TECHNICAL REPORT ON THE FEASIBILITY STUDY FOR THE SONORA LITHIUM PROJECT, MEXICO

January 2018

Prepared For

Bacanora Minerals Ltd

Prepared by

Ausenco Services Pty Ltd
144 Montague Rd
South Brisbane
Australia

Effective Date:	December 12, 2017
Issue Date:	January 25, 2018

CERTIFICATE OF QUALIFIED PERSON

I, Martin Frank Pittuck, CEng, MIMMM, FGS, do hereby certify that:

1. I am a Corporate Consultant (Mining Geology) of SRK Consulting (UK) Ltd with an office at 5th Floor, Churchill House, Churchill Way, Cardiff CF10 2HH;
2. This certificate applies to the technical report titled "Technical Report on the Feasibility Study for the Sonora Lithium Project, Mexico, January 2018" (the "Technical Report"), prepared for Bacanora Minerals Limited;
3. The Effective Date of the Technical Report is 12 December 2017;
4. I am a graduate with a Master of Science in Mineral Resources gained from Cardiff College, University of Wales in 1996 and I have practised my profession continuously since that time. Since graduating I have worked as a consultant at SRK on a wide range of mineral projects, specializing in precious and rare metals. I have undertaken many geological investigations, resource estimations, mine evaluation technical studies and due diligence reports. I am a member of the Institution of Materials Mining and Metallurgy (Membership Number 49186) and I am a Chartered Engineer;
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I visited the Sonora property between 24 and 27 March, 2015 and 20 and 25 June 2016.
7. I am co-author and reviewer of this report and have responsibility for the Mineral Resource estimate and sections 1.3, 1.4, 1.16.1, 4.1, 4.2, 4.3, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.1, 26.1, and 27 in the Technical Report, unless subsections are specifically identified by another Qualified Person.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report, other than previous independent consulting mandates.
10. I have read NI 43-101 and Form 43-101F1; the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
11. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th January 2018.

Martin Frank Pittuck, CEng, MIMMM, FGS
Corporate Consultant (Mining Geology)

CERTIFICATE OF QUALIFIED PERSON

I, Gregory Searle Lane, F. AusIMM, do hereby certify that:

1. I am Chief Technical Officer for Ausenco Pty Ltd.
2. This certificate applies to the technical report titled "Technical Report on the Feasibility Study for the Sonora Lithium Project, Mexico, January 2018" (the "Technical Report"), prepared for Bacanora Minerals Limited.
3. The Effective Date of the Technical Report is 12 December 2017.
4. I am a graduate of University of Tasmania with Master of Science and Bachelor of Applied Science (Applied Chemistry) degrees. I have worked as a chemist, metallurgist and process engineer continuously for a total of 36 years since my graduation from University.
5. I am a Fellow of the Australian Institute of Mining and Metallurgy (No 203005) and a Registered Professional Engineer of Queensland (RPEQ), RPEQ 06514.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I am co-ordinating author of the Technical Report, and co-author responsible specifically for sections 1.1, 1.7, 1.8, 1.9, 1.10, 1.12, 1.13, 1.14, 1.15, 1.16.3, 1.16.4, 2, 4.5, 11.6, 13, 17, 18, 19, 21, 22, 24, 25.3, 25.4, 26.3, 26.4, and 27, unless subsections are specifically identified by another Qualified Person.
8. I have not visited the property, but have visited the facilities where the bulk of the test work was completed.
9. I am independent of Bacanora applying all of the tests in section 1.5 of NI 43-101.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and Form 43-101F1; the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of January, 2018.

Gregory S. Lane, FAusIMM (No. 203005)

CERTIFICATE OF QUALIFIED PERSON

I, Herbert E. Welhener do hereby certify that:

1. I am a Vice President of Independent Mining Consultants, Inc. located at 3560 E. Gas Road, Tucson, Arizona, USA;
2. This certificate applies to the technical report titled "Technical Report on the Feasibility Study for the Sonora Lithium Project, Mexico, January 2018" (the "Technical Report"), prepared for Bacanora Minerals Limited;
3. The Effective Date of the Technical Report is 12 December 2017;
4. I am a graduate with a Bachelor of Science in Geology from the University of Arizona in 1973 and I have practiced my profession continuously since that time. Since graduating I have worked as a consultant on a wide range of mineral projects, specializing in precious, base and industrial metals. I have undertaken many mineral resource estimations, mine evaluation technical studies and due diligence reports in a variety of settings around the world. I am a registered member of the Society of Mining, Metallurgy and Exploration, Inc. (SME RM # 3434330).
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I last visited the Sonora Lithium property on the 28 July 2017.
7. I am co-author and reviewer of this report and have specific responsibility for the Mineral Reserve estimate and Sections 1.5, 1.6, 1.16.2, 15, 16, 25.2, 26.2, 27, and input for 18.11, 21.1 and 21.2 in the Technical Report, unless subsections are specifically identified by another Qualified Person.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have had prior involvement with the property that is the subject of the Technical Report by working on the Pre-Feasibility Study dated April 2016.
10. I have read NI 43-101 and Form 43-101F1; the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
11. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of January, 2018.

Herbert E. Welhener

Herbert E. Welhener, SME RM

SME
Society for Mining, Metallurgy & Exploration
Herbert E. Welhener
SME Registered Member No. 3434330
Signature Herbert E. Welhener
Date Signed 1-25-2018
Expiration date 12-31-2018

CERTIFICATE OF QUALIFIED PERSON

I, Joel A. Carrasco, P.E., do hereby certify that:

1. I am a Principal Engineer, Solum Consulting Group, 350 S Jackson st. #454 Denver, Colorado 80209 USA.
2. This certificate applies to the technical report titled "Technical Report on the Feasibility Study for the Sonora Lithium Project, Mexico, January 2018" (the "Technical Report"), prepared for Bacanora Minerals Limited;
3. The Effective Date of the Technical Report is 12 December 2017.
4. I am a graduate of Texas Tech University, Texas with a Bachelor of Science degree in Civil Engineering. I have worked as a Civil Engineer continuously for a total of 15 years since my graduation from University. My relevant experience has been working as Project Manager for multi-national engineering companies on feasibility studies and engineering designs of tailings facilities;
5. I am registered as a Professional Engineer in the State of Arizona (Licence # 52000).
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I am co-author and reviewer of this report and have specific responsibility for sections 1.11, 1.16.5, 4.4, 4.6, 4.7, 4.8, 5, 18.10, 18.14, 20, 25.5, 26.5, and 27, unless subsections are specifically identified by another Qualified Person.
8. I visited the Sonora property on the 19th August, 2015.
9. I am independent of Bacanora applying all of the tests in section 1.5 of NI 43-101.
10. I have had prior involvement with the property that is the subject of the Technical Report. I was a QP for the NI 43-101 PFS Technical Report.
11. I have read NI 43-101 and Form 43-101F1; the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of January, 2018.

Registered Professional Engineer (Civil)
CERTIFICATE NO.
52000
JOEL A.
CARRASCO
Date Signed 25/01/18
ARIZONA U.S.A.
Expires: 31/03/20

Joel A Carrasco, P.E.

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1 EXECUTIVE SUMMARY

1.1 Introduction

The Sonora Lithium Project (the "Project") is located in north-west Mexico, in the state of Sonora. The Project is located 170 km south of the USA – Mexico border and three hours drive north east of the state capital of Hermosillo, a city of approximately 700,000 people.

Access to the site is by road from either Hermosillo or the US border town of Agua Prieta.

The proposed Project consists of an open-pit mine and lithium carbonate processing facility, with a mine plan for 19 years. The yearly minimum design output for the project will commence at 17,500 tonnes per year ("t/y") of battery-grade Li_2CO_3 (Stage 1), for the first four years of the project, followed by a proposed expansion, by duplicating the plant, to produce a target minimum design output of 35,000 t/y (Stage 2). In addition, the Sonora Lithium Project has been designed to produce up to 28,800 t/y of potassium sulfate ("K_2SO_4"), for sale to the fertiliser industry.

A Technical Report, based on a Feasibility Study ("FS"), has been prepared for the Project in accordance with the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources and Mineral Reserves (May 2014) National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101").

Ausenco Pty Ltd ("Ausenco"), SRK Consulting (UK) Limited ("SRK"), Independent Mining Consultants Inc. ("IMC") and Solum Consulting Group ("Solum") were commissioned by Bacanora Minerals Limited ("Bacanora" or the "Company") to produce the FS of the Project.

Ausenco is the co-ordinating author of this NI 43-101 Technical Report.

1.2 Accessibility, Local Resources, Infrastructure and Physiography

The Project consists of seven exploration and mining concessions (the "concessions"). Within these concessions the 'La Ventana' portion of the Project is owned 99.9% by Bacanora. The other concessions are held in joint venture with Cadence Minerals PLC ("Cadence"), comprising 70% ownership by Bacanora and 30% by Cadence.

The Project is situated within the Sonoran Desert in the western portion of the Sierra Madre Occidental physiographic province, within the Basin and Range sub province. It lies between "Mesa de Enmedio", "Rincon del Sauz" and "El Capulin" mountain ranges. Average elevation at the Project area is 900 m above mean sea level ("amsl"). The concessions are surrounded by mountain peaks with elevations ranging up to 1,440 m amsl.

The Project area specifically is accessed by way of Federal Highway 14, a two-lane highway extending 225 km east of Hermosillo, to the intersection known as "El Coyote", then south from the intersection for 20 km on a recently paved, two-lane highway to the town of Bacadéhuachi. Bacanora has set up its local base of operations in this town and undertakes all drill core processing facilities from this location.

Access to the concessions from Bacadéhuachi is on secondary, dry-weather roads, crossing various privately owned ranches for approximately 11 km. Land owners have provided authorisation for the Company to access the concessions on these roads.

1.3 Geological Setting and Mineralisation

The geology on the property is dominated by the Oligocene and Miocene Sierra Madre Oriental volcanic complex comprising Miocene sediments and volcanics deposited in half graben basins. The mineralisation studied in this report is contained in a stratiform package dominated by pyroclastics including two distinct clay-rich tuffaceous layers. Some of the clay minerals in these units such as polylithionite are a potentially economic source of lithium. The clay units are separated by an ignimbrite layer and the upper clay layer is overlain by Miocene basalt flows.

The area has mountainous relief with deeply incised valleys where the clay units outcrop in some places; the outcrop geometry is affected by the topography and several faults which offset the deposit. A three dimensional model of the deposit and faults has been created based on outcrop mapping, aerial photography and drilling.

1.4 Mineral Resource Estimation

The majority of exploration on the Project has been completed under Bacanora's management since 2010. Following an early sampling and mapping phase, drilling initially took place on the La Ventana licence area and more recently on the El Sauz and Fleur licence areas. Infill drilling in 2016 focussed on upgrading the Mineral Resource categories in the La Ventana and Fleur licence areas. Approximately 18,000 m of core drilling and 141 m of trenching has been completed to date.

Refer to Table 1.4.1 for the Mineral Resource statement, prepared by SRK with an effective date of 13 December 2017. The Mineral Resource estimate is based on exploration results from mapping drilling and trenching made available to SRK on the 05 September 2016. The Mineral Resource statement is inclusive of the Mineral Reserve.

The Mineral Resource statement is the total for the Project. Of this total, 93%, 75% and 85% of metal in the Measured, Indicated and Inferred Mineral Resource categories, respectively, is attributable to Bacanora.

Table 1.4.1: SRK Mineral Resource Statement as of 13 December 2017

Classification	Tonnes (Mt)	Grade		Contained Metal		
		Li (ppm)	K (%)	kt Li	kt LCE	kt K
Measured	103	3,480	1.5	359	1,910	1,532
Indicated	188	3,120	1.3	588	3,130	2,460
Meas + Ind	**291**	**3,250**	**1.4**	**947**	**5,038**	**3,993**
Inferred	268	2,650	1.2	710	3,779	3,101

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.
2. Mineral Resources are reported inclusive of Mineral Reserves.
3. The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
4. Mineral Resources are reported on 100 percent basis for all Project areas.
5. SRK assumes the Sonora Lithium deposit to be amenable to surface mining methods. Using results from initial metallurgical test work, suitable surface mining and processing costs, and forecast LCE price SRK has reported the Mineral Resource contained within an optimistic open pit shell and above a cutoff grade of 1,000 ppm Li.
6. SRK completed a site inspection of the deposit by Mr. Martin Pittuck, CEng, MIMMM, FGS, an appropriate "independent qualified person" as such term is defined in NI 43-101.
7. LCE is the industry standard terminology for, and is equivalent to, Li_2CO_3. 1 ppm Li metal is equivalent to 5.323 ppm LCE / Li_2CO_3. Use of LCE is to provide data comparable with industry reports and assumes complete conversion of lithium in clays with no recovery or process losses.
8. Mt = million tonnes (metric).
9. kt = thousand tonnes (metric).

1.5 Mineral Reserve Estimation

Refer to Table 1.5.1 for the Mineral Reserve estimate which was prepared by IMC based on an open-pit operation using surface miners for ore mining and conventional truck/shovel mining methods for the waste removal. The Mineral Reserve estimate used a cutoff grade of 1,500 ppm Li, ore recovery factor of 100% and a mining dilution of 100 cm at the contacts with adjacent lithologies using the Li grade of the adjacent lithology.

Table 1.5.1: Open Pit Mineral Reserve Statement as of 13 December 2017

Category	Ore > = 1,500 ppm Li				Waste kt	Total kt	Waste : Ore Ratio	% LCE to Bacanora
	kt	Li ppm	LCE kt	K (%)				
Proven	80,146	3,905	1,666	1.64				93.03%
Probable	163,662	3,271	2,849	1.36				74.63%
Total	**243,808**	**3,480**	**4,515**	**1.45**	**2,298,844**	**2,542,652**	**9.43**	**81.42%**

Notes:
1. kt = thousand tonnes (metric)
2. The mining royalty by the Mexican government of 7.5% was not included in the economics for the pit definition algorithm used for the mineral reserve pit design (a 3% royalty was used instead, based on the information available at the time). As a check, a subsequent pit definition run was made which included the 7.5% royalty and a one half of one percent difference was noted in the two pit shells. The final mineral reserve pit design has less tonnage above cutoff than either of the pit shells used to guide the final pit design.

1.6 Mining Methods

Mining operations will be carried out with front end loaders and haul trucks for waste mining and an ancillary fleet of dozers, graders and water trucks. Ore mining will be done with surface miners which have better selectivity for mining the lithium clays up to the contact with adjacent lithologies.

The open pit phase designs are based on 10 m mining benches (sub-divided into five 2 m benches for scheduling), 20 m wide haul roads (includes allowance for berms and ditches) and 42° inter-ramp slope angle on the hanging wall (east) side of the pits. The lithium clay beds dip to the east and there are no haul ramps on the final east wall of the reserve pit so the inter-ramp slope angle and overall slope angle are the same at 42° for the final pit based on geotechnical investigations. The internal mining phases used for the 19 year production schedule have ramps on the east wall to facilitate waste stripping when mining extends beyond the 19 year mine plan.

The mine plan covers the first 19 years of production and there are additional mineral resources and reserves to extend mining and processing beyond 19 years. For the 19 year mine schedule, a total of 37.1 Mt of ore at a diluted grade of 4,151 Li ppm and 1.76% K and a stripping ratio of 3.4:1 will be mined. The Li cutoff grade for material sent to the plant is 1,500 ppm during years 1 to 18 and 2,000 ppm during year 19.

1.7 Metallurgical Testwork

The FS testwork program builds on the Pre-Feasibility Study program completed by SGS Lakefield in 2016. The purposes of the feasibility testwork program were:

- to define the process flow sheet to produce high quality battery-grade lithium carbonate
- provide engineering data for major equipment selection and sizing.

The FS testwork included flow sheet development testwork using Trench 4 material performed at SGS Malaga. This material was selected as it is sourced from an area that will be mined in the first few years of operation and has an elemental composition similar to what is expected to present to the plant in this time period. During the development of the FS there were significant changes to the flowsheet based on testwork findings resulting in improvements to project economics, including:

- Beneficiation Plant – ore comminution changed from scrubbing to SAG milling based on the higher power efficiency of SAG milling in scrubbing and coarse rejection at a coarser size to give higher lithium recovery
- Extraction Plant - roasting including recycled sodium sulfate to reduce use and cost of gypsum and lower roasting temperature.

Variability test work was done to verify robustness of the flowsheet for both the beneficiation and extraction sections. Locked cycle testwork was also completed to test flow sheet stability.

Key testwork outcomes are summarized below:

- Initial beneficiation circuit definition identified the requirement for a grinding circuit. Scrubbing was not sufficient to obtain the desired lithium upgrade.
- The overall lithium recovery of 78.0% for Stage 1 and 74.2% for Stage 2 is based on 94.2% and 89.5% respective lithium recovery in beneficiation and 82.8% and 83.0% recovery in extraction. Overall potassium recovery is 36.6% for Stage 1 and 33.4% for Stage 2.

- Beneficiation testwork on 14 composites using the selected flowsheet indicated overall lithium recovery ranged from 77.9% to 91.2%, and averaged 83.8%, for 10 of the 14 composites.
- Extraction testwork confirmed a robust roasting recipe consistently achieving >90% lithium extraction in the leach. Impurity removal successfully reduced contaminants in the pregnant leach solution (PLS).
- Extraction testwork on three production composites successfully followed the process flowsheet. Impurities were removed and battery grade lithium carbonate was produced for each of the three composites.
- Locked cycle testwork proved stability in the flow sheet and ability to produce battery grade lithium carbonate, by-products and remove key impurities.

The design criteria which were used to develop the mass balance are based on the testwork results from SGS and ANSTO.

1.8 Recovery Methods

Process engineering and design for the process plants and infrastructure was completed by Ausenco based on the SGS and ANSTO testwork.

The construction of the Sonora Lithium Plant will be in two stages. Stage 1 is designed to process 1.10 Mt/y of ROM feed, at 0.46% Li, to produce a design minimum 17,500 t/y battery grade Li_2CO_3 and 17,000 t/y K_2SO_4. The potassium sulfate produced is expected to be sold as a Sulfate of Potash fertiliser. About 42,000 t/y of Na_2SO_4 is produced in Stage 1. This is not expected to be saleable and is therefore gifted or stored in a lined tailings storage facility.

Stage 2 involves adding a duplicate 1.10 Mt/y train, to be constructed for production in Year 5, to treat a combined total of 2.21 Mt/y of ROM feed, at 0.41% Li, to produce a design minimum 35,000 t/y Li_2CO_3, 28,800 t/y K_2SO_4 and 73,000 t/y Na_2SO_4.

For clarity, Ausenco's SysCAD modelling for Stage 1 produces 21,113 t/y of battery grade Li_2CO_3 and 17,808 t/y K_2SO_4. Similarly, modelling for Stage 2 produces 35,918 t/y of battery grade Li_2CO_3 and 28,805 t/y K_2SO_4. These models were used as the basis of the steady state operating cost calculation.

Whilst the Stage 1 plant is largely duplicated for Stage 2 the modelled Li_2CO_3 and K_2SO_4 production for Stage 1does not simply double for Stage 2 due to a change in feed grade in Stage 2 when compared to the feed grade in Stage 1.

The operating schedule for the plant is a continuous 24 h/d operation, using two 12 h shifts per day, 365 d/y. Design plant availabilities are typical at 90% (7,884 h/y) for the beneficiation plant and 83% (7,270 h/y) for the extraction plant.

A summary of the selected flowsheet is:

- Beneficiation to recover lithium into a fine ground stream while rejecting coarse gangue using grinding, screening and hydrocyclone classification.
- Sodium sulfate roasting, which converts the lithium to water soluble lithium sulfate, in the presence of gypsum, sodium sulfate and limestone.
- A hydrometallurgical section where the roast product is repulped in water to form an impure lithium sulfate PLS. Impurities are then removed from the PLS using precipitation and ion exchange prior to the evaporation and precipitation of battery grade lithium carbonate.

- Potassium sulfate is recovered from the barren liquor using crystallisation and selective dissolution. The filtrate is returned to the sodium sulfate circuit.
- Sodium sulfate is produced from the PLS via crystallization and stockpiled for either reclaim for reuse in the roasting circuit or disposal or gifting.

1.9 Project Infrastructure

Infrastructure proposed for the Project includes:

- Site Access Road: 14.7 km, gravel road with six (6) concrete floodway crossings and 29 culvert crossings.
- Accommodation: modular, 'camp style' accommodation is proposed in close proximity to the project site to facilitate construction and operations for Stage 1. For Stage 2 it is assumed that the Stage 1 camp will be refurbished and that infrastructure and facilities at the local town of Bacadéhuachi will have developed privately to provide accommodation for the majority of local operations staff, ensuring the stage 1 camp is available primarily for Stage 2 construction personnel. During stage 2 operations it is assumed that local operating personnel will live in the local town and only expatriates and key staff will make use of the on-site accommodation. Existing lease boundaries need extending to include the proposed camp location
- Power Supply: A combined heat and power (CHP) natural gas turbine will generate 33 kV power on site. The power station will be operated by a build-own-operate (BOO) partner. The total connected load is estimated at approximately 25 MW for Stage 1 and approximately 50 MW for Stage 2. These loads should be rechecked prior to finalisation of build-own-operate contracts for the gas pipeline and gas turbine power station as there is currently believed to be limited capacity in the proposals received by Bacanora during the feasibility study to accommodate any potential increase in the estimated electrical loads.
- Power Distribution & Electrical: via 60 MVA 33 kV/13.8 kV transformers, supplied and installed by Bacanora Minerals Ltd. Power distribution around site is at 13.8 kV. The generator used to initially supply power to the camp is retained and used for backup emergency (diesel) power generation to the camp power network.

  It is intended that where equipment of similar requirements is to be procured, for the site and camp, that makes and models be standardised where possible. The following equipment has been identified as having similar requirements across multiple scopes/areas for the supply of electrical equipment:

  - 480 Vac switchboards (Process Plant)
  - 120 Vac distribution boards
  - 13.8 kV / 0.480 kV transformers (Process Plant)
  - 480 V / 208 Vac 75 kVA outdoor control distribution transformers, which are used to supply 120 Vac power
  - 120 V / 208 Vac UPS equipment
  - PLC equipment and SCADA systems.
- Mine Infrastructure: hardstand, tyre change pad, vehicle washdown bay, diesel fuel storage and distribution, explosives magazine and mine workshop.
- Water Supply: two water wells, located 7 km north of the plant site, will supply raw water to the process plant for Stage 1 and accommodation facility. A third well will be required for Stage 2. The third well will either be drilled in the same area as the Stage 1 wells or at a site closer to Bacadéhuachi, approximately 12 km from the project site, as measured along the proposed access road. For the FS capital cost assumptions it has been assumed the well is located within the same area as the Stage 1 wells.

- Buildings: administration building, process plant office, process plant workshop, warehouse, laboratory and gatehouse.
- Mobile equipment: includes light vehicles, front end loader, crane, forklifts, ambulance, fire truck and mine rescue vehicle.
- Tailings storage facility (filtered residue waste facility): A total of 0.69 Mt/y of residue waste are estimated to be produced in Stage 1 (Y0-Y4) and 1.6 Mt/y in Stage 2 (Y5-Y19) for a total of 25.3 Mt of filtered residue waste. The residue waste facility is to be located just adjacent to the plant and upstream of the final open pit limits. During the first years of operation, the residue waste disposal facility will be independent of the waste rock placement in a side valley northwest of the plant and contained by a waste rock buttress. The total residue waste placed independent of the waste rock will be 1.38 Mt. Based on the production rate of the waste rock and residue waste, the waste rock buttress and filtered residue waste facility will meet between Y02 and Y03 and the waste rock dump will provide the buttressing. Starting in Y03, the residue waste and waste rock will be placed as a co-disposal operation through the LOM.

1.10 Marketing Studies

Market information has been provided by Bacanora and SignumBox, a Chilean based natural resources research and consulting company with a specific focus on the lithium industry.

The lithium market (as expressed in terms of volume of LCE) is currently growing in excess of 15% p.a. and in value terms has more than doubled since 2014/15 to an estimated $2.2 – 2.5 billion.

SignumBOX has performed a bottom up demand forecast for lithium in which they have estimated the use of lithium in each of the applications in which it is used. They have estimated three different demand scenarios broadly varying based on different potential outcomes for general economic growth and, most importantly, the development of the electric vehicle (“EV”) market which is anticipated to be the primary driver of battery demand for lithium.

SignumBOX forecast that annual growth over the next 20 years will average 11.6% in their Base Case scenario. The bulk of the growth is anticipated to occur due to increasing demand from the battery sector implying continued strong growth for battery grade lithium carbonate and lithium hydroxide.

The Low Scenario still assumes robust growth in demand from battery applications but at less than half the current rate of growth over the 20 year forecast period. It also assumes a weak global economic outlook over the period and a concomitantly lower level of demand growth for non-battery demand. This scenario still sees a tripling of demand in terms of LCE over the next 20 years.

The High Scenario assumes a robust general economic development coupled with an extremely rapid adoption of EVs and a concomitant impact on lithium demand from this sector.

Prices have increased dramatically since the third quarter of 2015 from a global average price of lithium carbonate in the $6,000 per tonne range to over $12,000 per tonne in Q3 2017.

Another important element of lithium pricing is the pricing differential for different grades (i.e. technical grade, battery grade; battery grade commanding the premium price). In recent history this premium has been between $1,200 and $3,500 per tonne of LCE. Given that demand will be driven by the battery sector, SignumBOX anticipate that the premium will increase to over $5,000 per tonne over the next 20 years. The Sonora Project will produce

Battery Grade lithium carbonate. For the purposes of the FS, Bacanora has chosen to apply a price of $11,000 which represents a material discount to the spot prices in Q4, 2017 of $12,000 to $20,000 per tonne.

For sulfate of potash (SOP), Bacanora has commissioned a market study on the SOP market from Green Markets (Bloomberg). Green Markets forecast North American SOP pricing of $550 per tonne for the next ten years.

1.11 Environmental Studies

Environmental and social baseline studies, carried out by Solum, include protected natural areas, flora, fauna, surface water, ground water and social-economic activities. In additional numerous samples of mine rock (ore and waste), and residual waste have been studied for any potential of acid generation. No significant environmental or social issues have been identified.

The baseline collection studies follow guidelines and plans established by the authorities in Mexico, "International Lending Institution Standards" and International Council on Mining and Metals to satisfy potential financing interests and requirements for the project.

Social engagement with the communities has commenced as part of the baseline studies. In general the sorrunding comminties are in favor of the project and no major risks have been identified for the project. A social engagement plan is in progress for the construction and operation phase of the project.

Surface and groundwater sampling sites were established by Solum to characterize the pre-mining conditions at the project. Monitoring wells have been installed at several locations and are currently sampled regulary. Sampling of the groundwater sites consists of the collection of levels and chemical analysis. Sampling of surface water sites involve surface flow measurements and levels through installed stage gauging stations.

