0.09
Cs2O	-0.03	0.27	0.07	-0.03	0.84	0.19	1.32	3.38	2.25	-0.05	0.02	0.00	-0.01	0.01	0.00
Cl	-0.01	0.03	0.00	-0.01	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.03	0.01
F	-0.19	0.12	-0.01	-0.09	0.96	0.39	-0.15	0.03	-0.04	-0.10	0.02	-0.03	-0.16	0.01	-0.08
O	-0.05	0.08	0.00	-0.40	0.04	-0.16	-0.01	0.06	0.02	-0.01	0.04	0.01	-0.01	0.06	0.03
H2O*	0.00	0.00	0.00	4.24	4.50	4.37	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total	98.88	101.93	100.71	97.18	102.87	99.68	84.74	86.00	85.37	99.41	100.81	100.12	80.61	88.55	85.42

* calculated

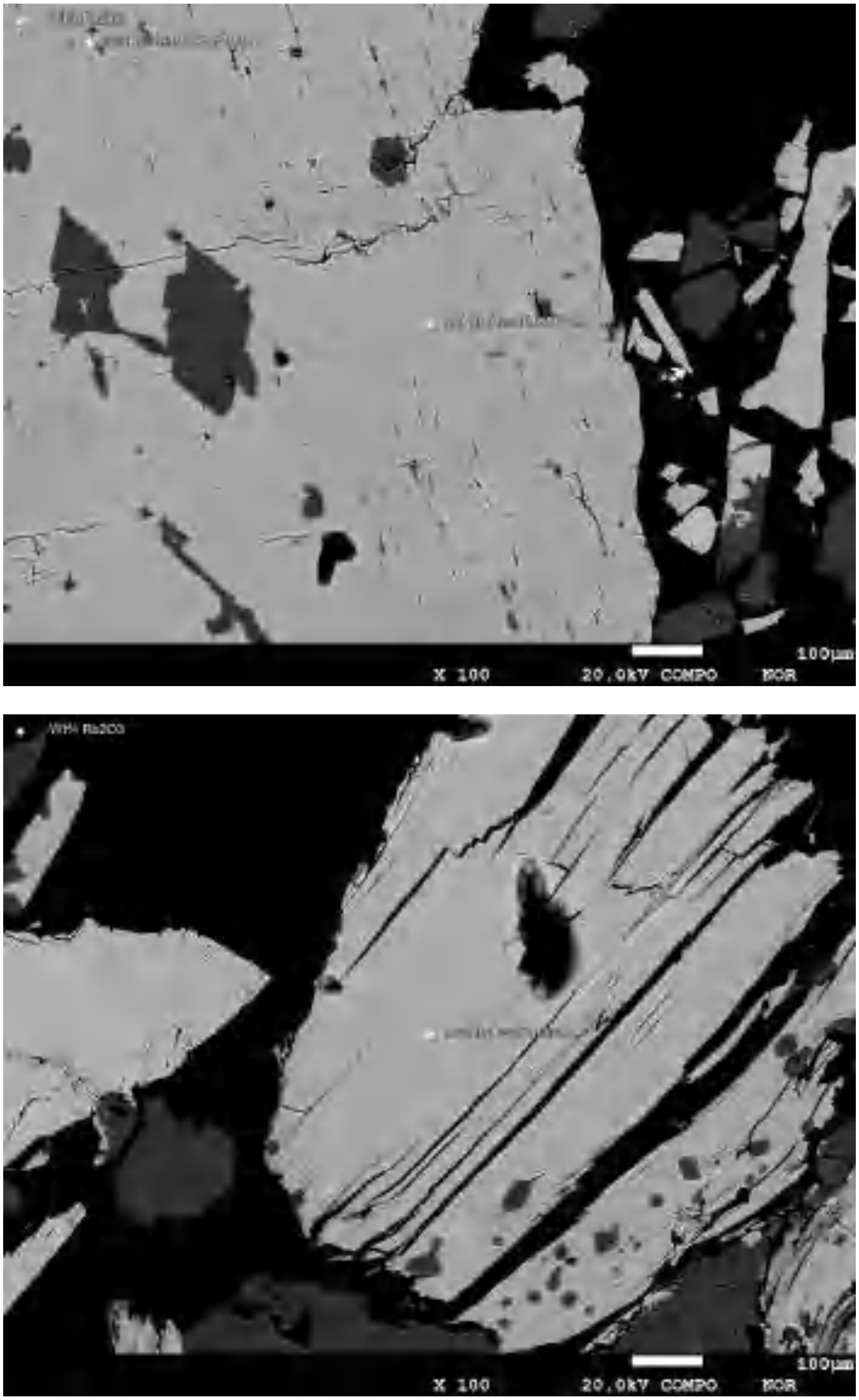


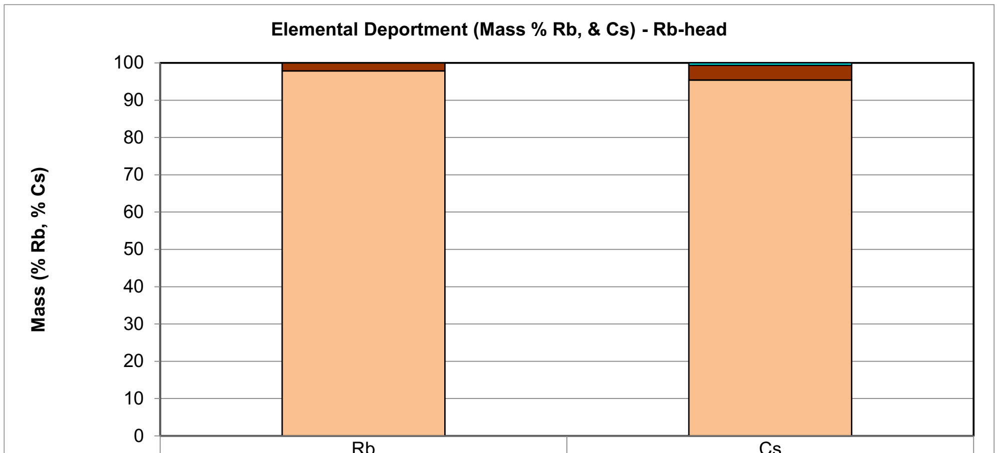
Figure 2: BSE Images of the Analyzed Minerals for the Rb Head

Images illustrate a K-feldspar grain 9above) and a mica grain (below) and the spots (white dots) that were analyzed with the Electron Microrprobe.

Rubidium and Cesium Deportment

Figure 3 illustrates the rubidium and cesium distribution by mineral in the sample. This is calculated based on the mass of the minerals from the TIMA-X analysis and mineral chemistry from the EPMA.

K-feldspars hosts ca. 97.8% of the total rubidium and 95.4% of the total cesium in the sample. The remainder is hosted by muscovite at 2.1% and 4.0%, and trace amounts by chlorite (nil and 0.6%, respectively). Note that both rubidium and cesium vary significantly in the grains analyzed. The average rubidium and cesium values are used or these calculations. Therefore, some deviations in the rubidium and cesium distribution might be expected.



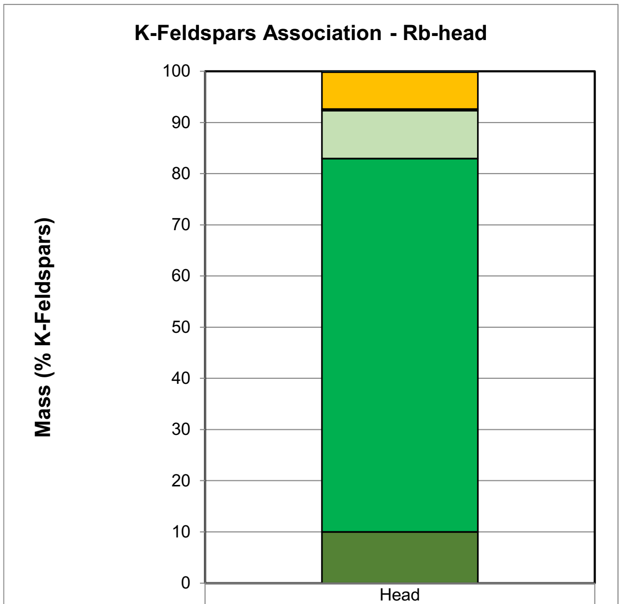
	Rb	Cs
Beryl	0.0	0.6
Chlorite	0.0	0.0
Muscovite	2.1	4.0
K-Feldspars	97.9	95.4

Figure 3: Rubidium and Cesium Deportment (Normalized Mass%) in the Rb Head

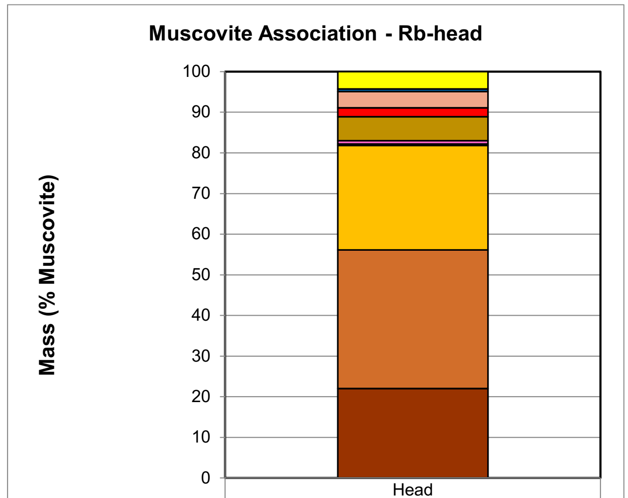
Liberation and Association of K-feldspars and Muscovite

Liberated (pure, free, and liberated) K-feldspars account for 92.4%. The non-liberated grains occur as middlings (binary) with plagioclase (7.2%), and other minerals in trace amounts (<1%) (Figure 4).

Liberated (pure, free, and liberated) muscovite accounts for 81.8%. The non-liberated grains occur as complex (ternary and quaternary middling particles) (4.3%), and middlings (binary) with K-feldspars (2.2%), ternary associations with quartz and feldspars (4.0%, both K-feldspars and plagioclase), and other minerals in minor associations (<1% each) (Figure 5).



	Head
K-Feld:Other	0.15
K-Feld:Plagioclase	7.21
K-Feld:Quartz	0.29
Lib K-Feldspars	9.40
Free K-Feldspars	73.0
Pure K-Feldspars	9.98



	Head
Complex	4.25
Ms:Other	0.66
Ms:Quartz:Feldspars	4.03
Ms:K-Feldspars	2.18
Ms:Plagioclase	5.87
Ms:Quartz	0.81
Ms:Beryl/Tourmaline	0.42
Lib Muscovite	25.7
Free Muscovite	34.1
Pure Muscovite	22.0

Figure 4: Liberation (Normalized Mass%) of K-feldspars and Muscovite in Rb Head

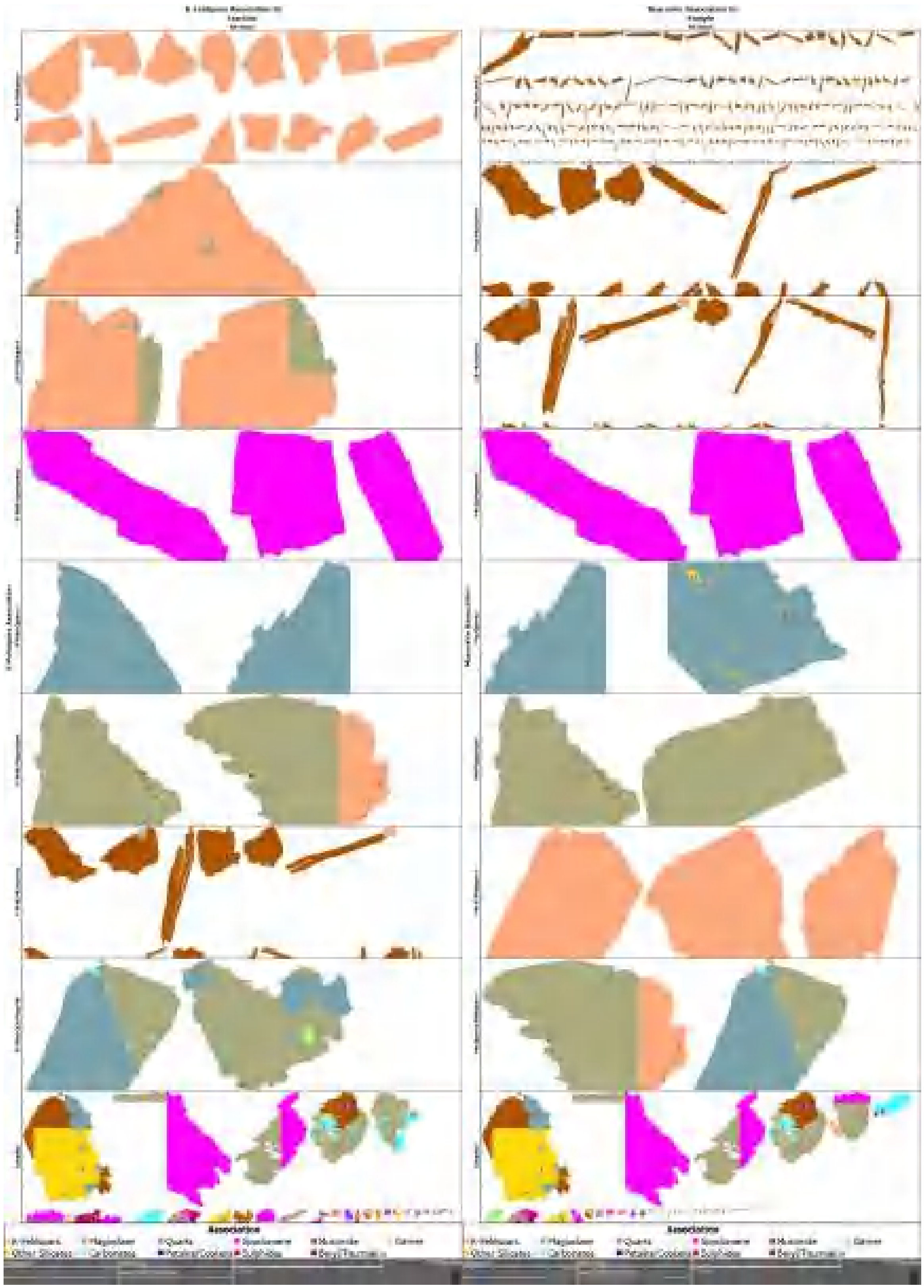
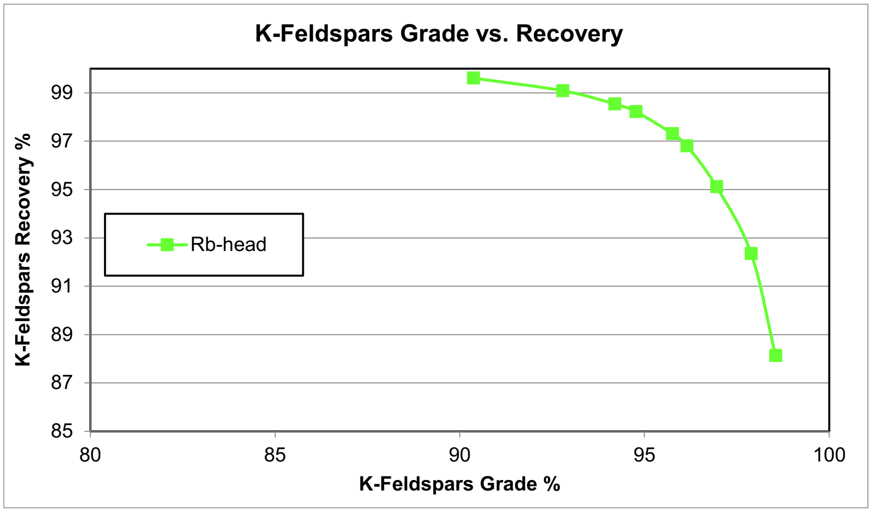


Figure 5: Image Grid Based on Liberation and Association of K-feldspars (left) and Muscovite (right) in the Rb Head

Grade and Recovery of K-feldspars and Muscovite

Another, more functional, method of presenting liberation is the mineralogically limiting grade-recovery curves, as shown below. They are based on the calculated mass of minerals and the total mass in each liberation category. Thus, the highest grade (>80% e.g., muscovite) is contained in the >80% liberated muscovite particles. Then the next category (60% to 80% liberation) is added and the combined grade is calculated. This is repeated until all muscovite is accounted for. Mineralogically limited grade-recovery analyses provides an indication of the theoretical maximum achievable elemental or mineral grade by recovery, based on individual particle liberation and grade. These results, of course, do not reflect any other recovery factors that could occur in the actual metallurgical process.

Figure illustrates the grade and recovery of K-feldspars and muscovite for the Rb Head sample. K-feldspar grades of ca. 99% to 94% for K-feldspar recoveries of 88% to ca. 99% are projected at the current grind size. Similarly, muscovite grades of ca. 98% to 94% for muscovite recoveries of 74% to ca. 87% are projected for the sample at the current grind size.



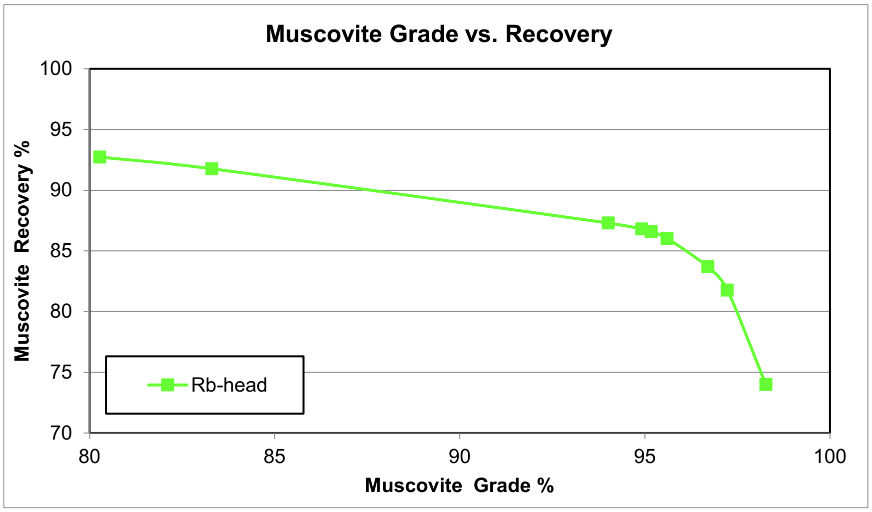


Figure 6: Grade-Recovery Curves for K-feldspars and Muscovite for the Rb Head

Mineralogical Results for the Li Head

Mass Balance and Lithium Distribution

The lithium values from the whole rock analysis, mass (wt%) and lithium distribution for each size fraction are shown in Table 4. The wt% and lithium distribution are graphically illustrated in Figure 7.

The lithium distribution reflects both the wt% distribution of the size fractions and the spodumene grade in each fraction. Most of the lithium is recorded in the +425 µm fraction (32.1%), followed by the -300/+150 µm (26.3%), -425/+300 µm (22.4%), and -150 µm (19.3%). The calculated lithium grade for the Li Head sample is 0.36%.

Table 4: Lithium Assays, Weight Distribution, and Lithium Distribution (Dist.) by Size Fraction for the Li Head

Fraction	Chemical Li%	wt% Dist.	Li% Dist.
+425 um	0.59	19.40	32.1
-425/+300 um	0.36	22.20	22.4
-300/+150 um	0.27	34.70	26.3
-150 um	0.29	23.70	19.3

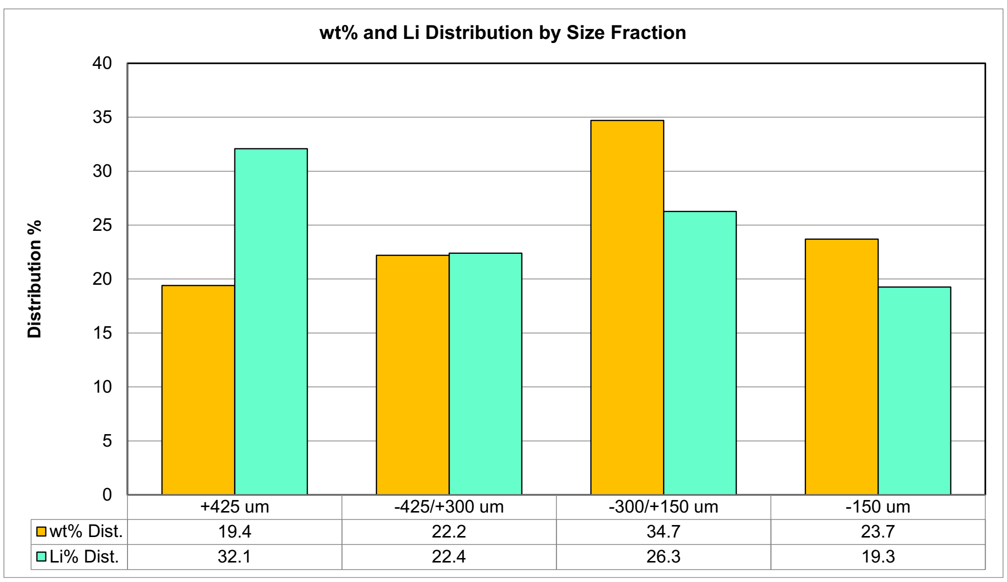


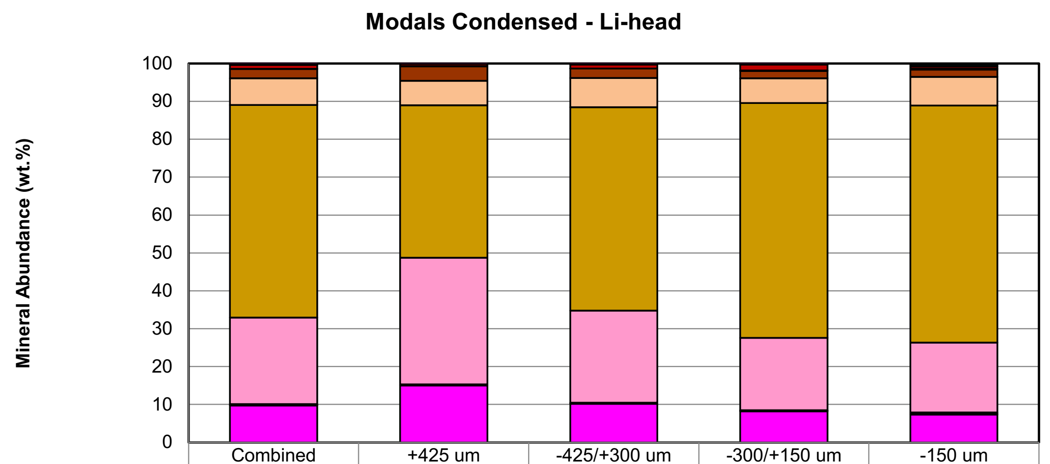
Figure 7: Weight% and Li% Distribution by Size Fraction for the Li Head

Modal Mineralogy

Mineral abundance is presented in wt% by size fraction and calculated Li Head (Figure 8). The Li Head (calculated head) consists of plagioclase (56.1%), quartz (22.9%), spodumene (9.7%) K-feldspars (7.0%), muscovite (2.4%), garnet (1.0%), and trace amounts of other minerals.

Beryl, petalite, and potentially lithium-bearing phosphates are tentatively identified and are present in <1% collectively. Ta-Nb-phases account for 0.02%.

The distribution of spodumene, absolute by sample, and calculated head is given in Figure 9. Spodumene varies between 1.7% and 2.9% among the fractions including +425 µm fraction (2.9%), -425/+300 µm (2.3%), -300/+150 µm (2.8%), and -150 µm (1.7%).



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Other	0.03	0.01	0.02	0.03	0.07
Sulphides	0.01	0.00	0.00	0.00	0.03
Nb-Ta-Oxides	0.02	0.00	0.00	0.02	0.03
Fe-(Ti)-Oxides	0.03	0.01	0.00	0.02	0.11
Phosphates	0.02	0.00	0.00	0.04	0.01
Carbonates	0.25	0.08	0.26	0.19	0.47
Garnet	1.03	0.58	1.02	1.56	0.63
Other Silicates	0.12	0.06	0.05	0.10	0.26
Muscovite	2.43	3.82	2.51	1.95	1.92
K-Feldspars	7.02	6.50	7.68	6.51	7.57
Plagioclase	56.1	40.2	53.7	62.0	62.6
Quartz	22.9	33.4	24.3	19.1	18.4
Beryl/Tourmaline	0.21	0.21	0.17	0.14	0.37
Petalite/Cookeite	0.15	0.09	0.14	0.13	0.25
Spodumene	9.73	15.0	10.2	8.19	7.29

Figure 8: Summary of Modal Mineralogy by Size Fraction and Calculated Head for the Li Head

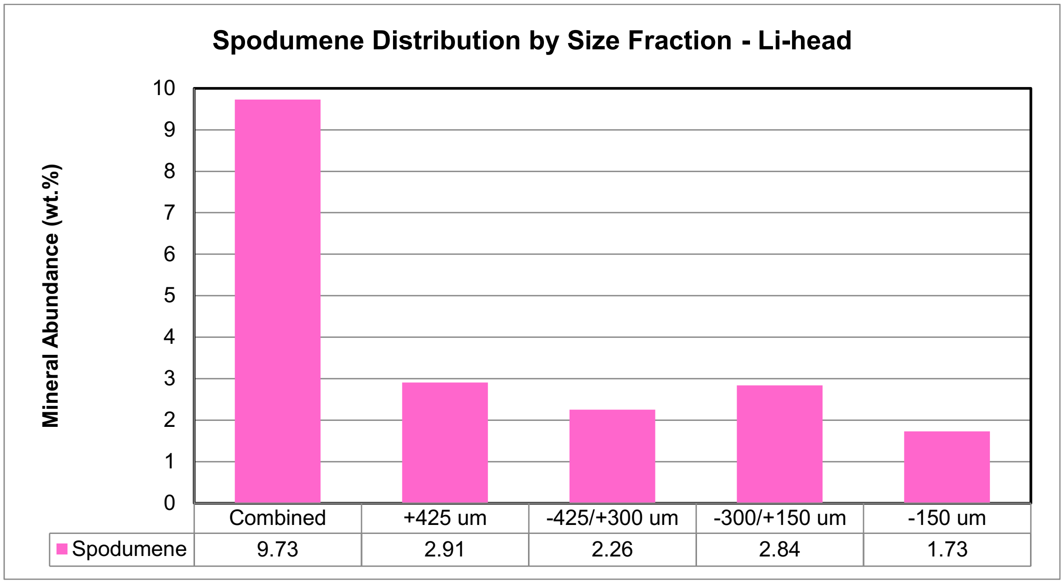


Figure 9: Spodumene Distribution by Sample and Calculated Head for the Li Head

Grain Size Distribution

The P_{80} (Table 5) for particle is 417 µm, for spodumene is 491 µm, quartz 476 µm, K-feldspars 401 µm, plagioclase 370 µm, and muscovite 336 µm.

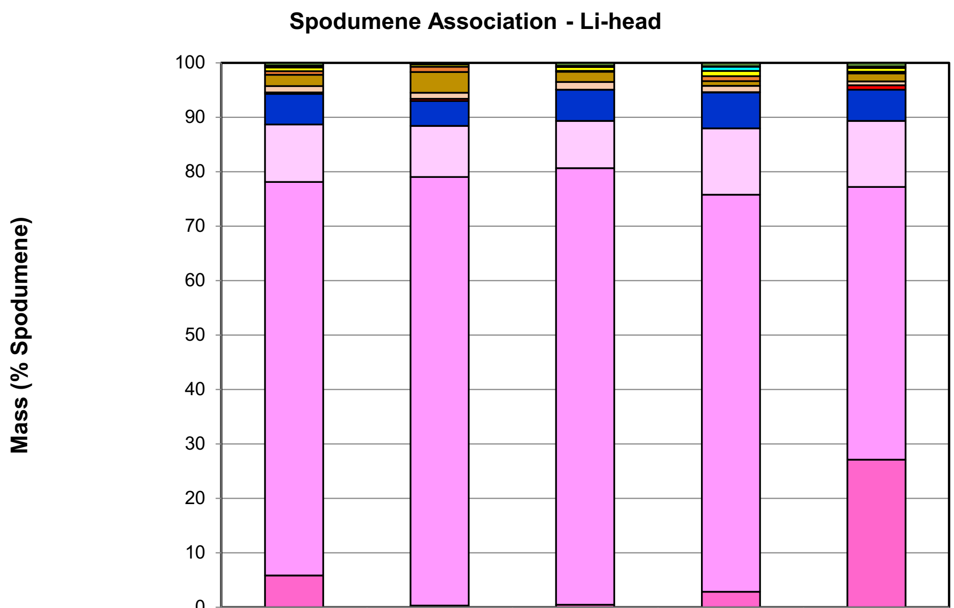
Table 5: P_{80} Size of Selected Minerals Calculated for the Head Sample

Sample	Li-head	
Grain Size (µm)	Median	P80
Spodumene	294	491
Petalite/ Cookeite	22	140
Beryl/ Tourmaline	46	259
Quartz	280	476
Plagioclase	215	370
K-Feldspars	228	401
Muscovite	179	336
Other Silicates	53	161
Garnet	260	333
Carbonates	120	231
Phosphates	308	308
Fe-(Ti)-Oxides	44	79
Nb-Ta-Oxides	124	144
Sulphides	40	63
Other	62	126
Particle	240	417

Liberation and Association of Spodumene, K-Feldspars, Plagioclase, and Muscovite

The liberation of spodumene is presented in Figure 10.

Liberated (pure, free, and liberated) spodumene accounts for 88.8% in the sample. The non-liberated grains occur as middling particles with petalite/cookeite (5.6%), middlings (binary) with quartz (1.1%), and plagioclase (2.0%). Liberation of spodumene is similar across all size fractions (88-89%).



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Complex	0.52	0.16	0.60	0.68	0.74
Spd:Other	0.14	0.16	0.16	0.08	0.18
Spd:Garnet	0.22	0.00	0.00	0.76	0.01
Spd:Quartz:Feldspars	0.68	0.39	0.75	0.92	0.70
Spd:K-Feldspars	0.66	0.98	0.14	0.92	0.34
Spd:Plagioclase	2.08	3.79	1.86	0.89	1.44
Spd:Quartz	1.15	1.15	1.42	1.18	0.74
Spd:Beryl/Tourmaline	0.28	0.38	0.02	0.05	0.82
Spd:Petalite/Cookeite	5.63	4.59	5.74	6.56	5.72
Lib Spodumene	10.5	9.38	8.65	12.2	12.1
Free Spodumene	72.3	78.7	80.2	72.9	50.1
Pure Spodumene	5.83	0.30	0.46	2.83	27.1

Figure 10: Liberation (Normalized Mass%) of Spodumene by Size Fraction and Calculated Head

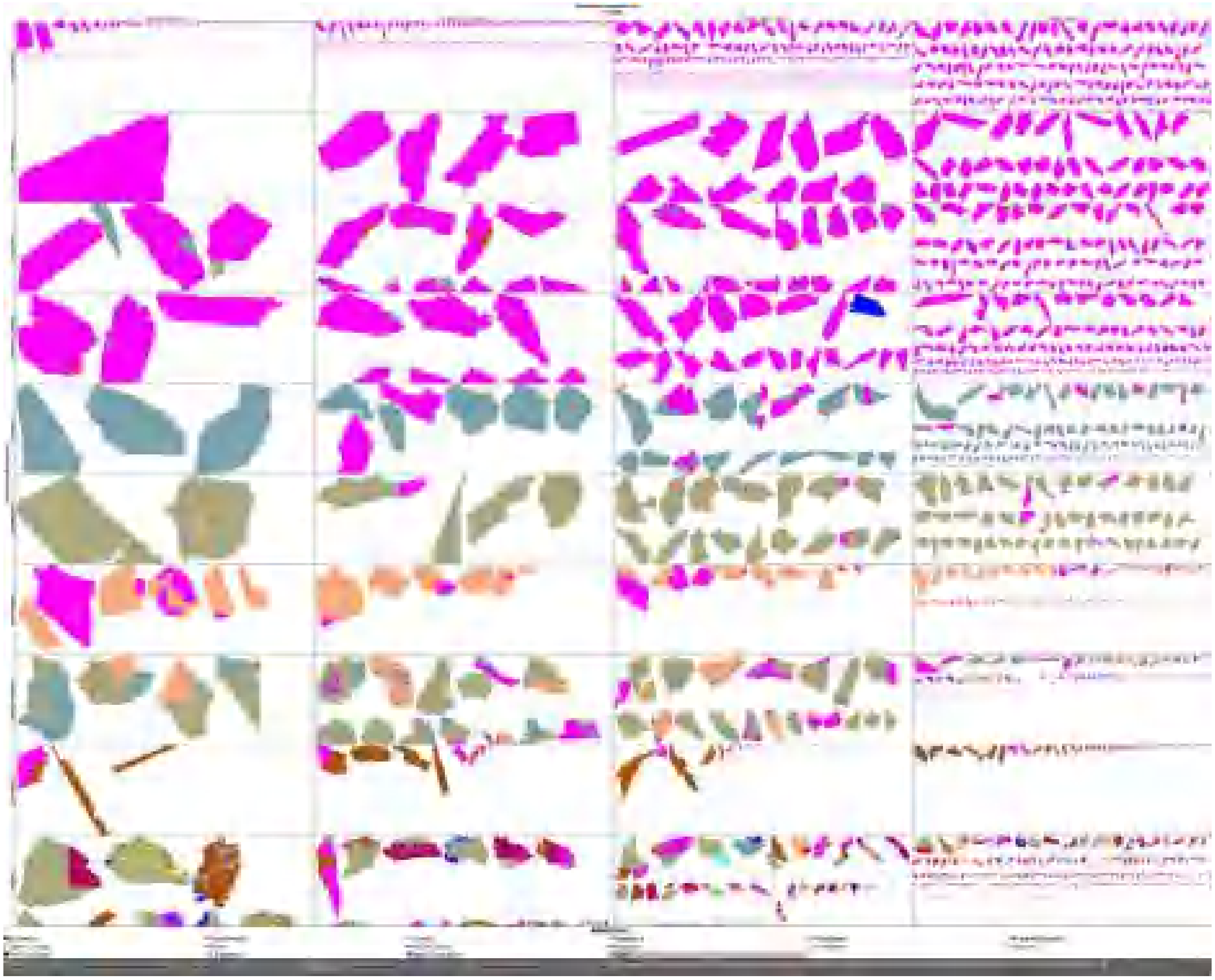
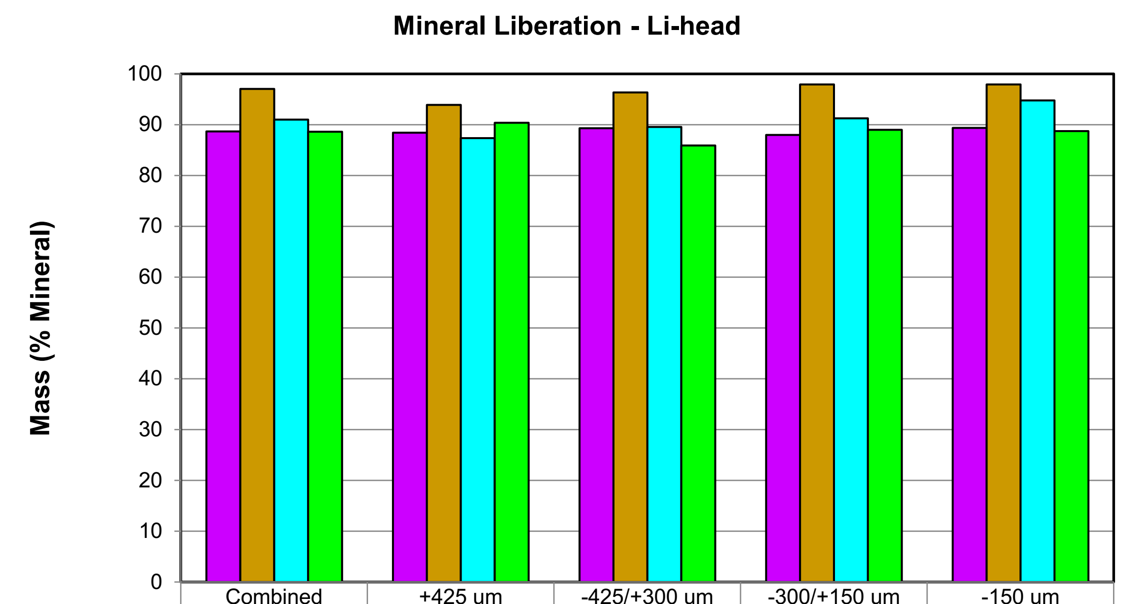


Figure 11: Image Grid Based on Liberation and Association of Spodumene in the Li Head

Liberation of K-feldspars accounts for 91% in the sample, and it increases from 87% in the coarse (+425 µm) to 90% in -425/+300 µm to 91% in -300/+150 µm and 95% in the fine (-150 µm) size fraction.

Liberation of plagioclase accounts for 97% in the sample, and it increases from 94% in the coarse (+425 µm) to 96% in -425/+300 µm to 98% in both the -300/+150 µm and -150 µm size fractions.

Liberation of muscovite accounts for 89% in the sample, and it decreases from 90% in the coarse (+425 µm) to 86% in -425/+300 µm, and it is similar in the finest two size fractions at 89%.



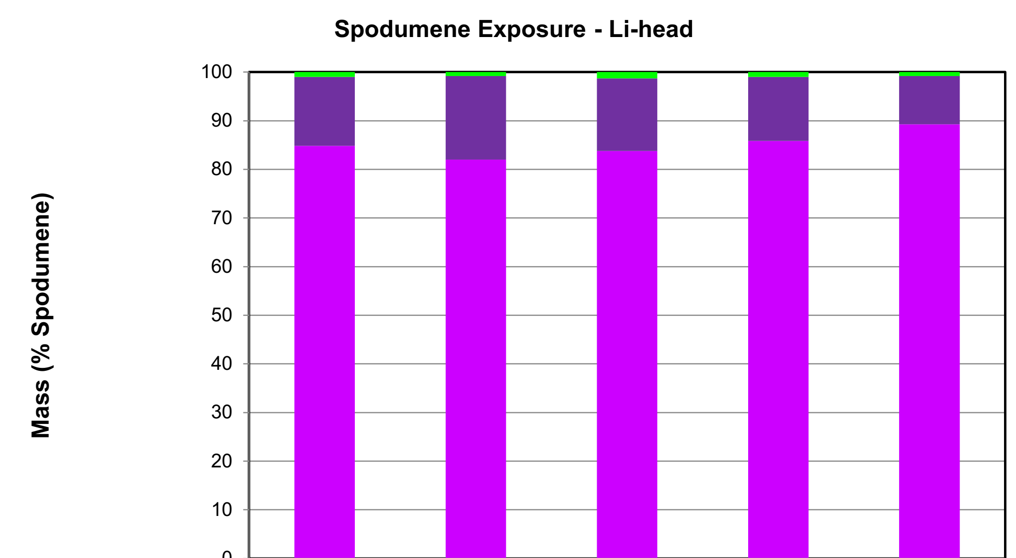
	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Spodumene	88.6	88.4	89.3	88.0	89.3
Plagioclase	97.0	93.9	96.4	97.9	97.9
K-Feldspars	91.0	87.3	89.5	91.2	94.7
Muscovite	88.6	90.3	85.9	88.9	88.7

Figure 12: Liberation (Normalized Mass%) of Pure, Free, and Liberated Spodumene, Quartz, K-Feldspars, Albite, and Muscovite by Size Fraction and Calculated Head

Exposure of Spodumene

The exposure of spodumene by surface area (i.e., exposure) characteristics for the Li Head by size fraction and calculated head are illustrated in Figure 13. Well exposed spodumene (>80% exposure) is 85%, between <80%->30% is 14%, and below 30% exposure is 1%

An image grid of illustrating the spodumene exposure by exposure class is given in Figure 14.



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
<30% Exposed	1.0	0.8	1.3	1.0	0.8
>30%-<80% Exposed	14.2	17.2	15.0	13.1	9.9
>80% Exposed	84.8	82.0	83.8	85.9	89.3

Figure 13: Exposure of Spodumene (Surface Area) (Mass%) by Size Fraction and Calculated for the Head for the Li Head

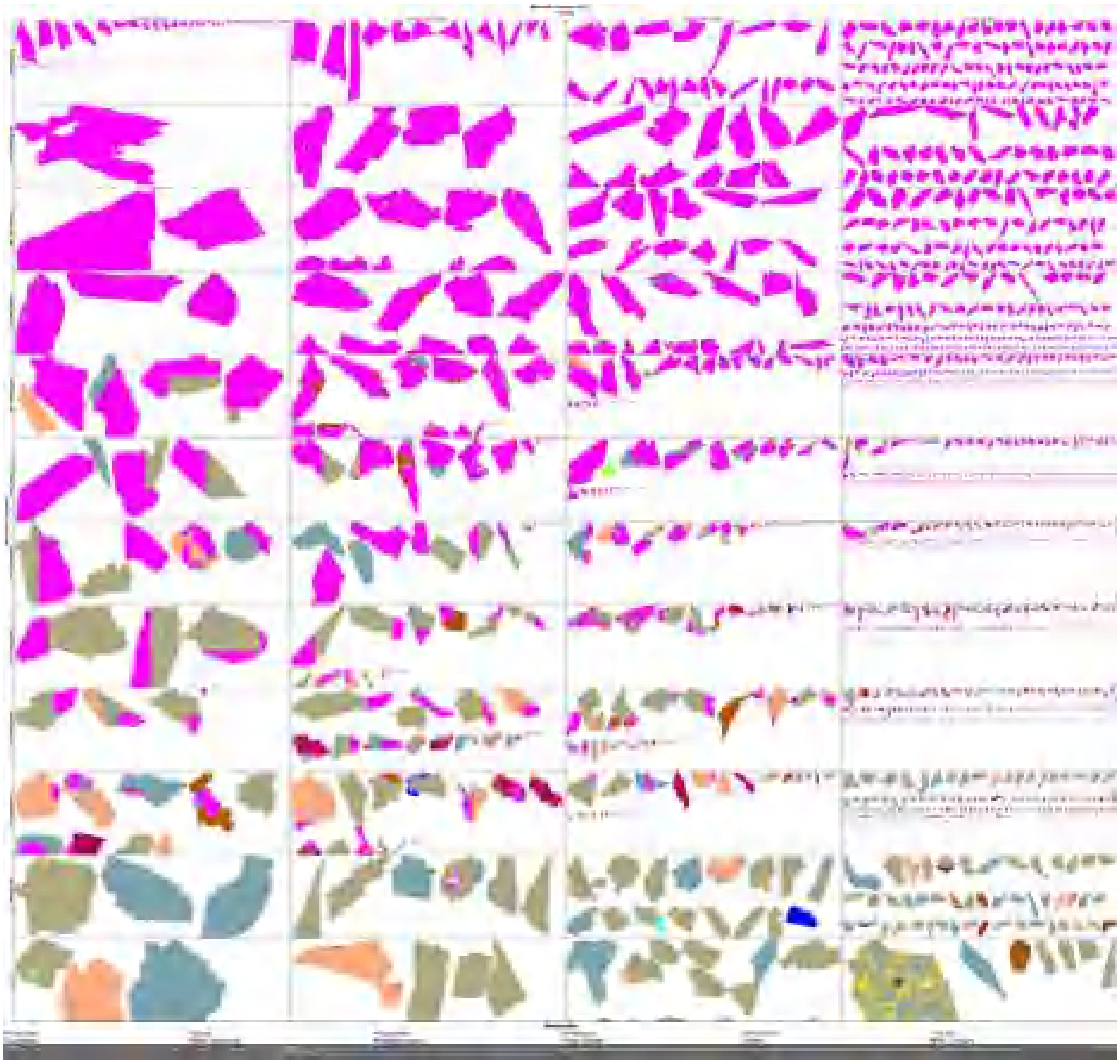


Figure 14: Image Grid Based on Exposure of Spodumene

Spodumene Grade and Recovery

Figure 15 illustrates that spodumene grades between 98% and 95% and recoveries of 89% to 99%, respectively, are projected for the sample (excludes the last 2 points of the curve).

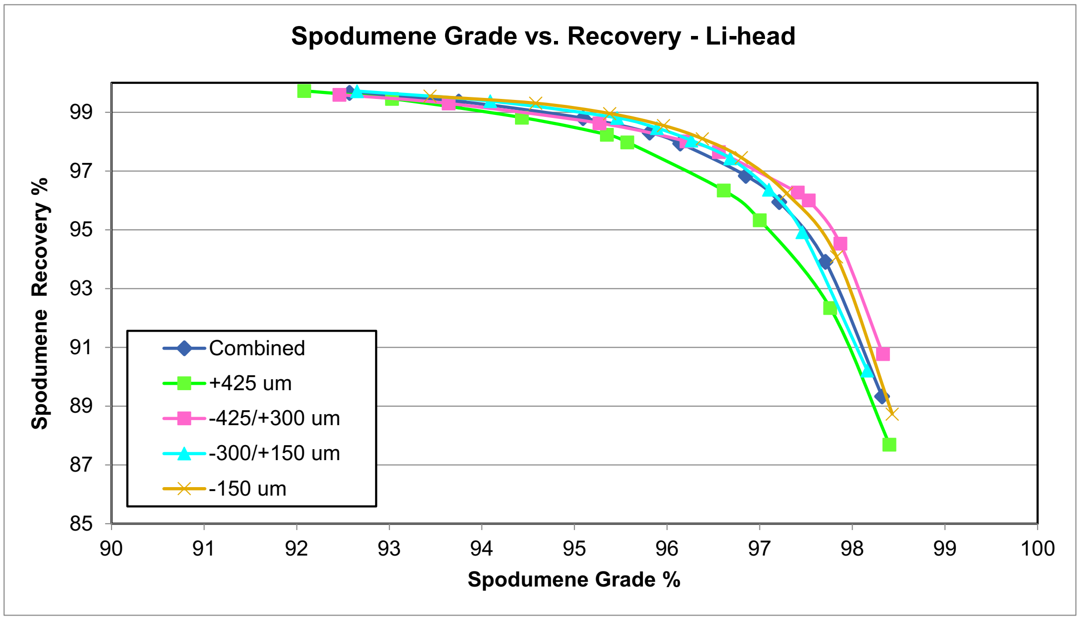


Figure 15: Grade-Recovery Curves for Spodumene by Size Fraction and Calculated Head for the Li Head

Mineral Chemistry

Electron Probe Micro-Analyses and LA-ICP-MS were conducted on spodumene, muscovite, K-feldspars, and garnet. Back scattered electron (BSE) microscope images of the minerals analyzed are given in Figure 16. The point analyses correspond to the analyses provided in the Appendix. For example, point analysis 1 is a muscovite, 48 and 50 is spodumene, and 72 is garnet. The letter (R) in the analyses indicates replicate polished mounts.

Spodumene averages (avg.) 3.47% lithium (Table 6). Note the presence of gallium (avg. 123 ppm).

Muscovite averages 1022 ppm lithium, Rb_2O 1.34%, Cs_2O 0.11%, Nb 146 ppm, Ta 66 ppm, Zn 403 ppm, Sn 223 ppm, Ga 330 ppm, Be 17 ppm and B 11 ppm (Table 7 and Table 8).

K-feldspars average 31 ppm lithium, Rb_2O 1.47%, Cs_2O 0.09%, Ga 31 ppm, B 245 ppm, and Pb 42 ppm (Table 9).

Garnet is spessartine-almandine solid solution. It averages150 ppm lithium, Ga 60 ppm, Ge 54 ppm, Sn 117 ppm, and Y 745 ppm (Table 10).

Table 6: Spodumene Chemistry from the EPMA and LA-ICP-MS for the Li Head

	EPMA												LA-ICP-MS (ppm)												
No.	**oxide weight %**	**SiO2**	**Al2O3**	**TiO2**	**Fe2O3**	**MnO**	**MgO**	**CaO**	**Na2O**	**K2O**	**Li2O***	**Total**	**Li**	***Li***	**Mg**	**Al**	**Si**	**Ca**	**Sc**	**Ti**	**V**	**Mn**	**Fe**	**Ga**	**Sn**
	LOD	0.049016	0.038418	0.025558	0.023798	0.027318	0.012065	0.015395	0.015641	0.012443															
46	Spodumene 01	64.51	26.45	0.01	1.87	0.24	0.00	0.01	0.14	0.01	7.95	101.18	35102.0	***3.51***	11.3	138710.2	304492.3	322.6	13.8	73.4	53.7	1624.3	9701.0	145.6	77.6
47	Spodumene 02	64.49	27.06	0.00	0.92	0.18	0.01	0.02	0.10	0.00	7.95	100.73	35666.9	***3.57***	13.8	141982.3	304217.0	399.6	5.8	83.3	12.7	1550.0	4531.3	141.9	24.7
48	Spodumene 03	64.86	26.86	0.00	1.15	0.09	0.00	0.00	0.11	0.00	7.95	101.03	34433.9	***3.44***	6.1	142679.1	302293.1	309.6	6.2	43.5	30.5	551.9	5596.3	130.8	8.8
49	Spodumene 04	64.65	27.07	0.02	1.13	0.28	0.01	0.02	0.14	0.01	7.95	101.28	34630.8	***3.46***	6.6	140607.3	308234.8	279.8	5.7	71.6	20.7	2035.7	5604.8	144.1	24.0
50	Spodumene 05	64.28	26.51	0.01	1.57	0.22	0.00	0.01	0.13	0.01	7.95	100.68	34799.3	***3.48***	11.8	139595.3	302228.5	324.0	8.1	54.3	21.1	1287.3	7115.0	129.8	13.5
51	Spodumene 06	64.58	26.96	0.02	0.96	0.25	0.00	0.00	0.12	0.01	7.95	100.86	34270.1	***3.43***	700.0	141852.1	303832.3	643.2	6.4	56.1	3.0	1668.9	4946.2	117.2	14.0
52	Spodumene 07	64.27	27.10	0.00	0.96	0.03	0.00	0.00	0.08	0.00	7.95	100.40	36171.2	***3.62***	2.0	139492.8	309367.8	325.8	6.7	32.6	19.1	288.8	4835.6	94.9	2.9
53	Spodumene 08	64.85	27.03	0.00	1.14	0.06	0.00	0.00	0.10	0.00	7.95	101.12	35030.0	***3.50***	9.5	139769.5	310600.2	332.0	6.5	36.1	38.0	377.2	5926.6	125.3	6.3
54	Spodumene 09	64.29	26.67	0.00	1.35	0.31	0.01	0.00	0.13	0.01	7.95	100.71	34451.1	***3.45***	6.0	138065.6	307549.4	397.0	11.6	56.8	22.2	2316.5	7868.9	119.3	24.5
55	Spodumene 10	64.46	27.04	0.00	1.00	0.05	0.00	0.00	0.06	0.00	7.95	100.58	35540.6	***3.55***	<LOD	140171.6	307994.5	328.5	7.9	34.4	17.8	225.6	5995.5	109.9	6.1
56	Spodumene 11	64.16	26.62	0.01	1.36	0.24	0.00	0.00	0.12	0.00	7.95	100.46	34775.4	***3.48***	22.2	139449.7	303231.8	308.8	7.7	69.4	38.2	1701.3	7127.5	136.5	55.7
57	Spodumene 12	64.83	26.74	0.00	1.22	0.06	0.00	0.01	0.08	0.01	7.95	100.89	34372.4	***3.44***	5.0	140266.5	305940.3	278.0	5.9	48.5	23.3	384.7	5786.7	135.3	7.0
58	Spodumene 13	64.34	26.87	0.02	1.07	0.14	0.00	0.00	0.12	0.00	7.95	100.50	34090.3	***3.41***	9.8	141144.5	303173.5	302.6	6.1	61.5	15.0	964.5	5530.8	115.0	5.7
59	Spodumene 14	64.62	26.80	0.00	1.03	0.16	0.00	0.01	0.12	0.01	7.95	100.72	34915.4	***3.49***	13.0	141939.8	302066.5	313.9	6.4	50.9	39.9	1112.9	5660.0	131.1	10.2
60	Spodumene 15	64.47	26.96	0.00	0.99	0.17	0.00	0.01	0.12	0.01	7.95	100.67	34583.4	***3.46***	19.4	138768.5	310308.6	310.5	6.8	50.6	5.2	1208.9	5054.2	123.5	22.4
61	Spodumene 16	64.62	27.05	0.00	1.17	0.12	0.00	0.00	0.12	0.00	7.95	101.03	34376.8	***3.44***	30.9	142910.8	302769.1	350.8	7.1	46.4	2.4	857.1	5864.1	127.4	5.8
62	Spodumene 17	64.36	26.78	0.02	1.17	0.10	0.00	0.00	0.10	0.00	7.95	100.49	32115.1	***3.21***	2.8	140642.2	303423.8	322.5	7.9	41.2	21.1	205.2	5126.7	125.4	11.0
63	Spodumene 18	64.69	26.96	0.00	1.17	0.07	0.00	0.01	0.09	0.01	7.95	100.95	34245.4	***3.42***	3.3	141864.4	304339.9	322.4	6.4	34.9	27.7	371.1	4781.5	108.6	5.4
64	Spodumene 19	64.59	27.03	0.00	1.07	0.09	0.00	0.01	0.10	0.00	7.95	100.85	34048.9	***3.40***	3.1	141042.8	306500.3	317.0	6.0	42.0	27.9	513.3	5547.8	101.0	14.9
65	R Spodumene 01	64.80	26.78	0.00	1.11	0.09	0.00	0.01	0.11	0.00	7.95	100.86	35254.7	***3.53***	21.3	142413.1	301663.4	349.6	7.0	43.2	28.4	598.4	5644.0	117.5	6.8
66	R Spodumene 02	64.74	27.21	0.01	1.08	0.11	0.00	0.01	0.11	0.01	7.95	101.22	34997.8	***3.50***	9.2	141405.3	308438.2	352.0	11.1	58.2	25.1	836.5	5899.4	127.5	7.1
67	R Spodumene 03	64.70	26.70	0.00	1.34	0.16	0.00	0.00	0.11	0.00	7.95	100.98	34328.7	***3.43***	14.2	140857.5	303589.0	362.1	6.6	47.3	<LOD	1155.4	6313.7	67.2	51.5
68	R Spodumene 04	64.63	27.00	0.00	1.13	0.07	0.00	0.01	0.08	0.00	7.95	100.89	34640.1	***3.46***	84.6	141931.6	304385.3	451.3	7.1	42.4	25.3	559.2	6075.6	128.4	21.9
69	R Spodumene 05	64.92	27.04	0.00	0.92	0.08	0.00	0.00	0.08	0.00	7.95	101.00	34538.9	***3.45***	<LOD	139926.0	310738.3	337.5	6.4	36.6	26.4	386.1	4973.5	117.6	6.3
70	R Spodumene 06	64.48	27.47	0.00	0.61	0.32	0.00	0.00	0.16	0.00	7.95	101.01	35267.6	***3.53***	12.6	141823.7	309406.5	329.7	6.1	63.8	125.8	2130.5	2981.1	152.9	33.5
	Min	64.16	26.45	0.00	0.61	0.03	0.00	0.00	0.06	0.00	7.95	100.40	32115.1	***3.21***	1.96	138065.6	301663.4	278.0	5.7	32.6	2.4	205.2	2981.1	67.2	2.9
	Max	64.92	27.47	0.02	1.87	0.32	0.01	0.02	0.16	0.01	7.95	101.28	36171.2	***3.62***	700.01	142910.8	310738.3	643.2	13.8	83.3	125.8	2316.5	9701.0	152.9	77.6
	Avg.	64.57	26.91	0.00	1.14	0.15	0.00	0.01	0.11	0.00	7.95	100.84	34665.9	***3.47***	44.28	140772.5	305631.4	346.8	7.3	51.2	27.9	996.4	5779.5	122.9	18.7

Li2O assumed; Li in red font indicates %*

Table 7: Muscovite Chemistry from the EPMA and LA-ICP-MS for the Li Head

	EPMA																		
No.	**oxide or element weight %**	**SiO2**	**Al2O3**	**TiO2**	**FeO**	**MnO**	**MgO**	**CaO**	**BaO**	**Na2O**	**K2O**	**Rb2O**	**Cs2O**	**F**	**Cl**	**H2O***	**O = F**	**O = Cl**	**Total**
	LOD	0.071765	0.059115	0.055186	0.048758	0.041532	0.022681	0.033264	0.075757	0.030732	0.034268	0.074563	0.057125	0.076207	0.014251				
1	Muscovite 01	44.30	35.22	0.14	2.50	0.26	0.04	0.00	0.02	0.48	10.09	1.30	0.05	0.23	0.00	4.27	-0.10	0.00	98.82
2	Muscovite 02	44.33	34.96	0.09	2.28	0.18	0.25	0.03	0.03	0.32	10.04	1.53	0.18	0.18	0.00	4.29	-0.07	0.00	98.60
3	Muscovite 03	44.29	34.98	0.05	2.61	0.12	0.00	0.00	0.00	0.48	10.26	1.10	0.04	0.35	0.00	4.20	-0.15	0.00	98.33
4	Muscovite 04	44.59	34.61	0.19	2.30	0.13	0.30	0.03	0.00	0.37	9.83	1.73	0.28	0.29	0.00	4.23	-0.12	0.00	98.75
5	Muscovite 05	44.18	34.91	0.13	2.86	0.19	0.09	0.01	0.02	0.33	10.50	1.22	0.04	0.29	0.00	4.23	-0.12	0.00	98.87
6	Muscovite 06	44.37	35.50	0.14	2.19	0.14	0.10	0.00	0.01	0.42	10.16	1.27	0.06	0.18	0.00	4.31	-0.07	0.00	98.77
7	Muscovite 07	44.28	34.99	0.00	2.24	0.18	0.11	0.02	0.00	0.22	10.49	1.33	0.06	0.12	0.00	4.30	-0.05	0.00	98.29
8	Muscovite 08	44.39	35.90	0.06	2.14	0.11	0.02	0.04	0.00	0.43	10.15	1.35	0.08	0.12	0.01	4.35	-0.05	0.00	99.09
9	Muscovite 09	44.58	34.70	0.13	2.44	0.18	0.11	0.03	0.00	0.39	10.15	1.17	0.04	0.23	0.01	4.26	-0.10	0.00	98.33
10	Muscovite 10	44.26	35.26	0.06	2.61	0.14	0.01	0.02	0.00	0.54	10.24	1.05	0.06	0.29	0.01	4.24	-0.12	0.00	98.67
11	Muscovite 11	45.04	36.34	0.07	1.47	0.09	0.03	0.03	0.00	0.27	10.96	0.19	0.07	0.12	0.00	4.39	-0.05	0.00	99.01
12	R Muscovite 01	45.05	35.80	0.02	1.94	0.13	0.08	0.02	0.03	0.23	10.56	0.90	0.07	0.12	0.01	4.37	-0.05	0.00	99.27
13	R Muscovite 02	44.09	34.33	0.14	2.91	0.16	0.12	0.02	0.01	0.24	10.08	1.75	0.28	0.23	0.01	4.22	-0.10	0.00	98.48
14	R Muscovite 03	44.58	34.71	0.19	2.19	0.10	0.24	0.00	0.00	0.35	10.10	1.75	0.08	0.23	0.00	4.26	-0.10	0.00	98.70
15	R Muscovite 04	43.62	30.52	0.64	5.47	0.09	0.61	0.01	0.00	0.16	9.74	2.26	0.44	0.39	0.01	4.03	-0.17	0.00	97.82
16	R Muscovite 05	44.18	36.51	0.02	1.80	0.10	0.02	0.00	0.02	0.31	10.28	1.39	0.13	0.23	0.00	4.30	-0.10	0.00	99.20
17	R Muscovite 06	44.33	35.27	0.20	2.29	0.13	0.27	0.06	0.05	0.37	9.90	1.74	0.13	0.23	0.01	4.28	-0.10	0.00	99.16
18	R Muscovite 07	44.05	34.93	0.19	2.46	0.18	0.17	0.02	0.00	0.29	10.40	1.22	0.04	0.35	0.00	4.19	-0.15	0.00	98.35
19	R Muscovite 08	44.04	35.17	0.05	2.38	0.11	0.05	0.00	0.00	0.29	10.37	1.45	0.03	0.18	0.00	4.27	-0.07	0.00	98.31
20	R Muscovite 09	44.46	34.56	0.10	2.92	0.16	0.07	0.00	0.00	0.41	10.21	1.29	0.07	0.35	0.01	4.20	-0.15	0.00	98.66
21	R Muscovite 10	44.95	32.06	0.45	3.73	0.61	0.07	0.00	0.00	0.13	10.26	1.69	0.21	0.46	0.01	4.09	-0.19	0.00	98.55
22	R Muscovite 11	44.53	35.79	0.11	2.29	0.16	0.05	0.00	0.00	0.49	10.19	1.13	0.02	0.23	0.00	4.30	-0.10	0.00	99.19
23	R Muscovite 12	44.20	35.80	0.23	1.68	0.10	0.17	0.03	0.02	0.47	9.89	1.83	0.16	0.35	0.00	4.23	-0.15	0.00	99.01
24	R Muscovite 13	44.76	35.17	0.09	2.49	0.14	0.01	0.01	0.04	0.45	10.34	1.00	0.07	0.29	0.00	4.26	-0.12	0.00	99.00
25	R Muscovite 14	44.58	36.08	0.11	2.26	0.13	0.06	0.03	0.02	0.53	10.29	0.77	0.01	0.12	0.00	4.38	-0.05	0.00	99.32
	Min	43.62	30.52	0.00	1.47	0.09	0.00	0.00	0.00	0.13	9.74	0.19	0.01	0.12	0.00	4.03	-0.19	0.00	97.82
	Max	45.05	36.51	0.64	5.47	0.61	0.61	0.06	0.05	0.54	10.96	2.26	0.44	0.46	0.01	4.39	-0.05	0.00	99.32
	Avg.	44.40	34.96	0.14	2.50	0.16	0.12	0.02	0.01	0.36	10.22	1.34	0.11	0.25	0.00	4.26	-0.10	0.00	98.74

Table 8: Muscovite Chemistry from the EPMA and LA-ICP-MS for the Li Head (continued)

	LA-ICP-MS (ppm)																							
No.	Element	Li	*Li%*	Be	B	Mg	Al	Si	K	Ca	Sc	Ti	V	Mn	Fe	Zn	Ga	Ge	Rb	Nb	Sn	Cs	Ba	Ta
	LOD																							
1	Muscovite 01	1157.5	***0.12***	21.5	8.8	304.5	182419.9	211775.0	83768.9	218.3	4.4	516.8	8.0	1798.1	12993.0	814.3	314.6	4.8	10671.9	222.0	169.5	378.6	1.5	59.0
2	Muscovite 02	779.9	***0.08***	13.3	11.3	1448.4	181132.6	211893.5	87222.6	225.0	5.8	293.0	42.4	1157.1	11368.3	322.9	369.6	4.5	13942.1	130.4	350.1	1206.6	62.8	24.5
3	Muscovite 03	1077.3	***0.11***	19.2	10.0	86.9	178736.6	214837.2	85671.6	255.6	4.1	338.7	5.9	988.9	14774.1	627.4	300.0	4.0	9786.3	215.8	90.5	420.2	2.5	40.2
4	Muscovite 04	1043.8	***0.10***	17.3	9.8	1883.3	178024.5	214782.4	86867.4	262.5	6.7	964.3	96.4	1040.2	12529.7	394.4	408.6	2.7	15309.8	151.1	435.2	2339.0	60.8	128.1
5	Muscovite 05	1013.7	***0.10***	20.2	9.5	492.3	177161.9	215868.6	88231.4	196.5	4.8	723.8	127.3	1086.2	15095.1	422.6	300.5	3.0	10522.8	220.5	204.0	579.2	2.2	63.4
6	Muscovite 06	707.0	***0.07***	24.5	9.1	510.8	181238.5	215395.6	88864.6	227.9	6.4	643.2	3.1	1275.8	11580.3	539.6	347.4	4.3	11270.6	175.9	179.9	411.8	2.9	76.3
7	Muscovite 07	665.1	***0.07***	12.2	10.8	737.0	177702.1	216201.9	90638.8	328.2	5.2	166.4	42.6	1192.2	12915.6	192.6	301.1	5.1	11332.2	62.4	183.1	481.2	132.2	15.6
8	Muscovite 08	683.4	***0.07***	20.5	9.5	68.7	185267.1	213027.7	87532.9	217.1	4.4	286.6	160.0	662.0	11442.0	265.4	330.1	3.6	12469.6	13.1	153.0	626.1	1.7	36.2
9	Muscovite 09	783.4	***0.08***	19.7	9.4	860.0	179673.5	213207.3	86993.7	200.6	8.4	523.0	14.2	1052.1	13061.9	483.2	327.7	3.8	10973.6	240.8	166.3	433.0	2.7	59.4
10	Muscovite 10	988.6	***0.10***	19.9	9.6	66.9	180794.8	213875.0	89000.9	204.9	4.0	294.9	2.3	1063.6	14363.8	622.6	283.9	2.6	8887.1	208.3	79.3	281.6	2.0	47.1
11	Muscovite 11	246.2	***0.02***	1.3	12.9	288.0	186867.7	217019.9	93092.6	203.9	19.0	491.3	18.5	536.4	6549.9	61.2	106.2	71.7	2042.8	<LOD	86.0	458.0	5.6	1.0
12	R Muscovite 01	607.2	***0.06***	10.0	12.0	1056.2	181517.7	220325.2	92688.9	228.2	9.6	283.3	22.4	757.1	11554.7	120.6	330.2	6.0	11224.9	44.0	141.4	520.1	44.7	13.7
13	R Muscovite 02	1165.7	***0.12***	15.3	11.1	901.5	178577.8	209883.1	86971.3	261.3	13.2	799.6	40.4	1208.4	15942.2	283.6	378.7	2.9	15221.2	179.7	479.5	1948.7	4.4	113.1
14	R Muscovite 03	1068.6	***0.11***	18.1	10.6	1754.3	177894.2	215512.5	87196.0	267.4	7.2	1328.7	111.1	1067.5	13318.7	432.0	388.9	2.7	17770.2	116.6	170.0	2408.6	11.3	149.9
15	R Muscovite 04	2953.5	***0.30***	15.3	8.4	3643.9	155959.5	211540.9	81975.1	261.0	25.6	3363.8	106.0	860.2	29717.7	195.9	321.7	4.0	20547.7	234.0	720.8	3384.6	80.0	125.2
16	R Muscovite 05	612.6	***0.06***	14.9	11.7	85.3	184420.7	216958.2	90193.5	209.0	3.9	101.7	97.4	642.0	10288.6	202.1	307.4	2.9	12828.7	1.1	110.5	784.4	4.6	2.3
17	R Muscovite 06	1142.6	***0.11***	20.2	11.9	2025.2	180401.4	214801.6	85221.9	279.5	5.5	1290.1	73.5	1013.2	12791.8	517.0	387.0	2.5	17784.1	84.4	132.3	1322.6	19.8	132.7
18	R Muscovite 07	936.9	***0.09***	19.2	10.6	883.9	179789.7	212012.4	91703.6	267.5	7.7	1440.7	112.4	1048.3	13618.8	395.6	367.3	3.4	11907.9	226.2	241.5	601.4	2.1	98.4
19	R Muscovite 08	863.5	***0.09***	18.0	9.8	304.5	183316.9	209226.9	88795.2	307.0	4.9	346.2	112.5	901.3	12498.5	281.1	298.7	2.9	12053.9	110.4	209.8	466.0	2.8	90.1
20	R Muscovite 09	1291.9	***0.13***	18.7	11.6	393.1	177480.2	214484.1	87699.0	288.4	15.8	713.0	61.5	1164.1	15791.9	523.4	335.7	2.4	11727.1	227.1	200.3	506.9	2.4	65.4
21	R Muscovite 10	1784.7	***0.18***	8.2	11.1	423.8	164388.7	217220.6	88687.7	273.6	4.0	2602.6	53.8	3898.4	20419.4	306.9	436.6	4.4	16155.5	59.3	580.4	1480.3	1.4	24.5
22	R Muscovite 11	901.8	***0.09***	23.1	11.2	254.1	183752.5	214867.1	89561.6	301.0	5.7	513.5	0.8	1234.8	12728.7	622.3	332.1	3.0	11026.9	201.4	158.9	390.7	1.6	61.4
23	R Muscovite 12	896.1	***0.09***	18.0	12.6	1141.2	181359.9	216434.3	86890.1	284.5	5.1	1189.6	41.4	787.0	10282.0	335.5	387.9	4.0	17373.9	83.3	143.2	1129.5	2.2	114.7
24	R Muscovite 13	967.7	***0.10***	18.2	11.8	88.8	181051.3	215372.5	93581.3	334.5	3.9	367.9	3.8	1151.7	14041.3	705.4	302.3	2.1	9914.3	244.5	106.5	356.8	2.7	41.6
25	R Muscovite 14	1213.7	***0.12***	22.1	12.9	303.6	183004.2	218009.8	88570.4	252.9	4.3	433.7	98.3	945.0	12580.2	413.0	276.8	2.2	9449.2	57.6	93.7	285.6	5.9	66.5
	Min	246.2	0.025	1.3	8.4	66.9	155959.5	209226.9	81975.1	196.5	3.9	101.7	0.8	536.4	6549.9	61.2	106.2	2.1	2042.8	1.1	79.3	281.6	1.4	1.0
	Max	2953.5	0.295	24.5	12.9	3643.9	186867.7	220325.2	93581.3	334.5	25.6	3363.8	160.0	3898.4	29717.7	814.3	436.6	71.7	20547.7	244.5	720.8	3384.6	132.2	149.9
	Avg.	1022.1	0.102	17.2	10.7	800.3	179277.3	214581.3	88304.8	254.3	7.6	800.7	58.2	1141.3	13689.9	403.2	329.6	6.2	12487.8	146.2	223.4	928.1	18.5	66.0

H2O calculated; Li in red font indicates %*

Table 9: K-feldspar Chemistry from the EPMA and LA-ICP-MS for the Li Head

		EPMA										LA-ICP-MS (ppm)												
No.	Oxide weight %	SiO2	Al2O3	Fe2O3	CaO	BaO	Na2O	K2O	Rb2O	Cs2O	Total	Li	B	Al	Si	P	K	Ca	Ga	Rb	Sr	Cs	Ba	Pb
	LOD	0.051738	0.038135	0.031014	0.023117	0.053009	0.020588	0.024615	0.04884	0.039816														
26	Alkali feldspar 01	64.07	18.29	0.00	0.02	0.04	0.46	15.31	1.57	0.06	99.82	33.6	266.8	96034.4	302039.6	81.2	131718.0	362.5	31.0	12716.7	16.3	299.3	25.6	54.4
27	Alkali feldspar 02	64.18	18.24	0.00	0.01	0.00	0.71	15.39	1.09	0.03	99.65	28.5	265.1	96493.0	300291.0	91.9	126492.1	319.9	31.9	9522.5	1.9	297.4	6.0	37.3
28	Alkali feldspar 03	64.53	18.16	0.01	0.02	0.01	0.65	15.15	1.27	0.10	99.91	23.5	258.5	96536.2	300506.2	84.4	130858.3	340.7	33.3	11740.0	6.5	978.6	9.7	45.6
29	Alkali feldspar 04	64.00	18.22	0.01	0.02	0.00	0.60	15.05	1.49	0.05	99.44	24.6	253.5	96956.7	297677.5	87.5	131860.9	377.4	31.2	13927.4	10.0	509.2	28.2	39.3
30	Alkali feldspar 05	64.06	18.18	0.01	0.02	0.00	0.56	15.37	1.28	0.04	99.52	24.5	266.2	97384.8	296053.9	105.6	128946.7	375.1	30.8	11832.8	2.4	322.6	4.0	32.4
31	Alkali feldspar 06	64.08	18.02	0.01	0.01	0.02	0.53	15.33	1.35	0.19	99.53	32.1	253.4	94769.4	301534.6	<LOD	127161.6	377.8	30.6	12579.4	4.9	1640.9	2.1	52.2
32	Alkali feldspar 07	63.85	18.25	0.00	0.02	0.01	0.50	15.21	1.57	0.15	99.55	30.6	248.9	94300.7	305950.1	99.2	131135.2	341.9	30.5	14799.6	8.1	1139.3	19.3	45.2
33	Alkali feldspar 08	64.45	18.36	0.00	0.01	0.03	0.57	15.25	1.35	0.06	100.07	29.2	224.3	96504.2	303531.0	69.0	127220.2	324.2	30.5	12966.8	5.5	629.9	11.8	51.3
34	Alkali feldspar 09	64.11	18.29	0.02	0.01	0.02	0.57	15.30	1.18	0.05	99.54	22.2	222.3	96467.7	300850.0	93.2	136989.4	793.5	30.2	11220.9	11.5	324.3	23.9	30.7
35	Alkali feldspar 10	63.80	18.43	0.02	0.02	0.01	0.62	15.01	1.54	0.04	99.48	45.0	224.5	97752.7	297736.2	96.2	122640.2	374.7	29.3	13429.0	7.2	458.6	22.2	44.9
36	Alkali feldspar 11	63.48	18.05	0.00	0.01	0.01	0.34	15.41	1.50	0.18	98.97	14.6	232.8	95654.8	296510.0	<LOD	133340.4	384.9	27.8	13071.6	3.5	803.0	16.1	22.3
37	Alkali feldspar 12	63.68	18.10	0.02	0.00	0.01	0.44	15.27	1.62	0.06	99.21	23.8	126.8	94383.3	302289.5	<LOD	124937.6	356.5	30.9	13650.9	9.3	297.8	12.7	47.1
38	R Alkali feldspar 01	64.10	18.33	0.00	0.02	0.00	0.53	15.50	1.29	0.03	99.81	45.5	258.5	96782.9	300537.7	104.2	125712.8	415.0	31.1	11535.3	5.6	308.1	21.4	32.6
38	R Alkali feldspar 02	64.12	18.06	0.02	0.01	0.02	0.48	15.21	1.68	0.14	99.73	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
49	R Alkali feldspar 03	63.74	17.92	0.02	0.02	0.00	0.41	15.24	1.96	0.16	99.47	21.3	273.0	93501.9	302475.5	90.5	130270.2	414.3	35.1	16697.7	13.2	1397.4	18.2	55.6
41	R Alkali feldspar 04	64.09	18.07	0.01	0.01	0.02	0.46	15.19	1.67	0.03	99.56	30.4	256.8	94305.4	303998.6	92.7	127142.4	368.9	31.0	14632.7	4.1	349.0	9.9	29.8
42	R Alkali feldspar 05	64.44	18.31	0.01	0.02	0.00	0.59	15.05	1.55	0.04	100.01	50.4	254.0	96456.9	302803.4	78.4	130893.4	415.4	30.7	14584.6	5.9	565.8	18.7	39.6
43	R Alkali feldspar 06	63.99	18.33	0.00	0.00	0.00	0.59	15.13	1.68	0.18	99.90	61.2	257.5	96617.9	300438.3	127.2	130045.1	429.3	28.0	15502.7	6.6	1568.6	19.0	47.5
44	R Alkali feldspar 07	64.10	18.22	0.00	0.02	0.00	0.43	15.38	1.65	0.12	99.93	20.3	269.2	96679.4	298963.4	92.7	129774.6	345.3	30.5	14722.5	7.1	1066.6	14.4	46.8
45	R Alkali feldspar 08	64.43	18.35	0.00	0.01	0.00	0.55	15.41	1.20	0.14	100.09	20.3	245.8	96408.3	303487.7	<LOD	124590.0	343.9	29.1	10930.0	7.2	746.3	19.7	49.7
	Min	63.48	17.92	0.00	0.00	0.00	0.34	15.01	1.09	0.03	98.97	14.6	126.8	93501.9	296053.9	69.0	122640.2	319.9	27.8	9522.5	1.9	297.4	2.1	22.3
	Max	64.53	18.43	0.02	0.02	0.04	0.71	15.50	1.96	0.19	100.09	61.2	273.0	97752.7	305950.1	127.2	136989.4	793.5	35.1	16697.7	16.3	1640.9	28.2	55.6
	Avg.	64.06	18.21	0.01	0.01	0.01	0.53	15.26	1.47	0.09	99.66	30.6	245.1	95999.5	300930.2	92.9	129038.4	392.7	30.7	13161.2	7.2	721.2	16.0	42.3

n/a: not analyzed;

Table 10: Garnet Chemistry from the EPMA and LA-ICP-MS for the Li Head

		EPMA										LA-ICP-MS (ppm)															
No.	Oxide weight %	SiO2	Al2O3	TiO2	Y2O3	FeO	MnO	MgO	CaO	Na2O	Total	Li	B	Mg	Al	Si	P	Sc	Ti	V	Mn	Fe	Zn	Ga	Ge	Y	Sn
	LOD	0.041964	0.035912	0.017983	0.034748	0.032934	0.027263	0.015567	0.013761	0.017559																	
71	Garnet 01	35.70	20.88	0.05	0.01	8.22	34.19	0.03	0.67	0.01	99.76	71.1	17.6	289.0	111017.8	204511.7	148.0	18.6	225.9	20.7	263740.3	59914.1	335.9	56.5	56.7	132.6	22.1
72	Garnet 02	35.75	20.63	0.06	0.09	11.91	31.09	0.01	0.32	0.00	99.85	94.8	15.9	65.0	108176.6	210399.5	155.5	7.1	193.6	6.6	243100.4	74697.7	311.8	64.6	22.4	239.8	240.0
73	Garnet 03	35.78	20.83	0.03	0.17	10.39	32.02	0.00	0.27	0.01	99.49	169.2	18.5	114.1	108653.1	208888.3	123.1	18.8	180.0	15.9	251735.9	60250.6	381.8	47.7	64.3	1106.4	32.9
74	Garnet 04	35.47	20.62	0.09	0.29	10.73	31.77	0.01	0.31	0.01	99.28	312.9	19.1	39.6	109369.9	205113.8	225.8	13.0	447.6	16.0	245607.6	72406.1	308.1	69.2	61.8	2165.5	192.0
75	Garnet 05	35.82	20.79	0.04	0.04	9.68	32.54	0.02	0.66	0.01	99.61	68.3	20.6	234.0	106962.3	209265.0	137.7	22.2	219.2	10.8	259602.6	52738.8	348.0	50.2	76.2	113.1	30.3
76	Garnet 06	35.61	20.54	0.03	0.02	11.27	31.55	0.01	0.30	0.00	99.32	72.1	10.5	178.3	118324.8	183667.3	168.0	3.4	179.4	2.7	226074.2	63840.9	67.0	56.7	15.2	90.3	117.2
77	Garnet 07	35.87	20.68	0.06	0.10	8.96	33.57	0.01	0.30	0.00	99.55	192.8	19.1	137.0	106458.1	205638.0	175.8	55.6	309.1	3.0	267610.4	56711.4	356.4	68.0	65.0	860.4	123.2
78	Garnet 08	36.01	20.47	0.05	0.03	10.05	33.08	0.01	0.24	0.01	99.95	55.8	21.0	15.4	109055.2	206424.9	239.3	4.1	113.5	2.5	254680.8	62305.8	348.6	65.0	18.6	22.1	87.8
79	Garnet 09	35.72	20.40	0.08	0.25	11.12	31.40	0.00	0.30	0.01	99.29	305.3	19.0	35.5	109198.5	204534.0	256.9	13.8	430.4	15.2	240589.6	69686.9	313.5	66.1	59.8	2131.7	171.7
80	Garnet 10	35.47	20.61	0.03	0.03	8.93	34.08	0.00	0.26	0.01	99.44	107.4	17.2	13.6	104765.5	197623.2	295.2	4.2	242.1	<LOD	275486.0	33149.8	354.7	53.6	55.4	176.5	112.0
81	Garnet 11	35.72	20.73	0.05	0.02	8.09	34.58	0.01	0.26	0.00	99.45	122.9	19.2	31.4	112788.7	209108.2	305.8	4.5	304.5	3.0	260786.8	63155.0	355.3	63.7	67.5	244.3	82.8
82	Garnet 12	35.87	20.70	0.03	0.10	9.88	32.82	0.02	0.34	0.01	99.77	103.2	21.1	217.6	114262.2	206206.4	173.1	62.4	157.1	6.5	244235.7	64964.5	358.4	41.6	81.3	402.2	17.0
83	R Garnet 01	35.71	20.43	0.09	0.33	10.68	32.01	0.01	0.29	0.01	99.56	174.4	22.2	92.9	109043.5	209376.1	213.6	55.1	318.3	19.7	245963.7	69075.9	312.7	76.2	36.5	832.8	241.0
84	R Garnet 02	36.00	20.83	0.04	0.19	9.56	32.87	0.02	0.34	0.00	99.85	174.7	19.7	103.7	109015.4	214859.2	138.9	33.1	179.2	3.2	257666.4	58303.4	355.5	52.9	69.0	1156.1	36.7
85	R Garnet 03	35.73	20.65	0.06	0.09	10.62	32.21	0.00	0.29	0.00	99.66	77.1	24.8	49.3	109260.6	205405.9	195.3	11.1	157.5	5.0	249603.2	68247.1	309.7	64.0	24.1	207.0	170.3
86	R Garnet 04	36.03	20.57	0.07	0.19	10.45	32.25	0.01	0.29	0.00	99.87	258.7	21.5	28.0	105230.4	210798.4	254.3	10.1	392.7	6.8	258827.0	66070.5	298.2	66.7	52.9	1485.8	186.8
87	R Garnet 05	35.95	20.79	0.04	0.05	5.57	37.26	0.00	0.25	0.01	99.91	95.4	22.5	13.4	110121.5	193818.7	333.6	4.7	190.1	0.5	288447.6	43540.2	380.9	54.7	64.5	184.1	120.4
88	R Garnet 06	35.62	20.53	0.09	0.35	11.23	31.42	0.01	0.31	0.00	99.55	313.0	22.1	39.0	109214.7	207345.1	261.5	14.8	459.9	10.2	242145.7	72446.1	311.3	72.0	61.4	2343.1	206.8
89	R Garnet 07	36.06	21.03	0.05	0.00	8.19	34.35	0.03	0.65	0.00	100.36	88.7	21.8	282.8	113383.1	207329.5	160.9	27.7	227.2	14.9	261301.5	59789.5	351.4	57.1	76.3	253.4	25.6
	Min	35.47	20.40	0.03	0.00	5.57	31.09	0.00	0.24	0.00	99.28	55.8	10.5	13.4	104765.5	183667.3	123.1	3.4	113.5	0.5	226074.2	33149.8	67.0	41.6	15.2	22.1	17.0
	Max	36.06	21.03	0.09	0.35	11.91	37.26	0.03	0.67	0.01	100.36	313.0	24.8	289.0	118324.8	214859.2	333.6	62.4	459.9	20.7	288447.6	74697.7	381.8	76.2	81.3	2343.1	241.0
	Avg.	35.78	20.67	0.05	0.12	9.77	32.90	0.01	0.35	0.01	99.66	150.4	19.7	104.2	109700.1	205279.6	208.5	20.2	259.3	9.1	254589.8	61647.1	324.2	60.4	54.2	744.6	116.7

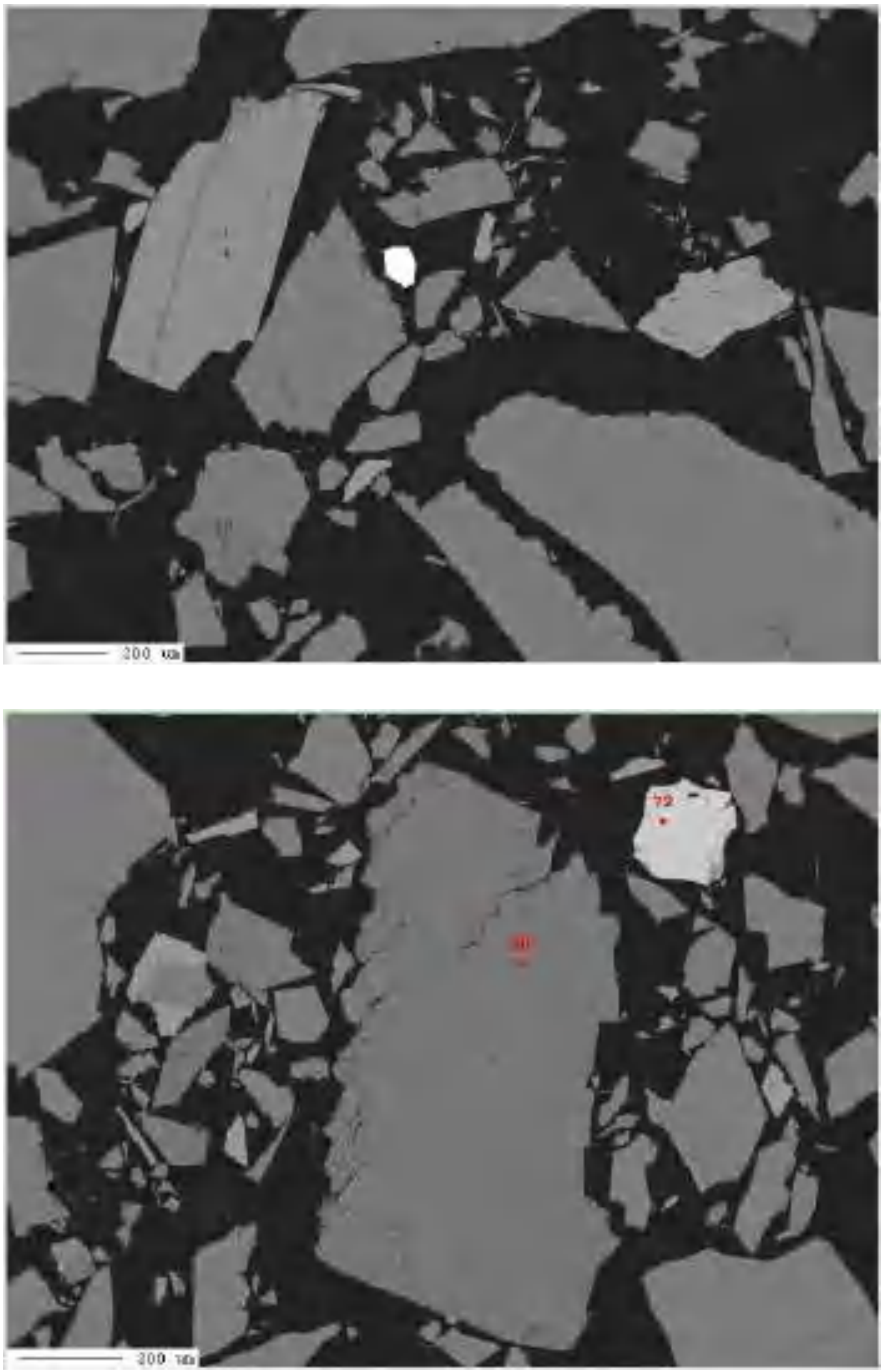
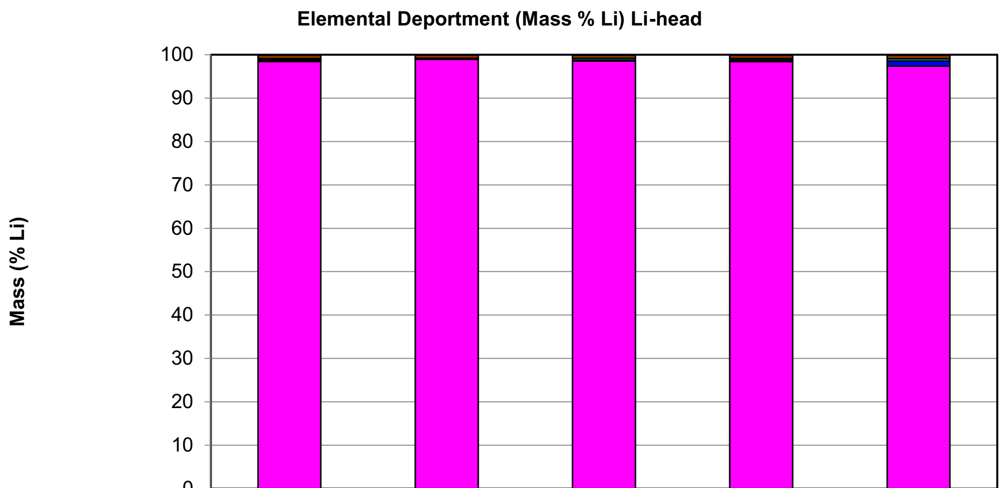


Figure 16: BSE Images of the Analyzed Minerals for the Li Head

Lithium Deportment

Figure 17 illustrates the lithium distribution by mineral in the sample. This is calculated based on the mass of the minerals from the TIMA-X analysis and theoretical lithium concentrations in other potential lithium-bearing minerals. These values are tentative and will be adjusted based on the LA-ICP-MS. Approximately 98.4% of the total lithium is hosted by spodumene.