The Manifestacion de Impacto Ambiental ("MIA") was prepared and submitted to SEMARNAT for approval on May 2017 ("Q2 2017"). The approval process usually takes 12-18 months but can be achieved in 6 months with properly completed documentation. The company received the approved resolution of the MIA on October 11, 2017.

The Estudio Tecnico Justificativo ("ETJ") for land use change of the Ventana construction site was prepared and submitted on January 10, 2018 ("Q1 2018). The approval process usually takes 60 days.

1.12 Capital Cost Estimate

The capital cost estimate covers the design and construction of the mine and process plant, together with on-site and off-site infrastructure to support the operation, including water and power distribution and support services. The capital costs associated with the gas supply pipeline and power station are included as part of the operating cost estimate, not the capital cost estimate, as these items are intended to be sourced on a build-own-operate ("BOO") basis.

Refer to Table 1.12.1 for a summary of the Stage 1 and Stage 2 capital cost estimates which have an accuracy of ±15% and a base date of Q4 2017. All amounts expressed are in US dollars unless otherwise indicated.

Table 1.12.1: Estimated Capital Cost - Summary for the Two Stages

Area	Stage 1 $M	Stage 2 $M
Mining Equipment	14.2	17.6
Mining Infrastructure	3.4	0
Beneficiation Plant	18.5	18.5
Lithium Processing (Extraction) Plant	158	158
Common Plant Services	55.3	55.3
On-Site Infrastructure	37.8	20.5
Off-Site Infrastructure	21.0	3.1
EPCM/Owner's Costs/Indirects	72.9	72.4
Contingency	38.1	34.6
Total	**420**	**380**

1.13 Operating Cost Estimate

The mining and processing operating costs were calculated for an operation achieving average annual production of approximately 21,113 t/y of battery-grade (99.5%) Li_2CO_3 in Stage 1 and 35,918 t/y in Stage 2, as per Ausenco's SyCAD model (steady state). The operating cost estimate covers the mine, process plant and general and administration facilities. These SysCAD model operating costs estimates, at an accuracy of ±15%, are summarized in Table 1.13.1.

The financial model included ramp up for Stage 1 and Stage 2 plants. This results in lower recoveries and thus, lower Li2CO3 production. The resulting increase in operating cost per tonne of Li2CO3 production is included presented in Table 1.13.2.

Table 1.13.1: Operating Cost Estimate – SysCAD

Category	$/t Li_2CO_3		
	Stage 1	Stage 2	LOM
Mining	295	499	471
Processing	3,093	3,266	3,243
General and Administration	263	209	216
Total	**3,651**	**3,974**	**3,930**

Table 1.13.2: Operating Cost Estimate – Financial Model

Category	$/t Li_2CO_3		
	Stage 1	Stage 2	LOM
Mining	325	511	490
Processing	3,418	3,169	3,198
General and Administration	296	212	222
Total	**4,039**	**3,893**	**3,910**

1.14 Financial Analysis

As shown in Table 1.14.1 the FS demonstrates the financial viability of the Sonora Lithium Project at an initial minimum design production rate of 17,500 t/y of battery-grade Li_2CO_3 in Stage 1 and expansion to 35,000 t/y in Year 5 (Stage 2).

The project is currently estimated to have a payback period for Stage 1 of four years. Cash flows are based on a 100% equity funding basis and show the average gross annual revenue is $363M over the 19 years of operations. The economic analysis indicates a pre-tax NPV, discounted at 8%, of approximately $1,253M and an Internal Rate of Return ("IRR") of approximately 26%. Post tax the NPV is approximately $802M and IRR 21%.

A sensitivity analysis has shown the Project is more sensitive to the lithium price than it is to either CAPEX or OPEX. An increase of 30% in the average lithium carbonate price, from $11,000 to $14,300, increases the Post-Tax NPV from $802M to $1,430M and the Post-Tax IRR to 30%. A decrease of 30% in the average lithium carbonate price, from $11,000 to $7,700, decreases the Post-Tax NPV from $802M to $148M and Post-Tax IRR to 11%.

Table 1.14.1: Sonora Lithium Project – Key Economic Parameters

Parameter	Unit	Value
Pre-tax NPV	$M	1,253
Pre-tax IRR	%	26
Simple Payback	y	4
Initial Construction Capital Cost	$M	420
Stage 2 Construction Capital Cost	$M	380
Average Life of Mine ("LOM") operating costs	$/t Li_2CO_3	3,910
Average LOM operating costs - net of K_2SO_4 credits	$/t Li_2CO_3	3,418
Average yearly EBITDA with co-products	$M/y	229
Post-tax NPV (at 8% discount)	$M	802
Post-tax IRR	%	21
Li_2CO_3 production (Years 1 to 4)	t	70,080
Li_2CO_3 production (Years 5 to 19)	t	530,847
K_2SO_4 production (Years 5 to 19)	t	466,612

1.15 Conclusions and Recommendations

Financial modelling carried out for the FS demonstrates that the Sonora Lithium Project is financially viable. Additional development is required at the start of the next phase (detailed design) for critical vendor packages to facilitate procurement activities. An allowance of $518,261 is included in the capital cost estimate to facilitate the costs associated with the necessary development work.

Prior to the completion of detailed design and commencement of construction complete geotechnical information will be required. Work to obtain this information is currently in progress and outcomes are expected April 2018.

Additional well testing is recommended to ensure the proposed borefield can supply the water needs during an extended drought condition in the area. As part of the hydrogeological investigation, another borefiled was identified and should be further investigated as a viable backup source for the project water supply.

Further process and testwork investigation is recommended for consideration to further de-risk technical aspects of the project and optimize operating parameters. The key technical aspects to be de-risked involve engagement with vendors for the kiln and crystalliser and evaporator packages. Bacanora has already commenced engagement and sharing of key feasibility testwork outcomes, where appropriate, with vendors to facilitate this. The costs for this development work have been included in the capital cost estimate. The costs associated with optimisation of process parameters have generally not been explicitly included. While useful, optimisation of process parameters is not viewed as critical to project development, especially when compared to the necessary development for the kiln and crystalliser and evaporator packages. Optimization work (i.e. reagent additions) would be reasonably expected to positively influence project economics.

The proposed execution schedule, whilst achievable, is considered ‘fast track’ and is reliant upon rapid decision making, unencumbered design processes, collaborative engagement, and no adverse outcomes from the recommended work. Delays or adverse outcomes during the next phase may delay the delivery schedule, especially where there is an impact on the long lead or critical procurement items.

The majority of the Measured and Indicated Mineral Resources have been converted to Mineral Reserves. There is potential to upgrade current Inferred Mineral Resources to Measured and Indicated Mineral Resources through additional exploration and infill drilling; these may convert to additional Mineral Reserves in future if they satisfy the relevant technical and economic criteria.

1.16 Forward Work Program

1.16.1 Geology

Not relevant.

1.16.2 Mining

As this project nears the mine operations stage the following tasks should be addressed in order to conduct an efficient and successful mining operation. The construction of the Stage 1 plant is expected to be approximately two years and most of the mine advance work can be done during the plant construction time period.

- Develop a mine operations block model of the clay zones to include any vertical changes in grade within the clay ore zones.

- Develop standard operating procedures for:
  - Mining ore and waste with the surface miner
  - Mining waste material adjacent to ore
  - Upper waste stripping.
- Evaluate and optimize waste storage locations.
- Hire open pit manager and prepare the RFP's for mine equipment purchase quotations or evaluate mine contractor bids if mine contractors are to be used.

Within approximately six months of mine start up:

- Hire mine planning engineers for short range mine planning
- Hire mine geologist for resource and geologic model updates and set up sampling protocol
- Develop sampling procedures to identify the Lithium grade trends in the host clay material to facilitate blending of different grade ranges in the plant feed
- Hire remaining technical staff
- Hire and train equipment operators and supervisory staff.

1.16.3 Process

The following process related activities are required to facilitate process plant design and operation:

- Additional testwork to quantify the breakthrough and determine the loading capacity of the activated alumina
- Testwork to confirm the design parameters for the boron removal columns
- Optimisation of the carbon dioxide sparging in order to reduce the dissolution time within bicarbonation
- Kiln design development and proof of concept are required to ensure mass throughput and design temperatures are achieved
- Additional evaporator / crystalliser testwork to improve the potassium ratio in the glaserite formed and to optimize yield of potassium sulphate when decomposing
- Practically demonstrate the mechanism for the distribution and harvesting of briquettes at the plant site.

1.16.4 Infrastructure

Infrastructure work is recommended in the following areas:

- The fieldwork for the geotechnical investigation was completed at the end of December 2017 and results are expected by April 2018. The outcomes from this investigation should be compared against the assumptions that were used to underpin the engineering work completed during the feasibility study.
- A detailed geotechnical field investigation program should be developed to obtain final design parameters for the plant area and TSF. Duration for the field program and testing is expected to take approximately six weeks for a total cost of approximately $200,000. This cost has not been explicitly included as a line item in the capital cost estimate.

- Further geotechnical testing of the co-mingled material should be performed and model for short term and long-term stability of the TSF based on how the different configurations the materials may be placed during operations. Duration and costs for these test is approximately six weeks and $20,000-$30,000. This cost has not been explicitly included as a line item in the capital cost estimate.
- Engagement with vendors for the build-own-operate gas pipeline and power station should continue to ensure availability in-time for the expected commissioning of the plant in late 2019.
- Engagement with vendors for the build-own-operate gas pipeline and power station should continue to ensure availability in-time for the expected commissioning of the plant in late 2019.

1.16.5 Environment

Environmental work is recommended in the following areas:

- A social engangment plan should be developed to ensure risks are mitigated as the project continues through construction and operation.
- A site wide project water balance should be developed to confirm water needs thoughout each phase of the project and quantify discharge quantities for permiting purposes.
- The proposed borefield should be further tested in order to quantify the flows for each well while minimizing long term impacts within the borefield.
- Pumps tests should be performed at the other borefields identified to ensure Stage 2 and a backup source of water can be confirmed. Total costs and duration for these test is approximately $250,000 and 8 weeks, respectively. This cost has not been explicitly included as a line item in the capital cost estimate.

2 INTRODUCTION

2.1 Background

The Sonora Lithium Project consists of seven exploration and mining concessions (the “concessions”). Within these concessions, the ‘La Ventana’ part of the project is owned 99.9% by Bacanora and the other concessions are owned jointly with Cadence Minerals PLC, comprising 70% ownership by Bacanora and 30% by Cadence. Refer to Section 4.3 for further details of mineral tenure.

This Technical Report has been prepared for Bacanora and summarizes the FS completed in December 2017.

2.2 Project Scope and Terms of Reference

The Project consists of an open pit mine and an associated processing facility along with on-site and off-site infrastructure to support the operation with a mine life of 19 years. The nominal yearly output for the project is 17,500 tonnes per year of battery-grade Li_2CO_3 (Stage 1), for the first four years of the project, followed by a proposed expansion, by duplicating the plant, to produce a total of 35,000 t/y (Stage 2). In addition, the Project has been designed to produce up to 28,805 t/y of potassium sulfate, for sale to the fertiliser industry.

A FS was completed for Bacanora in December 2017 to provide information to determine the economic feasibility of developing the Sonora Lithium Project, and to determine whether to proceed to project execution and the requirements necessary to do so. This Technical Report summarizes the outcomes of the FS in accordance with the disclosure requirements.

2.3 Study Participants

Ausenco was commissioned by Bacanora in May 2016 to prepare the FS and NI 43-101 compliant technical report on the Project. SRK was engaged to prepare the Mineral Resource estimate and to supervise geology inputs. IMC was engaged for mine design, mine operating costs, mine capital and operating costing and economic modelling. Solum was engaged to conduct environmental and social studies. Bacanora produced the economic model. Table 2.3.1 provides an overview of the key participants and their area of responsibility.

Environmental and social studies, carried out by Solum, are based upon the Sonora Lithium Project being located within the La Ventana basin which is a sub-basin of the Rio Bavispe Bajo. Investigations conducted include protected natural areas, flora, fauna, surface water, ground water and social-economic activities. As part of the ongoing permitting approval process, Solum is currently preparing a Manifestación de Impacto Ambiental (“MIA”) (Expression of Environmental Impact) and Estudio Tecnico Justificativo (“ETJ”) (Land-Use Change) for the site access road, borefield and borefield corridor. The ETJ for the plant site is also being prepared. These permits are to be submitted to local government authorities in Q1 2018.

Table 2.3.1: Study Participants

Area of Responsibility	Company
Geology and Mineral Resource Estimate	SRK
Mining	IMC
Testwork	SGS & ANSTO (managed by Ausenco)
Environmental and Social Impact Assessment	Solum
Flowsheet Development	Ausenco
Process Plant Design, Engineering, and Plant Layout	Ausenco
Infrastructure	Ausenco & Bacanora
Capital Cost (Mining)	IMC
Capital Cost (Process and Overall Compilation)	Ausenco
Operating Cost (Mining)	IMC
Operating Cost (Process and Overall Compilation)	Ausenco
Tailings Storage Facility	Ausenco & Solum
Implementation and Execution Planning	Ausenco
Marketing	SignumBOX–Bacanora
Financial Modelling	Bacanora

2.4 Site Visit

The site visit and inspection of the sample preparation facilities were undertaken initially between (PFS) 24 and 27 March 2015 and then again between (FS) 20 and 25 June 2016 by Martin Pittuck. Martin is a full time employee of SRK and supervised the resource estimation process.

Joel Carrasco previously visited the site on 19 August 2015 to select the location for the Tailings Storage Facility (“TSF”) and to inspect the site access road.

Herb Welhener, vice president of IMC, last visited the site on 28 July 2017.

Greg Lane, co-ordinating author and a full time employee of Ausenco, did not visit the project site. Greg visited the key laboratories undertaking the FS metallurgical testwork programs.

2.5 Frequently Used Abbreviations, Acronyms and Units of Measure

Where they are used in this report, abbreviations, acronyms, definitions and terms have the meaning shown in Table 2.5.1.

Table 2.5.1: Abbreviations, Acronyms and Units of Measure

Abbreviation	Description
A	Ampere
amsl	Above Mean Sea Level
ANSTO	Australian Nuclear Science and Technology Organisation
BCM	Bulk Cubic Meter
BG	Battery Grade
BOO	Build Own Operate
°C	Degrees Celsius
cm	Centimetre
CRM	Certified Reference Material
d	Day
DD	Detailed Design
d/y	Days per year
Datamine	Datamine Studio 3 software
DEM	Digital Elevation Model
EBITDA	Earnings before interest, taxes, depreciation and amortisation
EDTA	Ethylenediaminetetraacetic acid
ETJ	Estudio Tecnico Justificativo
FEED	Front End Engineering and Design
FEL	Front End Loader
FS	Feasibility study
h	Hour
h/d	Hours per day
ha	Hectare
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ICP-MS	Inductively Coupled Plasma Mass Spectrometer
IDW	Inverse-distance weighted algorithm
IRR	Internal rate of return
ISE	Ion-Selective Electrode
IX	Ion exchange
J	Joule (energy)
k	Kilo or thousand
kg	Kilogram
km	Kilometre
kt	Kilo tonne (thousand metric tonne)
kW	Kilowatt (power)
kWh	Kilowatt hour
L	Litre

Abbreviation	Description
LCE	Lithium Carbonate Equivalent
LCT	Locked Cycle Testwork
Leapfrog	Leapfrog geo software
LNG	Liquefied natural gas
LOM	Life of Mine
m	Metre
M	Million
m^2	Square metre
m^3	Cubic metre
MCC	Motor control centre
MIA	Manifestacion de Impacto Ambiental
mm	Millimetre
MRE	Mineral Resource Estimate
Mt	Million tonnes (metric)
Mt/y	Million tonnes per year
MOP	Muriate of Potash
MW	Megawatt
NPV	Net present value
OK	Ordinary kriging
P_{80}	80% passing size
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PLC	Programmable Logic Controller
PLS	Pregnant Liquor Solution
QA–QC	Quality Assurance and Quality Control
RCW	Radial Collector Well
ROM	Run-of-mine
s	Second
SCADA	Supervisory Control and Data Acquisition
SGS	Société Générale de Surveillance
SOP	Sulfate of Potash
Supervisor	Supervisor software
t	Tonne (metric)
t/h	Tonnes per hour
t/m^3	Tonnes per cubic metre
t/y	Tonnes per year
TSF	Tailings storage facility
µm	Micrometre or micron

Abbreviation	Description
UTM	Universal Transverse Mercator conformal projection
V	Volt
VAT	Value added tax
XRD	X-Ray Diffraction
XRF	X-Ray Fluoresence

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared for Bacanora by SRK, IMC, Ausenco and Solum (the “Authors”) based on assumptions as identified throughout the text and upon information and data supplied discussed below.

Bacanora has provided validation of mineral tenement and land and mineral tenure status, specifically location and ownership agreements, including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.

Ausenco is not an expert in the matters of pricing of lithium carbonate and potassium sulfate. For the information contained in Section 19 Ausenco has relied on information from Bacanora. Bacanora has advised that the information provided is based on a report (Lithium Market Report, 20 October 2017) by SignumBOX.

Bacanora and its consultants have used a forecast Battery Grade Lithium Carbonate price given in a report prepared by SignumBOX Inteligencia de Mercados (“SignumBOX”), a Chilean based research company that provides market intelligence reports and consulting services in the natural resources industries, with a specific focus on the lithium industry. A key focus of their business is Market Studies looking at demand estimation, supply and forecast of future production capacity, and price modelling and forecast. SignumBox has used its existing database and market intelligence on the lithium market to provide an expert opinion to Bacanora.

Ausenco is also not an expert in royalties and matter of taxation. For this information Ausenco has relied upon information provided by Bacanora.

4 PROJECT DESCRIPTION AND LOCATION

4.1 Property Area

As discussed in Section 4.3, the licence holding by the Company forms a continuous coverage over the Project area of 8,154 ha. This is illustrated in Figure 4.3.3 and Figure 4.3.4. La Ventana and La Ventana 1, covering approximately 1,820 ha. The five concessions El Sauz, El Sauz 1, El Sauz 2, Fleur and Fleur 1 cover approximately 6,334 ha in total.

4.2 Project Location

The Project is situated in the northwestern Mexican state of Sonora, some 11 km south of Bacadehuachi which is 180 km northeast of Hermosillo and approximately 170 km south of the USA – Mexico border. A location plan is given in Figure 4.3.2.

4.3 Mineral Tenure

For clarity of understanding, in March 2017 Rare Earth Minerals Plc ("REM") changed their name to Cadence Minerals Plc. Any legacy reference to REM in this technical report should be read as similarly referring to Cadence.

The Sonora Lithium Project is an exploration project, part of which is owned 99.9% by Bacanora and part of which is owned jointly by Cadence (30%) and Bacanora (70%). Cadence also owns 9.25% of Bacanora.

The Sonora Lithium Project consists of seven concessions which confer rights for exploration, mining and production. In addition, Bacanora is a 70% owner of an additional 3 concessions, which surround the Sonora Project, which are not part of this MRE or FS. The concessions are owned by a number of Cadence-Bacanora subsidiaries:

- Within Sonora Project:
  - Mexilit SA de CV ("Mexilit"), owned 70% by Bacanora
  - Minera Sonora Borax SA de CV ("MSB"), owned 99.9% by Bacanora.
- Outside Sonora Project:
  - Megalit SA de CV ("Megalit"), owned 70% by Bacanora.

Two concessions (La Ventana and La Ventana 1) are 100% owned by MSB. Another five concessions (El Sauz, El Sauz 1, El Sauz 2, Fleur and Fleur 1) are 100% owned by Mexilit. Three concessions (San Gabriel, Buenavista and Megalit) are 100% owned by Megalit. Mexilit and Megalit are owned 70% by Bacanora and 30% by Cadence.

The data and MRE described in this report relate only to the Mexalit and MSB concessions. The concessions held by Megalit have not been reviewed by SRK and the Mineral Resource statement does not include material from the Megalit concessions.

A separate subsidiary 'Minerales Industriales Tubutana SA de CV' is also owned under the Bacanora umbrella. However, this subsidiary deals solely with the Company's borate holding and as such is not referred to further in this report. The current ownership structure of the Project concessions can be seen in Figure 4.3.1.

Of the 10 concessions held within this company structure and dealt with in this program of study, 9 have been issued to the Company and one has been applied for and currently is 'Approved for Title'. The issued and Approved for Title concessions of Bacanora Minerals Ltd are set out in Table 4.3.1.

The “Approved for Title” stage of application applies to the Megalit concession which does not contain any of the Mineral Resource reported herein. A summary of the process of obtaining title to a concession from the Mexican Federal Mining Registry is as follows:

- Initially an application for title is submitted to the local registry where the property is located
- Following the submission of the application, the applicant has 60 days to file a survey with the local registry
- Upon receipt of the survey, the local registry reviews and either approves it or responds to the applicant and gives them a further 15 days to correct their survey
- If the survey is approved (that is, no objections are conveyed to the applicant), it is stamped “Approved for Title” and is submitted to the Federal Mining Registry in Mexico City for them to grant title to the applicant as a final administrative step.

In July 2014 and as part of Bacanora’s admission to the AIM market on the London Stock Exchange, a legal opinion was prepared in relation the mineral concession status. The opinion prepared by Melicoff & Asociados Abogados confirmed that:

- Each mining concession is in full force and effect and has been duly validated by the Mexican Mining Bureau and is free from any liens and encumbrances.
- Each mining concession was validly issued for a period of 50 years.
- Each of the mining concessions are in good standing, and they are not subject of any unusual or onerous conditions, and their existence or validity will not be effected by any change of control.
- Bacanora and Cadence do not see any reason why the pending applications which have been granted full concession status by the Ministry of Mining will not be approved by the Ministry of Mining and confirm that these transfers are being processed.

The Directors of Bacanora believe that there is minimal risk of title not being eventually granted for concessions currently “Approved for Title”. Further the Directors state that Bacanora is, and has been, appropriately able to conduct its exploration activities within these concessions consistent with Approved for Title status. Once the concession that is presently “Approved for Title” has been issued, the concessions will be transferred to Megalit in line with Mexican law and applicable regulations and in accordance with the contractual obligations under the agreements between Bacanora and Cadence.

The licence holding by the Company forms a continuous coverage over the Project area. This is illustrated in Figure 4.3.2 and Figure 4.3.3. La Ventana and La Ventana 1, covering approximately 1,820 ha. The five concessions El Sauz, El Sauz 1, El Sauz 2, Fleur and Fleur 1 cover approximately 6,334 ha in total and the additional three concessions Buenavista, Megalit and San Gabriel cover approximately 89,235 ha in total.

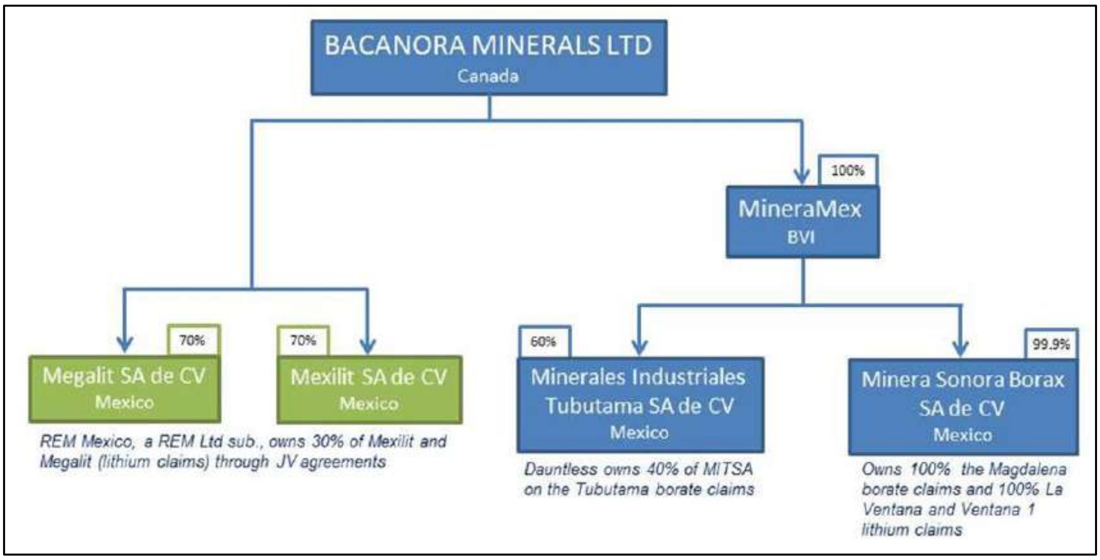


Figure 4.3.1: Current Project Ownership Structure

Table 4.3.1: Concessions of Bacanora Minerals Ltd (Note: Red Indicates Outside Sonora Project)

Company	Concession	Locality	Title ref.	Area (ha)	Licence Accepted	Expiry
Minera Sonora Borax	La Ventana	Bacadehuachi	235611	875	22-Jan-10	21-Jan-60
Minera Sonora Borax	La Ventana_1	Bacadehuachi	243127	945	10-Jul-14	09-Jul-64
Mexilit	El Sauz	Bacadehuachi	235614	1,025	22-Jan-10	21-Jan-60
Mexilit	Fleur	Bacadehuachi	243132	2,335	10-Jul-14	09-Jul-64
Mexilit	El Sauz_1	Bacadehuachi	244345	200	11-Aug-15	10-Aug-65
Mexilit	El Sauz_2	Bacadehuachi	243029	1,144	30-May-14	29-May-64
Mexilit	Fleur_1	Bacadehuachi	243133	1,630	10-Jul-14	09-Jul-64
Megalit	Buenavista	Huasabas	235613	649	22-May-10	21-May-60
Megalit	San Gabriel	Bacadehuachi	235816	1,500	12-Mar-10	11-Mar-60
Megalit	Megalit	Bacadehuachi		87,086	"Approved for title"	