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Other	0.10	0.10	0.10	0.10	0.10
Mn-Fe-Phosphates	0.00	0.00	0.00	0.00	0.00
Chlorite	0.00	0.00	0.00	0.00	0.00
Muscovite	0.78	0.80	0.77	0.75	0.81
Cookeite	0.34	0.14	0.41	0.38	0.52
Petalite	0.36	0.05	0.16	0.34	1.18
Spodumene	98.4	99.0	98.6	98.4	97.4

Figure 17: Lithium Deportment (Normalized Mass%) for the Li Head

Conclusions and Recommendations

- Rb Head
  - The Rb Head consists mainly of K-feldspars (49.7%), plagioclase (33.1%), and quartz (14.9%), and minor muscovite.
  - K-feldspars host an average of 1.42% Rb_2O and 0.07% Cs_2O. A pure feldspar concentrate is expected to have a chemical composition similar to the average mineral chemistry of the feldspar from the EPMA.
  - Muscovite averages 1.90% Rb_2O 0.19% Cs_2O.
  - K-feldspars hosts ca. 98% of the total rubidium and 95% of the total cesium in the sample.
  - K-feldspars are well liberated (92%), while middlings with plagioclase are minor (7%). Therefore, theoretically, a high purity K-feldspar concentrate is achievable (for the P_{80} of 700 µm).
- Li Head
  - Spodumene is the main lithium mineral in the sample. It is well liberated (89%) at this grind target (P_{80} of ca. 400 µm). Liberation of the spodumene is similar across the size fractions examined (+425 to -150 µm). The pure spodumene particles increase only below the -150 µm fraction.
  - Gangue silicates are also well-liberated.
  - Therefore, spodumene may be floated at a relatively coarse grain size. Theoretically, a high purity lithium concentrate is achievable.
  - Spodumene hosts greater than 98% of the total lithium in the sample.
  - A pure spodumene concentrate is expected to have ca. 3.5% Li.
- The current results reflect the samples examined.
- Additional mineralogical analyses are recommended to determine the variability of the both the rubidium and lithium ores. XRD analysis is a valuable tool to determine the bulk mineralogy of variability samples although it has a 1-2% detection limit. Additional analyses with the TIMA can enhance the XRD information in greater detail the identification of minor minerals, and textural characteristics (grain size, liberation/association etc.).

- The mineralogical data cannot substitute the results from the metallurgical testwork.

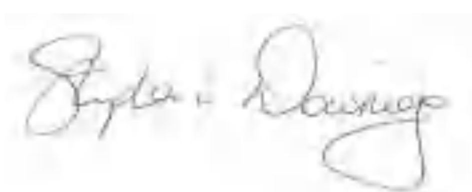
Note that the findings in this report are based on what is mineralogically possible, under ideal separation conditions. For instance, it assumes that it is possible to separate a spodumene grain with a minute attachment of another mineral from a particle that contains no inclusions or attachments. Practically, this separation might be more complex. Thus, the findings in this report should not be considered as a prediction of recovery performance. Rather, this provides insight into the limitations with respect to mineralogical characteristics.

It must be noted, that due to the difference in grain size, all size fractions contain particles that are close to the measurement area (~3 µm) and the spacing of the measurement points and therefore can encounter less precision in the measurements. In addition, the X-ray beam can scatter at the edges of particles and can lead to inaccurate analytical results. As the particles become smaller, the edges constitute a larger percentage of the total particle mass. Therefore, some bias might be introduced, especially in the fine fraction, and caution is advised in interpreting the results in this particular fraction.

Introduction

This report describes a mineralogical test program using High Definition Mineralogy, including TIMA-X (Tescan Integrated Mineral Analyzer), Electron Probe Micro-Analysis (EPMA), Laser Ablation by Inductively Coupled Plasma Mass Spectrometry (LA by ICP-MS), X-ray diffraction analysis (XRD), and chemical assays. The purpose of this test program was to determine the overall mineral assemblage of the samples and define the liberation and association attributes of lithium and rubidium minerals.

Tassos Grammatikopoulos, Ph.D., P.Geo.
Director Technical Services (Mineralogy), NAM



Stephanie Downing, M.Sc.
Mineralogy Manager, Canada

Sample Preparation by: Julian Legault-Seguin
TIMA Operation by: Margot Aldis
Data Processing by: Tassos Grammatikopoulos and Nicole Morton
Report preparation by: Nicole Morton and Tassos Grammatikopoulos
Report reviewed by: Chris Gunning, Cheryl Mina, and Stephanie Downing

Testwork Summary

1. Sample Receipt and Preparation

Two samples, referred to as Rb Head and Li Head, were received by the SGS Advanced Mineralogy Facility from the International Lithium Corp. for mineralogical examination. The samples are taken from the Raleigh Lake Ores, Ontario. The project number 19977-01 and LIMS number MI5018-SEP23 were assigned to the testwork. The testwork was conducted to support the metallurgical testwork currently underway.

Sample preparation and homogenization was conducted (more details in the metallurgical report). Sample charges were used from the Rb Head. A subsample of about 1 kg was stage crushed to a P_{80} of ca. 600-700 µm.

The Li Head represents the -1 mm material from the initial head sample. The Li Head was further stage crushed into 425 µm, screen and recombined into four size fractions including +425 µm, -425/+300 µm, -300/+150 µm and -150 µm.

A subsample from the as-received sample was submitted for XRD analysis by Rietveld refinement to calibrate the major minerals for the TIMA-X analysis. Subsamples were also prepared for EPMA and LA-ICP-MS analyses. Graphite-impregnated polished epoxy grain mounts were prepared from each sample and fraction.

A subsample was micro-riffled from the Rb Head for WRA by XRF, 4-aqcid acid ICP scan, and Rb by four acid digest.

A subsample was micro-riffled from each size fraction from the Li Head for WRA by XRF, and Li by sodium peroxide fusion, ICP-AES, and S by Leco.

The certificate of chemical analyses is given in Appendix A, the XRD report in Appendix B, the complete TIMA-X data in Appendix C, and the EPMA and LA_ICP-MS results in Appendix D.

2. Operational Modes and Quality Control

TIMA-X is an acronym for TESCAN Integrated Mineral Analyzer. It is based on four Energy Dispersive X-Ray (EDX) silicon drift detectors (SDD) attached to a TESCAN MIRA (field-emission gun – FEG) platform which also includes a backscattered electron (BSE) and secondary electron (SE) detectors. The TIMA system utilizes both the EDX and BSE signals to identify minerals at each measurement point, and it is optimized to deal with rapidly acquired low-count spectra. These EDX (and BSE) spectra (and BSE data) are compared to entries in a mineral library on a first match principle to identify the mineral phase, where this mineral library is based on theoretical mineral/phase composition or created by the user based from

BSE, X-ray spectral window counts, and/or ratios. TIMA-X has four X-ray analysis scanning modes to identify mineral/compounds including the High-Resolution Mapping (THRM) mode. The THRM collects a BSE signal and an X-ray spectrum at a set resolution to map the particles, and collect modal and textural information (i.e., liberation, exposure).

The mode of TIMA-X analysis used for this project was Dot Mapping (TDM). The TDM analysis mode uses a BSE grid at a predetermined pixel spacing to segment areas of homogenous BSE intensities and identifies the centre of the greatest inscribed circle (similar to the point spectroscopy), it then creates a grid for the X-ray acquisition with the specified resolution spacing the same as the BSE. The X-ray data from zones of similar BSE and EDS signals are summed to produce a single higher quality spectrum for each final segment, which is used for the mineral identification. This analysis mode is good for modal mineralogy, grain size, and liberation analysis.

Thus, the identification of spodumene, petalite and other potential Li-bearing was based on the aluminum-silica ratios since lithium cannot be detected with conventional instruments (TIMA-X or electron probe). Laser ablation analyses are required for proper lithium quantification.

2.1. TIMA-X Assay Reconciliation

Each polished section from each size fraction of sample was submitted for mineralogical analyses with TIMA-X using the Dot Mapping mode of measurement. All data was processed with the TIMA-X software version 2.9.0. A mineral list developed for the analyzed sample is shown in Table 11.

Table 11: Mineral List

Mineral	Formula
Amphibole	$(Ca,Na)_{2-3}(Mg,Fe,Al)_5Si_6(Si,Al)_2O_{22}(OH)_2$
Apatite	$Ca_5(PO_4)_3F$
Beryl	$Be_3Al_2(Si_6O_{18})$
Biotite	$K(Mg,Fe)_3(AlSi_3O_{10})(OH)_2$
Carbonates, i.e., calcite	$CaCO_3$
Chlorite	$(Fe,Mg)_5Al(Si_3Al)O_{10}(OH)_8$
Cookeite	$LiAl_5Si_3O_{10}(OH)_8$
Garnet	$(Ca,Mg,Mn)_3(V,Al, Fe)_2(SiO_4)_3$
Fe-(Ti)-Oxides	
Ilmenite	$FeTiO_3$
Magnetite	Fe_3O_4
Magnetite	Fe_2O_3
K-Feldspars	$KAlSi_3O_8$
Mn-Fe-Phosphates	Li (?) bearing Mn-Fe-Phosphates
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$
Petalite	$LiAlSi_4O_{10}$
Plagioclase	$(NaSi,CaAl)AlSi_2O_8$
Pyroxene	$(Ca,Na)(Mg,Fe,Al)(Si,Al)_2O_6$
Quartz	SiO_2
Spodumene	$LiAlSi_2O_6$
Sulphides, e.g., pyrite	FeS_2
Ta-Nb-Oxides, e.g., tantalite, columbite	$(Fe,Mn)Ta_2O_6$
Titanite	$CaTiO(SiO_4)$
Tourmaline	$(Na,Ca)(Li,Mg,Al)(Al,Fe,Mn)_6(BO_3)_3(Si_6O_{18})(OH)_4$

Key TIMA-X mineralogical assays have been regressed with the chemical assays, as presented in Table 12 and Figure 18. Overall correlation, as measured by R-squared criteria, was 1.0.

Table 12: TIMA-X Calculated and Direct Assay Reconciliation

Sample	Li-head				Rb-head
Element	+425 um	-425/+300 um	-300/+150 um	-150 um	Head
Li TIMA Calculated	0.53	0.36	0.29	0.26	0.03
Li Chemical Assay	0.59	0.36	0.27	0.29	0.06
Na TIMA Calculated	3.61	4.82	5.56	5.60	2.94
Na Chemical Assay	3.21	4.58	5.30	5.28	2.82
Al TIMA Calculated	7.87	8.55	9.00	8.98	8.66
Al Chemical Assay	7.94	8.36	8.73	8.89	8.89
Si TIMA Calculated	35.7	34.4	33.5	33.5	32.9
Si Chemical Assay	35.5	34.5	33.8	33.8	32.5
K TIMA Calculated	1.17	1.20	1.01	1.15	6.44
K Chemical Assay	1.34	1.24	1.11	1.23	6.61
Ca TIMA Calculated	0.25	0.39	0.42	0.55	0.23
Ca Chemical Assay	0.14	0.21	0.22	0.31	0.11
Mn TIMA Calculated	0.15	0.25	0.39	0.15	0.04
Mn Chemical Assay	0.14	0.19	0.17	0.15	0.02
Fe TIMA Calculated	0.29	0.26	0.30	0.28	0.09
Fe Chemical Assay	0.43	0.35	0.31	0.39	0.22
Rb TIMA Calculated	0.15	0.14	0.12	0.13	0.66
Rb Chemical Assay	-	-	-	-	0.73

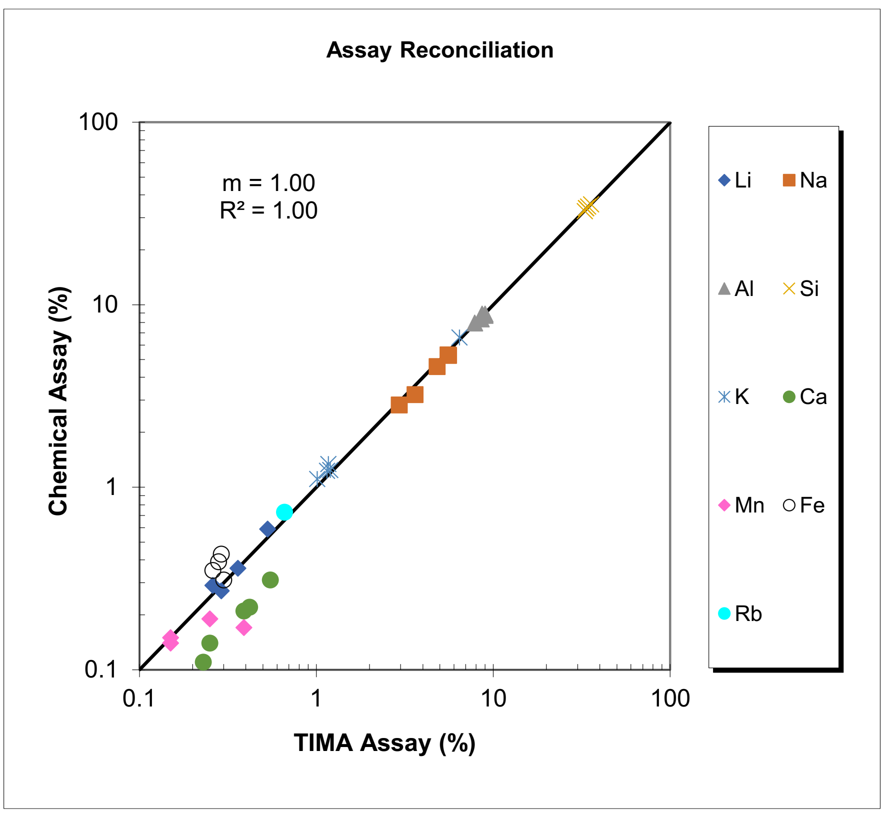


Figure 18: TIMA-X Calculated and Direct Assay Reconciliation

2.2. Mineral Identification Considerations

The mineral identification is based on the elemental composition of the minerals. Lithium cannot be detected using SEM based technologies (TIMA-X, Electron Microprobe). The identification of spodumene, and Li-phosphates is based on the major element ratios; for example, Al-Si for the spodumene and Al-P-Mn for the phosphates.

2.3. Liberation and Association

The liberation and association characteristics of spodumene, K-feldspars, plagioclase, and muscovite were examined. For the purposes of this analysis, particle liberation is defined based on 2D particle area percent. Particles are classified in the following groups (in descending order) based on mineral-of-interest area percent: pure (100% of the total particle area by volume), free (≥95%), and liberated (≥80%). The non-liberated grains have been classified according to association characteristics, where binary association groups refer to particle area percent greater than or equal to 95% of the two minerals or mineral groups. The complex groups refer to particles with ternary, quaternary, and greater mineral associations including the mineral of interest.

Association of spodumene classes were defined as shown below. The same category principle was applied for the other minerals for the liberation and association calculations.

- Barren – a particle that has 0% of spodumene
- Pure spodumene - particle that has 100% of spodumene
- Free spodumene - a particle that has ≥95% of spodumene
- Liberated spodumene - a particle that has ≤95% to ≥80% of spodumene
- Binary/ternary spodumene (Spd) : cookeite/petalite - a particle that has ≥95 area% of spodumene: cookeite/petalite
- Binary/ternary spodumene (Spd) : beryl/tourmaline - a particle that has ≥95 area% of spodumene: beryl/tourmaline
- Binary/ternary spodumene (Spd) : quartz - a particle that has ≥95 area% of spodumene: quartz
- Binary/ternary spodumene (Spd) : plagioclase - a particle that has ≥95 area% of spodumene: plagioclase
- Binary/ternary spodumene (Spd) : K-feldspars - a particle that has ≥95 area% of spodumene: K-feldspars
- Ternary spodumene (Spd) : Quartz : K-feldspars - a particle that has ≥95 area% of spodumene: Quartz : K-feldspars
- Binary/ternary spodumene (Spd) : muscovite - a particle that has ≥95 area% of spodumene: muscovite
- Binary/ternary spodumene (Spd) : other silicates - a particle that has ≥95 area% of spodumene : other silicates
- Binary/ternary spodumene (Spd) : garnet - a particle that has ≥95 area% of spodumene : garnet
- Binary/ternary spodumene (Spd) : carbonates - a particle that has ≥95 area% of spodumene : carbonates
- Binary/ternary spodumene (Spd) : phosphates - a particle that has ≥95 area% of spodumene : phosphates

- Binary/ternary spodumene (Spd) : Fe-(Ti)-Oxides - a particle that has ≥95 area% of spodumene : Fe-(Ti)-Oxides
- Binary/ternary spodumene (Spd) : Nb-Ta-Oxides - a particle that has ≥95 area% of spodumene : Fe-(Ti)-Oxides
- Binary/ternary spodumene (Spd) : sulphides - a particle that has ≥95 area% of spodumene: muscovite
- Binary/ternary spodumene (Spd):other minerals- a particle that has ≥95 area% of spodumene: other
- Spodumene: complex - particles that do not fall into the above categories

Note: the complex category refers to ternary and quaternary particles and does not necessarily reflect the complexity of the middling particles.

The liberation and association of the minerals is calculated based on their area% as a function of their volume and mass%, and the exposure based on the free surface (Figure 19).

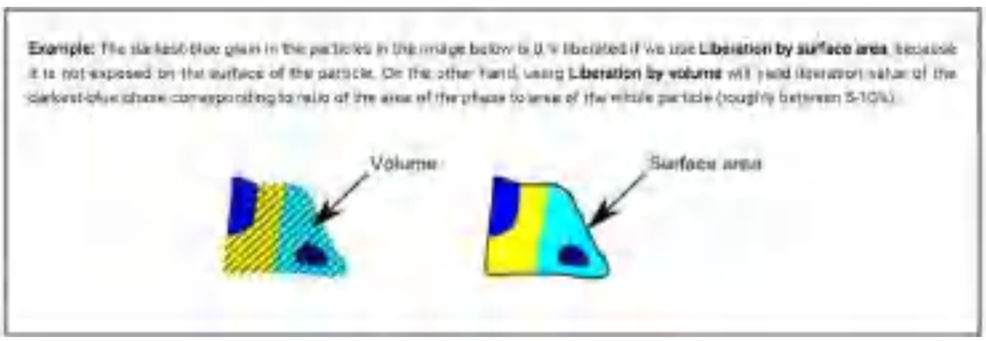


Figure 19: Liberation by Free Surface and Volume of a Mineral

Appendix A – Certificate of Analysis

SGS Canada Inc.
P.O. Box 4300 - 185 Concession St.
Lakefield - Ontario - KOL 2HO
Phone: 705-652-2000 FAX: 705-652-6365

LR Internal Dept 14
Attn : Tassos Grammatikopoulos

23-October-2023

Date Rec. : 13 September 2023
LR Report : **CA02192-SEP23**
Project : CA20M-00000-110-19977-0 1
Client Ref : MI5018-SEP23

CERTIFICATE OF ANALYSIS

Final Report

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %
1: Rb-head	69.6	16.8	0.31	0.02	0.15	3.80	7.96

Sample ID	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %	Sum %	Rb %
1: Rb-head	< 0.01	0.02	0.03	0.02	< 0.01	0.40	99.1	0.73

Jordan Graham
Project Coordinator

OnLine LIMS

0003508542

SGS Canada Inc.
P.O. Box 4300 - 185 Concession St.
Lakefield - Ontario - KOL 2HO
Phone: 705-652-2000 FAX: 705-652-6365

LR Internal Dept 14
Attn : Tassos Grammatikopoulos

23-October-2023

Date Rec. : 13 September 2023
LR Report : **CA02191-SEP23**
Project : CA20M-00000-110-19977-01
Client Ref : MI5018-SEP23

CERTIFICATE OF ANALYSIS

Final Report

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	Li g/t	Mg g/t
1: Li-head	< 2	63400	< 30	7	60.9	< 20	1910	< 2	< 4	63	13	2730	12000	3440	124

Sample ID	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	V g/t	Y g/t	Zn g/t
1: Li-head	1450	< 5	46700	< 20	< 50	< 20	< 10	< 30	< 20	14	36	< 30	4	4.4	64

Jordan Graham
Project Coordinator

OnLine LIMS

0003508553

SGS Canada Inc.
P.O. Box 4300 - 185 Concession St.
Lakefield - Ontario - KOL 2HO
Phone: 705-652-2000 FAX: 705-652-6365

LR Internal Dept 14
Attn : C. Gunning

23-October-2023

Date Rec. : 21 September 2023
LR Report : **CA02326-SEP23**
Project : CA20M-00000-110-19977-01
Client Ref : MI5018-SEP23

CERTIFICATE OF ANALYSIS

Final Report

Sample ID	Li %	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %
1: Li-head +425um	0.59	75.9	15.0	0.62	0.03	0.19	4.33
2: Li-head -425/+300um	0.36	73.7	15.8	0.50	0.03	0.29	6.17
3: Li-head -300/+150um	0.27	72.2	16.5	0.44	0.04	0.31	7.15
4: Li-head -150um	0.29	72.3	16.8	0.56	0.04	0.44	7.12

Sample ID	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %	Sum %
1: Li-head +425um	1.62	< 0.01	< 0.01	0.18	0.04	< 0.01	0.60	98.5
2: Li-head -425/+300um	1.49	< 0.01	0.01	0.24	0.03	< 0.01	0.59	98.9
3: Li-head -300/+150um	1.34	< 0.01	< 0.01	0.22	0.03	< 0.01	0.60	98.8
4: Li-head -150um	1.48	< 0.01	0.02	0.19	0.03	< 0.01	0.74	99.8

Jordan Graham
Project Coordinator

OnLine LIMS

0003508556

Appendix B – XRD Results

Quantitative X-Ray Diffraction by Rietveld Refinement

Report Prepared for:	*Metallurgical Operations*
Project Number/ LIMS No.	*19977-01/MI5018-SEP23*
Sample Receipt:	*September 12, 2023*
Sample Analysis:	*September 18, 2023*
Reporting Date:	*October 25, 2023*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 35 kV, 40 mA; Detector: LYNXEYE Regular Scanning: Step: 0.02°, Step time: 0.75s, 2θ range: 6-80°
Interpretations :	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva and Topas software.
Detection Limit :	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Quantitative XRD Results
3) XRD Pattern(s)

Zhihai (Adrian) Zhang, Ph.D
Mineralogist

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

ACCREDITATION: SGS Natural Resources Lakefield is accredited to the requirements of ISO/IEC 17025 for specific tests as listed on our scope of accreditation, including geochemical, mineralogical and trade mineral tests. To view a list of the accredited methods, please visit the following website and search SGS Canada Inc. - Minerals: https://www.scc.ca/en/search/palcan.

Method Summary

The Rietveld Method of Mineral Identification by XRD (ME-LR-MIN-MET-MN-D05) method used by SGS Natural Resources is accredited to the requirements of ISO/IEC 17025.

Mineral Identification and Interpretation:

Mineral identification and interpretation involves 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, except when internal standards have been added by request. Mineral proportions may be strongly influenced by crystallinity, crystal structure and preferred orientations. Mineral or compound identification and quantitative analysis results should be accompanied by supporting chemical assay data or other additional tests.

Quantitative Rietveld Analysis:

Quantitative Rietveld Analysis is performed by using Topas 4.2 (Bruker AXS), a graphics based profile analysis program built around a non-linear least squares fitting system, to determine the amount of different phases present in a multicomponent sample. Whole pattern analyses are predicated by the fact that the X-ray diffraction pattern is a total sum of both instrumental and specimen factors. Unlike other peak intensity-based methods, the Rietveld method uses a least squares approach to refine a theoretical line profile until it matches the obtained experimental patterns.

Rietveld refinement is completed with a set of minerals specifically identified for the sample. Zero values indicate that the mineral was included in the refinement calculations, but the calculated concentration was less than 0.05wt%. Minerals not identified by the analyst are not included in refinement calculations for specific samples and are indicated with a dash.

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

Summary of Rietveld Quantitative Analysis X-Ray Diffraction Results

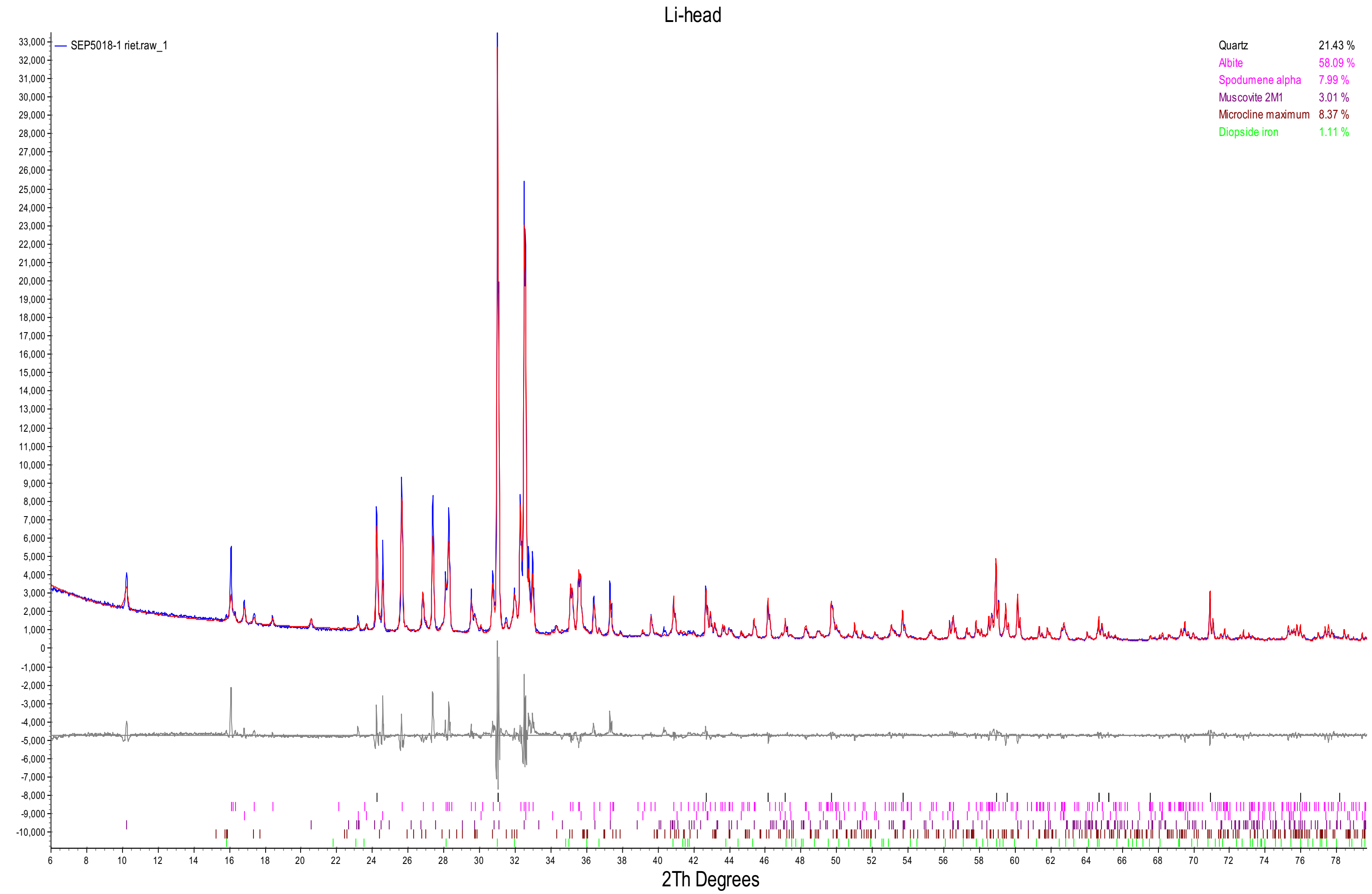
Mineral/Compound	Li-head SEP5018-1 (wt %)	Rb-head SEP5018-2 (wt %)
Quartz	21.4	13.2
Albite	58.1	33.2
Spodumene	8.0	2.5
Muscovite	3.0	2.0
Microcline	8.4	46.6
Diopside	1.1	2.6
TOTAL	100	100

Zero values indicate that the mineral was included in the refinement, but the calculated concentration is below a measurable value.

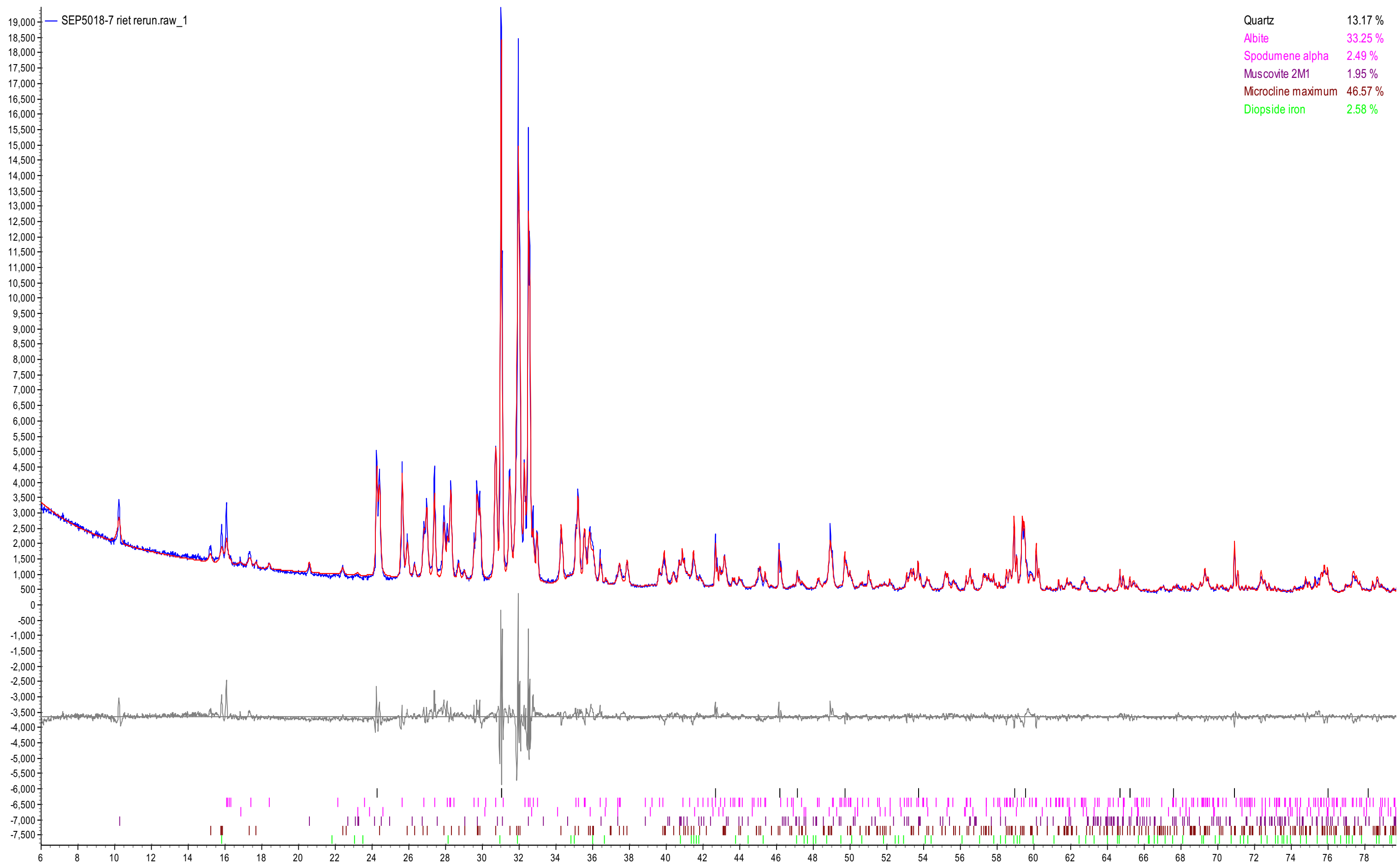
Dashes indicate that the mineral was not identified by the analyst and not included in the refinement calculation for the sample.

The weight percent quantities indicated have been normalized to a sum of 100%. The quantity of amorphous material has not been determined.

Mineral/Compound	Formula
Quartz	SiO_2
Albite	$NaAlSi_3O_8$
Spodumene	$LiAlSi_2O_6$
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$
Microcline	$KAlSi_3O_8$
Diopside	$CaMgSi_2O_6$


Li-head
SEP5018-1 riet.raw_1
Quartz 21.43 %
Albite 58.09 %
Spodumene alpha 7.99 %
Muscovite 2M1 3.01 %
Microcline maximum 8.37 %
Diopside iron 1.11 %
2Th Degrees

Rb-head


SEP5018-7 riet rerun.raw_1
Quartz 13.17 %
Albite 33.25 %
Spodumene alpha 2.49 %
Muscovite 2M1 1.95 %
Microcline maximum 46.57 %
Diopside iron 2.58 %
2Th Degrees

Appendix C – TIMA-X Data

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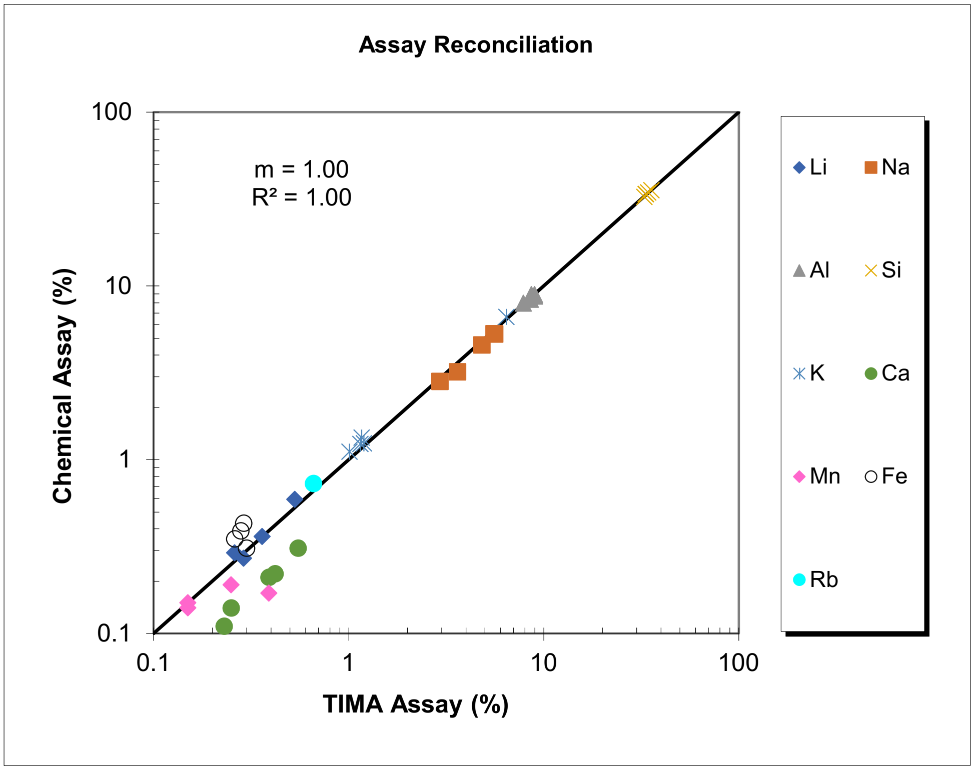
Assay Reconciliation



Sample	Li-head				Rb-head
Element	+425 um	-425/+300 um	-300/+150 um	-150 um	Head
Li TIMA Calculated	0.53	0.36	0.29	0.26	0.03
Li Chemical Assay	0.59	0.36	0.27	0.29	0.06
Na TIMA Calculated	3.61	4.82	5.56	5.60	2.94
Na Chemical Assay	3.21	4.58	5.30	5.28	2.82
Al TIMA Calculated	7.87	8.55	9.00	8.98	8.66
Al Chemical Assay	7.94	8.36	8.73	8.89	8.89
Si TIMA Calculated	35.7	34.4	33.5	33.5	32.9
Si Chemical Assay	35.5	34.5	33.8	33.8	32.5
K TIMA Calculated	1.17	1.20	1.01	1.15	6.44
K Chemical Assay	1.34	1.24	1.11	1.23	6.61
Ca TIMA Calculated	0.25	0.39	0.42	0.55	0.23
Ca Chemical Assay	0.14	0.21	0.22	0.31	0.11
Mn TIMA Calculated	0.15	0.25	0.39	0.15	0.04
Mn Chemical Assay	0.14	0.19	0.17	0.15	0.02
Fe TIMA Calculated	0.29	0.26	0.30	0.28	0.09
Fe Chemical Assay	0.43	0.35	0.31	0.39	0.22
Rb TIMA Calculated	0.15	0.14	0.12	0.13	0.66
Rb Chemical Assay	-	-	-	-	0.73

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MI5018-SEP23

Modals

Survey		CALR-19977-01 / MI5018-SEP23									
Project		**International Lithium**									
Sample		**Li-head**									**Rb-head**
Fraction		**Combined**	**+425 um**		**-425/+300 um**		**-300/+150 um**		**-150 um**		**Head**
Mass % of Size Fraction [%]		100.0	19.4		22.2		34.7		23.7		100.0
Median Particle Size (µm)		240	527		342		208		81		412
		Sample	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**
Mineral Mass (%)	Spodumene	9.73	2.91	15.0	2.26	10.2	2.84	8.19	1.73	7.29	0.94
	Petalite	0.06	0.00	0.01	0.01	0.03	0.02	0.05	0.04	0.15	0.03
	Cookeite	0.09	0.01	0.08	0.03	0.11	0.03	0.09	0.02	0.10	0.00
	Beryl	0.19	0.04	0.20	0.03	0.15	0.04	0.12	0.08	0.34	0.01
	Tourmaline	0.02	0.00	0.01	0.00	0.02	0.01	0.02	0.01	0.03	0.01
	Quartz	22.9	6.49	33.4	5.40	24.3	6.62	19.1	4.36	18.4	14.9
	Plagioclase	56.1	7.80	40.2	11.9	53.7	21.5	62.0	14.8	62.6	33.1
	K-Feldspars	7.02	1.26	6.50	1.71	7.68	2.26	6.51	1.79	7.57	49.7
	Muscovite	2.43	0.74	3.82	0.56	2.51	0.68	1.95	0.45	1.92	0.81
	Biotite	0.05	0.01	0.03	0.00	0.01	0.01	0.02	0.04	0.16	0.08
	Amphibole/Pyroxene	0.02	0.00	0.02	0.00	0.01	0.00	0.01	0.01	0.04	0.01
	Chlorite	0.02	0.00	0.00	0.00	0.01	0.01	0.02	0.01	0.03	0.01
	Titanite	0.04	0.00	0.01	0.01	0.02	0.02	0.05	0.01	0.04	0.01
	Garnet	1.03	0.11	0.58	0.23	1.02	0.54	1.56	0.15	0.63	0.17
	Carbonates	0.25	0.02	0.08	0.06	0.26	0.07	0.19	0.11	0.47	0.06
	Apatite	0.02	0.00	0.00	0.00	0.00	0.01	0.04	0.00	0.01	0.00
	Mn-Fe-Phosphates	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Fe-(Ti)-Oxides	0.03	0.00	0.01	0.00	0.00	0.01	0.02	0.03	0.11	0.01
	Nb-Ta-Oxides	0.02	0.00	0.00	0.00	0.00	0.01	0.02	0.01	0.03	0.00
	Sulphides	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.03	0.07
	Other	0.03	0.00	0.01	0.00	0.02	0.01	0.03	0.02	0.07	0.01
	Total	100.0	19.4	100.0	22.2	100.0	34.7	100.0	23.7	100.0	100.0
Mean Line Intercept Length (µm)	Spodumene	91	174		114		74		28		116
	Petalite	7	7		7		7		7		7
	Cookeite	23	24		29		25		14		11
	Beryl	19	33		25		16		12		15
	Tourmaline	9	8		10		12		7		7
	Quartz	102	231		145		87		28		119
	Plagioclase	89	219		151		96		35		93
	K-Feldspars	80	179		131		80		29		110
	Muscovite	37	48		43		35		17		22
	Biotite	13	11		9		14		14		39
	Amphibole/Pyroxene	14	20		8		12		12		28
	Chlorite	14	10		17		22		11		16
	Titanite	15	7		16		25		10		8
	Garnet	82	108		112		98		30		87
	Carbonates	37	31		64		40		26		25
	Apatite	68	8		17		123		22		17
	Mn-Fe-Phosphates	6	0		0		0		6		0
	Fe-(Ti)-Oxides	15	17		9		42		13		5
	Nb-Ta-Oxides	35	27		17		53		35		13
	Sulphides	16	6		10		9		20		34
	Other	14	8		16		15		15		9

SGS Canada Inc.
International Lithium
CALR-19977-01
MI5018-SEP23

Modals Condensed

Survey		**CALR-19977-01 / MI5018-SEP23**									
Project		**International Lithium**									
Sample		**Li-head**									**Rb-head**
Fraction		**Combined**	**+425 um**		**-425/+300 um**		**-300/+150 um**		**-150 um**		**Head**
Mass % of Size Fraction [%]		100.0	19.4		22.2		34.7		23.7		100.0
Median Particle Size (µm)		240	527		342		208		81		412
		Sample	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**
Mineral Mass (%)	Spodumene	9.73	2.91	15.0	2.26	10.2	2.84	8.19	1.73	7.29	0.94
	Petalite/Cookeite	0.15	0.02	0.09	0.03	0.14	0.05	0.13	0.06	0.25	0.04
	Beryl/Tourmaline	0.21	0.04	0.21	0.04	0.17	0.05	0.14	0.09	0.37	0.02
	Quartz	22.9	6.49	33.4	5.40	24.3	6.62	19.1	4.36	18.4	14.9
	Plagioclase	56.1	7.80	40.2	11.9	53.7	21.5	62.0	14.8	62.6	33.1
	K-Feldspars	7.02	1.26	6.50	1.71	7.68	2.26	6.51	1.79	7.57	49.7
	Muscovite	2.43	0.74	3.82	0.56	2.51	0.68	1.95	0.45	1.92	0.81
	Other Silicates	0.12	0.01	0.06	0.01	0.05	0.04	0.10	0.06	0.26	0.11
	Garnet	1.03	0.11	0.58	0.23	1.02	0.54	1.56	0.15	0.63	0.17
	Carbonates	0.25	0.02	0.08	0.06	0.26	0.07	0.19	0.11	0.47	0.06
	Phosphates	0.02	0.00	0.00	0.00	0.00	0.01	0.04	0.00	0.01	0.00
	Fe-(Ti)-Oxides	0.03	0.00	0.01	0.00	0.00	0.01	0.02	0.03	0.11	0.01
	Nb-Ta-Oxides	0.02	0.00	0.00	0.00	0.00	0.01	0.02	0.01	0.03	0.00
	Sulphides	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.03	0.07
	Other	0.03	0.00	0.01	0.00	0.02	0.01	0.03	0.02	0.07	0.01
	Total	100.0	19.4	100.0	22.2	100.0	34.7	100.0	23.7	100.0	100.0
Mean Line Intercept Length (µm)	Spodumene	91	174		114		74		28		116
	Petalite/Cookeite	13	18		18		13		8		7
	Beryl/Tourmaline	18	29		22		15		12		10
	Quartz	102	231		145		87		28		119
	Plagioclase	89	219		151		96		35		93
	K-Feldspars	80	179		131		80		29		110
	Muscovite	37	48		43		35		17		22
	Other Silicates	14	12		13		21		13		28
	Garnet	82	108		112		98		30		87
	Carbonates	37	31		64		40		26		25
	Phosphates	68	8		17		123		22		17
	Fe-(Ti)-Oxides	15	17		9		42		13		5
	Nb-Ta-Oxides	35	27		17		53		35		13
	Sulphides	16	6		10		9		20		34
	Other	14	8		16		15		15		9

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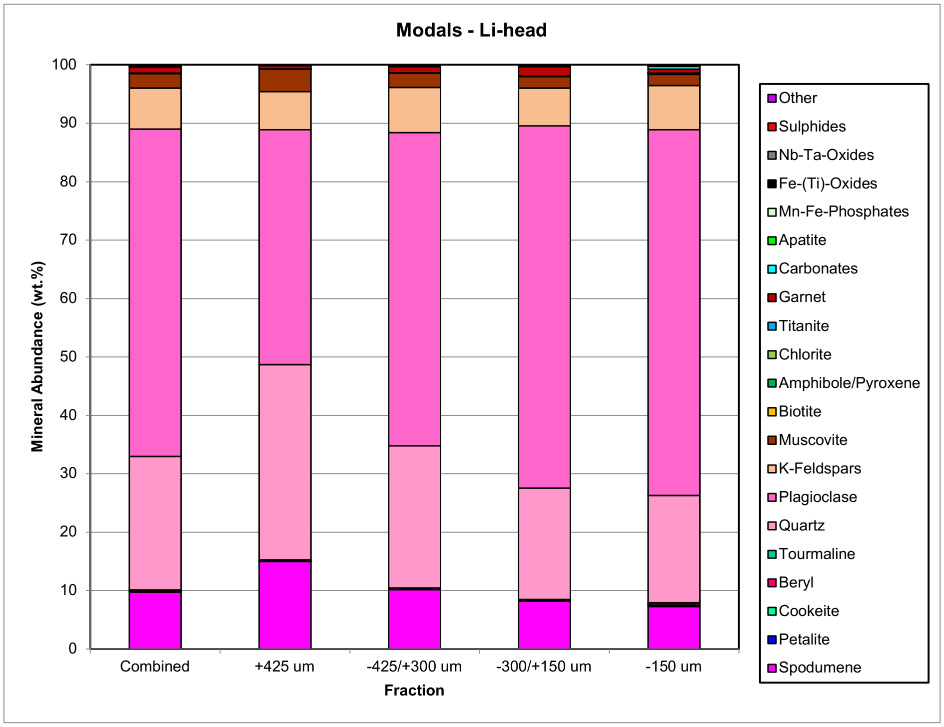
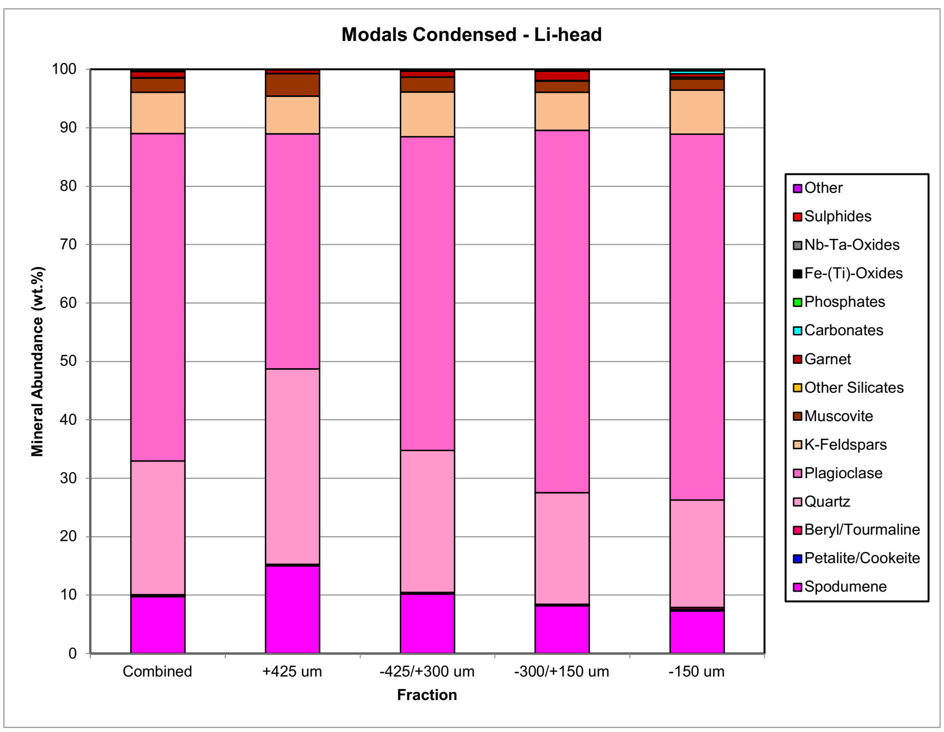
Modal Chart





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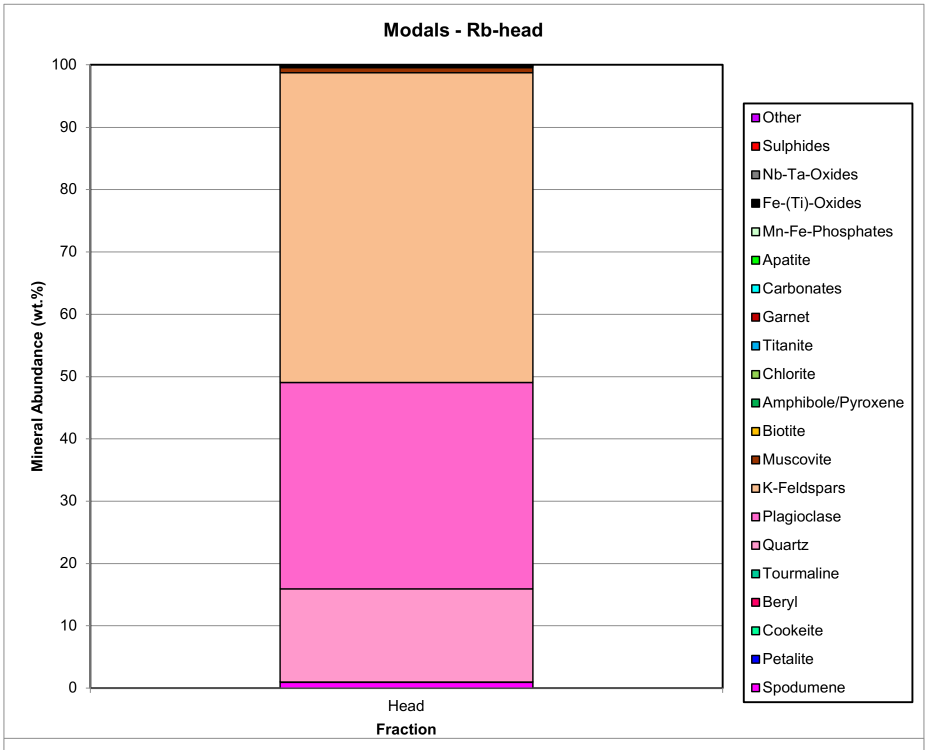
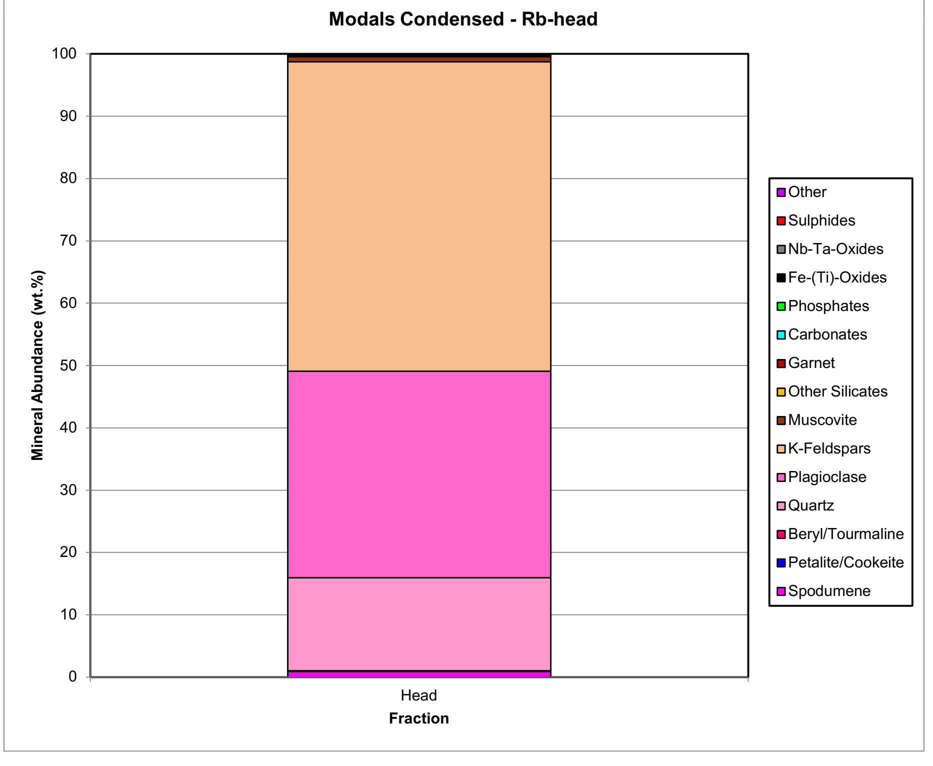
Modal Chart





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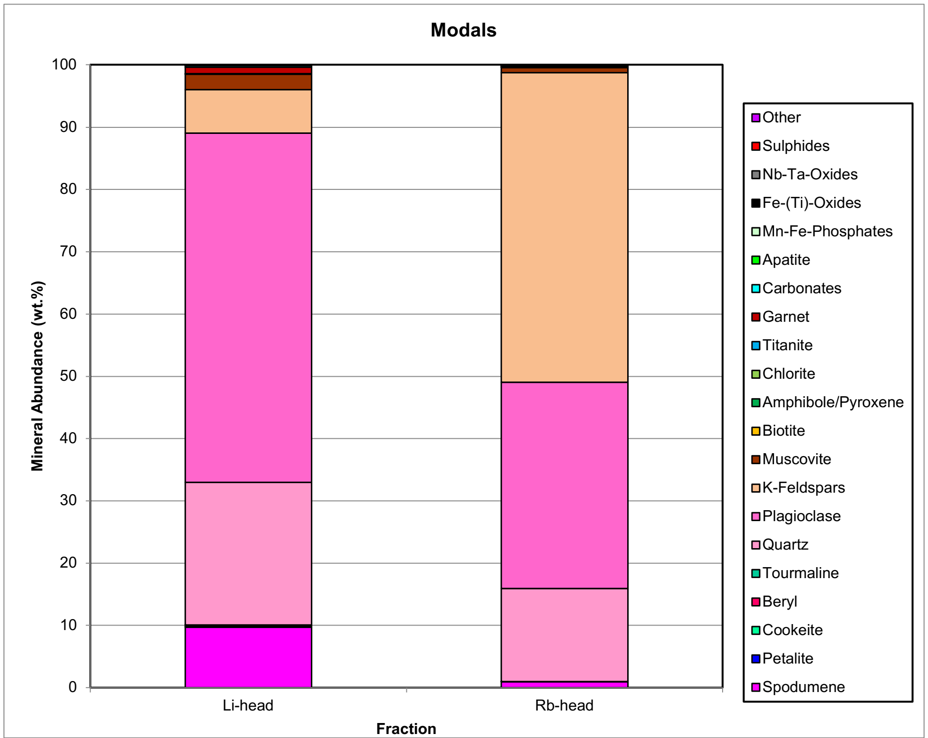
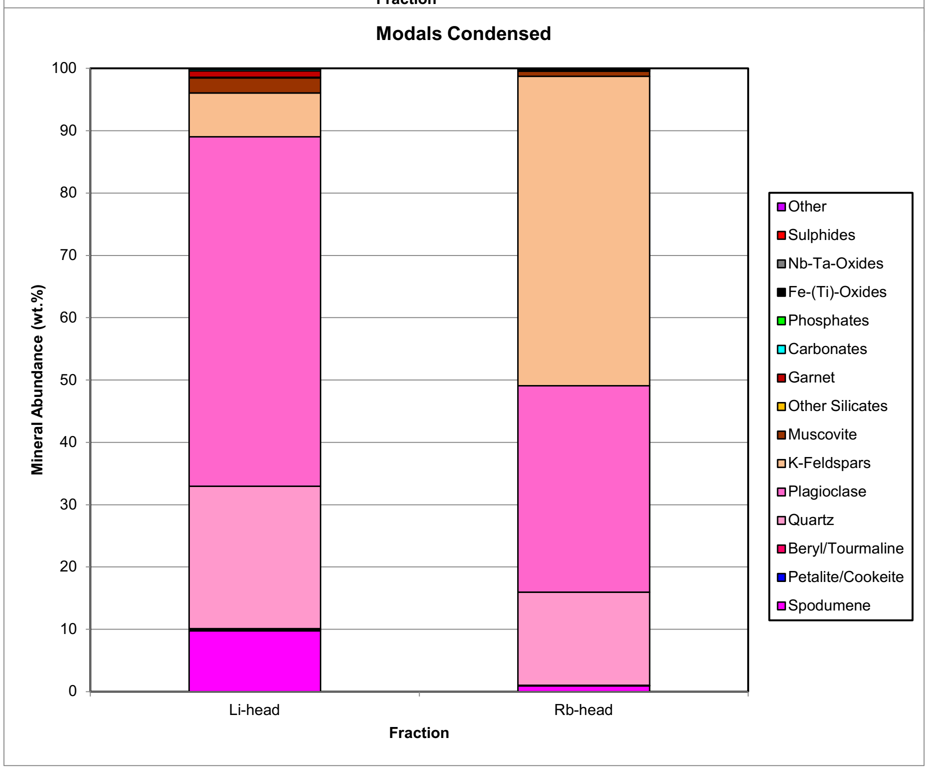
Modal Chart





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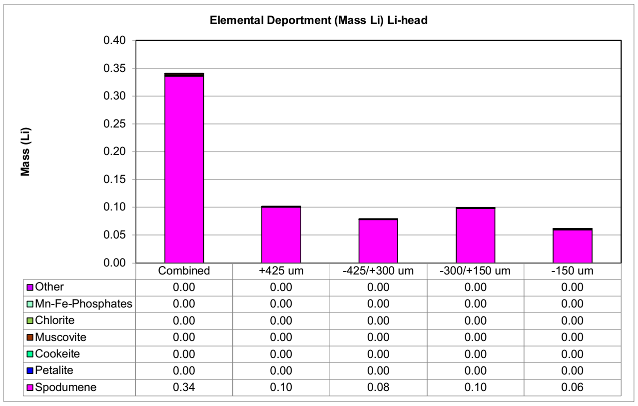
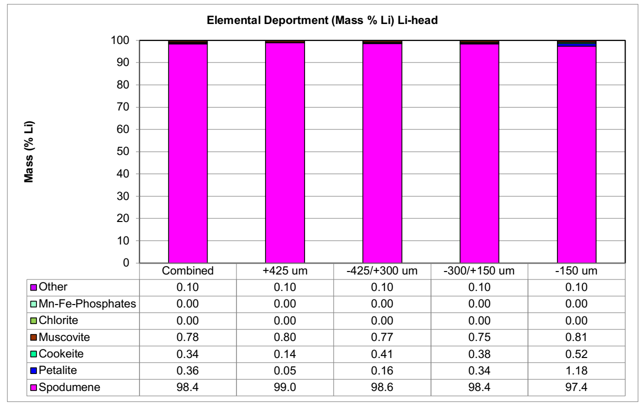
MI5018-SEP23

Li Deportment



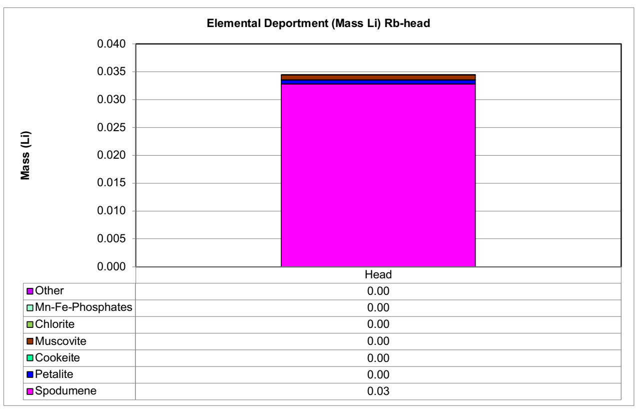


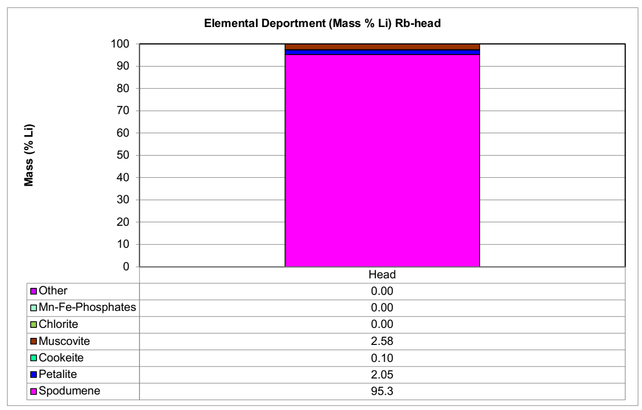
Elemental Deportment (Mass Li) Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Spodumene	0.34	0.10	0.08	0.10	0.06
Petalite	0.00	0.00	0.00	0.00	0.00
Cookeite	0.00	0.00	0.00	0.00	0.00
Muscovite	0.00	0.00	0.00	0.00	0.00
Chlorite	0.00	0.00	0.00	0.00	0.00
Mn-Fe-Phosphates	0.00	0.00	0.00	0.00	0.00
Other	0.00	0.00	0.00	0.00	0.00
Total	0.34	0.10	0.08	0.10	0.06
Total (% in fraction)	100.0	29.7	23.2	29.2	17.9

Elemental Deportment (Mass % Li) Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Spodumene	98.4	99.0	98.6	98.4	97.4
Petalite	0.36	0.05	0.16	0.34	1.18
Cookeite	0.34	0.14	0.41	0.38	0.52
Muscovite	0.78	0.80	0.77	0.75	0.81
Chlorite	0.00	0.00	0.00	0.00	0.00
Mn-Fe-Phosphates	0.00	0.00	0.00	0.00	0.00
Other	0.10	0.10	0.10	0.10	0.10
Total	100.0	100.0	100.0	100.0	100.0





Elemental Deportment (Mass Li) Rb-head

Mineral Name	Head
Spodumene	0.03
Petalite	0.00
Cookeite	0.00
Muscovite	0.00
Chlorite	0.00
Mn-Fe-Phosphates	0.00
Other	0.00
Total	0.03

Elemental Deportment (Mass % Li) Rb-head

Mineral Name	Head
Spodumene	95.3
Petalite	2.05
Cookeite	0.10
Muscovite	2.58
Chlorite	0.00
Mn-Fe-Phosphates	0.00
Other	0.00
Total	100.0

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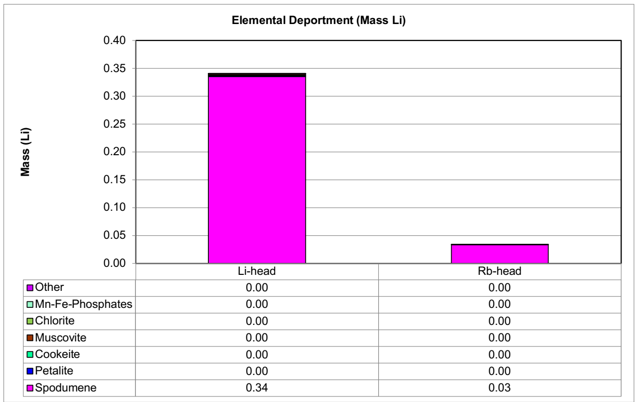
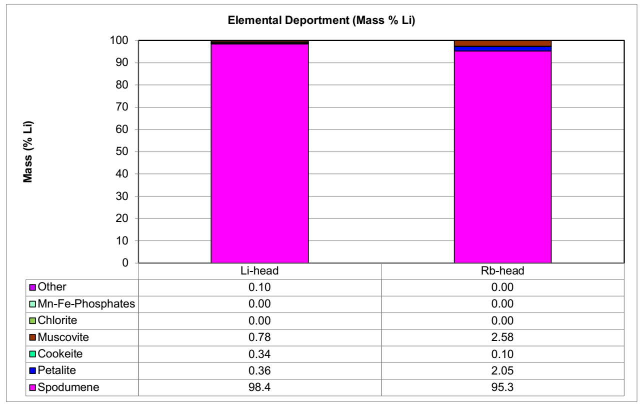
MI5018-SEP23

Li Deportment





Elemental Deportment (Mass Li)

Mineral Name	Li-head	Rb-head
Spodumene	0.34	0.03
Petalite	0.00	0.00
Cookeite	0.00	0.00
Muscovite	0.00	0.00
Chlorite	0.00	0.00
Mn-Fe-Phosphates	0.00	0.00
Other	0.00	0.00
Total	0.34	0.03

Elemental Deportment (Mass % Li)

Mineral Name	Li-head	Rb-head
Spodumene	98.4	95.3
Petalite	0.36	2.05
Cookeite	0.34	0.10
Muscovite	0.78	2.58
Chlorite	0.00	0.00
Mn-Fe-Phosphates	0.00	0.00
Other	0.10	0.00
Total	100.0	100.0

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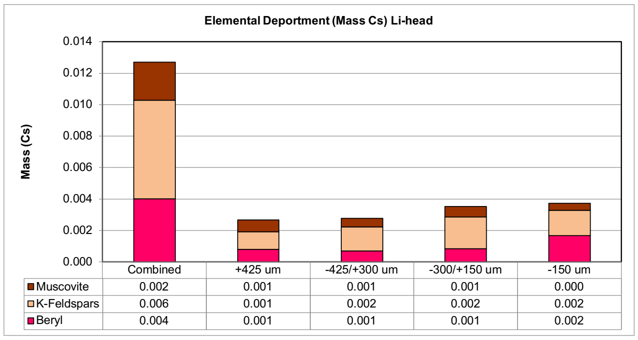
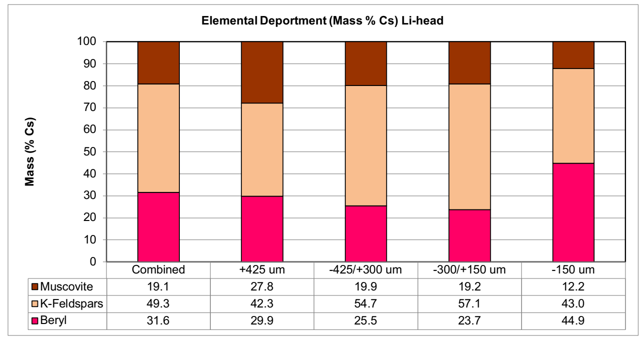
MI5018-SEP23

Cs Deportment



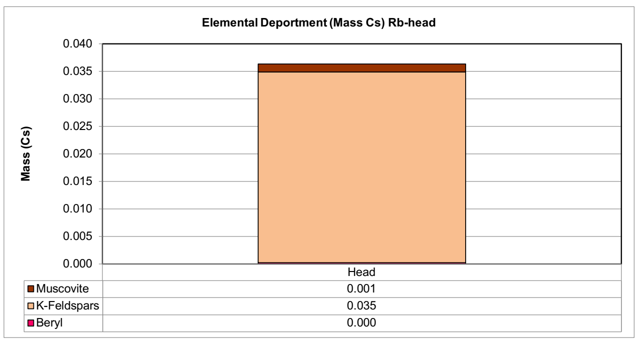


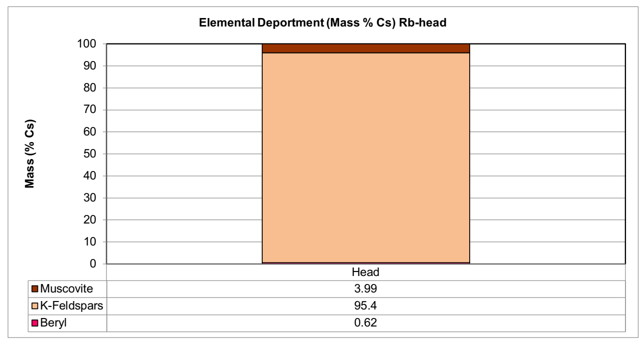
Elemental Deportment (Mass Cs) Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Beryl	0.004	0.001	0.001	0.001	0.002
K-Feldspars	0.006	0.001	0.002	0.002	0.002
Muscovite	0.002	0.001	0.001	0.001	0.000
Total	0.013	0.003	0.003	0.004	0.004
Total (% in fraction)	100.0	21.0	21.8	27.8	29.4

Elemental Deportment (Mass % Cs) Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Beryl	31.6	29.9	25.5	23.7	44.9
K-Feldspars	49.3	42.3	54.7	57.1	43.0
Muscovite	19.1	27.8	19.9	19.2	12.2
Total	100.0	100.0	100.0	100.0	100.0



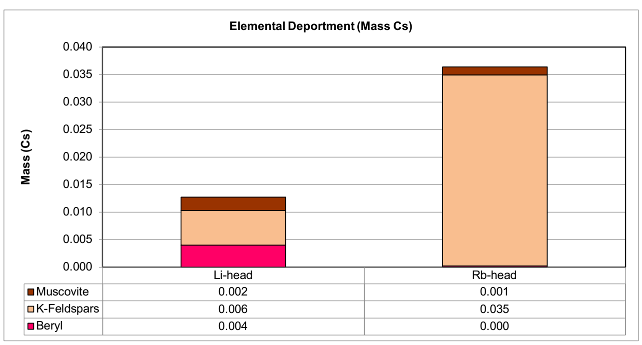


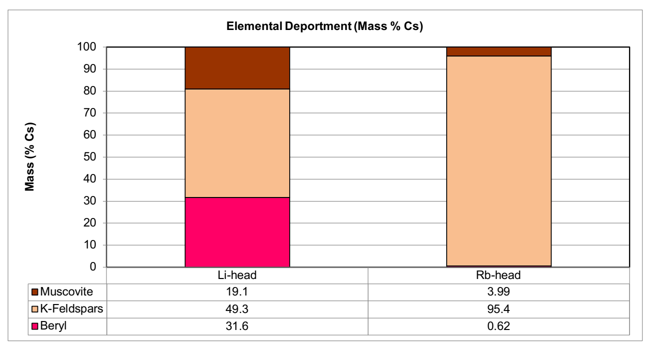
Elemental Deportment (Mass Cs) Rb-head

Mineral Name	Head
Beryl	0.000
K-Feldspars	0.035
Muscovite	0.001
Total	0.036

Elemental Deportment (Mass % Cs) Rb-head

Mineral Name	Head
Beryl	0.62
K-Feldspars	95.4
Muscovite	3.99
Total	100.0





Elemental Deportment (Mass Cs)

Mineral Name	Li-head	Rb-head
Beryl	0.004	0.000
K-Feldspars	0.006	0.035
Muscovite	0.002	0.001
Total	0.013	0.036

Elemental Deportment (Mass % Cs)

Mineral Name	Li-head	Rb-head
Beryl	31.6	0.62
K-Feldspars	49.3	95.4
Muscovite	19.1	3.99
Total	100.0	100.0

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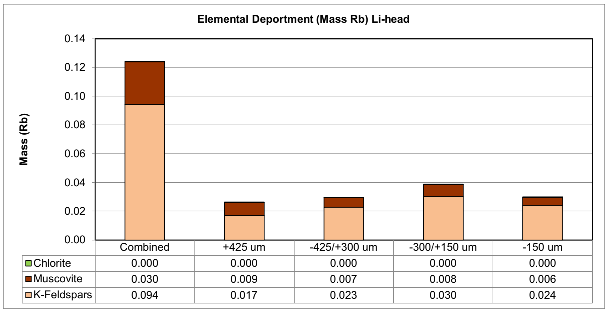
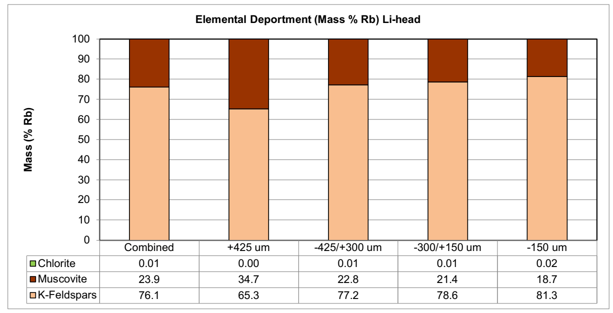
MI5018-SEP23

Rb Deportment



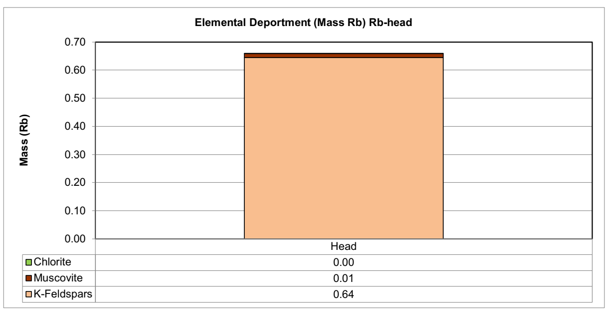


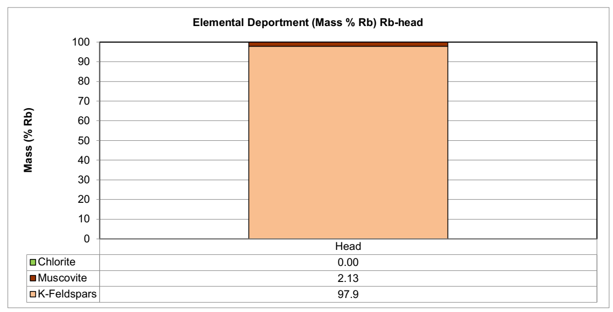
Elemental Deportment (Mass Rb) Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
K-Feldspars	0.094	0.017	0.023	0.030	0.024
Muscovite	0.030	0.009	0.007	0.008	0.006
Chlorite	0.000	0.000	0.000	0.000	0.000
Total	0.124	0.026	0.030	0.039	0.030
Total (% in fraction)	100.0	21.0	23.9	31.2	24.0

Elemental Deportment (Mass % Rb) Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
K-Feldspars	76.1	65.3	77.2	78.6	81.3
Muscovite	23.9	34.7	22.8	21.4	18.7
Chlorite	0.01	0.00	0.01	0.01	0.02
Total	100.0	100.0	100.0	100.0	100.0



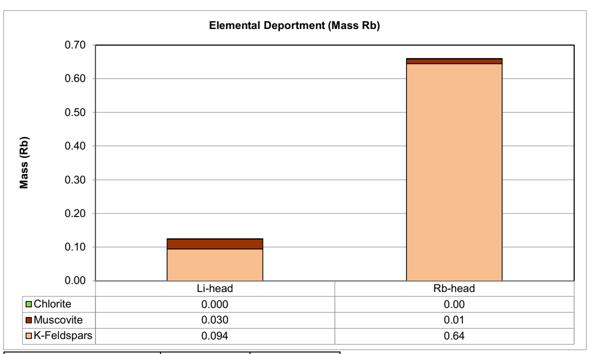


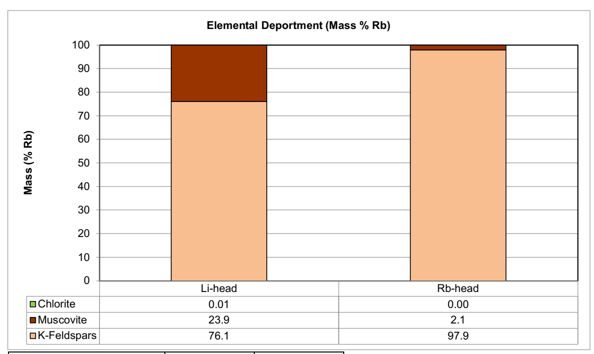
Elemental Deportment (Mass Rb) Rb-head

Mineral Name	Head
K-Feldspars	0.64
Muscovite	0.01
Chlorite	0.00
Total	0.66

Elemental Deportment (Mass % Rb) Rb-head

Mineral Name	Head
K-Feldspars	97.9
Muscovite	2.13
Chlorite	0.00
Total	100.0





Mineral Name	Li-head	Rb-head
K-Feldspars	0.094	0.64
Muscovite	0.030	0.01
Chlorite	0.000	0.00
Total	0.124	0.66

Mineral Name	Li-head	Rb-head
K-Feldspars	76.1	97.9
Muscovite	23.9	2.1
Chlorite	0.01	0.00
Total	100.0	100.0

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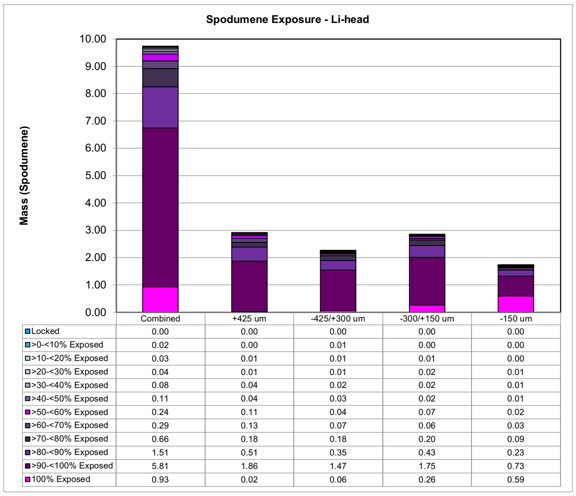
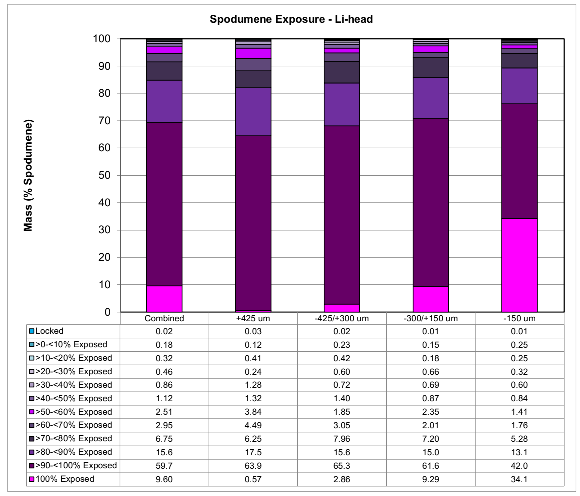
MI5018-SEP23

Spodumene Exposure



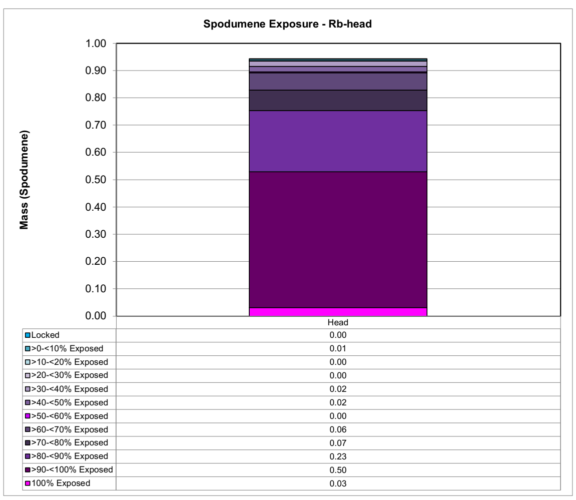


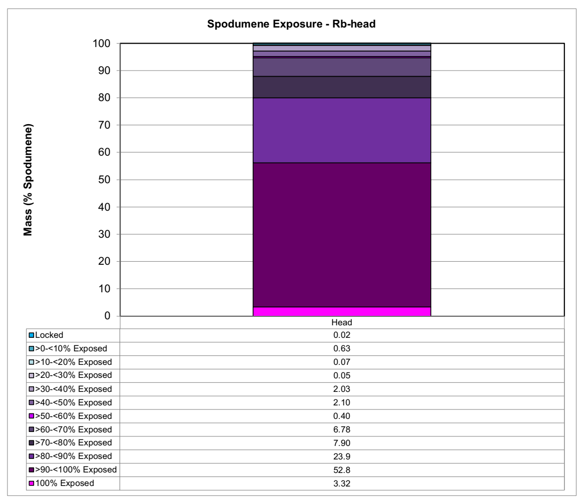
Absolute Mass of Spodumene Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	0.93	0.02	0.06	0.26	0.59
>90-<100% Exposed	5.81	1.86	1.47	1.75	0.73
>80-<90% Exposed	1.51	0.51	0.35	0.43	0.23
>70-<80% Exposed	0.66	0.18	0.18	0.20	0.09
>60-<70% Exposed	0.29	0.13	0.07	0.06	0.03
>50-<60% Exposed	0.24	0.11	0.04	0.07	0.02
>40-<50% Exposed	0.11	0.04	0.03	0.02	0.01
>30-<40% Exposed	0.08	0.04	0.02	0.02	0.01
>20-<30% Exposed	0.04	0.01	0.01	0.02	0.01
>10-<20% Exposed	0.03	0.01	0.01	0.01	0.00
>0-<10% Exposed	0.02	0.00	0.01	0.00	0.00
Locked	0.00	0.00	0.00	0.00	0.00
Total	9.73	2.91	2.26	2.84	1.73
Total (% in fraction)	100.0	29.9	23.2	29.2	17.7

Normalized Mass of Spodumene Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	9.60	0.57	2.86	9.29	34.1
>90-<100% Exposed	59.7	63.9	65.3	61.6	42.0
>80-<90% Exposed	15.6	17.5	15.6	15.0	13.1
>70-<80% Exposed	6.75	6.25	7.96	7.20	5.28
>60-<70% Exposed	2.95	4.49	3.05	2.01	1.76
>50-<60% Exposed	2.51	3.84	1.85	2.35	1.41
>40-<50% Exposed	1.12	1.32	1.40	0.87	0.84
>30-<40% Exposed	0.86	1.28	0.72	0.69	0.60
>20-<30% Exposed	0.46	0.24	0.60	0.66	0.32
>10-<20% Exposed	0.32	0.41	0.42	0.18	0.25
>0-<10% Exposed	0.18	0.12	0.23	0.15	0.25
Locked	0.02	0.03	0.02	0.01	0.01
Total	100.0	100.0	100.0	100.0	100.0
>20% Exposed	99.5	99.4	99.3	99.7	99.5





Absolute Mass of Spodumene Across Fraction Rb-head

Mineral Name	Head
100% Exposed	0.03
>90-<100% Exposed	0.50
>80-<90% Exposed	0.23
>70-<80% Exposed	0.07
>60-<70% Exposed	0.06
>50-<60% Exposed	0.00
>40-<50% Exposed	0.02
>30-<40% Exposed	0.02
>20-<30% Exposed	0.00
>10-<20% Exposed	0.00
>0-<10% Exposed	0.01
Locked	0.00
Total	0.94

Normalized Mass of Spodumene Across Fraction Rb-head

Mineral Name	Head
100% Exposed	3.32
>90-<100% Exposed	52.8
>80-<90% Exposed	23.9
>70-<80% Exposed	7.90
>60-<70% Exposed	6.78
>50-<60% Exposed	0.40
>40-<50% Exposed	2.10
>30-<40% Exposed	2.03
>20-<30% Exposed	0.05
>10-<20% Exposed	0.07
>0-<10% Exposed	0.63
Locked	0.02
Total	100.0
>20% Exposed	99.3

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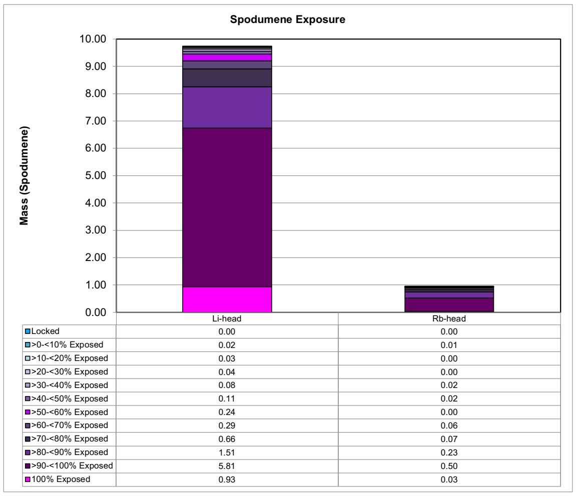
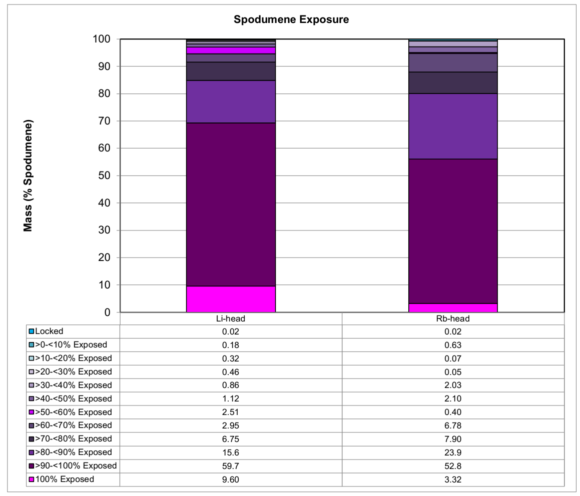
MI5018-SEP23

Spodumene Exposure





Absolute Mass of Spodumene Across Samples

Mineral Name	Li-head	Rb-head
100% Exposed	0.93	0.03
>90-<100% Exposed	5.81	0.50
>80-<90% Exposed	1.51	0.23
>70-<80% Exposed	0.66	0.07
>60-<70% Exposed	0.29	0.06
>50-<60% Exposed	0.24	0.00
>40-<50% Exposed	0.11	0.02
>30-<40% Exposed	0.08	0.02
>20-<30% Exposed	0.04	0.00
>10-<20% Exposed	0.03	0.00
>0-<10% Exposed	0.02	0.01
Locked	0.00	0.00
Total	9.73	0.94

Normalized Mass of Spodumene Across Samples

Mineral Name	Li-head	Rb-head
100% Exposed	9.60	3.32
>90-<100% Exposed	59.7	52.8
>80-<90% Exposed	15.6	23.9
>70-<80% Exposed	6.75	7.90
>60-<70% Exposed	2.95	6.78
>50-<60% Exposed	2.51	0.40
>40-<50% Exposed	1.12	2.10
>30-<40% Exposed	0.86	2.03
>20-<30% Exposed	0.46	0.05
>10-<20% Exposed	0.32	0.07
>0-<10% Exposed	0.18	0.63
Locked	0.02	0.02
Total	100.0	100.0
>20% Exposed	99.5	99.3

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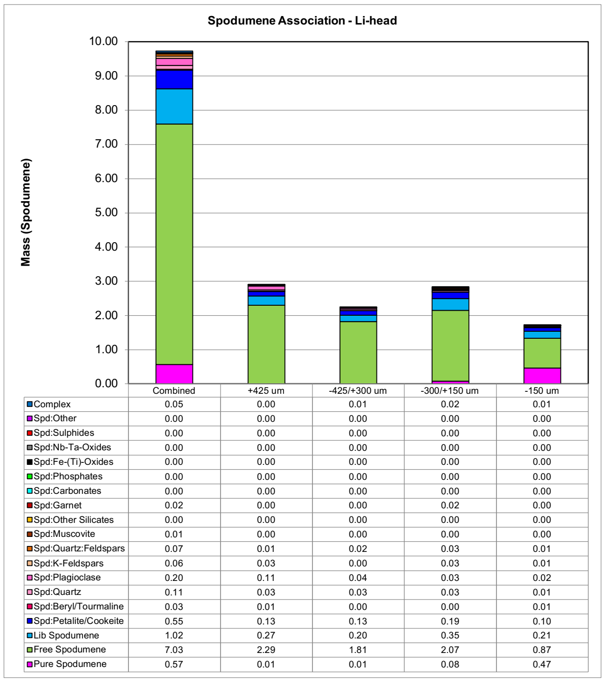
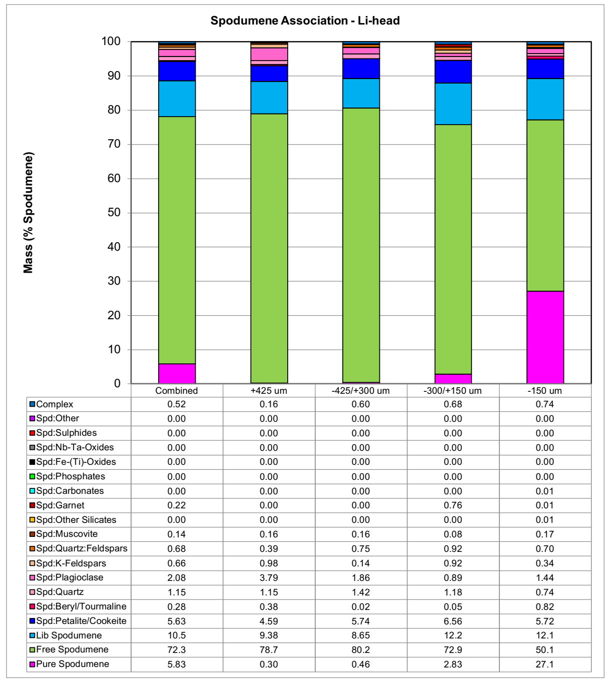
CALR-19977-01
MI5018-SEP23

Spodumene Association





Absolute Mass of Spodumene Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure Spodumene	0.57	0.01	0.01	0.08	0.47
Free Spodumene	7.03	2.29	1.81	2.07	0.87
Lib Spodumene	1.02	0.27	0.20	0.35	0.21
Spd:Petalite/Cookeite	0.55	0.13	0.13	0.19	0.10
Spd:Beryl/Tourmaline	0.03	0.01	0.00	0.00	0.01
Spd:Quartz	0.11	0.03	0.03	0.03	0.01
Spd:Plagioclase	0.20	0.11	0.04	0.03	0.02
Spd:K-Feldspars	0.06	0.03	0.00	0.03	0.01
Spd:Quartz:Feldspars	0.07	0.01	0.02	0.03	0.01
Spd:Muscovite	0.01	0.00	0.00	0.00	0.00
Spd:Other Silicates	0.00	0.00	0.00	0.00	0.00
Spd:Garnet	0.02	0.00	0.00	0.02	0.00
Spd:Carbonates	0.00	0.00	0.00	0.00	0.00
Spd:Phosphates	0.00	0.00	0.00	0.00	0.00
Spd:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
Spd:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
Spd:Sulphides	0.00	0.00	0.00	0.00	0.00
Spd:Other	0.00	0.00	0.00	0.00	0.00
Complex	0.05	0.00	0.01	0.02	0.01
Total	9.73	2.91	2.26	2.84	1.73
Total (% in fraction)	100.0	29.9	23.2	29.2	17.7

Normalized Mass of Spodumene Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure Spodumene	5.83	0.30	0.46	2.83	27.1
Free Spodumene	72.3	78.7	80.2	72.9	50.1
Lib Spodumene	10.5	9.38	8.65	12.2	12.1
Spd:Petalite/Cookeite	5.63	4.59	5.74	6.56	5.72
Spd:Beryl/Tourmaline	0.28	0.38	0.02	0.05	0.82
Spd:Quartz	1.15	1.15	1.42	1.18	0.74
Spd:Plagioclase	2.08	3.79	1.86	0.89	1.44
Spd:K-Feldspars	0.66	0.98	0.14	0.92	0.34
Spd:Quartz:Feldspars	0.68	0.39	0.75	0.92	0.70
Spd:Muscovite	0.14	0.16	0.16	0.08	0.17
Spd:Other Silicates	0.00	0.00	0.00	0.00	0.01
Spd:Garnet	0.22	0.00	0.00	0.76	0.01
Spd:Carbonates	0.00	0.00	0.00	0.00	0.01
Spd:Phosphates	0.00	0.00	0.00	0.00	0.00
Spd:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
Spd:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
Spd:Sulphides	0.00	0.00	0.00	0.00	0.00
Spd:Other	0.00	0.00	0.00	0.00	0.00
Complex	0.52	0.16	0.60	0.68	0.74
Total	100.0	100.0	100.0	100.0	100.0
Total Liberated	88.6	88.4	89.3	88.0	89.3
	94	93	95	95	95

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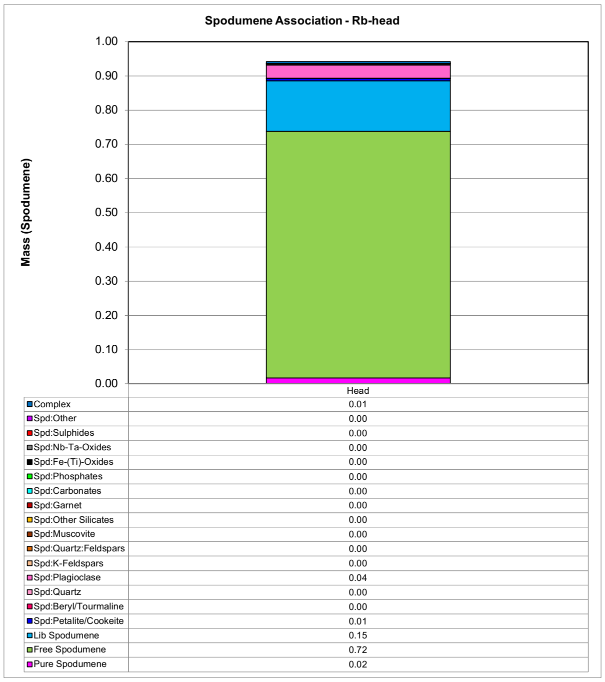
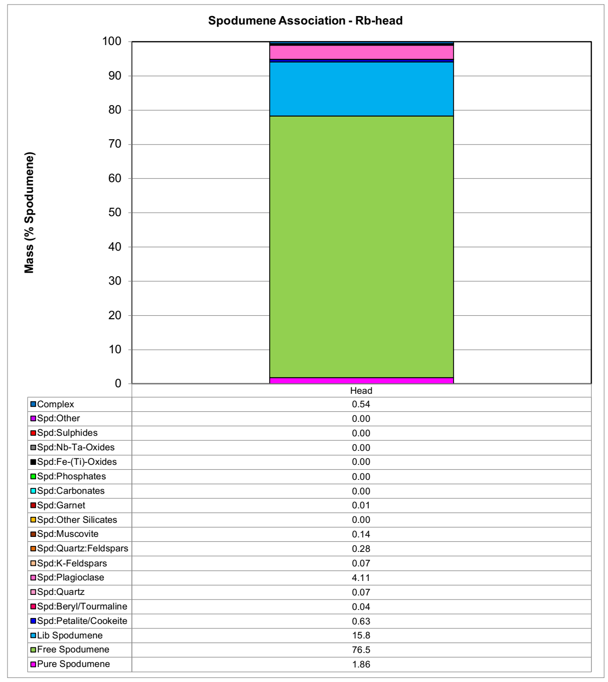
Spodumene Association





Absolute Mass of Spodumene Across Fraction Rb-head

Mineral Name	Head
Pure Spodumene	0.02
Free Spodumene	0.72
Lib Spodumene	0.15
Spd:Petalite/Cookeite	0.01
Spd:Beryl/Tourmaline	0.00
Spd:Quartz	0.00
Spd:Plagioclase	0.04
Spd:K-Feldspars	0.00
Spd:Quartz:Feldspars	0.00
Spd:Muscovite	0.00
Spd:Other Silicates	0.00
Spd:Garnet	0.00
Spd:Carbonates	0.00
Spd:Phosphates	0.00
Spd:Fe-(Ti)-Oxides	0.00
Spd:Nb-Ta-Oxides	0.00
Spd:Sulphides	0.00
Spd:Other	0.00
Complex	0.01
Total	0.94

Normalized Mass of Spodumene Across Fraction Rb-head

Mineral Name	Head
Pure Spodumene	1.86
Free Spodumene	76.5
Lib Spodumene	15.8
Spd:Petalite/Cookeite	0.63
Spd:Beryl/Tourmaline	0.04
Spd:Quartz	0.07
Spd:Plagioclase	4.11
Spd:K-Feldspars	0.07
Spd:Quartz:Feldspars	0.28
Spd:Muscovite	0.14
Spd:Other Silicates	0.00
Spd:Garnet	0.01
Spd:Carbonates	0.00
Spd:Phosphates	0.00
Spd:Fe-(Ti)-Oxides	0.00
Spd:Nb-Ta-Oxides	0.00
Spd:Sulphides	0.00
Spd:Other	0.00
Complex	0.54
Total	100.0
Total Liberated	94.1

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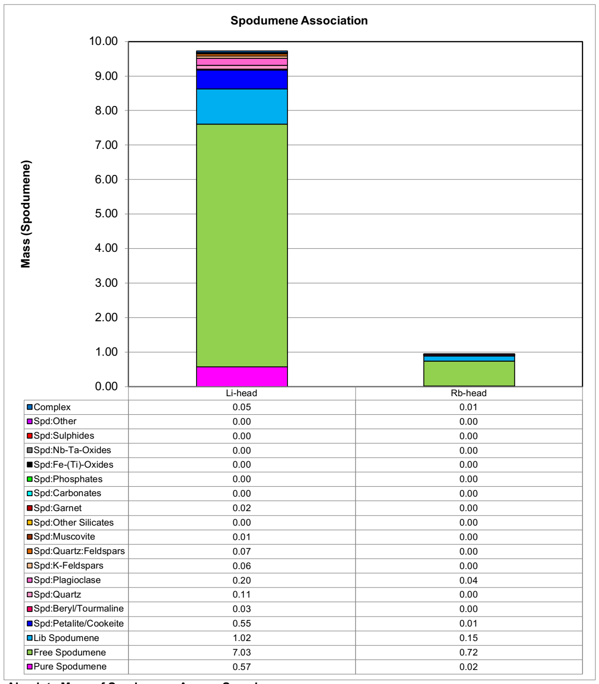
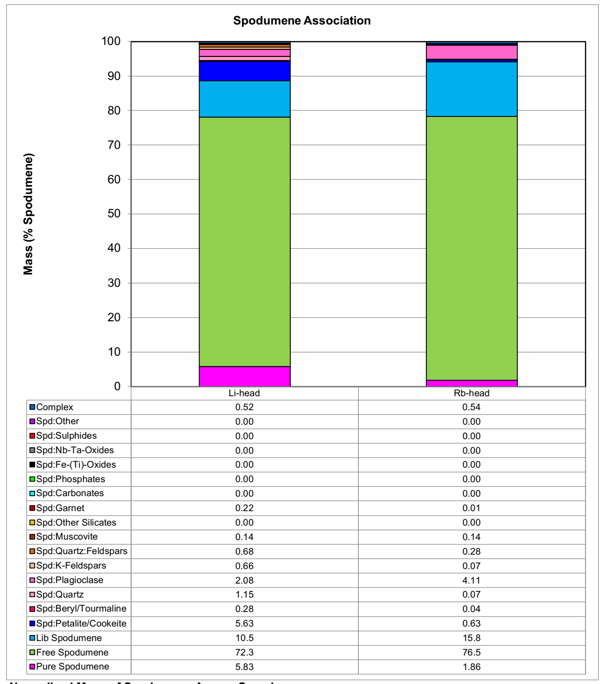
Spodumene Association





Absolute Mass of Spodumene Across Samples

Mineral Name	Li-head	Rb-head
Pure Spodumene	0.57	0.02
Free Spodumene	7.03	0.72
Lib Spodumene	1.02	0.15
Spd:Petalite/Cookeite	0.55	0.01
Spd:Beryl/Tourmaline	0.03	0.00
Spd:Quartz	0.11	0.00
Spd:Plagioclase	0.20	0.04
Spd:K-Feldspars	0.06	0.00
Spd:Quartz:Feldspars	0.07	0.00
Spd:Muscovite	0.01	0.00
Spd:Other Silicates	0.00	0.00
Spd:Garnet	0.02	0.00
Spd:Carbonates	0.00	0.00
Spd:Phosphates	0.00	0.00
Spd:Fe-(Ti)-Oxides	0.00	0.00
Spd:Nb-Ta-Oxides	0.00	0.00
Spd:Sulphides	0.00	0.00
Spd:Other	0.00	0.00
Complex	0.05	0.01
Total	9.73	0.94

Normalized Mass of Spodumene Across Samples

Mineral Name	Li-head	Rb-head
Pure Spodumene	5.83	1.86
Free Spodumene	72.3	76.5
Lib Spodumene	10.5	15.8
Spd:Petalite/Cookeite	5.63	0.63
Spd:Beryl/Tourmaline	0.28	0.04
Spd:Quartz	1.15	0.07
Spd:Plagioclase	2.08	4.11
Spd:K-Feldspars	0.66	0.07
Spd:Quartz:Feldspars	0.68	0.28
Spd:Muscovite	0.14	0.14
Spd:Other Silicates	0.00	0.00
Spd:Garnet	0.22	0.01
Spd:Carbonates	0.00	0.00
Spd:Phosphates	0.00	0.00
Spd:Fe-(Ti)-Oxides	0.00	0.00
Spd:Nb-Ta-Oxides	0.00	0.00
Spd:Sulphides	0.00	0.00
Spd:Other	0.00	0.00
Complex	0.52	0.54
Total	100.0	100.0
Total Liberated	88.6	94.1

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Image Grid of Spodumene Exposure

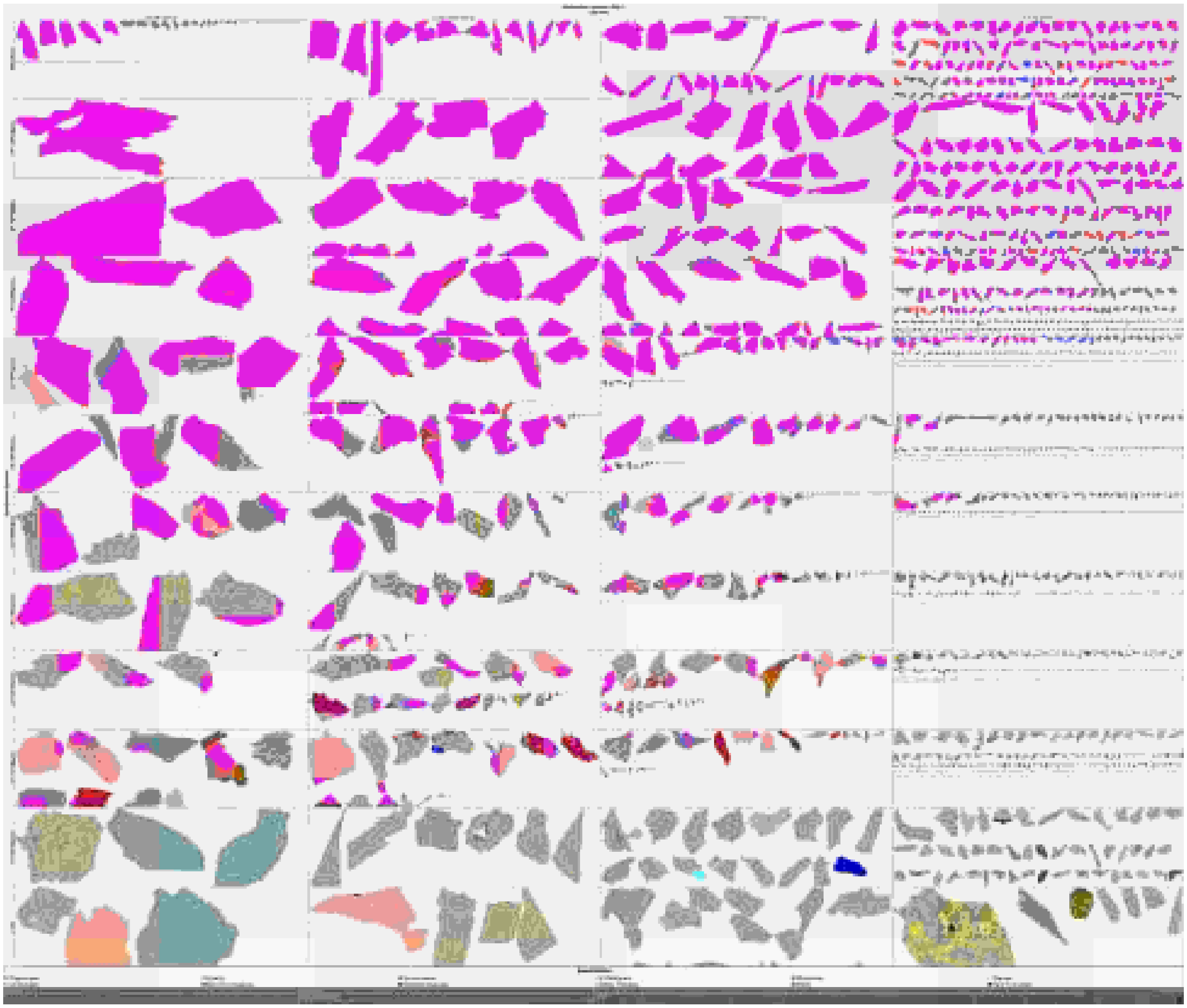

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Image Grid of Spodumene Exposure

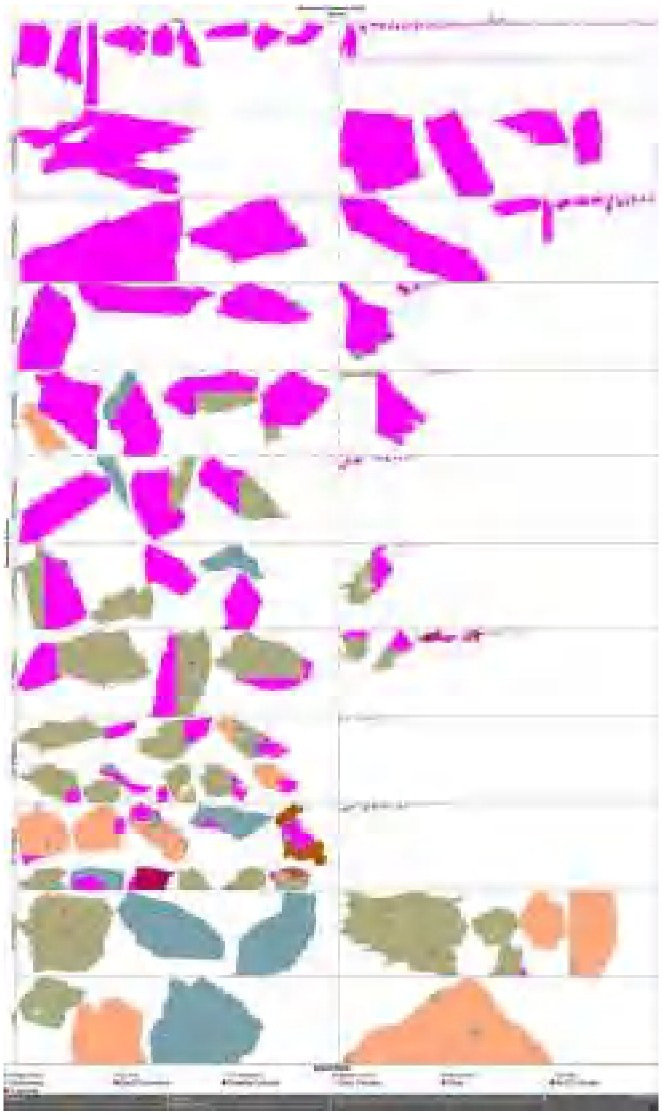

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Image Grid of Spodumene Association

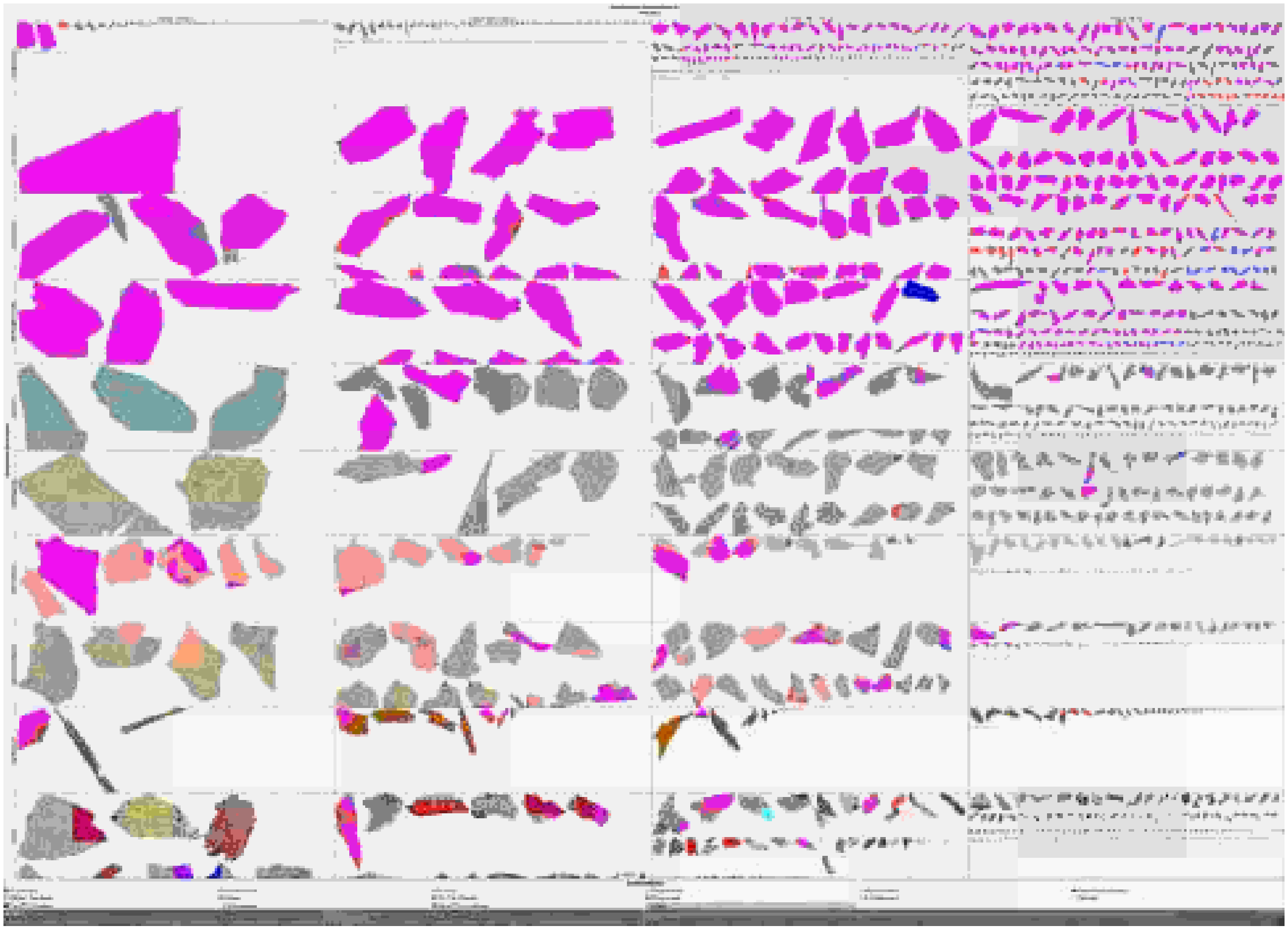

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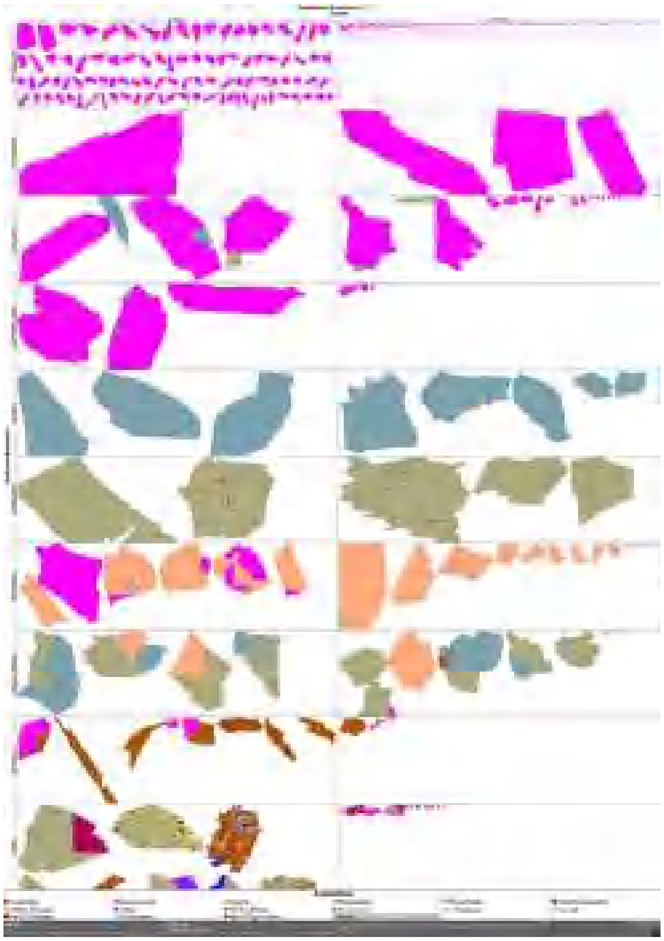
Image Grid of Spodumene Association



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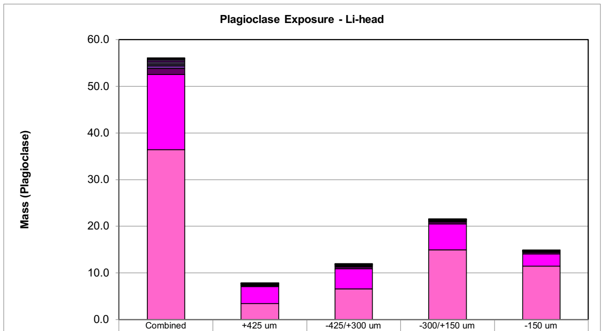
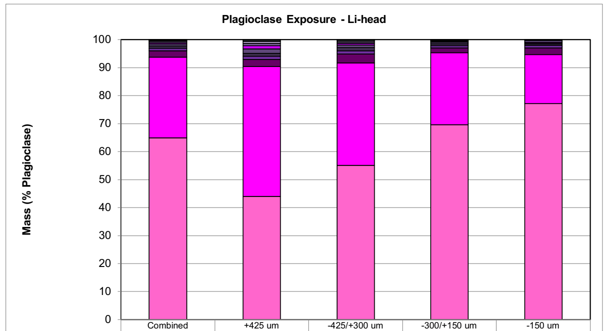
Plagioclase Exposure



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Locked	0.01	0.00	0.00	0.00	0.00
>0-<10% Exposed	0.10	0.03	0.03	0.03	0.02
>10-<20% Exposed	0.09	0.02	0.03	0.02	0.02
>20-<30% Exposed	0.14	0.05	0.05	0.03	0.02
>30-<40% Exposed	0.26	0.07	0.06	0.05	0.09
>40-<50% Exposed	0.25	0.09	0.07	0.05	0.04
>50-<60% Exposed	0.40	0.12	0.10	0.13	0.05
>60-<70% Exposed	0.45	0.08	0.14	0.15	0.08
>70-<80% Exposed	0.51	0.09	0.13	0.16	0.13
>80-<90% Exposed	1.32	0.19	0.39	0.39	0.36
>90-<100% Exposed	16.1	3.63	4.36	5.53	2.60
100% Exposed	36.4	3.43	6.56	15.0	11.4



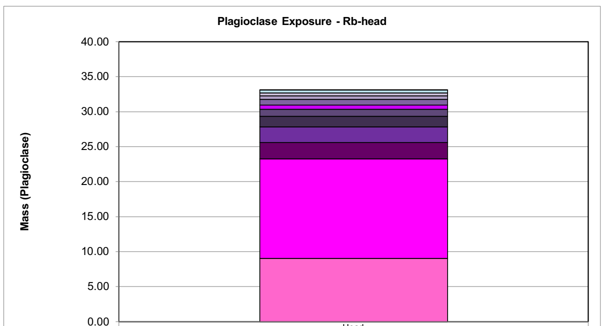
	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Locked	0.02	0.03	0.02	0.02	0.01
>0-<10% Exposed	0.19	0.33	0.24	0.15	0.11
>10-<20% Exposed	0.16	0.32	0.22	0.11	0.11
>20-<30% Exposed	0.25	0.62	0.38	0.12	0.13
>30-<40% Exposed	0.46	0.88	0.48	0.22	0.59
>40-<50% Exposed	0.45	1.16	0.60	0.24	0.26
>50-<60% Exposed	0.72	1.60	0.85	0.60	0.33
>60-<70% Exposed	0.80	0.97	1.20	0.69	0.54
>70-<80% Exposed	0.92	1.20	1.10	0.75	0.86
>80-<90% Exposed	2.36	2.42	3.26	1.80	2.41
>90-<100% Exposed	28.7	46.5	36.6	25.7	17.5
100% Exposed	64.9	44.0	55.1	69.6	77.1

Absolute Mass of Plagioclase Across Fraction Li-hea

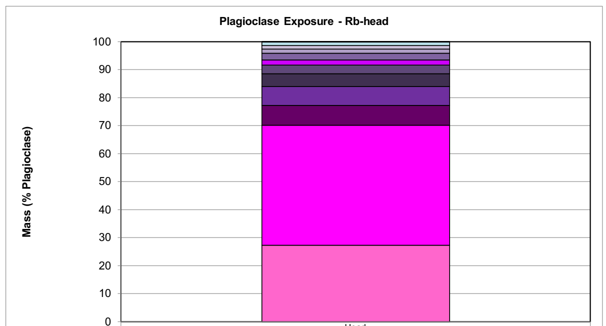
Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	36.4	3.43	6.56	15.0	11.4
>90-<100% Exposed	16.1	3.63	4.36	5.53	2.60
>80-<90% Exposed	1.32	0.19	0.39	0.39	0.36
>70-<80% Exposed	0.51	0.09	0.13	0.16	0.13
>60-<70% Exposed	0.45	0.08	0.14	0.15	0.08
>50-<60% Exposed	0.40	0.12	0.10	0.13	0.05
>40-<50% Exposed	0.25	0.09	0.07	0.05	0.04
>30-<40% Exposed	0.26	0.07	0.06	0.05	0.09
>20-<30% Exposed	0.14	0.05	0.05	0.03	0.02
>10-<20% Exposed	0.09	0.02	0.03	0.02	0.02
>0-<10% Exposed	0.10	0.03	0.03	0.03	0.02
Locked	0.01	0.00	0.00	0.00	0.00
Total	56.1	7.80	11.9	21.5	14.8
Total (% in fraction)	100.0	13.9	21.2	38.4	26.5

Normalized Mass of Plagioclase Across Fraction Li-hea

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	64.9	44.0	55.1	69.6	77.1
>90-<100% Exposed	28.7	46.5	36.6	25.7	17.5
>80-<90% Exposed	2.36	2.42	3.26	1.80	2.41
>70-<80% Exposed	0.92	1.20	1.10	0.75	0.86
>60-<70% Exposed	0.80	0.97	1.20	0.69	0.54
>50-<60% Exposed	0.72	1.60	0.85	0.60	0.33
>40-<50% Exposed	0.45	1.16	0.60	0.24	0.26
>30-<40% Exposed	0.46	0.88	0.48	0.22	0.59
>20-<30% Exposed	0.25	0.62	0.38	0.12	0.13
>10-<20% Exposed	0.16	0.32	0.22	0.11	0.11
>0-<10% Exposed	0.19	0.33	0.24	0.15	0.11
Locked	0.02	0.03	0.02	0.02	0.01
Total	100.0	100.0	100.0	100.0	100.0
>20% Exposed	99.6	99.3	99.5	99.7	99.8



	Head
Locked	0.03
>0-<10% Exposed	0.43
>10-<20% Exposed	0.41
>20-<30% Exposed	0.50
>30-<40% Exposed	0.80
>40-<50% Exposed	0.60
>50-<60% Exposed	1.02
>60-<70% Exposed	1.50
>70-<80% Exposed	2.26
>80-<90% Exposed	2.33
>90-<100% Exposed	14.2
100% Exposed	9.03



	Head
Locked	0.08
>0-<10% Exposed	1.28
>10-<20% Exposed	1.25
>20-<30% Exposed	1.50
>30-<40% Exposed	2.42
>40-<50% Exposed	1.81
>50-<60% Exposed	3.08
>60-<70% Exposed	4.53
>70-<80% Exposed	6.81
>80-<90% Exposed	7.04
>90-<100% Exposed	42.9
100% Exposed	27.3

Absolute Mass of Plagioclase Across Fraction Rb-hea

Mineral Name	Head
100% Exposed	9.03
>90-<100% Exposed	14.2
>80-<90% Exposed	2.33
>70-<80% Exposed	2.26
>60-<70% Exposed	1.50
>50-<60% Exposed	1.02
>40-<50% Exposed	0.60
>30-<40% Exposed	0.80
>20-<30% Exposed	0.50
>10-<20% Exposed	0.41
>0-<10% Exposed	0.43
Locked	0.03
Total	33.1

Normalized Mass of Plagioclase Across Fraction Rb-hea

Mineral Name	Head
100% Exposed	27.3
>90-<100% Exposed	42.9
>80-<90% Exposed	7.04
>70-<80% Exposed	6.81
>60-<70% Exposed	4.53
>50-<60% Exposed	3.08
>40-<50% Exposed	1.81
>30-<40% Exposed	2.42
>20-<30% Exposed	1.50
>10-<20% Exposed	1.25
>0-<10% Exposed	1.28
Locked	0.08
Total	100.0
>20% Exposed	97.4

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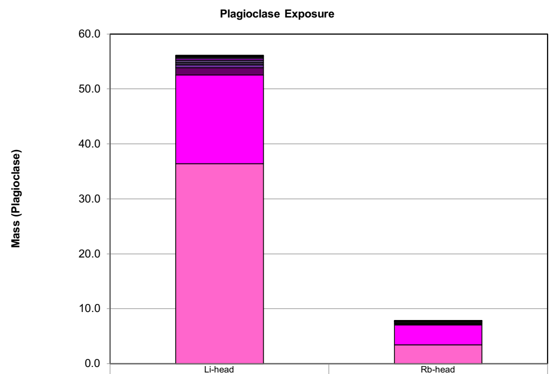
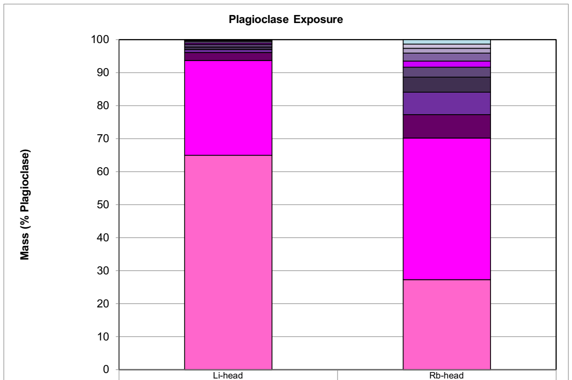
Plagioclase Exposure



	Li-head	Rb-head
Locked	0.01	0.00
>0-<10% Exposed	0.10	0.03
>10-<20% Exposed	0.09	0.02
>20-<30% Exposed	0.14	0.05
>30-<40% Exposed	0.26	0.07
>40-<50% Exposed	0.25	0.09
>50-<60% Exposed	0.40	0.12
>60-<70% Exposed	0.45	0.08
>70-<80% Exposed	0.51	0.09
>80-<90% Exposed	1.32	0.19
>90-<100% Exposed	16.1	3.63
100% Exposed	36.4	3.43

100% Exposed	36.4	3.43
>90-<100% Exposed	16.1	3.63
>80-<90% Exposed	1.32	0.19
>70-<80% Exposed	0.51	0.09
>60-<70% Exposed	0.45	0.08
>50-<60% Exposed	0.40	0.12
>40-<50% Exposed	0.25	0.09
>30-<40% Exposed	0.26	0.07
>20-<30% Exposed	0.14	0.05
>10-<20% Exposed	0.09	0.02
>0-<10% Exposed	0.10	0.03
Locked	0.01	0.00
Total	56.1	7.80



	Li-head	Rb-head
Locked	0.02	0.08
>0-<10% Exposed	0.19	1.28
>10-<20% Exposed	0.16	1.25
>20-<30% Exposed	0.25	1.50
>30-<40% Exposed	0.46	2.42
>40-<50% Exposed	0.45	1.81
>50-<60% Exposed	0.72	3.08
>60-<70% Exposed	0.80	4.53
>70-<80% Exposed	0.92	6.81
>80-<90% Exposed	2.36	7.04
>90-<100% Exposed	28.7	42.9
100% Exposed	64.9	27.3

100% Exposed	64.9	27.3
>90-<100% Exposed	28.7	42.9
>80-<90% Exposed	2.36	7.04
>70-<80% Exposed	0.92	6.81
>60-<70% Exposed	0.80	4.53
>50-<60% Exposed	0.72	3.08
>40-<50% Exposed	0.45	1.81
>30-<40% Exposed	0.46	2.42
>20-<30% Exposed	0.25	1.50
>10-<20% Exposed	0.16	1.25
>0-<10% Exposed	0.19	1.28
Locked	0.02	0.08
Total	100.0	100.0
>20% Exposed	99.6	97.4

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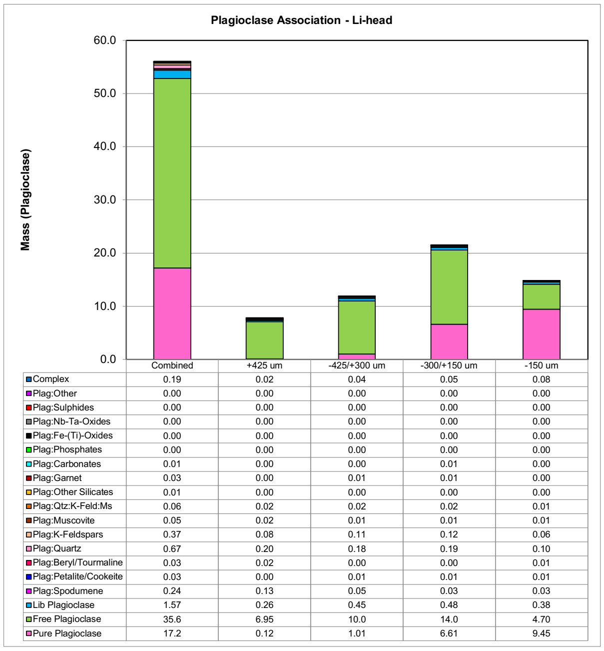
Plagioclase Association



Absolute Mass of Plagioclase Across Fraction Li-head

Normalized Mass of Plagioclase Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure Plagioclase	17.2	0.12	1.01	6.61	9.45
Free Plagioclase	35.6	6.95	10.0	14.0	4.70
Lib Plagioclase	1.57	0.26	0.45	0.48	0.38
Plag:Spodumene	0.24	0.13	0.05	0.03	0.03
Plag:Petalite/Cookeite	0.03	0.00	0.01	0.01	0.01
Plag:Beryl/Tourmaline	0.03	0.02	0.00	0.00	0.01
Plag:Quartz	0.67	0.20	0.18	0.19	0.10
Plag:K-Feldspars	0.37	0.08	0.11	0.12	0.06
Plag:Muscovite	0.05	0.02	0.01	0.01	0.01
Plag:Qtz:K-Feld:Ms	0.06	0.02	0.02	0.02	0.01
Plag:Other Silicates	0.01	0.00	0.00	0.00	0.00
Plag:Garnet	0.03	0.00	0.01	0.01	0.00
Plag:Carbonates	0.01	0.00	0.00	0.01	0.00
Plag:Phosphates	0.00	0.00	0.00	0.00	0.00
Plag:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
Plag:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
Plag:Sulphides	0.00	0.00	0.00	0.00	0.00
Plag:Other	0.00	0.00	0.00	0.00	0.00
Complex	0.19	0.02	0.04	0.05	0.08
Total	56.1	7.80	11.9	21.5	14.8
Total (% in fraction)	100.0	13.9	21.2	38.4	26.5

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure Plagioclase	30.7	1.55	8.52	30.7	63.7
Free Plagioclase	63.5	89.0	84.0	64.9	31.7
Lib Plagioclase	2.80	3.31	3.80	2.23	2.56
Plag:Spodumene	0.43	1.63	0.46	0.15	0.19
Plag:Petalite/Cookeite	0.06	0.00	0.06	0.06	0.08
Plag:Beryl/Tourmaline	0.05	0.23	0.00	0.02	0.04
Plag:Quartz	1.19	2.50	1.55	0.90	0.66
Plag:K-Feldspars	0.66	1.02	0.88	0.56	0.42
Plag:Muscovite	0.09	0.21	0.06	0.07	0.09
Plag:Qtz:K-Feld:Ms	0.10	0.21	0.14	0.07	0.05
Plag:Other Silicates	0.01	0.00	0.00	0.01	0.02
Plag:Garnet	0.05	0.06	0.11	0.04	0.01
Plag:Carbonates	0.02	0.00	0.00	0.04	0.02
Plag:Phosphates	0.00	0.00	0.00	0.00	0.00
Plag:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
Plag:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
Plag:Sulphides	0.00	0.00	0.00	0.00	0.00
Plag:Other	0.00	0.00	0.00	0.00	0.01
Complex	0.33	0.28	0.38	0.21	0.51
Total	100.0	100.0	100.0	100.0	100.0
Total Liberated	97.0	93.9	96.4	97.9	97.9

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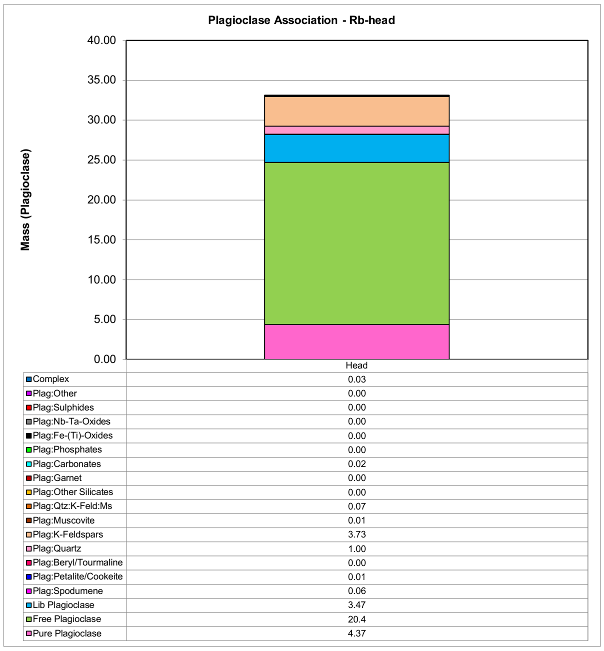
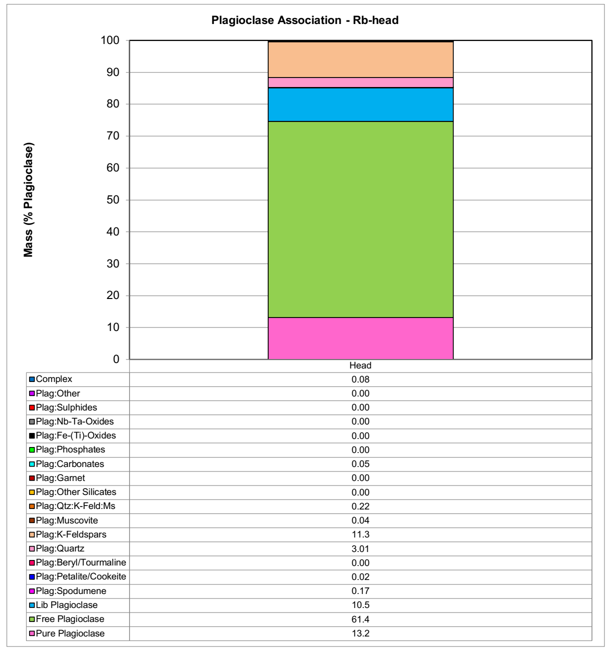
Plagioclase Association





Absolute Mass of Plagioclase Across Fraction Rb-head

Mineral Name	Head
Pure Plagioclase	4.37
Free Plagioclase	20.4
Lib Plagioclase	3.47
Plag:Spodumene	0.06
Plag:Petalite/Cookeite	0.01
Plag:Beryl/Tourmaline	0.00
Plag:Quartz	1.00
Plag:K-Feldspars	3.73
Plag:Muscovite	0.01
Plag:Qtz:K-Feld:Ms	0.07
Plag:Other Silicates	0.00
Plag:Garnet	0.00
Plag:Carbonates	0.02
Plag:Phosphates	0.00
Plag:Fe-(Ti)-Oxides	0.00
Plag:Nb-Ta-Oxides	0.00
Plag:Sulphides	0.00
Plag:Other	0.00
Complex	0.03
Total	33.1

Normalized Mass of Plagioclase Across Fraction Rb-head

Mineral Name	Head
Pure Plagioclase	13.2
Free Plagioclase	61.4
Lib Plagioclase	10.5
Plag:Spodumene	0.17
Plag:Petalite/Cookeite	0.02
Plag:Beryl/Tourmaline	0.00
Plag:Quartz	3.01
Plag:K-Feldspars	11.3
Plag:Muscovite	0.04
Plag:Qtz:K-Feld:Ms	0.22
Plag:Other Silicates	0.00
Plag:Garnet	0.00
Plag:Carbonates	0.05
Plag:Phosphates	0.00
Plag:Fe-(Ti)-Oxides	0.00
Plag:Nb-Ta-Oxides	0.00
Plag:Sulphides	0.00
Plag:Other	0.00
Complex	0.08
Total	100.0
Total Liberated	85.1

85

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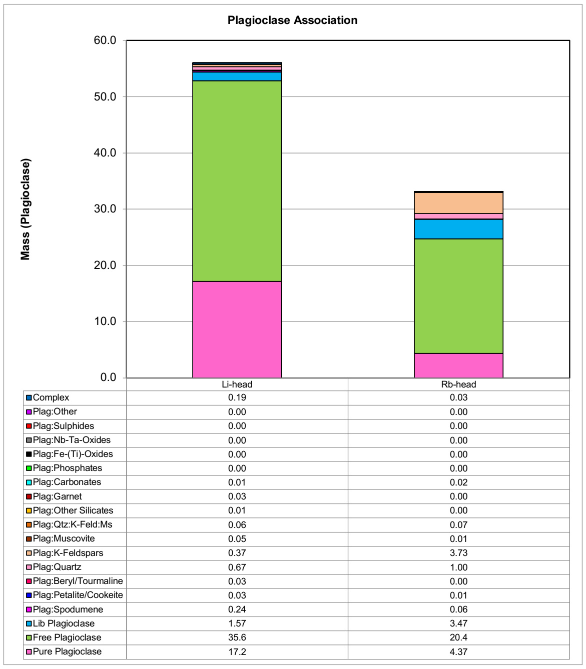
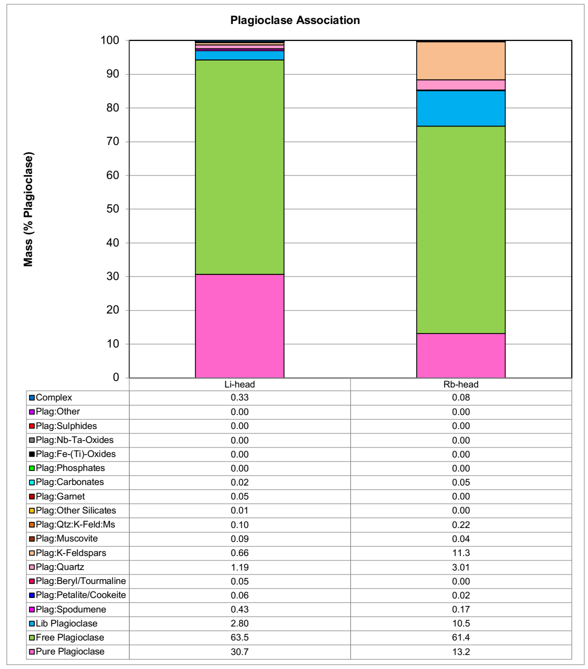
Plagioclase Association





Mineral Name	Li-head	Rb-head
Pure Plagioclase	17.2	4.37
Free Plagioclase	35.6	20.4
Lib Plagioclase	1.57	3.47
Plag:Spodumene	0.24	0.06
Plag:Petalite/Cookeite	0.03	0.01
Plag:Beryl/Tourmaline	0.03	0.00
Plag:Quartz	0.67	1.00
Plag:K-Feldspars	0.37	3.73
Plag:Muscovite	0.05	0.01
Plag:Qtz:K-Feld:Ms	0.06	0.07
Plag:Other Silicates	0.01	0.00
Plag:Garnet	0.03	0.00
Plag:Carbonates	0.01	0.02
Plag:Phosphates	0.00	0.00
Plag:Fe-(Ti)-Oxides	0.00	0.00
Plag:Nb-Ta-Oxides	0.00	0.00
Plag:Sulphides	0.00	0.00
Plag:Other	0.00	0.00
Complex	0.19	0.03
Total	56.1	33.1

Mineral Name	Li-head	Rb-head
Pure Plagioclase	30.7	13.2
Free Plagioclase	63.5	61.4
Lib Plagioclase	2.80	10.5
Plag:Spodumene	0.43	0.17
Plag:Petalite/Cookeite	0.06	0.02
Plag:Beryl/Tourmaline	0.05	0.00
Plag:Quartz	1.19	3.01
Plag:K-Feldspars	0.66	11.3
Plag:Muscovite	0.09	0.04
Plag:Qtz:K-Feld:Ms	0.10	0.22
Plag:Other Silicates	0.01	0.00
Plag:Garnet	0.05	0.00
Plag:Carbonates	0.02	0.05
Plag:Phosphates	0.00	0.00
Plag:Fe-(Ti)-Oxides	0.00	0.00
Plag:Nb-Ta-Oxides	0.00	0.00
Plag:Sulphides	0.00	0.00
Plag:Other	0.00	0.00
Complex	0.33	0.08
Total	100.0	100.0
Total Liberated	97.0	85.1

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Image Grid of Plagioclase Exposure

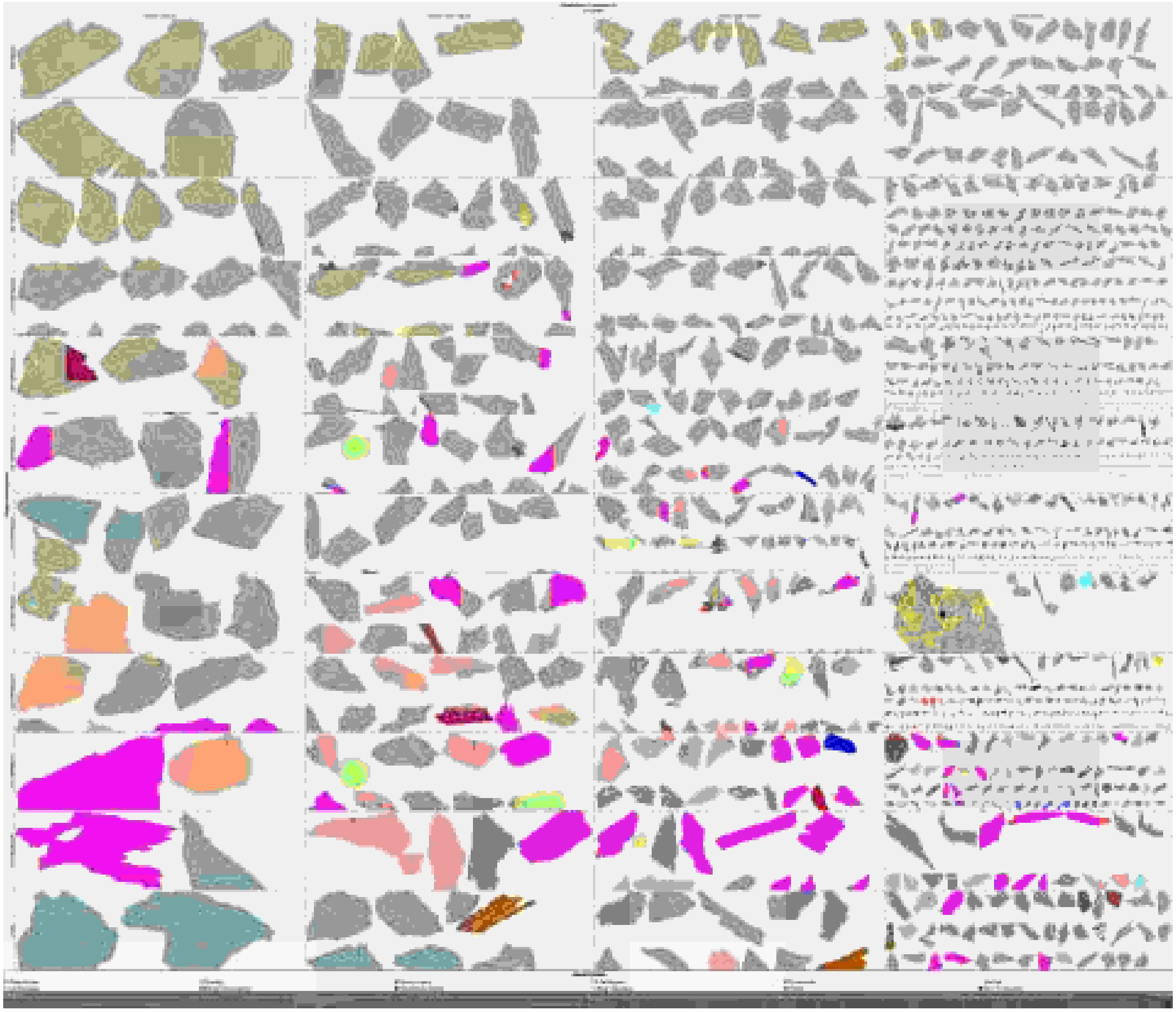

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Image Grid of Plagioclase Exposure

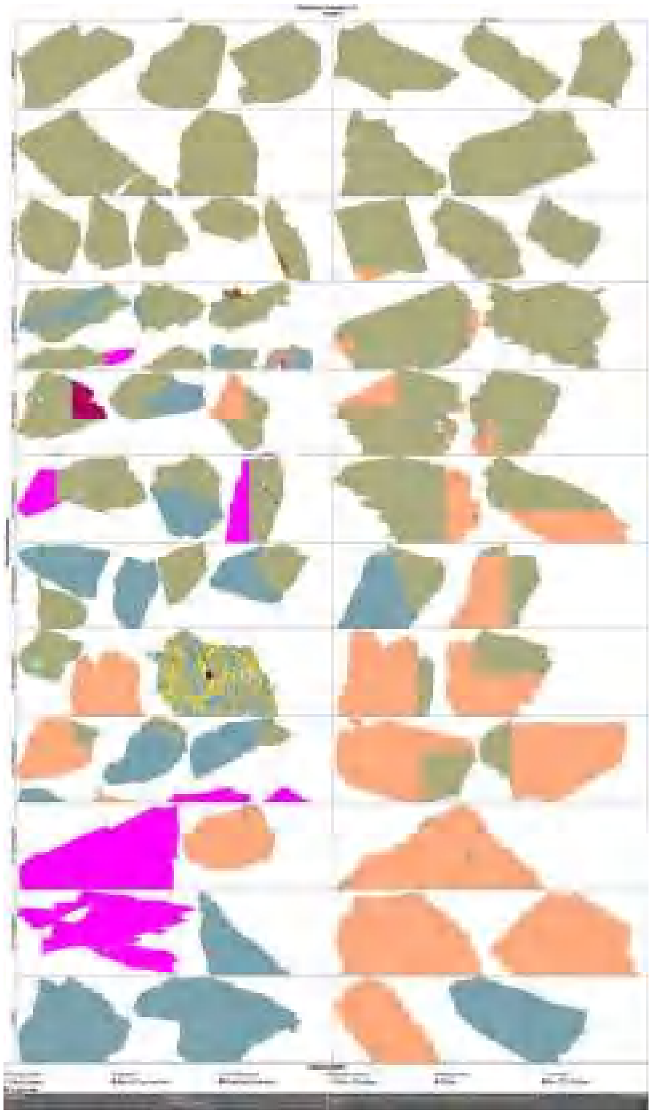

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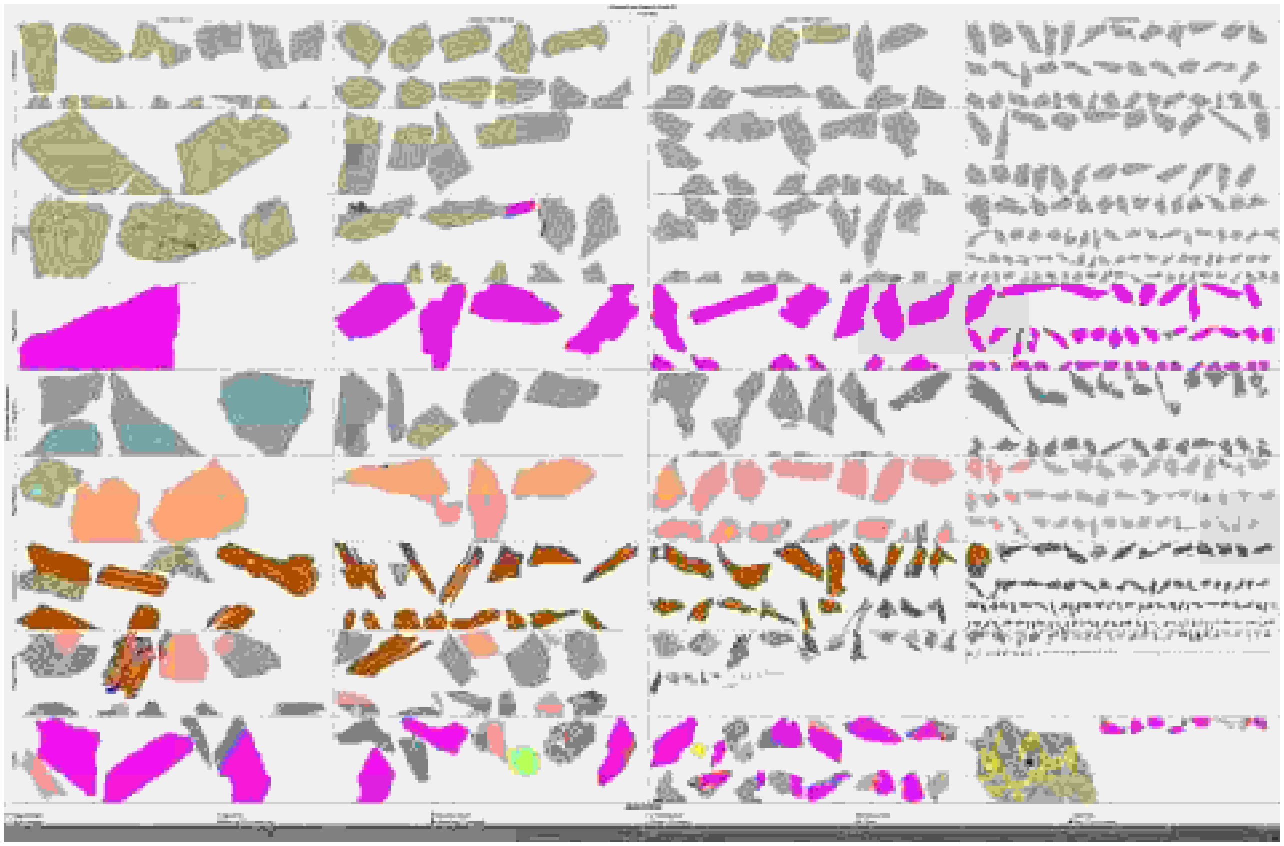
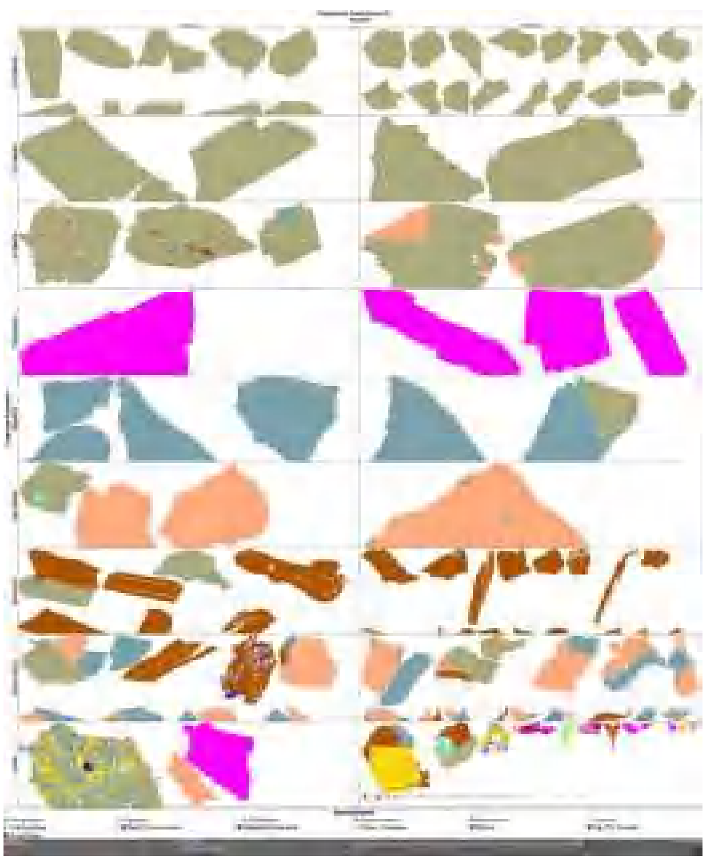
Image Grid of Plagioclase Association





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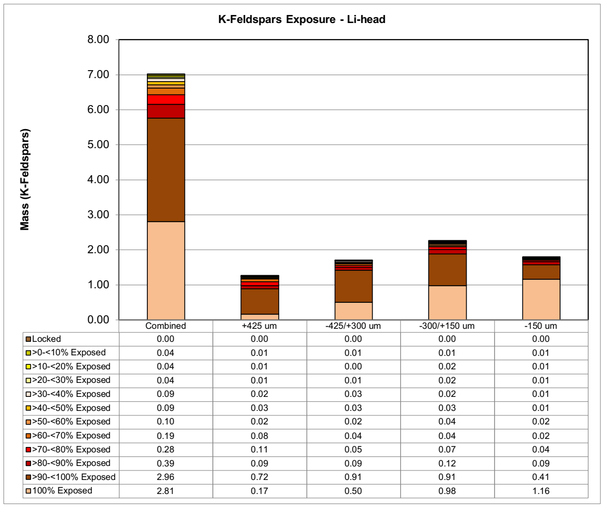
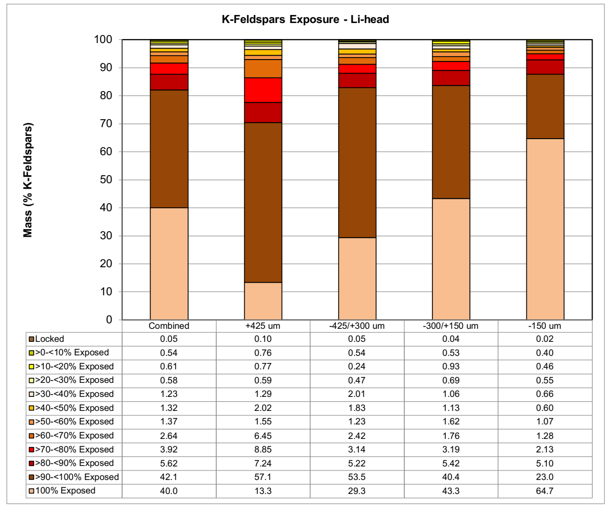
K-Feldspars Exposure



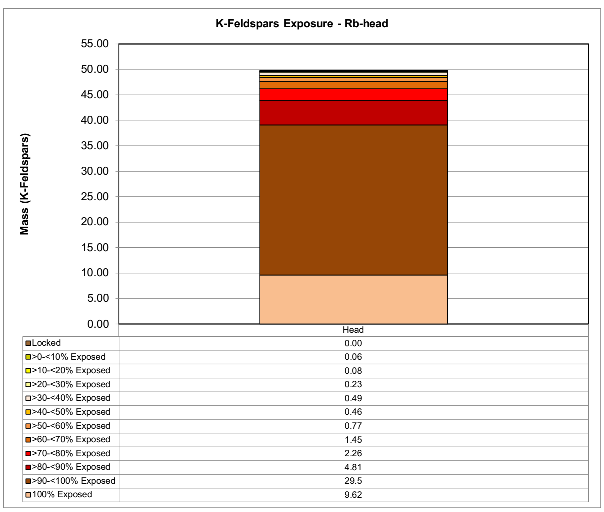


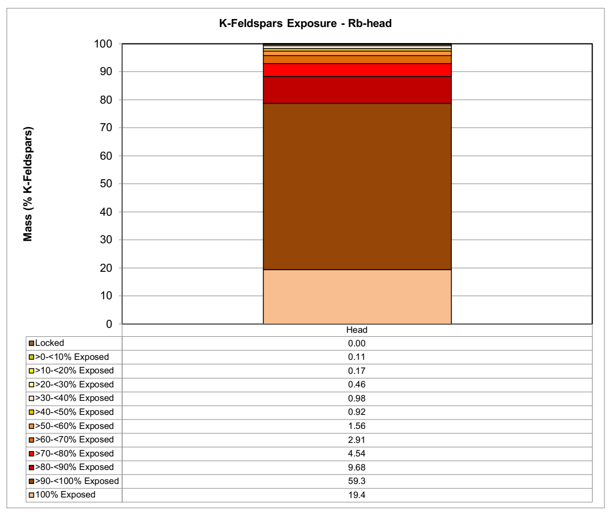
Absolute Mass of K-Feldspars Across Fraction Li-hea

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	2.81	0.17	0.50	0.98	1.16
>90-<100% Exposed	2.96	0.72	0.91	0.91	0.41
>80-<90% Exposed	0.39	0.09	0.09	0.12	0.09
>70-<80% Exposed	0.28	0.11	0.05	0.07	0.04
>60-<70% Exposed	0.19	0.08	0.04	0.04	0.02
>50-<60% Exposed	0.10	0.02	0.02	0.04	0.02
>40-<50% Exposed	0.09	0.03	0.03	0.03	0.01
>30-<40% Exposed	0.09	0.02	0.03	0.02	0.01
>20-<30% Exposed	0.04	0.01	0.01	0.02	0.01
>10-<20% Exposed	0.04	0.01	0.00	0.02	0.01
>0-<10% Exposed	0.04	0.01	0.01	0.01	0.01
Locked	0.00	0.00	0.00	0.00	0.00
Total	7.02	1.26	1.71	2.26	1.79
Total (% in fraction)	100.0	18.0	24.3	32.2	25.6

Normalized Mass of K-Feldspars Across Fraction Li-hea

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	40.0	13.3	29.3	43.3	64.7
>90-<100% Exposed	42.1	57.1	53.5	40.4	23.0
>80-<90% Exposed	5.62	7.24	5.22	5.42	5.10
>70-<80% Exposed	3.92	8.85	3.14	3.19	2.13
>60-<70% Exposed	2.64	6.45	2.42	1.76	1.28
>50-<60% Exposed	1.37	1.55	1.23	1.62	1.07
>40-<50% Exposed	1.32	2.02	1.83	1.13	0.60
>30-<40% Exposed	1.23	1.29	2.01	1.06	0.66
>20-<30% Exposed	0.58	0.59	0.47	0.69	0.55
>10-<20% Exposed	0.61	0.77	0.24	0.93	0.46
>0-<10% Exposed	0.54	0.76	0.54	0.53	0.40
Locked	0.05	0.10	0.05	0.04	0.02
Total	100.0	100.0	100.0	100.0	100.0
>20% Exposed	98.8	98.4	99.2	98.5	99.1





Absolute Mass of K-Feldspars Across Fraction Rb-hea

Mineral Name	Head
100% Exposed	9.62
>90-<100% Exposed	29.5
>80-<90% Exposed	4.81
>70-<80% Exposed	2.26
>60-<70% Exposed	1.45
>50-<60% Exposed	0.77
>40-<50% Exposed	0.46
>30-<40% Exposed	0.49
>20-<30% Exposed	0.23
>10-<20% Exposed	0.08
>0-<10% Exposed	0.06
Locked	0.00
Total	49.7