Figure 4.3.2: Project Location Plan

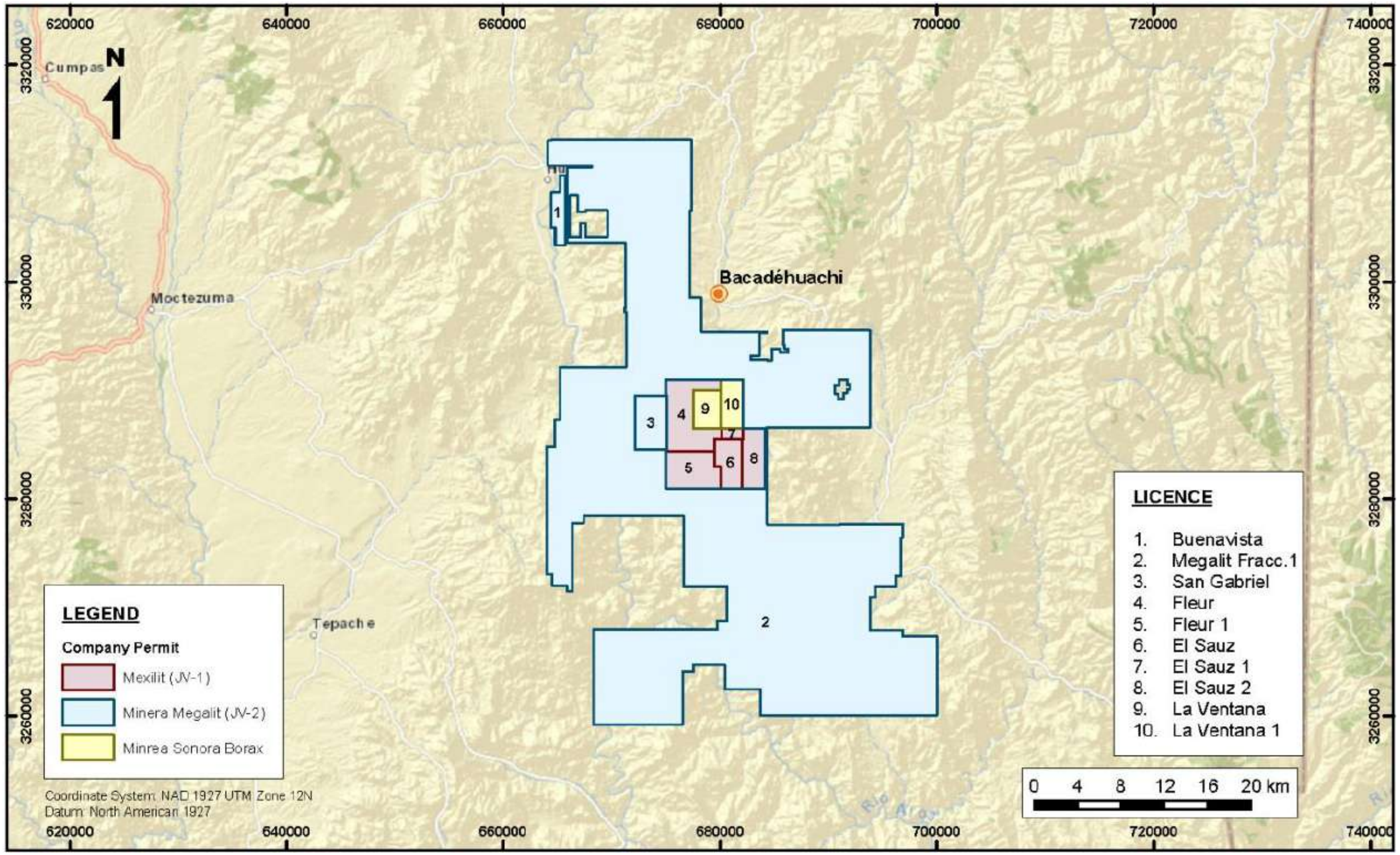


Figure 4.3.3: Location of the Sonora Lithium Project Concessions, Mexico (Note: Only Mexalit and MSB are Described in This Report)

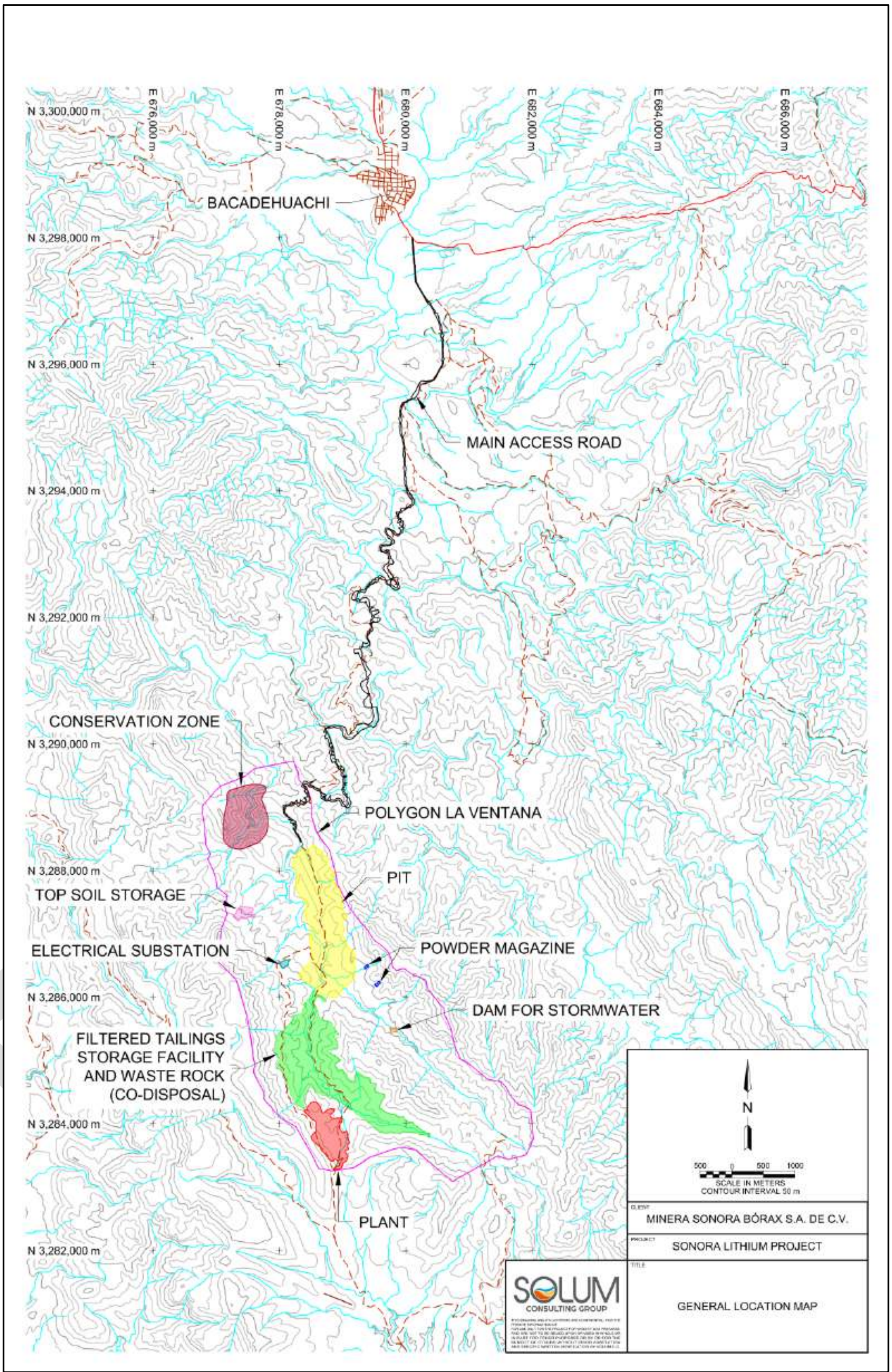


Figure 4.3.4: Project Plan

4.4 Surface Rights

Surface rights sufficient for mining operations are obtainable have been purchased from local landowners, should such activities develop on the concessions and title is currently held by MSB.

4.5 Royalties

Bacanora has advised that a 7.5% Mining Royalty tax is due based solely on the mining parts of the operations. In addition, there is a 3% royalty due on all product sales to Mr Colin Orr-Ewing, which has been included in the Life of Mine cashflows.

4.5.1 Orr Ewing Royalty

The Company has previously disclosed the existence of an agreement between the late Mr. Colin Orr- Ewing, the past Chairman of the Company, and the Company subjecting the Sonora Lithium Project to a 3% gross overriding royalty (the "Royalty") on production from certain concessions within the Sonora Lithium Project. The Company understands that the Royalty is now held by the estate of Mr. Colin Orr-Ewing. On November 17, 2017, the Company filed a statement of claim with the Court of Queen's Bench (Alberta) seeking to void ab initio, the Royalty. The basis of the Company's claim is that the Royalty was originally granted based on the misrepresentation of Mr. Colin Orr-Ewing that he held a pre-existing royalty granted prior to the acquisition of the lithium properties by the Company. The Board of Directors of Bacanora has completed a review of the historical background and concluded that no such pre-existing royalty existed and accordingly there was no basis for the grant of the royalty by the Company. In relation to the economic model, until the legal matter is resolved, the Company has included the 3% Gross royalty on all sales of Lithium carbonate.

4.5.2 Mexican Mining Royalty

In 2014, the Mexican Government introduced a Mining Royalty of 7.5% on mining revenues based on the same taxable base as for calculating income tax with the exception of excluding depreciation, amortisation and interest (i.e.: 7.5% x EBITDA). This royalty is itself deductible against the net income used to calculate general corporation income tax at 30%. The royalty is applicable to the holder of the mining concessions. There have been considerable legal challenges by the industry in terms of determining the taxable base that should be subject to the mining royalty. In relation to the economic model, the Company has included the 7.5% royalty on the net income from the revenues directly attributable to the Company's Mining operations. The Company has taken appropriate Transfer Pricing advice to comply with OECD tax guidelines on the pricing of the mined ore as it is sold to the Chemical Processing part of its business.

4.6 Environmental Liabilities

This is a greenfields site which has had exploration drilling carried out. No environmental liabilities are known to exist at the Project.

4.7 Permits

Federal and State permits include:

- The Preventative Notice (Informe Preventivo)
- The Environmental Impact Assessment (Manifestación de Impacto Ambiental) - refer to Section 20.1 for the details and schedule associated with the Manifestación de Impacto Ambiental

- The Permit for Change of Land Use (Estudio Tecnico Justificativo) in Forested Area issued by the State Delegations of Secretary of the Environment, Natural Resources and Fisheries (SEMARNAT)
- A PPA (Accident Prevention Program)
- A water use permit (Comisión Nacional del Agua)
- An archaeological land liberation, based on authorization by the Instituto Nacional de, Antropología e Historia
- Explosives Use Permit (SEDENA)
- A notice to the state and municipal authorities (i.e., local construction permits, land use change, etc.).

4.8 Site Access Risk Factors

The Project area specifically is accessed by way of Federal Highway 14, a two-lane highway extending 225 km east of Hermosillo, to the intersection known as “El Coyote”, then south from the intersection for 20 km on a recently paved, two-lane highway to the town of Bacadéhuachi. Bacanora has set up its local base of operations in this town and undertakes all core processing facilities from this location.

Access to the concessions from Bacadéhuachi is on secondary, dry-weather roads, crossing various privately owned ranches for approximately 11 km. There are two (2) crossing where during a large storm there will be significant water in the streams. These storms are high intensity and short duration storms and should not have negative impacts to the access to the site as the water level will drop typically within a few hours.

The region is well known for cattle ranching, and ranches and fenced zones cross the area. The ranchers have created a network of secondary dirt roads to access other areas, and these roads provide access to the concessions.

Land owners have provided authorisation for the Company to access the concessions on these roads. Renewable easements (concurrent with concession renewal) for site and borefield access are currently under negotiation and expected to be complete in Q1 2018. Land purchase negotiations for the borefield, solar drying area and accommodation facility are also in progress and expected to be complete in Q1 2018.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography

The Sonora Lithium Project is situated within the Sonoran Desert in the western portion of the Sierra Madre Occidental physiographic province, within the Basin and Range sub province. It lies between “Mesa de Enmedio”, “Rincon del Sauz” and “El Capulin” mountain ranges. Average elevation at the Project area is 900 m above mean sea level (“amsl”). The concessions are surrounded by mountain peaks with elevations ranging up to 1,440 m amsl.

The area has mountainous relief with deeply incised valleys where the clay units outcrop in some places; the outcrop geometry is affected by the topography and several faults which offset the deposit.

A detailed 1 m resolution topographic survey has been undertaken, covering the extent of the known lithium deposit included in this study. Topographic data was collected using LiDAR simultaneously with high resolution aerial photography.

5.2 Site Access

Access to the concession from Bacadehauchi is on secondary, dry-weather roads, crossing various privately owned ranches for approximately 11 km. The region is well known for cattle ranching, and ranches and fenced zones cross the area. The ranchers have created a network of secondary dirt roads to access other areas, and these roads provide access to the concessions. Land owners have provided authorisation for the Company to access the concessions on these roads.

5.3 International Access

Sonora lies on the geographic corridor connecting the central Mexican highlands (Mexico City) north into the United States of America along the Pacific Coast. It is a major corridor for travel and shipping.

The state has four airports in the cities of Hermosillo, Puerto Peñasco, Ciudad Obregón and Nogales. These airports connect the state with various locations within Mexico and international.

Railway lines mostly consist of those which lead into the USA.

Guaymas is a city located in the southwest part of the state of Sonora; it is the principal port for the state. Figure 4.3.2 shows the location of Guaymas in relation to the site. The port has road and rail access and container and bulk handling capabilities.

It is expected that the Port of Guaymas will be utilised for the export of products from the project to Asia. The Port of Guaymas will be accessed via the Federal Highway 14 and 15. Trucked product in containers will be taken from site to El Coyote, which is situated on Federal Highway 14, then south west to Hermosillo and then south to the Port of Guaymas via Federal Highway 15.

Product being delivered to North America would be trucked in containers to Hermosillo using Federal Highway 14 where they will be loaded onto trains and transported to the USA and Canada.

5.4 Regional and Local Access

Sonora State and therefore the Project area has well developed infrastructure with more than 24,000 km of highways including a four-lane highway (Highway 15) that crosses the state from south to north

The Project area specifically is accessed by way of Federal Highway 14, a two-lane highway extending 225 km east of Hermosillo, to the intersection known as “El Coyote”, then south from the intersection for 20 km on a recently paved, two-lane highway to the town of Bacadehuachi. Bacanora has set up its local base of operations in this town and undertakes all core processing facilities from this location.

5.4.1 Proximity to Population Centres

Bacadéhuachi, approximately 11 km from the Project, is the closest town to the Project. It is a small farming and ranching community with a population of approximately 2,010. Basic services capable of supporting early stage exploration projects are available in the town.

The Project is approximately three hours' drive north east of the state capital of Hermosillo, a city of approximately 700,000 people. Rail, road and natural gas networks join Hermosillo to the United States of America and Mexico.

5.5 Physiography

The Sonora Lithium Project is situated within the Sonoran Desert in the western portion of the Sierra Madre Occidental physiographic province, within the Basin and Range sub province. It lies between “Mesa de Enmedio”, “Rincon del Sauz” and “El Capulin” mountain ranges. Average elevation at the Project area is 900 m amsl. The concessions are surrounded by mountain peaks with elevations ranging up to 1,440 m amsl.

5.6 Climate

The average ambient temperature is 21°C, with minimum and maximum temperatures of -5ºC and 50ºC, respectively in the project area. Extreme high temperatures, upwards of 49ºC occur in summer. Winters are considered cool compared to most of Mexico.

The accumulated annual rainfall for the area is approximately 450 mm. The wet season or desert “monsoon” season occurs between the months of July and September, and heavy rainfall can hamper exploration at times. The Sonoran Desert, because of its seasonal rainfall pattern, hosts plants from the agave, palm, cactus and legume family, as well as many others. The local climate provides no incumbents to undertaking field programs and as such the length of the operating season is 365 days a year.

5.7 Infrastructure

Bacadehuachi historically is a small farming and ranching community with a population of approximately 2,010. Basic services capable of supporting early stage exploration projects are available in the town.

The closest electric power line is about 10 km north of the concessions, passing very close to Bacadehuachi. The power line then heads toward Nacori Chico, the next village southeast from Bacadehuachi.

The town is served by municipal water drawn provided by a community well. Propane for heating and cooking is delivered by truck.

6 HISTORY

There are no records of mineral exploration or mineral occurrences on the Property prior to 1992, when an American group, US Borax, initiated regional exploration work in the search for borate deposits.

6.1 Previous Mapping and Surface Sampling

In 1996, US Borax conducted detailed field work in the area which consisted of geological mapping and rock sampling. The mapping resulted in the discovery of sequences of calcareous, fine-grained sandstones to mudstones intercalated with tuffaceous bands that are locally gypsiferous. Rock sampling across representative sections of the sequence at intervals along the strike extensions of these units returned weakly anomalous boron values, consequently US Borax abandoned exploration in the area.

6.2 Drilling by Previous Explorers

No drilling has been undertaken on the licence concessions prior to Bacanora commencing operations in 2010.

6.3 Previous Mineral Resource Estimation

6.3.1 Amerlin Exploration Services 2014

Bacanora has completed mapping, chip sampling, trenching, metallurgical testwork and drilling on the Project. Mineral Resources have been previously estimated by Bacanora for the lithium bearing clays on the Company's concessions which were reported in *'Updated and reclassified Lithium resources, Sonora Lithium project, Sonora Mexico'* produced for Bacanora Minerals Ltd on 24 June, 2014 (C Verley of Amerlin Exploration Services Ltd). Within this document, Verley updated earlier estimates based on additional drilling in 2013 and 2014; in the process, reclassifying all resources from inferred to indicated (not reported using NI 43-101 guidelines).

6.3.1.1 El Sauz and Fleur Concessions

A Mineral Resource estimate was undertaken for the area drilled on the El Sauz and Fleur concessions using a polygonal estimation method. Grade and thickness continuity were assumed in an area of influence around each drill such that: (i) in the north-south direction the influence area is half of the distance between holes; and (ii) in the east-west direction a distance from outcrop and extending down dip for 150 m was used. Plan views illustrating the areas of the polygons used in the estimate are provided in Figure 6.3.1. Dry density values of 2.38 and 2.35 tonnes per cubic metre (t/m^3) were assumed for the estimate for the Upper and Lower Clay units respectively. The resulting grade and tonnage estimates were reported at cutoffs of 1,000, 2,000 and 3,000 ppm Li, with a cutoff of 2,000 ppm Li used as a base case scenario for future study work.

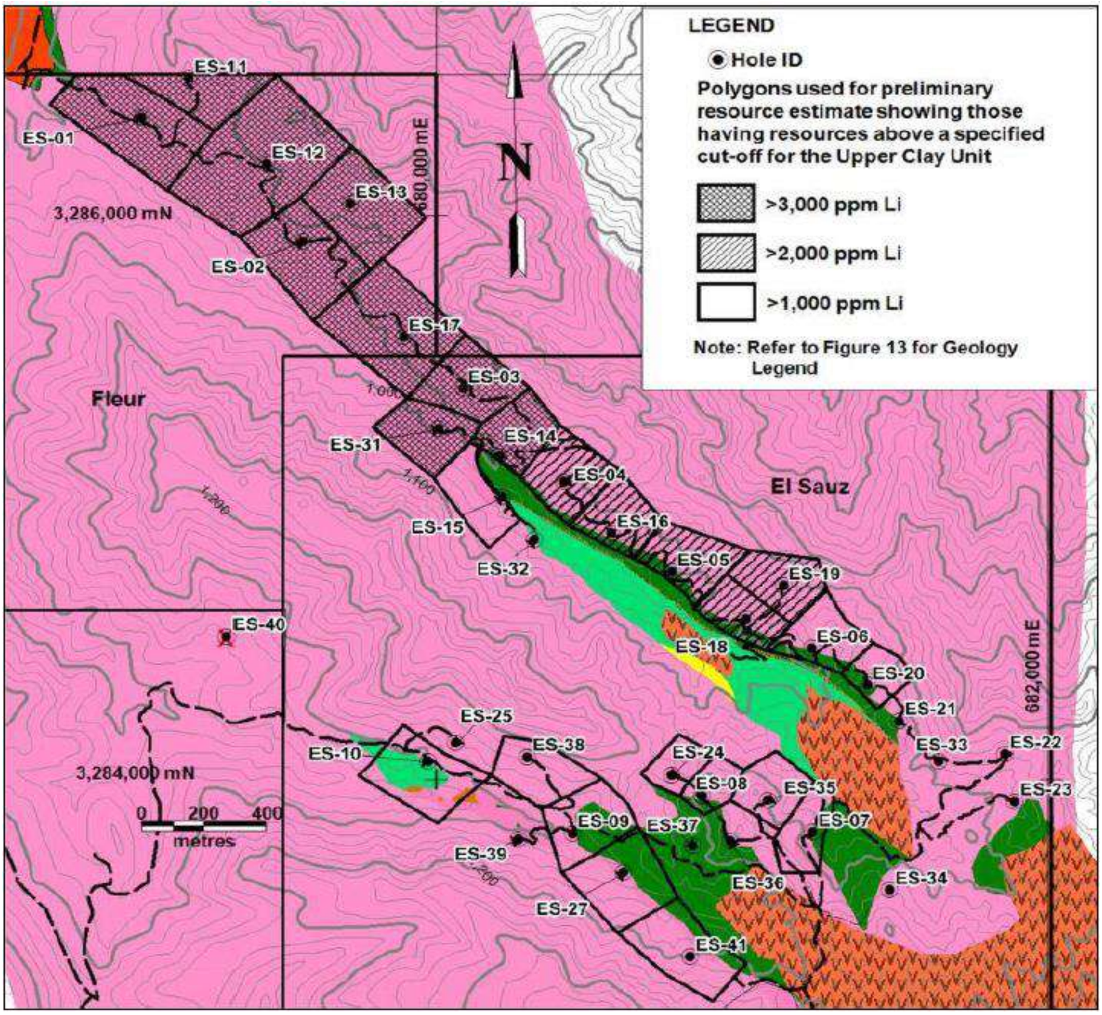


Figure 6.3.1: Plan of Resource Polygons and Base Geological Map for the Fleur and El Sauz Concessions

A total Indicated Mineral Resource, based on CIM Definition Standards for Mineral Resources and Reserves (2010), was estimated for each of the lithium-bearing units and is given in Table 6.3.1. At a cutoff of 2,000 ppm Li, the base case Indicated Mineral Resource for the Upper Clay unit is estimated to be 47 Mt averaging 2,222 ppm Li, and for the Lower Clay unit the Indicated Mineral Resource is 74 Mt averaging 3,698 ppm Li, giving a total Indicated Mineral Resource of 121 Mt averaging 3,120 ppm Li. A distinct zone of higher grade lithium occurs in the northern part of El Sauz and Fleur and continues through Fleur onto the southern half of La Ventana. In the Mineral Resource statement, the lithium metal content is also given as a Lithium Carbonate Equivalent ("LCE"); using a conversion factor of 1 unit of lithium metal is equivalent to 5.32 units of LCE.

Table 6.3.1: Historic Indicated Mineral Resources for El Sauz and Fleur (Verley, 2014)

Lithological Unit	Li (ppm) Cutoff	Tonnage (Mt)²	Li (ppm)	LCE (%)¹	LCE Tonnage (kt)²
Upper Clay	1,000	97	1,657	0.88	856
	2,000	47	2,222	1.18	560
	3,000	18	3,773	2.01	369
Lower Clay	1,000	98	3,028	1.61	1,584
	2,000	74	3,698	1.97	1,450
	3,000	59	4,140	2.20	1,298
Combined	1,000	195	2,347	1.25	2,440
	2,000	121	3,120	1.66	2,010
	3,000	77	4,053	2.15	1,667

[1]LCE = Lithium carbonate equivalent and assumes that all lithium can be converted to lithium carbonate with no recovery or processing losses.
[2] Dry bulk density = 2.38 t/m^3

6.3.1.2 La Ventana

Based upon drilling undertaken during 2010, 2011 and 2013, Verley used a polygonal estimation method to produce an Indicated Mineral Resource for the La Ventana concession based upon the same logic and processes as presented for the El Sauz and Fleur concessions. Plan views illustrating the areas of the polygons used in the estimate are provided in Figure 6.3.2.

A total Indicated Mineral Resource, based on CIM Definition Standards for Mineral Resources and Reserves (2010), was estimated for each of the lithium-bearing units and is given in Table 6.3.2. Using a 2,000 ppm Li cutoff, an Indicated Mineral Resource for the Upper and Lower Clay Units of 75 Mt averaging 3,174 ppm Li (1.69% LCE) or 1,273 kt LCE was estimated. Both the Upper and Lower Clay Units were considered to be open down-dip.

Table 6.3.2: Historic Indicated Mineral Resources for La Ventana Concessions (Verley, 2014)

Lithological Unit	Li (ppm) Cutoff	Tonnage (Mt)²	Li (ppm)	LCE (%)¹	LCE Tonnage (kt)²
Upper Clay	1,000	31	1,824	0.97	289
	2,000	21	2,256	1.2	258
	3,000	10	3,186	1.7	170
Lower Clay	1,000	61	3,247	1.73	1,055
	2,000	54	3,540	1.88	1,015
	3,000	38	4,510	2.40	917
Combined	1,000	92	2,771	1.48	1,353
	2,000	75	3,174	1.69	1,273
	3,000	48	4,235	2.25	1,087

[1]LCE = Lithium carbonate equivalent and assumes that all lithium can be converted to lithium carbonate with no recovery or processing losses.
[2] Dry bulk density = 2.38 t/m^3

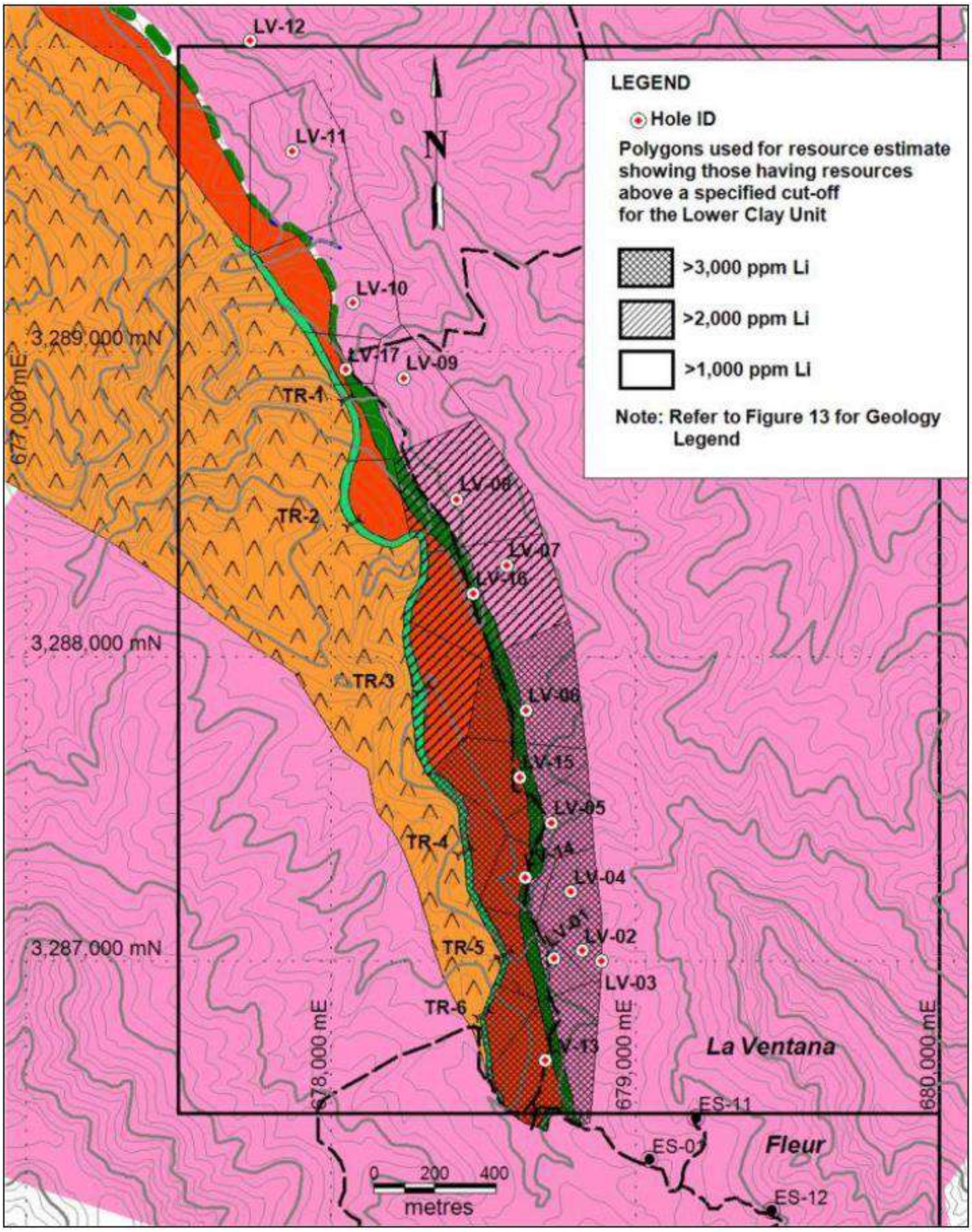


Figure 6.3.2: Plan of Resource Polygons and Base Geological Map for La Ventana

6.3.2 SRK May 2015

SRK completed an MRE in May 2015 ("May 2015 MRE") using all data collected prior to the August/September 2015 drilling campaign. The May 2015 MRE utilized 3D wireframing techniques and block modelling with grades interpolated using Ordinary Kriging ("OK"). A pit optimization was run on the block model to assess the 'reasonable prospects for economic extraction' and the Mineral Resource is stated within the maximum profit pit. The Mineral Resource statement produced by SRK is provided in Table 6.3.3. The methodology and results of the May 2015 MRE were described in a NI 43-101 technical report (SRK, 2015). Grades of potassium were not estimated in 2015.