Normalized Mass of K-Feldspars Across Fraction Rb-hea

Mineral Name	Head
100% Exposed	19.4
>90-<100% Exposed	59.3
>80-<90% Exposed	9.68
>70-<80% Exposed	4.54
>60-<70% Exposed	2.91
>50-<60% Exposed	1.56
>40-<50% Exposed	0.92
>30-<40% Exposed	0.98
>20-<30% Exposed	0.46
>10-<20% Exposed	0.17
>0-<10% Exposed	0.11
Locked	0.00
Total	100.0
>20% Exposed	99.7

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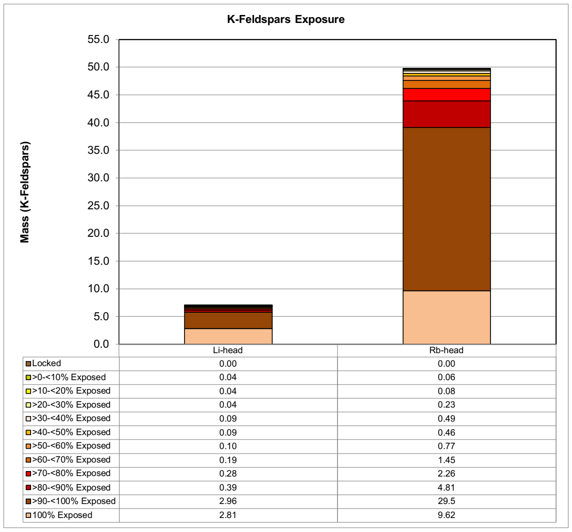
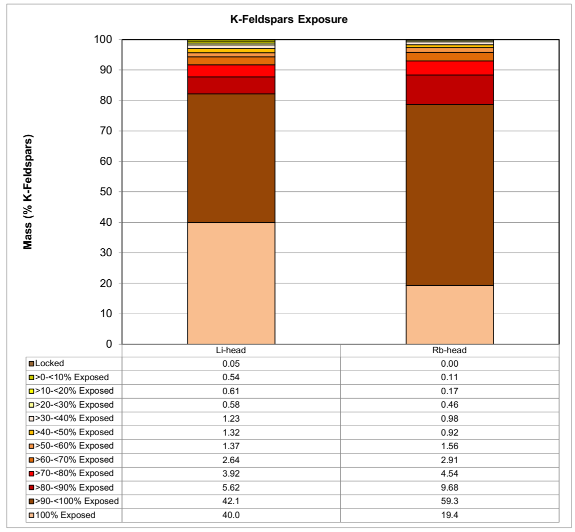
K-Feldspars Exposure





Mineral Name	Li-head	Rb-head
100% Exposed	2.81	9.62
>90-<100% Exposed	2.96	29.5
>80-<90% Exposed	0.39	4.81
>70-<80% Exposed	0.28	2.26
>60-<70% Exposed	0.19	1.45
>50-<60% Exposed	0.10	0.77
>40-<50% Exposed	0.09	0.46
>30-<40% Exposed	0.09	0.49
>20-<30% Exposed	0.04	0.23
>10-<20% Exposed	0.04	0.08
>0-<10% Exposed	0.04	0.06
Locked	0.00	0.00
Total	7.02	49.7

Mineral Name	Li-head	Rb-head
100% Exposed	40.0	19.4
>90-<100% Exposed	42.1	59.3
>80-<90% Exposed	5.62	9.68
>70-<80% Exposed	3.92	4.54
>60-<70% Exposed	2.64	2.91
>50-<60% Exposed	1.37	1.56
>40-<50% Exposed	1.32	0.92
>30-<40% Exposed	1.23	0.98
>20-<30% Exposed	0.58	0.46
>10-<20% Exposed	0.61	0.17
>0-<10% Exposed	0.54	0.11
Locked	0.05	0.00
Total	100.0	100.0
>20% Exposed	98.8	99.7

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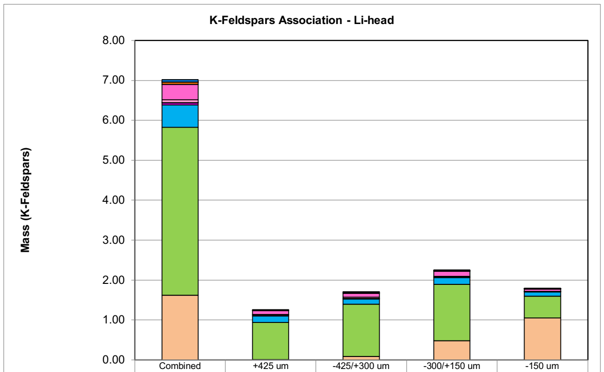
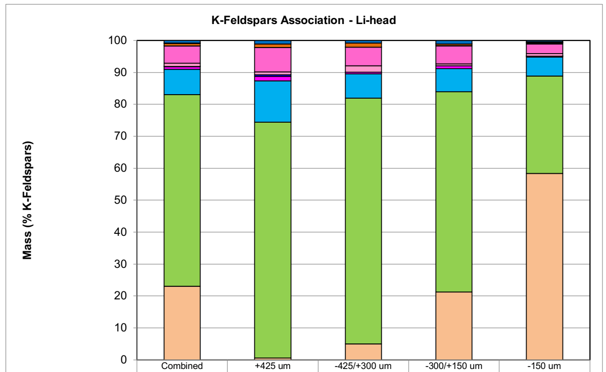
MI5018-SEP23

K-Feldspars Association



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Complex	0.06	0.01	0.01	0.02	0.01
K-Feld:Other	0.00	0.00	0.00	0.00	0.00
K-Feld:Sulphides	0.00	0.00	0.00	0.00	0.00
K-Feld:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Phosphates	0.00	0.00	0.00	0.00	0.00
K-Feld:Carbonates	0.00	0.00	0.00	0.00	0.00
K-Feld:Garnet	0.00	0.00	0.00	0.00	0.00
K-Feld:Other Silicates	0.00	0.00	0.00	0.00	0.00
K-Feld:Qtz:Plag:Ms	0.05	0.01	0.02	0.01	0.00
K-Feld:Muscovite	0.00	0.00	0.00	0.00	0.00
K-Feld:Plagioclase	0.38	0.10	0.10	0.13	0.06
K-Feld:Quartz	0.07	0.01	0.03	0.01	0.01
K-Feld:Beryl/Tourmaline	0.00	0.00	0.00	0.00	0.00
K-Feld:Petalite/Cookeite	0.01	0.01	0.00	0.00	0.00
K-Feld:Spodumene	0.05	0.02	0.01	0.02	0.01
Lib K-Feldspars	0.56	0.16	0.13	0.16	0.10
Free K-Feldspars	4.20	0.93	1.31	1.42	0.55
Pure K-Feldspars	1.62	0.01	0.09	0.48	1.05



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Complex	0.83	1.12	0.77	1.10	0.36
K-Feld:Other	0.00	0.00	0.00	0.00	0.00
K-Feld:Sulphides	0.00	0.00	0.00	0.00	0.00
K-Feld:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Phosphates	0.00	0.00	0.00	0.00	0.00
K-Feld:Carbonates	0.06	0.00	0.00	0.00	0.24
K-Feld:Garnet	0.01	0.00	0.00	0.00	0.06
K-Feld:Other Silicates	0.00	0.00	0.00	0.00	0.01
K-Feld:Qtz:Plag:Ms	0.76	1.06	1.37	0.53	0.25
K-Feld:Muscovite	0.06	0.02	0.01	0.03	0.17
K-Feld:Plagioclase	5.40	7.64	5.82	5.68	3.08
K-Feld:Quartz	1.03	0.88	2.00	0.62	0.72
K-Feld:Beryl/Tourmaline	0.01	0.00	0.02	0.00	0.02
K-Feld:Petalite/Cookeite	0.11	0.57	0.00	0.00	0.02
K-Feld:Spodumene	0.72	1.36	0.50	0.82	0.35
Lib K-Feldspars	8.01	13.0	7.56	7.30	5.85
Free K-Feldspars	59.9	73.8	76.9	62.7	30.5
Pure K-Feldspars	23.1	0.62	5.05	21.2	58.4

Absolute Mass of K-Feldspars Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure K-Feldspars	1.62	0.01	0.09	0.48	1.05
Free K-Feldspars	4.20	0.93	1.31	1.42	0.55
Lib K-Feldspars	0.56	0.16	0.13	0.16	0.10
K-Feld:Spodumene	0.05	0.02	0.01	0.02	0.01
K-Feld:Petalite/Cookeite	0.01	0.01	0.00	0.00	0.00
K-Feld:Beryl/Tourmaline	0.00	0.00	0.00	0.00	0.00
K-Feld:Quartz	0.07	0.01	0.03	0.01	0.01
K-Feld:Plagioclase	0.38	0.10	0.10	0.13	0.06
K-Feld:Muscovite	0.00	0.00	0.00	0.00	0.00
K-Feld:Qtz:Plag:Ms	0.05	0.01	0.02	0.01	0.00
K-Feld:Other Silicates	0.00	0.00	0.00	0.00	0.00
K-Feld:Garnet	0.00	0.00	0.00	0.00	0.00
K-Feld:Carbonates	0.00	0.00	0.00	0.00	0.00
K-Feld:Phosphates	0.00	0.00	0.00	0.00	0.00
K-Feld:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Sulphides	0.00	0.00	0.00	0.00	0.00
K-Feld:Other	0.00	0.00	0.00	0.00	0.00
Complex	0.06	0.01	0.01	0.02	0.01
Total	7.02	1.26	1.71	2.26	1.79
Total (% in fraction)	100.0	18.0	24.3	32.2	25.6

Normalized Mass of K-Feldspars Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure K-Feldspars	23.1	0.62	5.05	21.2	58.4
Free K-Feldspars	59.9	73.8	76.9	62.7	30.5
Lib K-Feldspars	8.01	13.0	7.56	7.30	5.85
K-Feld:Spodumene	0.72	1.36	0.50	0.82	0.35
K-Feld:Petalite/Cookeite	0.11	0.57	0.00	0.00	0.02
K-Feld:Beryl/Tourmaline	0.01	0.00	0.02	0.00	0.02
K-Feld:Quartz	1.03	0.88	2.00	0.62	0.72
K-Feld:Plagioclase	5.40	7.64	5.82	5.68	3.08
K-Feld:Muscovite	0.06	0.02	0.01	0.03	0.17
K-Feld:Qtz:Plag:Ms	0.76	1.06	1.37	0.53	0.25
K-Feld:Other Silicates	0.00	0.00	0.00	0.00	0.01
K-Feld:Garnet	0.01	0.00	0.00	0.00	0.06
K-Feld:Carbonates	0.06	0.00	0.00	0.00	0.24
K-Feld:Phosphates	0.00	0.00	0.00	0.00	0.00
K-Feld:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
K-Feld:Sulphides	0.00	0.00	0.00	0.00	0.00
K-Feld:Other	0.00	0.00	0.00	0.00	0.00
Complex	0.83	1.12	0.77	1.10	0.36
Total	100.0	100.0	100.0	100.0	100.0
Total Liberated	91.0	87.3	89.5	91.2	94.7

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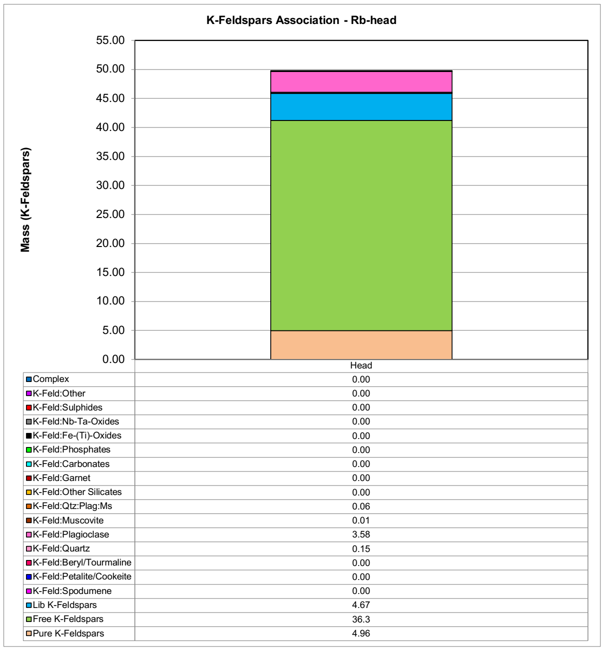
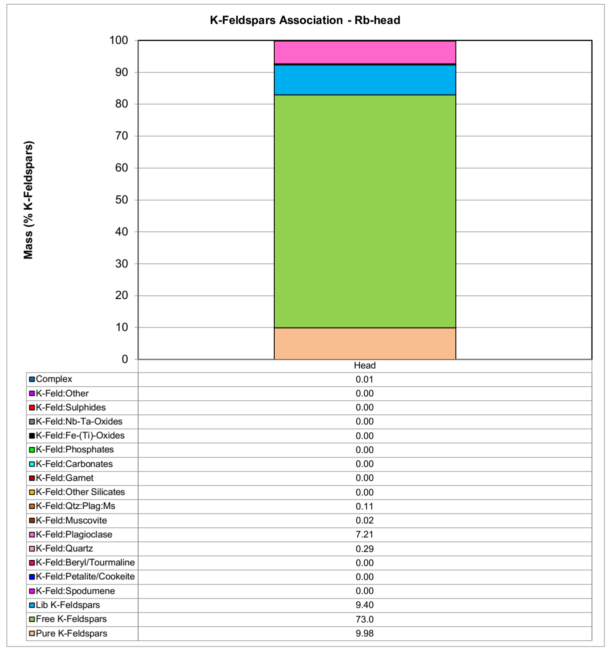
K-Feldspars Association



Absolute Mass of K-Feldspars Across Fraction Rb-head

Mineral Name	Head
Pure K-Feldspars	4.96
Free K-Feldspars	36.3
Lib K-Feldspars	4.67
K-Feld:Spodumene	0.00
K-Feld:Petalite/Cookeite	0.00
K-Feld:Beryl/Tourmaline	0.00
K-Feld:Quartz	0.15
K-Feld:Plagioclase	3.58
K-Feld:Muscovite	0.01
K-Feld:Qtz:Plag:Ms	0.06
K-Feld:Other Silicates	0.00
K-Feld:Garnet	0.00
K-Feld:Carbonates	0.00
K-Feld:Phosphates	0.00
K-Feld:Fe-(Ti)-Oxides	0.00
K-Feld:Nb-Ta-Oxides	0.00
K-Feld:Sulphides	0.00
K-Feld:Other	0.00
Complex	0.00
Total	49.7



Normalized Mass of K-Feldspars Across Fraction Rb-head

Mineral Name	Head
Pure K-Feldspars	9.98
Free K-Feldspars	73.0
Lib K-Feldspars	9.40
K-Feld:Spodumene	0.00
K-Feld:Petalite/Cookeite	0.00
K-Feld:Beryl/Tourmaline	0.00
K-Feld:Quartz	0.29
K-Feld:Plagioclase	7.21
K-Feld:Muscovite	0.02
K-Feld:Qtz:Plag:Ms	0.11
K-Feld:Other Silicates	0.00
K-Feld:Garnet	0.00
K-Feld:Carbonates	0.00
K-Feld:Phosphates	0.00
K-Feld:Fe-(Ti)-Oxides	0.00
K-Feld:Nb-Ta-Oxides	0.00
K-Feld:Sulphides	0.00
K-Feld:Other	0.00
Complex	0.01
Total	100.0
Total Liberated	92.4

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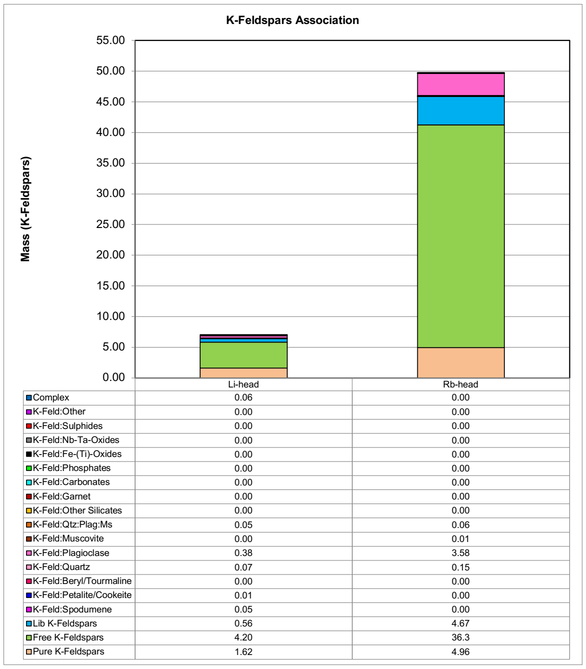
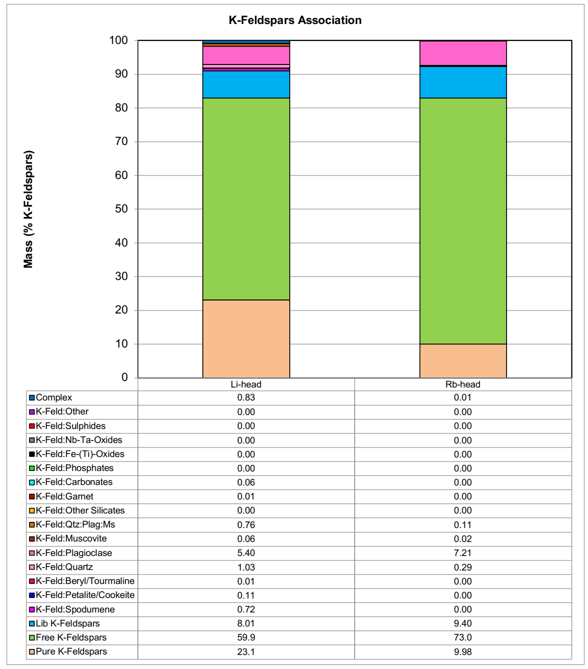
K-Feldspars Association





Mineral Name	Li-head	Rb-head
Pure K-Feldspars	1.62	4.96
Free K-Feldspars	4.20	36.3
Lib K-Feldspars	0.56	4.67
K-Feld:Spodumene	0.05	0.00
K-Feld:Petalite/Cookeite	0.01	0.00
K-Feld:Beryl/Tourmaline	0.00	0.00
K-Feld:Quartz	0.07	0.15
K-Feld:Plagioclase	0.38	3.58
K-Feld:Muscovite	0.00	0.01
K-Feld:Qtz:Plag:Ms	0.05	0.06
K-Feld:Other Silicates	0.00	0.00
K-Feld:Garnet	0.00	0.00
K-Feld:Carbonates	0.00	0.00
K-Feld:Phosphates	0.00	0.00
K-Feld:Fe-(Ti)-Oxides	0.00	0.00
K-Feld:Nb-Ta-Oxides	0.00	0.00
K-Feld:Sulphides	0.00	0.00
K-Feld:Other	0.00	0.00
Complex	0.06	0.00
Total	7.02	49.7

Mineral Name	Li-head	Rb-head
Pure K-Feldspars	23.1	9.98
Free K-Feldspars	59.9	73.0
Lib K-Feldspars	8.01	9.40
K-Feld:Spodumene	0.72	0.00
K-Feld:Petalite/Cookeite	0.11	0.00
K-Feld:Beryl/Tourmaline	0.01	0.00
K-Feld:Quartz	1.03	0.29
K-Feld:Plagioclase	5.40	7.21
K-Feld:Muscovite	0.06	0.02
K-Feld:Qtz:Plag:Ms	0.76	0.11
K-Feld:Other Silicates	0.00	0.00
K-Feld:Garnet	0.01	0.00
K-Feld:Carbonates	0.06	0.00
K-Feld:Phosphates	0.00	0.00
K-Feld:Fe-(Ti)-Oxides	0.00	0.00
K-Feld:Nb-Ta-Oxides	0.00	0.00
K-Feld:Sulphides	0.00	0.00
K-Feld:Other	0.00	0.00
Complex	0.83	0.01
Total	100.0	100.0
Total Liberated	91.0	92.4

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Image Grid of K-Feldspars Exposure

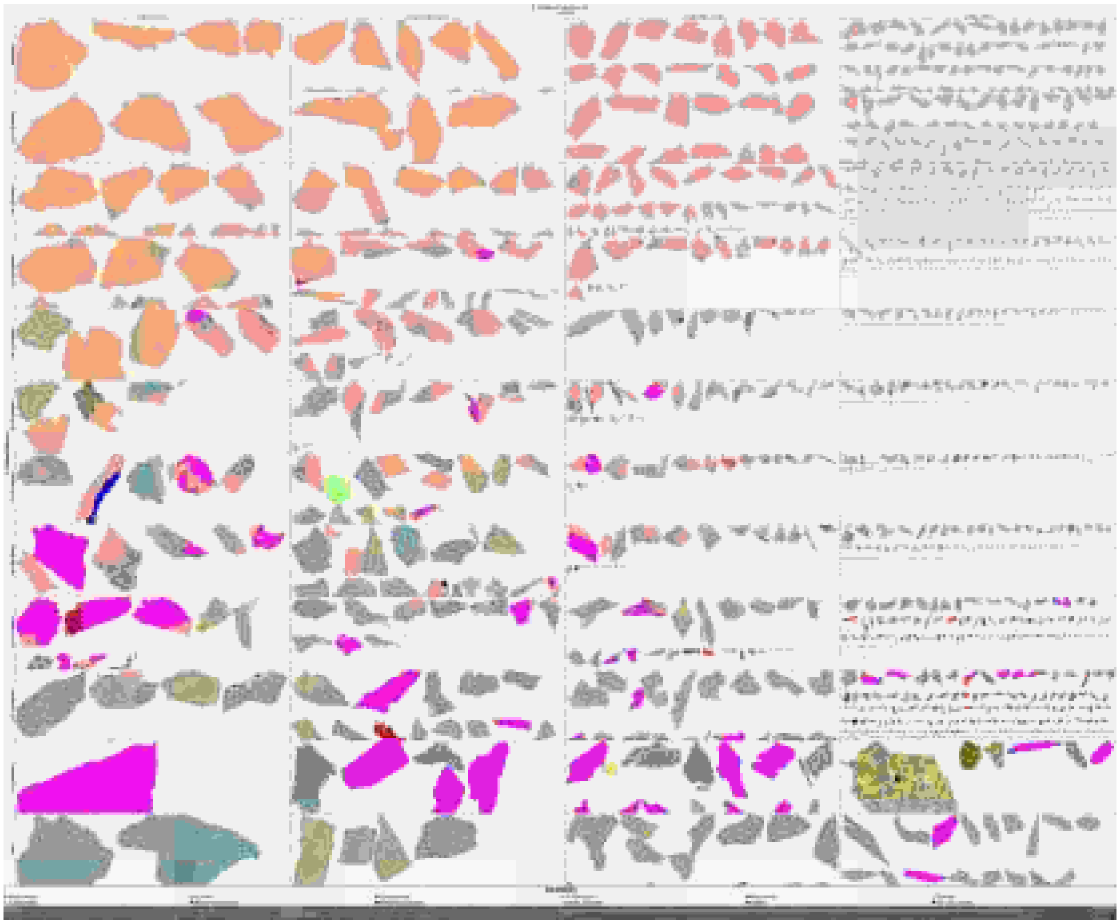

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Image Grid of K-Feldspars Exposure

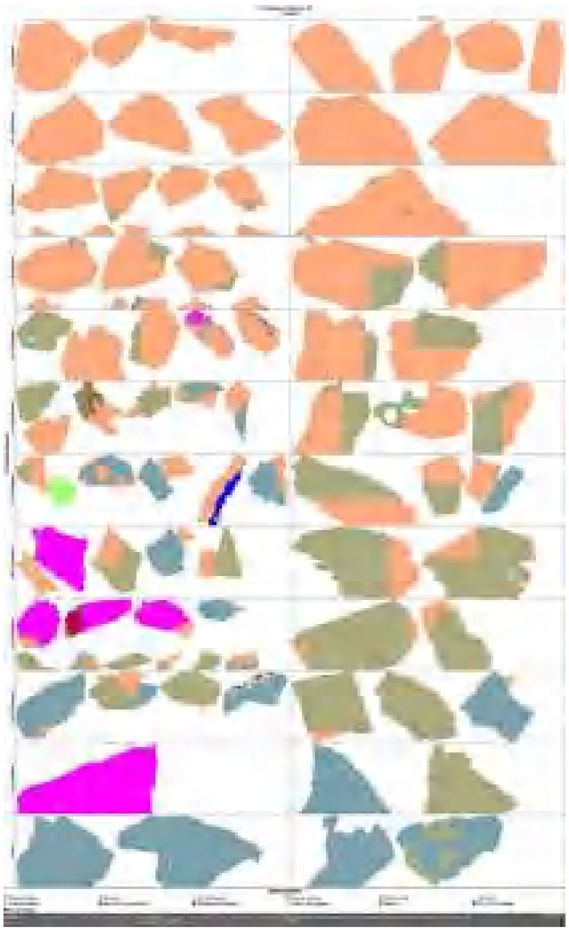

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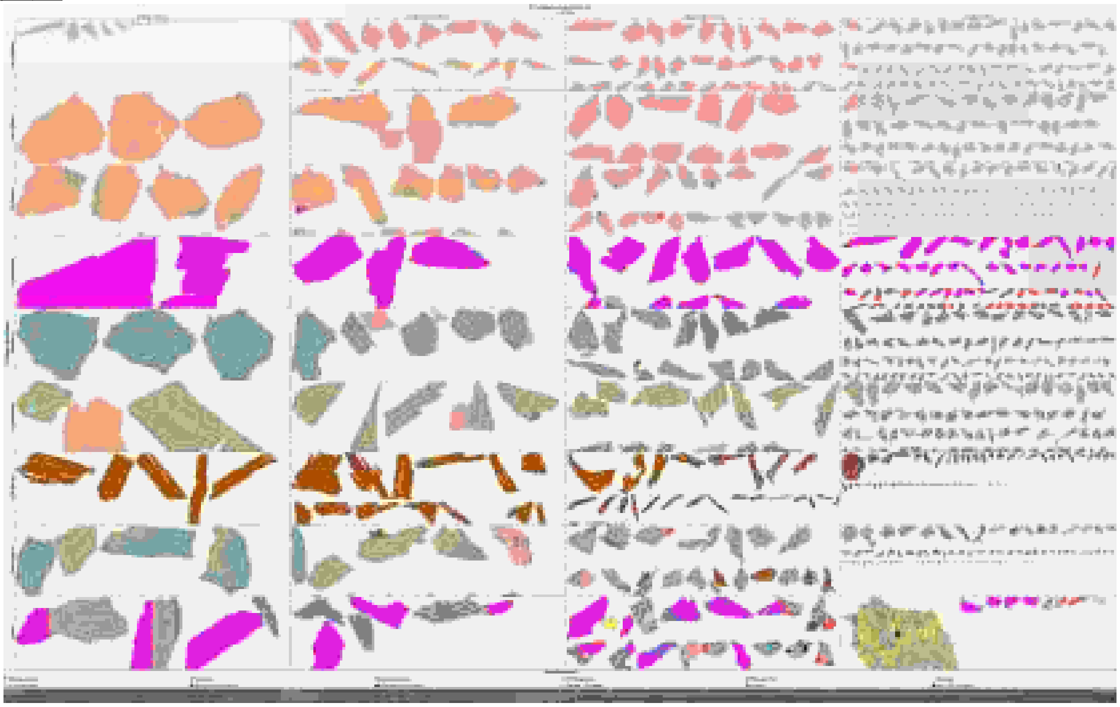
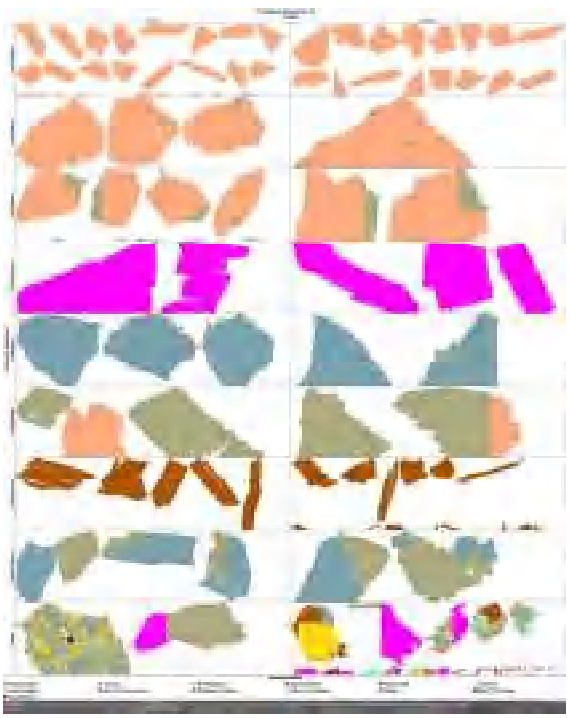
Image Grid of K-Feldspars Association





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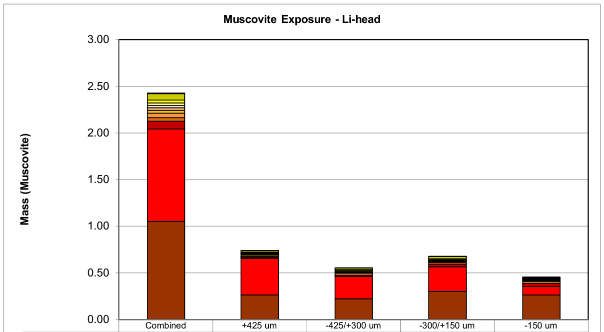
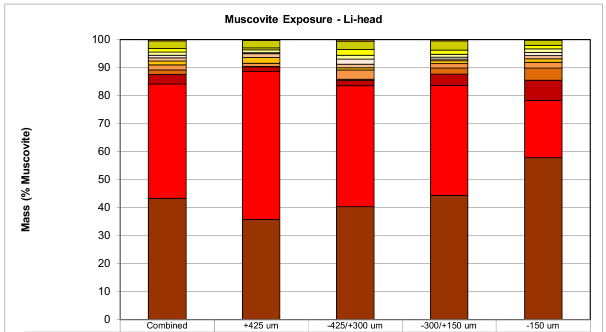
Muscovite Exposure



	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Locked	0.01	0.00	0.00	0.00	0.00
>0-<10% Exposed	0.07	0.02	0.02	0.02	0.01
>10-<20% Exposed	0.03	0.01	0.01	0.01	0.01
>20-<30% Exposed	0.03	0.01	0.01	0.01	0.01
>30-<40% Exposed	0.02	0.00	0.01	0.00	0.01
>40-<50% Exposed	0.03	0.01	0.01	0.00	0.01
>50-<60% Exposed	0.03	0.02	0.00	0.01	0.01
>60-<70% Exposed	0.05	0.01	0.02	0.01	0.01
>70-<80% Exposed	0.04	0.00	0.00	0.02	0.02
>80-<90% Exposed	0.08	0.01	0.01	0.03	0.03
>90-<100% Exposed	0.99	0.39	0.24	0.27	0.09
100% Exposed	1.05	0.27	0.22	0.30	0.26



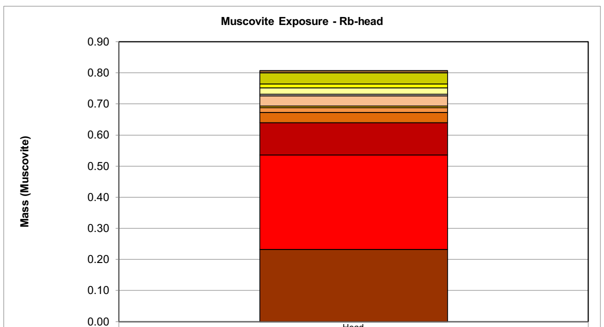
	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Locked	0.38	0.30	0.56	0.43	0.22
>0-<10% Exposed	2.75	2.61	2.99	3.33	1.85
>10-<20% Exposed	1.33	0.76	2.01	1.50	1.17
>20-<30% Exposed	1.13	1.01	1.35	0.96	1.30
>30-<40% Exposed	0.92	0.31	1.83	0.68	1.15
>40-<50% Exposed	1.10	1.38	1.31	0.62	1.12
>50-<60% Exposed	1.35	2.05	0.81	1.08	1.28
>60-<70% Exposed	1.91	1.16	3.24	1.50	2.09
>70-<80% Exposed	1.58	0.14	0.47	2.26	4.29
>80-<90% Exposed	3.43	1.73	1.92	3.98	7.20
>90-<100% Exposed	40.8	52.8	43.2	39.3	20.4
100% Exposed	43.4	35.8	40.3	44.4	57.9

Absolute Mass of Muscovite Across Fraction Li-hea

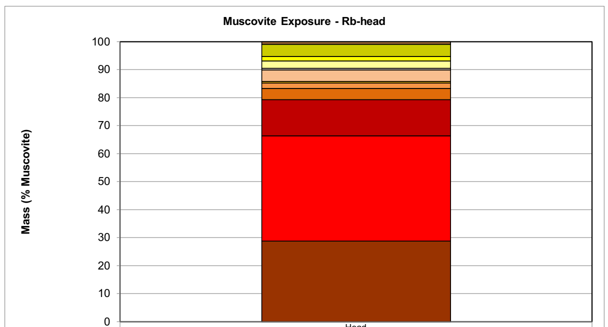
Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	1.05	0.27	0.22	0.30	0.26
>90-<100% Exposed	0.99	0.39	0.24	0.27	0.09
>80-<90% Exposed	0.08	0.01	0.01	0.03	0.03
>70-<80% Exposed	0.04	0.00	0.00	0.02	0.02
>60-<70% Exposed	0.05	0.01	0.02	0.01	0.01
>50-<60% Exposed	0.03	0.02	0.00	0.01	0.01
>40-<50% Exposed	0.03	0.01	0.01	0.00	0.01
>30-<40% Exposed	0.02	0.00	0.01	0.00	0.01
>20-<30% Exposed	0.03	0.01	0.01	0.01	0.01
>10-<20% Exposed	0.03	0.01	0.01	0.01	0.01
>0-<10% Exposed	0.07	0.02	0.02	0.02	0.01
Locked	0.01	0.00	0.00	0.00	0.00
Total	2.43	0.74	0.56	0.68	0.45
Total (% in fraction)	100.0	30.5	22.9	27.9	18.7

Normalized Mass of Muscovite Across Fraction Li-hea

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
100% Exposed	43.4	35.8	40.3	44.4	57.9
>90-<100% Exposed	40.8	52.8	43.2	39.3	20.4
>80-<90% Exposed	3.43	1.73	1.92	3.98	7.20
>70-<80% Exposed	1.58	0.14	0.47	2.26	4.29
>60-<70% Exposed	1.91	1.16	3.24	1.50	2.09
>50-<60% Exposed	1.35	2.05	0.81	1.08	1.28
>40-<50% Exposed	1.10	1.38	1.31	0.62	1.12
>30-<40% Exposed	0.92	0.31	1.83	0.68	1.15
>20-<30% Exposed	1.13	1.01	1.35	0.96	1.30
>10-<20% Exposed	1.33	0.76	2.01	1.50	1.17
>0-<10% Exposed	2.75	2.61	2.99	3.33	1.85
Locked	0.38	0.30	0.56	0.43	0.22
Total	100.0	100.0	100.0	100.0	100.0
>20% Exposed	95.5	96.3	94.4	94.7	96.8



	Head
Locked	0.01
>0-<10% Exposed	0.04
>10-<20% Exposed	0.01
>20-<30% Exposed	0.02
>30-<40% Exposed	0.00
>40-<50% Exposed	0.03
>50-<60% Exposed	0.00
>60-<70% Exposed	0.02
>70-<80% Exposed	0.03
>80-<90% Exposed	0.10
>90-<100% Exposed	0.30
100% Exposed	0.23



	Head
Locked	0.89
>0-<10% Exposed	4.40
>10-<20% Exposed	1.56
>20-<30% Exposed	2.60
>30-<40% Exposed	0.61
>40-<50% Exposed	4.10
>50-<60% Exposed	0.56
>60-<70% Exposed	1.96
>70-<80% Exposed	4.04
>80-<90% Exposed	12.9
>90-<100% Exposed	37.6
100% Exposed	28.7

Absolute Mass of Muscovite Across Fraction Rb-hea

Mineral Name	Head
100% Exposed	0.23
>90-<100% Exposed	0.30
>80-<90% Exposed	0.10
>70-<80% Exposed	0.03
>60-<70% Exposed	0.02
>50-<60% Exposed	0.00
>40-<50% Exposed	0.03
>30-<40% Exposed	0.00
>20-<30% Exposed	0.02
>10-<20% Exposed	0.01
>0-<10% Exposed	0.04
Locked	0.01
Total	0.81

Normalized Mass of Muscovite Across Fraction Rb-hea

Mineral Name	Head
100% Exposed	28.7
>90-<100% Exposed	37.6
>80-<90% Exposed	12.9
>70-<80% Exposed	4.04
>60-<70% Exposed	1.96
>50-<60% Exposed	0.56
>40-<50% Exposed	4.10
>30-<40% Exposed	0.61
>20-<30% Exposed	2.60
>10-<20% Exposed	1.56
>0-<10% Exposed	4.40
Locked	0.89
Total	100.0
>20% Exposed	93.1

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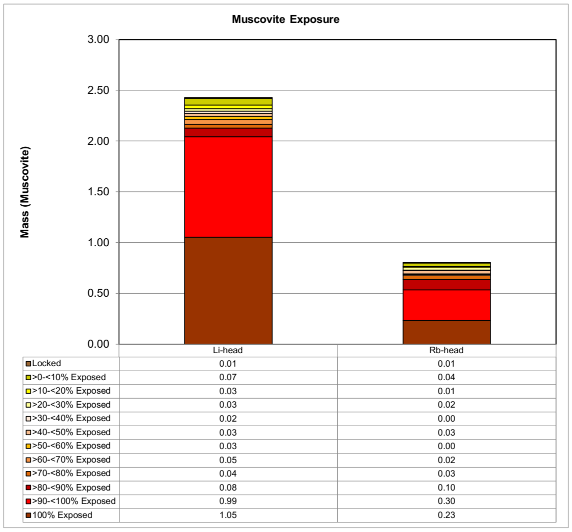
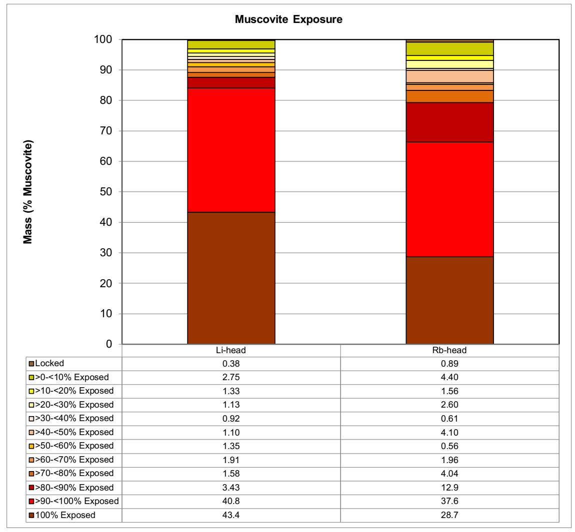
Muscovite Exposure





Mineral Name	Li-head	Rb-head
100% Exposed	1.05	0.23
>90-<100% Exposed	0.99	0.30
>80-<90% Exposed	0.08	0.10
>70-<80% Exposed	0.04	0.03
>60-<70% Exposed	0.05	0.02
>50-<60% Exposed	0.03	0.00
>40-<50% Exposed	0.03	0.03
>30-<40% Exposed	0.02	0.00
>20-<30% Exposed	0.03	0.02
>10-<20% Exposed	0.03	0.01
>0-<10% Exposed	0.07	0.04
Locked	0.01	0.01
Total	2.43	0.81

Mineral Name	Li-head	Rb-head
100% Exposed	43.4	28.7
>90-<100% Exposed	40.8	37.6
>80-<90% Exposed	3.43	12.9
>70-<80% Exposed	1.58	4.04
>60-<70% Exposed	1.91	1.96
>50-<60% Exposed	1.35	0.56
>40-<50% Exposed	1.10	4.10
>30-<40% Exposed	0.92	0.61
>20-<30% Exposed	1.13	2.60
>10-<20% Exposed	1.33	1.56
>0-<10% Exposed	2.75	4.40
Locked	0.38	0.89
Total	100.0	100.0
>20% Exposed	95.5	93.1

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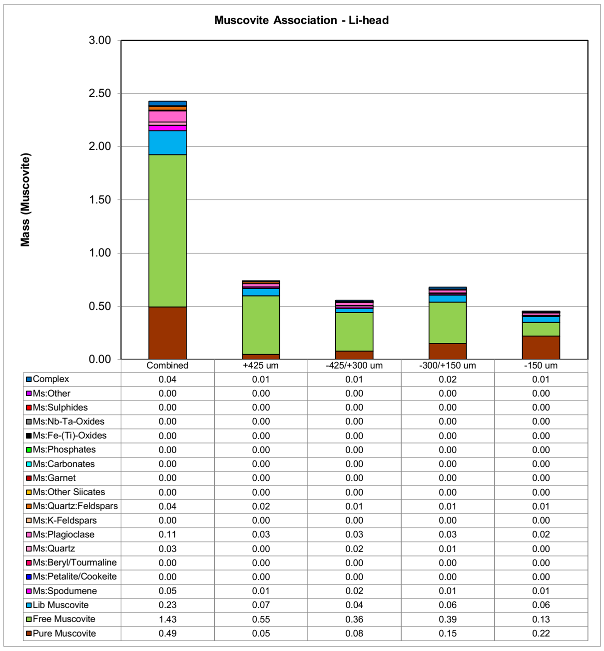
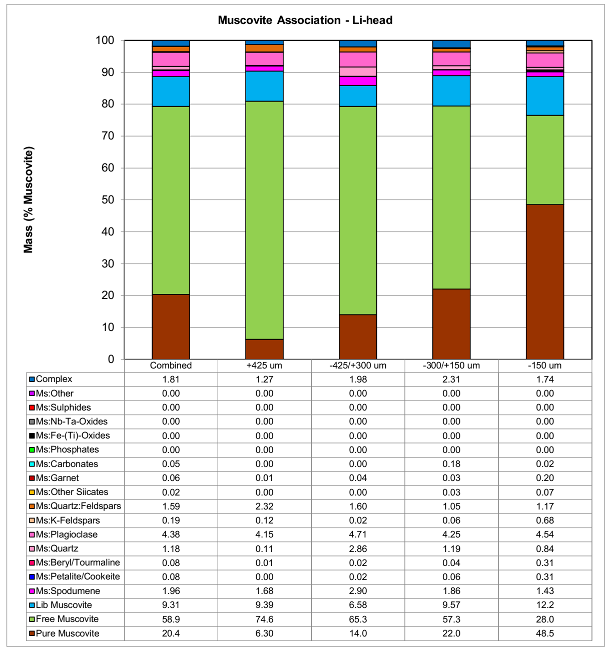
MI5018-SEP23

Muscovite Association





Absolute Mass of Muscovite Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure Muscovite	0.49	0.05	0.08	0.15	0.22
Free Muscovite	1.43	0.55	0.36	0.39	0.13
Lib Muscovite	0.23	0.07	0.04	0.06	0.06
Ms:Spodumene	0.05	0.01	0.02	0.01	0.01
Ms:Petalite/Cookeite	0.00	0.00	0.00	0.00	0.00
Ms:Beryl/Tourmaline	0.00	0.00	0.00	0.00	0.00
Ms:Quartz	0.03	0.00	0.02	0.01	0.00
Ms:Plagioclase	0.11	0.03	0.03	0.03	0.02
Ms:K-Feldspars	0.00	0.00	0.00	0.00	0.00
Ms:Quartz:Feldspars	0.04	0.02	0.01	0.01	0.01
Ms:Other Siicates	0.00	0.00	0.00	0.00	0.00
Ms:Garnet	0.00	0.00	0.00	0.00	0.00
Ms:Carbonates	0.00	0.00	0.00	0.00	0.00
Ms:Phosphates	0.00	0.00	0.00	0.00	0.00
Ms:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
Ms:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
Ms:Sulphides	0.00	0.00	0.00	0.00	0.00
Ms:Other	0.00	0.00	0.00	0.00	0.00
Complex	0.04	0.01	0.01	0.02	0.01
Total	2.43	0.74	0.56	0.68	0.45
Total (% in fraction)	100.0	30.5	22.9	27.9	18.7

Normalized Mass of Muscovite Across Fraction Li-head

Mineral Name	Combined	+425 um	-425/+300 um	-300/+150 um	-150 um
Pure Muscovite	20.4	6.30	14.0	22.0	48.5
Free Muscovite	58.9	74.6	65.3	57.3	28.0
Lib Muscovite	9.31	9.39	6.58	9.57	12.2
Ms:Spodumene	1.96	1.68	2.90	1.86	1.43
Ms:Petalite/Cookeite	0.08	0.00	0.02	0.06	0.31
Ms:Beryl/Tourmaline	0.08	0.01	0.02	0.04	0.31
Ms:Quartz	1.18	0.11	2.86	1.19	0.84
Ms:Plagioclase	4.38	4.15	4.71	4.25	4.54
Ms:K-Feldspars	0.19	0.12	0.02	0.06	0.68
Ms:Quartz:Feldspars	1.59	2.32	1.60	1.05	1.17
Ms:Other Siicates	0.02	0.00	0.00	0.03	0.07
Ms:Garnet	0.06	0.01	0.04	0.03	0.20
Ms:Carbonates	0.05	0.00	0.00	0.18	0.02
Ms:Phosphates	0.00	0.00	0.00	0.00	0.00
Ms:Fe-(Ti)-Oxides	0.00	0.00	0.00	0.00	0.00
Ms:Nb-Ta-Oxides	0.00	0.00	0.00	0.00	0.00
Ms:Sulphides	0.00	0.00	0.00	0.00	0.00
Ms:Other	0.00	0.00	0.00	0.00	0.00
Complex	1.81	1.27	1.98	2.31	1.74
Total	100.0	100.0	100.0	100.0	100.0
Total Liberated	88.6	90.3	85.9	88.9	88.7

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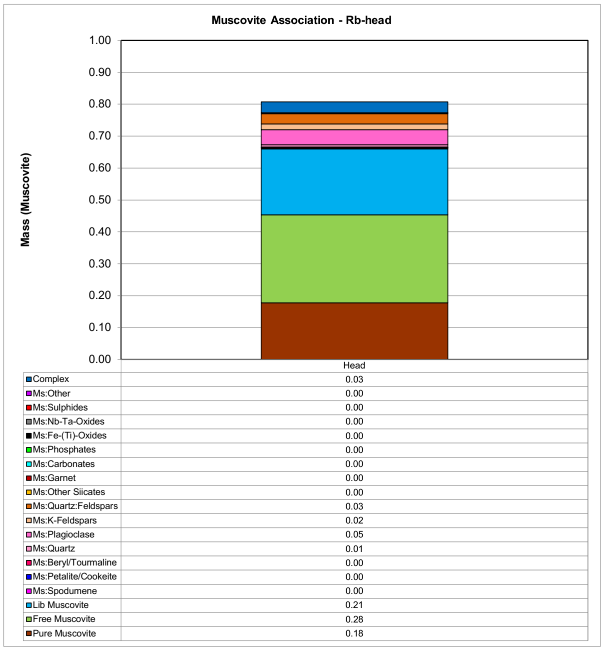
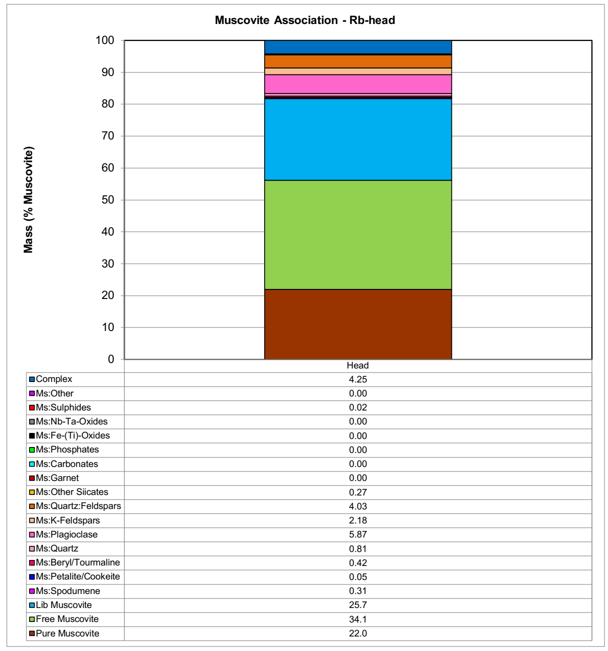
Muscovite Association





Absolute Mass of Muscovite Across Fraction Rb-head

Mineral Name	Head
Pure Muscovite	0.18
Free Muscovite	0.28
Lib Muscovite	0.21
Ms:Spodumene	0.00
Ms:Petalite/Cookeite	0.00
Ms:Beryl/Tourmaline	0.00
Ms:Quartz	0.01
Ms:Plagioclase	0.05
Ms:K-Feldspars	0.02
Ms:Quartz:Feldspars	0.03
Ms:Other Siicates	0.00
Ms:Garnet	0.00
Ms:Carbonates	0.00
Ms:Phosphates	0.00
Ms:Fe-(Ti)-Oxides	0.00
Ms:Nb-Ta-Oxides	0.00
Ms:Sulphides	0.00
Ms:Other	0.00
Complex	0.03
Total	0.81

Normalized Mass of Muscovite Across Fraction Rb-head

Mineral Name	Head
Pure Muscovite	22.0
Free Muscovite	34.1
Lib Muscovite	25.7
Ms:Spodumene	0.31
Ms:Petalite/Cookeite	0.05
Ms:Beryl/Tourmaline	0.42
Ms:Quartz	0.81
Ms:Plagioclase	5.87
Ms:K-Feldspars	2.18
Ms:Quartz:Feldspars	4.03
Ms:Other Siicates	0.27
Ms:Garnet	0.00
Ms:Carbonates	0.00
Ms:Phosphates	0.00
Ms:Fe-(Ti)-Oxides	0.00
Ms:Nb-Ta-Oxides	0.00
Ms:Sulphides	0.02
Ms:Other	0.00
Complex	4.25
Total	100.0
Total Liberated	81.8

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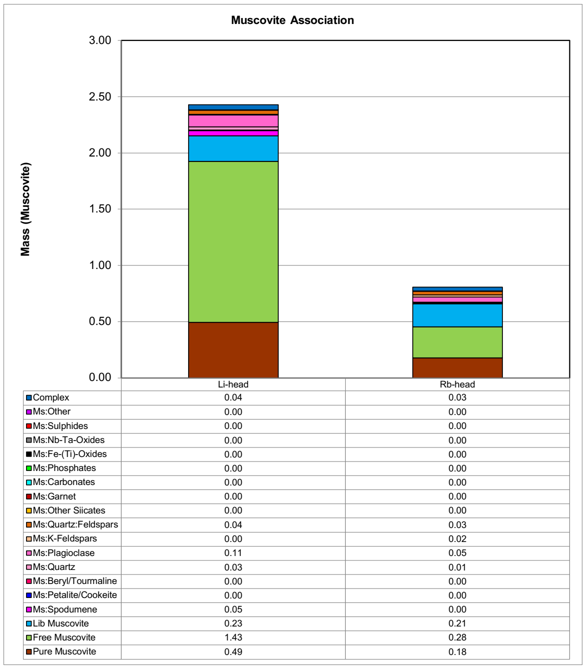
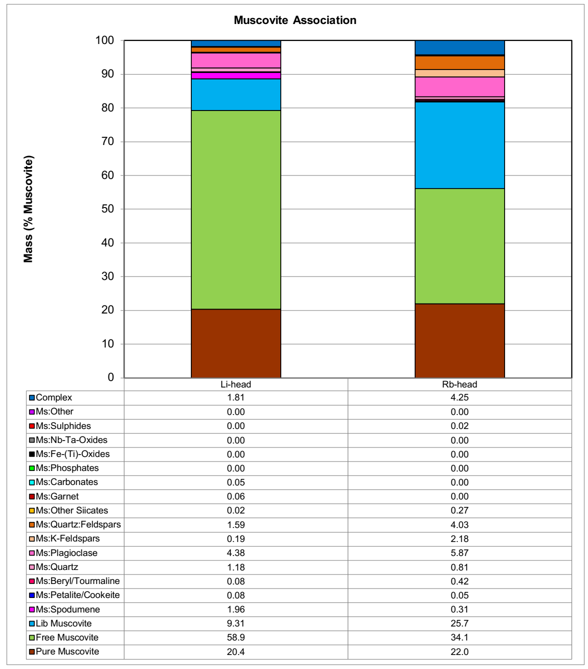
Muscovite Association





Mineral Name	Li-head	Rb-head
Pure Muscovite	0.49	0.18
Free Muscovite	1.43	0.28
Lib Muscovite	0.23	0.21
Ms:Spodumene	0.05	0.00
Ms:Petalite/Cookeite	0.00	0.00
Ms:Beryl/Tourmaline	0.00	0.00
Ms:Quartz	0.03	0.01
Ms:Plagioclase	0.11	0.05
Ms:K-Feldspars	0.00	0.02
Ms:Quartz:Feldspars	0.04	0.03
Ms:Other Siicates	0.00	0.00
Ms:Garnet	0.00	0.00
Ms:Carbonates	0.00	0.00
Ms:Phosphates	0.00	0.00
Ms:Fe-(Ti)-Oxides	0.00	0.00
Ms:Nb-Ta-Oxides	0.00	0.00
Ms:Sulphides	0.00	0.00
Ms:Other	0.00	0.00
Complex	0.04	0.03
Total	2.43	0.81

Mineral Name	Li-head	Rb-head
Pure Muscovite	20.4	22.0
Free Muscovite	58.9	34.1
Lib Muscovite	9.31	25.7
Ms:Spodumene	1.96	0.31
Ms:Petalite/Cookeite	0.08	0.05
Ms:Beryl/Tourmaline	0.08	0.42
Ms:Quartz	1.18	0.81
Ms:Plagioclase	4.38	5.87
Ms:K-Feldspars	0.19	2.18
Ms:Quartz:Feldspars	1.59	4.03
Ms:Other Siicates	0.02	0.27
Ms:Garnet	0.06	0.00
Ms:Carbonates	0.05	0.00
Ms:Phosphates	0.00	0.00
Ms:Fe-(Ti)-Oxides	0.00	0.00
Ms:Nb-Ta-Oxides	0.00	0.00
Ms:Sulphides	0.00	0.02
Ms:Other	0.00	0.00
Complex	1.81	4.25
Total	100.0	100.0
Total Liberated	88.6	81.8

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Image Grid of Muscovite Exposure

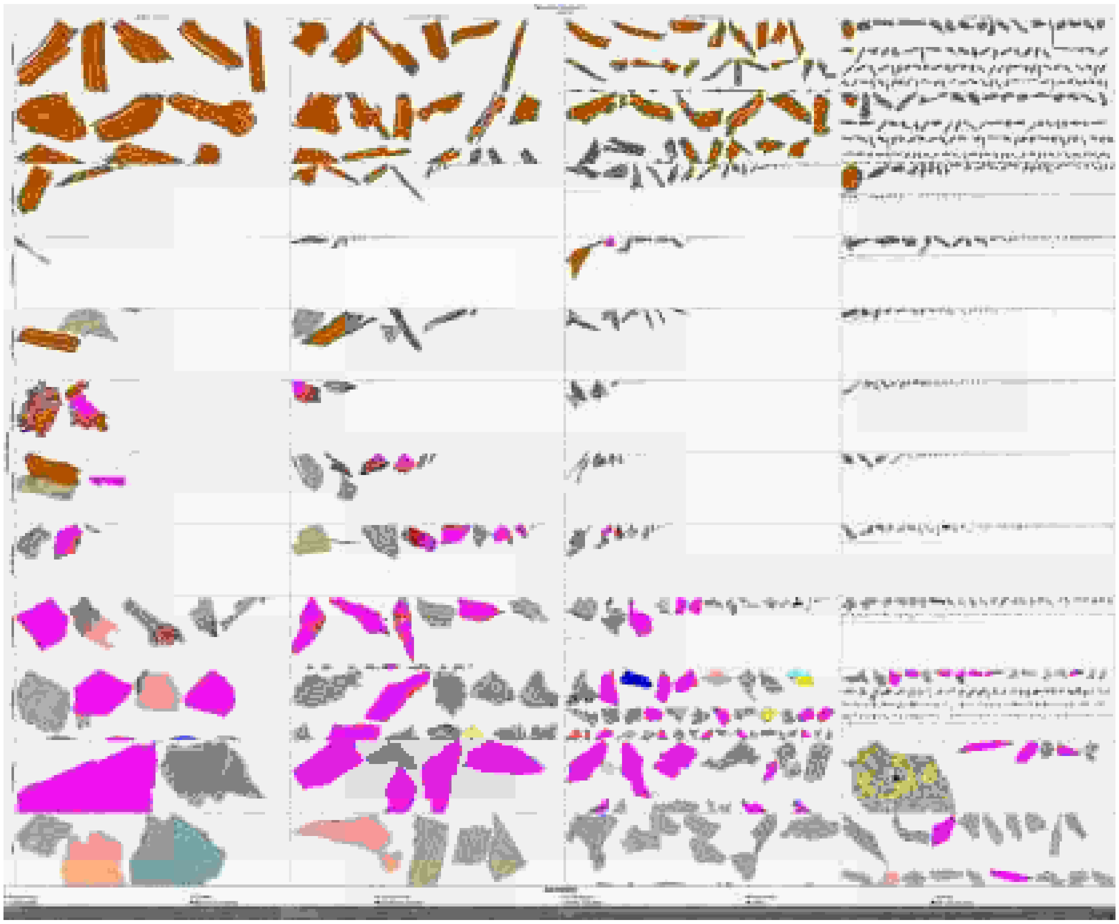

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Image Grid of Muscovite Exposure

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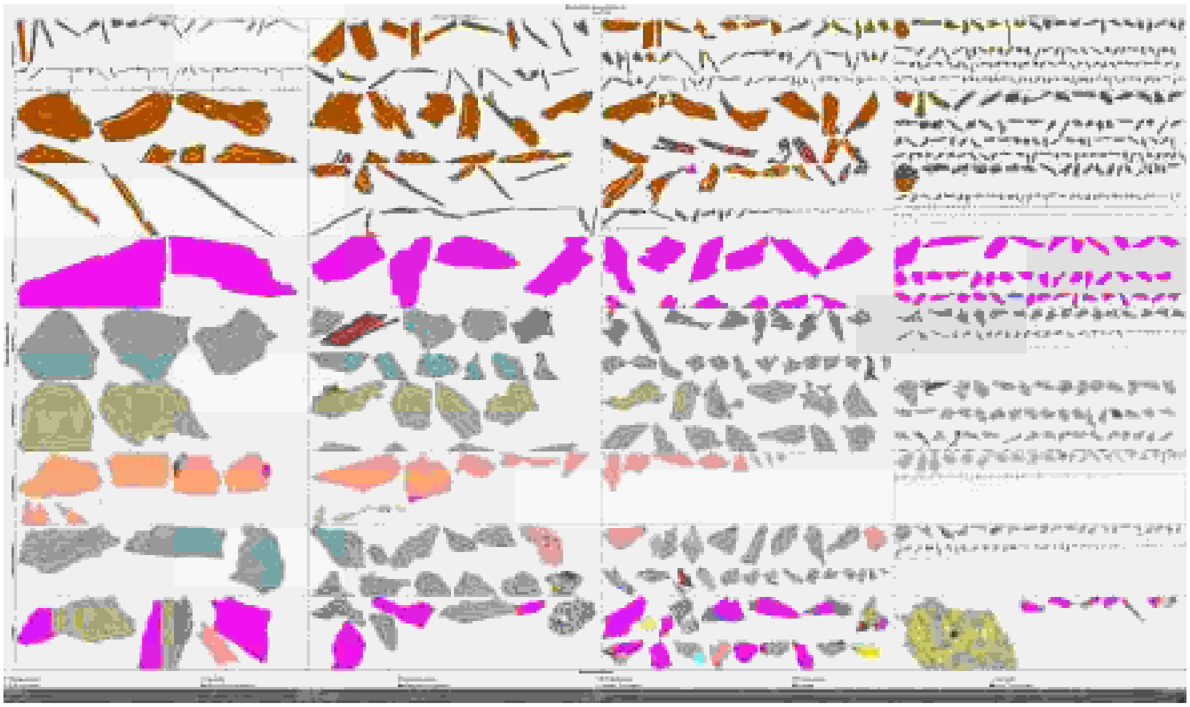
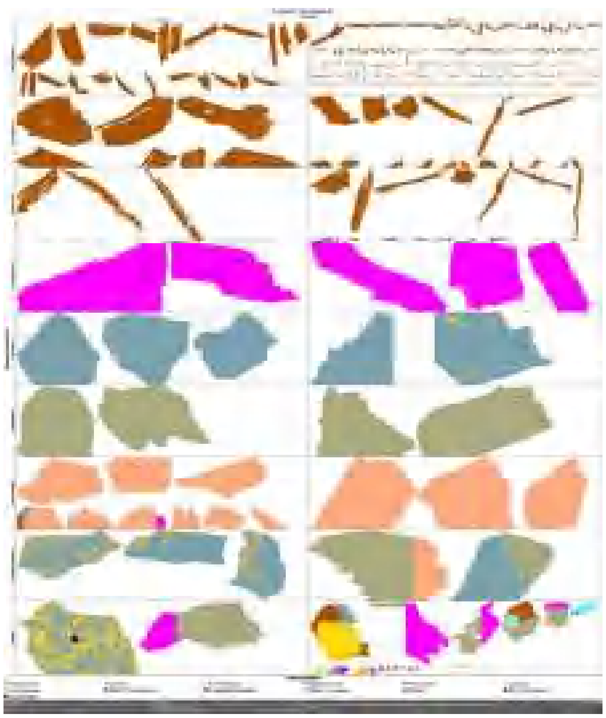
Image Grid of Muscovite Association





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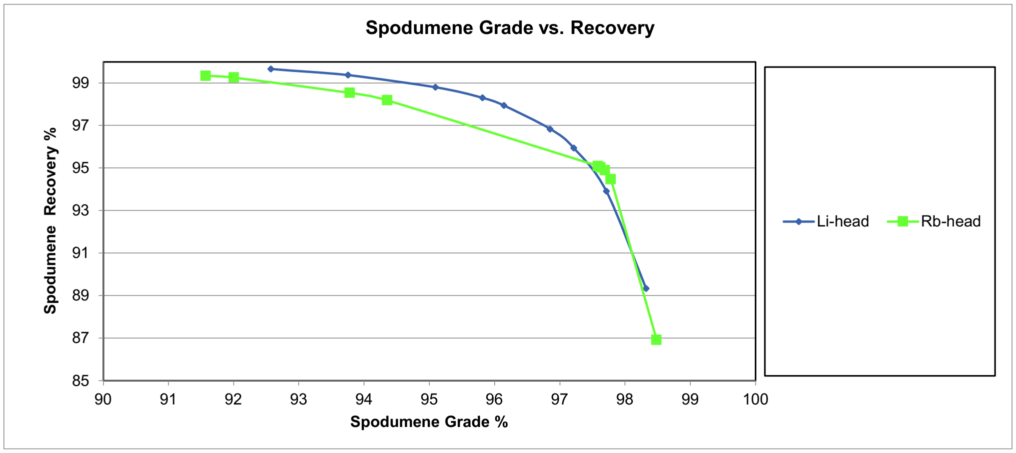
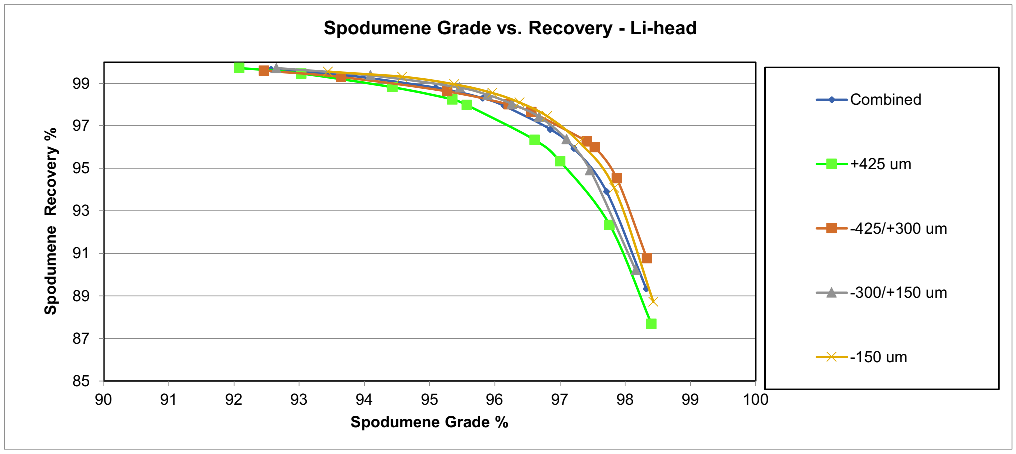
Spodumene Grade vs. Recovery:





Volume % of Spodumene / Sample	Li-head		Rb-head	
	Grade	Recovery	Grade	Recovery
All particles	9.73	100.0	0.94	100.0
≥10	92.6	99.7	91.6	99.3
≥20	93.8	99.4	92.0	99.3
≥30	95.1	98.8	93.8	98.5
≥40	95.8	98.3	94.4	98.2
≥50	96.1	97.9	97.6	95.1
≥60	96.9	96.8	97.6	95.0
≥70	97.2	96.0	97.7	94.9
≥80	97.7	93.9	97.8	94.5
≥90	98.3	89.3	98.5	86.9

Volume % of Spodumene / Li-head	Combined		+425 um		-425/+300 um		-300/+150 um		-150 um	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	9.73	100.0	15.0	100.0	10.2	100.0	8.19	100.0	7.29	100.0
≥10	93	100	92.1	99.7	92.5	99.6	92.7	99.7	93.4	99.6
≥20	94	99	93.0	99.5	93.6	99.3	94.1	99.4	94.6	99.3
≥30	95	99	94.4	98.8	95.3	98.6	95.5	98.8	95.4	99.0
≥40	96	98	95.4	98.2	96.2	98.0	95.9	98.5	96.0	98.6
≥50	96	98	95.6	98.0	96.6	97.7	96.3	98.0	96.4	98.1
≥60	97	97	96.6	96.3	97.4	96.3	96.7	97.4	96.8	97.5
≥70	97	96	97.0	95.3	97.5	96.0	97.1	96.4	97.3	96.2
≥80	98	94	97.8	92.3	97.9	94.5	97.5	94.9	97.8	94.1
≥90	98	89	98.4	87.7	98.3	90.8	98.2	90.2	98.4	88.7

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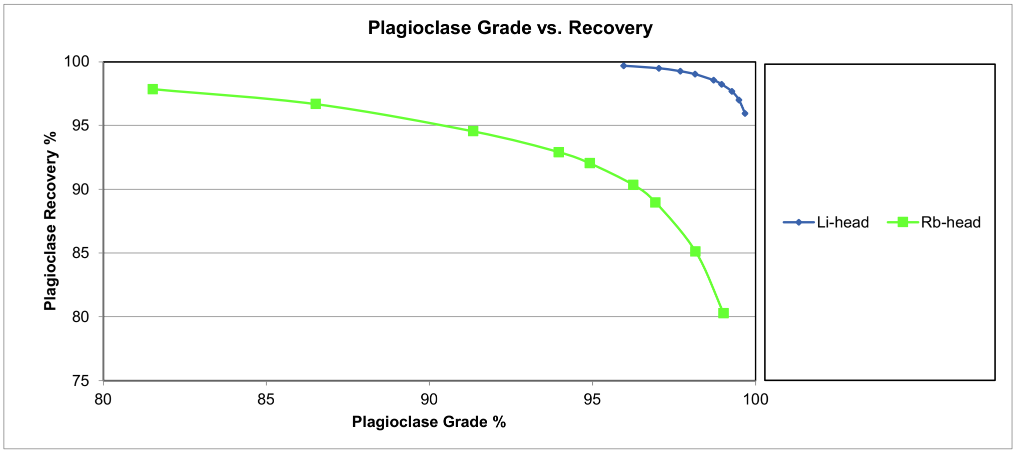
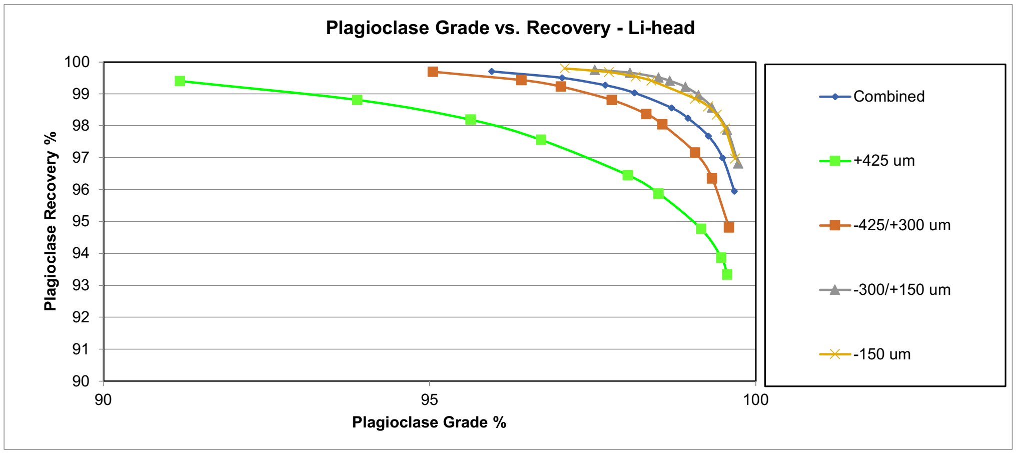
Plagioclase Grade vs. Recovery:





Volume % of Plagioclase/ Sample	Li-head		Rb-head	
	Grade	Recovery	Grade	Recovery
All particles	56.1	100.0	33.1	100.0
≥10	96.0	99.7	81.5	97.9
≥20	97.0	99.5	86.5	96.7
≥30	97.7	99.3	91.3	94.6
≥40	98.1	99.0	94.0	92.9
≥50	98.7	98.6	94.9	92.0
≥60	99.0	98.2	96.3	90.4
≥70	99.3	97.7	96.9	89.0
≥80	99.5	97.0	98.2	85.1
≥90	99.7	96.0	99.0	80.3

Volume % of Plagioclase / Li-head	Combined		+425 um		-425/+300 um		-300/+150 um		-150 um	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	56.1	100.0	40.2	100.0	53.7	100.0	62.0	100.0	62.6	100.0
≥10	96.0	99.7	91.2	99.4	95.1	99.7	97.5	99.8	97.1	99.8
≥20	97.0	99.5	93.9	98.8	96.4	99.4	98.1	99.7	97.8	99.7
≥30	97.7	99.3	95.6	98.2	97.0	99.2	98.5	99.5	98.2	99.5
≥40	98.1	99.0	96.7	97.6	97.8	98.8	98.7	99.4	98.4	99.4
≥50	98.7	98.6	98.0	96.5	98.3	98.4	98.9	99.2	99.1	98.9
≥60	99.0	98.2	98.5	95.9	98.6	98.0	99.1	99.0	99.3	98.6
≥70	99.3	97.7	99.2	94.8	99.1	97.2	99.3	98.6	99.4	98.3
≥80	99.5	97.0	99.5	93.9	99.3	96.4	99.6	97.9	99.5	97.9
≥90	99.7	96.0	99.6	93.3	99.6	94.8	99.7	96.8	99.7	97.0

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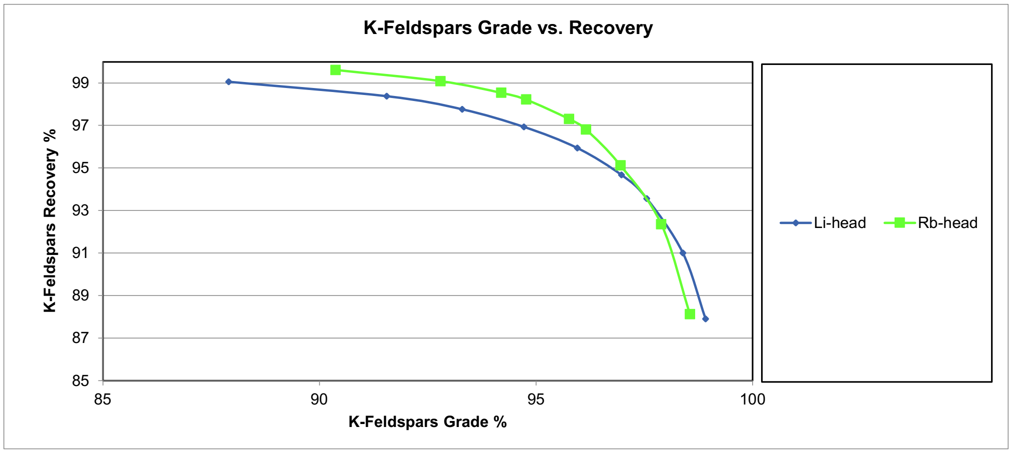
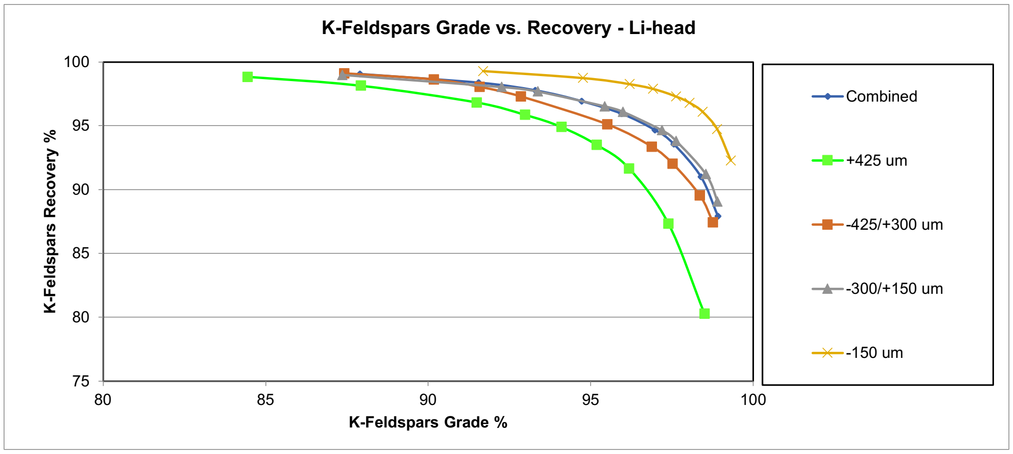
K-Feldspars Grade vs. Recovery:





Volume % of K-Feldspars / Sample	Li-head		Rb-head	
	Grade	Recovery	Grade	Recovery
All particles	7.02	100.0	49.7	100.0
≥10	87.9	99.1	90.4	99.6
≥20	91.6	98.4	92.8	99.1
≥30	93.3	97.8	94.2	98.5
≥40	94.7	96.9	94.8	98.2
≥50	96.0	96.0	95.8	97.3
≥60	97.0	94.7	96.2	96.8
≥70	97.6	93.6	97.0	95.1
≥80	98.4	91.0	97.9	92.4
≥90	98.9	87.9	98.6	88.1

Volume % of K-Feldspars / Li-head	Combined		+425 um		-425/+300 um		-300/+150 um		-150 um	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	7.02	100.0	6.50	100.0	7.68	100.0	6.51	100.0	7.57	100.0
≥10	87.9	99.1	84.4	98.8	87.4	99.1	87.4	99.0	91.7	99.3
≥20	91.6	98.4	87.9	98.1	90.2	98.6	92.3	98.0	94.8	98.7
≥30	93.3	97.8	91.5	96.8	91.6	98.1	93.4	97.7	96.2	98.3
≥40	94.7	96.9	93.0	95.9	92.9	97.3	95.4	96.5	96.9	97.9
≥50	96.0	96.0	94.1	94.9	95.5	95.1	96.0	96.1	97.6	97.3
≥60	97.0	94.7	95.2	93.5	96.9	93.4	97.2	94.7	98.0	96.8
≥70	97.6	93.6	96.2	91.7	97.5	92.0	97.6	93.8	98.4	96.1
≥80	98.4	91.0	97.4	87.3	98.4	89.5	98.5	91.2	98.9	94.7
≥90	98.9	87.9	98.5	80.3	98.8	87.4	98.9	89.1	99.3	92.3

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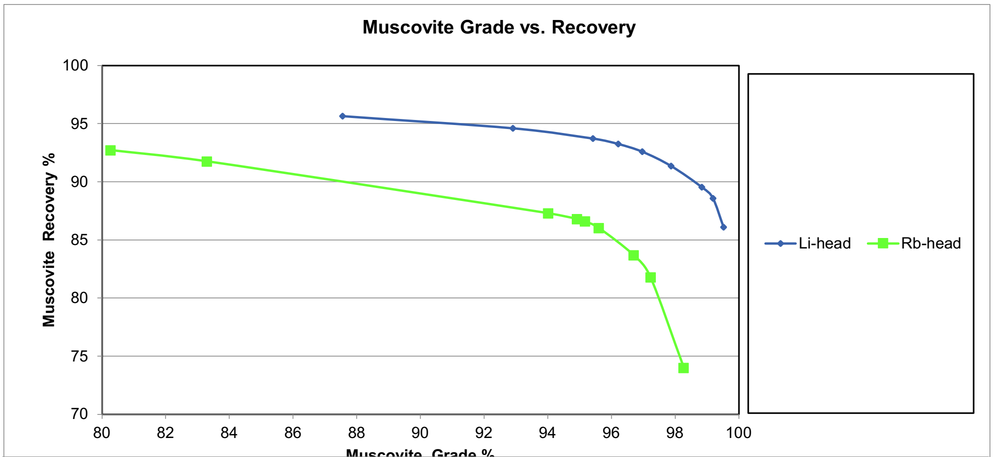
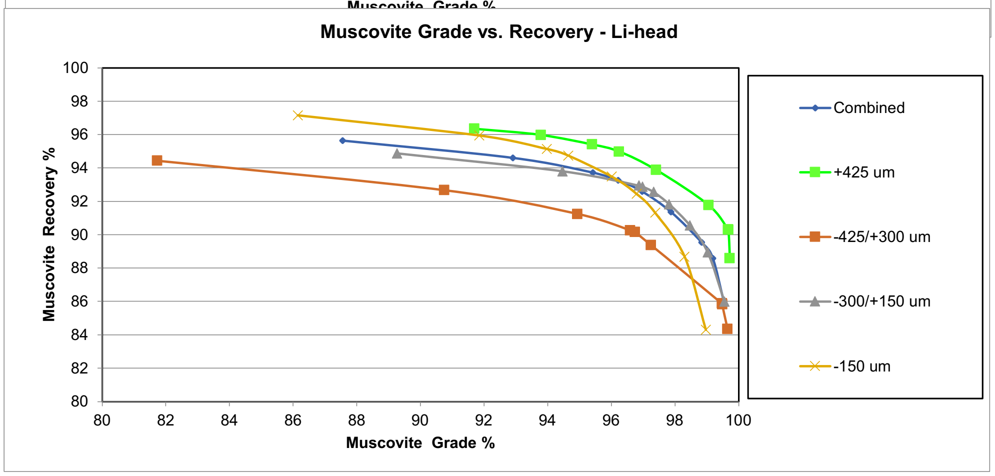
Muscovite Grade vs. Recovery:





Volume % of Muscovite / Sample	Li-head		Rb-head	
	Grade	Recovery	Grade	Recovery
All particles	2.43	100.0	0.81	100.0
≥10	87.6	95.7	80.3	92.7
≥20	92.9	94.6	83.3	91.8
≥30	95.4	93.7	94.0	87.3
≥40	96.2	93.3	94.9	86.8
≥50	97.0	92.6	95.2	86.6
≥60	97.9	91.4	95.6	86.0
≥70	98.8	89.6	96.7	83.7
≥80	99.2	88.6	97.2	81.8
≥90	99.5	86.1	98.3	74.0

Volume % of Muscovite / Li-head	Combined		+425 um		-425/+300 um		-300/+150 um		-150 um	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	2.43	100.0	3.82	100.0	2.51	100.0	1.95	100.0	1.92	100.0
≥10	87.6	95.7	91.7	96.4	81.7	94.4	89.3	94.9	86.2	97.2
≥20	92.9	94.6	93.8	96.0	90.8	92.7	94.5	93.8	91.9	96.0
≥30	95.4	93.7	95.4	95.4	94.9	91.3	96.9	92.9	94.0	95.1
≥40	96.2	93.3	96.2	95.0	96.6	90.3	97.0	92.9	94.7	94.7
≥50	97.0	92.6	97.4	93.9	96.7	90.2	97.3	92.6	96.0	93.5
≥60	97.9	91.4	99.1	91.8	97.2	89.4	97.8	91.8	96.8	92.5
≥70	98.8	89.6	99.7	90.3	99.5	85.9	98.5	90.6	97.4	91.3
≥80	99.2	88.6	99.7	90.3	99.5	85.8	99.0	88.9	98.3	88.7
≥90	99.5	86.1	99.7	88.6	99.7	84.4	99.6	86.0	99.0	84.3

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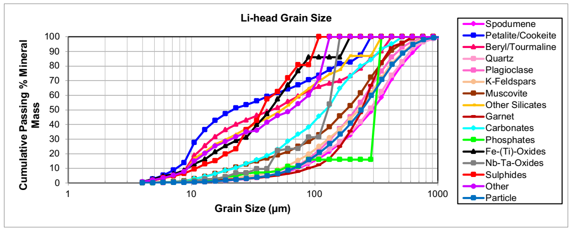
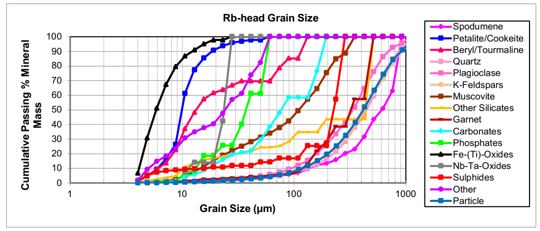
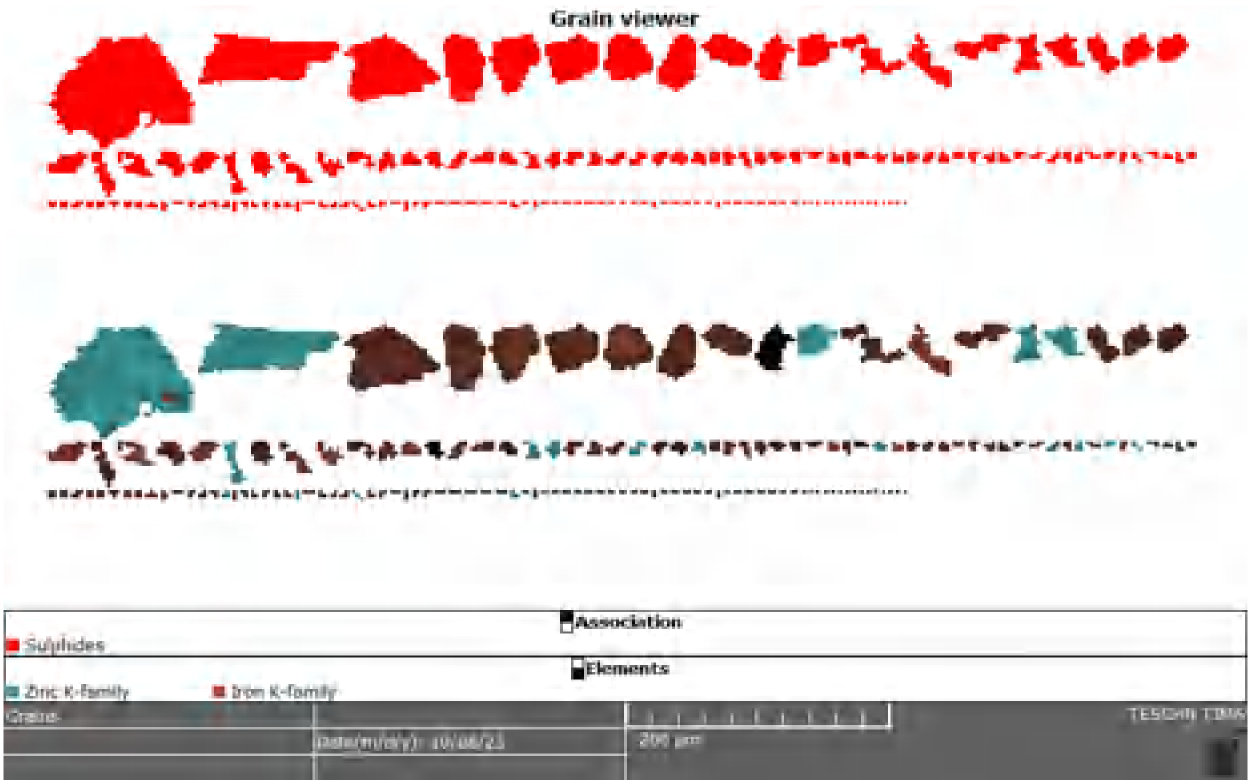
Cumulative Passing Grain Size Distribution





Sample	Grain Size (µm)	Spodumene	Petalite/ Cookeite	Beryl/ Tourmaline	Quartz	Plagioclase	K-Feldspars	Muscovite	Other Silicates	Garnet	Carbonates	Phosphates	Fe-(Ti)-Oxides	Nb-Ta-Oxides	Sulphides	Other	Particle
Li-head	Minimum	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
	P10	72	8	10	69	66	54	21	10	89	22	60	9	41	11	10	67
	P20	124	10	11	117	100	90	55	12	143	43	307	15	50	19	13	107
	P30	178	11	15	170	134	129	97	20	177	65	308	25	74	25	21	146
	P40	231	14	23	226	172	178	137	32	211	93	308	34	124	29	38	192
	Median	294	22	46	280	215	228	179	53	260	120	308	44	124	40	62	240
	P60	359	43	83	335	259	275	229	77	275	142	308	52	144	42	94	290
	P70	413	81	181	394	310	333	272	108	304	174	308	62	144	57	99	347
	P80	**491**	**140**	**259**	**476**	**370**	**401**	**336**	**161**	**333**	**231**	**308**	**79**	**144**	**63**	**126**	**417**
	P90	639	253	325	591	467	528	407	312	386	340	308	176	144	91	127	533
	Maximum	1263	270	419	1103	1083	973	642	312	578	451	308	176	144	91	127	1282
	Mean	34	9	10	29	35	28	17	10	23	19	19	10	22	11	10	34
Rb-head	Minimum	3	3	3	3	3	3	3	3	3	3	6	3	9	3	3	3
	P10	135	7	7	134	96	114	15	10	116	17	14	5	12	17	6	119
	P20	283	8	8	227	158	206	25	31	149	31	22	5	21	119	8	200
	P30	398	9	10	291	217	285	48	90	227	51	29	5	21	232	10	274
	P40	490	10	11	367	274	357	84	184	304	62	34	6	21	232	19	343
	Median	515	10	14	453	335	422	112	475	304	77	38	6	25	232	23	412
	P60	660	10	18	520	391	498	144	475	457	146	38	7	25	269	28	485
	P70	809	11	66	593	463	612	177	476	457	149	50	8	27	269	36	583
	P80	**810**	**14**	**84**	**684**	**544**	**723**	**221**	**476**	**457**	**161**	**53**	**9**	**27**	**269**	**42**	**708**
	P90	819	19	127	860	701	896	300	476	458	175	53	12	27	270	59	902
	Maximum	819	60	127	1220	1154	1639	335	476	457	175	53	27	27	269	59	1663
	Mean	24	8	8	21	27	30	14	9	19	18	18	5	17	6	7	29

calcite is the main carbonate mineral



Sulphides include pyrite, pyrrhotite and sphalerite; rare other sulphides

Appendix D – EPMA & LA-ICP-MS Data

SAMPLE	Mineral	No.	REL. LINE	SiO2	TiO2	ZnO	Al2O3	V2O3	Cr2O3	Sc2O3	FeO	MnO	BeO	MgO	CaO	SrO	BaO	Li2O	Na2O	K2O	Rb2O	Cs2O	Cl	F	O	H2O	TOTAL
Rb-Head	KSP	371	3	63.90	0.01	-	18.20	0.01	-0.03	-	0.02	0.00	-	-0.02	0.01	-0.00	-0.06	-	0.29	15.47	1.50	0.14	0.02	-0.04	0.01	-	99.4278
Rb-Head	KSP	372	4	64.55	0.04	-	18.06	-0.00	-0.08	-	0.02	0.02	-	-0.02	0.01	0.06	-0.03	-	0.65	15.17	1.28	0.08	-0.00	0.05	-0.02	-	99.8318
Rb-Head	KSP	373	5	65.03	0.04	-	18.29	-0.03	-0.05	-	-0.00	0.01	-	-0.01	0.02	0.00	0.01	-	0.47	15.29	1.64	0.06	0.02	0.03	-0.02	-	100.798
Rb-Head	KSP	374	6	64.93	-0.01	-	18.21	0.01	-0.07	-	0.01	-0.02	-	-0.00	0.01	0.09	0.11	-	0.22	15.65	1.64	0.07	0.00	-0.06	0.03	-	100.804
Rb-Head 2	KSP	377	3	64.55	0.10	-	17.97	0.01	-0.03	-	-0.00	0.01	-	-0.00	0.00	0.04	0.08	-	0.23	15.83	1.45	0.08	0.00	-0.01	0.00	-	100.309
Rb-Head 2	KSP	378	4	65.04	0.02	-	18.01	0.01	-0.02	-	-0.00	-0.02	-	-0.01	-	0.09	-0.03	-	0.33	15.13	2.20	0.11	0.00	-0.03	0.01	-	100.854
Rb-Head 2	KSP	379	5	64.82	0.03	-	18.10	0.00	-0.07	-	-0.03	-0.00	-	0.00	-0.00	0.06	0.11	-	0.35	15.36	1.99	0.27	-0.00	-0.03	0.01	-	100.951
Rb-Head 2	KSP	383	9	65.38	0.03	-	18.17	-0.01	-0.02	-	-0.02	-0.01	-	-0.00	0.00	0.03	0.03	-	0.55	15.45	1.32	0.03	-0.00	0.07	-0.03	-	100.992
Rb-Head 2	KSP	384	10	65.49	0.01	-	18.21	0.01	-0.04	-	0.01	-0.00	-	-0.00	-0.01	0.05	0.07	-	0.61	15.31	1.20	0.03	-0.00	0.05	-0.02	-	100.993
Rb-Head 2	KSP	390	16	65.47	-0.03	-	18.18	0.02	-0.05	-	-0.01	0.02	-	-0.01	0.10	0.07	-0.03	-	0.08	16.28	0.80	-0.03	0.00	-0.06	0.02	-	100.817
Rb-Head 2	KSP	391	17	65.42	-0.07	-	18.23	-0.01	-0.03	-	0.00	-0.01	-	-0.01	-0.00	0.04	0.01	-	0.56	15.42	1.39	0.08	-0.00	-0.06	0.02	-	100.983
Rb-Head 2	KSP	394	20	65.74	0.03	-	18.45	-0.04	-0.05	-	-0.01	-0.01	-	-0.01	0.01	0.05	0.10	-	0.18	16.56	0.14	0.00	0.01	0.05	-0.02	-	101.188
Rb-Head 2	KSP	395	21	65.49	-0.01	-	18.13	-0.01	-0.08	-	0.04	0.02	-	0.00	-0.00	0.06	0.14	-	0.60	14.91	2.37	0.09	-0.01	0.03	-0.01	-	101.782
Rb-Head 2	KSP	396	22	65.95	0.04	-	18.39	0.02	-0.04	-	0.09	-0.01	-	0.00	-0.00	0.04	0.13	-	0.13	16.49	0.13	0.00	0.00	0.06	-0.03	-	101.384
Rb-Head 2	KSP	397	23	65.50	0.06	-	18.32	-0.03	-0.07	-	0.00	0.01	-	-0.02	-0.00	0.10	0.11	-	0.26	15.71	1.67	0.05	0.00	-0.07	0.03	-	101.64
Rb-Head 2	KSP	398	24	65.13	-0.02	-	18.29	0.02	-0.02	-	0.04	-0.02	-	-0.01	0.01	0.04	-0.01	-	0.20	16.43	0.13	0.02	0.01	-0.11	0.04	-	100.172
Rb-Head 2	KSP	399	25	65.30	-0.04	-	18.22	-0.01	-0.05	-	-0.02	-0.00	-	-0.00	-0.00	0.09	-0.01	-	0.59	14.73	2.39	0.15	-0.00	-0.00	0.00	-	101.338
Rb-Head 2	KSP	407	33	65.41	-0.02	-	18.44	-0.00	-0.08	-	-0.01	0.00	-	-0.00	0.02	0.03	-0.07	-	0.62	15.23	1.70	0.06	-0.00	-0.15	0.06	-	101.234
Rb-Head 2	KSP	408	34	65.87	-0.02	-	18.46	0.02	-0.03	-	-0.01	0.02	-	-0.00	-0.01	0.06	-0.13	-	0.45	15.63	1.37	0.04	0.00	0.04	-0.02	-	101.722
Rb-Head 2	KSP	411	37	65.63	-0.03	-	18.46	0.01	-0.01	-	0.01	0.01	-	-0.02	-0.00	0.05	-0.01	-	0.44	15.36	1.60	0.19	-0.00	-0.19	0.08	-	101.57
Rb-Head 2	KSP	412	38	65.43	-0.01	-	18.43	-0.01	-0.07	-	0.05	0.02	-	-0.01	0.01	0.04	-0.02	-	0.29	15.72	1.58	0.11	-0.00	-0.03	0.01	-	101.541
Rb-Head 2	KSP	413	39	65.52	-0.05	-	18.44	0.01	-0.07	-	0.03	-0.01	-	0.00	0.01	0.03	0.01	-	0.17	16.18	0.91	0.05	0.00	0.09	-0.04	-	101.29
Rb-Head 2	KSP	414	40	65.62	0.00	-	18.44	0.03	-0.04	-	-0.01	0.01	-	-0.00	-0.00	0.05	-0.02	-	0.65	15.26	1.41	0.03	0.00	-0.04	0.02	-	101.402
Rb-Head 2	KSP	415	41	65.76	-0.03	-	18.61	0.00	-0.06	-	0.02	0.01	-	-0.01	-0.00	0.03	0.05	-	0.16	16.61	0.06	-0.03	0.00	0.03	-0.01	-	101.193
Rb-Head 2	KSP	416	42	64.83	-0.00	-	18.41	-0.00	-0.06	-	-0.03	0.02	-	-0.01	0.00	0.06	0.01	-	0.52	15.06	2.11	0.08	-0.00	-0.06	0.02	-	100.955
Rb-Head 2	KSP	419	45	66.01	0.01	-	18.40	0.01	-0.07	-	-0.01	0.00	-	-0.01	0.00	0.03	-0.03	-	0.45	15.43	1.58	0.11	0.00	0.04	-0.02	-	101.927
Rb-Head 2	KSP	420	46	64.58	-0.06	-	18.49	-0.00	-0.07	-	-	0.02	-	-0.00	0.01	0.07	0.13	-	0.53	15.23	1.60	0.04	0.00	-0.02	0.01	-	100.569
Rb-Head 2	KSP	429	55	66.17	-0.01	-	18.71	0.02	-0.07	-	-0.00	0.01	-	0.00	0.03	0.04	0.03	-	1.33	14.32	1.09	0.05	-0.00	-0.09	0.04	-	101.661
Rb-Head R 1	KSP	435	4	63.09	0.03	-	18.56	0.01	-0.08	-	0.11	0.02	-	0.04	0.02	0.02	0.17	-	0.37	15.82	0.72	0.01	0.00	-0.02	0.01	-	98.8841
Rb-Head R 1	KSP	437	6	64.88	-0.04	-	18.32	-0.02	-0.07	-	-0.01	-0.01	-	-0.01	0.00	0.03	0.03	-	0.37	15.58	1.33	-0.01	0.00	0.01	-0.00	-	100.371
Rb-Head R 1	KSP	438	7	65.90	0.02	-	18.38	-0.04	-0.07	-	-0.00	-0.02	-	-0.01	0.01	0.03	0.05	-	0.38	15.55	1.27	0.06	0.00	-0.12	0.05	-	101.451
Rb-Head R 1	KSP	441	10	63.30	-0.01	-	18.41	0.01	-0.06	-	0.02	0.01	-	-0.01	-0.00	0.03	-0.06	-	0.64	14.91	2.13	0.16	-0.00	0.10	-0.04	-	99.5404
Rb-Head R 1	KSP	442	11	65.44	-0.03	-	18.40	-0.01	-0.06	-	0.01	-0.01	-	0.01	0.01	0.06	-0.02	-	0.16	16.55	0.01	0.01	0.03	0.11	-0.05	-	100.604
Rb-Head R 1	KSP	443	12	65.05	0.02	-	18.25	-0.00	-0.08	-	0.00	-0.02	-	-0.00	0.00	0.07	0.01	-	0.53	14.99	2.13	0.14	-0.01	0.00	0.00	-	101.072
Rb-Head R 1	KSP	451	20	64.31	0.02	-	18.29	0.02	0.01	-	0.01	-0.00	-	0.00	-0.01	0.04	0.09	-	0.27	15.90	1.13	0.05	0.00	0.06	-0.02	-	100.151
Rb-Head R 1	KSP	452	21	64.80	-0.01	-	18.40	0.01	-0.04	-	0.00	0.01	-	-0.01	0.00	0.08	-0.00	-	0.23	15.47	1.68	0.14	0.00	-0.16	0.07	-	100.68
Rb-Head R 1	KSP	460	29	65.46	0.01	-	18.43	-0.01	-0.05	-	0.01	-0.02	-	-0.00	0.01	0.03	-0.07	-	0.72	15.40	1.09	0.04	-0.01	0.03	-0.01	-	101.048
Rb-Head R 1	KSP	463	32	64.00	0.04	-	18.28	-0.03	-0.08	-	0.03	0.03	-	-0.00	0.46	0.07	0.05	-	0.21	15.73	1.44	0.02	0.00	0.07	-0.03	-	100.289
Rb-Head R 1	KSP	466	35	63.89	-0.06	-	18.14	-0.02	-0.05	-	0.07	0.02	-	-0.01	0.00	0.05	0.01	-	0.45	15.18	1.71	0.12	0.01	0.03	-0.01	-	99.5204
Rb-Head R 1	KSP	469	38	65.15	0.03	-	18.49	0.02	-0.04	-	0.01	0.00	-	-0.01	0.00	0.06	-0.01	-	0.59	15.23	1.34	0.08	-0.00	-0.08	0.04	-	100.907
Rb-Head R 1	KSP	472	41	65.02	-0.01	-	18.37	0.00	-0.04	-	0.03	0.02	-	-0.01	0.00	0.04	-0.01	-	0.33	15.62	1.31	0.08	0.00	-0.12	0.05	-	100.689
Rb-Head R 1	KSP	474	43	64.91	-0.01	-	18.52	0.01	-0.05	-	0.01	0.01	-	-0.00	0.00	0.04	0.05	-	0.66	15.15	1.35	0.11	0.00	-0.04	0.02	-	100.747
Rb-Head R 1	KSP	475	44	64.07	-0.01	-	18.27	0.01	-0.08	-	0.00	0.02	-	0.00	-0.01	0.07	0.12	-	0.71	14.65	1.95	0.09	-0.00	0.12	-0.05	-	99.9378
Rb-Head R 1	KSP	478	47	64.03	0.00	-	18.29	-0.04	-0.04	-	0.02	0.00	-	0.01	0.01	0.04	-0.03	-	0.61	15.17	1.14	0.07	-0.00	-0.01	0.00	-	99.2772
Rb-Head R 1	KSP	479	48	65.16	0.00	-	18.54	-0.02	-0.03	-	-0.00	-0.01	-	-0.01	0.05	0.07	-0.05	-	0.61	15.30	1.12	0.10	-0.00	-0.02	0.01	-	100.838
Rb-Head R 1	KSP	483	52	63.85	-0.02	-	18.39	0.04	-0.04	-	-0.01	0.01	-	-0.00	0.01	0.06	-0.07	-	0.40	15.12	2.11	0.11	0.00	0.12	-0.05	-	100.024
Rb-Head R 1	KSP	484	53	64.20	0.04	-	18.33	-0.00	-0.03	-	0.01	-0.01	-	-0.01	-0.01	0.10	0.01	-	0.60	14.71	2.32	0.16	-	0.01	-0.00	-	100.42
Rb-Head R 1	KSP	487	56	64.72	0.02	-	18.32	0.01	-0.05	-	0.02	0.03	-	-0.00	0.00	0.06	0.09	-	0.28	15.43	1.84	0.10	-0.00	0.06	-0.02	-	100.895
Rb-Head R 1	KSP	488	57	63.82	-0.06	-	18.24	0.01	-0.07	-	0.01	0.00	-	0.00	0.01	0.06	0.04	-	0.42	15.12	1.79	0.03	0.00	-0.13	0.05	-	99.3491
Rb-Head R 1	KSP	489	58	65.36	0.01	-	18.35	-0.01	-0.04	-	0.04	-0.02	-	0.00	0.01	0.04	-0.07	-	0.18	16.06	1.07	0.09	0.01	0.06	-0.03	-	101.119
Rb-Head R 1	KSP	492	61	62.94	0.01	-	18.49	0.00	-0.06	-	-0.01	0.00	-	-0.01	0.01	0.06	0.01	-	0.55	14.94	2.06	0.11	-0.00	0.12	-0.05	-	99.1683
			Average	64.94	0.00	0.00	18.34	0.00	-0.05	0.00	0.01	0.00	0.00	0.00	0.02	0.05	0.02	0.00	0.43	15.47	1.42	0.07	0.00	-0.01	0.00	0.00	100.71
			Std.Dev.	0.77	0.03	0.00	0.15	0.02	0.02	0.00	0.03	0.01	0.00	0.01	0.07	0.02	0.07	0.00	0.22	0.52	0.60	0.06	0.01	0.08	0.03	0.00	0.75
			Std. Err.	0.11	0.00	0.00	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.01	0.00	0.03	0.07	0.08	0.01	0.00	0.01	0.00	0.00	0.10
			Minimum	62.942	-0.070	0.000	17.972	-0.044	-0.085	0.000	-0.028	-0.025	0.000	-0.021	-0.014	-0.004	-0.132	0.000	0.076	14.319	0.014	-0.033	-0.005	-0.195	-0.054	0.000	98.884
			Maximum	66.173	0.103	0.000	18.715	0.037	0.008	0.000	0.105	0.032	0.000	0.043	0.456	0.098	0.172	0.000	1.327	16.609	2.391	0.267	0.034	0.123	0.082	0.000	101.927

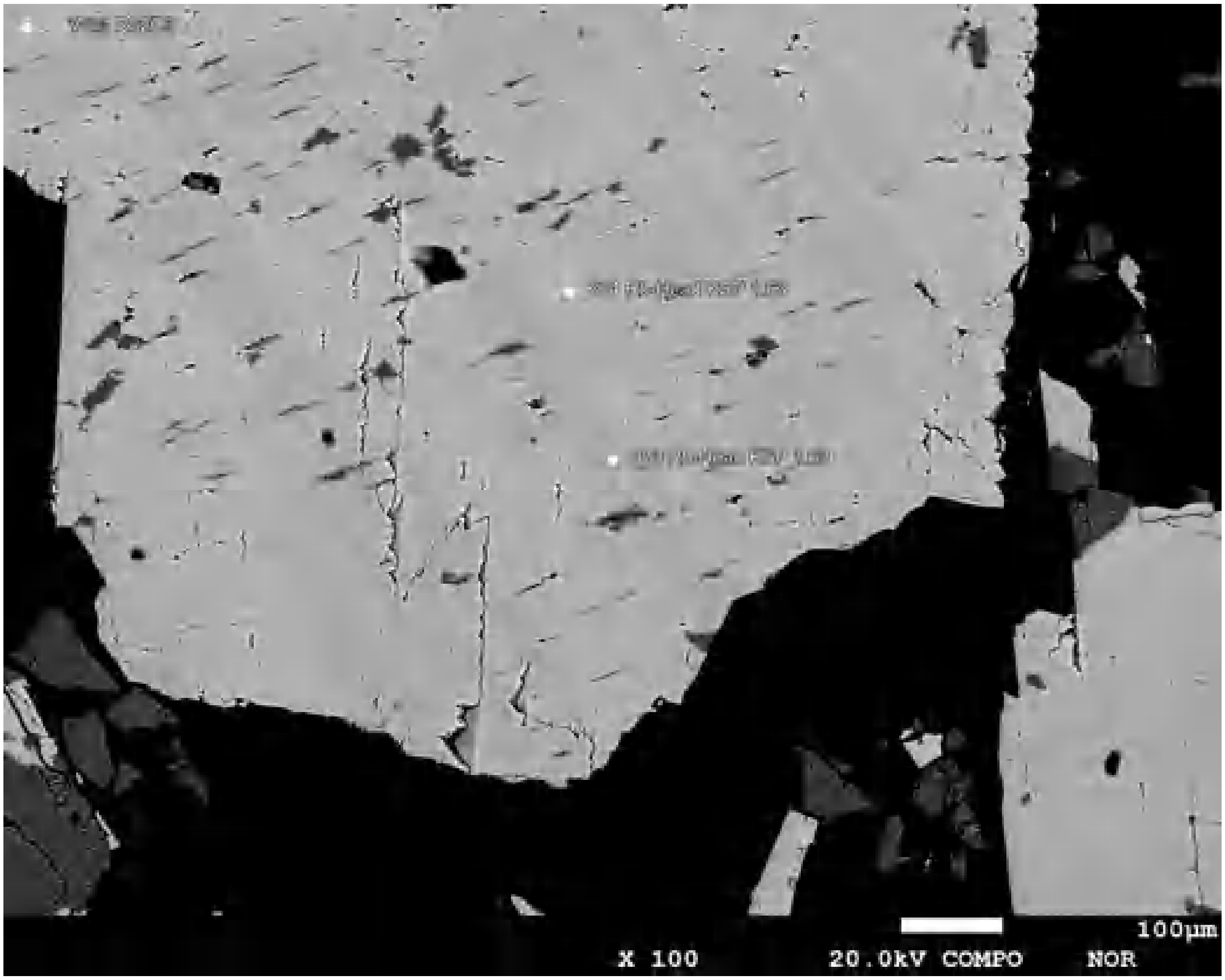
SAMPLE	Mineral	No.	REL. LINE	SiO2	TiO2	ZnO	Al2O3	V2O3	Cr2O3	Sc2O3	FeO	MnO	BeO	MgO	CaO	SrO	BaO	Li2O	Na2O	K2O	Rb2O	Cs2O	Cl	F	O	H2O	TOTAL
Rb-Head	Mica	370	2	43.60	0.11	0.00	32.02	-0.03	-0.06	0.00	4.61	0.16	0.00	0.32	0.01	0.06	0.01	0.00	0.24	10.19	1.35	0.06	0.00	0.28	-0.12	4.36	97.18
Rb-Head 2	Mica	375	1	45.90	0.00	0.00	35.63	-0.01	-0.06	0.00	2.42	0.14	0.00	0.07	0.02	0.04	-0.01	0.00	0.19	10.26	1.86	0.09	0.00	0.07	-0.03	4.43	101.02
Rb-Head 2	Mica	376	2	44.73	0.16	0.00	32.59	0.01	-0.06	0.00	4.54	0.15	0.00	0.28	0.01	0.05	0.03	0.00	0.20	10.10	1.86	0.20	0.01	0.18	-0.08	4.36	99.32
Rb-Head 2	Mica	380	6	45.03	0.25	0.00	32.45	0.04	-0.06	0.00	3.28	0.19	0.00	0.85	-0.01	0.07	0.02	0.00	0.22	9.98	2.12	0.14	0.01	0.81	-0.34	4.34	99.39
Rb-Head 2	Mica	381	7	45.33	0.31	0.00	31.93	0.03	-0.10	0.00	3.42	0.27	0.00	0.95	0.00	0.05	-0.07	0.00	0.22	9.84	2.18	0.34	0.00	0.74	-0.31	4.33	99.49
Rb-Head 2	Mica	382	8	44.61	0.01	0.00	34.85	0.01	-0.03	0.00	2.68	0.15	0.00	0.05	0.01	0.07	-0.09	0.00	0.53	10.02	1.21	0.07	0.00	0.23	-0.10	4.43	98.74
Rb-Head 2	Mica	392	18	44.65	0.16	0.00	32.13	0.03	-0.05	0.00	3.40	0.24	0.00	0.80	0.00	0.04	-0.01	0.00	0.19	10.27	1.82	0.23	-0.01	0.69	-0.29	4.34	98.62
Rb-Head 2	Mica	393	19	45.35	0.42	0.00	32.04	0.02	-0.05	0.00	3.59	0.28	0.00	0.88	0.03	0.05	0.16	0.00	0.25	10.08	1.85	0.06	0.00	0.81	-0.34	4.33	99.79
Rb-Head 2	Mica	400	26	44.44	0.28	0.00	32.81	0.04	-0.07	0.00	3.53	0.17	0.00	0.42	0.01	0.06	0.03	0.00	0.21	10.03	1.84	0.31	0.00	0.20	-0.08	4.37	98.58
Rb-Head 2	Mica	401	27	44.63	0.26	0.00	32.76	0.00	-0.07	0.00	3.68	0.13	0.00	0.57	0.00	0.08	-0.10	0.00	0.27	9.75	2.15	0.54	0.00	0.31	-0.13	4.35	99.18
Rb-Head 2	Mica	402	28	44.34	0.16	0.00	33.02	-0.01	-0.04	0.00	3.79	0.20	0.00	0.55	0.01	0.04	-0.05	0.00	0.31	9.61	2.12	0.51	0.00	0.34	-0.15	4.35	99.10
Rb-Head 2	Mica	403	29	45.99	0.08	0.00	34.15	0.01	-0.05	0.00	2.34	0.05	0.00	0.39	0.04	0.07	0.09	0.00	0.08	9.83	1.64	0.08	0.00	0.07	-0.03	4.45	99.27
Rb-Head 2	Mica	404	30	44.46	0.25	0.00	33.12	0.01	-0.06	0.00	3.43	0.15	0.00	0.43	-0.01	0.05	0.01	0.00	0.22	9.98	1.91	0.39	0.00	0.36	-0.15	4.36	98.91
Rb-Head 2	Mica	405	31	45.72	0.14	0.00	34.36	0.00	-0.10	0.00	3.19	0.16	0.00	0.34	0.00	0.04	-0.02	0.00	0.24	10.05	1.65	0.14	0.00	0.12	-0.05	4.41	100.41
Rb-Head 2	Mica	406	32	46.05	0.21	0.00	34.21	0.00	-0.05	0.00	2.76	0.24	0.00	0.66	0.01	0.08	-0.03	0.00	0.27	9.81	2.15	0.24	0.00	0.48	-0.20	4.38	101.28
Rb-Head 2	Mica	409	35	45.75	0.02	0.00	36.70	-0.02	-0.07	0.00	1.73	0.12	0.00	0.00	-0.01	0.03	-0.02	0.00	0.24	10.37	1.76	0.12	0.00	-0.09	0.04	4.46	101.14
Rb-Head 2	Mica	410	36	45.93	0.07	0.00	32.24	-0.01	-0.04	0.00	4.38	0.15	0.00	0.29	0.00	0.05	0.03	0.00	0.13	10.03	2.43	0.12	0.00	0.27	-0.12	4.35	100.30
Rb-Head 2	Mica	417	43	46.50	0.43	0.00	34.04	0.00	-0.09	0.00	2.94	0.23	0.00	0.84	0.00	0.06	-0.01	0.00	0.29	10.00	1.91	0.18	0.00	0.60	-0.25	4.37	102.04
Rb-Head 2	Mica	418	44	46.36	0.38	0.00	33.88	0.00	-0.06	0.00	2.99	0.24	0.00	0.77	0.05	0.08	0.07	0.00	0.25	9.98	1.91	0.22	0.00	0.59	-0.25	4.37	101.84
Rb-Head 2	Mica	421	47	45.39	0.39	0.00	33.70	0.00	-0.03	0.00	2.76	0.21	0.00	0.68	0.01	0.08	-0.01	0.00	0.23	9.97	2.01	0.35	0.01	0.52	-0.22	4.37	100.42
Rb-Head 2	Mica	422	48	45.22	0.24	0.00	33.23	0.04	-0.04	0.00	2.91	0.22	0.00	0.73	0.02	0.06	0.07	0.00	0.17	9.77	2.27	0.33	0.00	0.78	-0.33	4.34	100.06
Rb-Head 2	Mica	423	49	44.70	0.16	0.00	31.87	-0.02	-0.07	0.00	5.30	0.18	0.00	0.25	0.04	0.04	-0.04	0.00	0.36	10.09	1.39	0.07	0.01	0.39	-0.16	4.35	98.86
Rb-Head 2	Mica	424	50	44.35	0.37	0.00	31.13	0.04	-0.06	0.00	5.75	0.16	0.00	0.25	0.01	0.06	0.06	0.00	0.22	9.89	1.71	0.18	0.01	0.56	-0.24	4.31	98.76
Rb-Head 2	Mica	425	51	44.77	0.14	0.00	33.87	0.03	-0.03	0.00	4.54	0.08	0.00	0.24	0.03	0.07	-0.02	0.00	0.34	10.27	1.28	0.04	0.00	0.11	-0.05	4.39	100.10
Rb-Head 2	Mica	426	52	44.20	0.19	0.00	33.20	0.04	-0.06	0.00	4.72	0.09	0.00	0.23	0.01	0.03	-0.03	0.00	0.34	10.22	1.38	0.04	0.01	0.24	-0.10	4.37	99.11
Rb-Head 2	Mica	427	53	45.03	0.23	0.00	35.35	-0.02	-0.08	0.00	2.75	0.11	0.00	0.08	0.05	0.03	0.03	0.00	0.35	10.18	1.49	0.02	0.01	0.02	-0.01	4.43	100.05
Rb-Head 2	Mica	428	54	44.97	0.22	0.00	32.53	0.03	-0.03	0.00	4.75	0.27	0.00	0.22	0.02	0.04	-0.05	0.00	0.18	9.80	2.30	0.08	0.00	0.10	-0.04	4.36	99.73
Rb-Head 2	Mica	430	56	43.81	0.37	0.00	29.64	0.01	-0.05	0.00	6.65	0.19	0.00	0.57	0.02	0.06	0.23	0.00	0.12	9.32	2.41	0.84	0.00	0.48	-0.20	4.24	98.73
Rb-Head 2	Mica	431	57	43.85	0.68	0.00	30.24	0.06	-0.08	0.00	5.94	0.17	0.00	0.53	0.02	0.01	0.14	0.00	0.12	9.69	2.08	0.39	0.00	0.32	-0.14	4.30	98.32
Rb-Head R 1	Mica	432	1	45.61	0.09	0.00	33.18	0.01	-0.05	0.00	3.64	0.28	0.00	0.11	-0.01	0.07	-0.01	0.00	0.18	9.91	2.40	0.17	0.00	0.20	-0.08	4.37	100.08
Rb-Head R 1	Mica	433	2	46.09	0.13	0.00	33.38	0.03	-0.08	0.00	3.53	0.15	0.00	0.18	0.00	0.05	-0.07	0.00	0.10	9.88	2.87	0.12	0.00	0.17	-0.07	4.37	100.84
Rb-Head R 1	Mica	434	3	53.20	-0.02	0.00	28.06	0.03	-0.03	0.00	2.54	0.13	0.00	2.96	0.12	0.01	0.08	0.00	0.03	10.21	0.50	0.03	0.01	0.08	-0.03	4.50	102.41
Rb-Head R 1	Mica	436	5	45.17	0.02	0.00	36.86	0.00	-0.06	0.00	1.34	0.28	0.00	0.01	0.02	0.07	-0.05	0.00	0.18	10.63	1.23	0.05	0.00	0.12	-0.05	4.46	100.28
Rb-Head R 1	Mica	444	13	44.58	0.22	0.00	32.02	0.01	-0.08	0.00	3.61	0.25	0.00	0.90	-0.01	0.06	0.04	0.00	0.25	9.93	2.23	0.23	0.00	0.83	-0.35	4.32	99.04
Rb-Head R 1	Mica	445	14	44.56	0.22	0.00	31.60	0.00	-0.06	0.00	3.75	0.26	0.00	0.96	-0.01	0.08	0.03	0.00	0.28	9.97	1.71	0.14	0.00	0.86	-0.36	4.33	98.32
Rb-Head R 1	Mica	446	15	45.32	0.44	0.00	32.27	0.00	-0.05	0.00	3.60	0.26	0.00	0.96	0.03	0.05	0.03	0.00	0.28	9.99	1.99	0.15	0.01	0.35	-0.15	4.36	99.88
Rb-Head R 1	Mica	447	16	45.33	0.40	0.00	32.30	0.05	-0.04	0.00	3.67	0.26	0.00	0.98	0.02	0.05	0.10	0.00	0.29	9.87	1.97	0.12	0.01	0.27	-0.12	4.37	99.90
Rb-Head R 1	Mica	448	17	45.45	0.30	0.00	32.32	0.06	-0.06	0.00	3.74	0.31	0.00	0.93	0.01	0.05	-0.01	0.00	0.26	9.91	2.08	0.19	0.00	0.88	-0.37	4.32	100.37
Rb-Head R 1	Mica	449	18	43.76	0.23	0.00	31.95	0.01	-0.08	0.00	3.36	0.24	0.00	0.97	0.01	0.01	0.10	0.00	0.18	10.13	1.80	0.16	0.00	0.96	-0.40	4.32	97.70
Rb-Head R 1	Mica	450	19	45.03	0.31	0.00	32.17	0.02	-0.08	0.00	3.32	0.28	0.00	0.95	0.02	0.02	0.11	0.00	0.17	10.13	1.71	0.11	0.00	0.78	-0.33	4.35	99.07
Rb-Head R 1	Mica	453	22	43.93	0.30	0.00	34.78	0.02	-0.06	0.00	2.67	0.13	0.00	0.07	0.02	0.03	0.03	0.00	0.31	9.79	1.83	0.39	0.00	0.27	-0.11	4.39	98.78
Rb-Head R 1	Mica	454	23	44.88	0.15	0.00	35.99	0.00	-0.05	0.00	2.43	0.16	0.00	0.11	0.01	0.03	-0.01	0.00	0.46	10.19	1.31	0.05	0.00	0.22	-0.09	4.43	100.28
Rb-Head R 1	Mica	455	24	44.37	0.29	0.00	32.18	0.05	-0.06	0.00	3.03	0.23	0.00	0.76	0.00	0.06	0.09	0.00	0.16	9.87	2.29	0.36	0.01	0.52	-0.22	4.34	98.32
Rb-Head R 1	Mica	456	25	46.17	0.29	0.00	33.20	0.02	-0.09	0.00	3.15	0.24	0.00	0.80	0.01	0.06	-0.05	0.00	0.20	10.27	1.70	0.15	0.00	0.71	-0.30	4.36	100.91
Rb-Head R 1	Mica	457	26	46.94	-0.02	0.00	31.47	0.06	-0.06	0.00	3.85	0.38	0.00	0.27	0.00	0.03	-0.04	0.00	0.10	9.71	2.77	0.13	0.00	0.32	-0.13	4.36	100.13
Rb-Head R 1	Mica	458	27	47.53	0.01	0.00	31.47	0.00	-0.10	0.00	3.14	0.39	0.00	0.27	0.03	0.08	0.05	0.00	0.28	9.13	2.84	0.16	0.01	0.24	-0.10	4.39	99.82
Rb-Head R 1	Mica	459	28	45.11	0.00	0.00	37.16	0.04	-0.03	0.00	1.00	0.08	0.00	0.02	0.24	0.04	0.04	0.00	0.20	10.49	1.17	-0.03	0.01	0.03	-0.02	4.48	100.05
Rb-Head R 1	Mica	461	30	46.81	0.02	0.00	37.35	-0.02	-0.05	0.00	1.57	0.18	0.00	0.04	0.03	0.01	-0.05	0.00	0.14	10.47	1.80	0.07	0.00	0.06	-0.03	4.45	102.87
Rb-Head R 1	Mica	464	33	44.87	0.14	0.00	33.04	0.04	-0.06	0.00	4.81	0.17	0.00	0.20	0.00	0.07	0.10	0.00	0.33	9.99	1.53	0.08	0.00	0.25	-0.11	4.36	99.81
Rb-Head R 1	Mica	465	34	44.67	0.03	0.00	34.84	0.03	-0.06	0.00	3.01	0.13	0.00	0.11	0.01	0.02	0.13	0.00	0.24	10.27	1.59	0.16	0.00	0.11	-0.04	4.41	99.65
Rb-Head R 1	Mica	467	36	45.12	0.29	0.00	32.46	-0.01	-0.05	0.00	3.40	0.24	0.00	0.85	0.02	0.08	-0.06	0.00	0.33	9.86	1.80	0.08	0.00	0.83	-0.35	4.35	99.24
Rb-Head R 1	Mica	468	37	45.32	0.25	0.00	32.33	-0.01	-0.08	0.00	3.55	0.30	0.00	0.88	0.00	0.08	0.05	0.00	0.15	9.82	2.11	0.40	0.00	0.91	-0.38	4.32	100.02
Rb-Head R 1	Mica	470	39	44.24	0.48	0.00	30.74	0.06	-0.05	0.00	5.83	0.14	0.00	0.23	0.00	0.06	0.03	0.00	0.13	10.11	1.89	0.21	0.01	0.43	-0.18	4.30	98.67
Rb-Head R 1	Mica	471	40	45.01	0.22	0.00	29.45	0.03	-0.04	0.00	5.86	0.17	0.00	0.28	0.01	0.08	0.01	0.00	0.12	9.70	2.35	0.40	0.01	0.30	-0.13	4.30	98.13
Rb-Head R 1	Mica	473	42	45.68	0.17	0.00	33.57	0.03	-0.06	0.00	3.11	0.22	0.00	0.78	0.00	0.02	0.03	0.00	0.41	9.87	1.85	0.11	0.00	0.77	-0.32	4.36	100.61
Rb-Head R 1	Mica	476	45	44.16	3.08	0.00	29.95	0.04	-0.11	0.00	4.45	0.13	0.00	0.27	0.01	0.05	-0.03	0.00	0.12	9.77	2.29	0.40	0.00	0.14	-0.06	4.33	98.98
Rb-Head R 1	Mica	477	46	44.57	0.13	0.00	32.26	-0.01	-0.06	0.00	4.77	0.14	0.00	0.29	0.01	0.06	0.05	0.00	0.16	10.07	1.96	0.26	0.01	0.26	-0.11	4.34	99.14
Rb-Head R 1	Mica	481	50	45.50	0.38	0.00	30.26	0.03	-0.06	0.00	5.82	0.23	0.00	0.75	0.03	0.07	0.03	0.00	0.13	9.67	2.41	0.07	0.00	0.38	-0.16	4.31	99.84
Rb-Head R 1	Mica	482	51	45.19	0.50	0.00	31.02	0.06	-0.06	0.00	5.28	0.18	0.00	0.68	0.02	0.00	-0.08	0.00	0.14	9.92	2.21	0.05	0.00	0.46	-0.19	4.32	99.69
Rb-Head R 1	Mica	485	54	44.82	0.07	0.00	32.99	-0.04	-0.04	0.00	5.03	0.19	0.00	0.26	0.03	0.06	0.01	0.00	0.28	10.02	1.88	0.20	0.01	0.21	-0.09	4.34	100.25
Rb-Head R 1	Mica	486	55	43.80	0.27	0.00	31.81	0.01	-0.03	0.00	4.90	0.12	0.00	0.23	0.01	0.01	-0.01	0.00	0.26	9.73	1.66	0.11	0.02	0.21	-0.09	4.36	97.38
		Average	30.02	45.22	0.26	0.00	32.85	0.02	-0.06	0.00	3.70	0.20	0.00	0.51	0.02	0.05	0.02	0.00	0.23	9.98	1.90	0.19	0.00	0.39	-0.16	4.37	99.68
		Std.Dev.	16.87	1.33	0.40	0.00	1.87	0.02	0.02	0.00	1.19	0.07	0.00	0.45	0.04	0.02	0.06	0.00	0.09	0.26	0.43	0.15	0.00	0.27	0.12	0.05	1.13
		Std. Err.	2.16	0.17	0.05	0.00	0.24	0.00	0.00	0.00	0.15	0.01	0.00	0.06	0.00	0.00	0.01	0.00	0.01	0.03	0.05	0.02	0.00	0.04	0.01	0.01	0.14
		Minimum	1.000	43.599	-0.019	0.000	28.062	-0.038	-0.113	0.000	1.004	0.046	0.000	0.000	-0.014	0.003	-0.105	0.000	0.033	9.125	0.502	-0.026	-0.006	-0.088	-0.405	4.245	97.183
		Maximum	57.000	53.202	3.082	0.000	37.346	0.064	-0.026	0.000	6.653	0.391	0.000	2.964	0.242	0.084	0.226	0.000	0.532	10.625	2.867	0.838	0.024	0.960	0.037	4.500	102.868

SAMPLE	Mineral	No.	REL. LINE	SiO2	TiO2	ZnO	Al2O3	V2O3	Cr2O3	Sc2O3	FeO	MnO	BeO	MgO	CaO	SrO	BaO	Li2O	Na2O	K2O	Rb2O	Cs2O	Cl	F	O	H2O	TOTAL
Rb-Head 2	Beryl	385	11	63.87	- 0.02	-	17.10	0.04	- 0.05	-	0.13	- 0.01	-	0.01	0.02	0.01	- 0.01	-	1.46	0.05	0.00	3.38	- 0.00	0.01	- 0.00	-	86.0002
Rb-Head 2	Beryl	386	12	64.34	0.03	-	17.23	0.02	- 0.01	-	0.15	0.00	-	0.04	0.01	0.02	- 0.07	-	1.70	0.06	- 0.00	1.32	0.00	- 0.15	0.06	-	84.7386
Rb-Head 2	Beryl	387	13	64.64	0.01	-	17.07	- 0.00	- 0.06	-	0.19	- 0.01	-	0.09	0.00	0.00	0.06	-	1.17	0.04	0.06	2.07	0.00	0.03	- 0.01	-	85.3601
			Average	64.29	0.01	0.00	17.13	0.02	-0.04	0.00	0.16	-0.01	0.00	0.04	0.01	0.01	0.00	0.00	1.44	0.05	0.02	2.25	0.00	-0.04	0.02	0.00	85.37
			Std.Dev.	0.39	0.02	0.00	0.08	0.02	0.02	0.00	0.03	0.01	0.00	0.04	0.01	0.01	0.06	0.00	0.26	0.01	0.04	1.04	0.00	0.10	0.04	0.00	0.63
			Std. Err.	0.22	0.01	0.00	0.05	0.01	0.01	0.00	0.02	0.00	0.00	0.02	0.00	0.00	0.04	0.00	0.15	0.00	0.02	0.60	0.00	0.06	0.02	0.00	0.36
			Minimum	63.872	-0.016	0.000	17.069	0.000	-0.056	0.000	0.132	-0.010	0.000	0.006	0.005	0.005	-0.065	0.000	1.173	0.044	-0.004	1.320	-0.001	-0.150	-0.012	0.000	84.739
			Maximum	64.641	0.027	0.000	17.227	0.039	-0.013	0.000	0.189	0.002	0.000	0.086	0.017	0.018	0.059	0.000	1.697	0.059	0.062	3.378	0.002	0.029	0.063	0.000	86.000

SAMPLE	Mineral	No.	REL. LINE	SiO2	TiO2	ZnO	Al2O3	V2O3	Cr2O3	Sc2O3	FeO	MnO	BeO	MgO	CaO	SrO	BaO	Li2O	Na2O	K2O	Rb2O	Cs2O	Cl	F	O	H2O	TOTAL
Rb-Head 2	Plagioclase	388	14	62.75	0.06	-	23.47	0.01	- 0.05	-	0.03	0.01	-	- 0.01	5.07	0.05	- 0.04	-	8.85	0.05	- 0.08	0.02	0.00	0.02	- 0.01	-	100.206
Rb-Head 2	Plagioclase	389	15	59.52	0.04	-	25.59	- 0.02	- 0.01	-	0.02	0.01	-	- 0.02	7.31	0.02	0.09	-	7.52	0.06	- 0.05	0.02	- 0.00	- 0.07	0.03	-	100.06
Rb-Head R 1	Albite	490	59	68.03	0.03	-	19.44	- 0.01	- 0.03	-	0.02	0.01	-	0.01	0.25	0.07	- 0.08	-	11.77	0.06	- 0.09	- 0.05	0.00	0.00	- 0.00	-	99.4076
Rb-Head R 1	Albite	493	62	68.84	- 0.02	-	19.85	0.02	0.00	-	0.01	0.00	-	- 0.00	0.24	0.03	0.02	-	11.90	0.13	- 0.14	- 0.00	- 0.00	- 0.10	0.04	-	100.809
			Average	64.79	0.03	0.00	22.09	0.00	-0.02	0.00	0.02	0.01	0.00	-0.01	3.22	0.04	0.00	0.00	10.01	0.08	-0.09	0.00	0.00	-0.03	0.01	0.00	100.12
			Std.Dev.	4.43	0.03	0.00	2.96	0.02	0.02	0.00	0.01	0.01	0.00	0.01	3.55	0.02	0.07	0.00	2.18	0.04	0.04	0.03	0.00	0.06	0.02	0.00	0.58
			Std. Err.	2.21	0.02	0.00	1.48	0.01	0.01	0.00	0.00	0.00	0.00	0.01	1.78	0.01	0.04	0.00	1.09	0.02	0.02	0.02	0.00	0.03	0.01	0.00	0.29
			Minimum	59.525	-0.021	0.000	19.438	-0.023	-0.049	0.000	0.011	0.001	0.000	-0.019	0.238	0.015	-0.084	0.000	7.519	0.050	-0.144	-0.050	-0.002	-0.097	-0.009	0.000	99.408
			Maximum	68.840	0.060	0.000	25.590	0.018	0.003	0.000	0.028	0.015	0.000	0.006	7.310	0.066	0.085	0.000	11.898	0.131	-0.050	0.020	0.002	0.021	0.041	0.000	100.809

Chlorite

SAMPLE	Mineral	No.	REL. LINE	SiO2	TiO2	ZnO	Al2O3	V2O3	Cr2O3	Sc2O3	FeO	MnO	BeO	MgO	CaO	SrO	BaO	Li2O	Na2O	K2O	Rb2O	Cs2O	Cl	F	O	H2O	TOTAL
Rb-Head R 1	Chlorite	439	8	32.46	0.03	-	23.20	0.04	- 0.09	-	21.39	1.26	-	6.54	0.26	0.02	- 0.01	-	0.04	1.86	0.16	0.01	0.03	- 0.16	0.06	-	87.1118
Rb-Head R 1	Chlorite	462	31	21.49	- 0.02	-	20.02	0.03	- 0.53	-	29.37	7.19	-	3.15	0.01	- 0.01	- 0.04	-	- 0.00	0.04	- 0.03	- 0.00	0.01	- 0.10	0.04	-	80.6108
Rb-Head R 1	Chlorite	491	60	33.85	0.04	-	21.38	0.03	- 0.12	-	17.90	0.85	-	11.55	0.10	0.03	0.07	-	0.02	2.70	0.14	- 0.01	0.01	0.01	- 0.01	-	88.5491
			Average	29.27	0.02	0.00	21.54	0.03	-0.25	0.00	22.89	3.10	0.00	7.08	0.12	0.01	0.01	0.00	0.02	1.53	0.09	0.00	0.01	-0.08	0.03	0.00	85.42
			Std.Dev.	6.77	0.03	0.00	1.60	0.01	0.25	0.00	5.88	3.55	0.00	4.23	0.12	0.02	0.05	0.00	0.02	1.36	0.10	0.01	0.01	0.09	0.03	0.00	4.23
			Std. Err.	3.91	0.02	0.00	0.92	0.00	0.14	0.00	3.39	2.05	0.00	2.44	0.07	0.01	0.03	0.00	0.01	0.78	0.06	0.00	0.01	0.05	0.02	0.00	2.44
			Minimum	21.489	-0.023	0.000	20.018	0.026	-0.533	0.000	17.899	0.853	0.000	3.146	0.014	-0.011	-0.037	0.000	0.000	0.045	-0.028	-0.009	0.006	-0.155	-0.008	0.000	80.611
			Maximum	33.854	0.041	0.000	23.205	0.041	-0.092	0.000	29.368	7.191	0.000	11.552	0.260	0.028	0.068	0.000	0.044	2.697	0.161	0.006	0.029	0.014	0.059	0.000	88.549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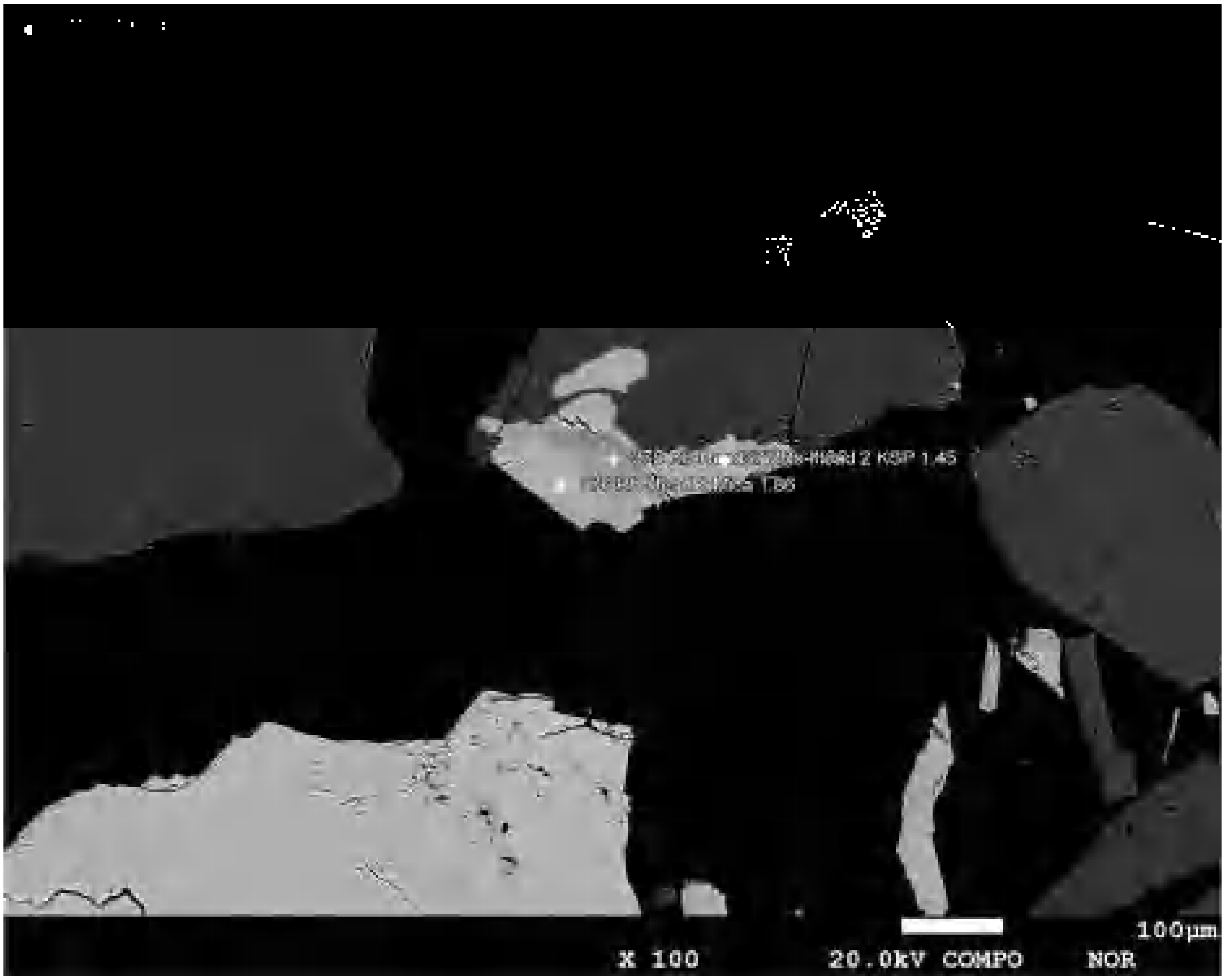
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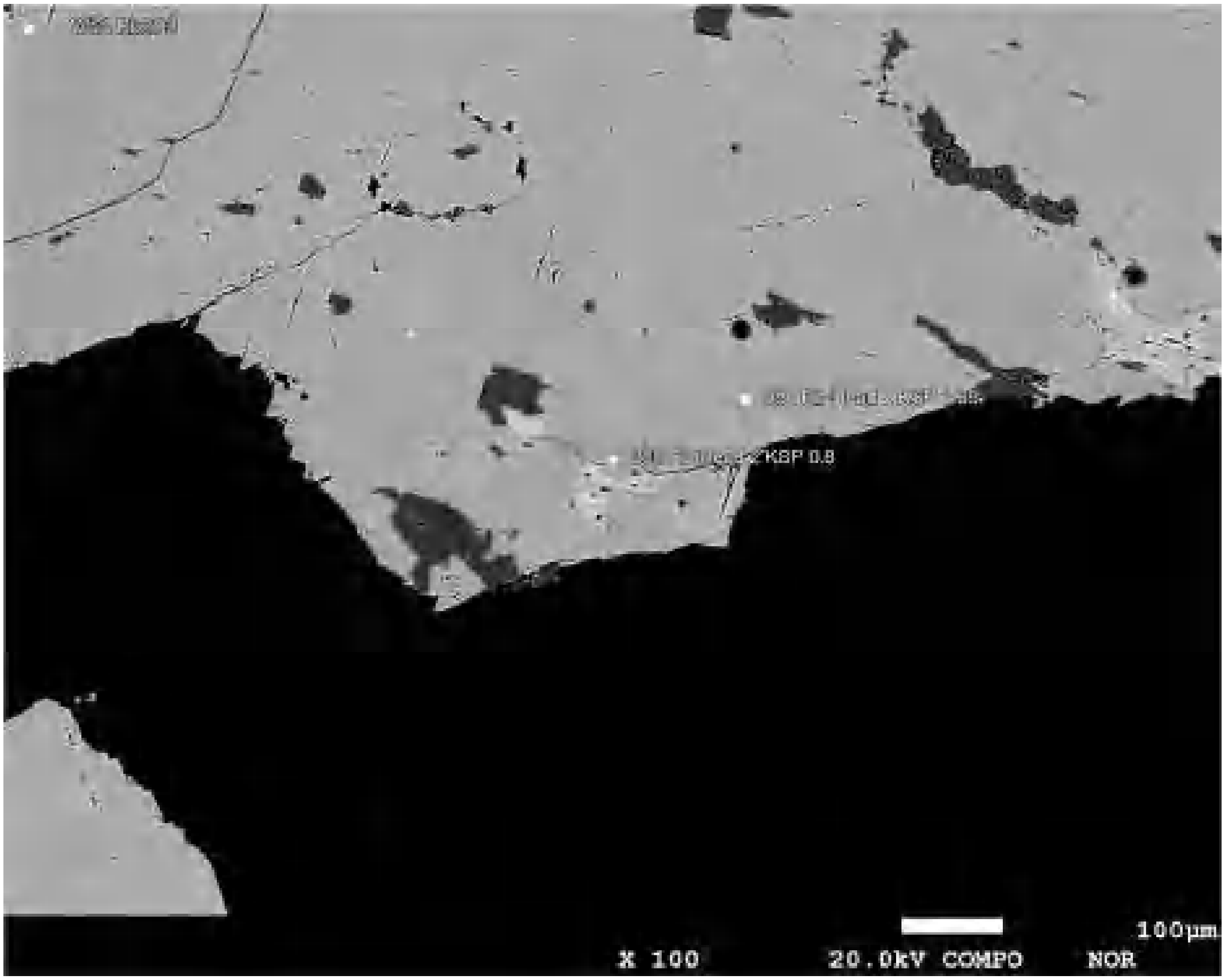
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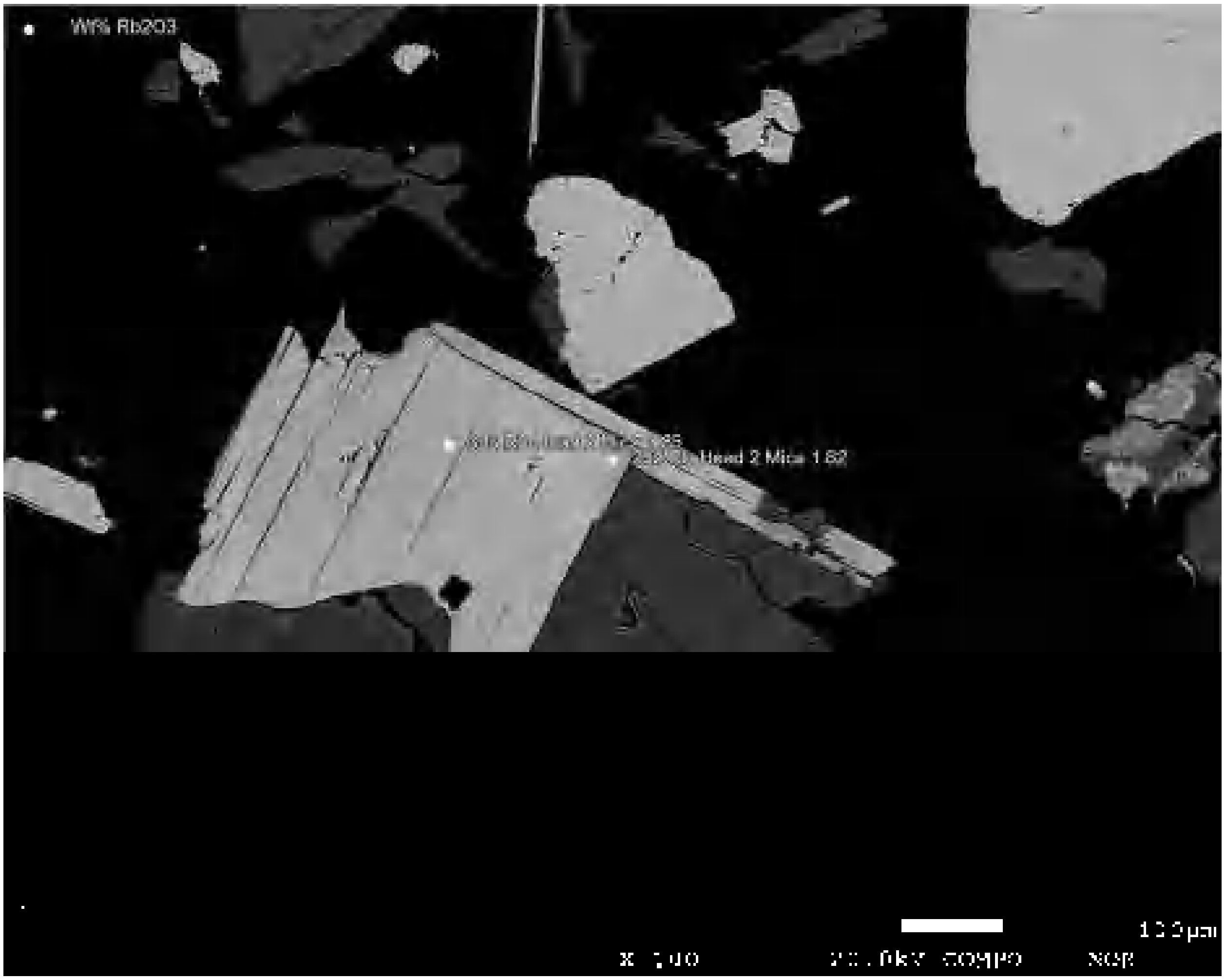
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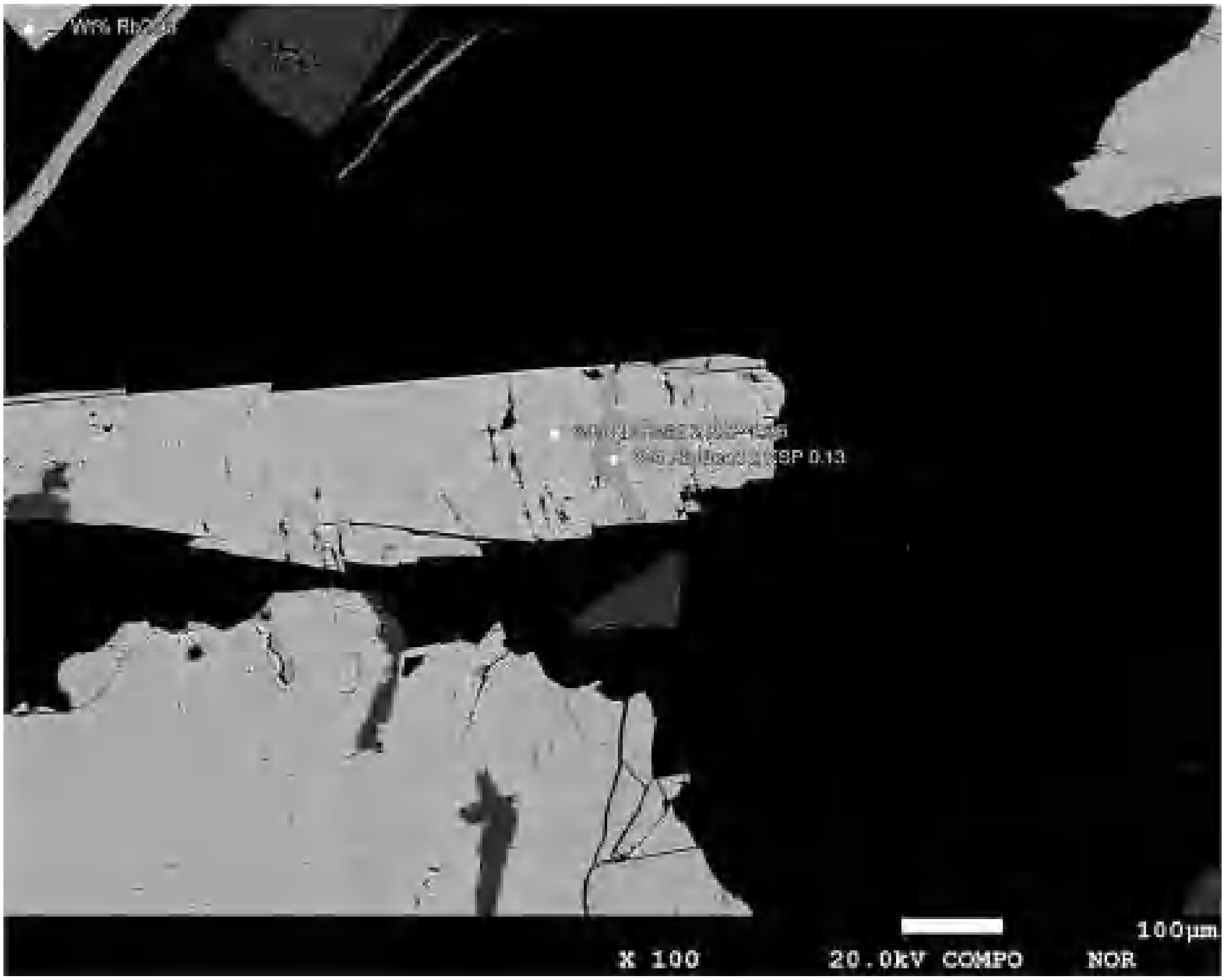
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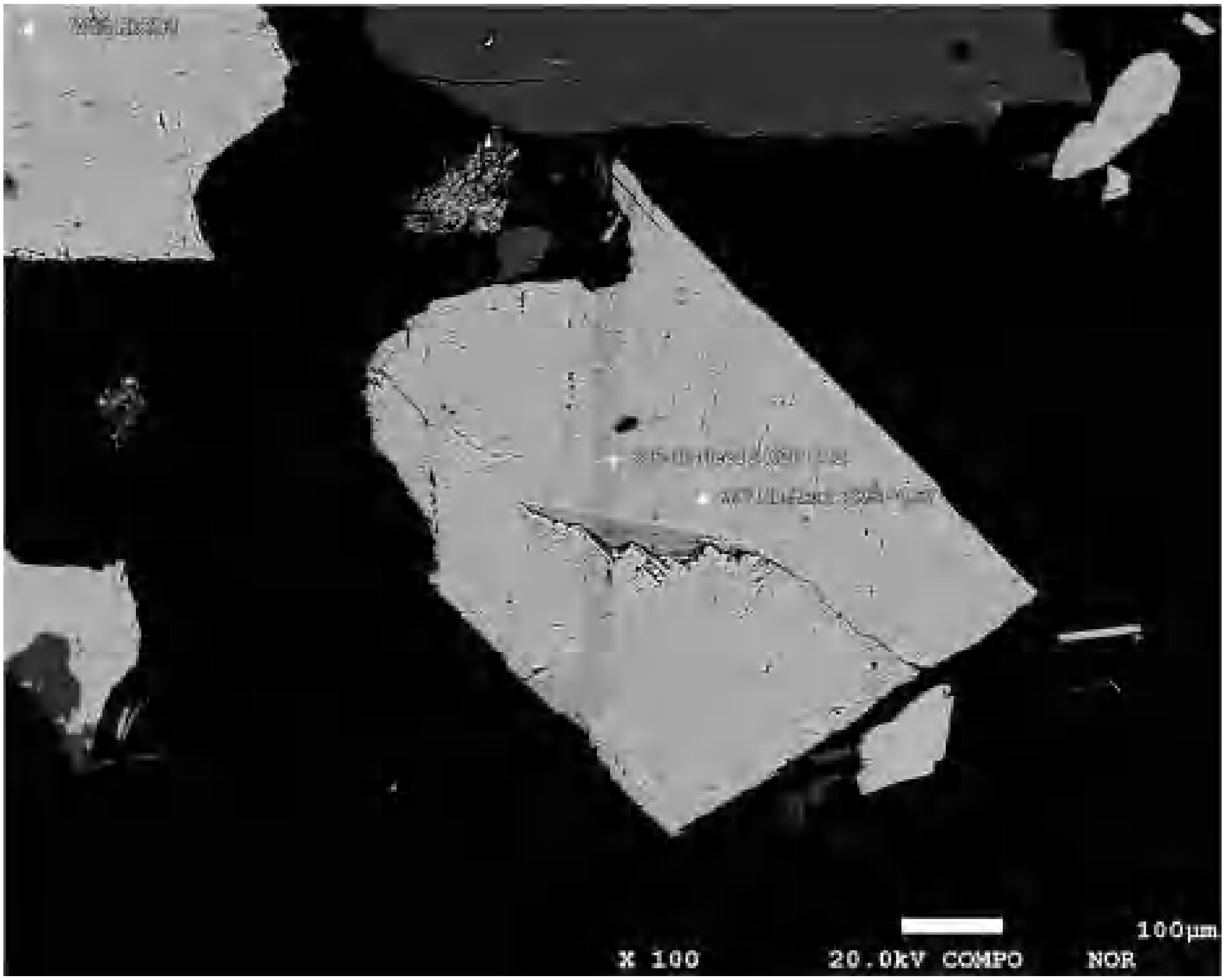
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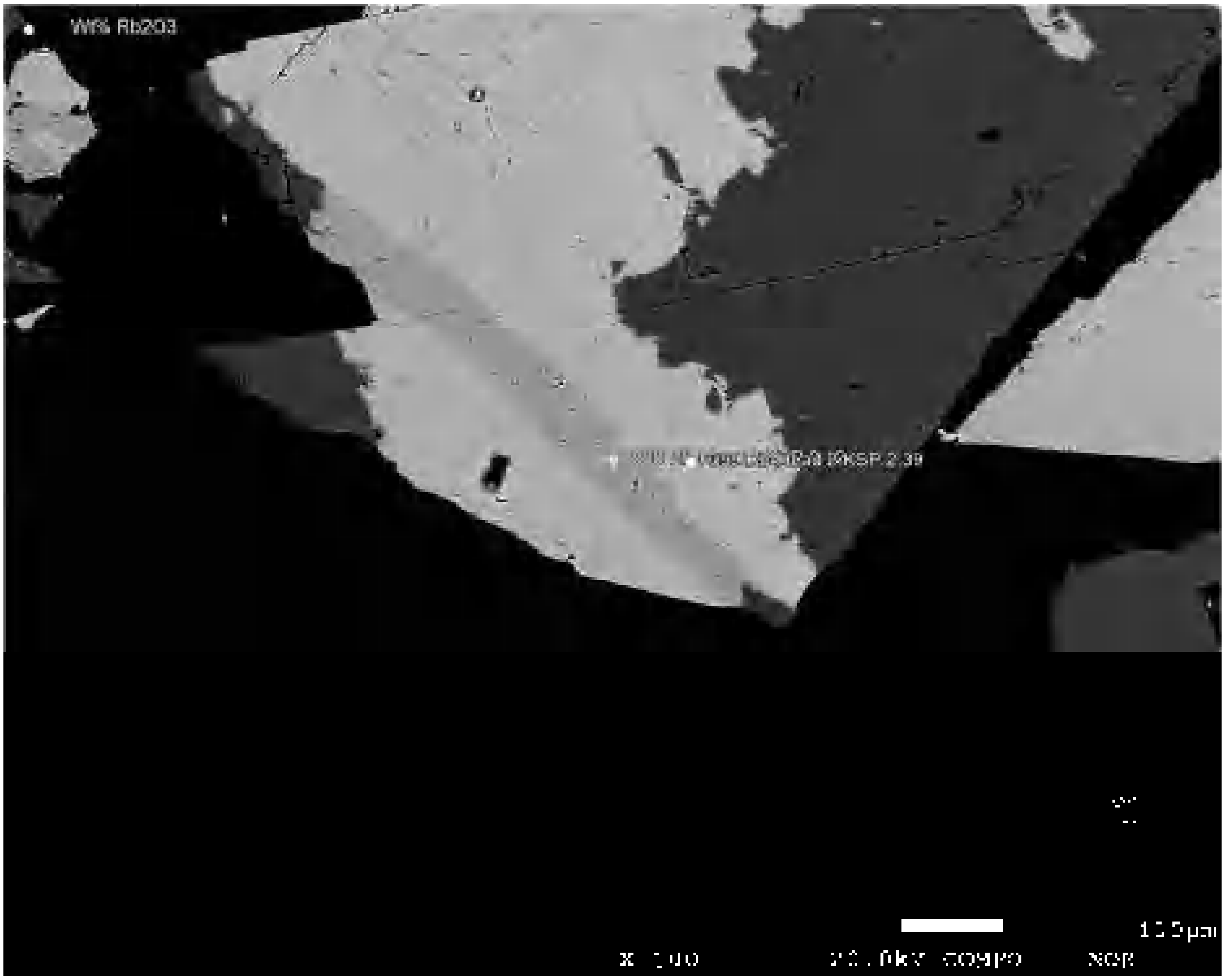
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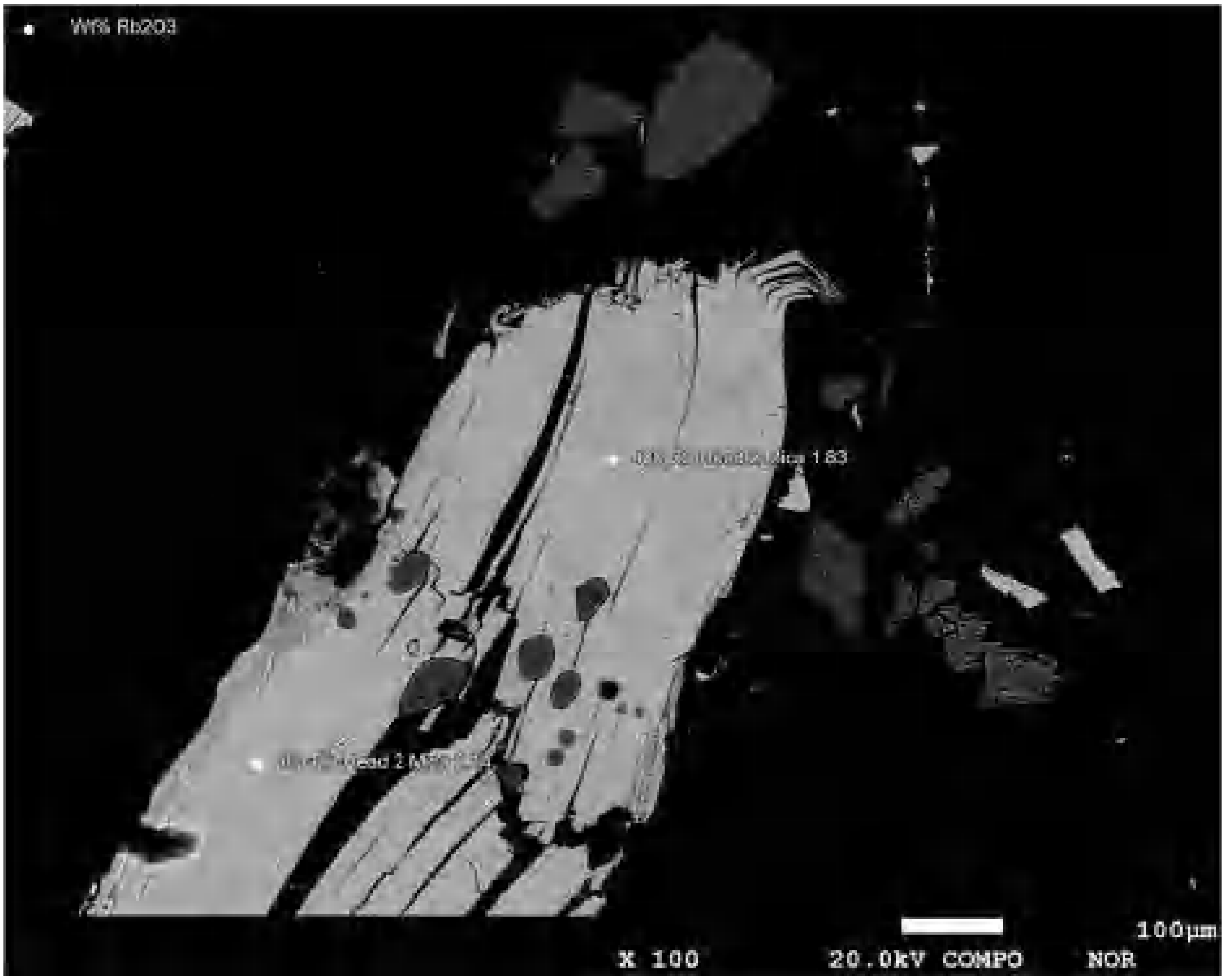