Table 6.3.3: Previous SRK Mineral Resource Statement (SRK, May 2015)*

Classification	Mt	Li (ppm)	K (%)	LCE Mt
Measured	-	-	-	-
Indicated	95	2,200	-	1.1
Meas+Ind	**95**	**2,200**	**-**	**1.1**
Inferred	510	2,400	-	6.3
Total	**606**	**2,300**	**-**	**7.5**

***Notes:**

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.
2. The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101 and JORC.
3. The MRE is reported on 100 percent basis for all project areas.
4. SRK assumes the Sonora Lithium deposit to be amenable to surface mining methods. Using results from initial metallurgical test work, suitable surface mining and processing costs, and forecast LCE price SRK has reported the Mineral Resource at a cutoff 450 ppm Li (2,400 ppm Li_2CO_3).
5. SRK completed a site inspection of the deposit by Mr. Martin Pittuck, MSc, C.Eng, MIMMM, an appropriate "independent qualified person" as such term is defined in NI 43-101

6.3.3 SRK April 2016

SRK completed an MRE in April 2016 ("April 2016 MRE") using all data collected prior to the August/September 2016 drilling campaign. The April 2016 MRE utilised the same techniques and reporting procedures as with May 2015. The Mineral Resource statement produced by SRK is provided in Table 6.3.4. The methodology and results of the April 2016 MRE were described in the PFS report in April 2016 (Ausenco, 2016).

Table 6.3.4: Previous SRK Mineral Resource Statement (SRK, April 2016)*

Classification	Mt	Li (ppm)	K (%)	LCE Mt
Measured	-	-	-	-
Indicated	259	3,200	1.4	4.5
Meas+Ind	**259**	**3,200**	**1.4**	**4.5**
Inferred	160	3,200	1.3	2.7
Total	**419**	**3,200**	**1.4**	**7.2**

***Notes:**

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.
2. The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101 and JORC.
3. The MRE is reported on 100 percent basis for all project areas.
4. SRK assumes the Sonora Lithium deposit to be amenable to surface mining methods. Using results from initial metallurgical test work, suitable surface mining and processing costs, and forecast LCE price SRK has reported the Mineral Resource at a cutoff 1,000 ppm Li (5,323 ppm Li_2CO_3).
5. SRK completed a site inspection of the deposit by Mr. Martin Pittuck, CEng, MIMMM, FGS, an appropriate "independent qualified person" as such term is defined in NI 43-101.

7 GEOLOGICAL SETTING AND MINERALISATION

The content of this section is largely based on the following report; *Updated and Reclassified Lithium Resource, Sonora Lithium project* by C Verley (Amerlin Exploration Services Ltd), which was lodged with the Canadian Securities Administrators 24 June 2014.

7.1 Regional Geology and Tectonics

The Property is underlain by Oligocene to Miocene age rhyolitic tuffs, ignimbrites and breccias of the upper volcanic complex of the Sierra Madre Occidental. This succession was subjected to basin and range extensional normal faulting during the Miocene that resulted in the development of a series of half-grabens. The half-grabens locally filled with fluvial-lacustrine sediments and intercalated tuffs. Alkaline volcanism around this time is thought to have contributed lithium and other alkali metals into these basin deposits. Quaternary basalt flows unconformably cover the basin sediment-volcaniclastic succession, except where later stage faulting and uplift have exposed the basin succession at surface. Mineralisation on the Property consists of lithium-bearing clays localized within these basins.

7.2 Deposit Stratigraphy

Geological mapping has defined the following stratigraphic sequence, outlined in Table 7.2.1. The lithium-bearing sedimentary sequences are well defined and are distinct from the surrounding volcanics by their pale colour and fine to medium bedding, they have been recorded and characterised as dominantly north striking, easterly dipping, Li-bearing sediments. Controls for the lithium sedimentary sequence and resulting mineralisation are considered to follow the shape of a lake in which the clays became entrained. Faults underlying the lake may have served as channel ways for lithium-rich solutions to percolate into the lake basin and possibly alter and enrich the existing clays in lithium. Alternatively, the lithium may have been sourced from underlying volcanics and remobilised into the basin sequence at a later date; however, rhyolites with sufficient lithium-rich melt inclusions to act as source material have not yet been identified in the sequence presented or regionally.

The lithium-bearing clays occur in two discreet units: an upper clay unit and a lower clay unit. The Lower Clay Unit is underlain by basaltic flows, breccias and tuffaceous rocks and is overlain by an ignimbrite sheet. The average thickness of the Lower Clay Unit is approximately 20 m reaching 40 m in places. The ignimbrite sheet is typically 6 m thick and is overlain by the Upper Clay Unit which averages 22 m and reaches over 70 m in thickness; the Upper Clay Unit is overlain by a sequence of basalt flows and intercalated flow top breccias.

These stratigraphic units are reasonably continuous across the La Ventana, Fleur and El Sauz concessions.

Both the Upper and Lower clay units are considered to consist of several mineralised subunits. The Lower Clay Unit consists of a basal red siltstone-sandstone-conglomerate unit, tuffaceous sediments, thin lapilli tuff layers and reworked tuff layers interbedded with lithium-rich clay layers.

The Upper clay unit, consists several subunits of thin, rhythmically laminated clay and silica layers, coarse-grained, poorly sorted brown sandstone beds with a clayey and calcareous matrix; yellowish green clay beds with silica nodules; dark grey clay bands with distinct slump features and local calcite masses; light grey claystone layers interbedded with reddish sandstone beds; reddish medium to coarse-grained sandstone with calcite veinlets.

Table 7.2.1: Stratigraphic Succession on the El Sauz Concession (Verley, 2014)

Unit	True Thickness (m)	Unit/Subunit Description
Capping basalt	Not determined	Basalt. Contains greenish olivine crystals. Veinlets of kaolinite/alunite (white/greenish, powdery).
Upper clay unit	28.0 (14.10 – 40.39)	Reddish, medium-coarse grained sandstone with calcite veinlets.
		Pale grey tuffaceous claystone intercalated with reddish, sandy layers. Scarce FeOx layers (black).
		Dark grey slumping breccias. Dark, clayey groundmass with tuffaceous fragments. Calcite in masses.
		Green-yellowish silica nodules in a clayey waxy, tuffaceous matrix.
		Brown sandstone. Poorly bedded. Highly calcareous. Reddish tuffaceous coarse grained sandstone. Clay matrix. Soft.
		Pale green-pinkish, fine grained sequence of clays and silica nodules. Waxy in zones. Calcite in masses.
Ignimbrite	5.58 (1.29 – 11.89)	Ignimbrite: orange coloured, welded lapilli tuff. Locally brecciated.
Lower clay unit	27.78 (21.57 – 42.11)	Pale grey reworked tuff with abundant lithium-bearing clay zones.
		Pale green tuffaceous sediments. K-feldspar groundmass with quartz and biotite. Indurated. Contains lapilli tuff.
Basement Volcanics	Not determined	Dark green basalt, andesitic basalt and rhyolite tuff.

7.3 Deposit Structure

The lithium-bearing sedimentary sequences are considered distinct and easily distinguished in the field from the surrounding volcanics by their pale colour and thin to medium bedding, as illustrated in the northeast view of gently, northeasterly dipping, lithium-bearing sediments near the centre of the El Sauz concession (Figure 7.3.1). On the La Ventana concession, lithium-bearing clay units are exposed from the northwest corner of the concession to the southeast of the concession, a distance of 3.6 km. The sediments dip approximately 20° to the northeast. A mapped northwesterly striking oblique slip fault has down thrown the clay units to the south of La Ventana under basalt cover so they no longer remain exposed at surface. Drilling has confirmed the continuity of the clay units under the basalt cover for a distance of 2.0 km to the southeast where they are again exposed at surface, on the El Sauz concession for a further distance of 2.0 km to the southeast. In total a 7.6 km strike length of the clay unit from the north end of La Ventana to the southern part of El Sauz has been established in both the upper and lower clay units. The deposit is open at depth; however, the down dip extent to the northeast, southwest and south is not known at present and remains to be tested by further drilling.

The more southerly exposures of the clay units occurring on the western extent of the oblique slip fault and exposed on the El Sauz concession dip gently westerly probably as a result of offsets and rotation on faults. In addition, exposures of the basement volcanics consist of rhyolite tuff on the southern part of El Sauz versus andesitic basalt on La Ventana.

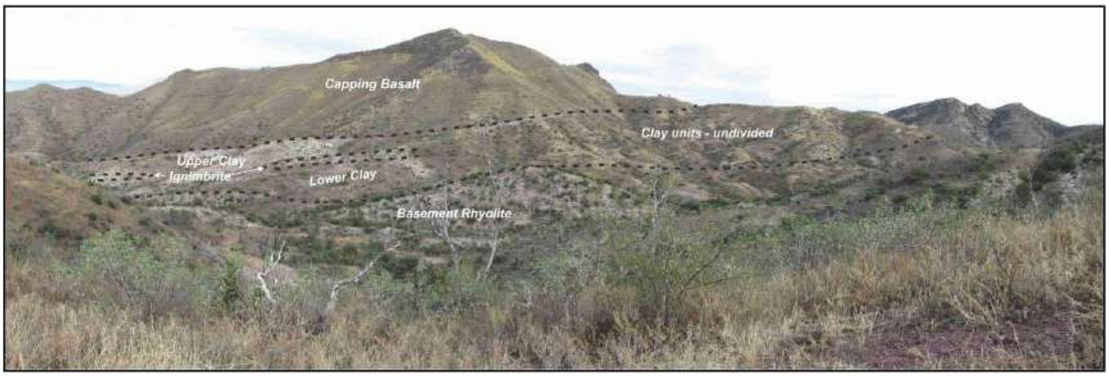


Figure 7.3.1: Northeast View of Gently Dipping Lithium-Bearing Sediments Near the Centre of the El Sauz Concession

7.4 Mineralisation

Mineralisation on the concessions consists of a series of lithium-bearing clays that occur within two bedded sequences, the Upper and the Lower Clay units, which are separated by an ignimbrite sheet.

Bacanora understands there to be a number of lithium-bearing clay minerals, with polylithionite being the only one currently positively identified. The clay units are believed to have formed from supergene or diagenetic alteration of volcanic ash. The clay layers also contain relict quartz and feldspar crystal shards, lithic fragments and silica bands (Figure 7.4.1), and traces of other minerals. The layers are locally interbedded with reddish terrigenous beds composed of sand and silt-sized material.

Initial interpretation has indicated a high grade lithium core in the area covered by the La Ventana, El Sauz and Fleur concessions where the lithium grades are generally above 3,000 ppm Li. This high grade zone extends from the middle of La Ventana southward across Fleur and approximately a third of the distance south into El Sauz. The best grades of lithium are associated with elevated levels of calcium, caesium, magnesium, potassium, rubidium and strontium; however, the correlation (especially for magnesium) is not one-to-one.

On La Ventana, the best grades of lithium are co-incident with elevated levels of potassium and caesium and are found in the southern part of the deposit. Magnesium appears to be irregularly distributed and does not follow lithium or the other alkalis. Mineralised intervals within the clay units vary for the Upper Clay Unit from 25% to 80% of the overall thickness and from 40% to 100% for the Lower Clay Unit, depending on the cutoff used. Vertical grade variation is noted in places, but with the exception of the Upper Clay Unit in the main eastern fault block it has not been identified with sufficient continuity between drillholes to have been reflected in the 3D modelling process described herein.

Further mineralogical studies are recommended to determine what minerals host the various alkalis in the clay units. Results of such studies could have an impact on beneficiation of these minerals and recovery of the alkalis.

Figure 7.4.1: Alternating Clay and Silica Bands Within an Outcrop on the La Ventana Concession

8 DEPOSIT TYPE

8.1 Deposit type

The Sonora deposit is believed to have formed by hydrothermal alteration as a result of alkaline volcanism effecting layers of volcaniclastic sedimentary rocks deposited in a basin environment. The origin and timing of the mineralised content remains unclear with regard to source and whether the alteration was essentially syn-genetic with deposition of the sedimentary rocks or whether the alteration is a post depositional event. Additional work is required to clarify the origin of these deposits.

The Western Lithium Kings Valley development project, Humbolt County, Nevada, has similar mineralogy and deposit geology to the Sonora Project, but the exact lithium clay mineralogy and regional geological setting is significantly different.

There are no directly analogous deposits known to be in operation.

8.2 Adjacent/Regional Deposits

The Sonora region plays a large part in Mexican production of ore minerals, predominantly silver, celestite and bismuth. The state has the largest mining surface in Mexico, and three of the country’s largest mines: La Caridad, Cananea, and Mineria María. Sonora also remains the leading Mexican producer of gold, copper, graphite, molybdenum, and wollastonite, as well as one of the largest coal reserves in the country. This has resulted in established and well maintained resources, specifically infrastructure which services the existing mining industry through the region.

9 EXPLORATION

9.1 Introduction

There are no records of mineral exploration or mineral occurrences in the Project area prior to 1992, when US Borax initiated regional exploration work in the search for industrial minerals. In 1996, US Borax conducted detailed field work in the area, which consisted of geological mapping and rock sampling. The mapping resulted in the discovery of sequences of calcareous, fine-grained sandstones to mudstones intercalated with tuffaceous bands that are locally gypsiferous. Rock sampling across representative sections of the sequence at intervals along the strike extensions of these units returned weakly anomalous boron values. Consequently, US Borax abandoned exploration in the area.

In 2010, Bacanora initiated a program of limited rock sampling on the La Ventana concession this work led to the discovery of lithium-bearing clays. Follow-up work in 2011 on the El Sauz concession led to the discovery of the lithium-bearing clays within this concession.

9.2 Surface Sampling Program

9.2.1 2010 La Ventana Concession

Bacanora's initial exploration efforts were focused on testing the clay exposures located on the La Ventana concession. In 2010, a series of six continuous chip samples were taken perpendicular to the strike of upper clay unit at the south end of the concession.

Each sample was placed in a numbered, fibre-weave sack. The samples were then taken to ALS Chemex facility in Hermosillo for lithium analysis and a multi-element scan using ICP-MS techniques.

The results of this work confirmed the elevated lithium concentrations in the clay unit. Values for the six samples ranged from 1,710 to 4,680 ppm Li (0.91 to 2.49% LCE).

Bacanora then conducted a diamond drilling campaign at La Ventana in 2010. A total of four holes were drilled as an initial test of the lithium-bearing clay units.

9.2.2 2011 El Sauz Concession

A geological reconnaissance and rock-sampling program was conducted on the El Sauz concession by Bacanora's geologists during the period 28 September to 11 November 2011. A total of 116 rock samples were collected from exposures of a pale coloured, clay-bearing sequence of sediments and intercalated tuffaceous rocks. The samples were collected across outcrops as continuous chip samples ranging in width from 0.9 to 2.2 m. and averaging 2.0 m. perpendicular to the strike direction of the sediments. Sample spacing was dependent on exposure; consequently, it was difficult to ascertain how representative the samples were of the overall clay-bearing units on the El Sauz concession.

The sampled exposures occur in the northern half of El Sauz and dip to the east, in the case of the northeastern most outcrops and to the west in the case of the more southerly exposures. These opposing dips appear to indicate an anticlinal structure. The initial mapping of the Fleur and El Sauz concessions is shown in Figure 9.2.1.

Results of analyses performed on the samples by ALS Chemex ranged from 49 to 7,220 ppm Li, with 39 samples greater than 1,000 ppm Li. The results indicated that significant lithium-bearing clay units occur on El Sauz.

A total of 94 rock samples averaging 1.7 kg were taken from outcrops of the clay units exposed on the El Sauz concession. The samples were collected across outcrops as continuous chip samples perpendicular to the strike direction of the sediments. Results of analyses performed on the samples by ALS Chemex ranged from 10 to 2,130 ppm Li, with 15 samples greater than 1,000 ppm Li. The results further confirmed the 2011 work, which indicated that significant lithium-bearing clay units occur on El Sauz warranting further work to more accurately assess the extent of the units and the concentration of.

In conjunction with the rock sampling, the geology of the area around the clay units on El Sauz and Fleur were mapped (Figure 9.2.2). Structurally, the clay units on El Sauz and Fleur dip to the northeast at approximately 20° and in the central part of El Sauz the clay units crop out in an arcuate form, with the more easterly arm of the arc dipping to the northeast and the westerly arm dipping westerly.

The geological mapping and Stage 1 drill program suggested that the strata on El Sauz were a continuation of those found on the La Ventana concession. From this comparison it was concluded that the lithium-bearing clay units on the El Sauz are a southern extension of the sedimentary basin from La Ventana onto the Fleur and El Sauz concessions.

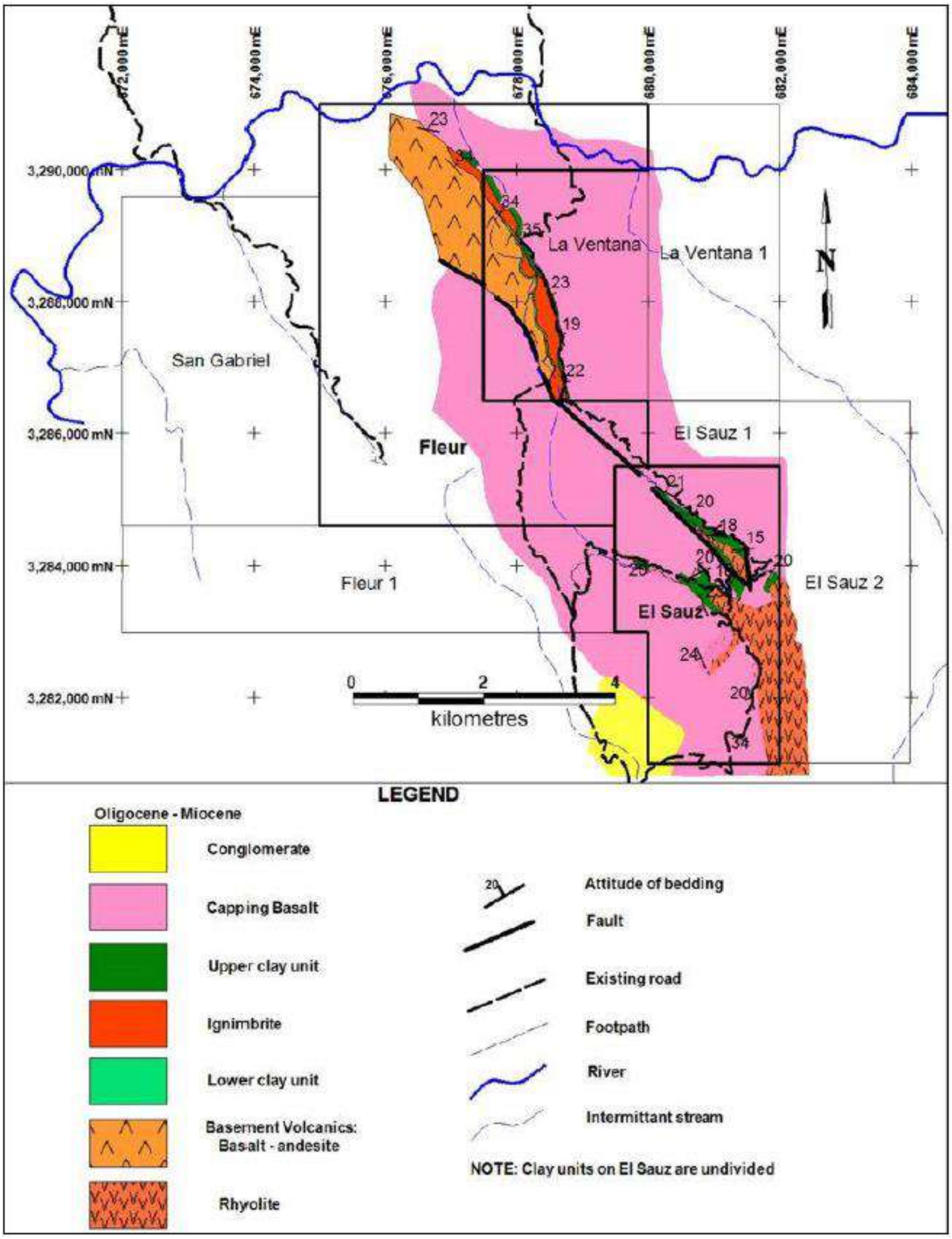


Figure 9.2.1: Initial Mapping Undertaken for the Sonora Lithium Project

9.2.3 2013 – El Sauz Concession

From February to April, 2013, the mapping and rock sampling campaign continued on the Fleur and El Sauz concessions, as shown in Figure 9.2.2.

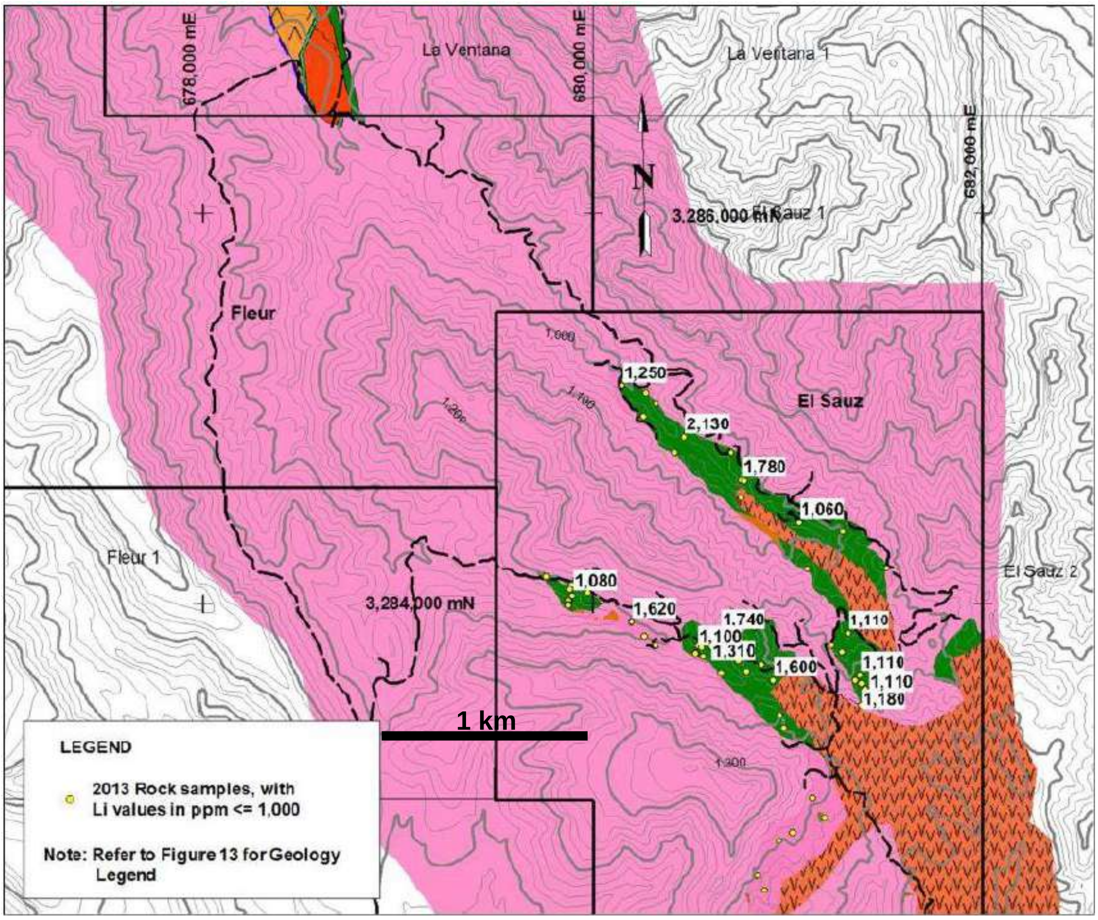


Figure 9.2.2: 2013 Surface Sampling and Mapping Undertaken on the El Sauz and Fleur Concessions

9.3 Trenching

In early 2014, six trenches for a combined length of 140.8 m were excavated across exposures of the Lower Clay Unit on La Ventana to provide additional grade control. Continuous chip samples were taken at intervals averaging 1.5 m in length. Figure 9.3.1 shows TR-6 excavated across the Lower Clay Unit in La Ventana. Collar locations of the trench samples are listed in Table 9.3.1 and illustrated Figure 9.3.2.