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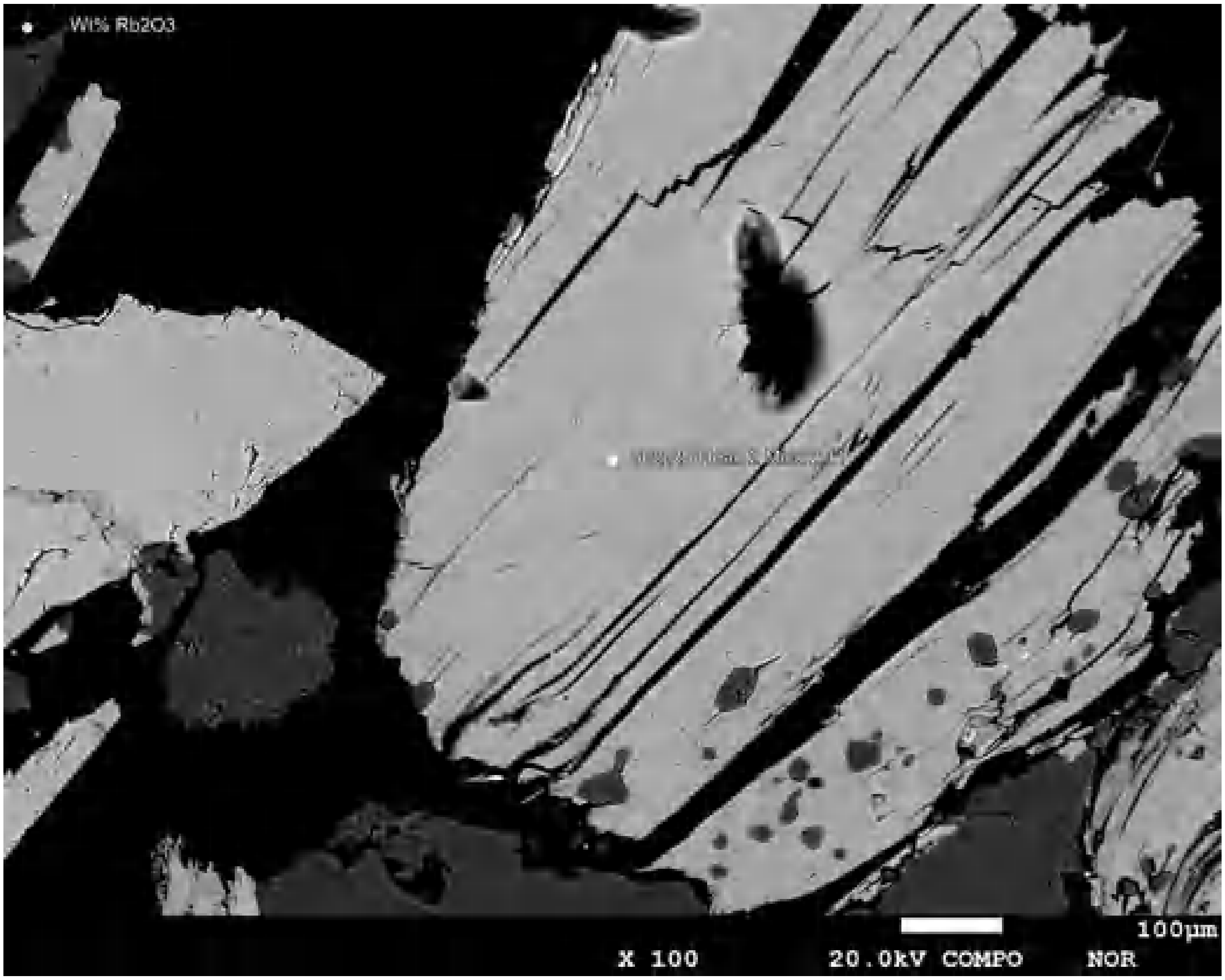
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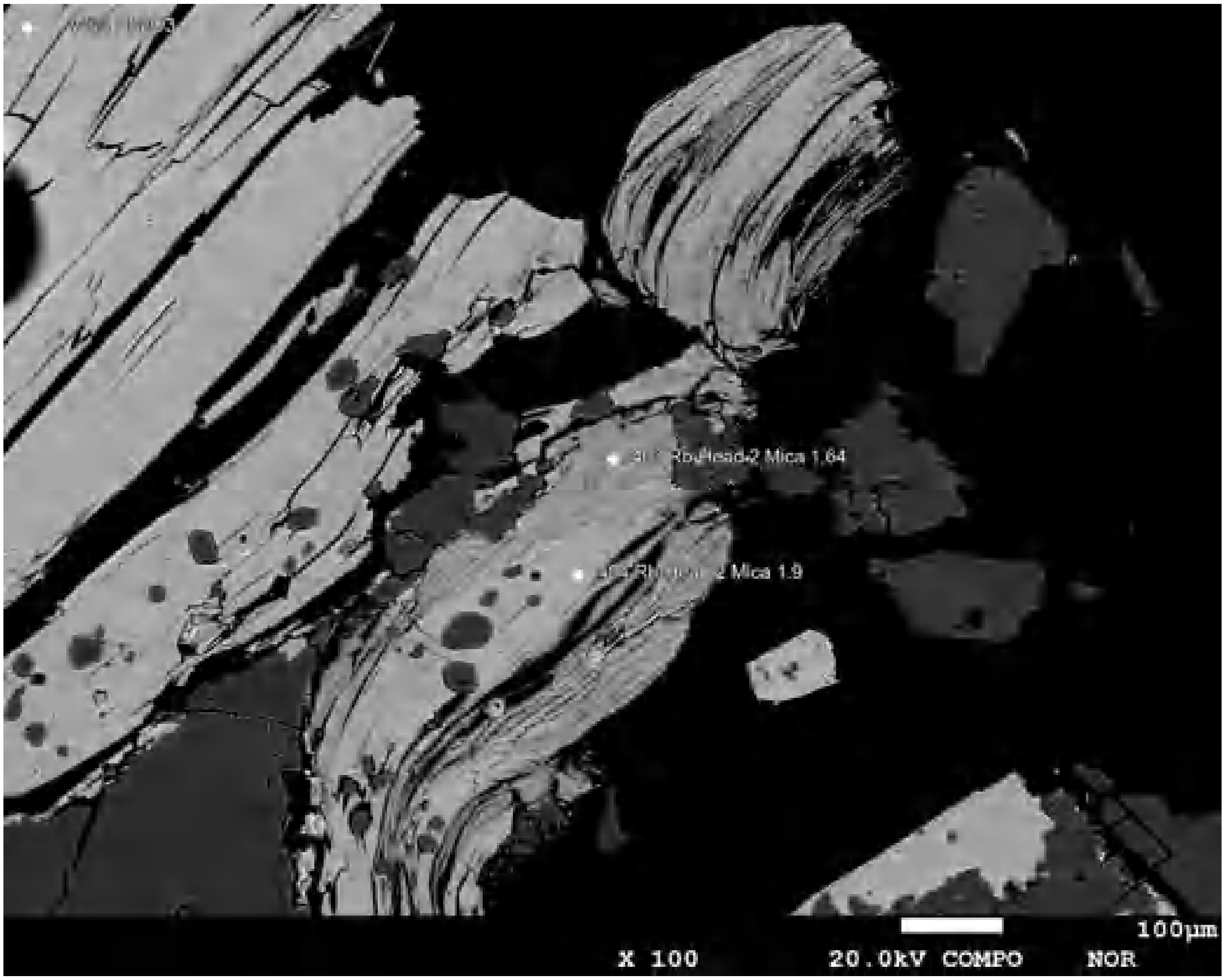
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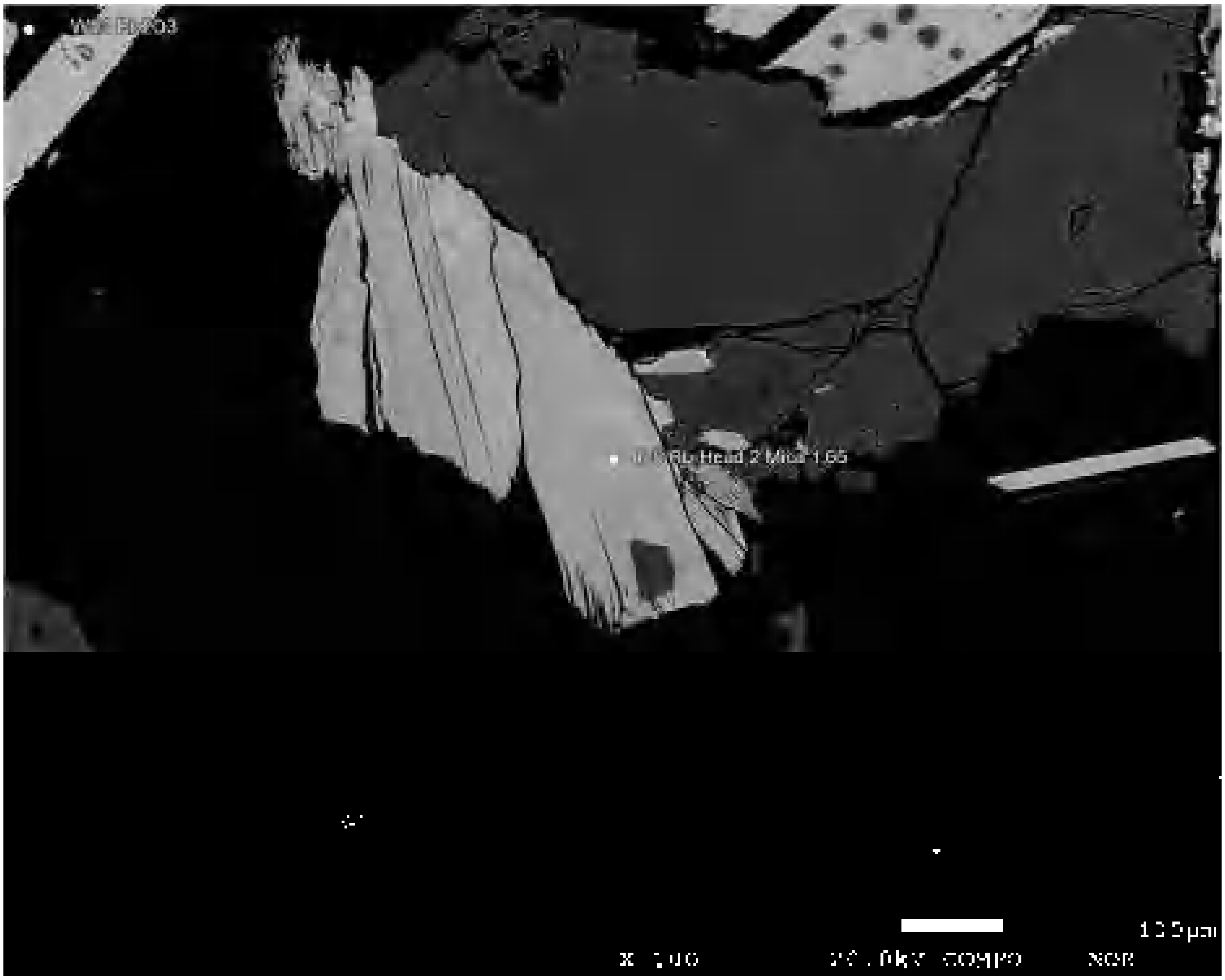
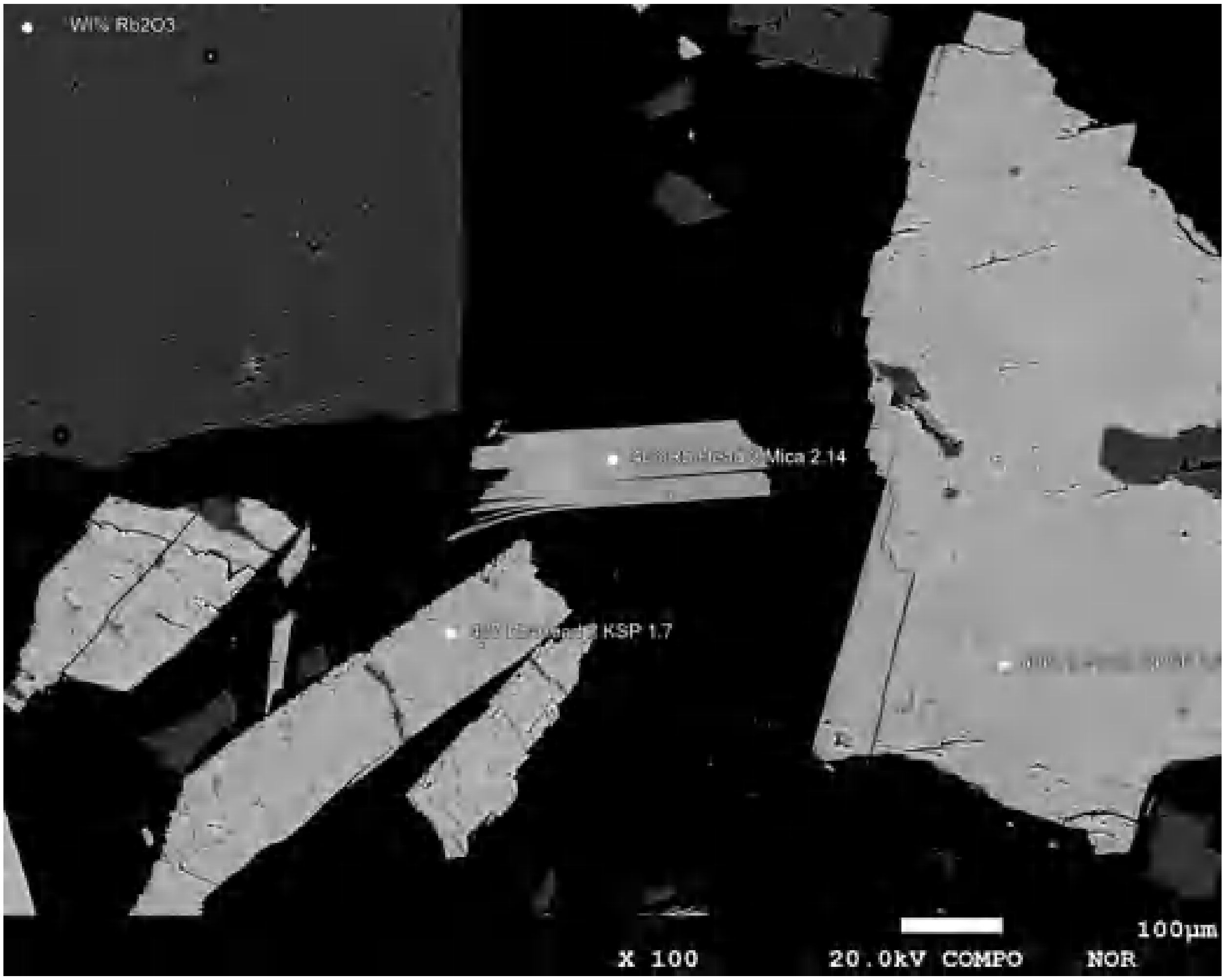
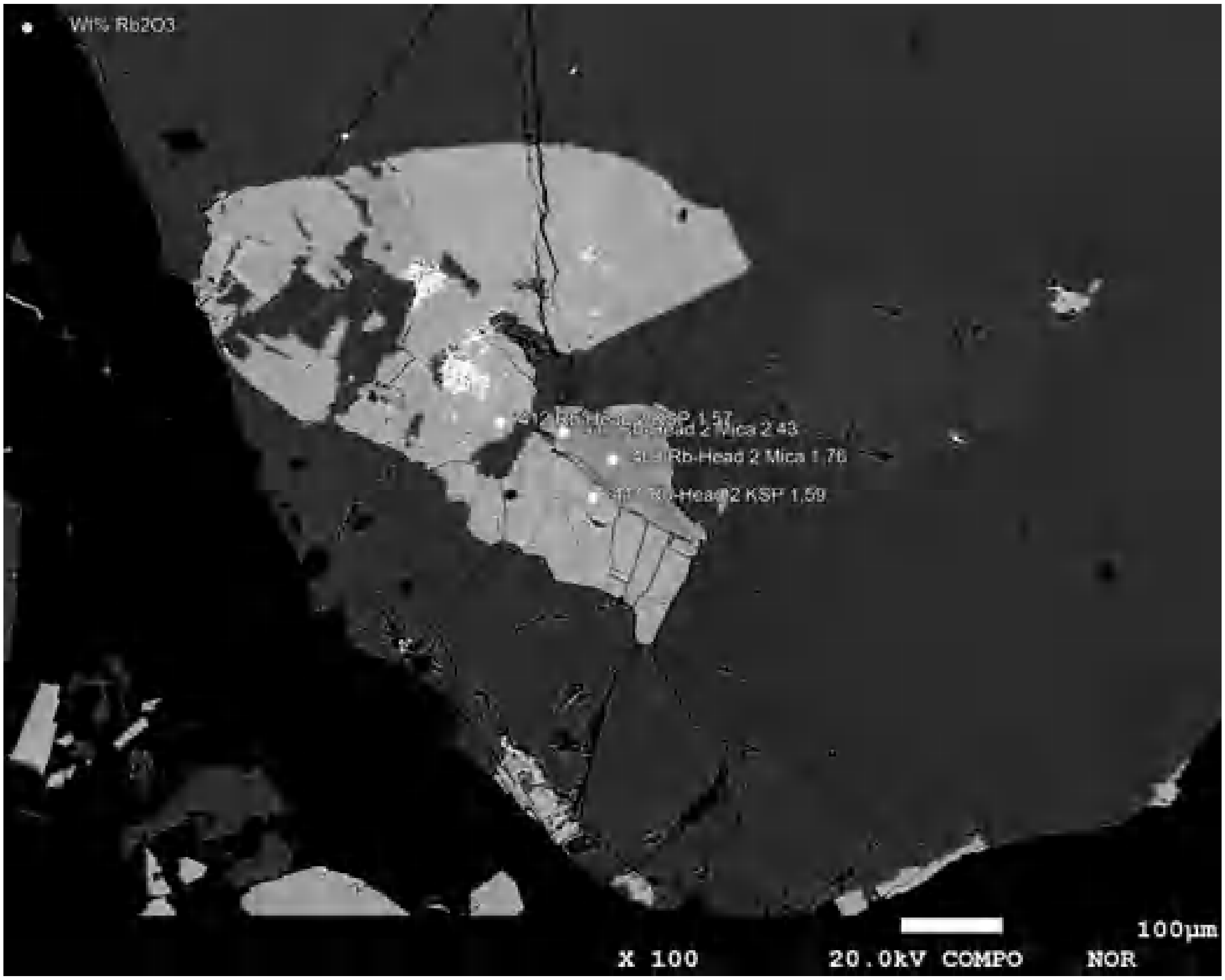
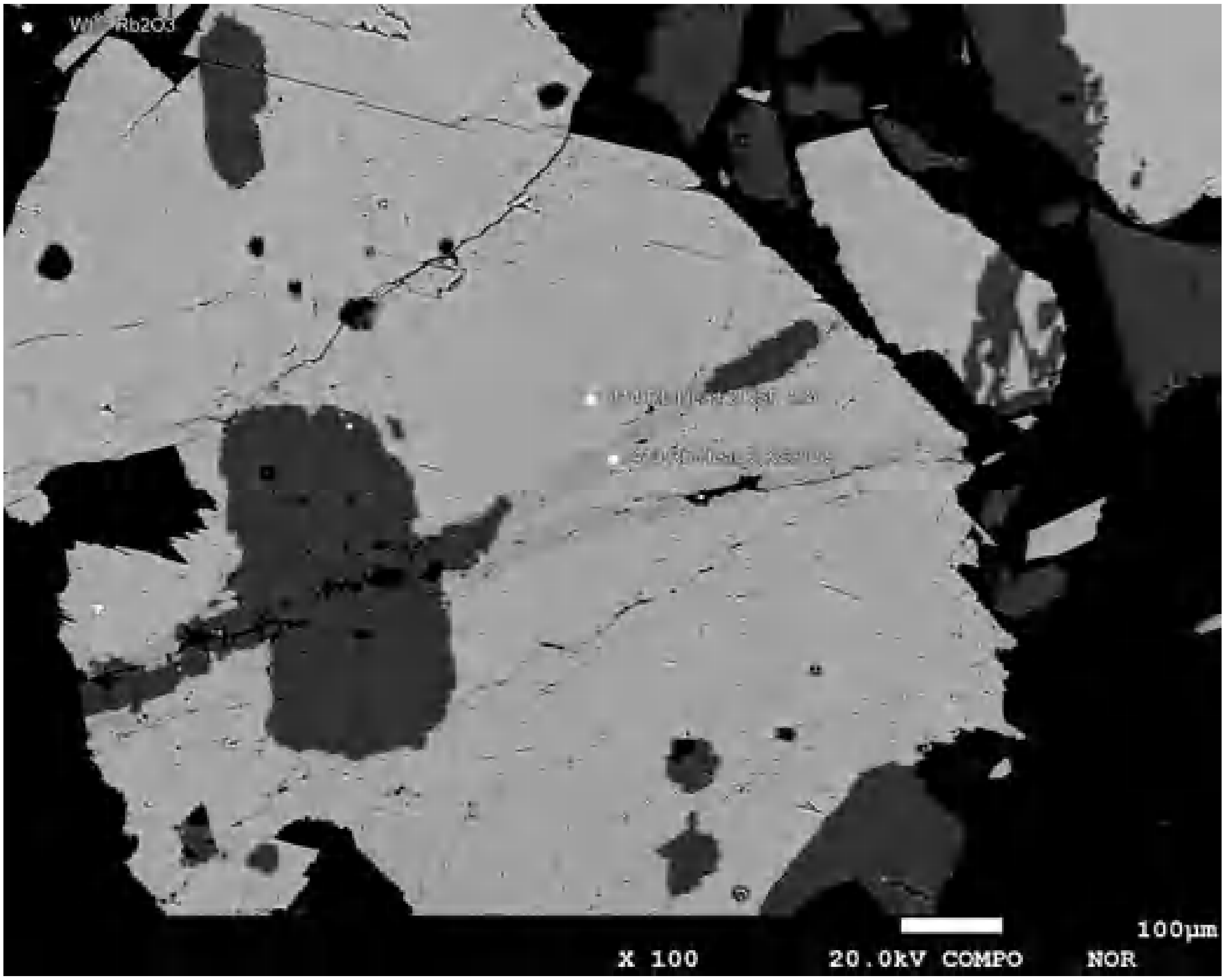
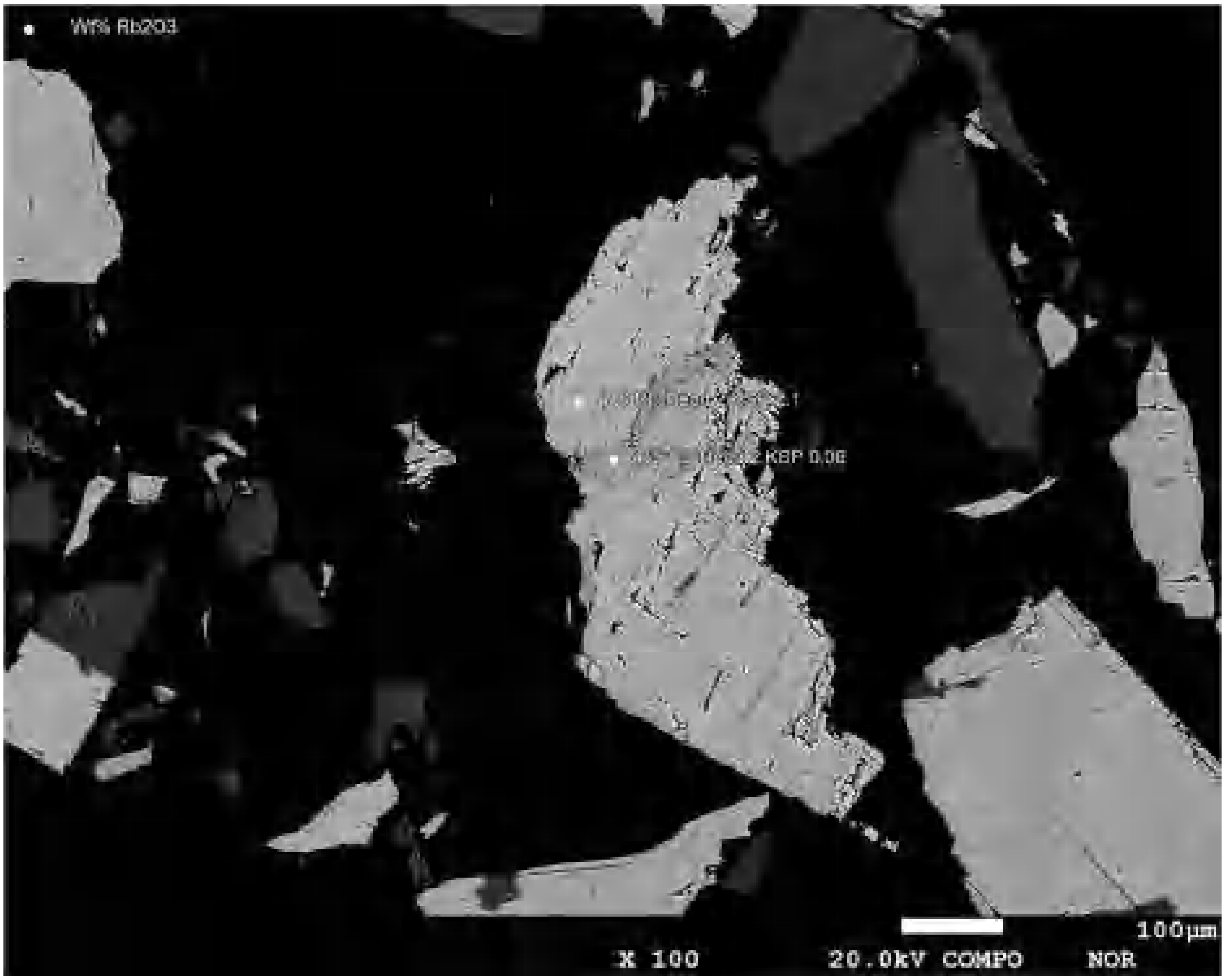
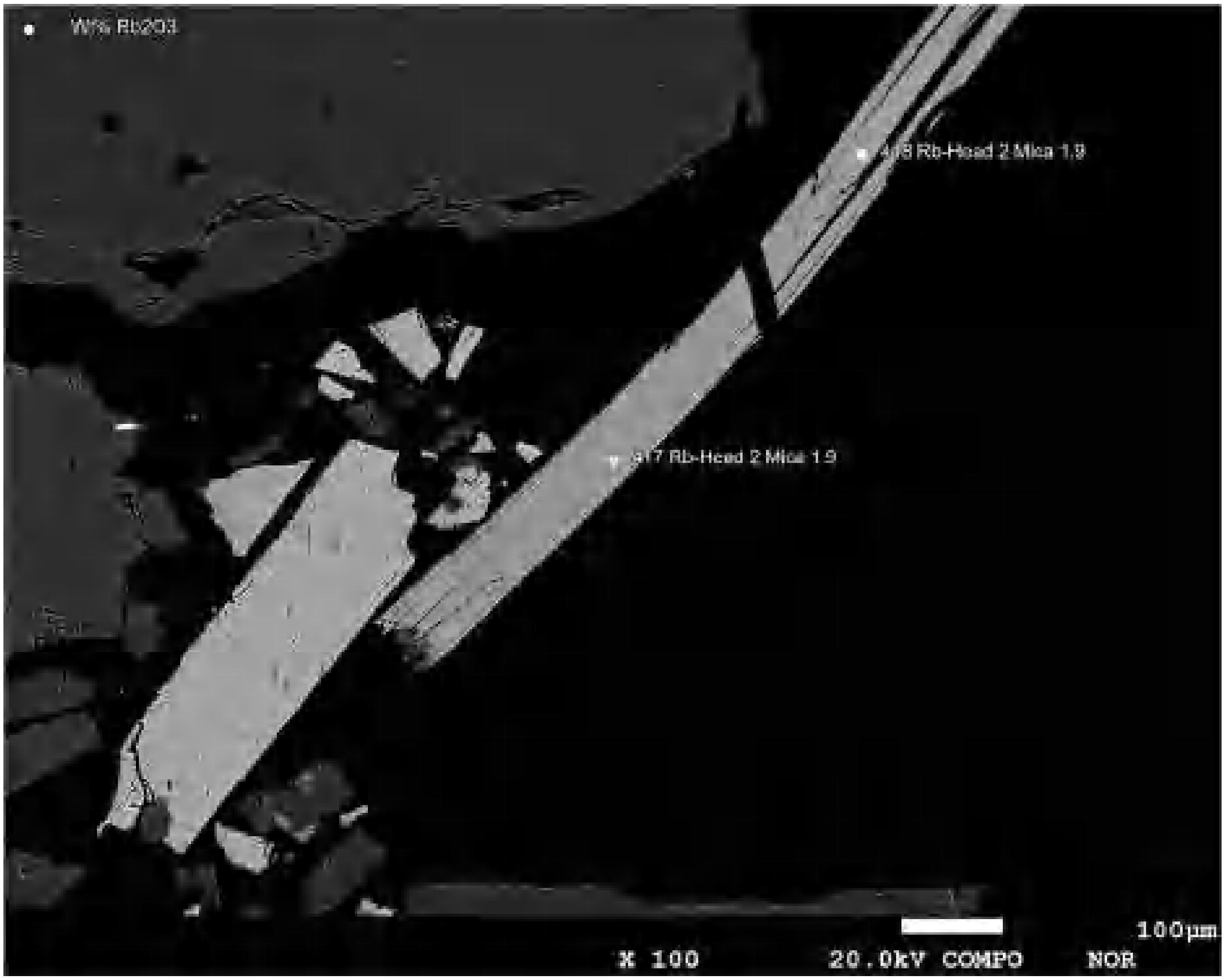

Head 2 Mica 1.65

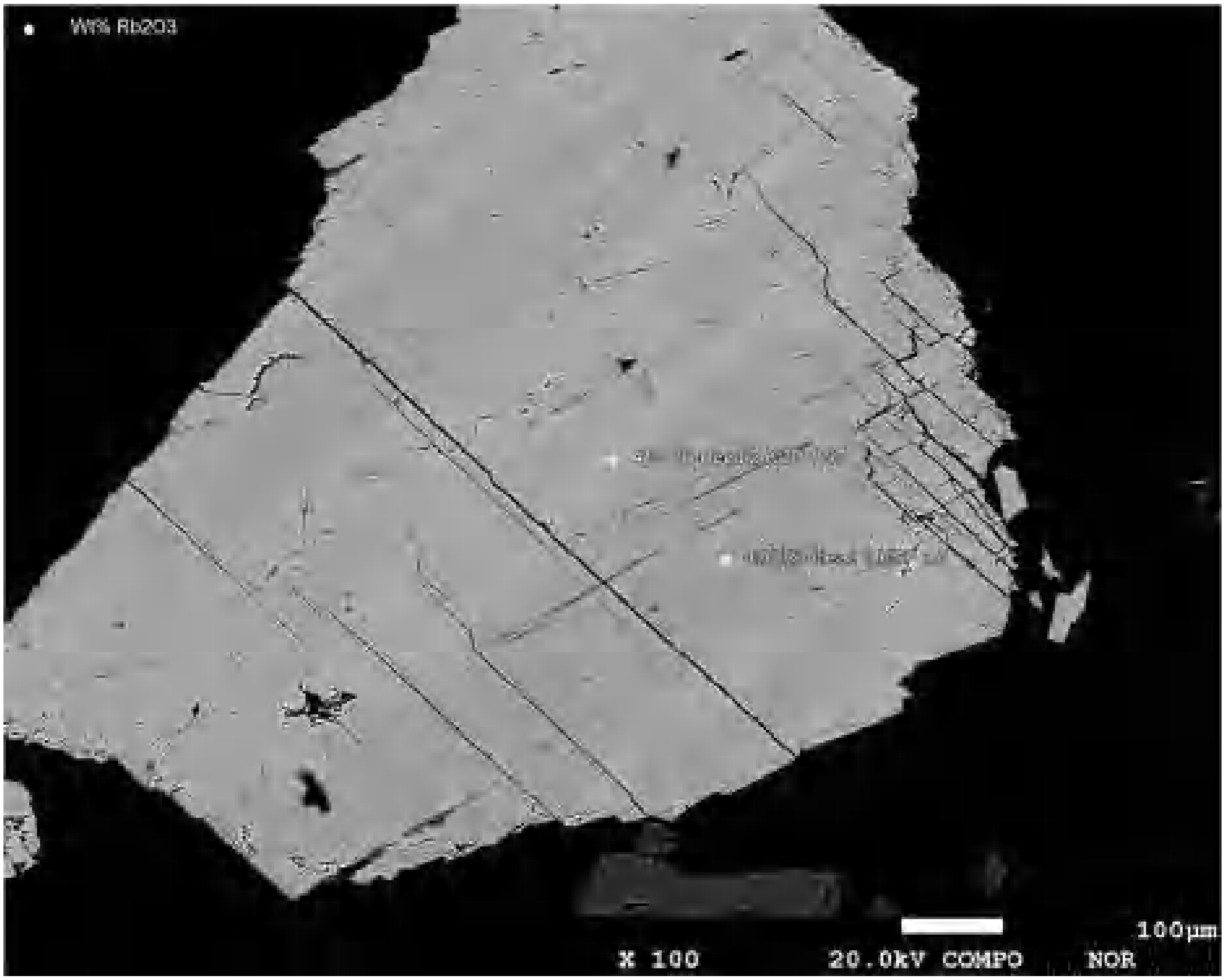

Wt% Rb2O3
Mica 2.14
KSP 1.7
100µm
X 100
20.0kV COMPO
NOR

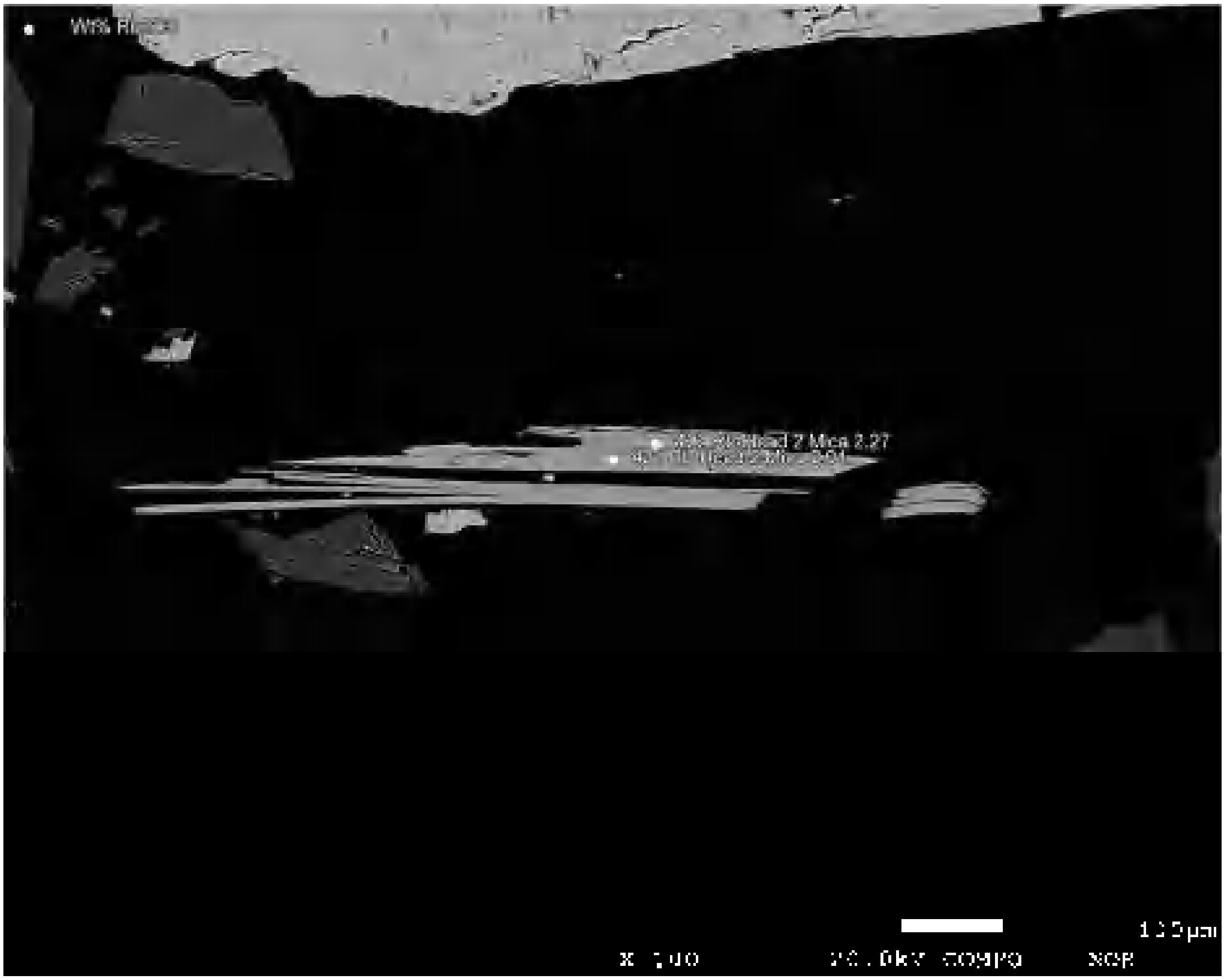

Wt% Rb2O3
Rb-Head 2 Mica 1.76
100µm
X 100
20.0kV COMPO
NOR

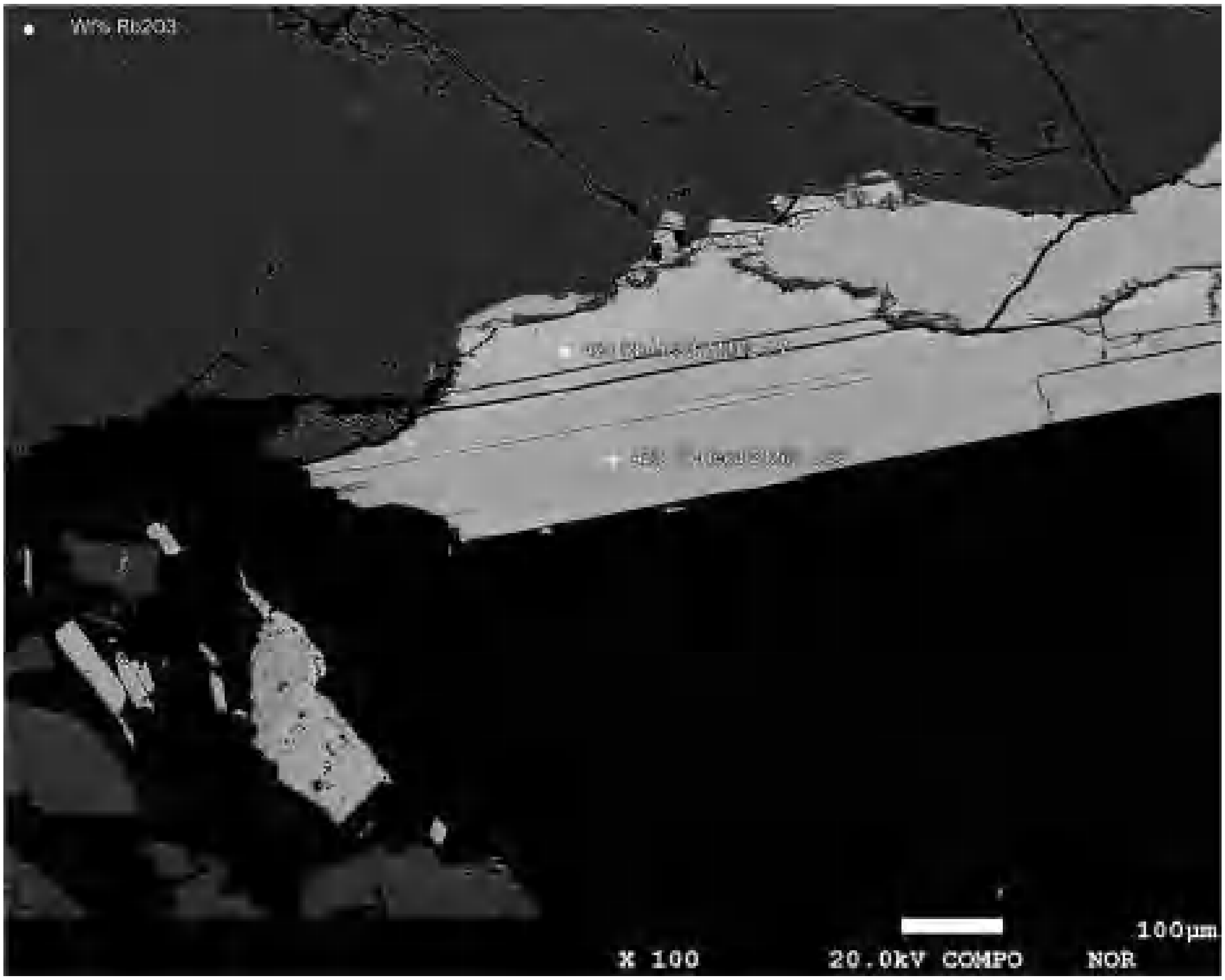

Rb2O3
100µm
X 100
20.0kV COMPO
NOR

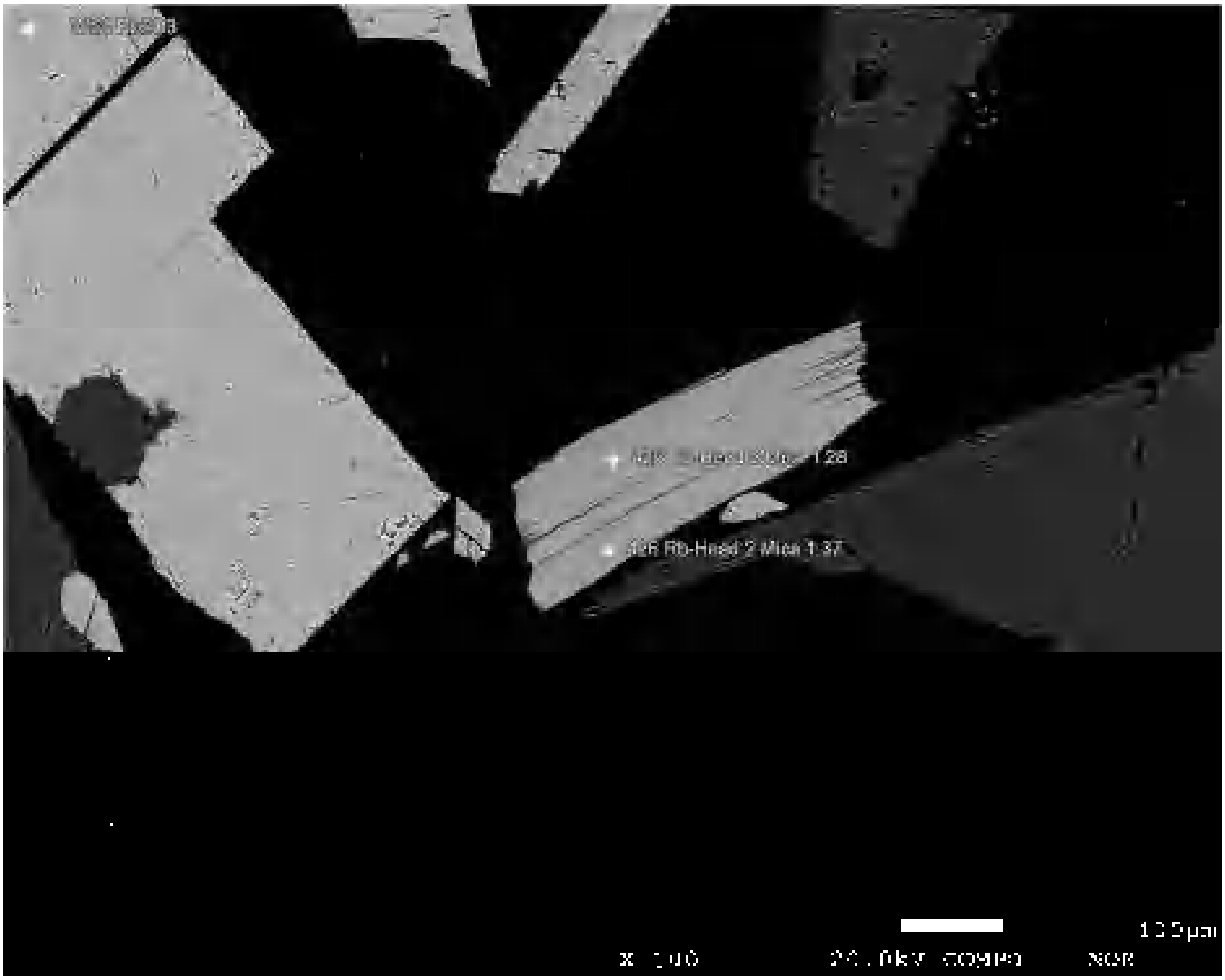

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

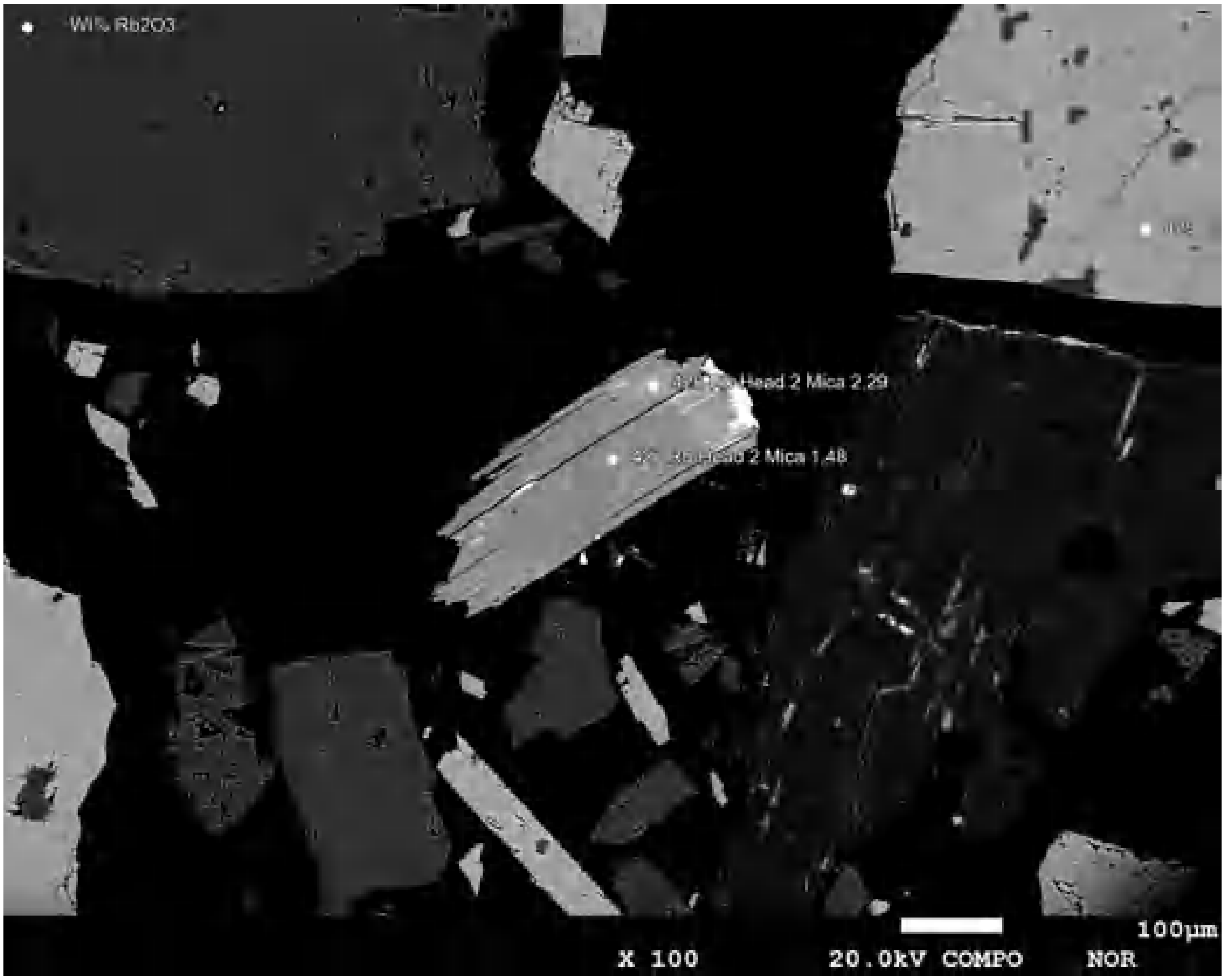

Wt% Rb2O3
Rb-Head 2 Mica 1.9
Rb-Head 2 Mica 1.9
100µm
X 100
20.0kV COMPO
NOR

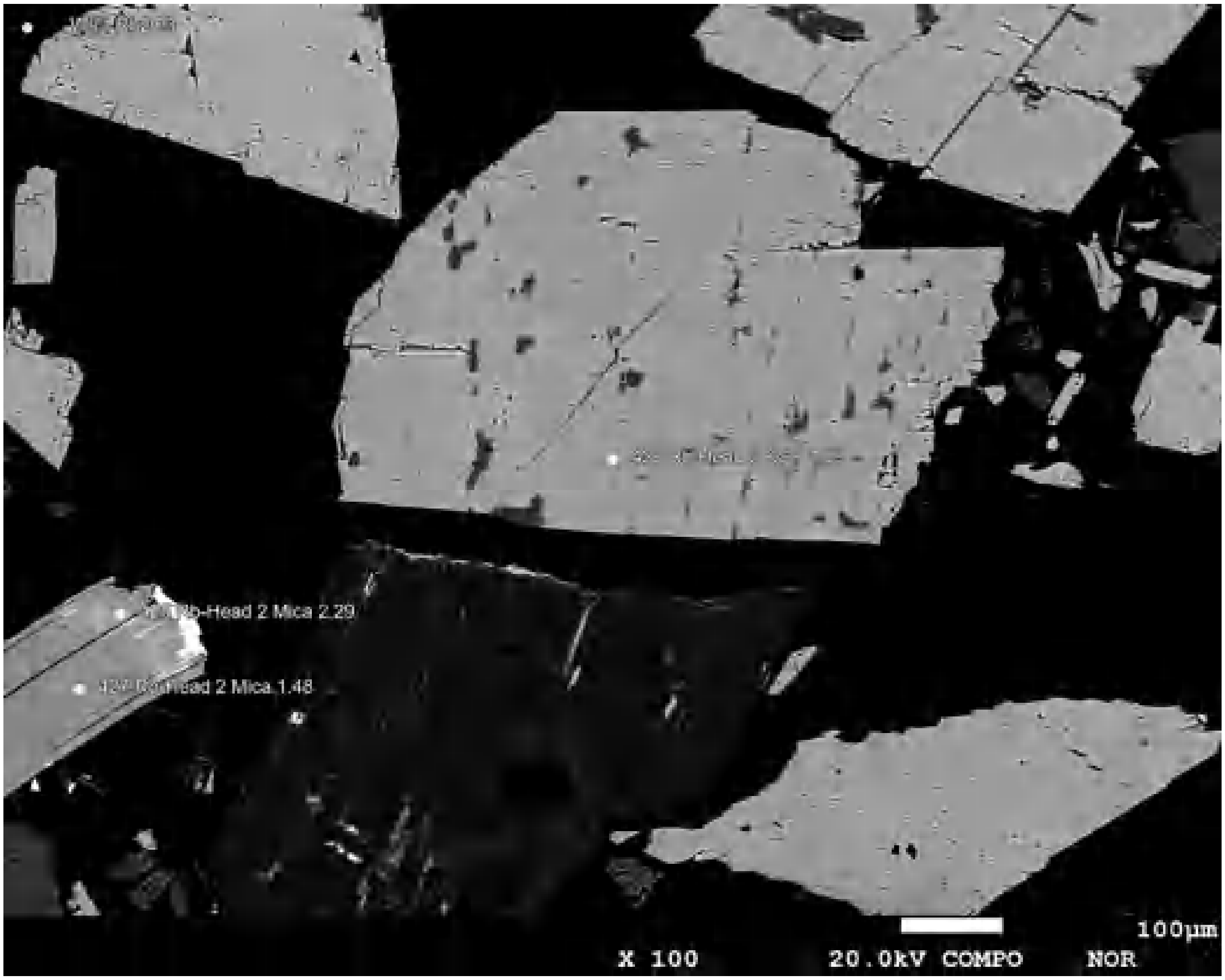

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

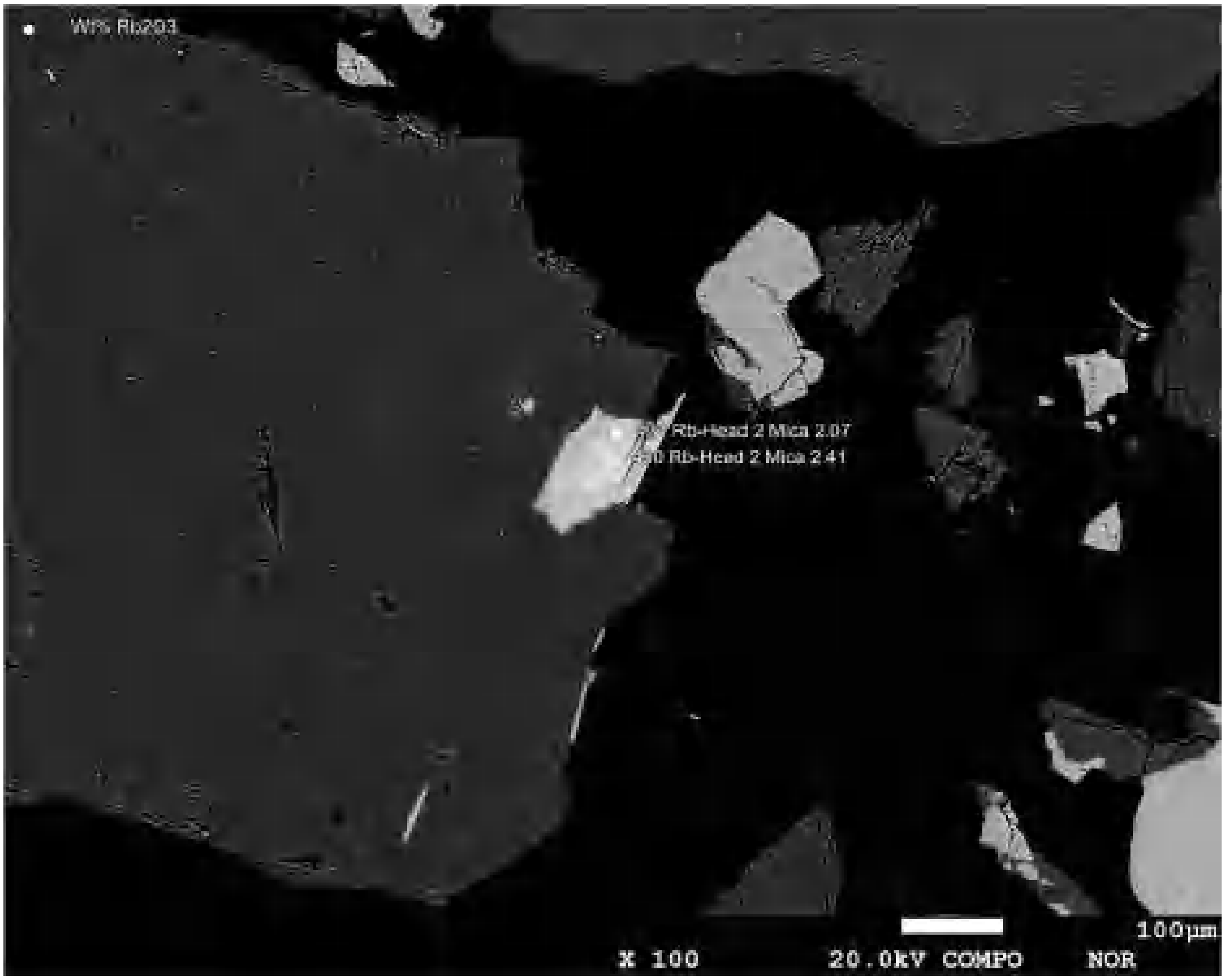


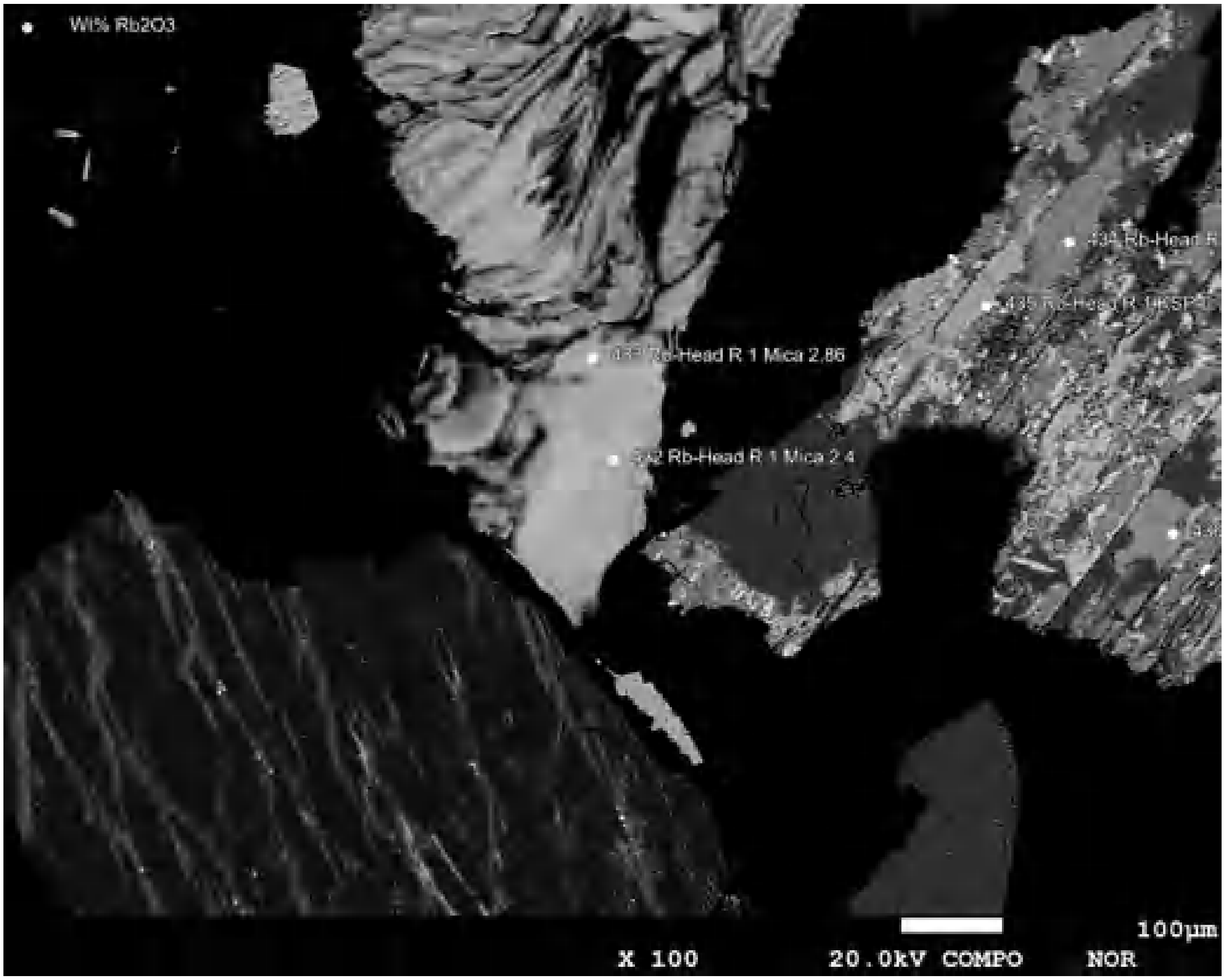

Wt% Rb2O3
100µm
x 100
20.0kV COMPO
NOR

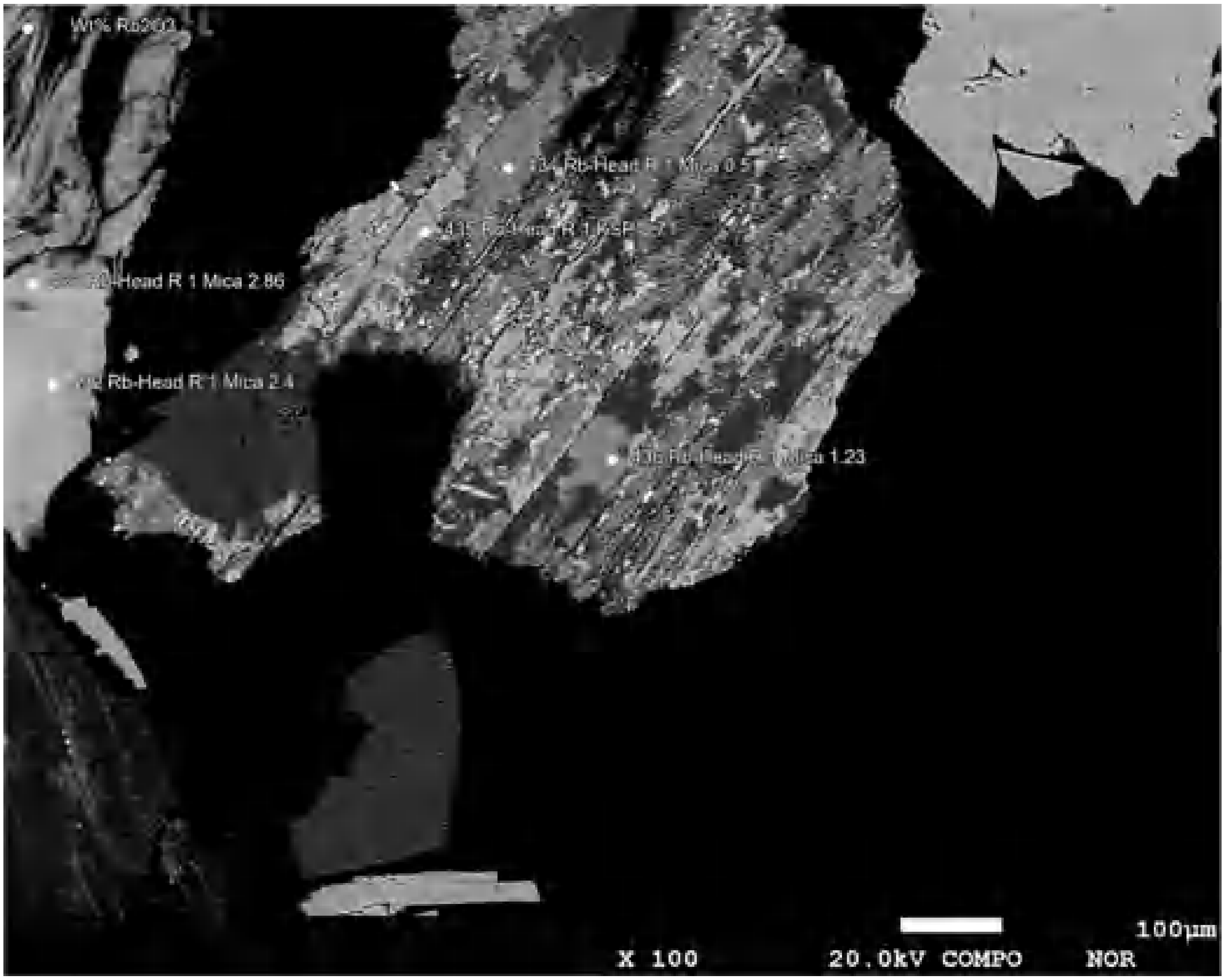


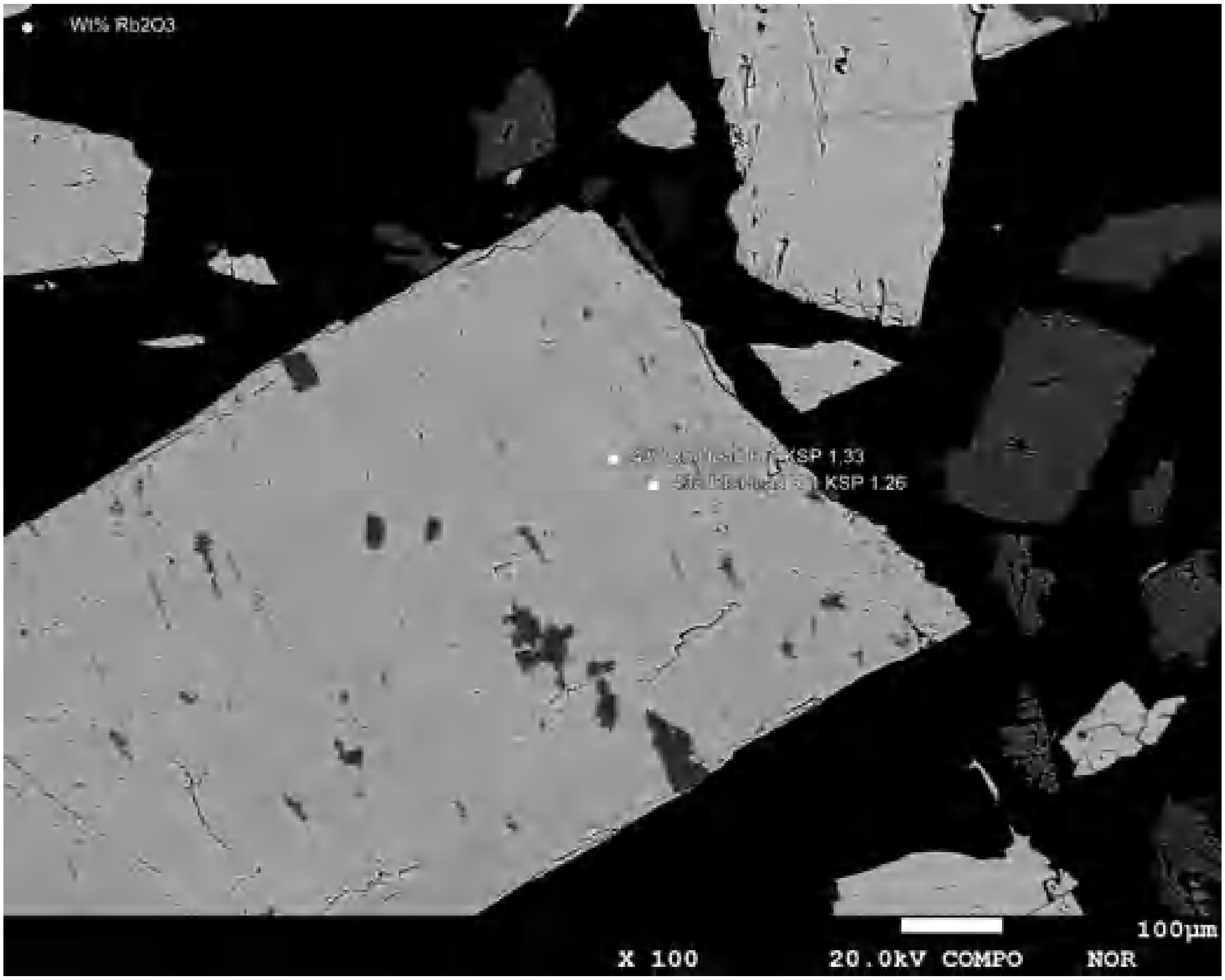

Wt% Rb2O3
Head 2 Mica 2.29
Head 2 Mica 1.48
100μm
X 100
20.0kV COMPO
NOR


b-Head 2 Mica 2.29
Head 2 Mica 1.48
100μm
X 100
20.0kV COMPO
NOR

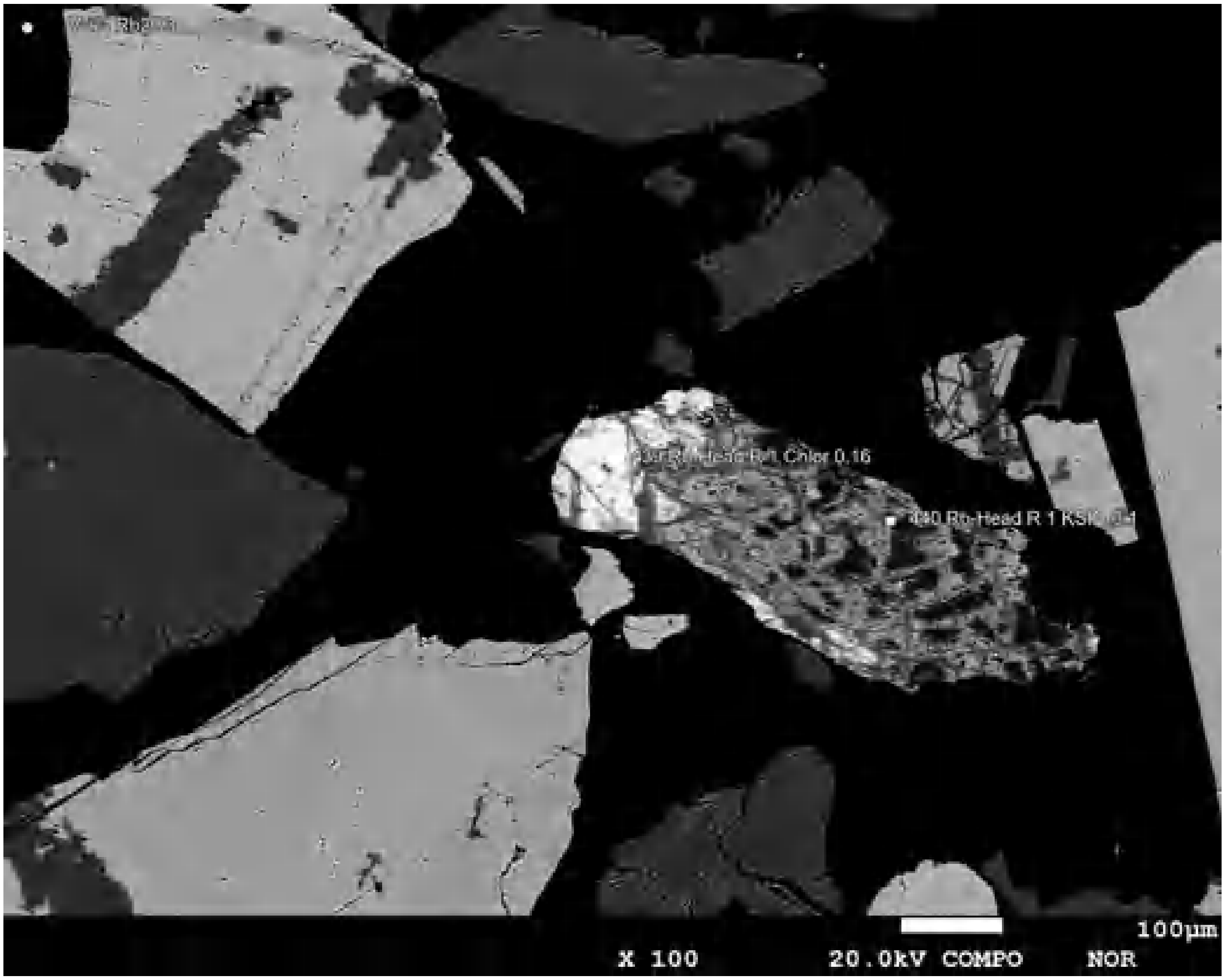

Wt% Rb2O3
Rb-Head 2 Mica 2.07
Rb-Head 2 Mica 2.41
100μm
x 100
20.0kV COMPO
NOR

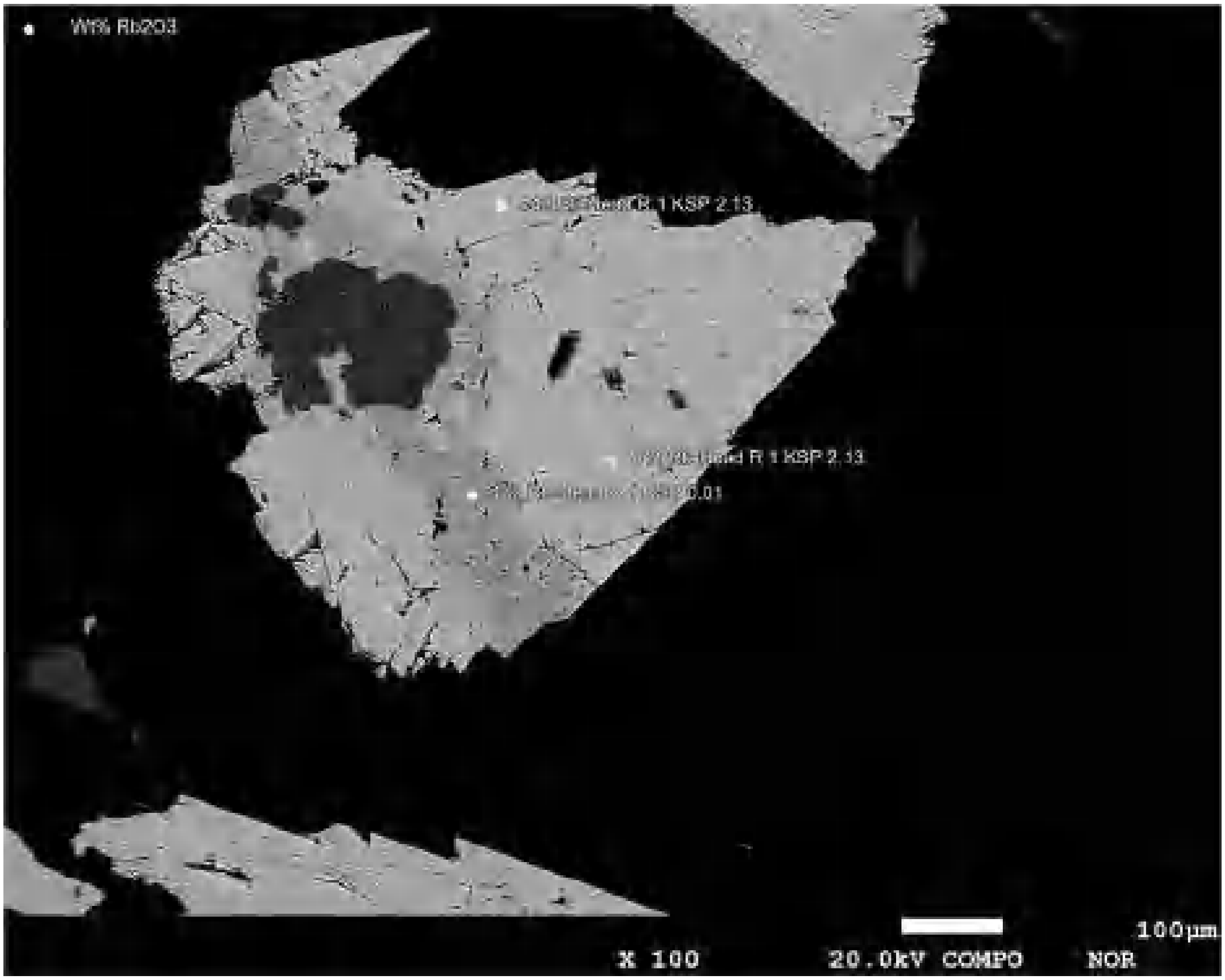

Wt% Rb2O3
Rb-Head R
Head R 1 Mica 2.86
Rb-Head R 1 Mica 2.4
100µm
X 100
20.0kV COMPO
NOR

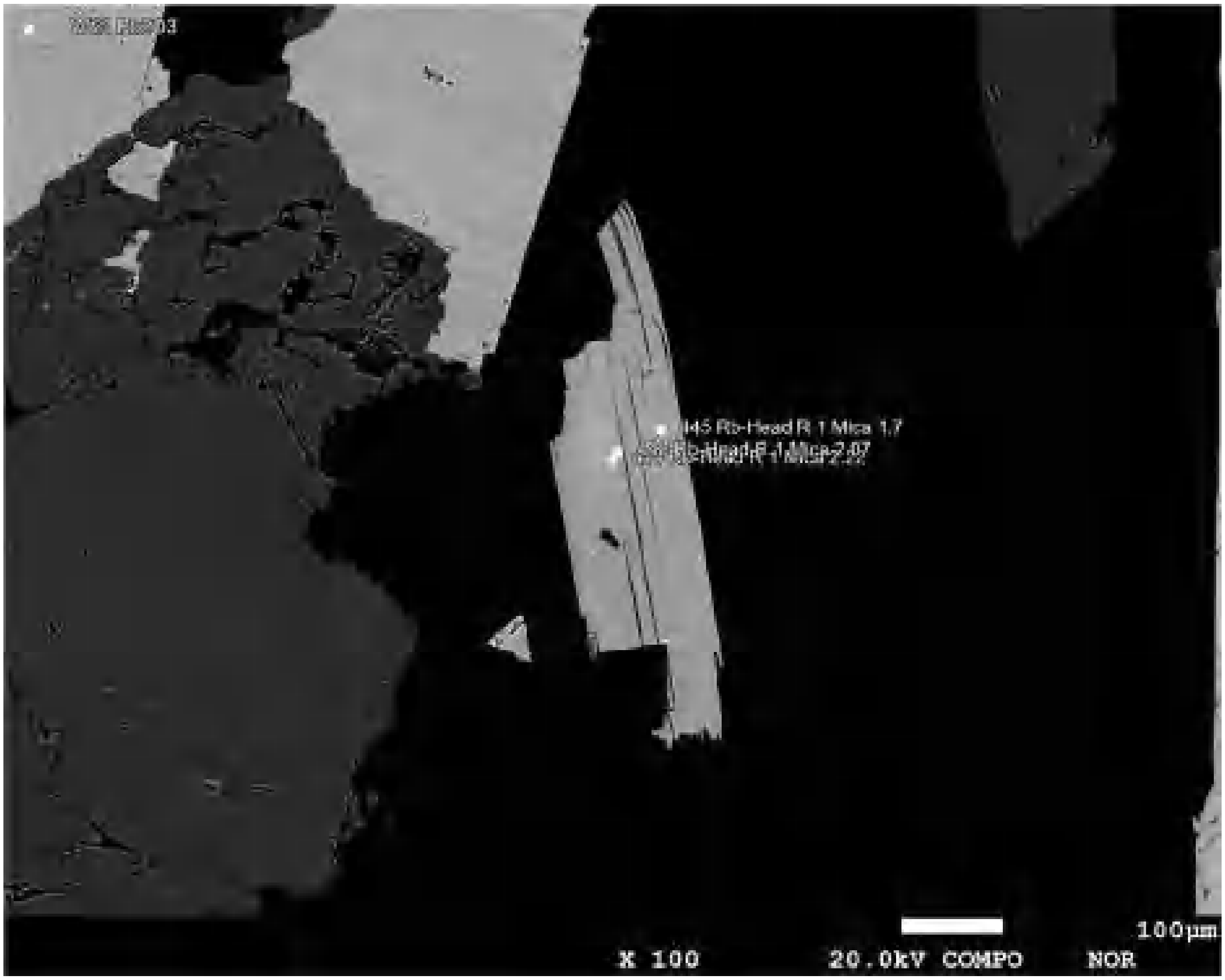

Rb-Head R 1 Mica 2.86
Rb-Head R 1 Mica 2.4
100µm
X 100
20.0kV COMPO
NOR



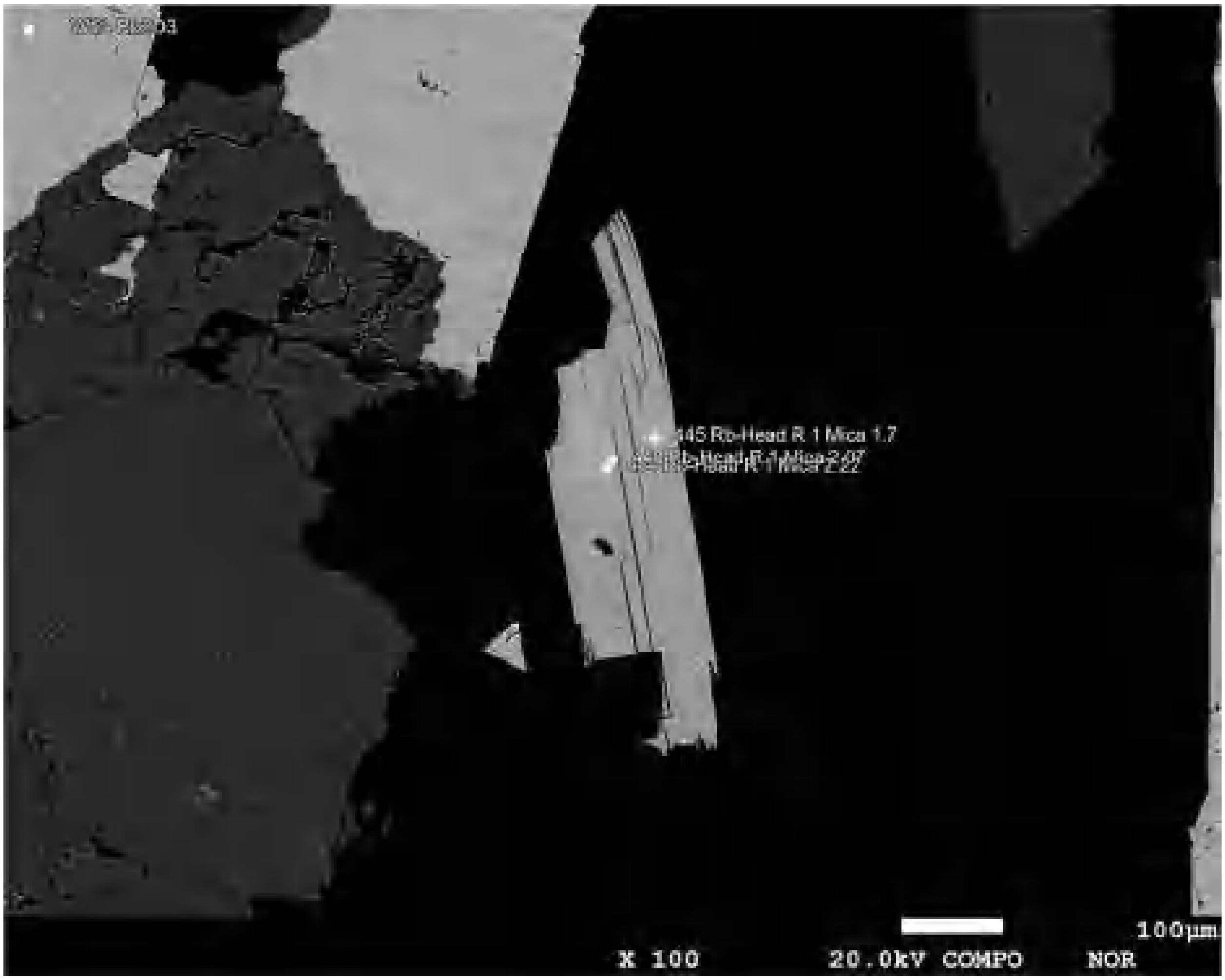
Wt% Rb2O3
KSP 1.33
KSP 1.26
100µm
X 100
20.0kV COMPO
NOR

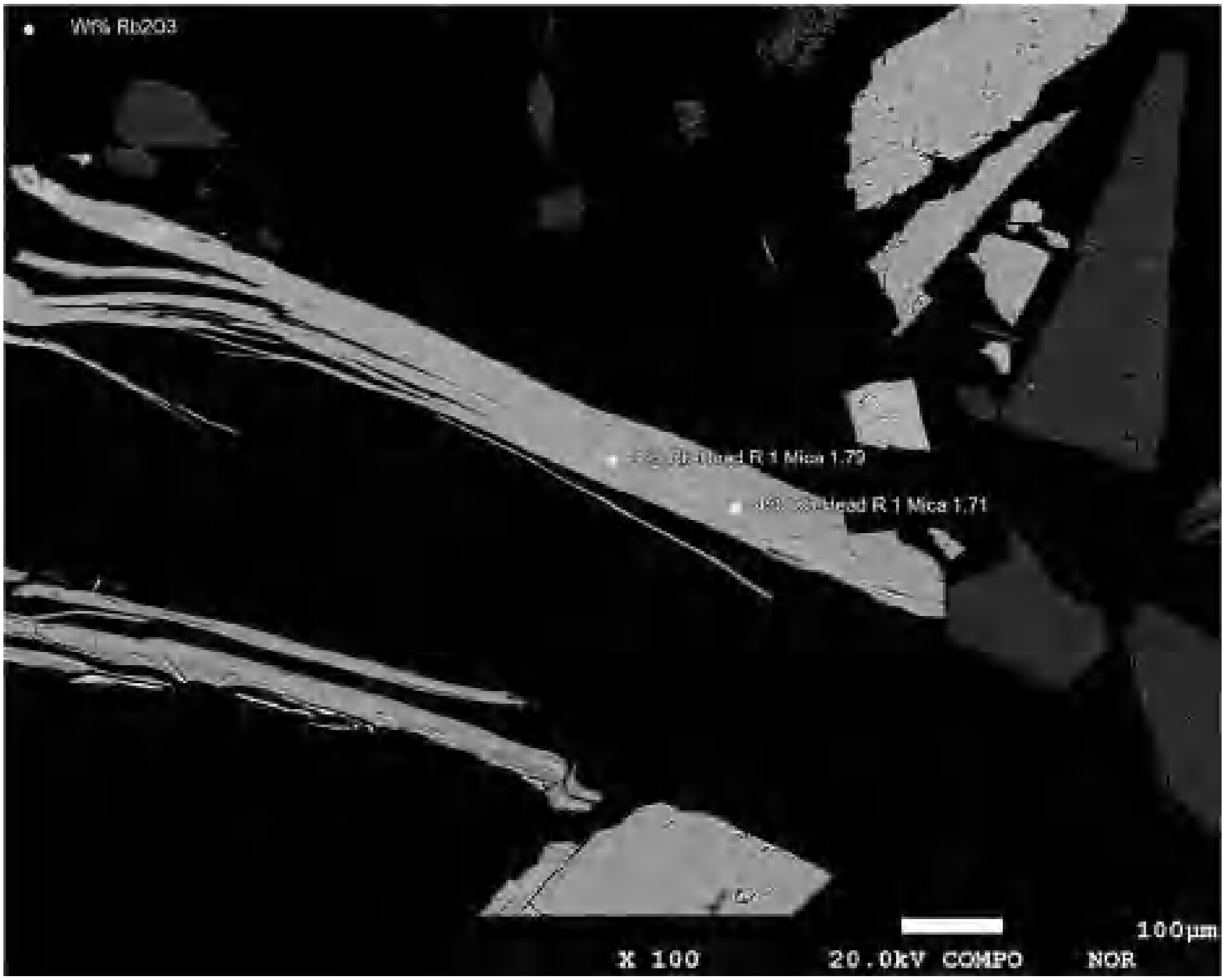

100µm
X 100
20.0kV COMPO
NOR

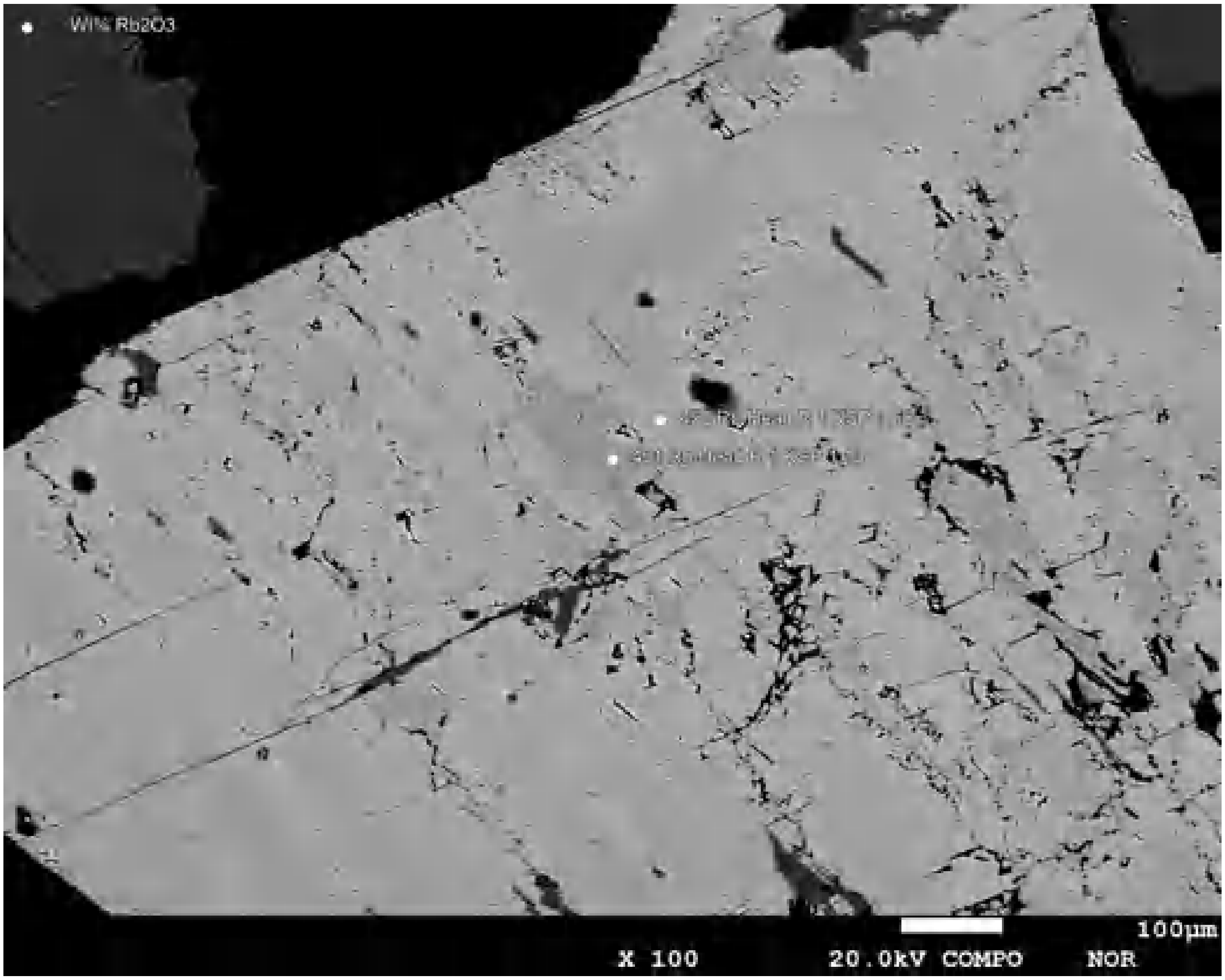

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

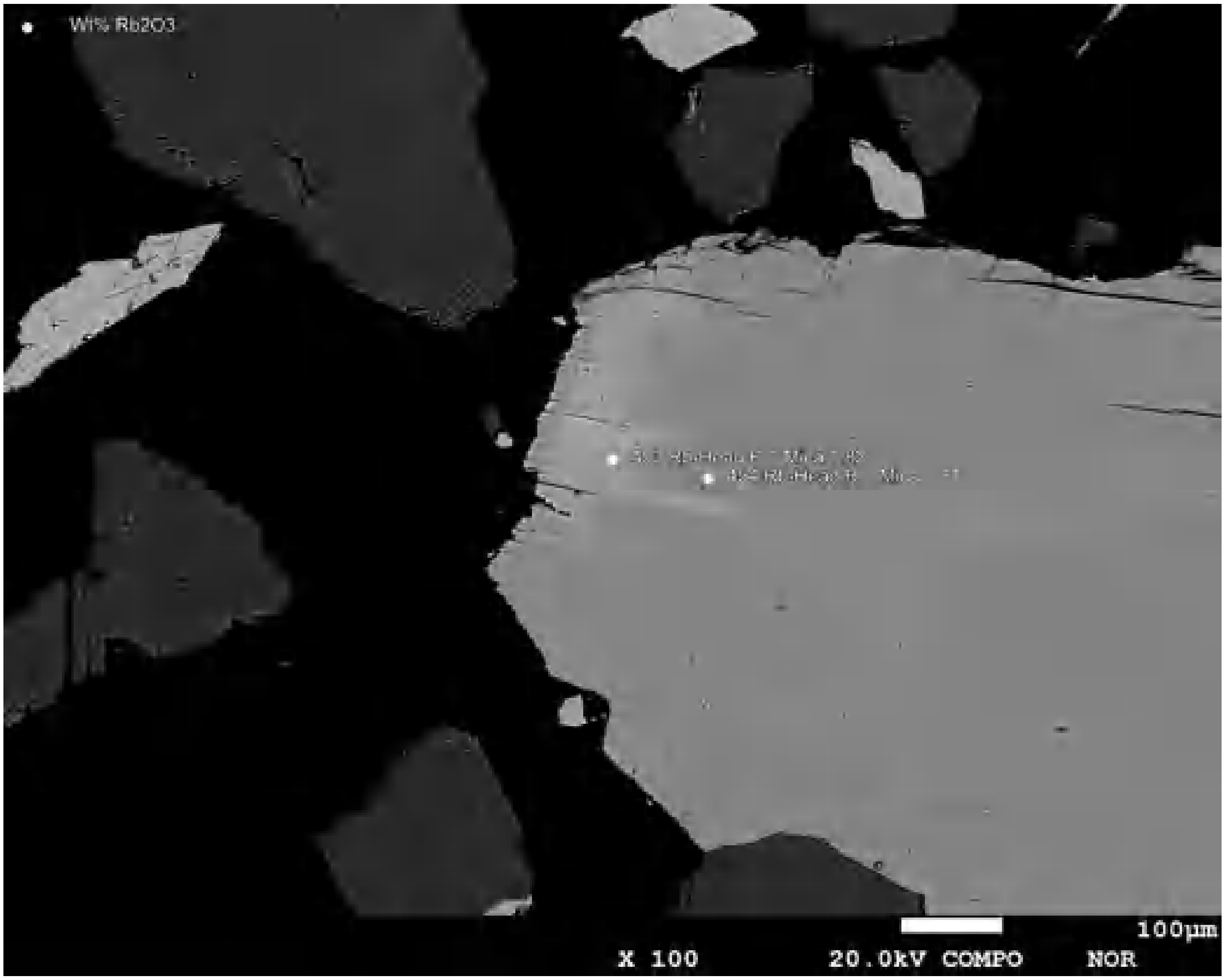

145 Rb-Head R 1 Mica 1.7
100µm
X 100
20.0kV COMPO
NOR

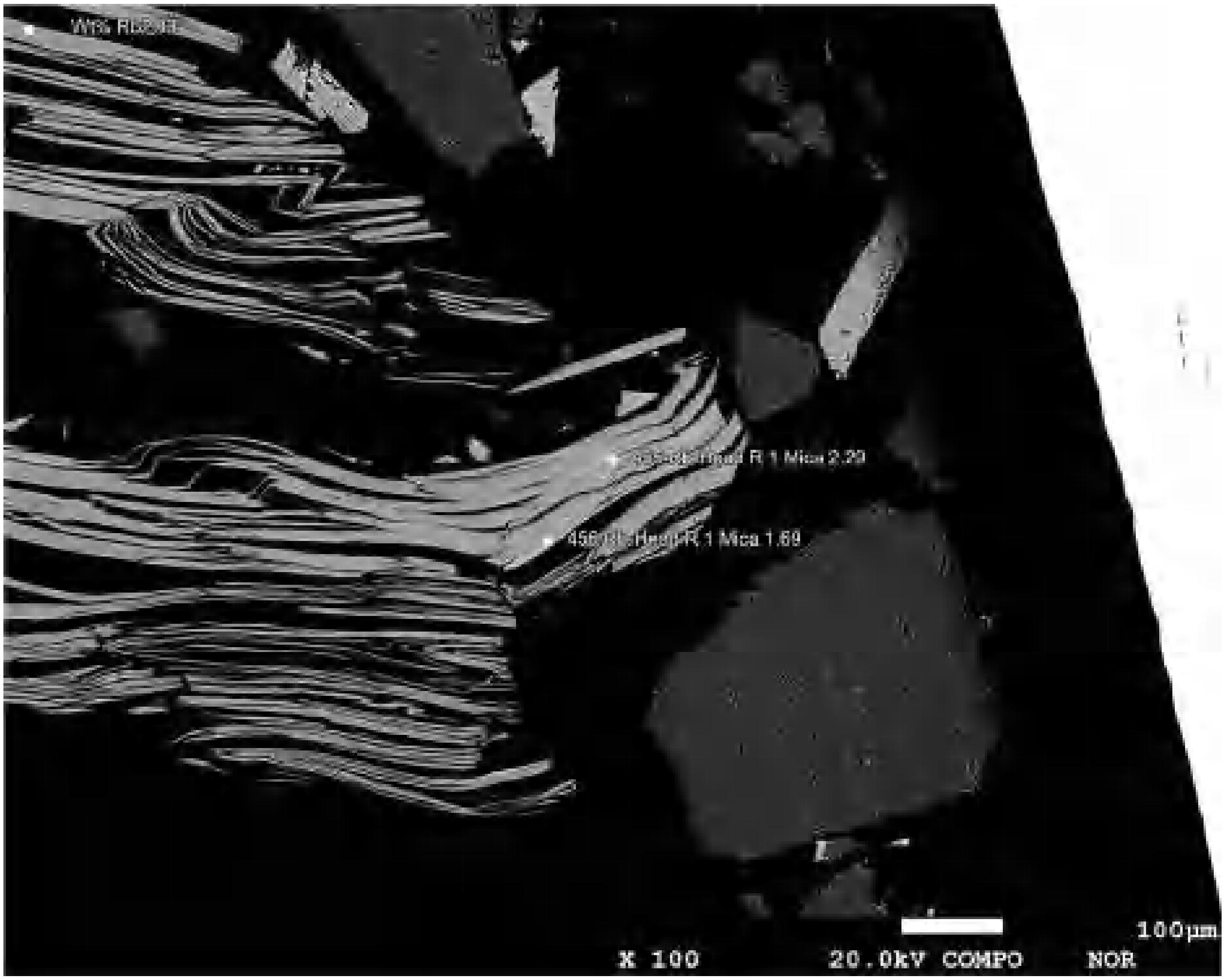

Wt% Rb2O3
446 Rb-Head R 1 Mica 1.99
100µm
x 100
20.0kV COMPO
NOR

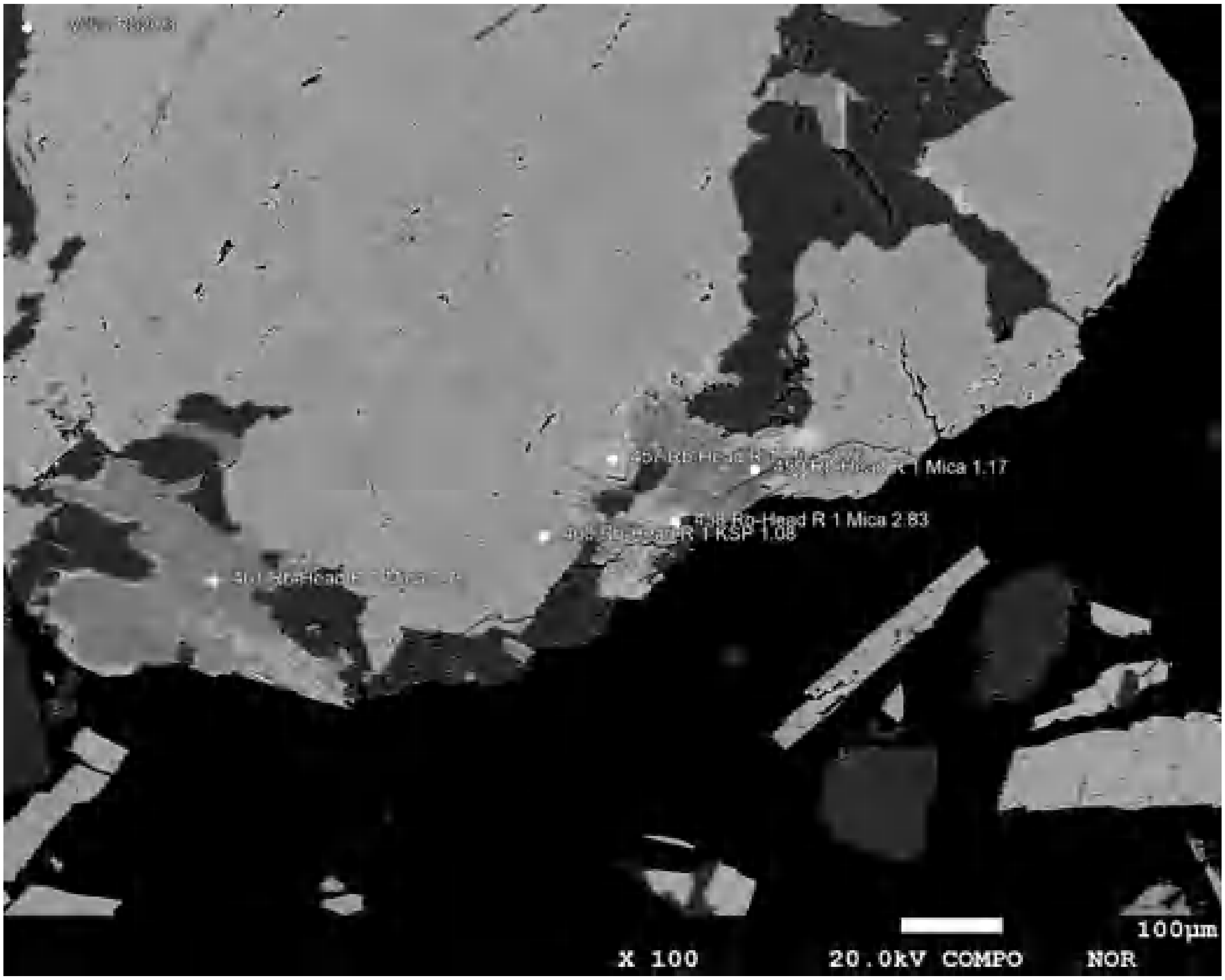

445 Rb-Head R 1 Mica 1.7
100μm
X 100
20.0kV COMPO
NOR

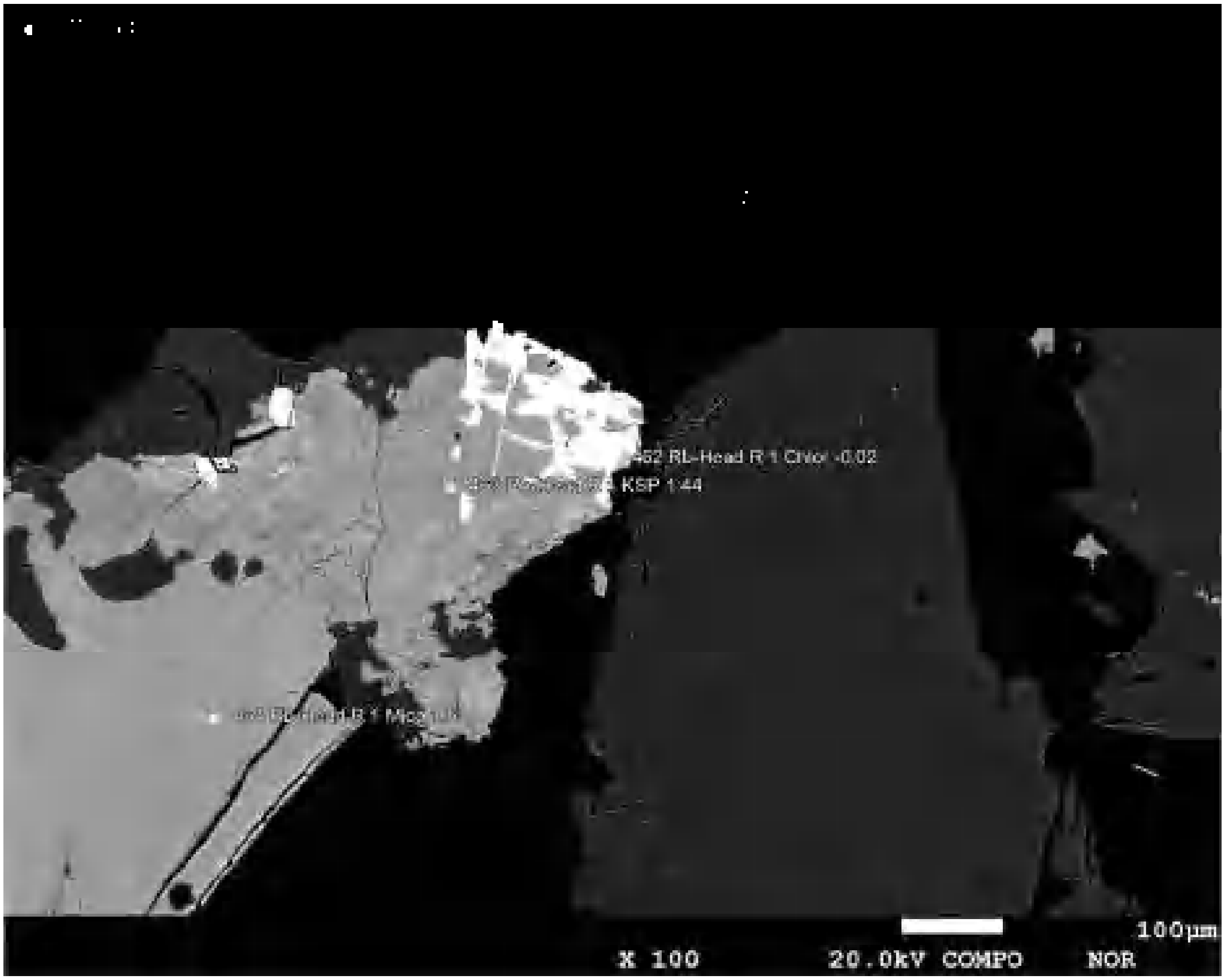

Wt% Rb2O3
R 1 Mica 1.79
R 1 Mica 1.71
100µm
X 100
20.0kV COMPO
NOR

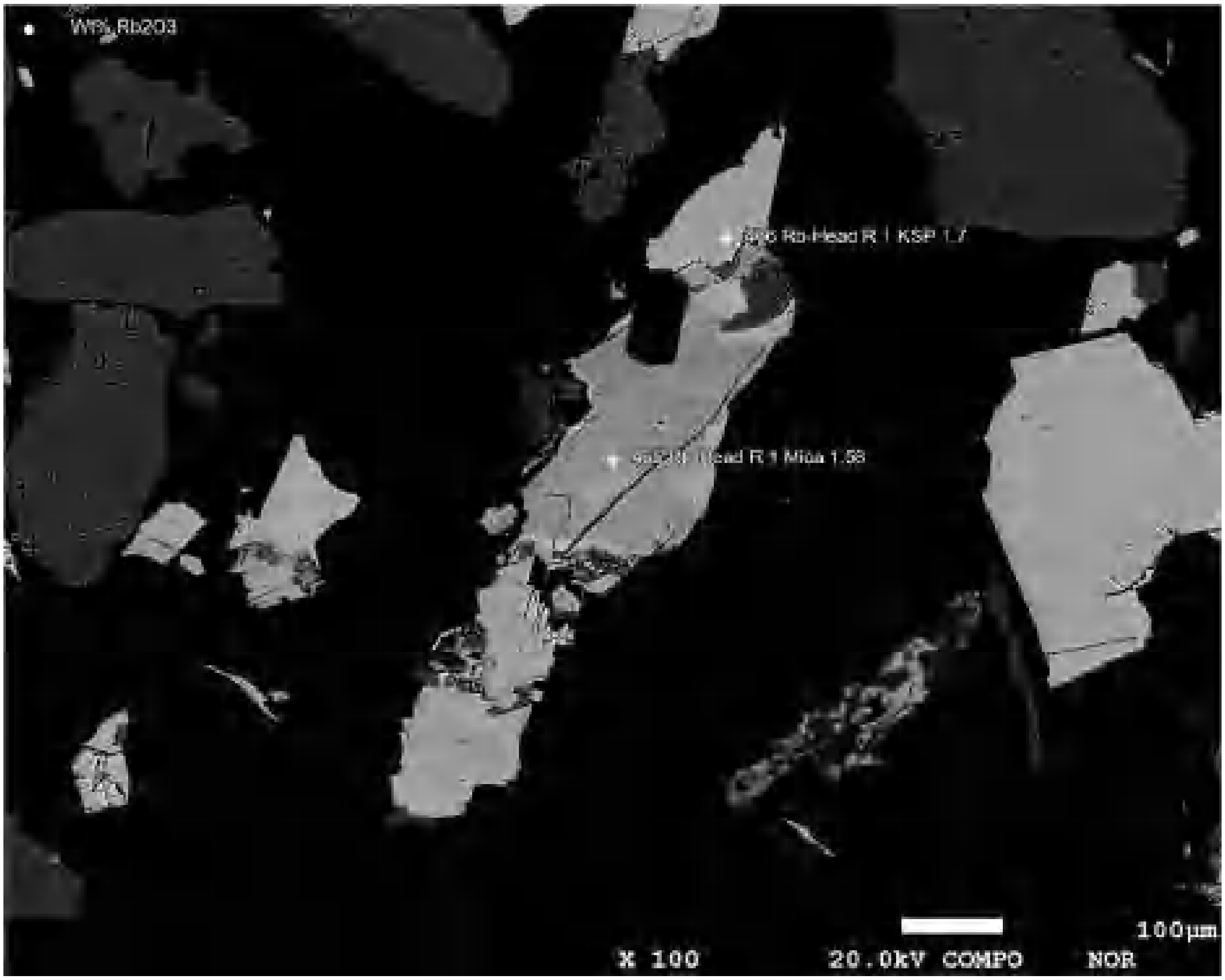

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

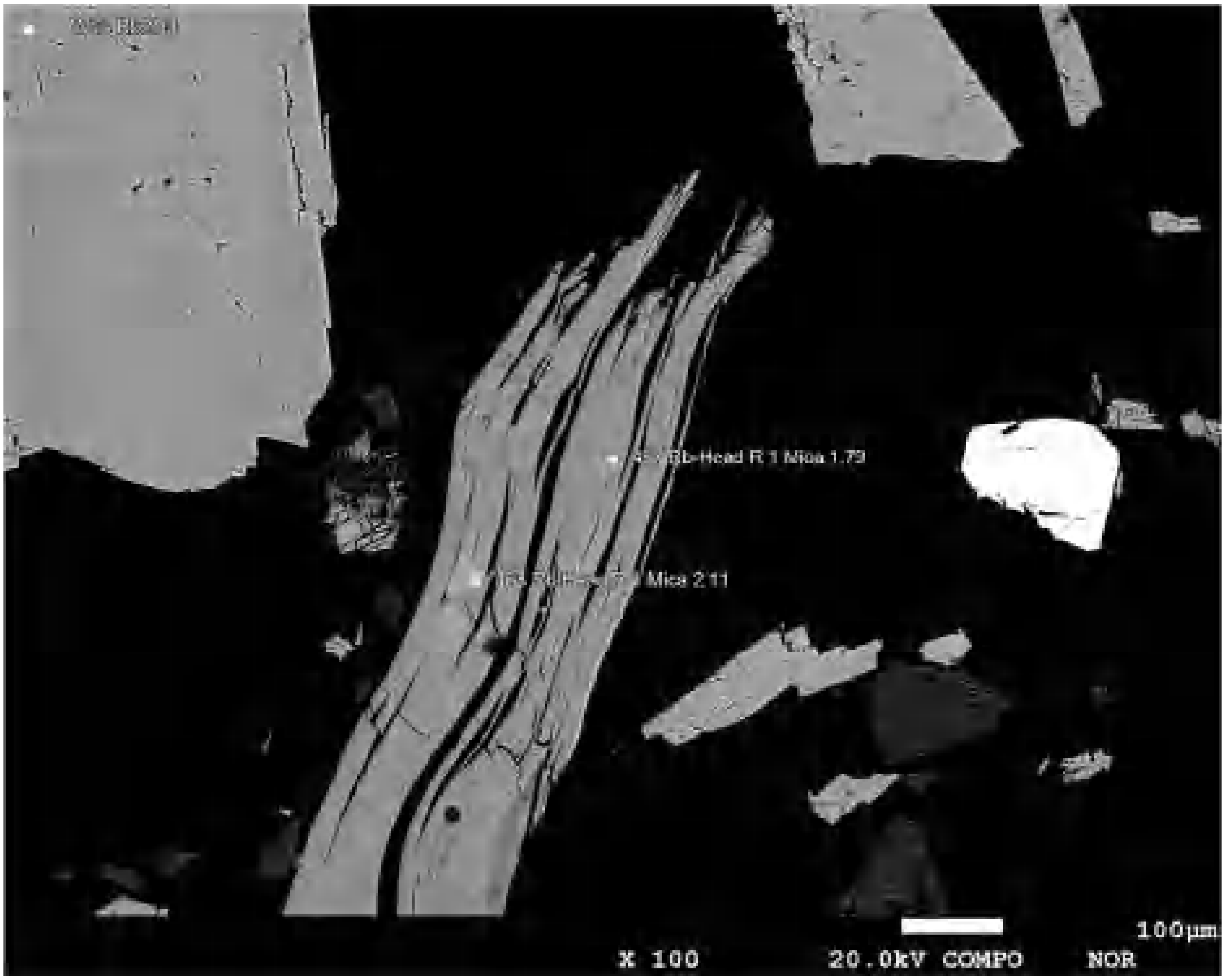

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

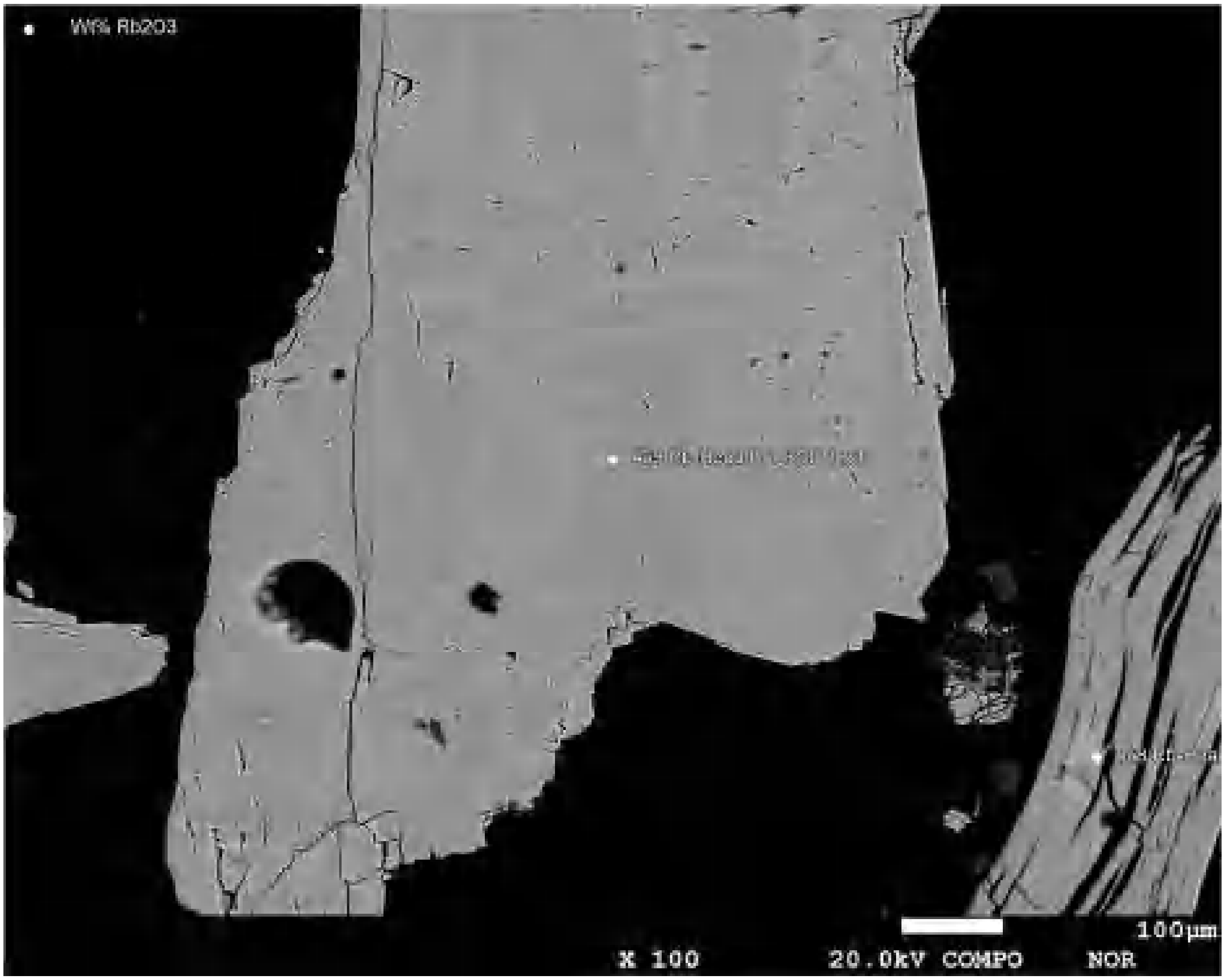

100μm
X 100
20.0kV COMPO
NOR

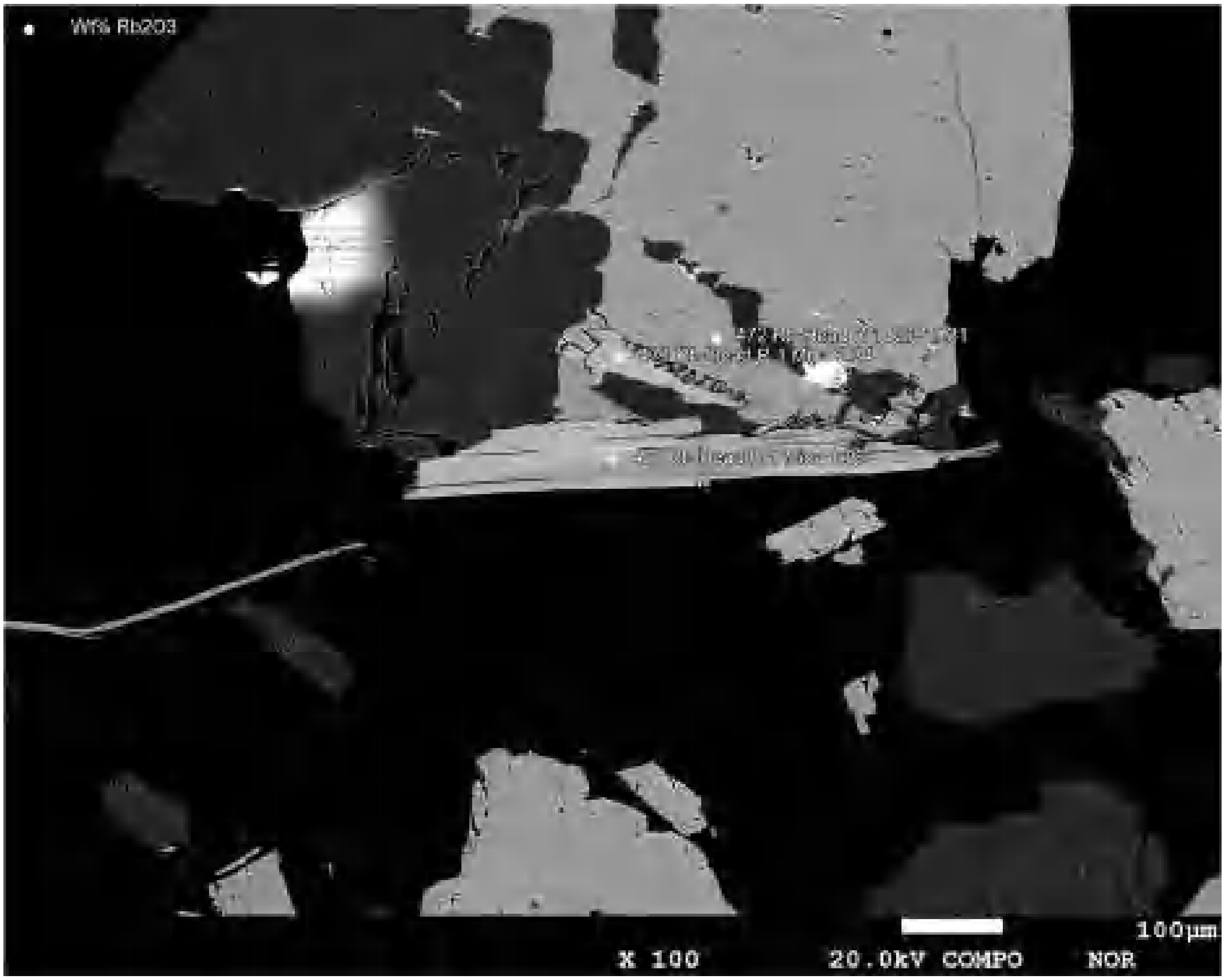

R 1 Mica 1.17
458 Rb-Head R 1 Mica 2.83
R 1 KSP 1.08
100µm
X 100
20.0kV COMPO
NOR

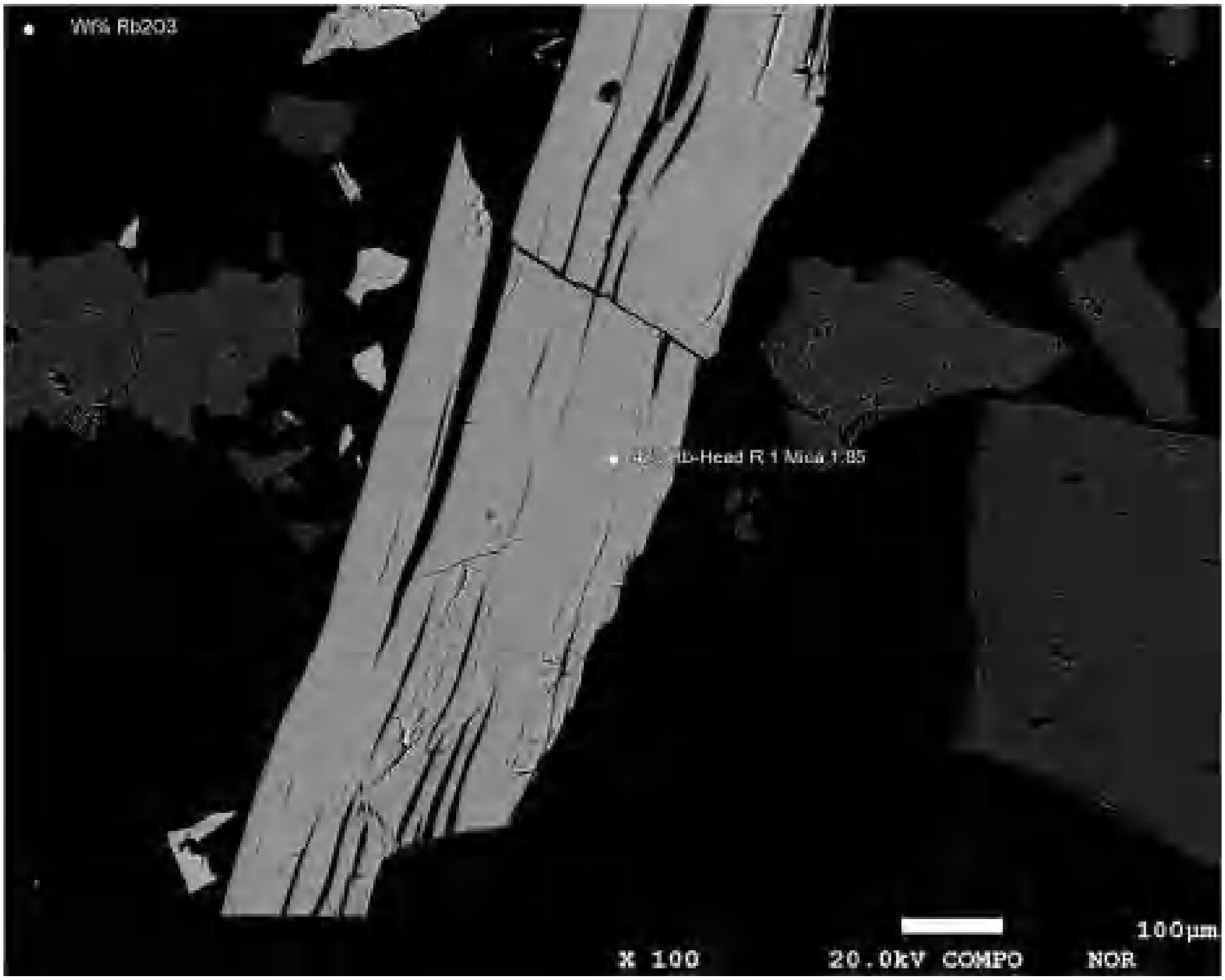

462 RL-Head R 1 Chlor -0.02
100µm
X 100
20.0kV COMPO
NOR

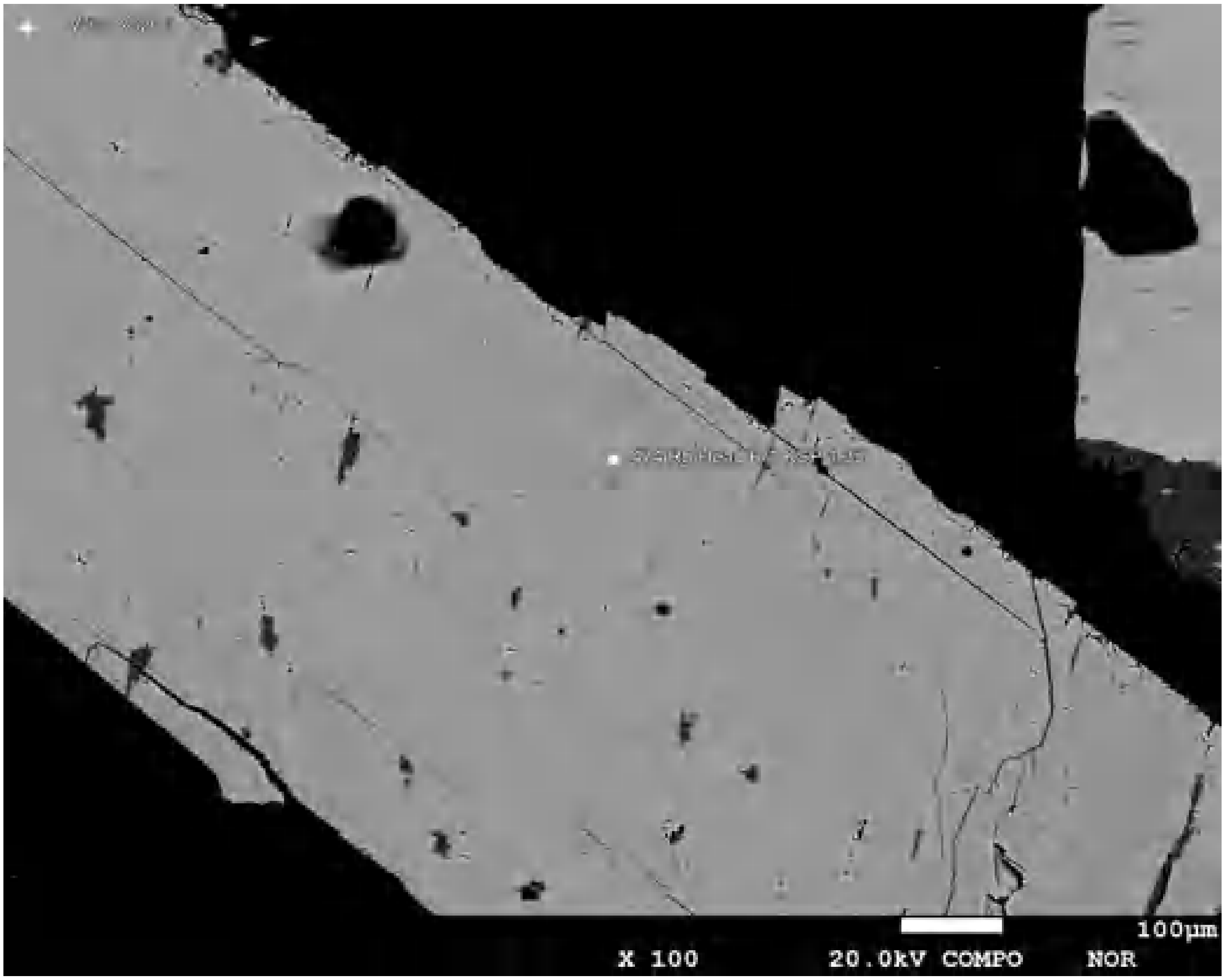

Wt% Rb2O3
Rb-Head R 1 KSP 1.7
Head R 1 Mica 1.58
100µm
X 100
20.0kV COMPO
NOR


100µm
X 100
20.0kV COMPO
NOR

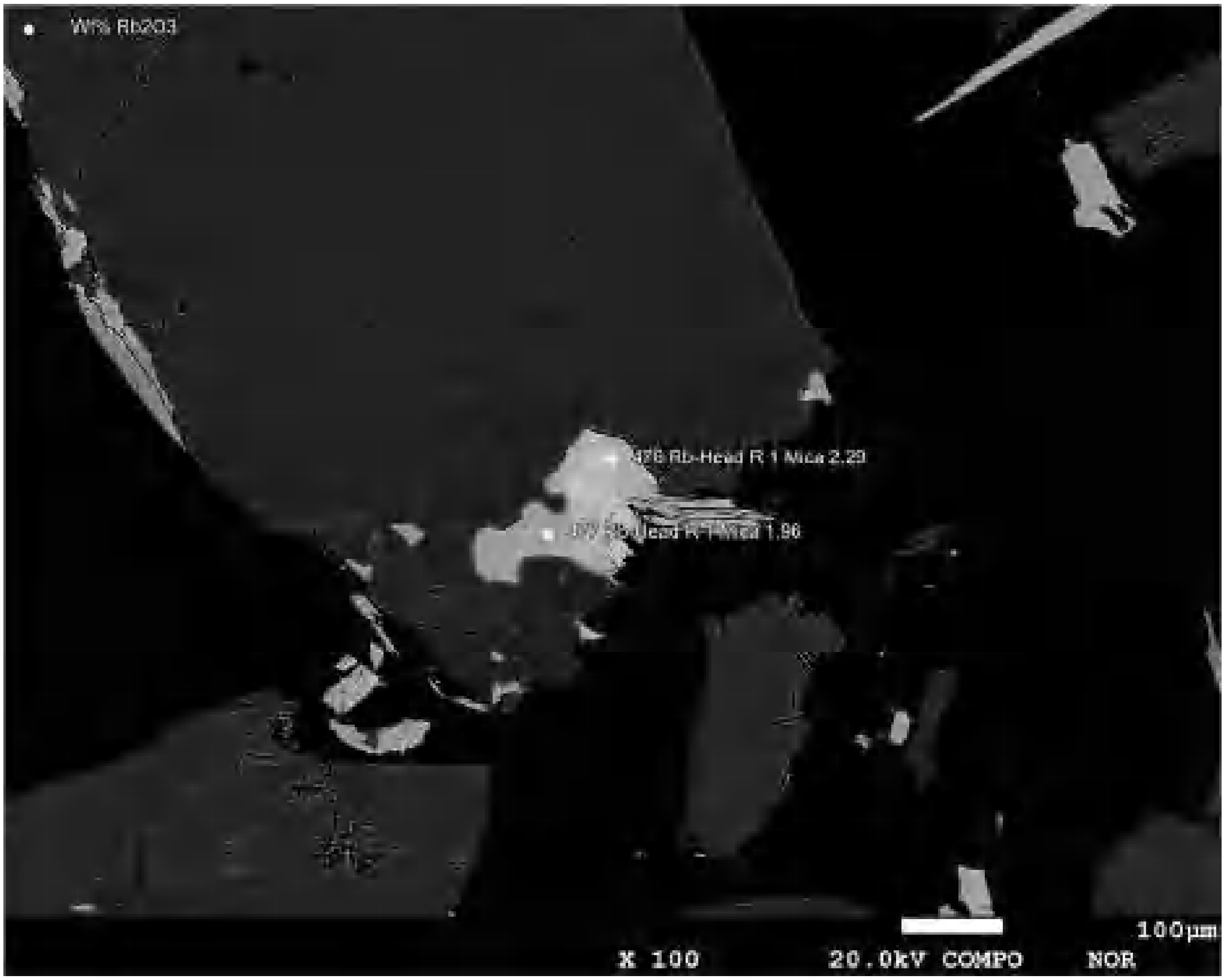

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

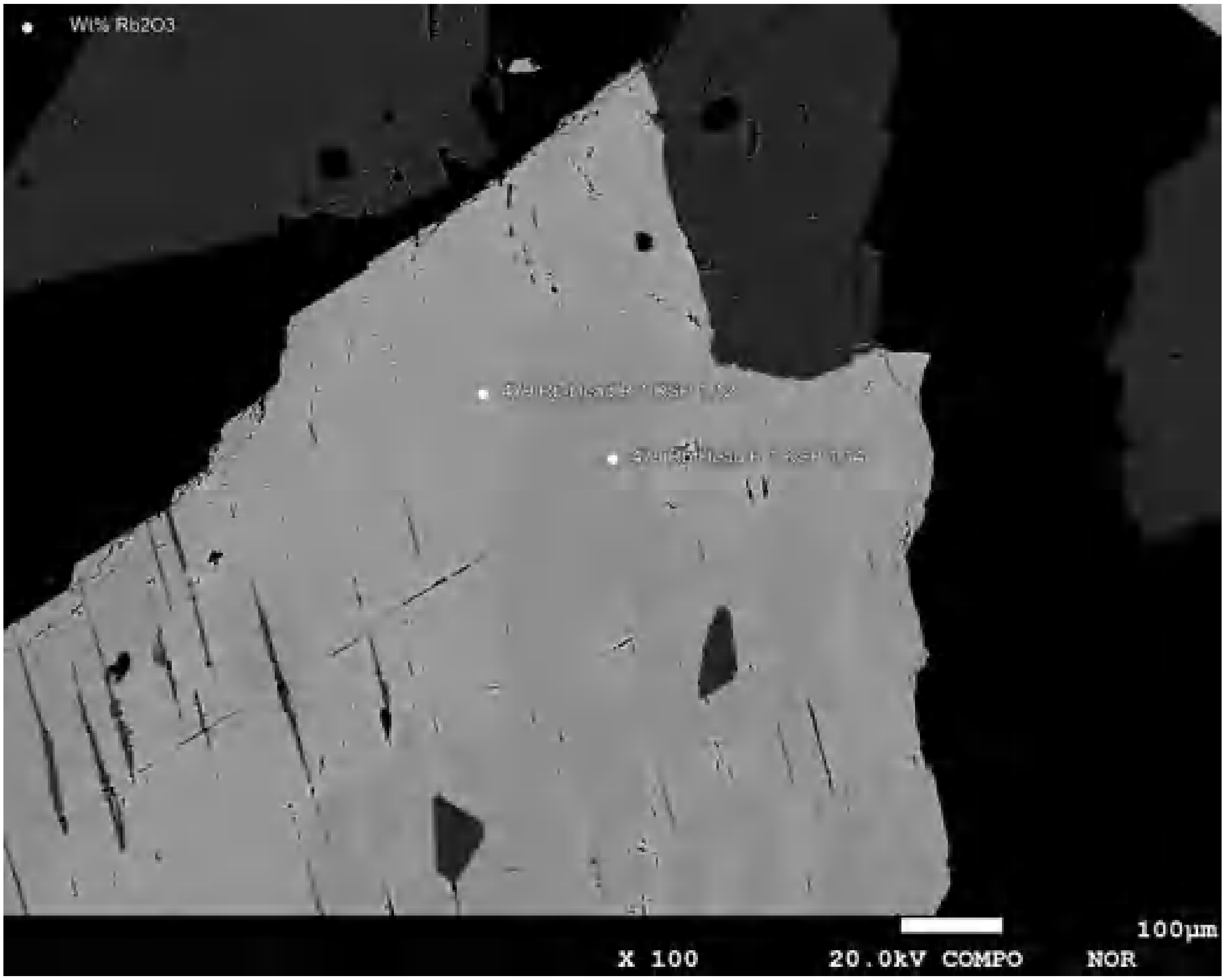

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

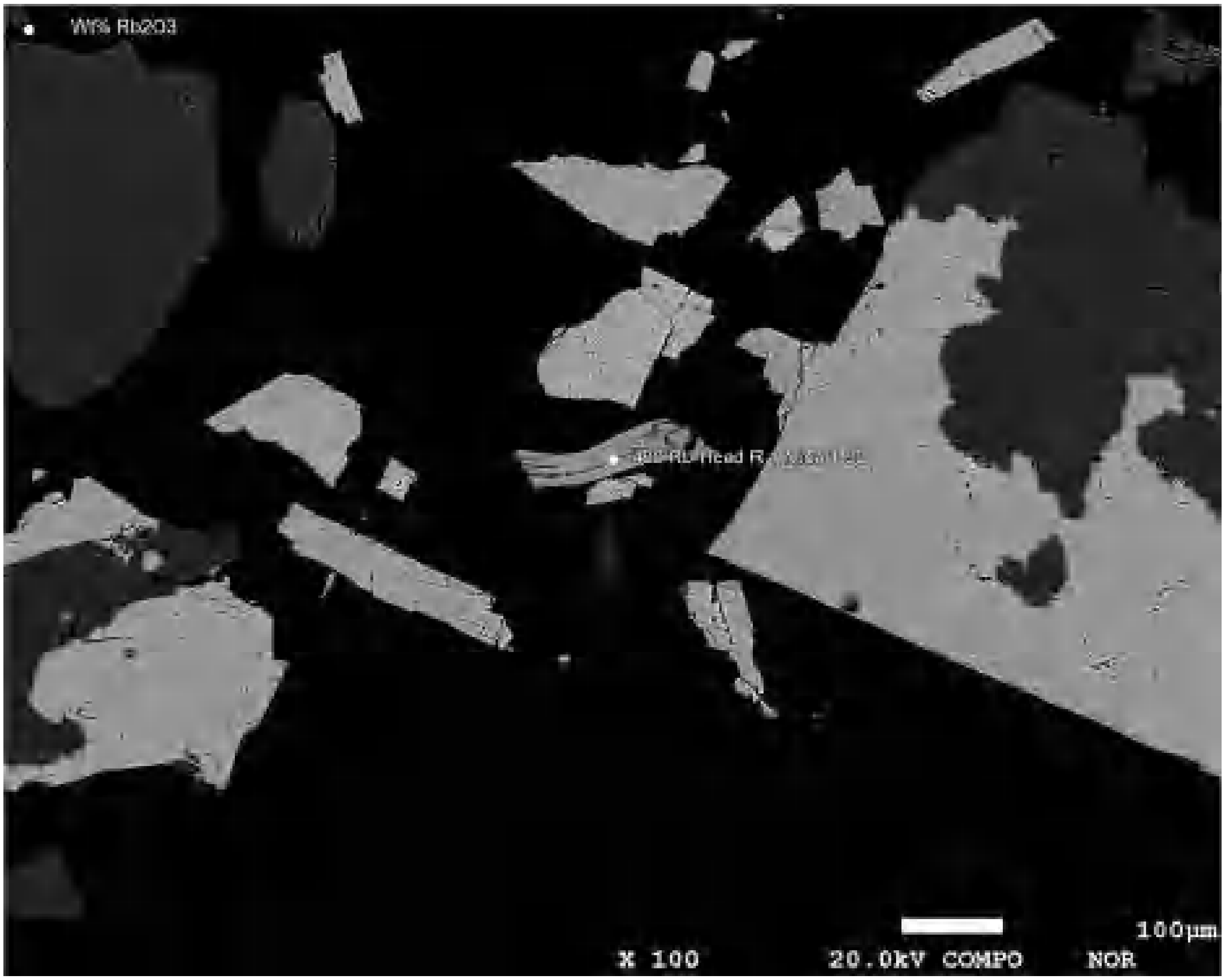

Wt% Rb2O3
Head R 1 Mica 1.85
100µm
X 100
20.0kV COMPO
NOR

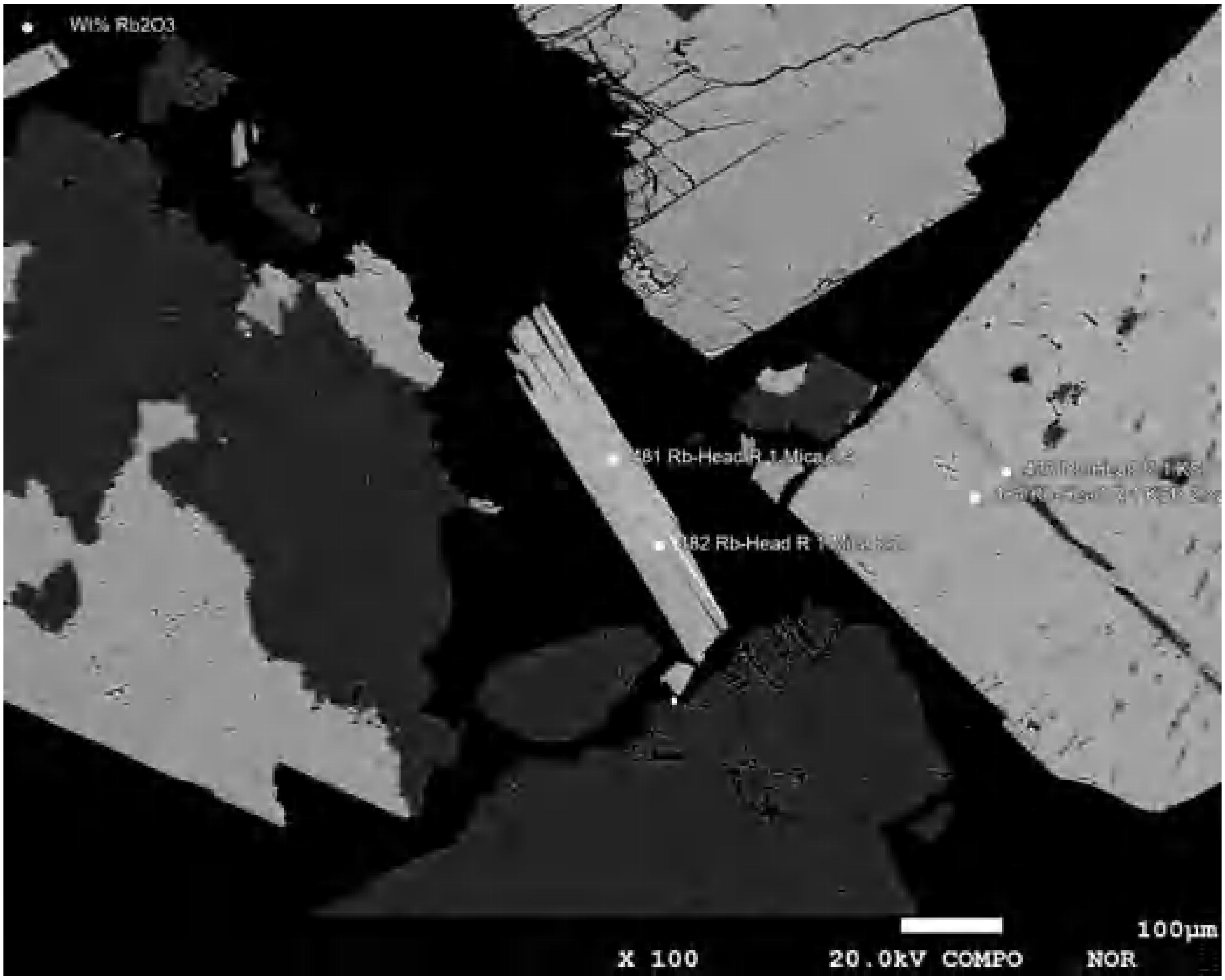

100µm
X 100
20.0kV COMPO
NOR


Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

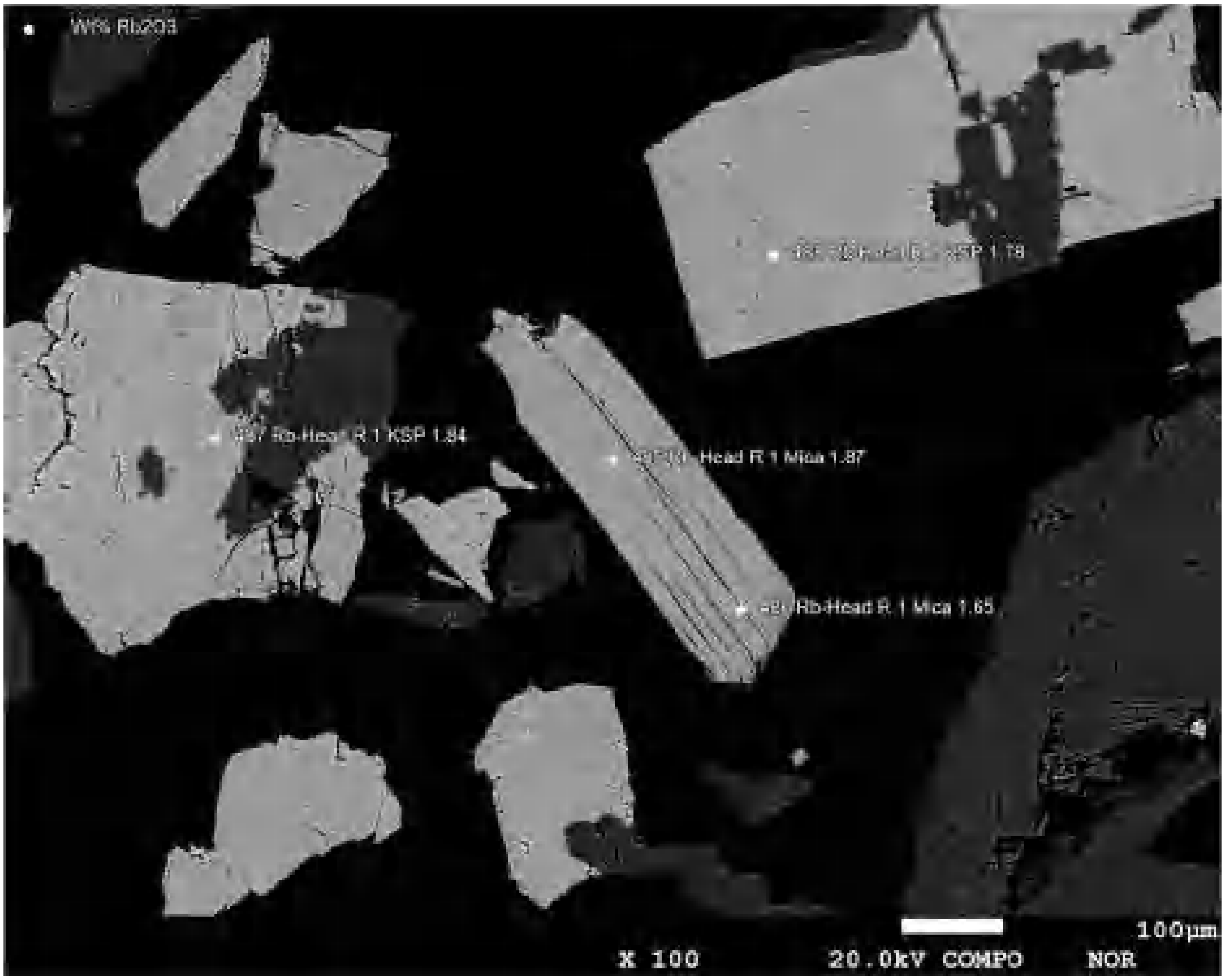

Wt% Rb2O3
476 Rb-Head R 1 Mica 2.29
100µm
X 100
20.0kV COMPO
NOR

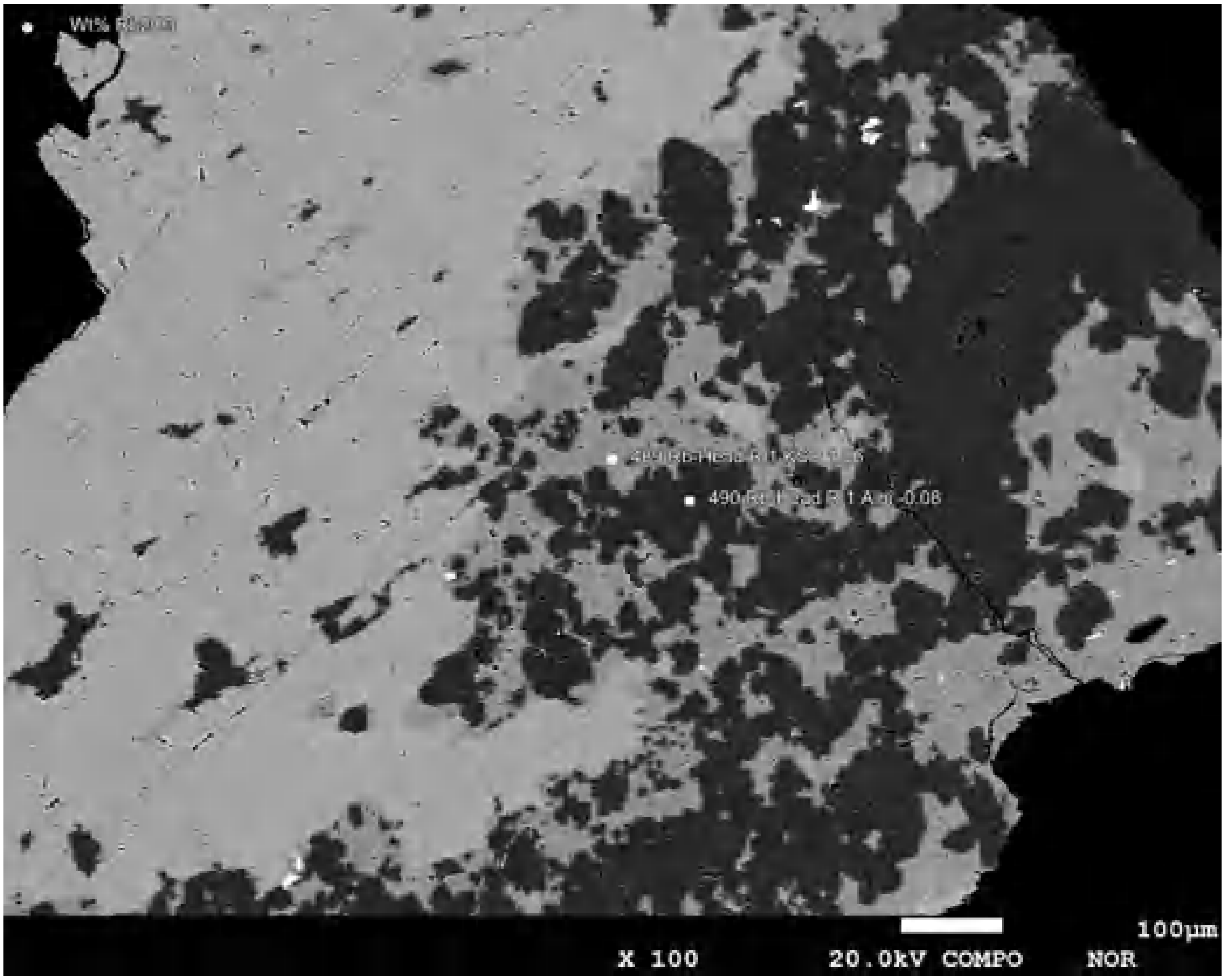

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

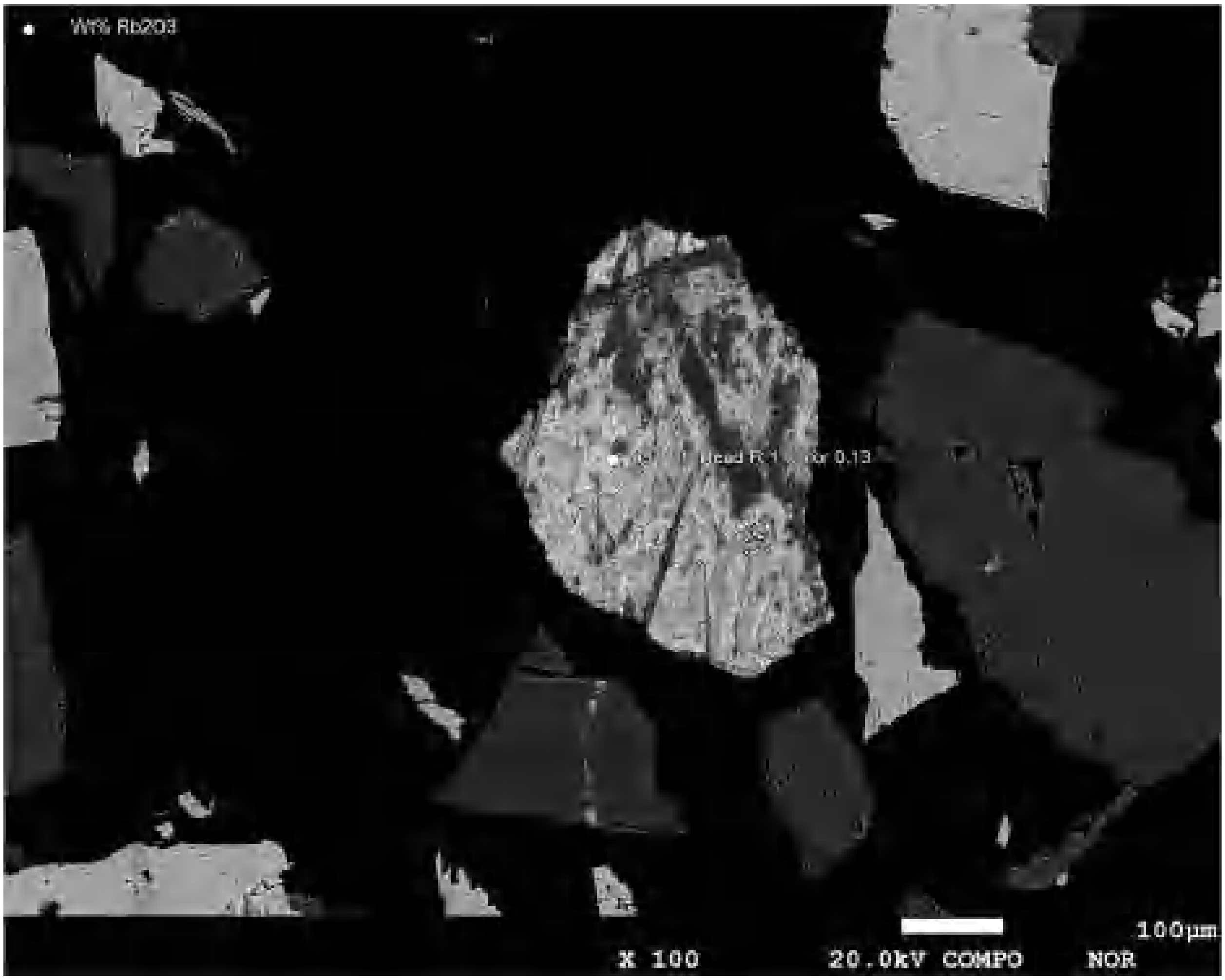

Wt% Rb2O3
100µm
x 100
20.0kV COMPO
NOR

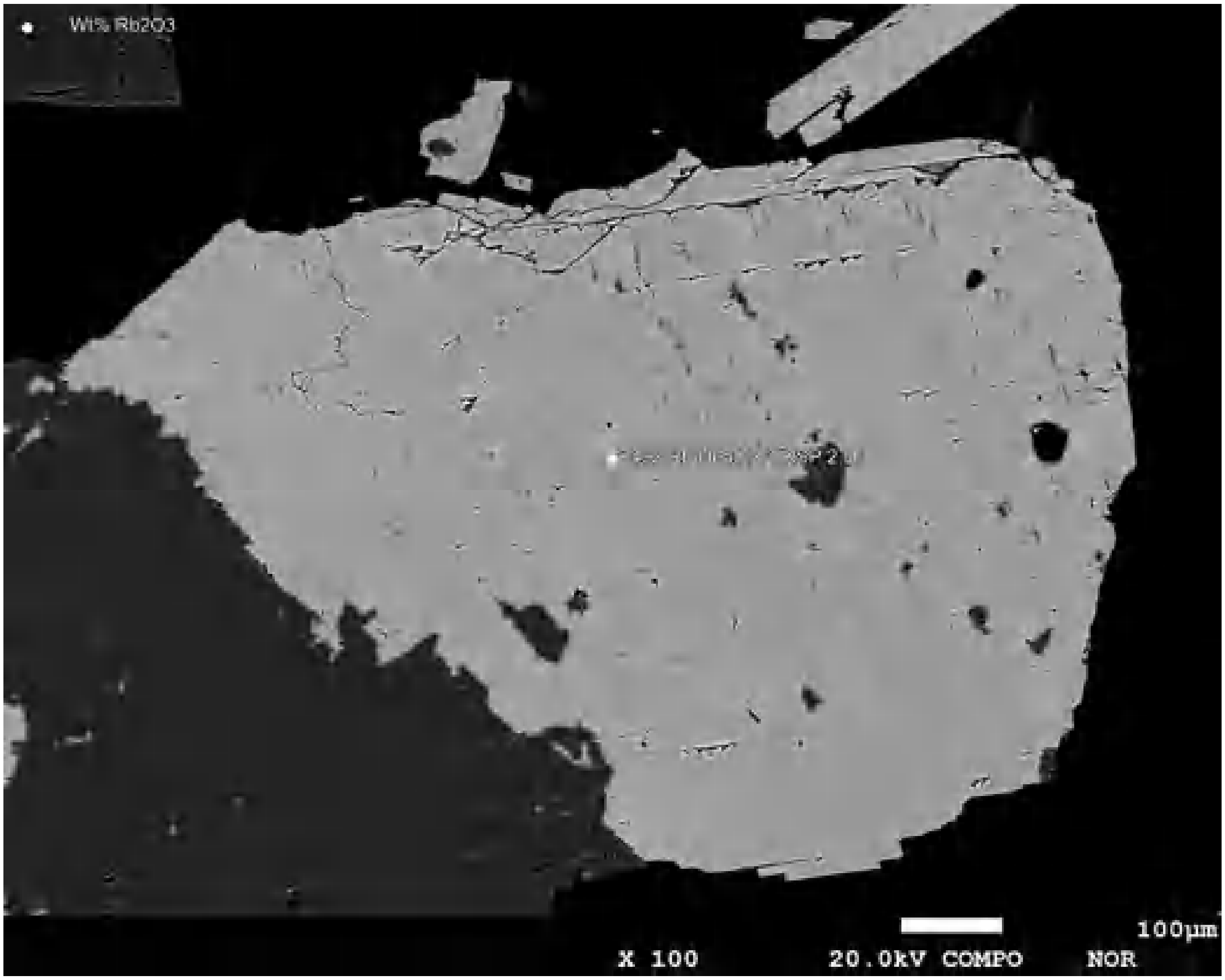


Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

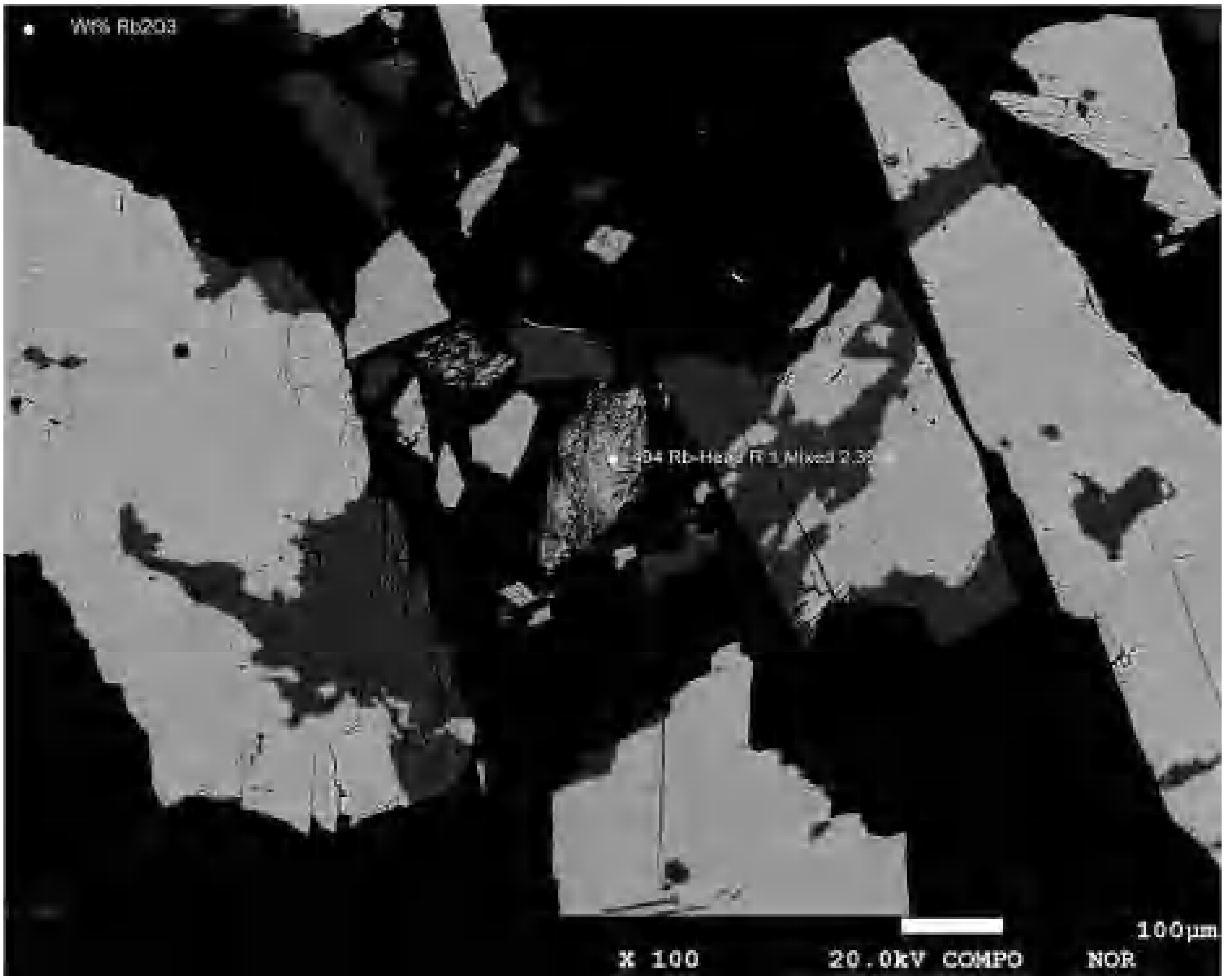

Wt% Rb2O3
KSP 1.78
R 1 KSP 1.84
Head R 1 Mica 1.87
Rb-Head R 1 Mica 1.65
100µm
X 100
20.0kV COMPO
NOR