Table 9.3.1: Trench Collar Locations

Trench	Easting	Northing	Elevation	Length (m)
TR-2	678073.4	3288432	874.7755	30
TR-3	678298.8	3287890	883.1865	27.7
TR-4	678436.1	3287359	925.7235	27
TR-5	678569.9	3287025	882.845	22.5
TR-6	678487.2	3286830	929.467	33.6
Total				**140.8**

Figure 9.3.1: TR-6 Excavated Through Clay Horizon in the South of La Ventana

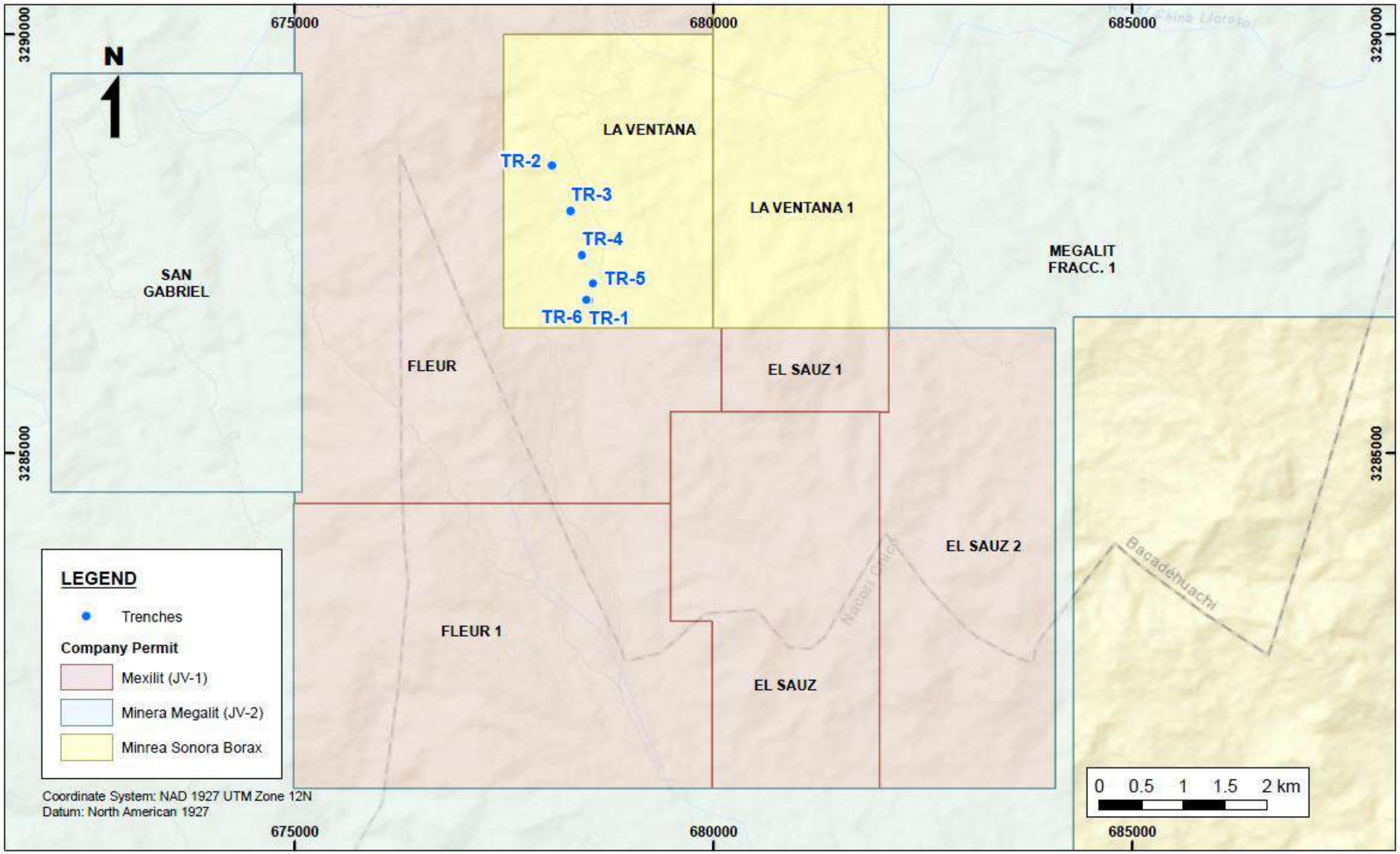


Figure 9.3.2: 2014 Trench Locations

10 DRILLING

10.1 Introduction

In 2010, Bacanora commenced a diamond drill program on the La Ventana concession before expanding the targeted area to include the El Sauz and Fleur concessions in 2013. Further drilling was conducted in 2015 and in 2016 to improve the drilling grid density. At the time of writing, a total of 17,965 m in 128 holes has been completed on the Sonora Lithium Project.

The drilling has demonstrated that the lithium mineralisation exists in two units along approximately 7.2 km of strike length.

All the drilling conducted to up until 2016 was undertaken by Perforaciones Godbe de Mexico SA de CV, a Mexican subsidiary of Godbe Drilling LLC, based in Montrose, Colorado. The drilling for the 2016 campaign was completed by Toro Drilling, based in Hermosillo, Mexico. The drill rig used for the 2016 drilling is shown in Figure 10.1.1, observed in operation during the 2016 site visit by SRK.

Drilling has been completed on a 100 to 250 m grid basis with locations frequently constrained by access and topography.

10.1.1 La Ventana Concession

Bacanora´s first drilling campaign on the La Ventana concession was conducted from May to September 2010. Four holes totalling 458 m were completed in this initial program using NQ-core diamond drilling. Drill sites were laid out to optimally test a section of the lithium-bearing clays exposed at the south end of the La Ventana concession with holes completed on 200 m spacing along strike.

A second campaign in 2011 totalled 1,453 m in 8 drillholes and extended the known strike length of the deposit to over 2.5 km. The culmination of a successful surface mapping program (outlined in Section 9.2) and sub-surface intercepts established the continuity of both the upper and lower clay mineralised units down dip and along strike.

Drilling in the La Ventana concession continued through 2014 and 2015. This program comprised 27 holes for 3,154 m of NQ drill core. This drilling has increased the depth extent of the upper and lower clay units and further confirmed the lithological continuity along strike.

The 2016 campaign aimed to infill drill the La Ventana area to ensure a consistent grid of data across the most economically attractive of the Project where the lower clay unit outcrops. This resulted in 31 holes for 3,896 m of NQ drill core.

Figure 10.1.1: 2016 Toro Drilling Rig Producing NQ Drill Core

10.1.2 Fleur and El Sauz Concessions

In addition to the drilling undertaken on the La Ventana licence, Bacanora has undertaken a number of drill programs aimed at extending the known strike of the mineralised clay units towards the southeast through the Fleur and El Sauz concession areas, driven by the continuity established in the La Ventana concession and supported by a positive surface mapping and sampling programs which are outlined in Section 9.2.

An initial drilling campaign was undertaken from May to September 2013 in which a total of 1,470 m of NQ-core was completed in 10 holes. Drill sites were laid out with the objective of testing the extension of the lithium-bearing clays on the La Ventana concessions which outcrop in El Sauz.

A second drill program on the Fleur and El Sauz concessions commenced in October 2013 and was completed in February 2014. A total of 2,436 m of NQ drilling was completed in 20 holes extending the strike extent of the known lithium mineralisation. This drilling also defined the southern and southwestern extents of the mineralised unit. This area is considered to be more structurally complexity as a result of numerous offset fault sets and potential rotation or folded movement within the stratigraphic sequence.

A third drill program along with field mapping was undertaken on the Fleur and El Sauz concessions from late 2014 to early 2015 comprising 12 drillholes totalling 1,164 m. This program targeted this structurally complex area to test continuity using a 200 m drill spacing as used in La Ventana and along the eastern extent of El Sauz and Fleur. This drilling and additional mapping established that the mineralisation dips gently toward the east in this area.

A four drill program was completed in summer 2015 which comprised 16 drillholes totalling 3,934 m. This program aimed to provide more detail in the southeastern area of the Fleur concession and northern area of the El Sauz concession, where the majority of higher grade lithium is situated.

10.2 Collar Surveys

All collars were surveyed using a handheld GPS unit (Garmin 62S) taking an average waypoint over a time lapse of five minutes. Due to the higher resolution of the LIDAR topographic survey, the elevation (Z) values of the collars were taken from the LIDAR survey. All collar related coordinates are reported in UTMNAD27 Z12.

10.3 Down-hole Surveys

All but four drillholes have been drilled vertically. None of the holes have been surveyed with down-hole survey or core orientation technology. The four holes are relatively short (<200 m) and it is not expected that significant deviation would have occurred.

10.4 Summary of Drillhole Locations

Figure 10.5.1 shows the locations of the drillhole collars across the Sonora concessions. These holes have been coded based on year drilled and as such reflects the development of the project over time. A smaller-scale map of just the La Ventana area with the 2016 drilling is shown in Figure 10.5.2.

10.5 Summary of Major Mineralisation Intersections

A summary of all major lithium mineralisation intersections within the modelled resource wireframes is provided in Appendix A.

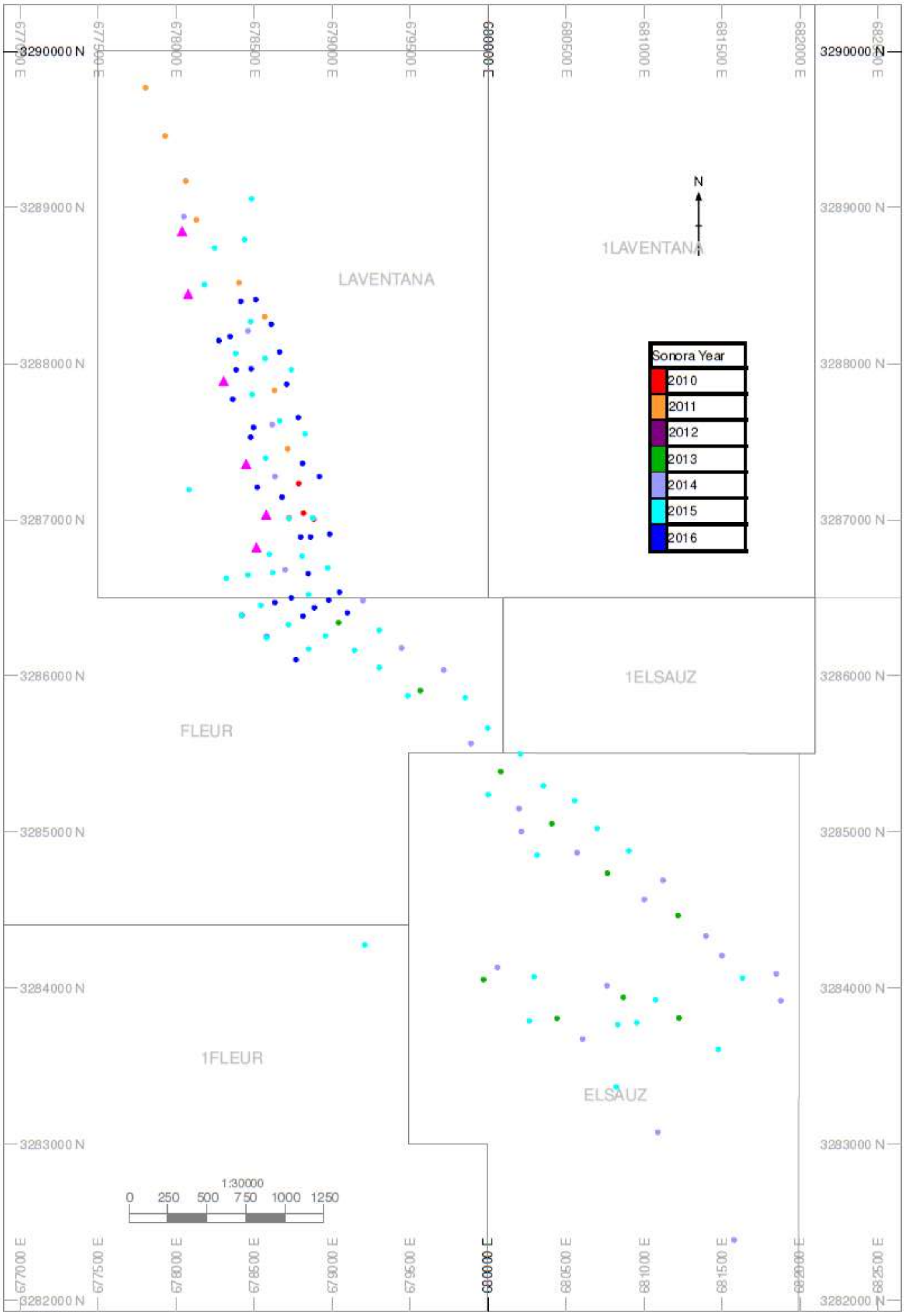


Figure 10.5.1: Sonora Concessions Drillhole Collars (Pink Triangles = Trenches)

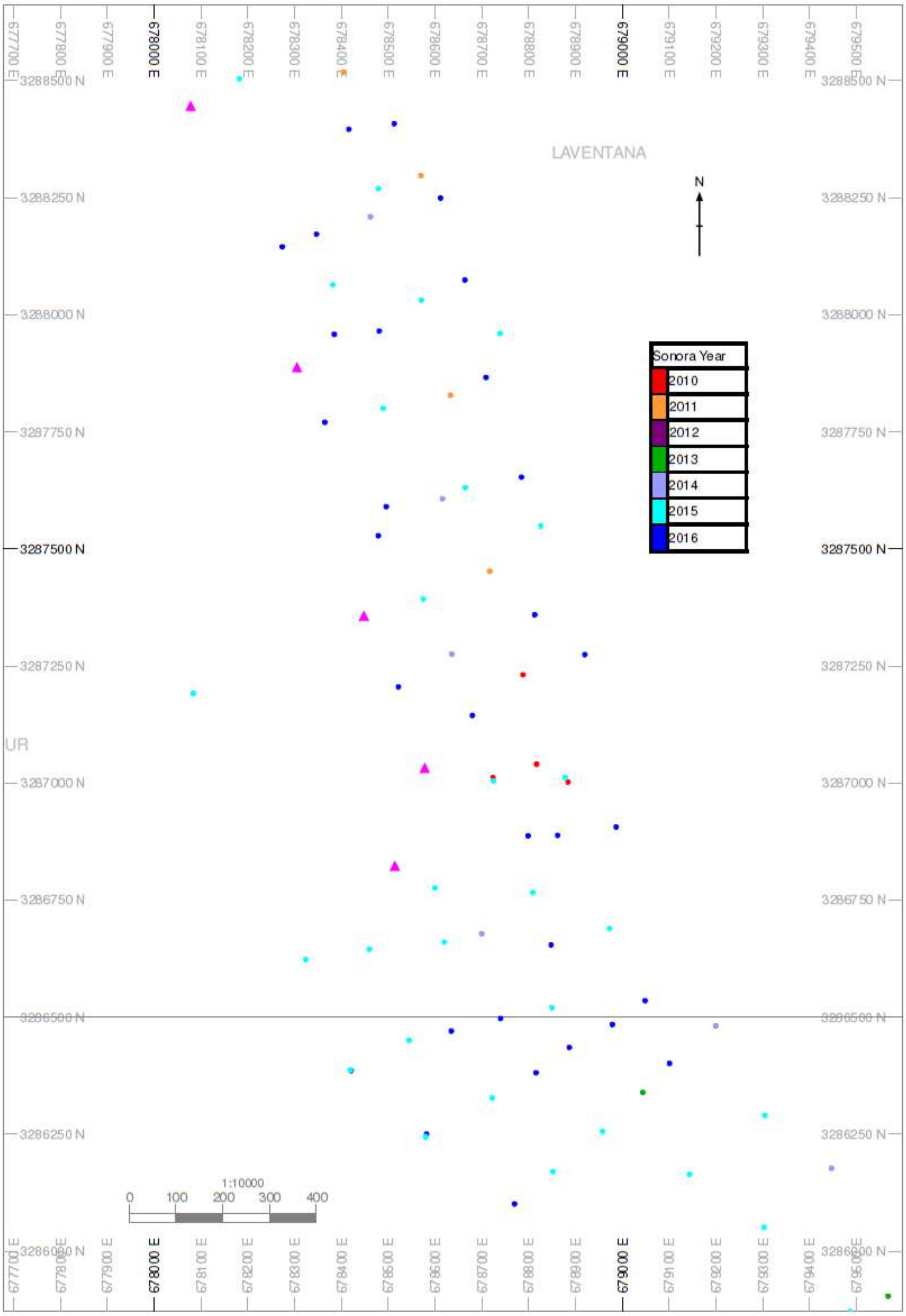


Figure 10.5.2: La Ventana Drillhole Collars (Pink Triangles = Trenches)

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Sampling Methodology and Approach

All core drilled on site was arranged into referenced core boxes and moved from the drill sites by Bacanora personnel to a secure compound in Bacadehuachi where under the supervision of the onsite geologist, it was logged, split and sampled (Figure 11.1.1). Core was then moved to Bacanora's secured facility in Magdalena de Kino for storage. In addition to logging of geological parameters in drill core, core recovery was also measured and recorded.

Figure 11.1.1: Bacanora Staff Preparing Core in a Dedicated and Secure Compound, Bacadehuachi

11.1.1 Core Presentation and Photography

Core and core blocks are placed in core boxes by the driller. Upon receipt in the core shed, the drill core is cleaned or washed, if required, and core blocks are checked by Bacanora staff. The core is split using a hydraulic splitter and then photographed wet and dry in a frame ensuring a constant angle and distance from the camera (Figure 11.1.2).

Figure 11.1.2: Drill Core Presented After Cut and Sampling Procedures

11.1.2 Logging

Geological logging is undertaken once core photography is complete. Logging includes recording from-to intervals and brief descriptions of the lithological units as well as observations and measurements regarding core recovery. The key logging codes used by Bacanora are summarized in Table 11.1.1.

Table 11.1.1: Key Logging Codes Summarized Based on Bacanora Core Logging Procedures

Geological Unit	Code	Lithology	Description
Capping basalt	UBAS	Capping Basalt	Dark grey olivine basalt. Massive.
Upper Sandstone	UPP_SS	Reddish sediments	Reddish-grey medium to coarse grained sandstone. Poorly bedded to massive. Abundant calcite, some iron oxides.
Upper clay	UTC	Upper Tuffaceous sequence	White to light grey claystone. Oxidized. Lithic and reworked. Contains sanidine crystals. Slightly calcareous
	CALCLS	Calcareous sequence	Pink to dark breccias, silty-muddy matrix. Abundant calcite in masses and veinlets. Feldspar altered to clays
	WAXCLS	Tuffaceous sequence	Light green-white altered tuff. Feldspar is being converted into clays (light green honey). Contains glass crystals (sanidine) and biotite. Waxy.
	BRSS	Brown/reddish sandstone	Brown sandstone. Poorly bedded. From 112 to 113. highly calcareous. Reddish tuffaceous coarse grained sandstone. Clay matrix. Soft.
	HS	Hot Spring Type Section	Light green-pink fine grained sequence composed of clays and silica nodules. Waxy in zones. Folded. Friable. Abundant calcite in masses and veinlets. Thin bedded.
Ignimbrite	IGNIMBRITE	Tuffaceous sequence	Orange to pink welded tuff. Well indurated. Brecciated. Highly silicified. Contains pumice flames.
Lower Clay	LWR-T-SED	Lake-beds-altered	Dark green sequence composed of rhythmic beds of clay-silica-marls with abundant calcite in masses and veinlets. Some dark zones with clay and organic matter. Thin to medium bedded.
	LART	Lower Sediments	Grey well indurated sandstone. Reworked andesitic tuff?
	LCGL	Lower conglomerate	Polymictic conglomerate. Reddish matrix to the top and greenish to the bottom. Purple-greenish-white fragments.
Basement	LBAS_AND	Lower Basalt Andesite	Dark green basalt. Biotite rich (black) in a fine grained groundmass. In some holes tuff with andesite frags.

11.1.3 Density Measurements

Dry in situ density readings are taken at regular intervals within each lithology and on every lithological break. The methodology involves weighing dry samples in air and then in water, all porous samples being wrapped in plastic first. Measurements are carried out on competent whole core using a balance with top and modified under-slung measuring capabilities with a detection limit of ±1 g. The following protocols were being followed for the 2016 drilling campaign, with the additional step of the oven-dried values included only for this campaign:

1. At the drill rig, density samples (~20 -30 cm in length) are selected and wrapped in cellophane to ensure in-situ moisture is maintained until measured at the core logging facility. Two density measurements per distinct lithology (including internal lithologies within the generalised Upper Clay/Lower Clay units) is taken from every hole
2. Record down-hole from and to depths, lithology (“UNIDAD”) and assign a density sample number

3. Unwrap sample, discard cellophane and coat in acrylic spray. Ensure table is level and steady and ensure scales are set to zero, then record weight (grams, g) on scales ($WEIGHT_{moist}$ or $WEIGHT_{humedeo}$)
4. Ensure table is level and steady, then fill large beaker fill known volume of water (to a level that will ensure the sample can be fully immersed and volume still measured), insert sample and record the volume of water plus sample. Calculate the sample volume (millilitres, ml or cubic centimeters, cm^3) by subtracting the original volume of water ($VOLUME_{moist}$ or $VOLUME_{humedeo}$) using the bottom of the meniscus and reading to the nearest 5 ml
5. Allow the sample to dry in the sun for at least 24 hours to allow it to be subject to the same sun drying process as the rest of the drill core
6. Weigh and record the “sun-dried” sample weight ($WEIGHT_{sun\text{-}dry}$)
7. Immerse the sample and record the volume of water plus sample, calculate the sample volume by subtracting the original volume of water, record the “sun-dried” sample volume ($VOLUME_{sun\text{-}dry}$)
8. Place in aluminium tray and bake in oven at 100^oC for 24 hours
9. Record weight in air ($WEIGHT_{oven\text{-}dry}$ or $WEIGHT_{seco}$)
10. Measure volume (same as point 4; $VOLUME_{oven\text{-}dry}$ or $VOLUME_{seco}$). or if the sample is vuggy (contains numerous voids), use cellophane wrap to ensure pores and voids do not fill with water
11. Calculate ‘moist’ in-situ density (g/cm^3) using $DENSITY_{moist} = WEIGHT_{moist} / VOLUME_{moist}$
12. Calculate ‘sun-dried’ density using $DENSITY_{sun\text{-}dry} = WEIGHT_{sun\text{-}dry} / VOLUME_{sun\text{-}dry}$
13. Calculate ‘oven-dried’ density using $DENSITY_{oven\text{-}dry} = WEIGHT_{oven\text{-}dry} / VOLUME_{oven\text{-}dry}$
14. Calculate swelling factor by dividing original sample volume ($VOLUME_{moist}$) by oven-dried sample volume ($VOLUME_{oven\text{-}dry}$)
15. Return the sample to the correct place in the core box.

The equipment used for the density determinations is shown in Figure 11.1.3.

a. Samples wrapped in cellophane

b. Density measurement equipment

c. Oven for drying samples

d. Sample ready for drying in oven

Figure 11.1.3: Density Measurement Equipment

11.2 Chain of Custody, Sample Preparation, and Analyses

11.2.1 Sampling Procedure Overview

Sampling was based on lithological intervals and extended 2-3 samples either side the identified lithium clay contacts. Samples ranged from a reported 0.3 – 8.68 m. The average sample length remains 1.5 m, reflecting the targeted sample length.

Sample intervals are measured by the project geologists, who mark the sample length on the core to indicate where it should be cut. The cut line along the core axis is positioned at 90º to the predominant structure to ensure that both halves of the core represent the same geological feature.

The core is then transferred to the core shed for sampling. Samples are then collected by splitting the core in half with a manual core splitter.

11.2.2 Sample Preparation

The samples are bagged and labelled with a sequential, unique sample identification number. Mr Martin Vidal (former Managing Director of Bacanora) supervised drilling of the first 12 holes on La Ventana; Daniel Calles, geologist under contract to Bacanora, supervised the core sampling during the later campaigns.

Split drill-core samples were shipped to an ALS Chemex Laboratories (“ALS Chemex Hermosillo”) sample preparation facility in Hermosillo, Mexico, for preparation. Sample preparation was conducted according to the ALS Chemex rock, drill-core and chip-sampling procedures (PREP-31). This consists of crushing the sample to minus 5.0 mm sized material, splitting off 250 g and pulverizing the split sample so that greater than 85% passed through a 75 micron aperture screen.

11.2.3 Analytical Procedures

Sample pulps were then shipped to ALS Chemex Laboratory in North Vancouver, Canada (“ALS Chemex Vancouver”), for assay and analysis. ALS Chemex is an ISO 14001-2004 certified laboratory in Canada and its preparation facility in Mexico has received ISO 17025 certification.

All core samples were digested using aqua regia methodology and analysed by inductively coupled plasma – mass spectrographic (ICP-MS: ME-MS41) method to provide data for a suite of 51 elements (Ag, Al, As, Au, B, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, Hg, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn.

11.2.3.1 Discussion on Aqua Regia

Bacanora chose to use aqua regia as the digestion substrate. An explanation of the different digestion methods utilized is described below (taken from ALS Chemex, the primary assaying laboratory).

Aqua regia is a popular acid digestion for exploration projects. It is optimised to attack sulphides and sulphosalts, and also dissolves most of the carbonates, oxides, and phosphates in the sample. However, some mineral phases in the sample are resistive to an aqua regia attack and will yield only a partial determination for those associated elements. The most resistate minerals can be largely unaffected by aqua regia. Many of the elements that are partially digested using aqua regia are much closer to total with the four acid digestion.

A four acid digestion (HF-HNO_3-$HClO_4$ digestion, followed by HCl leach) is much more rigorous than the aqua regia attack. Virtually all of the rock-forming minerals are completely destroyed with this digestion. This method provides near total elemental analyses for all but the most refractory minerals, such as spinels, some sulfates, and some rare earth element minerals. As noted above, for many elements the total content of a rock can be determined with a four-acid digestion.

It has been shown through round-robin analysis of standard materials (discussed below) that the aqua regia method has resulted in under-reporting of total lithium content. It seems that a certain proportion of lithium metal is bound within minerals which are resistant to aqua regia attack and do not therefore get entrained into solution for analysis. It should therefore be noted that all lithium grades relating to exploration samples refer to aqua regia-soluble lithium metal only, not total lithium.

11.3 Quality Assurance and Quality Control Procedures

11.3.1 Introduction

The Quality Assurance and Quality Control ("QA/QC") procedures include standards, duplicates and blank materials inserted into the sample stream blind to the laboratory.

For all samples analysed prior to 2016, the in-house standards used were not certified and did not represent the grade range typically found in the deposit. In the previous MREs, SRK recommended that new in-house standards with are created and certified. This was achieved prior to initiating the 2016 drilling campaign.

Additional confidence in the accuracy of grade determinations in the grade range of the deposit was established by independent duplicate samples submitted to an umpire laboratory (ACME Laboratory in Vancouver, Canada ("ACME Vancouver")).