100µm
X 100
20.0kV COMPO
NOR

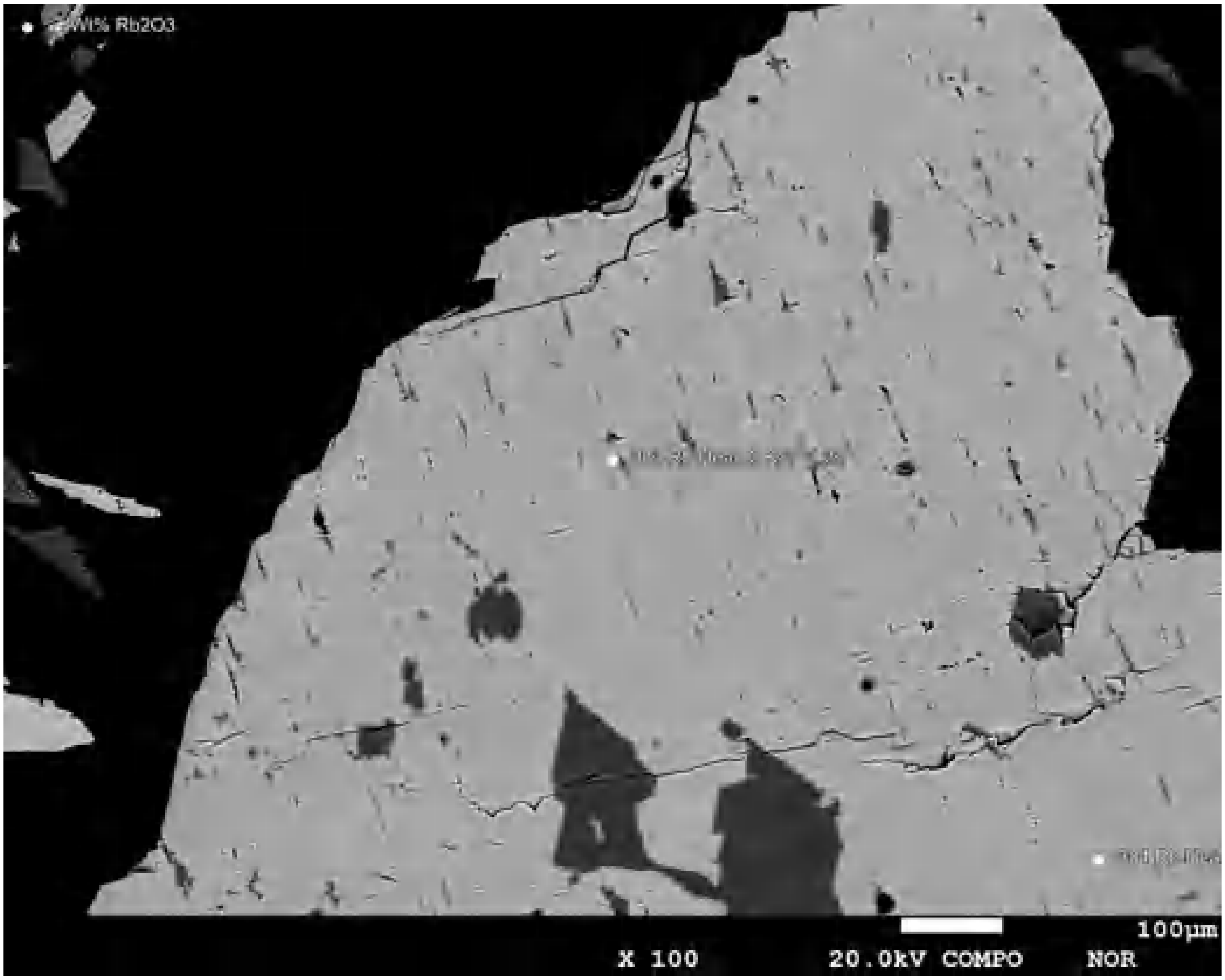

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

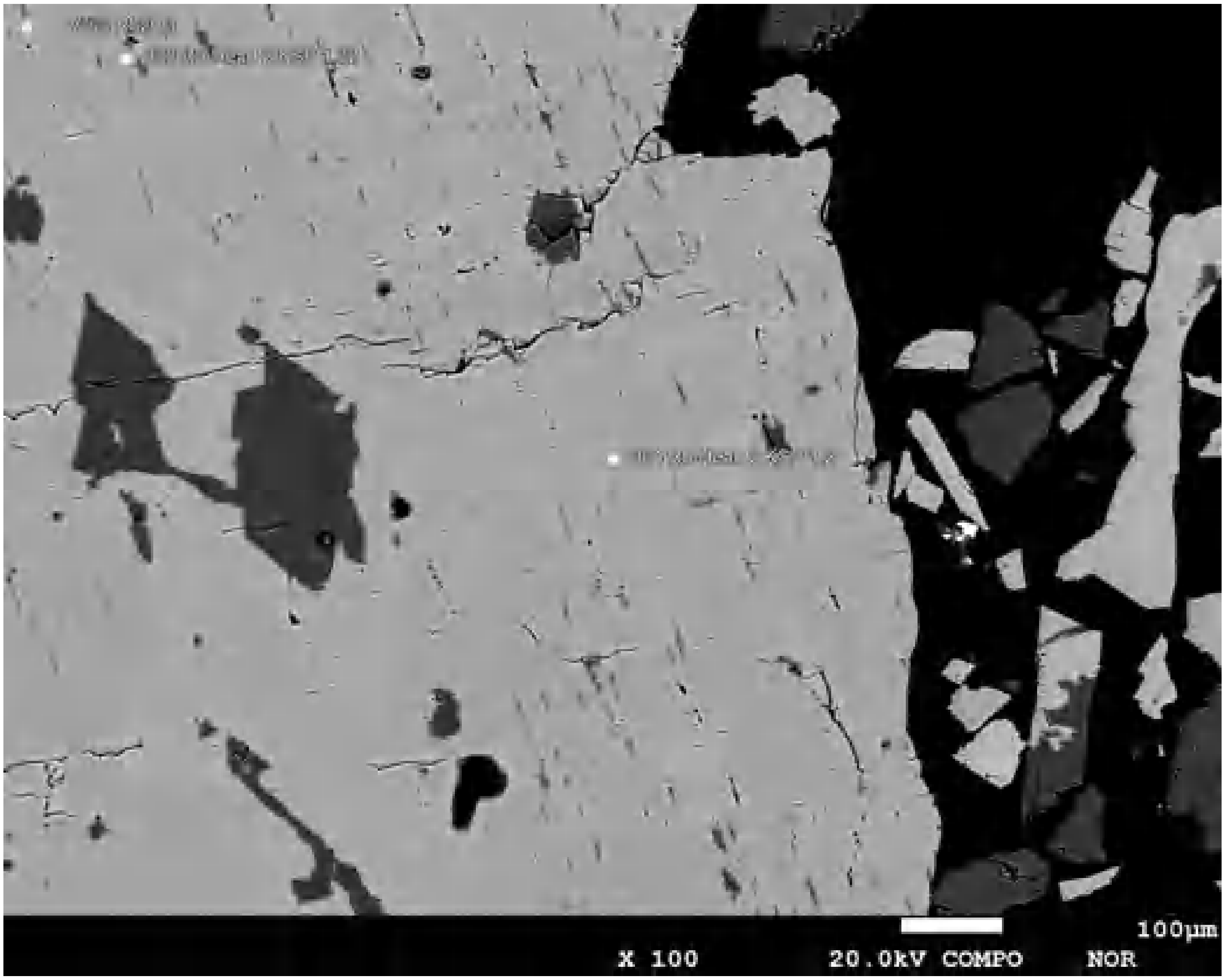

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

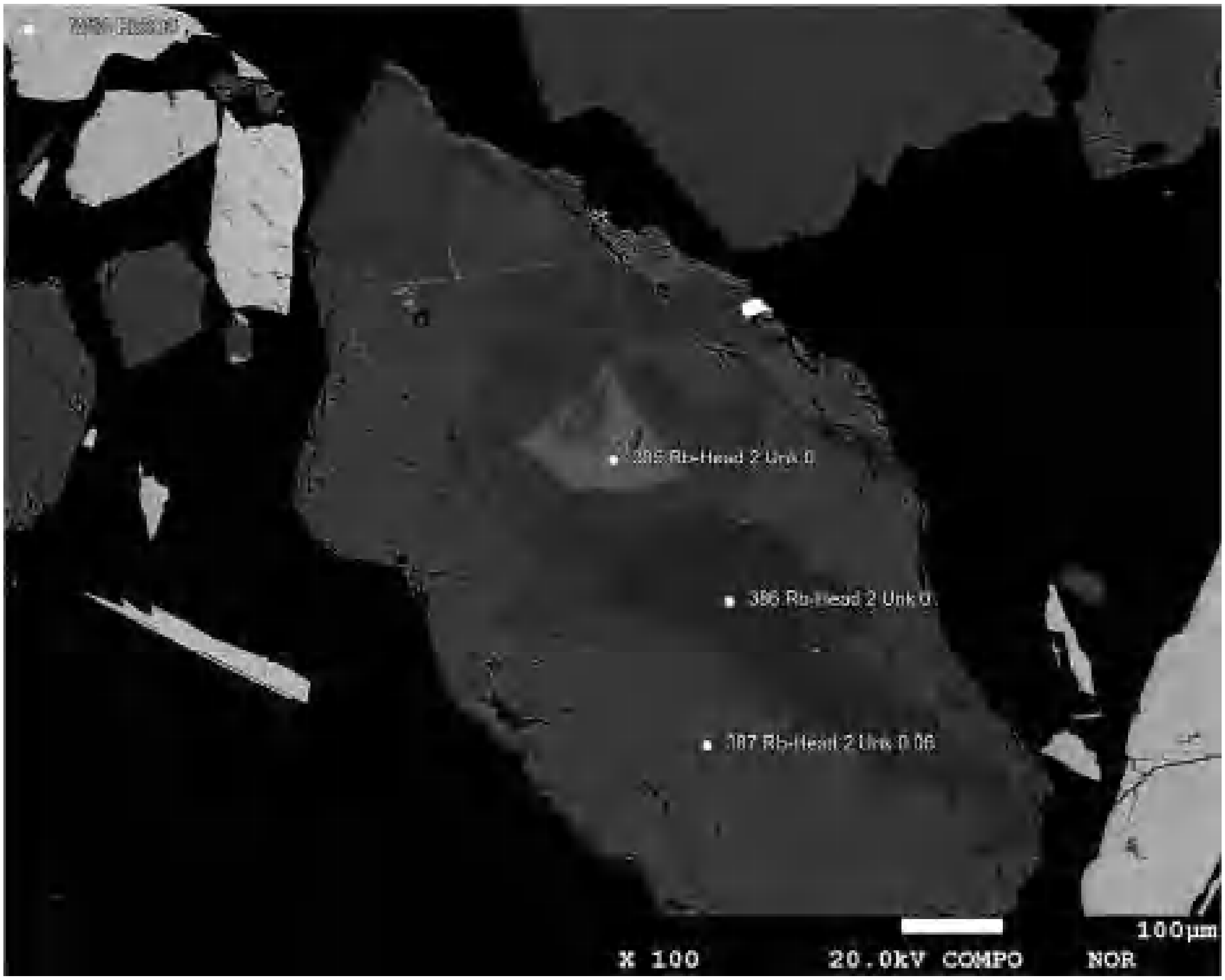

Wt% Rb2O3


Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

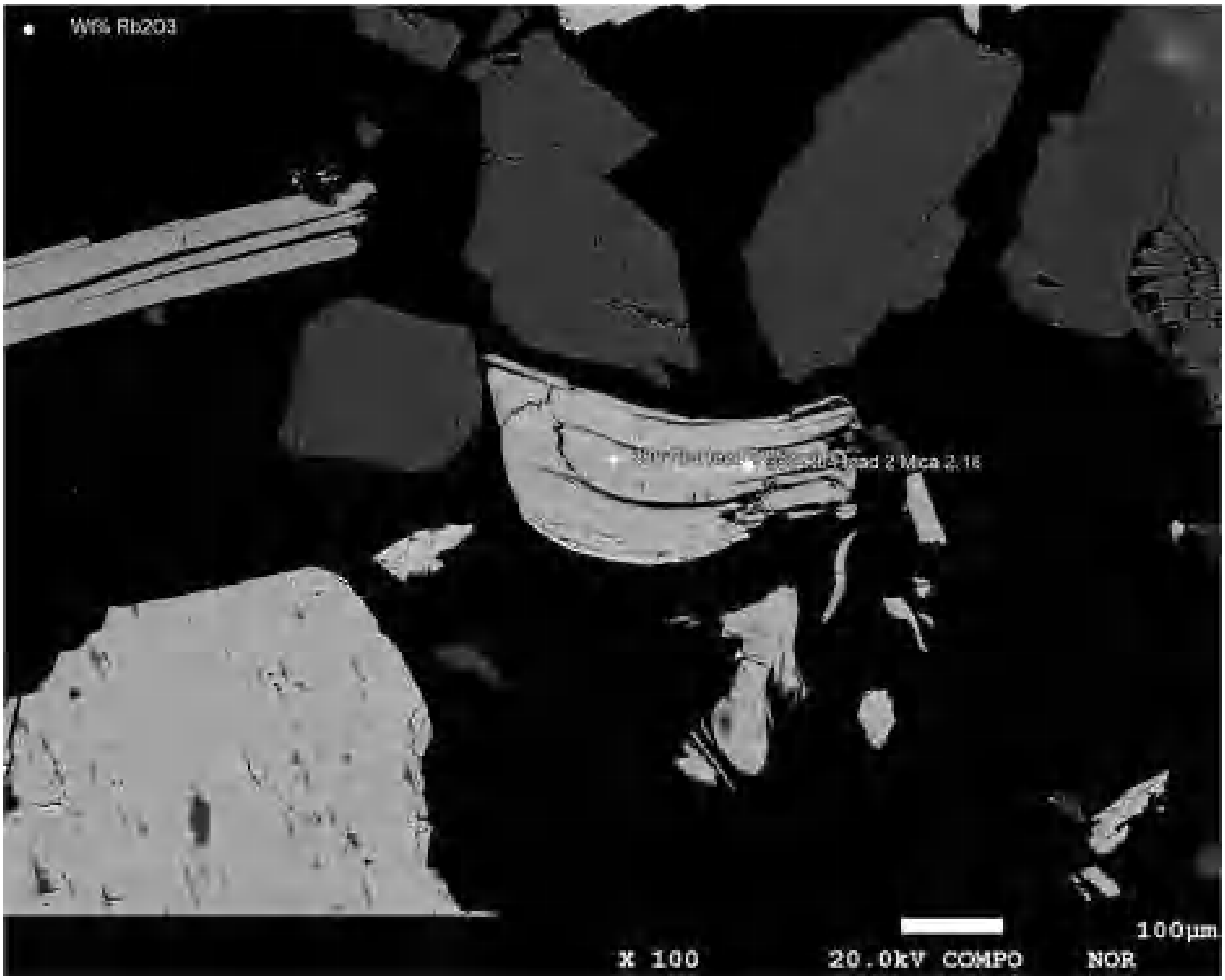

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

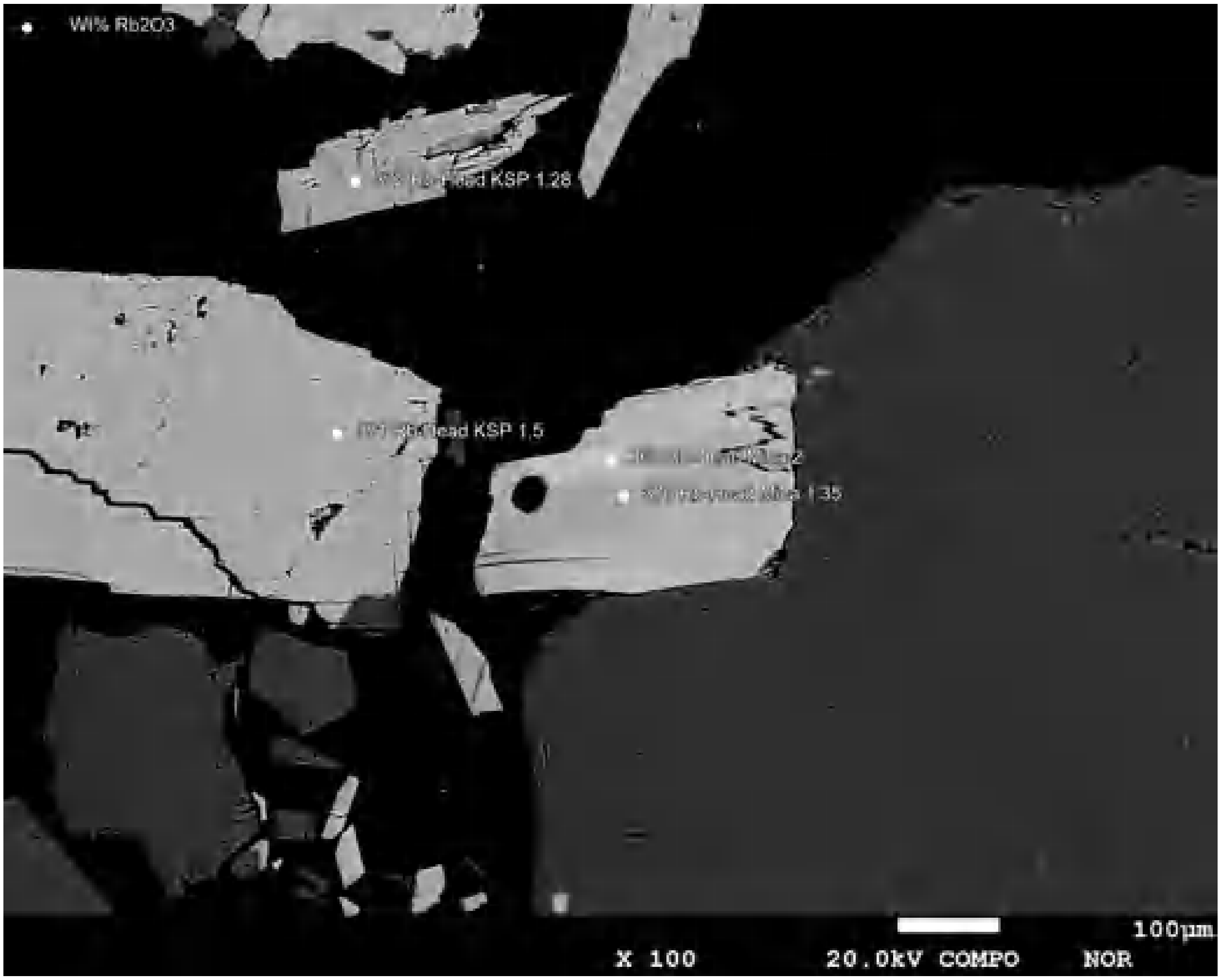

Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR


100µm
X 100
20.0kV COMPO
NOR

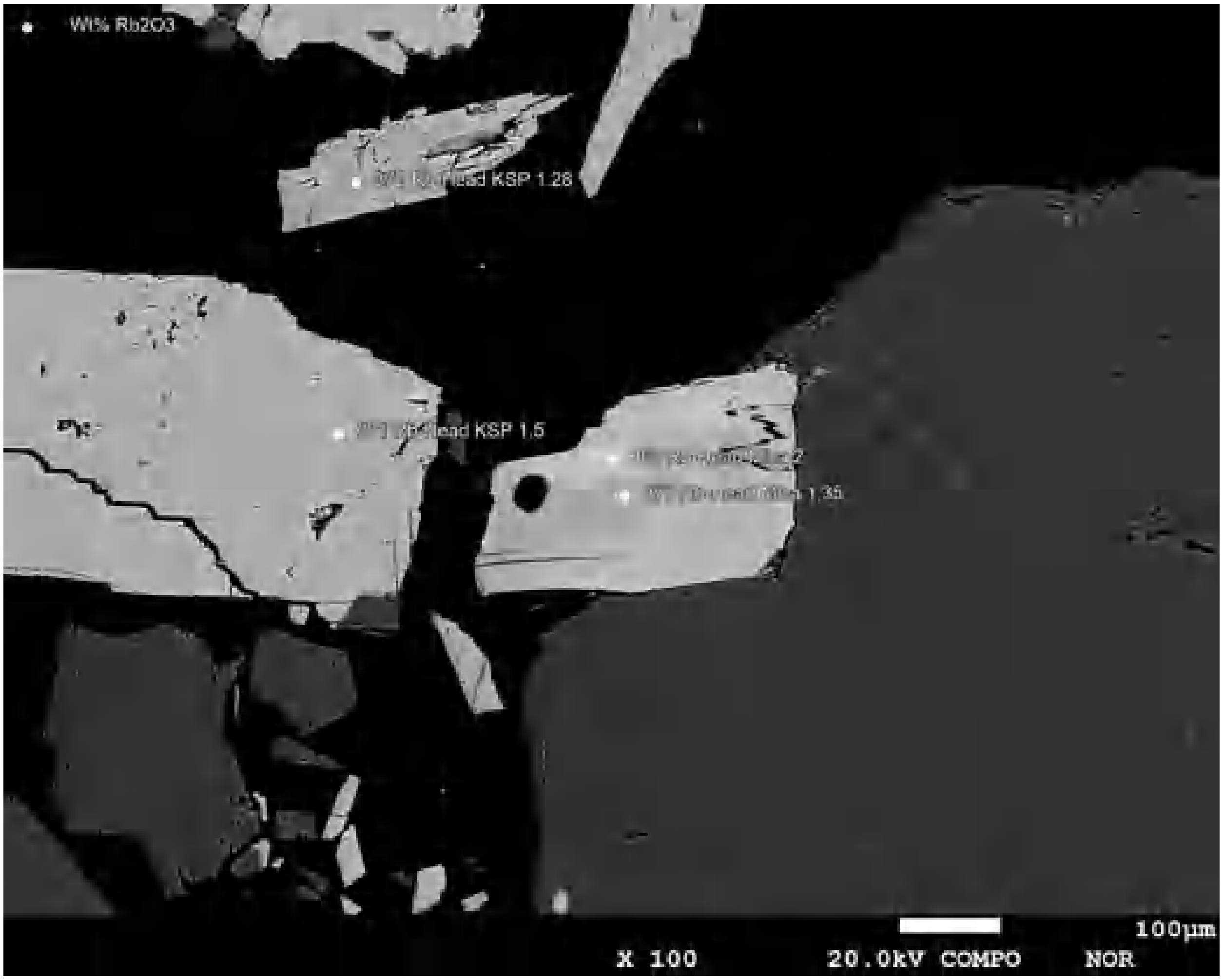

385 Rb-Head 2 Unk 0
386 Rb-Head 2 Unk 0.
387 Rb-Head 2 Unk 0.06
100µm
X 100
20.0kV COMPO
NOR


Wt% Rb2O3
X 100
20.0kV COMPO
NOR
100μm


Wt% Rb2O3
100µm
x 100
20.0kV COMPO
NOR



Wt% Rb2O3
KSP 1.28
KSP 1.5
100µm
X 100
20.0kV COMPO
NOR


Wt% Rb2O3
100µm
X 100
20.0kV COMPO
NOR

	EPMA												LA-ICP-MS (ppm)												
No.	oxide weight %	SiO2	Al2O3	TiO2	Fe2O3	MnO	MgO	CaO	Na2O	K2O	Li2O*	Total	Li	*Li*	Mg	Al	Si	Ca	Sc	Ti	V	Mn	Fe	Ga	Sn
	LOD	0.049016	0.038418	0.025558	0.023798	0.027318	0.012065	0.015395	0.015641	0.012443															
46	19977-01 Li-Head	64.51	26.45	0.01	1.87	0.24	0.00	0.01	0.14	0.01	7.95	101.18	35102.0	3.51	11.3	138710.2	304492.3	322.6	13.8	73.4	53.7	1624.3	9701.0	145.6	77.6
47	19977-01 Li-Head	64.49	27.06	0.00	0.92	0.18	0.01	0.02	0.10	0.00	7.95	100.73	35666.9	3.57	13.8	141982.3	304217.0	399.6	5.8	83.3	12.7	1550.0	4531.3	141.9	24.7
48	19977-01 Li-Head	64.86	26.86	0.00	1.15	0.09	0.00	0.00	0.11	0.00	7.95	101.03	34433.9	3.44	6.1	142679.1	302293.1	309.6	6.2	43.5	30.5	551.9	5596.3	130.8	8.8
49	19977-01 Li-Head	64.65	27.07	0.02	1.13	0.28	0.01	0.02	0.14	0.01	7.95	101.28	34630.8	3.46	6.6	140607.3	308234.8	279.8	5.7	71.6	20.7	2035.7	5604.8	144.1	24.0
50	19977-01 Li-Head	64.28	26.51	0.01	1.57	0.22	0.00	0.01	0.13	0.01	7.95	100.68	34799.3	3.48	11.8	139595.3	302228.5	324.0	8.1	54.3	21.1	1287.3	7115.0	129.8	13.5
51	19977-01 Li-Head	64.58	26.96	0.02	0.96	0.25	0.00	0.00	0.12	0.01	7.95	100.86	34270.1	3.43	700.0	141852.1	303832.3	643.2	6.4	56.1	3.0	1668.9	4946.2	117.2	14.0
52	19977-01 Li-Head	64.27	27.10	0.00	0.96	0.03	0.00	0.00	0.08	0.00	7.95	100.40	36171.2	3.62	2.0	139492.8	309367.8	325.8	6.7	32.6	19.1	288.8	4835.6	94.9	2.9
53	19977-01 Li-Head	64.85	27.03	0.00	1.14	0.06	0.00	0.00	0.10	0.00	7.95	101.12	35030.0	3.50	9.5	139769.5	310600.2	332.0	6.5	36.1	38.0	377.2	5926.6	125.3	6.3
54	19977-01 Li-Head	64.29	26.67	0.00	1.35	0.31	0.01	0.00	0.13	0.01	7.95	100.71	34451.1	3.45	6.0	138065.6	307549.4	397.0	11.6	56.8	22.2	2316.5	7868.9	119.3	24.5
55	19977-01 Li-Head	64.46	27.04	0.00	1.00	0.05	0.00	0.00	0.06	0.00	7.95	100.58	35540.6	3.55	<LOD	140171.6	307994.5	328.5	7.9	34.4	17.8	225.6	5995.5	109.9	6.1
56	19977-01 Li-Head	64.16	26.62	0.01	1.36	0.24	0.00	0.00	0.12	0.00	7.95	100.46	34775.4	3.48	22.2	139449.7	303231.8	308.8	7.7	69.4	38.2	1701.3	7127.5	136.5	55.7
57	19977-01 Li-Head	64.83	26.74	0.00	1.22	0.06	0.00	0.01	0.08	0.01	7.95	100.89	34372.4	3.44	5.0	140266.5	305940.3	278.0	5.9	48.5	23.3	384.7	5786.7	135.3	7.0
58	19977-01 Li-Head	64.34	26.87	0.02	1.07	0.14	0.00	0.00	0.12	0.00	7.95	100.50	34090.3	3.41	9.8	141144.5	303173.5	302.6	6.1	61.5	15.0	964.5	5530.8	115.0	5.7
59	19977-01 Li-Head	64.62	26.80	0.00	1.03	0.16	0.00	0.01	0.12	0.01	7.95	100.72	34915.4	3.49	13.0	141939.8	302066.5	313.9	6.4	50.9	39.9	1112.9	5660.0	131.1	10.2
60	19977-01 Li-Head	64.47	26.96	0.00	0.99	0.17	0.00	0.01	0.12	0.01	7.95	100.67	34583.4	3.46	19.4	138768.5	310308.6	310.5	6.8	50.6	5.2	1208.9	5054.2	123.5	22.4
61	19977-01 Li-Head	64.62	27.05	0.00	1.17	0.12	0.00	0.00	0.12	0.00	7.95	101.03	34376.8	3.44	30.9	142910.8	302769.1	350.8	7.1	46.4	2.4	857.1	5864.1	127.4	5.8
62	19977-01 Li-Head	64.36	26.78	0.02	1.17	0.10	0.00	0.00	0.10	0.00	7.95	100.49	32115.1	3.21	2.8	140642.2	303423.8	322.5	7.9	41.2	21.1	205.2	5126.7	125.4	11.0
63	19977-01 Li-Head	64.69	26.96	0.00	1.17	0.07	0.00	0.01	0.09	0.01	7.95	100.95	34245.4	3.42	3.3	141864.4	304339.9	322.4	6.4	34.9	27.7	371.1	4781.5	108.6	5.4
64	19977-01 Li-Head	64.59	27.03	0.00	1.07	0.09	0.00	0.01	0.10	0.00	7.95	100.85	34048.9	3.40	3.1	141042.8	306500.3	317.0	6.0	42.0	27.9	513.3	5547.8	101.0	14.9
65	19977-01 Li-Head	64.80	26.78	0.00	1.11	0.09	0.00	0.01	0.11	0.00	7.95	100.86	35254.7	3.53	21.3	142413.1	301663.4	349.6	7.0	43.2	28.4	598.4	5644.0	117.5	6.8
66	19977-01 Li-Head	64.74	27.21	0.01	1.08	0.11	0.00	0.01	0.11	0.01	7.95	101.22	34997.8	3.50	9.2	141405.3	308438.2	352.0	11.1	58.2	25.1	836.5	5899.4	127.5	7.1
67	19977-01 Li-Head	64.70	26.70	0.00	1.34	0.16	0.00	0.00	0.11	0.00	7.95	100.98	34328.7	3.43	14.2	140857.5	303589.0	362.1	6.6	47.3	<LOD	1155.4	6313.7	67.2	51.5
68	19977-01 Li-Head	64.63	27.00	0.00	1.13	0.07	0.00	0.01	0.08	0.00	7.95	100.89	34640.1	3.46	84.6	141931.6	304385.3	451.3	7.1	42.4	25.3	559.2	6075.6	128.4	21.9
69	19977-01 Li-Head	64.92	27.04	0.00	0.92	0.08	0.00	0.00	0.08	0.00	7.95	101.00	34538.9	3.45	<LOD	139926.0	310738.3	337.5	6.4	36.6	26.4	386.1	4973.5	117.6	6.3
70	19977-01 Li-Head	64.48	27.47	0.00	0.61	0.32	0.00	0.00	0.16	0.00	7.95	101.01	35267.6	3.53	12.6	141823.7	309406.5	329.7	6.1	63.8	125.8	2130.5	2981.1	152.9	33.5
	Min	64.16	26.45	0.00	0.61	0.03	0.00	0.00	0.06	0.00	7.95	100.40	32115.15	3.21	2.0	138065.6	301663.4	278.0	5.7	32.6	2.4	205.2	2981.1	67.2	2.9
	Max	64.92	27.47	0.02	1.87	0.32	0.01	0.02	0.16	0.01	7.95	101.28	36171.16	3.62	700.0	142910.8	310738.3	643.2	13.8	83.3	125.8	2316.5	9701.0	152.9	77.6
	Avg.	64.57	26.91	0.00	1.14	0.15	0.00	0.01	0.11	0.00	7.95	100.84	34665.88	3.47	44.3	140772.5	305631.4	346.8	7.3	51.2	27.9	996.4	5779.5	122.9	18.7

*Li2O (assumed)

		EPMA																	
No.	oxide or element weight %	SiO2	Al2O3	TiO2	FeO	MnO	MgO	CaO	BaO	Na2O	K2O	Rb2O	Cs2O	F	Cl	H2O*	O = F	O = Cl	Total
	LOD	0.071765	0.059115	0.055186	0.048758	0.041532	0.022681	0.033264	0.075757	0.030732	0.034268	0.074563	0.057125	0.076207	0.014251				
1	Muscovite 01	44.30	35.22	0.14	2.50	0.26	0.04	0.00	0.02	0.48	10.09	1.30	0.05	0.23	0.00	4.27	-0.10	0.00	98.82
2	Muscovite 02	44.33	34.96	0.09	2.28	0.18	0.25	0.03	0.03	0.32	10.04	1.53	0.18	0.18	0.00	4.29	-0.07	0.00	98.60
3	Muscovite 03	44.29	34.98	0.05	2.61	0.12	0.00	0.00	0.00	0.48	10.26	1.10	0.04	0.35	0.00	4.20	-0.15	0.00	98.33
4	Muscovite 04	44.59	34.61	0.19	2.30	0.13	0.30	0.03	0.00	0.37	9.83	1.73	0.28	0.29	0.00	4.23	-0.12	0.00	98.75
5	Muscovite 05	44.18	34.91	0.13	2.86	0.19	0.09	0.01	0.02	0.33	10.50	1.22	0.04	0.29	0.00	4.23	-0.12	0.00	98.87
6	Muscovite 06	44.37	35.50	0.14	2.19	0.14	0.10	0.00	0.01	0.42	10.16	1.27	0.06	0.18	0.00	4.31	-0.07	0.00	98.77
7	Muscovite 07	44.28	34.99	0.00	2.24	0.18	0.11	0.02	0.00	0.22	10.49	1.33	0.06	0.12	0.00	4.30	-0.05	0.00	98.29
8	Muscovite 08	44.39	35.90	0.06	2.14	0.11	0.02	0.04	0.00	0.43	10.15	1.35	0.08	0.12	0.01	4.35	-0.05	0.00	99.09
9	Muscovite 09	44.58	34.70	0.13	2.44	0.18	0.11	0.03	0.00	0.39	10.15	1.17	0.04	0.23	0.01	4.26	-0.10	0.00	98.33
10	Muscovite 10	44.26	35.26	0.06	2.61	0.14	0.01	0.02	0.00	0.54	10.24	1.05	0.06	0.29	0.01	4.24	-0.12	0.00	98.67
11	Muscovite 11	45.04	36.34	0.07	1.47	0.09	0.03	0.03	0.00	0.27	10.96	0.19	0.07	0.12	0.00	4.39	-0.05	0.00	99.01
12	R Muscovite 01	45.05	35.80	0.02	1.94	0.13	0.08	0.02	0.03	0.23	10.56	0.90	0.07	0.12	0.01	4.37	-0.05	0.00	99.27
13	R Muscovite 02	44.09	34.33	0.14	2.91	0.16	0.12	0.02	0.01	0.24	10.08	1.75	0.28	0.23	0.01	4.22	-0.10	0.00	98.48
14	R Muscovite 03	44.58	34.71	0.19	2.19	0.10	0.24	0.00	0.00	0.35	10.10	1.75	0.08	0.23	0.00	4.26	-0.10	0.00	98.70
15	R Muscovite 04	43.62	30.52	0.64	5.47	0.09	0.61	0.01	0.00	0.16	9.74	2.26	0.44	0.39	0.01	4.03	-0.17	0.00	97.82
16	R Muscovite 05	44.18	36.51	0.02	1.80	0.10	0.02	0.00	0.02	0.31	10.28	1.39	0.13	0.23	0.00	4.30	-0.10	0.00	99.20
17	R Muscovite 06	44.33	35.27	0.20	2.29	0.13	0.27	0.06	0.05	0.37	9.90	1.74	0.13	0.23	0.01	4.28	-0.10	0.00	99.16
18	R Muscovite 07	44.05	34.93	0.19	2.46	0.18	0.17	0.02	0.00	0.29	10.40	1.22	0.04	0.35	0.00	4.19	-0.15	0.00	98.35
19	R Muscovite 08	44.04	35.17	0.05	2.38	0.11	0.05	0.00	0.00	0.29	10.37	1.45	0.03	0.18	0.00	4.27	-0.07	0.00	98.31
20	R Muscovite 09	44.46	34.56	0.10	2.92	0.16	0.07	0.00	0.00	0.41	10.21	1.29	0.07	0.35	0.01	4.20	-0.15	0.00	98.66
21	R Muscovite 10	44.95	32.06	0.45	3.73	0.61	0.07	0.00	0.00	0.13	10.26	1.69	0.21	0.46	0.01	4.09	-0.19	0.00	98.55
22	R Muscovite 11	44.53	35.79	0.11	2.29	0.16	0.05	0.00	0.00	0.49	10.19	1.13	0.02	0.23	0.00	4.30	-0.10	0.00	99.19
23	R Muscovite 12	44.20	35.80	0.23	1.68	0.10	0.17	0.03	0.02	0.47	9.89	1.83	0.16	0.35	0.00	4.23	-0.15	0.00	99.01
24	R Muscovite 13	44.76	35.17	0.09	2.49	0.14	0.01	0.01	0.04	0.45	10.34	1.00	0.07	0.29	0.00	4.26	-0.12	0.00	99.00
25	R Muscovite 14	44.58	36.08	0.11	2.26	0.13	0.06	0.03	0.02	0.53	10.29	0.77	0.01	0.12	0.00	4.38	-0.05	0.00	99.32
	Min	43.62	30.52	0.00	1.47	0.09	0.00	0.00	0.00	0.13	9.74	0.19	0.01	0.12	0.00	4.03	-0.19	0.00	97.82
	Max	45.05	36.51	0.64	5.47	0.61	0.61	0.06	0.05	0.54	10.96	2.26	0.44	0.46	0.01	4.39	-0.05	0.00	99.32
	Avg.	44.40	34.96	0.14	2.50	0.16	0.12	0.02	0.01	0.36	10.22	1.34	0.11	0.25	0.00	4.26	-0.10	0.00	98.74

		LA-ICP-MS (ppm)																						
No.	oxide or element weight %	Li	Li%	Be	B	Mg	Al	Si	K	Ca	Sc	Ti	V	Mn	Fe	Zn	Ga	Ge	Rb	Nb	Sn	Cs	Ba	Ta
1	Muscovite 01	1157.5	***0.12***	21.5	8.8	304.5	182419.9	211775.0	83768.9	218.3	4.4	516.8	8.0	1798.1	12993.0	814.3	314.6	4.8	10671.9	222.0	169.5	378.6	1.5	59.0
2	Muscovite 02	779.9	***0.08***	13.3	11.3	1448.4	181132.6	211893.5	87222.6	225.0	5.8	293.0	42.4	1157.1	11368.3	322.9	369.6	4.5	13942.1	130.4	350.1	1206.6	62.8	24.5
3	Muscovite 03	1077.3	***0.11***	19.2	10.0	86.9	178736.6	214837.2	85671.6	255.6	4.1	338.7	5.9	988.9	14774.1	627.4	300.0	4.0	9786.3	215.8	90.5	420.2	2.5	40.2
4	Muscovite 04	1043.8	***0.10***	17.3	9.8	1883.3	178024.5	214782.4	86867.4	262.5	6.7	964.3	96.4	1040.2	12529.7	394.4	408.6	2.7	15309.8	151.1	435.2	2339.0	60.8	128.1
5	Muscovite 05	1013.7	***0.10***	20.2	9.5	492.3	177161.9	215868.6	88231.4	196.5	4.8	723.8	127.3	1086.2	15095.1	422.6	300.5	3.0	10522.8	220.5	204.0	579.2	2.2	63.4
6	Muscovite 06	707.0	***0.07***	24.5	9.1	510.8	181238.5	215395.6	88864.6	227.9	6.4	643.2	3.1	1275.8	11580.3	539.6	347.4	4.3	11270.6	175.9	179.9	411.8	2.9	76.3
7	Muscovite 07	665.1	***0.07***	12.2	10.8	737.0	177702.1	216201.9	90638.8	328.2	5.2	166.4	42.6	1192.2	12915.6	192.6	301.1	5.1	11332.2	62.4	183.1	481.2	132.2	15.6
8	Muscovite 08	683.4	***0.07***	20.5	9.5	68.7	185267.1	213027.7	87532.9	217.1	4.4	286.6	160.0	662.0	11442.0	265.4	330.1	3.6	12469.6	13.1	153.0	626.1	1.7	36.2
9	Muscovite 09	783.4	***0.08***	19.7	9.4	860.0	179673.5	213207.3	86993.7	200.6	8.4	523.0	14.2	1052.1	13061.9	483.2	327.7	3.8	10973.6	240.8	166.3	433.0	2.7	59.4
10	Muscovite 10	988.6	***0.10***	19.9	9.6	66.9	180794.8	213875.0	89000.9	204.9	4.0	294.9	2.3	1063.6	14363.8	622.6	283.9	2.6	8887.1	208.3	79.3	281.6	2.0	47.1
11	Muscovite 11	246.2	***0.02***	1.3	12.9	288.0	186867.7	217019.9	93092.6	203.9	19.0	491.3	18.5	536.4	6549.9	61.2	106.2	71.7	2042.8	<LOD	86.0	458.0	5.6	1.0
12	R Muscovite 01	607.2	***0.06***	10.0	12.0	1056.2	181517.7	220325.2	92688.9	228.2	9.6	283.3	22.4	757.1	11554.7	120.6	330.2	6.0	11224.9	44.0	141.4	520.1	44.7	13.7
13	R Muscovite 02	1165.7	***0.12***	15.3	11.1	901.5	178577.8	209883.1	86971.3	261.3	13.2	799.6	40.4	1208.4	15942.2	283.6	378.7	2.9	15221.2	179.7	479.5	1948.7	4.4	113.1
14	R Muscovite 03	1068.6	***0.11***	18.1	10.6	1754.3	177894.2	215512.5	87196.0	267.4	7.2	1328.7	111.1	1067.5	13318.7	432.0	388.9	2.7	17770.2	116.6	170.0	2408.6	11.3	149.9
15	R Muscovite 04	2953.5	***0.30***	15.3	8.4	3643.9	155959.5	211540.9	81975.1	261.0	25.6	3363.8	106.0	860.2	29717.7	195.9	321.7	4.0	20547.7	234.0	720.8	3384.6	80.0	125.2
16	R Muscovite 05	612.6	***0.06***	14.9	11.7	85.3	184420.7	216958.2	90193.5	209.0	3.9	101.7	97.4	642.0	10288.6	202.1	307.4	2.9	12828.7	1.1	110.5	784.4	4.6	2.3
17	R Muscovite 06	1142.6	***0.11***	20.2	11.9	2025.2	180401.4	214801.6	85221.9	279.5	5.5	1290.1	73.5	1013.2	12791.8	517.0	387.0	2.5	17784.1	84.4	132.3	1322.6	19.8	132.7
18	R Muscovite 07	936.9	***0.09***	19.2	10.6	883.9	179789.7	212012.4	91703.6	267.5	7.7	1440.7	112.4	1048.3	13618.8	395.6	367.3	3.4	11907.9	226.2	241.5	601.4	2.1	98.4
19	R Muscovite 08	863.5	***0.09***	18.0	9.8	304.5	183316.9	209226.9	88795.2	307.0	4.9	346.2	112.5	901.3	12498.5	281.1	298.7	2.9	12053.9	110.4	209.8	466.0	2.8	90.1
20	R Muscovite 09	1291.9	***0.13***	18.7	11.6	393.1	177480.2	214484.1	87699.0	288.4	15.8	713.0	61.5	1164.1	15791.9	523.4	335.7	2.4	11727.1	227.1	200.3	506.9	2.4	65.4
21	R Muscovite 10	1784.7	***0.18***	8.2	11.1	423.8	164388.7	217220.6	88687.7	273.6	4.0	2602.6	53.8	3898.4	20419.4	306.9	436.6	4.4	16155.5	59.3	580.4	1480.3	1.4	24.5
22	R Muscovite 11	901.8	***0.09***	23.1	11.2	254.1	183752.5	214867.1	89561.6	301.0	5.7	513.5	0.8	1234.8	12728.7	622.3	332.1	3.0	11026.9	201.4	158.9	390.7	1.6	61.4
23	R Muscovite 12	896.1	***0.09***	18.0	12.6	1141.2	181359.9	216434.3	86890.1	284.5	5.1	1189.6	41.4	787.0	10282.0	335.5	387.9	4.0	17373.9	83.3	143.2	1129.5	2.2	114.7
24	R Muscovite 13	967.7	***0.10***	18.2	11.8	88.8	181051.3	215372.5	93581.3	334.5	3.9	367.9	3.8	1151.7	14041.3	705.4	302.3	2.1	9914.3	244.5	106.5	356.8	2.7	41.6
25	R Muscovite 14	1213.7	***0.12***	22.1	12.9	303.6	183004.2	218009.8	88570.4	252.9	4.3	433.7	98.3	945.0	12580.2	413.0	276.8	2.2	9449.2	57.6	93.7	285.6	5.9	66.5
	Min	246.2	0.025	1.3	8.4	66.9	155959.5	209226.9	81975.1	196.5	3.9	101.7	0.8	536.4	6549.9	61.2	106.2	2.1	2042.8	1.1	79.3	281.6	1.4	1.0
	Max	2953.5	0.295	24.5	12.9	3643.9	186867.7	220325.2	93581.3	334.5	25.6	3363.8	160.0	3898.4	29717.7	814.3	436.6	71.7	20547.7	244.5	720.8	3384.6	132.2	149.9
	Avg.	1022.1	0.102	17.2	10.7	800.3	179277.3	214581.3	88304.8	254.3	7.6	800.7	58.2	1141.3	13689.9	403.2	329.6	6.2	12487.8	146.2	223.4	928.1	18.5	66.0

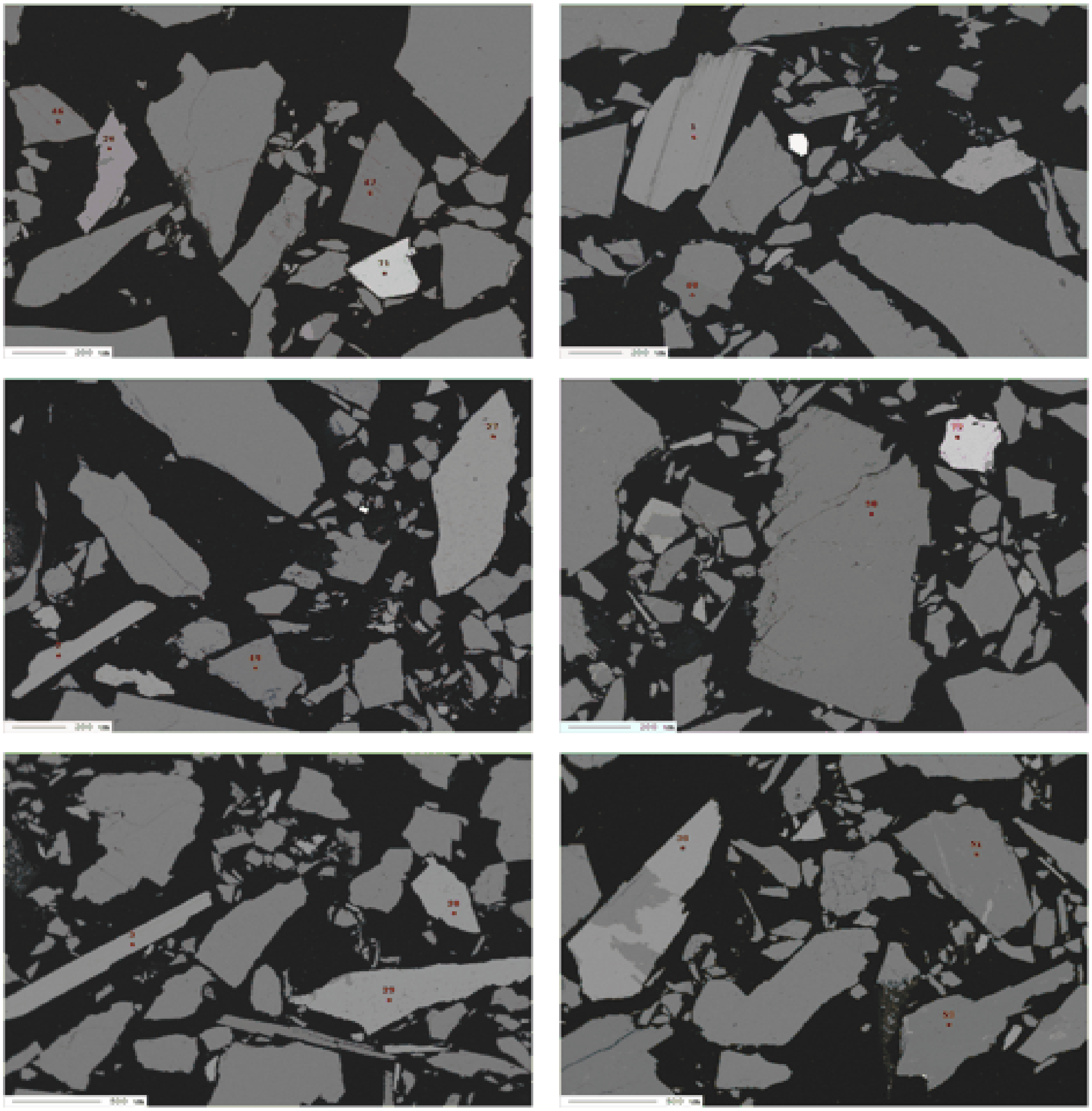
***H2O (calculated)**

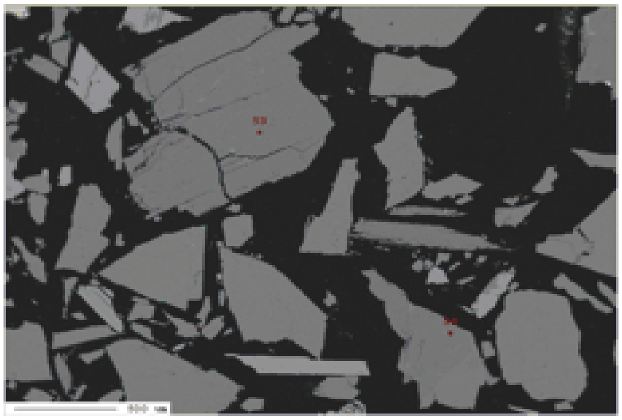
	EPMA											LA-ICP-MS (ppm)												
No.	Oxide weight %	SiO2	Al2O3	Fe2O3	CaO	BaO	Na2O	K2O	Rb2O	Cs2O	Total	Li	B	Al	Si	P	K	Ca	Ga	Rb	Sr	Cs	Ba	Pb
	LOD	0.051738	0.038135	0.031014	0.023117	0.053009	0.020588	0.024615	0.04884	0.039816														
26	Alkali feldspar 01	64.07	18.29	0.00	0.02	0.04	0.46	15.31	1.57	0.06	99.82	33.6	266.8	96034.4	302039.6	81.2	131718.0	362.5	31.0	12716.7	16.3	299.3	25.6	54.4
27	Alkali feldspar 02	64.18	18.24	0.00	0.01	0.00	0.71	15.39	1.09	0.03	99.65	28.5	265.1	96493.0	300291.0	91.9	126492.1	319.9	31.9	9522.5	1.9	297.4	6.0	37.3
28	Alkali feldspar 03	64.53	18.16	0.01	0.02	0.01	0.65	15.15	1.27	0.10	99.91	23.5	258.5	96536.2	300506.2	84.4	130858.3	340.7	33.3	11740.0	6.5	978.6	9.7	45.6
29	Alkali feldspar 04	64.00	18.22	0.01	0.02	0.00	0.60	15.05	1.49	0.05	99.44	24.6	253.5	96956.7	297677.5	87.5	131860.9	377.4	31.2	13927.4	10.0	509.2	28.2	39.3
30	Alkali feldspar 05	64.06	18.18	0.01	0.02	0.00	0.56	15.37	1.28	0.04	99.52	24.5	266.2	97384.8	296053.9	105.6	128946.7	375.1	30.8	11832.8	2.4	322.6	4.0	32.4
31	Alkali feldspar 06	64.08	18.02	0.01	0.01	0.02	0.53	15.33	1.35	0.19	99.53	32.1	253.4	94769.4	301534.6	<LOD	127161.6	377.8	30.6	12579.4	4.9	1640.9	2.1	52.2
32	Alkali feldspar 07	63.85	18.25	0.00	0.02	0.01	0.50	15.21	1.57	0.15	99.55	30.6	248.9	94300.7	305950.1	99.2	131135.2	341.9	30.5	14799.6	8.1	1139.3	19.3	45.2
33	Alkali feldspar 08	64.45	18.36	0.00	0.01	0.03	0.57	15.25	1.35	0.06	100.07	29.2	224.3	96504.2	303531.0	69.0	127220.2	324.2	30.5	12966.8	5.5	629.9	11.8	51.3
34	Alkali feldspar 09	64.11	18.29	0.02	0.01	0.02	0.57	15.30	1.18	0.05	99.54	22.2	222.3	96467.7	300850.0	93.2	136989.4	793.5	30.2	11220.9	11.5	324.3	23.9	30.7
35	Alkali feldspar 10	63.80	18.43	0.02	0.02	0.01	0.62	15.01	1.54	0.04	99.48	45.0	224.5	97752.7	297736.2	96.2	122640.2	374.7	29.3	13429.0	7.2	458.6	22.2	44.9
36	Alkali feldspar 11	63.48	18.05	0.00	0.01	0.01	0.34	15.41	1.50	0.18	98.97	14.6	232.8	95654.8	296510.0	<LOD	133340.4	384.9	27.8	13071.6	3.5	803.0	16.1	22.3
37	Alkali feldspar 12	63.68	18.10	0.02	0.00	0.01	0.44	15.27	1.62	0.06	99.21	23.8	126.8	94383.3	302289.5	<LOD	124937.6	356.5	30.9	13650.9	9.3	297.8	12.7	47.1
38	R Alkali feldspar 01	64.10	18.33	0.00	0.02	0.00	0.53	15.50	1.29	0.03	99.81	45.5	258.5	96782.9	300537.7	104.2	125712.8	415.0	31.1	11535.3	5.6	308.1	21.4	32.6
38	R Alkali feldspar 02	64.12	18.06	0.02	0.01	0.02	0.48	15.21	1.68	0.14	99.73	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
49	R Alkali feldspar 03	63.74	17.92	0.02	0.02	0.00	0.41	15.24	1.96	0.16	99.47	21.3	273.0	93501.9	302475.5	90.5	130270.2	414.3	35.1	16697.7	13.2	1397.4	18.2	55.6
41	R Alkali feldspar 04	64.09	18.07	0.01	0.01	0.02	0.46	15.19	1.67	0.03	99.56	30.4	256.8	94305.4	303998.6	92.7	127142.4	368.9	31.0	14632.7	4.1	349.0	9.9	29.8
42	R Alkali feldspar 05	64.44	18.31	0.01	0.02	0.00	0.59	15.05	1.55	0.04	100.01	50.4	254.0	96456.9	302803.4	78.4	130893.4	415.4	30.7	14584.6	5.9	565.8	18.7	39.6
43	R Alkali feldspar 06	63.99	18.33	0.00	0.00	0.00	0.59	15.13	1.68	0.18	99.90	61.2	257.5	96617.9	300438.3	127.2	130045.1	429.3	28.0	15502.7	6.6	1568.6	19.0	47.5
44	R Alkali feldspar 07	64.10	18.22	0.00	0.02	0.00	0.43	15.38	1.65	0.12	99.93	20.3	269.2	96679.4	298963.4	92.7	129774.6	345.3	30.5	14722.5	7.1	1066.6	14.4	46.8
45	R Alkali feldspar 08	64.43	18.35	0.00	0.01	0.00	0.55	15.41	1.20	0.14	100.09	20.3	245.8	96408.3	303487.7	<LOD	124590.0	343.9	29.1	10930.0	7.2	746.3	19.7	49.7
	Min	63.48	17.92	0.00	0.00	0.00	0.34	15.01	1.09	0.03	98.97	14.6	126.8	93501.9	296053.9	69.0	122640.2	319.9	27.8	9522.5	1.9	297.4	2.1	22.3
	Max	64.53	18.43	0.02	0.02	0.04	0.71	15.50	1.96	0.19	100.09	61.2	273.0	97752.7	305950.1	127.2	136989.4	793.5	35.1	16697.7	16.3	1640.9	28.2	55.6
	Avg.	64.06	18.21	0.01	0.01	0.01	0.53	15.26	1.47	0.09	99.66	30.6	245.1	95999.5	300930.2	92.9	129038.4	392.7	30.7	13161.2	7.2	721.2	16.0	42.3

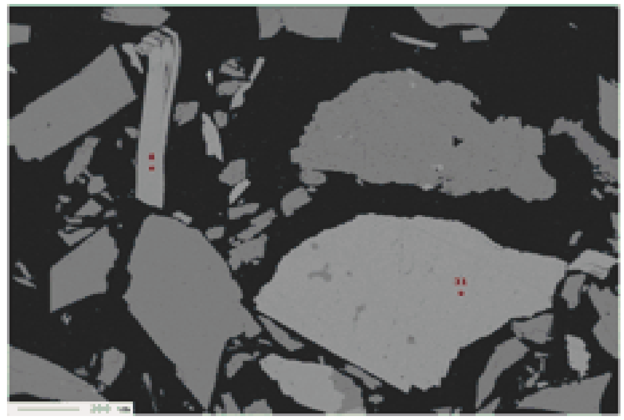
ı: not analyzed

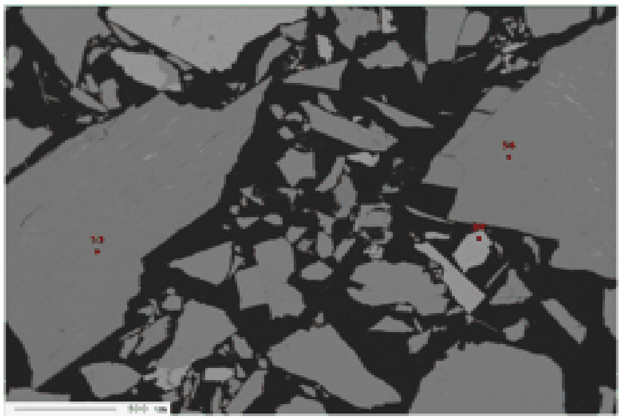
	EPMA											LA-ICP-MS (ppm)																
No.	**Oxide weight %**	**SiO2**	**Al2O3**	**TiO2**	**Y2O3**	**FeO**	**MnO**	**MgO**	**CaO**	**Na2O**	**Total**	**Li**	**B**	**Mg**	**Al**	**Si**	**P**	**Sc**	**Ti**	**V**	**Mn**	**Fe**	**Zn**	**Ga**	**Ge**	**Y**	**Sn**
	LOD	0.041964	0.035912	0.017983	0.034748	0.032934	0.027263	0.015567	0.013761	0.017559																	
71	Garnet 01	35.70	20.88	0.05	0.01	8.22	34.19	0.03	0.67	0.01	99.76	71.1	17.6	289.0	111017.8	204511.7	148.0	18.6	225.9	20.7	263740.3	59914.1	335.9	56.5	56.7	132.6	22.1
72	Garnet 02	35.75	20.63	0.06	0.09	11.91	31.09	0.01	0.32	0.00	99.85	94.8	15.9	65.0	108176.6	210399.5	155.5	7.1	193.6	6.6	243100.4	74697.7	311.8	64.6	22.4	239.8	240.0
73	Garnet 03	35.78	20.83	0.03	0.17	10.39	32.02	0.00	0.27	0.01	99.49	169.2	18.5	114.1	108653.1	208888.3	123.1	18.8	180.0	15.9	251735.9	60250.6	381.8	47.7	64.3	1106.4	32.9
74	Garnet 04	35.47	20.62	0.09	0.29	10.73	31.77	0.01	0.31	0.01	99.28	312.9	19.1	39.6	109369.9	205113.8	225.8	13.0	447.6	16.0	245607.6	72406.1	308.1	69.2	61.8	2165.5	192.0
75	Garnet 05	35.82	20.79	0.04	0.04	9.68	32.54	0.02	0.66	0.01	99.61	68.3	20.6	234.0	106962.3	209265.0	137.7	22.2	219.2	10.8	259602.6	52738.8	348.0	50.2	76.2	113.1	30.3
76	Garnet 06	35.61	20.54	0.03	0.02	11.27	31.55	0.01	0.30	0.00	99.32	72.1	10.5	178.3	118324.8	183667.3	168.0	3.4	179.4	2.7	226074.2	63840.9	67.0	56.7	15.2	90.3	117.2
77	Garnet 07	35.87	20.68	0.06	0.10	8.96	33.57	0.01	0.30	0.00	99.55	192.8	19.1	137.0	106458.1	205638.0	175.8	55.6	309.1	3.0	267610.4	56711.4	356.4	68.0	65.0	860.4	123.2
78	Garnet 08	36.01	20.47	0.05	0.03	10.05	33.08	0.01	0.24	0.01	99.95	55.8	21.0	15.4	109055.2	206424.9	239.3	4.1	113.5	2.5	254680.8	62305.8	348.6	65.0	18.6	22.1	87.8
79	Garnet 09	35.72	20.40	0.08	0.25	11.12	31.40	0.00	0.30	0.01	99.29	305.3	19.0	35.5	109198.5	204534.0	256.9	13.8	430.4	15.2	240589.6	69686.9	313.5	66.1	59.8	2131.7	171.7
80	Garnet 10	35.47	20.61	0.03	0.03	8.93	34.08	0.00	0.26	0.01	99.44	107.4	17.2	13.6	104765.5	197623.2	295.2	4.2	242.1	<LOD	275486.0	33149.8	354.7	53.6	55.4	176.5	112.0
81	Garnet 11	35.72	20.73	0.05	0.02	8.09	34.58	0.01	0.26	0.00	99.45	122.9	19.2	31.4	112788.7	209108.2	305.8	4.5	304.5	3.0	260786.8	63155.0	355.3	63.7	67.5	244.3	82.8
82	Garnet 12	35.87	20.70	0.03	0.10	9.88	32.82	0.02	0.34	0.01	99.77	103.2	21.1	217.6	114262.2	206206.4	173.1	62.4	157.1	6.5	244235.7	64964.5	358.4	41.6	81.3	402.2	17.0
83	R Garnet 01	35.71	20.43	0.09	0.33	10.68	32.01	0.01	0.29	0.01	99.56	174.4	22.2	92.9	109043.5	209376.1	213.6	55.1	318.3	19.7	245963.7	69075.9	312.7	76.2	36.5	832.8	241.0
84	R Garnet 02	36.00	20.83	0.04	0.19	9.56	32.87	0.02	0.34	0.00	99.85	174.7	19.7	103.7	109015.4	214859.2	138.9	33.1	179.2	3.2	257666.4	58303.4	355.5	52.9	69.0	1156.1	36.7
85	R Garnet 03	35.73	20.65	0.06	0.09	10.62	32.21	0.00	0.29	0.00	99.66	77.1	24.8	49.3	109260.6	205405.9	195.3	11.1	157.5	5.0	249603.2	68247.1	309.7	64.0	24.1	207.0	170.3
86	R Garnet 04	36.03	20.57	0.07	0.19	10.45	32.25	0.01	0.29	0.00	99.87	258.7	21.5	28.0	105230.4	210798.4	254.3	10.1	392.7	6.8	258827.0	66070.5	298.2	66.7	52.9	1485.8	186.8
87	R Garnet 05	35.95	20.79	0.04	0.05	5.57	37.26	0.00	0.25	0.01	99.91	95.4	22.5	13.4	110121.5	193818.7	333.6	4.7	190.1	0.5	288447.6	43540.2	380.9	54.7	64.5	184.1	120.4
88	R Garnet 06	35.62	20.53	0.09	0.35	11.23	31.42	0.01	0.31	0.00	99.55	313.0	22.1	39.0	109214.7	207345.1	261.5	14.8	459.9	10.2	242145.7	72446.1	311.3	72.0	61.4	2343.1	206.8
89	R Garnet 07	36.06	21.03	0.05	0.00	8.19	34.35	0.03	0.65	0.00	100.36	88.7	21.8	282.8	113383.1	207329.5	160.9	27.7	227.2	14.9	261301.5	59789.5	351.4	57.1	76.3	253.4	25.6
	Min	35.47	20.40	0.03	0.00	5.57	31.09	0.00	0.24	0.00	99.28	55.8	10.5	13.4	104765.5	183667.3	123.1	3.4	113.5	0.5	226074.2	33149.8	67.0	41.6	15.2	22.1	17.0
	Max	36.06	21.03	0.09	0.35	11.91	37.26	0.03	0.67	0.01	100.36	313.0	24.8	289.0	118324.8	214859.2	333.6	62.4	459.9	20.7	288447.6	74697.7	381.8	76.2	81.3	2343.1	241.0
	Avg.	35.78	20.67	0.05	0.12	9.77	32.90	0.01	0.35	0.01	99.66	150.4	19.7	104.2	109700.1	205279.6	208.5	20.2	259.3	9.1	254589.8	61647.1	324.2	60.4	54.2	744.6	116.7

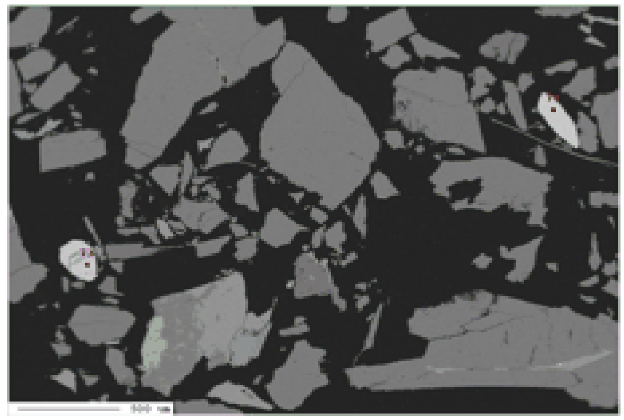
Li-Head

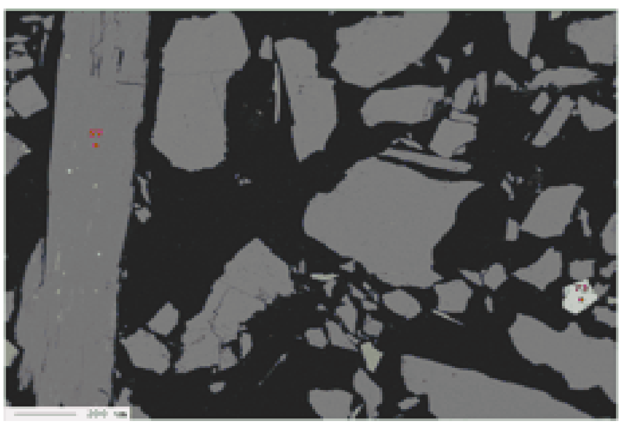


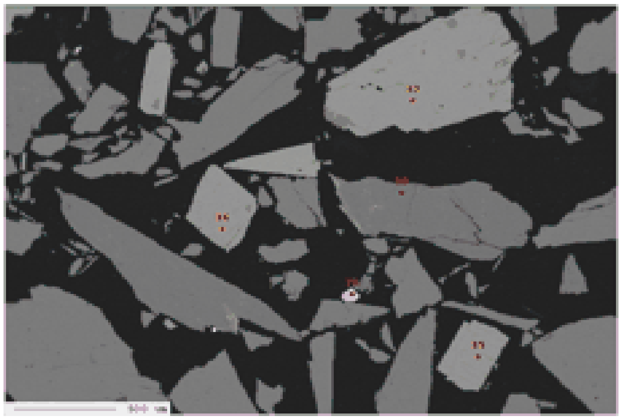


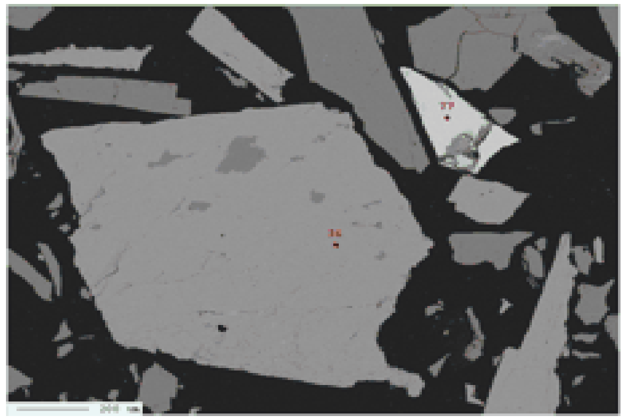


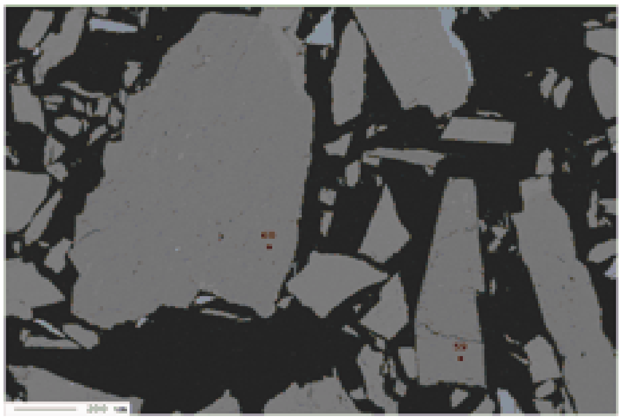


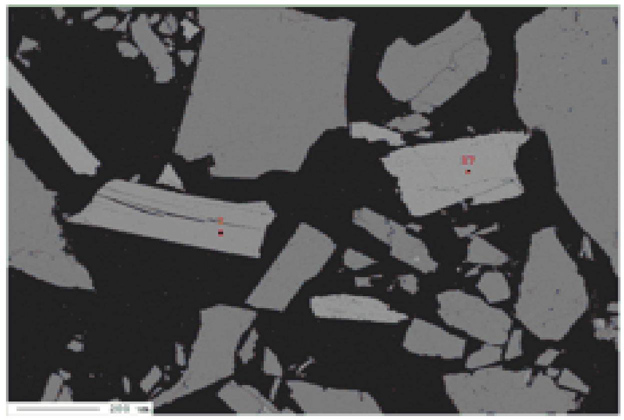


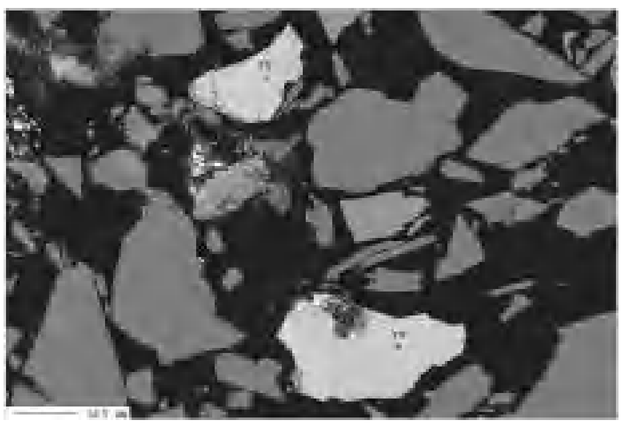


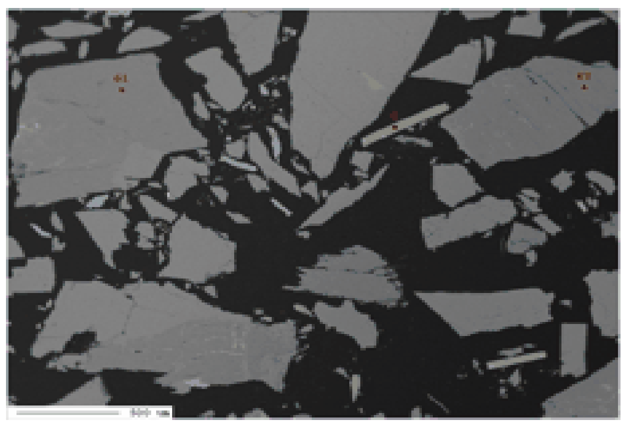


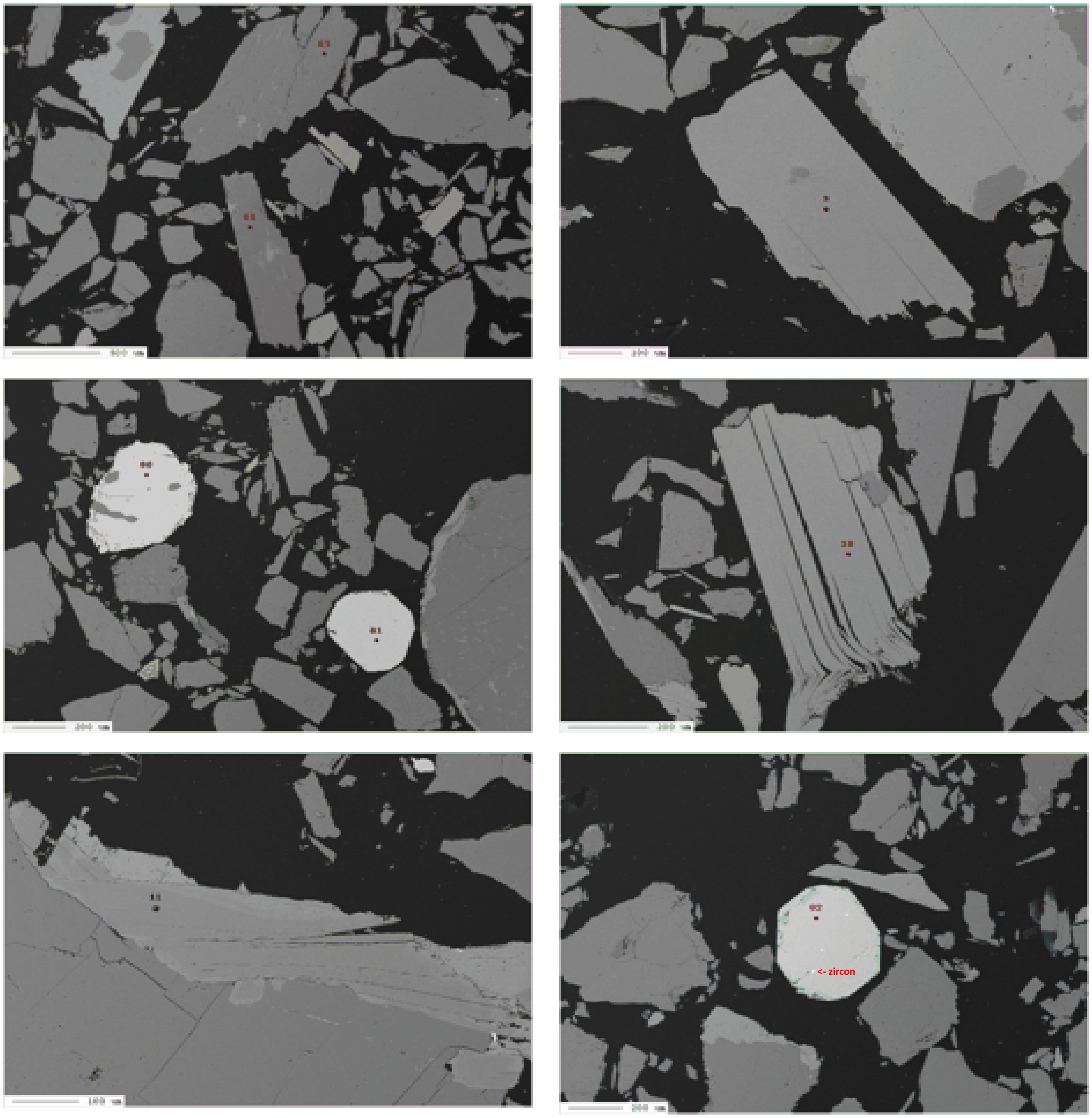








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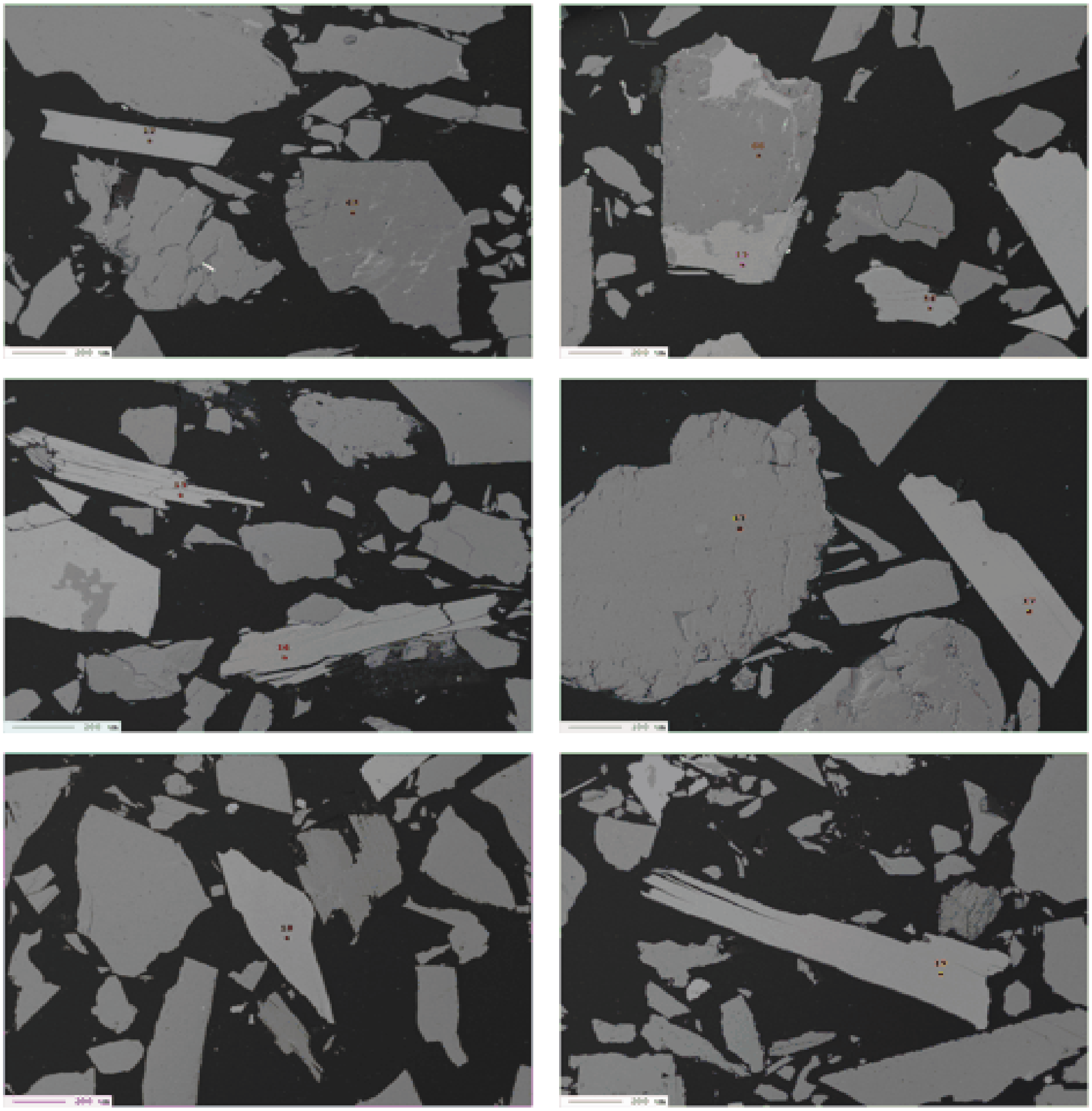
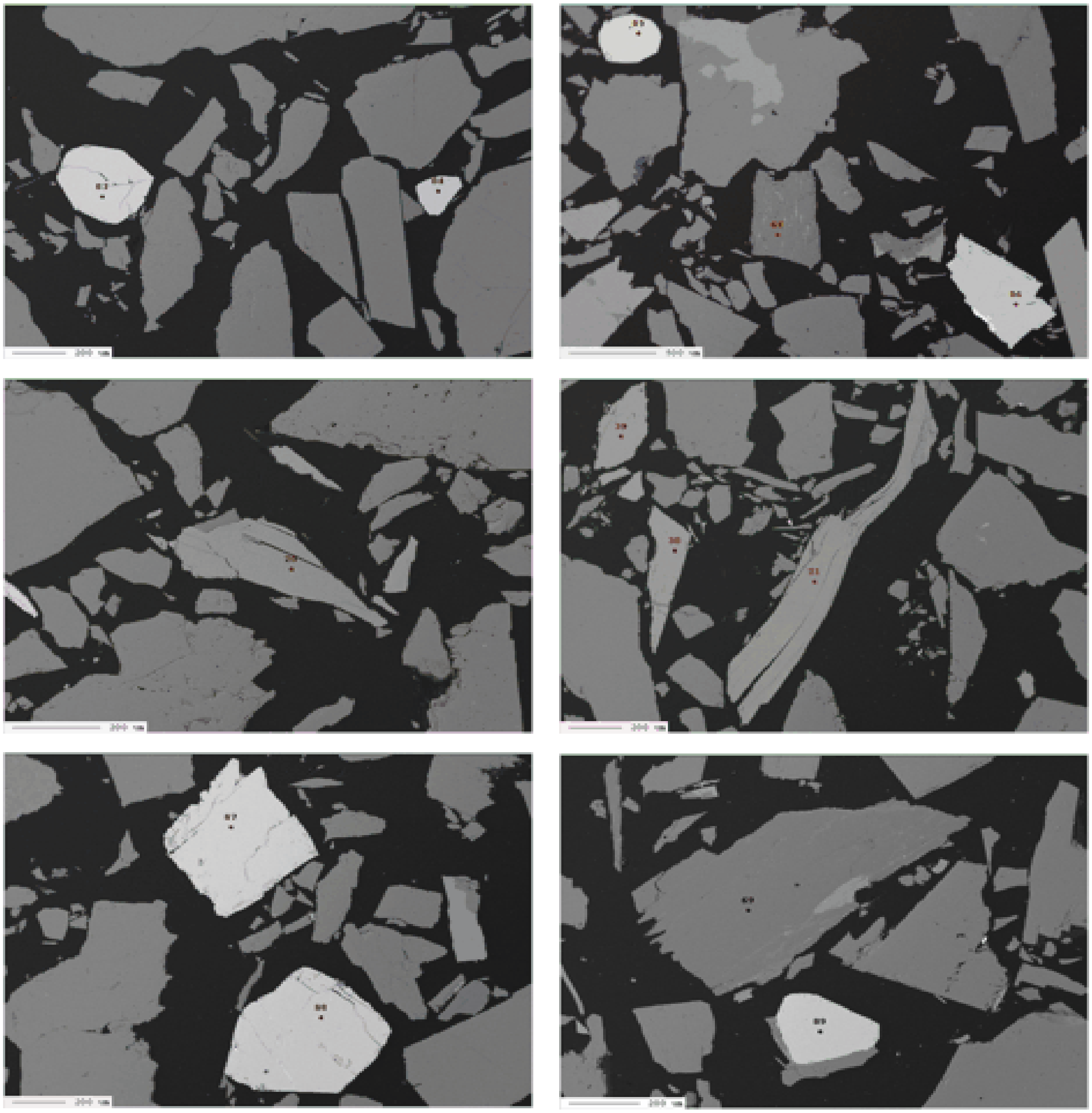
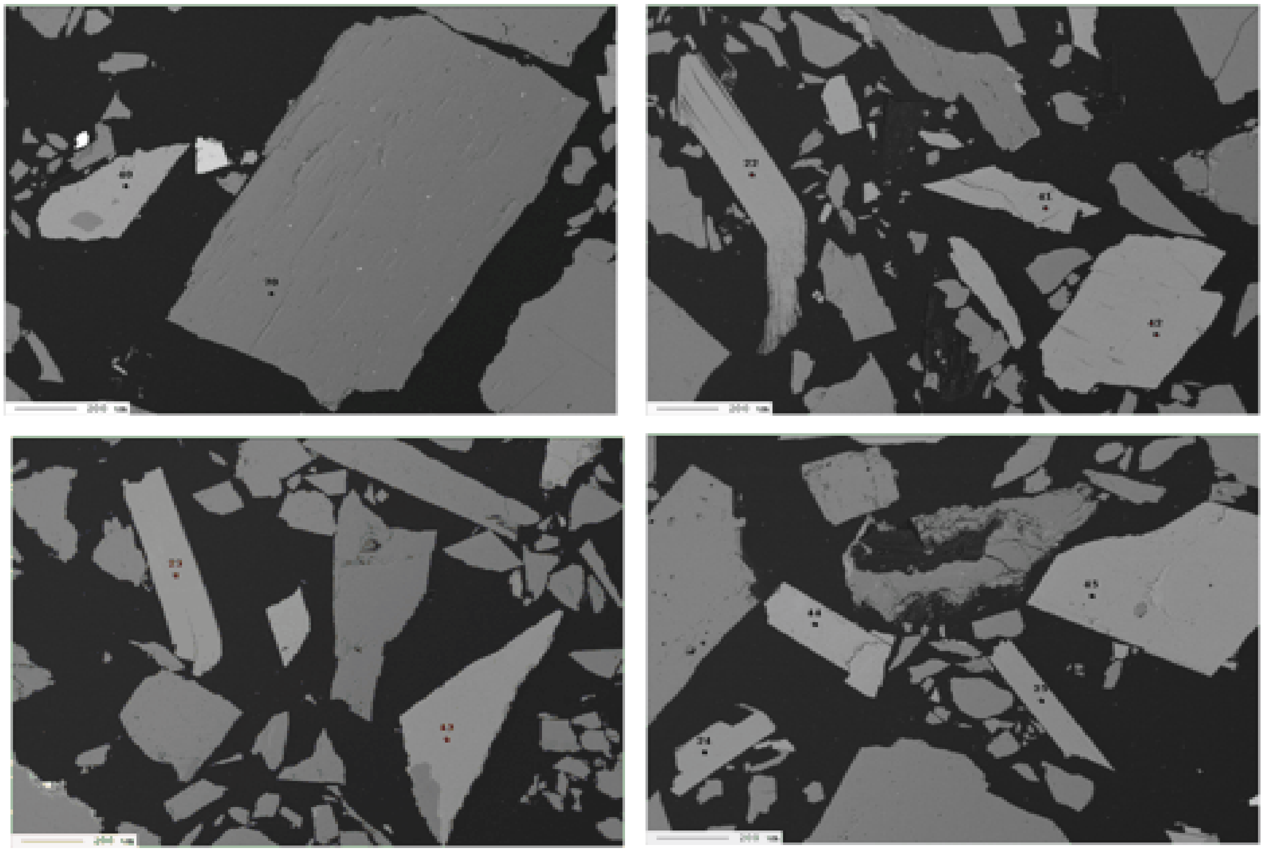
Li-Head R







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