11.3.2 Pre-2016 Standards

Bacanora produced three in-house lithium standards through localised bulk sampling. These were inserted into the regular sample stream to provide information on the accuracy of the laboratory results. The standards were prepared at Laboratorio Metalurgico LTM SA de CV in Hermosillo. Approximately 50 kg of bulk sample was milled to <100 µm and homogenised in a single batch in a drum mixer for 24 hours, after which 100 g sub-samples were split and sealed in plastic bags ready for insertion into sample batches.

Two different low grade standards and one higher grade standard were produced. These standards were not used concurrently; instead, each was used to completion before generation of a new standard material. Table 11.3.1 summarizes the insertion rates of the three different standard samples. Table 11.3.2 summarizes SRK's calculated means and standard deviations of the three reference samples.

Table 11.3.1: Summary of Reference Sample Insertion

Reference Sample	Total Number	Insertion Rate (%)
TT	26	1
MY-TT	56	2
High Grade Sample	77	2
Total Samples	**159**	**4**

Table 11.3.2: Summary of Reference Sample Calculated Means and Standard Deviations

Reference Sample	SRK Calculated Mean (ppm)	SRK Calculated Std Dev
TT	256	14.5
MY-TT	175	15.9
High Grade Sample	6,709	875.3

The performance of each standard is shown in Figure 11.3.1, Figure 11.3.2 and Figure 11.3.3; each shows a scattering around the calculated mean grades. Figure 11.3.3 also shows that over time there was a general trend from higher to lower assays within the range of 7,500 ppm to 6,000 ppm. SRK was satisfied at this stage the standard assays are within acceptable parameters and is not a cause for concern.

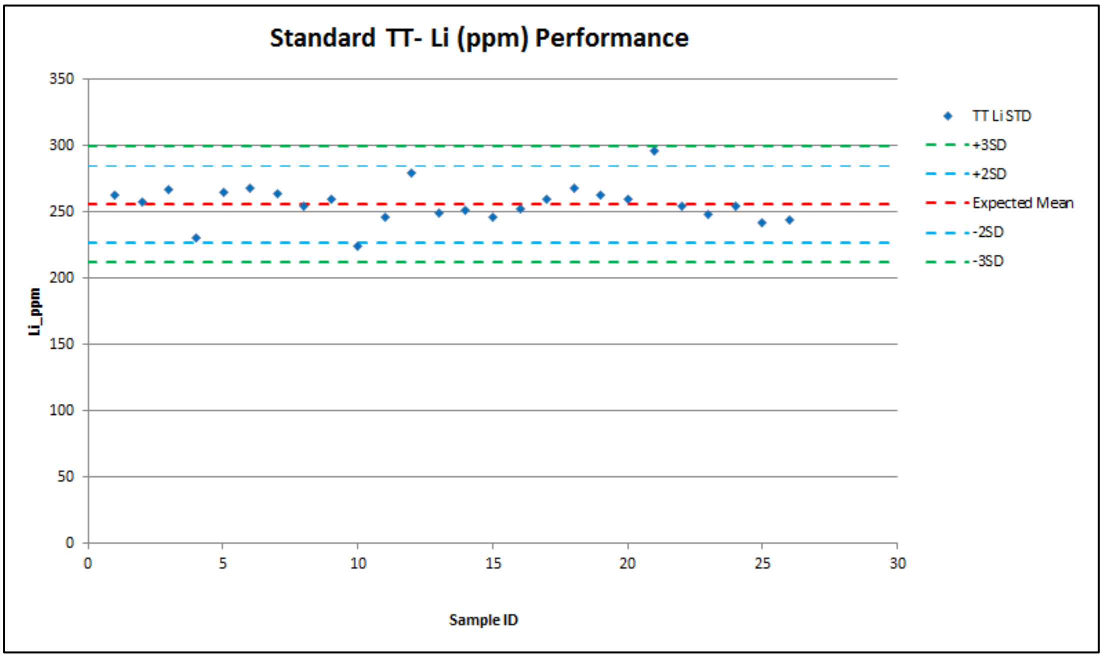


Figure 11.3.1: Low Grade Lithium Reference Standard TT

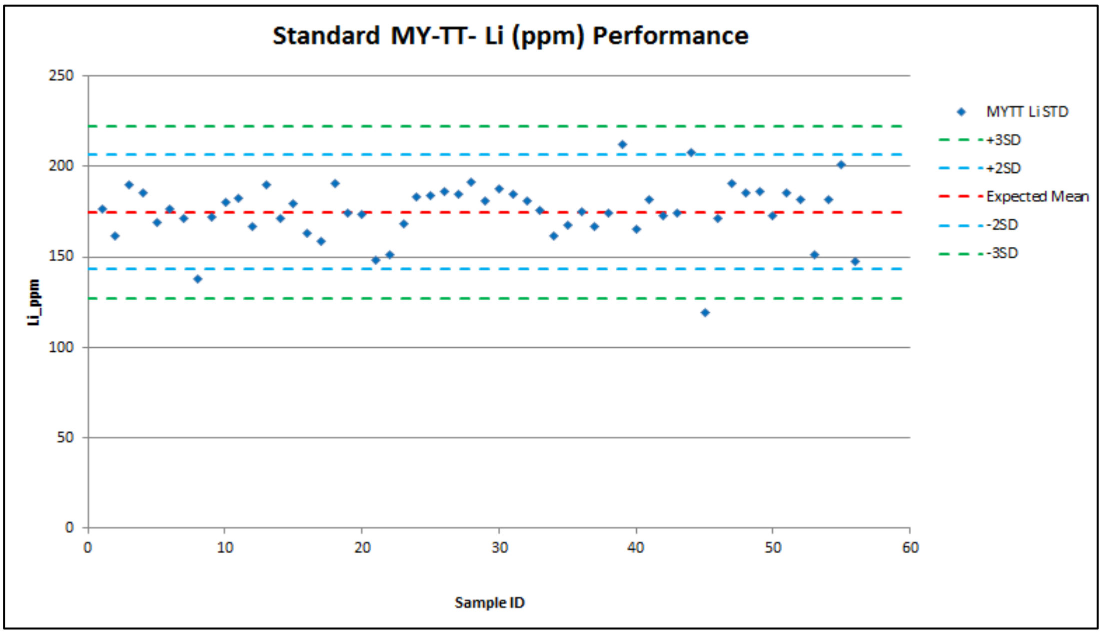


Figure 11.3.2: Low Grade Lithium Reference Standard MY-TT

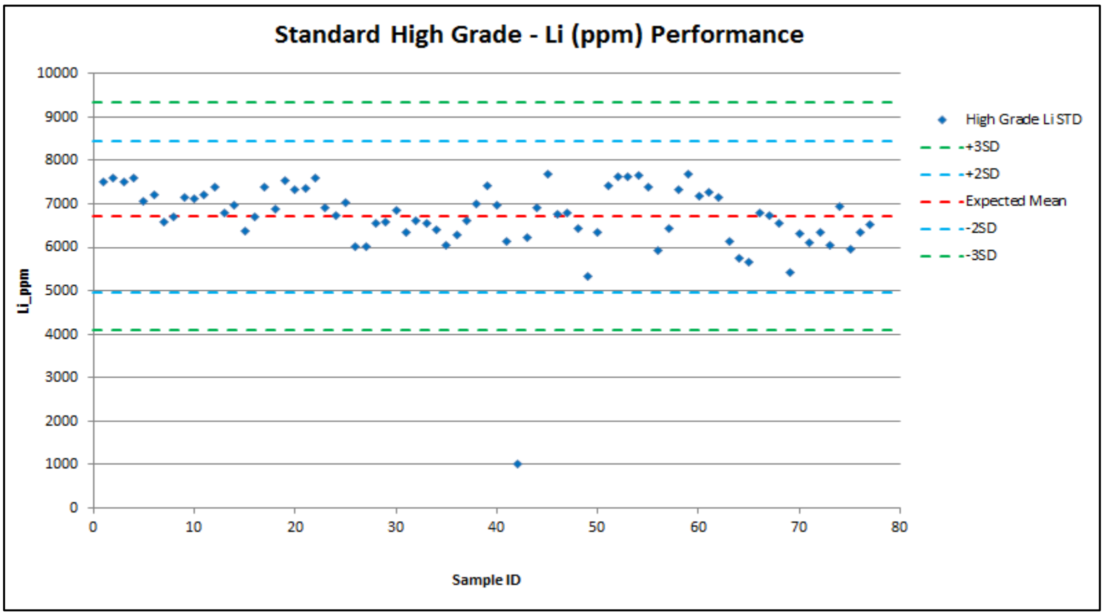


Figure 11.3.3: High Grade Lithium Reference Standard

11.3.3 2016 Standards

11.3.3.1 Description

Bacanora produced three new in-house lithium standards through localised bulk sampling in 2016. Bulk samples were collected from Bacanora's pilot plant in Hermosillo. The pilot plant processed material from Trench-4 to test a pre-concentration (cleaning) process. Two products were obtained: a pre-concentrate and a reject. A third standard was created using material previously collected for standard LUV (described above). A description of each standard is given below:

- Standard SPRET-4 (~0.5% Li) was collected from a pulverised pre-concentrate at -75 microns that is produced at the plant and that is used for roasting. A total of 15.9 kg was collected.

- Standard SREJT-4 (~0.1% Li) was collected from the rejects and then bulk pulverised to -75 microns. A total of 7.8 kg was collected.
- Standard SLUV-1 (~0.6% Li) was collected from the previously created LUV standard (prepared as above previously). The 25 g sachets were opened and all of the samples were mixed and followed the same homogenization subsample procedure. 13.6 kg was collected.

Bulk samples of 8 to 16 kg were homogenized in a single batch in a drum mixer for 24 hours at the third-party Sonora Sample Preparation Lab (ISO certified) in Hermosillo. Sub-samples of 25 g were then sealed in HDPE sachets and submitted to 4 commercial laboratories and the Company's on-site laboratory at the pilot plant in Hermosillo.

Certification

Bacanora employed independent geochemist Lynda Bloom of Analytical Solutions Ltd ("ASL") to provided certified values for the new standards. The certification report by ASL is provided in Appendix B.

The laboratories that received the standards for the round robin analysis and the methodologies used are summarized in Table 11.3.3.

Table 11.3.3: Summary of Round-Robin Laboratories and Methods Used

Laboratory	Method Description
ALS (Vancouver)	ME-MS41 (Aqua regia)
ALS (Vancouver)	ME-MS81 (Metaborate fusion)
Bureau Veritas (Vancouver)	AQ270/AQ250-X (Aqua regia)
Skyline (Tucson)	AQR (Aqua regia)
Skyline (Tucson)	TE-3 (Aqua regia)
SGS (Lakefield)	TE-5 (4 acid digest)
Bacanora pilot plant	GE_ICM (4 acid digest)

Table 11.3.4 summarizes ASL's calculated means and standard deviations of the three reference samples. The performance gate set by ASL was the relative standard deviation ("RSD" – standard deviation / mean).

Table 11.3.4: Summary of Reference Sample Calculated Means and Standard Deviations

Standard	Li (ppm)		K (%)		Ca (%)		Mg (%)		Sr (ppm)	
	Mean	RSD	Mean	RSD	Mean	RSD	Mean	RSD	Mean	RSD
SPRET-4	4764	6	2.3	6	12.0	2	2.0	6	464	4
SREJT-4	1343	6	0.8	6	4.5	5	0.8	7	270	2
SLUV-1	7089	11	2.5	9	1.8	4	1.1	7	90	4

Table 11.3.5 summarizes the insertion rates of the three different standard samples. SLUV-1 was not used due to its high-grade nature not considered appropriate for the majority of the samples sent to the laboratory.

Table 11.3.5: Summary of Reference Sample Insertion

Reference Sample	Total Number	Insertion Rate (%)
SPRET-4	31	2
SREJT-4	32	2
SLUV-1	0	0
Total Samples	**63**	**4**

11.3.3.2 Results

The assay results for the two standards submitted with the 2016 drilling are displayed in Figure 11.3.4 to Figure 11.3.7 for Li and K. The results show that the majority of assays returned grades within the performance gates set by ASL. It can be seen that the K grades display a slight negative bias overall, however, the majority of results are still within the expected ranges and so this is not deemed to be a material issue.

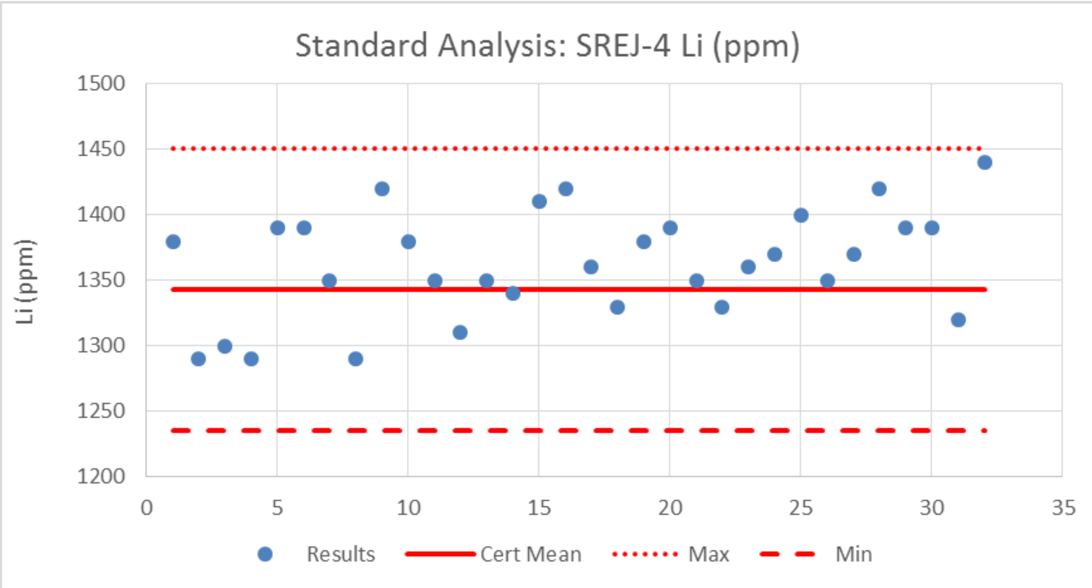


Figure 11.3.4: Standard SREJT-4 Li (ppm) Results

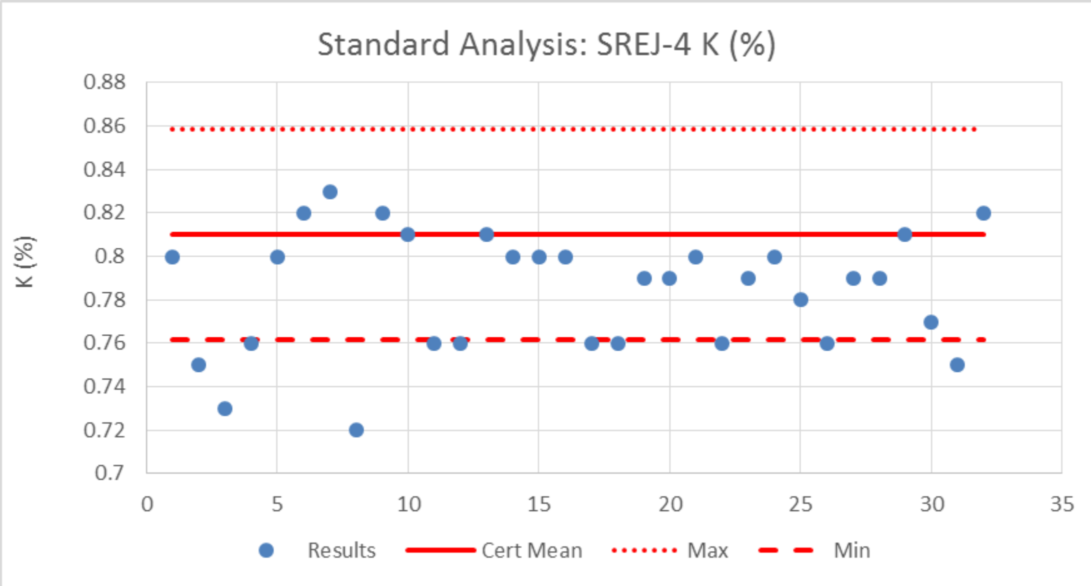


Figure 11.3.5: Standard SREJT-4 K (%) Results

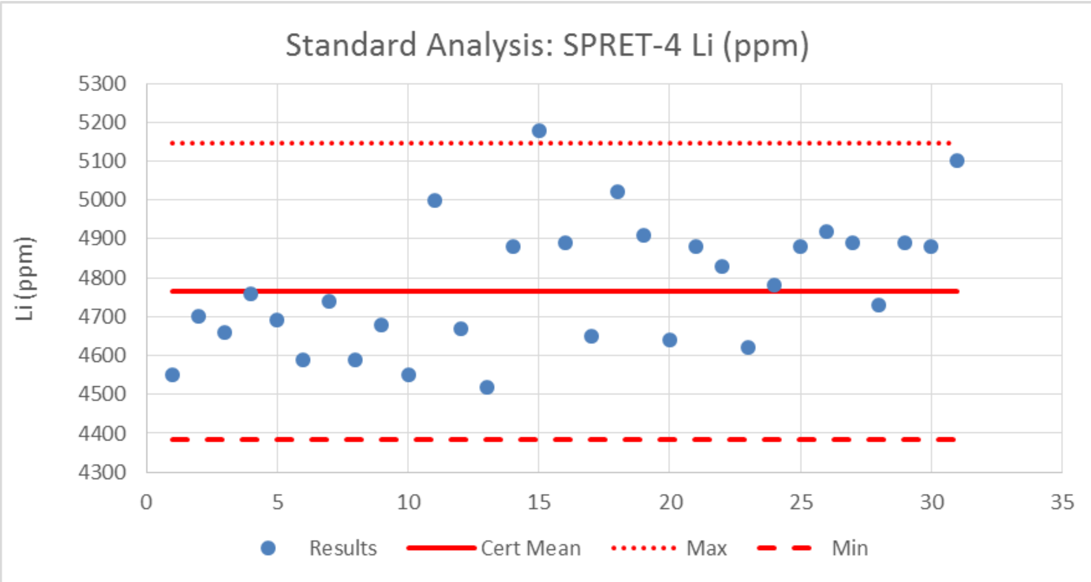


Figure 11.3.6: Standard SPRET-4 Li (ppm) Results

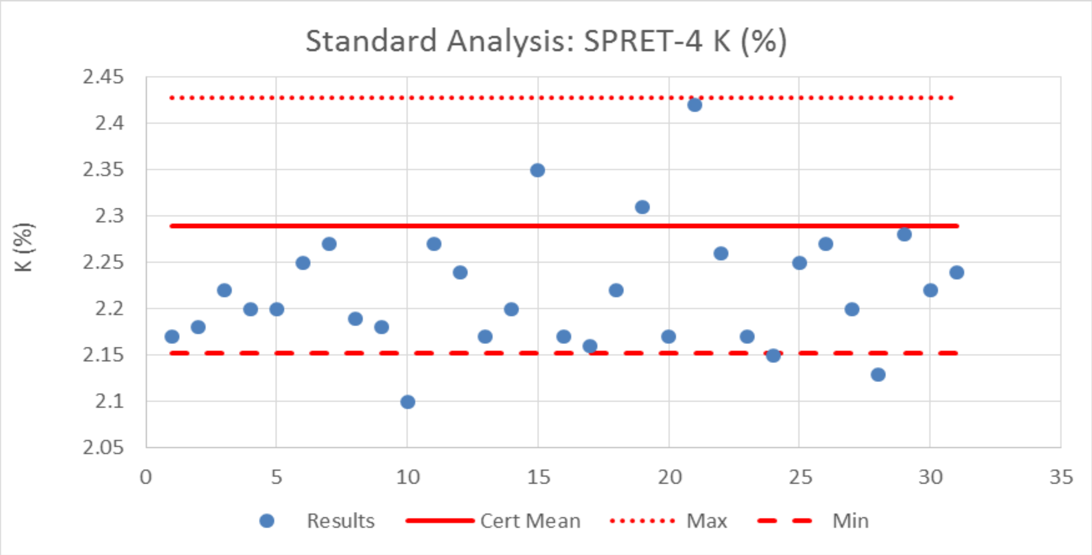


Figure 11.3.7: Standard SPRET-4 K (%) Results

11.3.4 Blanks

A total of 95 blanks were submitted as part of the QA/QC process by Bacanora during the 2014 to 2016 drilling campaigns. Prior to this, blank samples were not submitted as part of the QA/QC program. The overall performance of the blanks is considered to be acceptable. The insertion rate for blank samples in the most recent phase of drilling is approximately 1 in 20; this is considered to be in line industry best practice. Blank performance plots are presented in Figure 11.3.8.

SRK notes that almost all the samples fall above the analytical detection limit stated for lithium by ALS Chemex. The highest reported Li value for the blank follows samples with mid-level Li concentrations. There are other cases where the blank follows samples with higher concentrations and there is less carry-over. Bacanora uses a commercially available silica sand as blank material. The fine-grained blanks are not crushed and pulverized as for the samples. Therefore, sample cross-contamination would be attributed to sample solution carry-over during ICP analysis.

SRK recommends that the practice of submitting blank samples as part of the standard analytical submission sequence is maintained in further programs and that certified blank material is sourced.

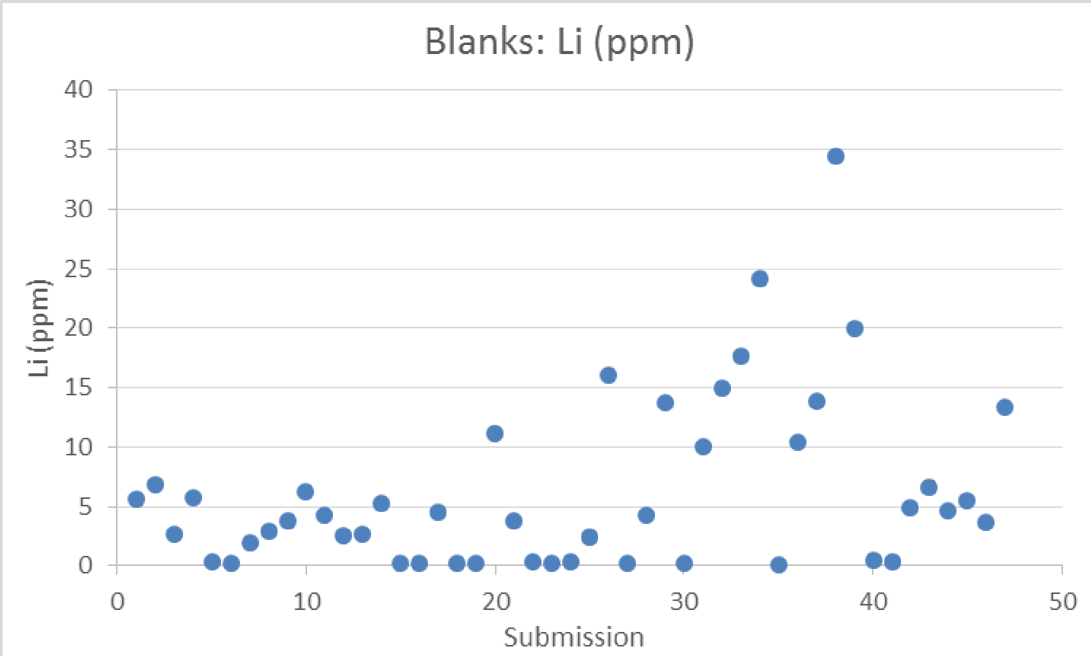


Figure 11.3.8: Blank Performance Plot

11.3.5 Duplicates

A total of 77 quarter-core duplicate samples were submitted as part of the QA/QC process by Bacanora during the 2014 to 2016 campaigns. Prior to this, duplicate samples were not submitted as part of the QA/QC program. The overall performance of the duplicates is considered to be acceptable as they show that there is little difference between the assays when one half core is compared to the other. Figure 11.3.9 and Figure 11.3.10 show a scatter plot of original versus duplicate samples for Li and K, respectively, highlighting a good correlation.

The insertion rate for duplicate samples in the most recent phase of drilling is approximately 1 in 20; this is considered to be in-line with industry best practice.

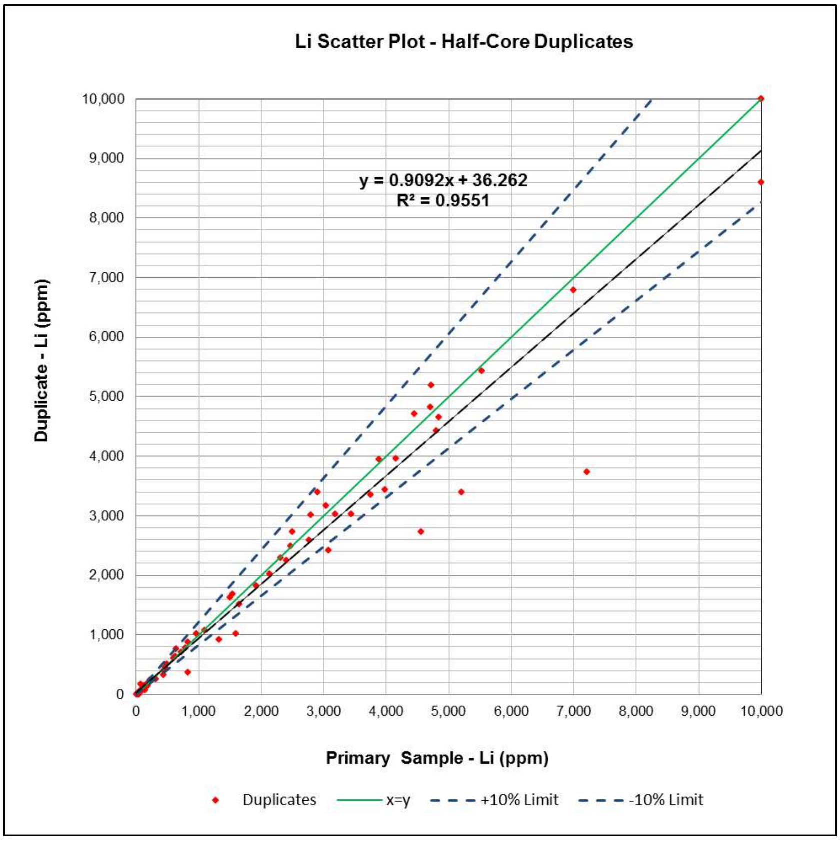


Figure 11.3.9: Duplicate Assay Comparison (Li)

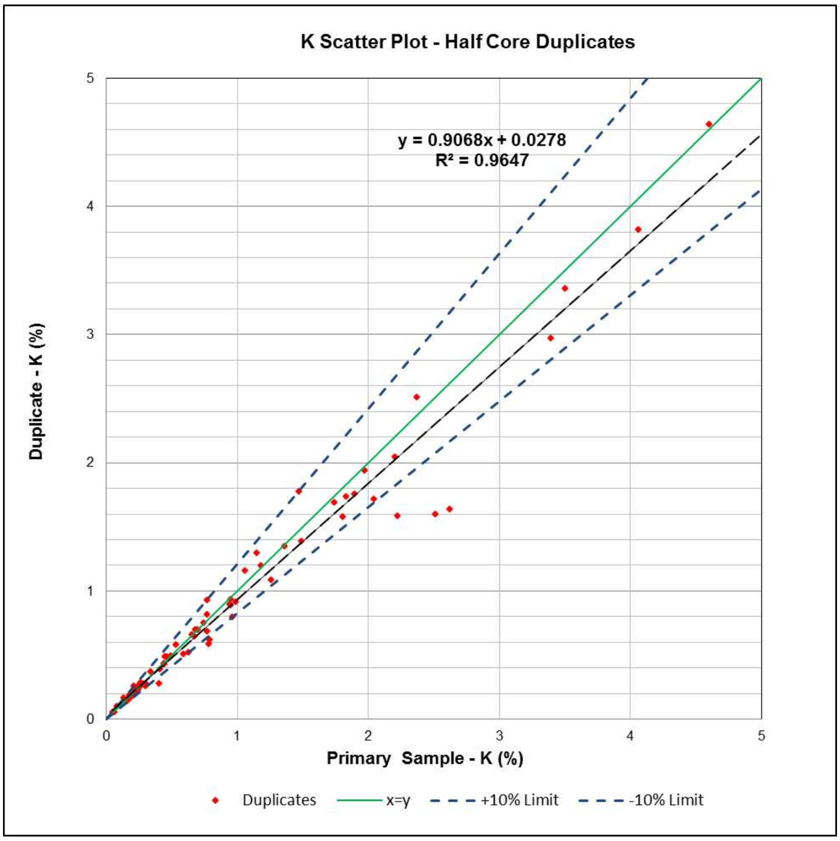


Figure 11.3.10: Duplicate Assay Comparison (K)

11.3.6 Comparative Laboratory Techniques

In addition to the ME-MS41 method, 280 samples were submitted as pulp duplicates in 2014 for further analysis using the Li-OG63 analytical method at ALS Chemex Vancouver, using a 4-acid digest with an ICP finish. Figure 11.3.11 shows an excellent correlation between the two methods.

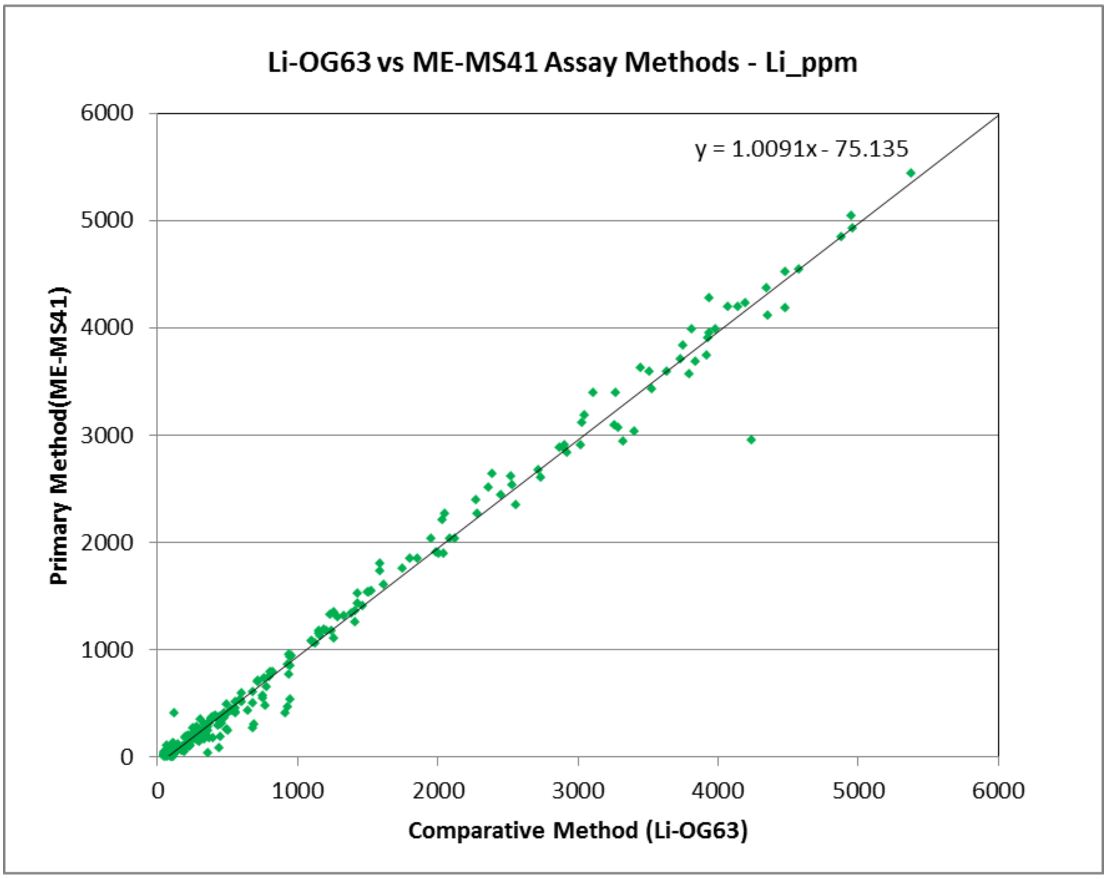


Figure 11.3.11: Duplicate Sample Method Comparison

11.3.7 Umpire Laboratory

The work undertaken by C Verley to verify the original analytical results included submitting 82 duplicate samples derived from quarter core to an umpire laboratory (ACME Vancouver) which is 2% of the total sample population. A 4-acid digest analysis was undertaken by ACME Vancouver (method MA270) with an ICP-ES/ICP-MS finish. The results in Figure 11.3.12 show that there is a good correlation between the two laboratories over the range of grades found in the deposit.

SRK recommends that in the future that at least 5% of the total sample population is routinely sent for verification at an umpire laboratory.

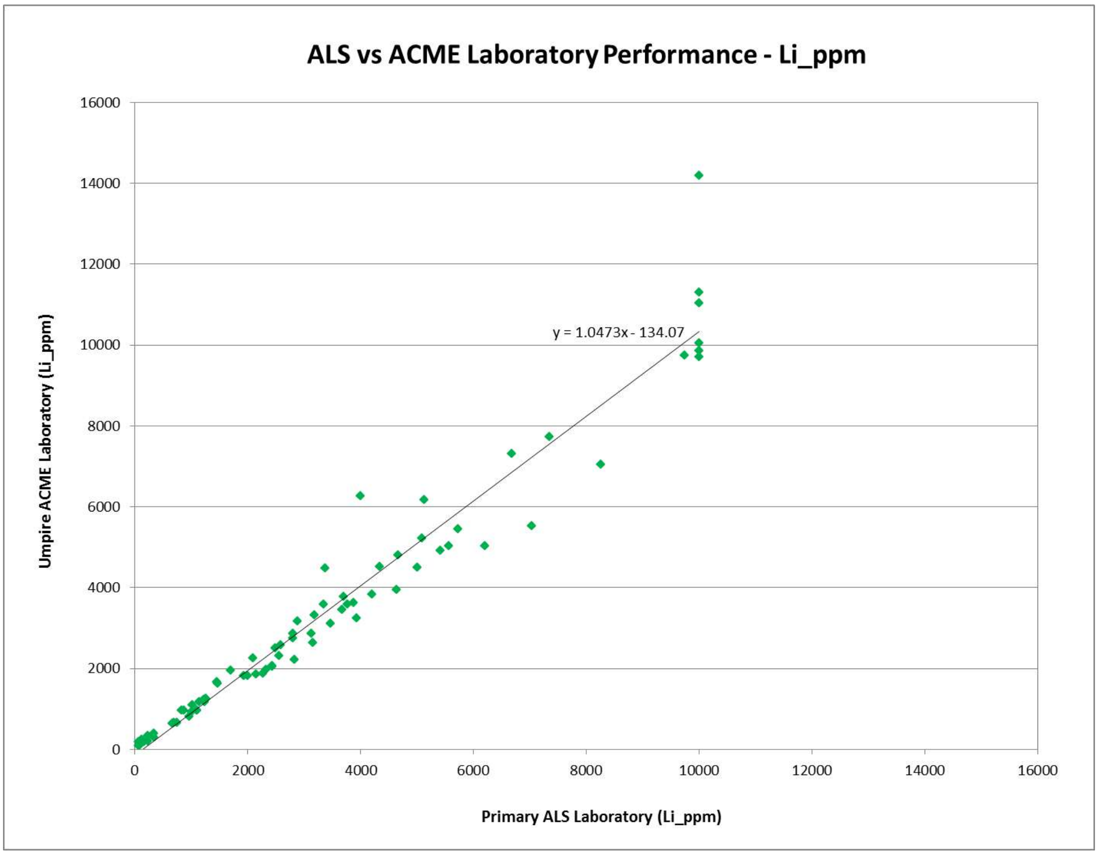


Figure 11.3.12: Duplicate Sample Laboratory Comparison

11.4 Core Recovery Analysis

Core recovery for the sampled intervals averages greater than 92%, based on core measurements undertaken by the Company. The core recovery is not believed to negatively affect the reliability of the results. SRK notes that a small drop in recovery was observed in the summer 2015 drilling, although this is also not believed to negatively affect the reliability of the results.

11.5 QA/QC Summary

SRK has reviewed the QA/QC and is confident that the quality of the data is sufficient for use a Mineral Resource estimate. SRK recommends that during future exploration drilling programs continue to submit a full suite of QA/QC samples for analysis including standards, blanks, and duplicate samples at a rate of at least 1 per 10 samples overall and increasing the submission of samples to umpire laboratories to at least 5% of the total sample population.

11.6 Testwork QA/QC Summary

11.6.1 Flowsheet development testwork and beneficiation variability testwork

Ore shipments were made by courier from the mine site directly to the laboratory (SGS Malaga). SGS Malaga is NATA and ISO-IEC 17025 certified. Trench samples were shipped in sealed plastic 200 L drums. Core samples were shipped in labelled plastic core trays. The twinned hole core samples used in the variability testwork were individually wrapped in plastic in the trays, to preserve the in-situ moisture content in the clays. Samples were unwrapped and re-wrapped for photographing and logging at SGS Malaga and remained wrapped until immediately prior to testwork commencing on the sample.

For head assay, samples were crushed to 25 mm and homogenised, so that a representative sample could be taken for assay.

- Li, B, Ni, Pb, Sr, Zn, Sc by peroxide fusion, HCl digest ICP-OES (method ICP90Q)
- Rb by peroxide fusion, HCl digest ICP-MS (method IMS90Q)
- Al, Ca, Cr, Fe, K, Si, Mg, Mn, Na, P, LOI by XRF (method XRF78S)
- C, Org C by Leco Furnace and CO3 digest (method CSA06V, CSA03V)
- S by Leco Furnace (method CSA06V)
- F by fusion digest ISE (method ISE07A)
- Cl by digest titration (method CLA74V).

Additionally, SGS has been involved with Ore Research (OREAS) in a certification program for three CRMs (OREAS 147, 148 and 149).

Table 11.6.1: Peroxide Fusion ICPOES Analysis

CRM	Certified Li %	Lab Result Li %	Absolute Z-Score
OREAS147	0.227	0.232	0.040
OREAS148	0.476	0.479	0.090
OREAS149	1.030	1.030	0.240

SGS Malaga has provided complete QA/QC reports, detailing performance of repeat assays, blanks and standards throughout the testwork program.

Greg Lane (QP) and other Ausenco personnel visited SGS Malaga on 12 September, 2017, to review sample preparation and assay procedures, QA/QC and testwork procedures. A review of the QA/QC analytical results for the standards, blanks and core duplicates did not highlight any analytical issues.

11.6.2 Locked cycle testwork and extraction variability testwork

This testwork was performed at ANSTO (Australian Nuclear Science Technology Organisation). Sample buckets were labelled and sealed. Samples were couriered overnight from SGS Malaga to ANSTO.

ANSTO participates in internal audits and external quality audits on a regular basis. The Minerals Facility is accredited with the current ISO 9001 (2015) standard.

Samples were entered into the Minerals Analytical database and given a unique job number when submitted for analysis. Samples were generally submitted either as neat or in a one in 10 dilution in nitric acid for analysis on the ICP-MS or ICP-OES. Further dilutions were carried out for both assays depending on the detection limit required.

Calibration standards were used on set up using multi element certified standards and several independent certified reference materials as check standards. ANSTO used certified standards suitable for the samples analysed which were run on a regular basis to validate our procedures / analysis. All samples were bracketed with check standards throughout the run and monitored.

Solids submitted were dried and pulverised before analysis. The XRF instrument uses elemental standards for calibrations and routine drift corrections were carried out along with certified reference standard checks.

Greg Lane (QP) and Ausenco personnel including Grant Harman, metallurgist with significant lithium metallurgical experience, visited ANSTO Minerals on 6 November, 2017, to review sample preparation and assay procedures, QA/QC, testwork flowsheets and procedures. A review of the QA/QC analytical results for the standards, blanks and core duplicates did not highlight any analytical issues.

11.6.3 Conclusion

Following site visits and a diligent review of standard procedures, the author is of the opinion that the sample preparation, analysis and QA/QC protocol used by SGS and ANSTO for the Sonora Lithium Project follow generally accepted industry standards and that the project data is of a sufficient quality.

12 DATA VERIFICATION

As QP for Mineral Resources, Martin Pittuck has verified that the data provided by the Company appears to be correct and viable for use in a MRE. This involved viewing some drillholes at the core shed to check the quality of the logging, along with cross-checking assay certificates against the database. Further statistical validation of the database was undertaken upon final receipt.

12.1 Data Received

The Company provided SRK with all requested technical information and data which SRK took in good faith as being accurate to the best of their knowledge.

SRK was provided with a package of electronic and paper based data by the Company. This included:

- Raw drillhole data sheets in Microsoft Excel format covering the drillhole collars, associated assay results and geology
- MapInfo data files relating to:
  - topography
  - licence tenure
  - geological and structural interpretation
- pdf documents relating to Resource Estimates including:
  - Initial Lithium Resource Estimate for the El Sauz and Fleur Concession, Sonora lithium project, C Verley, 11 October 2013
  - Updated and Reclassified Lithium Resources, Sonora lithium project, C Verley, 24 June 2013.

12.2 Database Validation

All available data has been validated through the production of histograms and scatterplots. All data was validated by an SRK geologist.

12.3 QA/QC

The quality control measures that have been put in place are discussed in the previous section. It is SRK's opinion that the procedures adopted have led to a reliable database and SRK is confident that the quality of the data is sufficient for use in an Indicated Mineral Resource.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The purposes of the feasibility testwork program were:

- To develop the process flow sheet to produce battery-grade lithium carbonate
- To confirm that the flow sheet is able to produce battery glade lithium with all recycle streams included and that the flowsheet is robust for the expected ore variability
- Provide engineering data for major equipment selection and sizing.

At the end of the PFS, the flowsheet consisted of beneficiation and extraction sections. Beneficiation was based on scrubbing, followed by upgrading by coarse particle rejection and flotation of mid-size producing a calcite and silica reject stream. Extraction was based on a gypsum and limestone roast followed by water leach, solution purification and evaporation and crystallisation to produce lithium carbonate.

There were modifications to the PFS flow sheet resulting from the FS testwork outcomes. These improved project economics and expected plant operability.

- A grinding circuit replaced the PFS scrubbing circuit to beneficiate the ROM ore using energy more efficiently.
- The roast feed material is briquetted to allow for ion exchange within the briquette at roasting temperature by enabling effective heat transfer through the mass of briquettes.
- The roast recipe was changed to a sodium sulfate and gypsum roast, from the PFS gypsum only roast, to reduce the operating costs associated with gypsum consumption.
- The glauber salt step was moved ahead of PLS evaporation to remove the sodium sulfate load in the PLS and allow an increase of the lithium sulfate concentration.
- A fluoride removal step is included where activated alumina removes trace fluoride.
- A split stream is included to remove caesium and rubidium from the PLS with alum precipitation.

The final feasibility flow sheet was established by design work on Trench 4 material and confirmed by variability testwork in both the beneficiation and extraction sections, as well as locked cycle testwork.

Key testwork outcomes are summarized below:

- Initial beneficiation circuit definition identified the requirement for a grinding circuit. Scrubbing was not sufficient to obtain the desired lithium upgrade.
- Beneficiation testwork on 14 composites using the selected flowsheet indicated overall lithium recovery ranged from 77.9% to 91.2%, and averaged 83.8%, for 10 of the 14 composites.
- Extraction testwork confirmed a robust roasting recipe consistently achieving >90% lithium extraction. Impurity removal successfully reduced levels of fluoride, caesium and rubidium from the PLS.

- Extraction testwork on three production composites successfully followed the process flowsheet. Impurities were removed and battery grade lithium carbonate was produced for each of the three composites. The purity of the three refined lithium carbonates, expressed as percent lithium carbonate, was 99.82% (Composite A), 99.80% (Composite B) and 99.85% (Composite C).
- Locked cycle testwork proved stable operation and a robust flow sheet, as well as the ability to produce battery grade lithium carbonate and remove key impurities. Overall lithium recovery from the extraction circuit during locked cycles was between 87% and 91%. Therefore the lithium recovery in the process is expected to be higher than the process design value of 82.8%, which accounts for other plant losses.

The testwork results served as inputs to the process design criteria, which were used to develop the mass balance.

13.2 Testwork Sample Selection and Feed Grades

Samples were derived from Trench 4 and drill core intervals. Composite samples were defined based on ore from within the mine schedule, Mine Schedule 33.

The main grades of the head samples are shown in Table 13.2.1. Refer to Figure 13.2.1 for lithology identifiers.

- Composites 14, 12 and 13, were based on samples selected from production Years 1-2, 3-5 and 6-10; and also used for extraction variability testwork
- Composites 1 and 2 were from intervals of altered tuff, LTC1 and LTC2, interbedded in the lower clay and have low Li grade
- Composite 3 is from intervals of LC2, the lower clay bottom near the basement
- Composite 4 is from intervals of lower clay between ignimbrite and altered tuff LTC1
- Composite 5 is of intervals from top LC1 Year 1
- Composite 6 is of intervals from bottom LC1 Year 2
- Composite 9, 10 and 11 are of intervals of lower clay with high Li, high Ca and low Ca grades respectively
- Composite 15 was of selected low competency material
- Compostie 16 was from dried original core parallel to composite 13 intervals from twinned holes.

Table 13.2.1: Head Samples Analyses

Sample	Li ppm	Si %	Al %	F %	Mg %	Ca %	K %
Trench 4	6,010	26.0	2.37	2.02	3.08	11.1	2.69
Composite 1	1,020	34.7	6.32	0.42	0.81	2.18	5.52
Composite 2	485	33.1	6.46	0.15	0.27	1.27	5.44
Composite 3	1,971	26.9	4.00	0.87	1.33	8.75	2.64
Composite 4	2,127	31.3	5.49	0.89	1.33	4.93	5.09
Composite 5	4,846	28.2	3.27	1.84	2.06	6.39	2.42
Composite 6	5,521	29.6	2.91	2.12	2.04	7.25	3.34
Composite 9	8,175	29.3	3.84	2.69	1.99	5.76	3.70
Composite 10	3,548	25.6	2.58	1.50	2.78	11.2	2.13
Composite 11	5,692	30.2	5.95	1.77	1.42	2.90	5.08
Composite 12	5,087	27.8	2.52	2.15	2.78	7.85	2.64
Composite 13	3,829	29.4	3.76	1.60	1.98	5.87	3.26
Composite 14	5,260	28.8	3.44	1.89	1.78	5.43	3.12
Composite 15	7,050	24.7	2.60	2.67	3.76	9.73	2.11
Composite 16	4,018	29.0	3.35	1.67	2.21	6.88	3.07

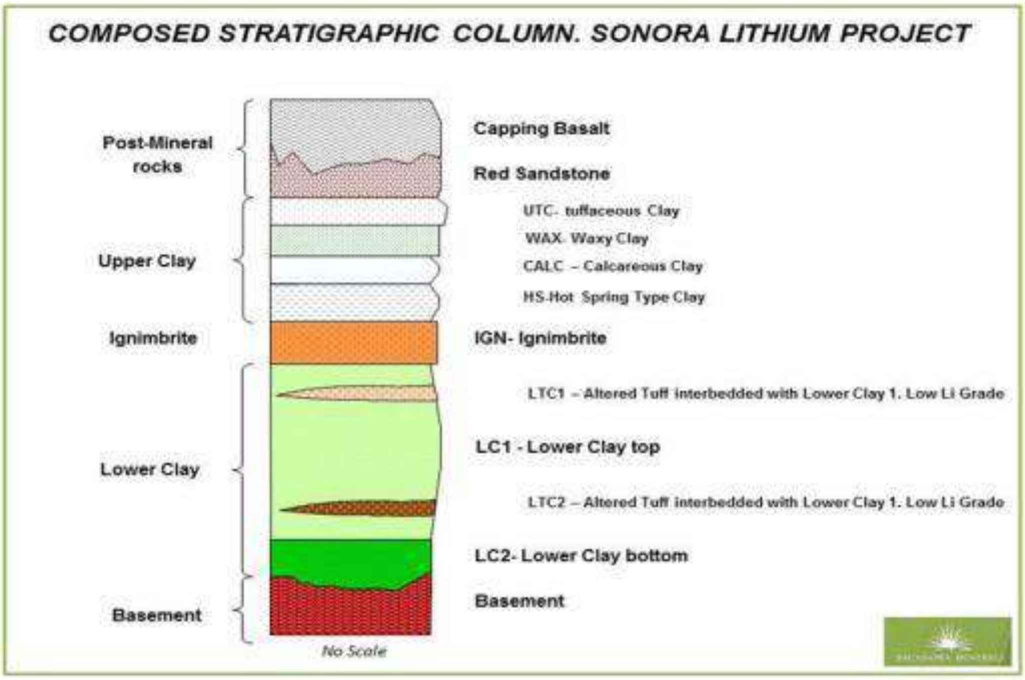


Figure 13.2.1 Composed Stratigraphic Column

13.3 Mineralogical Testwork

Mineralogical examination of the Trench 4 clay sample, stage crushed dry to a P_{80} of approximately 300 µm, was conducted with XRD, QEMSCAN, electron microscopy, EPMA, chemical analysis and LA ICP-MS.

The feed sample consists of major amounts of calcite and quartz, and minor K-feldspar and montmorillonite, with trace amounts of illite, mica, swinefordite and plagioclase.

The clays are mixtures of smectites and illite, but other clay minerals might also be present (polylithionite, hectorite). They are locally intergrown with quartz, feldspars and calcite.

Free and liberated Li-clays account for 44%, the remainder typically occur as complex middling particles (32%), and in binary associations with quartz (18%), and micas/clays (3%). Liberation increases from 28% to 44% and 50% from the +212 µm, -212/+53 µm and -53 µm fractions.

A significant portion (70%) of the Li-clays is in the -75 µm fraction with 22% of the Li-clays liberated and free in the -25 µm fraction.

With the delivery of freshly drilled core for the FS testwork variability composites were compiled for testing. The testing included XRD analysis of the head of each composite. See Table 13.3.1 for the XRD results.

Table 13.3.1: Composite Head XRD Results

Comp	Calcite $CaCO_3$	Plagioclase $(Na,Ca)(Si,Al)_4O_8$	Fluorite CaF_2	Mica/Illite $KAl_3Si_3O10(OH)_{1.8}F_{0.2}$	K-feldspar $KAlSi_3O_8$	Quartz SiO_2	Analcime $NaAl(Si_2O_6)\cdot(H_2O)$	Smectite $Na_{0.2}Ca_{0.1}Al_2Si_4O_{10}(OH)_2(H_2O)_{10}$	Amorphous
1	6.1	7.5		9.8	30.7	23.3	0.6		22.0
2	4.0				47.8	15.0	19.2		14.0
3	22.5	7.1		4.4	20.3	18.8			26.9
4	11.7	7.7		2.9	44.8	17.9			15.0
5	17.9	2.3		8.7	11.3	25.1			34.7
6	19.8	2.5		3.5	16.0	24.5			33.7
9	14.5	3.2	0.6		30.4	17.1	0.9	15.7	17.6
10	30.4	5.4		5.9	13.1	17.6	1.1		26.5
11	10.5	4.1		18.8	21.7	18.9			26.0
12	14.9	1.2	0.5	10.5	11.0	20.9		0.2	40.8
13	14.0	4.9		7.0	23.8	15.0		3.2	32.1
14	11.3	2.1		11.4	12.9	25.4			36.9
15	27.4	1.3		13.3	4.2	21.5		0.5	31.8
16	20.0	6.5		10.0	16.7	20.5	0.7		25.6

Analysis of these XRD results showed that:

- In most composites there is very little calcium that is not present as calcite
- In the worst cases of composites 12 and 14, as it happens two of the production period composites, there is at least 80% of the calcium present as calcite and perhaps more depending on the content of the amorphous fraction
- More calcite is evident than indicated by assays with factors from 0.76 to 1.45. The presence of calcium in minerals other than calcite includes amorphous content for Composites 12 and 14 and likely in plagioclase for Composites 4, 12, 13 and 14
- Silica distribution is not able to be well defined because of the high amount of amorphous material. The silica that is in the amorphous fraction is more likely to be less stable and thus more reactive than the crystalline phases.

13.4 Beneficiation Testwork

Beneficiation testwork was conducted to improve project economics by increasing lithium grade and reducing the throughput in the lithium extraction plant. PFS testwork showed 60% Li recovery was possible in 30% mass into the -20 µm fraction. FS indicated that the ore is more competent than previously understood and required substantially more power to disassociate the lithium bearing clays.

Tests were undertaken to enable design of the beneficiation circuit configuration using a Semi Autogenous Grinding ('SAG') and ball mill circuit incorporating coarse rejection and calcite flotation.

The beneficiation testwork flowsheet is depicted in Figure 13.4.1. The SAG mill circuit is represented by "SPI" in the figure as this is a laboratory technique with a dedicated mill type that characterises ore through SAG mills including product sizing. Calcite flotation was envisaged to be required when the ore contains calcite in excess of roast requirements. The calcite would then be stockpiled. At times when ore contains insufficient calcite for roasting the stockpiled calcite would be utilised.

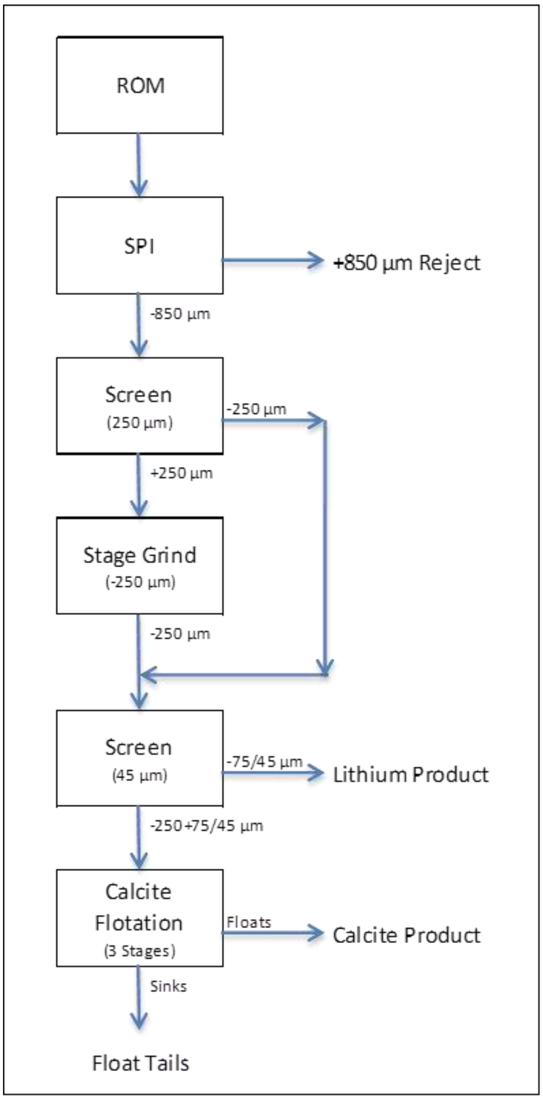


Figure 13.4.1: Testwork Beneficiation Flowsheet

As beneficiation testwork progressed in parallel with extraction testwork, it was determined that the beneficiation product needed only be 100% passing 125 µm, equivalent to 80% passing 93 µm. Subsequent work used the 125 µm screen for product and flotation feed splitting. It has been determined that calcite flotation is not needed due to ore blending resulting in calcite in ore always being less than roasting requirement.

Beneficiation variability testwork results indicated that:

- For LTC-1 and LTC-2, tuffaceous ore types, it may not be economic to process due to low head grade, low upgrading to the beneficiation product and lower recovery in leaching
- The coarse reject size from the SAG mill screen should be increased from 0.85 mm to 4 mm
- Overall Li recovery for beneficiation, roast and leach testwork ranged from 77.9 – 91.2% and averaged 83.8% for 10 of the 14 composites.

13.5 Extraction Testwork

13.5.1 Roasting

To maximize project economics, it was critical to reduce lithium losses in the process. The point of greatest lithium loss in the circuit was the unextracted lithium remaining in the leach residue, which was sent to tailings. Therefore in the FS testwork, it was imperative to optimise roast and leach conditions so that the maximum amount of lithium was extracted in this step. Initially, roasting testwork was performed by SGS Malaga using the PFS gypsum roast procedure and confirmed a feed mixture of ore, gypsum and limestone at a maximum feed size of P_{80} 75 µm. Thermogravimetric analysis (TGA) and differential scanning calorimetry (DSC) testwork confirmed that an exothermic reaction occurs, beginning at 1005°C, and that in the plant, a lower temperature of 950°C is therefore required to avoid vitrification of the material. These tests achieved 86.9% lithium extraction, which was consistent with the PFS work.

Roast testwork was subsequently carried out on beneficiated fines material obtained from the scrubbing testwork (-20 µm fraction). This showed that the reagent consumption increased relative to the contained lithium. To achieve the same lithium recovery a significant increase in gypsum consumption occurred. To reduce operating expenses, additional work was done to decrease overall reagent addition.

The roast recipe was modified to substitute part of the gypsum with sodium sulfate. XRD testwork performed at CSIRO in Perth had indicated that sodium sulfate is a key ingredient in the transfer of sulfate ions, so the gypsum in the roast recipe was partially substituted with this other sulfate source. Preliminary tests were successful and the sodium sulfate roast increased lithium recovery from 89% to 93% and reduced overall reagent consumption/costs. The roast recipe was confirmed by test GR136B using beneficiated ore : gypsum : limestone : sodium sulfate at target temperature for 1 hour. The leach was performed at ambient temperature at 50% solids. Additionally, testwork showed that the roast and leach feed could be coarsened to a P_{100} 125 µm and leach feed to P_{100} 250 µm, without reducing lithium extraction, which allows for more efficient filtration in the concentrate filters and leach residue filters.

Due to the tight limits on roast temperature to ensure conversion to lithium sulfate but avoid vitrification, the kiln used to roast this material industrially must have good temperature control to achieve the required bed temperature for 1 hour but not over-cook as this was seen to have a detrimental impact on lithium extraction.

The plant is designed to roast material as briquettes. The bulk of the FS roasting testwork was done in crucibles in a muffle furnace. For additional roasting and heat transfer testwork, briquettes were formed by a briquetting machine at Bacanora's demonstration plant and tested. The briquettes were prepared according to the sodium sulfate roast recipe and roasted at temperature for 1 hour, resulting in 90.6% lithium extraction. Performance of the briquetted material is therefore consistent with the crucible testwork.

Kilns typically used for high throughput mineral processing are generally not suitable to process briquetted material; modifications to equipment are required. Further engineering work is required to adapt conventional kilns to roast briquettes.

13.5.2 Leach residue thickening and filtering

Thickening and filtration testwork was performed on the leach residue of the three production composites at Outotec in Perth, WA. The optimum dilution, using Outotec's optimum dilution test method for efficient flocculation with Outotec's Vane feedwell, was determined to be approximately 15% w/w solids. Dynamic test flux rates as high as 0.8 t/h/m^2 and down to 0.35 t/h/m^2 produced underflows in the range 60.5 to 55.3% solids with yield stresses ranging 19 to 49 Pa. A high rate thickener was selected for the process.

Subsequent vacuum filtration tests were performed using feed at 59% solids. Testing was performed at 0.7 bar. The Sample A test with hot filtration and hot washing at 1.23 m^3/t solids required 15 seconds of drying time to reach 24% moisture. This corresponded to an overall filtration rate of 1,100 kg/h/m^2 and achieved a total was efficiency of 99.7%. These results enabled the number of filters to be reduced from two to one.

13.5.3 Impurity Removal

The ore contains fluoride which is leached in small amounts into the PLS. As this element will carry through to the final product and may impact the saleability of the lithium carbonate, it was decided to remove the contaminant. Testwork was done at SGS Malaga and ALS Balcatta. Fluoride was successfully removed to <1 ppm which resulted in the lithium carbonate containing less than 50 ppm fluoride. Further kinetic testwork is advisable to optimise sizing of the activated alumina columns.

Caesium and rubidium are two other elements that require removal from the PLS so that they do not build up in the circuit. Testwork showed that 40 g/L alum addition could remove 97% of caesium and 50% of rubidium.

13.5.4 Evaporation and Crystallisation

Evaporation and crystallisation testwork were performed by three different vendors to determine the phase equilibria and chemistry to design the equipment for the glauber's salt removal, PLS evaporator and glaserite crystalliser.

PLS and barren liquor solutions were provided from the demonstration plant in Hermosillo. The solution supplied was spiked with chemicals to match the composition in the mass balance model which included recycle streams.

The glauber's salt crystalliser is based on flash crystallisation, as it has lower capital and operating cost than the surface crystalliser and also reduces scale formation. Flash crystallisation does however impose a lower crystallisation temperature limit of 10°C. When limiting cooling to 10°C approximately 52% of the sodium sulfate is removed.

The role of the PLS evaporator is to increase the lithium concentration in solution without initiating any crystallisation. Testwork confirmed that a lithium tenor of 8,400 mg/L was acceptable and would minimize scaling of the equipment.

The purpose of the glaserite crystalliser is to remove potassium from the circuit and produce a saleable SOP product. Glaserite was successfully produced by the crystalliser vendors, as well as during the LCT testwork.

The glaserite system recovers approximately 50 to 60% of the potassium value, as glaserite, by a combination of evaporation and cooling. The glaserite was successfully decomposed which involved the dissolving of the glaserite in limited water and the simultaneous crystallising out of SOP from solution. Assay of the SOP showed that it contained minimal impurities.

13.5.5 Extraction variability testwork

The extraction variability testwork program performed at ANSTO, was designed to test the flowsheet against variations in composition and the impact on the production of battery grade lithium carbonate. Three production composites were prepared as per the beneficiation production composites and are identical to beneficiation composites 12, 13 and 14:

- Composite A (14): Representative of Years 1–2
- Composite B (12): Representative of Years 3–5
- Composite C (13): Representative of Years 6–10.

No recycle streams or by-product streams were to be included in this program. Battery grade lithium carbonate was successfully produced for all three composites. A summary of theANSTO results are presented below. The complete testwork program is detailed in the ANSTO report “Sonora Lithium Project, Mexico – Production Composite Extration Test Work” by C.S. Griffith.

13.5.5.1 Roasting

Initially, several variations on the roast recipe were tested for each composite. However, the standard recipe achieved > 90% lithium extraction for all the composites and was selected for use in the bulk roast and PLS testwork. The maximum extraction of lithium achievable from Composites A, B and C are 92%, 93% and 90%, respectively. This indicates that lithium is either hosted in mineral phases not responsive to alkaline roasting, or that roasting ‘locks up’ a small proportion of lithium or that homogeneous reagent blending was not achieved.

The lithium extraction did not vary significantly within the reagent ratios tested (between 89.9% and 93.5%). This demonstrates that the alkaline roasting approach is robust with respect to the extraction of lithium. However, there is an opportunity to manipulate the ratio of reagents to limit the extraction of other elements, such as boron and rubidium, without affecting the extraction of lithium.

Mixing and roasting of the 10 kg batches confirmed scalability and reproducibility of extraction results for all elements. Good mixing was observed to be critical and in plant operation and is expected that the pug mixer included in design will be suitable. In leach, ease of dispersion of ~50 wt% solids leach slurry and excellent filterability of the resultant leach residue was observed.

13.5.5.2 Glauber Salt Crystallisation

Glauber salt cyrstallisation demonstrated the rejection of sodium sulfate (as precipitation of glauber salt) for all three composites. Between 38–59% sodium removal was observed for the three composite liquors. There was little to no discernible difference in the behaviour between the composites.

13.5.5.3 Evaporation

Overall, the sulfate loading in solution limits the extent of the evaporation. Therefore, the lithium concentration in the PLS. Composite C had higher potassium and sodium concentrations, which limited the lithium concentration in the PLS to 8 g/L, unlike the Composite A and B liquors which achieved 9 to 9.5 g/L. The design allows for a process recycle to lower concentrated sodium solutions by recirculating the material to glauber salt crystallization.

13.5.5.4 Fluoride Ion Exchange

Alum-conditioned activated alumina displays sufficient selectivity for F at pH 6 to remove 150–170 mg/L F from solution to achieve ≤1 mg/L F in the IX barrens. The selectivity of activated alumina for F is comparable for each of the composites. However, the capacity for F under these conditions is not exceptional as ~5 L/Lwsr (litre/litre wet settled resin) is required to achieve this level of removal;

Although activated alumina is able to remove F to ≤1 mg/L F, the relatively low L/Lwsr required to achieve this suggests that an enlarged IX circuit (principally resin (alumina) inventory) may be required, and specific analysis of this is warranted. The current activated laumina column design is intentionally conservative to accommodate this.

13.5.5.5 Calcium (Al) Ion Exchange

Due to elevated K (in particular) and Li tenors, the selectivity of aminophosphonic acid resins for Ca and Mg are severely impacted and are only capable of removing trace amounts of these elements. All of the commercially available aminophosphonic acid resins from Purolite, Lanxess, Dow and Mitsubishi will display this type of behaviour.

Due to the absence of Al in any of the liquors, no conclusion can be made about the resins suitability for Al ‘bleeding’ from the upstream AA IX circuit.

These results along with the locked cycle results justified the removal of this ion exchange step from the process flowsheet.

13.5.5.6 Boron Ion Exchange

N-methyl glucamine resin displays excellent selectivity for B at pH 11 to remove >97% B from solution. Due to the relatively low maximum capacity (~3 g/L wet settled resin) of the N-methyl glucamine resin will require an enlarged IX circuit (principally resin inventory) to remove the majority of boron from each of the composite liquors.

Further testwork is recommended:

- The concentration of boron tolerable in the IX barrens needs to be established in order to accurately size the boron IX circuit. Based on the test work performed, the impact of greater than ~30 mg/L B cannot be judged.
- Further consideration should be given to additional B rejection in the roast – water leach steps, especially for Composite A and B, to reduce the reliance on boron IX. This is an opportunity to reduce operating costs.

13.5.5.7 Primary Lithium Carbonate Precipitation

Primary lithium carbonate precipitation resulted in lithium carbonate purity of Composite A (98.6 %) Composite B (97.7%) and Composite C (99.41%). One explanation for the higher purity of Composite C could be the reduced lithium tenor, and consequently reduced potassium, sodium and sulfate in the Composite C feed liquor. With the more dilute conditions, supersaturation of lithium carbonate and the propensity to rapidly precipitate, occluding potassium and sodium, would have been reduced. Seeding was not performed and is suggested to improve purity.

Such behaviour suggests that a lower stage efficiency (lower lithium tenor in the PLS from ~9.5 g/L to 7.5 g/L) may warrant an economic trade-off study prior to detailed design.

13.5.5.8 Refined Lithium Carbonate

Refining of primary lithium carbonate via bicarbonation, ion exchange and lithium carbonate re-crystallisation is a suitable approach to reject a large proportion of the major (potassium, sodium and sulfur) and minor (calcium, magnesium and iron) present in the primary lithium carbonate. The purity of the three refined lithium carbonates, expressed as percent lithium carbonate, was 99.82 (Composite A), 99.80% (Composite B) and 99.85% (Composite C). All three composite produced battery grade lithium carbonate at >99.8% lithium carbonate.

13.5.6 Locked Cycle Testwork

Locked cycle testwork was undertaken at ANSTO in NSW. Trench 4 ore was used for the LCT due to similarity to the first years' ore for plant production and the depth of testwork previously performed on this material. This material was beneficiated according to the flowsheet. A summary of the ANSTO results are presented below. Complete testwork methodology and results are provided in the ANSTO report "Sonora Lithium Project, Mexico – Lock Cycle Test Work" by P. Freeman and S. Burling.

The testwork program was designed to test the impact of recycle streams on elemental concentrations in the flowsheet. Six locked cycles were completed for the main section of the flowsheet comprising the roast through to glaserite crystallisation. The by-products section of the flowsheet, the Rb and Cs removal and bicarbonation were only conducted in selected cycles.

In each cycle all the unit operations from the roast through to producing a lithium carbonate precipitate were completed. These unit operations are shown in the yellow coloured cell in the flowsheet shown in Figure 13.5.2. The white coloured unit operations were decoupled, which means that only they were only conducted a limited number of times in order to simplify the overall LCT program.

The locked cycle program was continued until the Glaserite Crystalliser Centrifuge centrate recycle back to the feed of glauber's salt [Stream 66] had stabilised. The comparison between the composition of [Stream 66] for the successive locked cycles is shown in Figure 13.5.1.

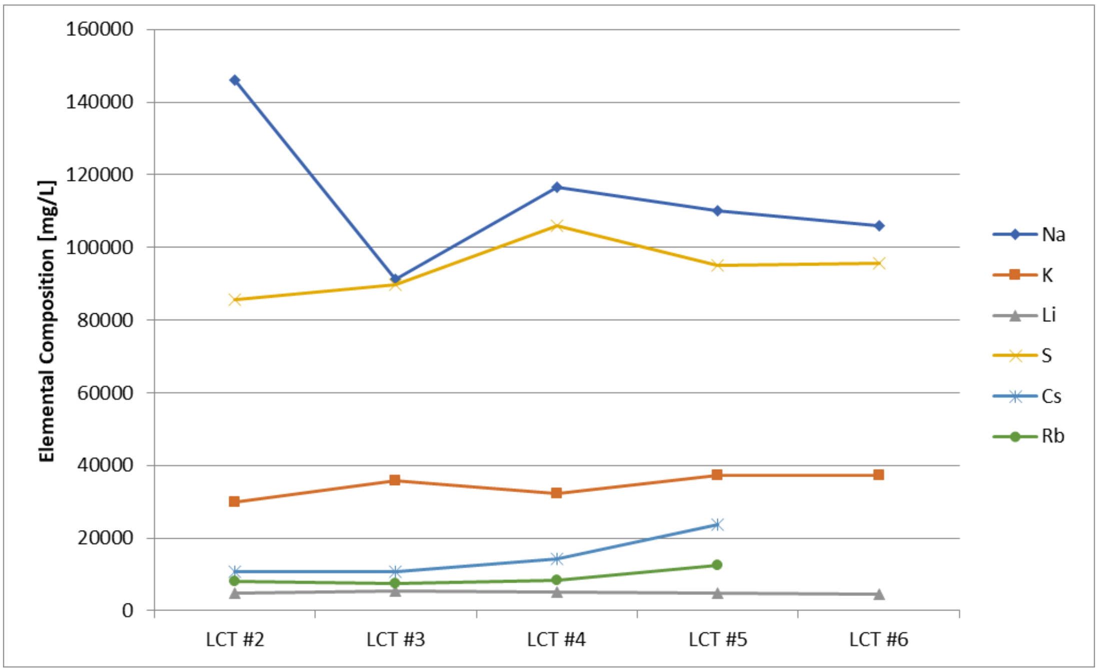


Figure 13.5.1: Comparison Between the Successive Compositions of Stream 66

Figure 13.5.1 shows that the Glauber salt, Lithium Carbonate Precipitation and Glaserite Crystallisation steps had all stabilised by LCT6 as shown by the lithium, sodium and potassium curves all becoming flat.

The curves for Cs, Rb and Cl (not shown) all are trending up which is due the build up of these elements in the overall system. In the commercial plant the Cs and Rb would be removed in the Cs and Rb removal step and the chloride level controlled by a small bleed stream.

Based on the data in Figure 13.5.1, the LCT program was terminated once the results for LCT6 were available and showed that the process had stabilized.

The key outcomes for the LCT program were:

- Confirmation of the impact of key recycle streams on the technical viability of the process
- Production of battery grade lithium carbonate
- Confirmation that all by-products can be produced and an indication of the quality of the by-products
- Confirmation of the reagent additions/ usage
- Confirmation of the overall lithium recovery of the process.

All these outcomes were successfully achieved and are detailed individually below.

13.5.6.1 Impact of Major Recycle Streams

No problems were incurred with the introduction of recycles in the process and the stream compositions were similar to those predicted by the SysCAD process model. Importantly there was no indication that the liquors were unstable and/or that unexpected double salts were being formed. There was no indication that lithium was being lost above design levels with the by-products or the residues.

As stated above the composition of stream 66 stabilised as shown in Figure 13.5.1.

13.5.6.2 Lithium Carbonate Production

Lithium carbonate is produced by reacting the lithium sulfate stream with sodium carbonate. This is a crude lithium carbonate and still contains levels of various impurities which are higher than most lithium carbonate specifications. For this reason the crude lithium carbonate must be bicarbonated, which involves dissolution and recrystallisation, to produce a refined battery grade quality lithium carbonate.

In Table 13.5.1 the compositions of the refined lithium carbonate produced in the various locked cycles are compared with the Chinese standard for battery grade lithium carbonate. This shows that in all cases the bicarbonate lithium carbonate met the specification except for slightly exceeding the potassium level in most cases. The slightly elevated potassium level is not expected to be a problem due to the low sodium level in the lithium carbonate product.

Further optimisation testwork might be able to further reduce the potassium level, if this is required. Overall the lithium carbonate produced in test work is considered to be battery grade based on the comparison between the lithium carbonate produced by ANSTO, the Chinese specification and the other manufacture’s specifications.

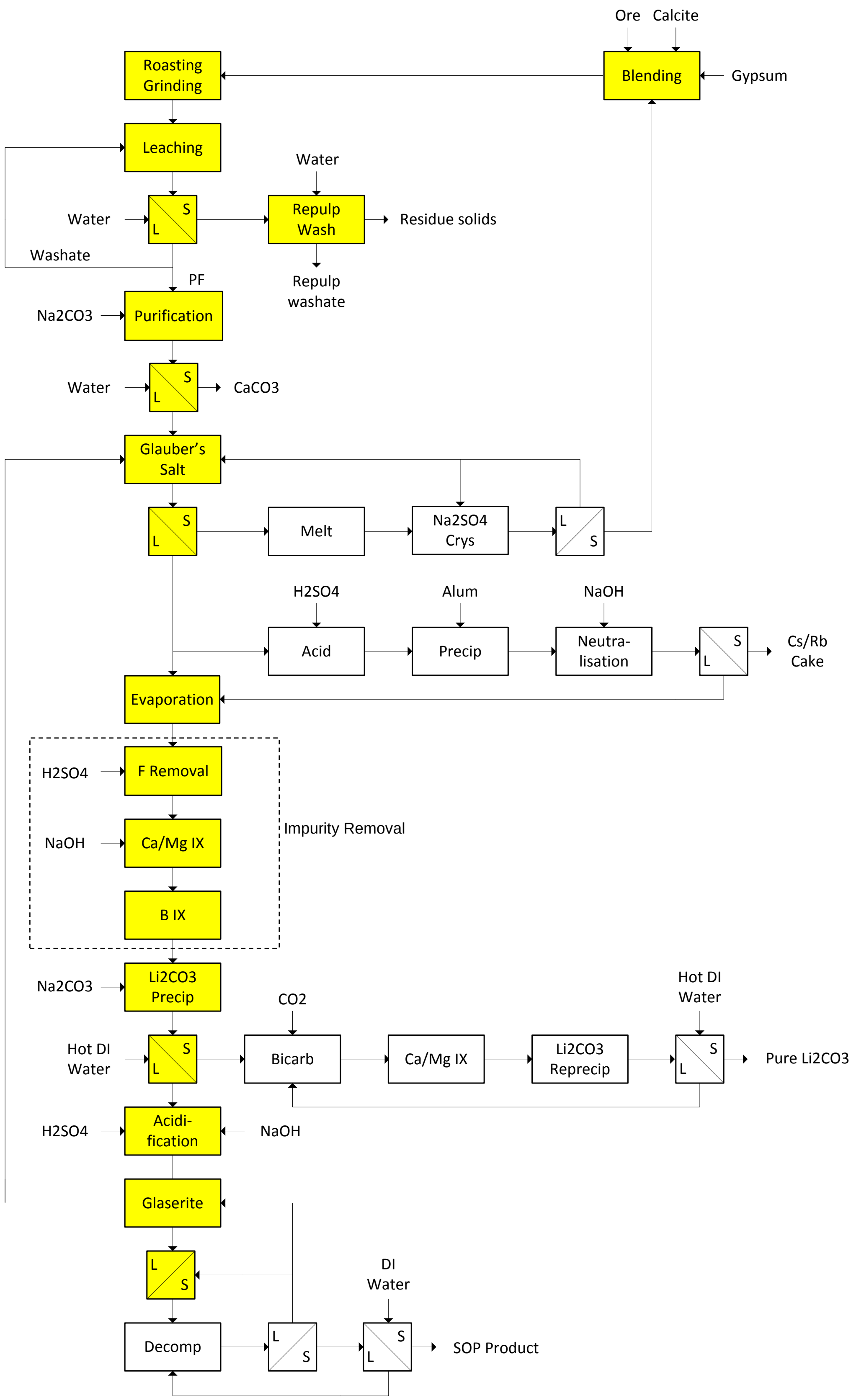


Figure 13.5.2: Simplified Bacanora Flowsheet

Table 13.5.1: Elemental Analyses for Purifies Lithium Carbonate Precipitates (a and b Indicate Duplicate Assays)

Element	Unit	Chinese Battery Grade YS/T582-(2006) specification	Cycle 2 a	C Cycle 2 b	C Cycle 4	C Cycle 6 a	C Cycle 6 b
Li_2CO_3	%	>99.5	>99.94	>99.93	>99.93	>99.95	>99.94
Moisture	%	<0.35	<0.1	<0.1	<0.1	<0.1	<0.1
Cl	ppm	<150	<24	29	<25	55	97
F	ppm	-	<24	<24	<25	<25	<35
Na	ppm	<650	215	217	191	128	132
Ca	ppm	<160	8	10	8	5	9
Mg	ppm	<70	2	2	<2	<2	<3
S	ppm	<166	16	15	25	20	23
Fe	ppm	<10	<2.4	<2.4	<4.0	5.0	5.9
B	ppm	-	<2.4	<2.4	<2.5	<2.5	<3.5
K	ppm	<10	14	22	37	9	16
Pb	ppm	<10	<0.2	<0.2	<0.2	0.3	<0.3
Ni	ppm	<30	<0.4	<0.4	<0.5	<5.0	<6.9
Al	ppm	<50	<2.7	<4.2	<2.5	6.0	8.0
Cr	ppm	-	<2.4	<2.4	<2.5	<2.5	<3.5
Mn	ppm	<10	<0.2	<0.2	<0.3	<2.5	<3.5
Mo	ppm	-	<0.2	<0.2	<0.2	<0.2	<0.3
As	ppm	-	<2	<2	<2.5	<2.5	<3.5
Si	ppm	<50	11	11	<11.2	<2.5	<3.5
Cs	ppm	-	<0.2	<0.2	0.3	<2.5	<3.5
Cu	ppm	<10	<0.4	<0.4	0.8	<5.0	<6.9
Rb	ppm	-	<0.2	<0.2	<0.2	<0.2	<0.3
Sr	ppm	-	<0.2	<0.2	<0.2	<0.2	<0.3
Zn	ppm	<10	<0.2	<0.2	13	<5.0	<6.9

13.5.6.3 Reagent Consumptions

The reagent consumptions were not optimised. It is likely that optimisation test work would produce lower reagent consumptions in several cases and this may be an opportunity to reduce operating costs.

The sodium hydroxide consumption in Cs and Rb removal test work was not recorded. However, the actual reagent consumption in the test would not reflect actual plant conditions as there should have been a filtration step before the addition of sodium hydroxide, which would result in a significant reduction in the sodium hydroxide consumption. It is likely that required sodium hydroxide would be no more than 1 g/L of feed solution.

The assumed reagent usage in the FS process design criteria (“PDC”) is supported by the testwork results. The Na_2CO_3 feed to Purification 1 was higher in the LCT, but this was due to the Ca in the feed solution being close to twice that predicted in the SysCAD model (model prediction 400 mg/L, actual 690 mg/L). Also, excess Na_2CO_3 was added to ensure maximum Ca removal.

13.5.6.4 Lithium Recovery

Figure 13.5.3 shows the potential lithium recovery from Roast feed to Lithium Carbonate Precipitation achieved in the LCT, compared with the process design criteria extraction plant lithium recovery.

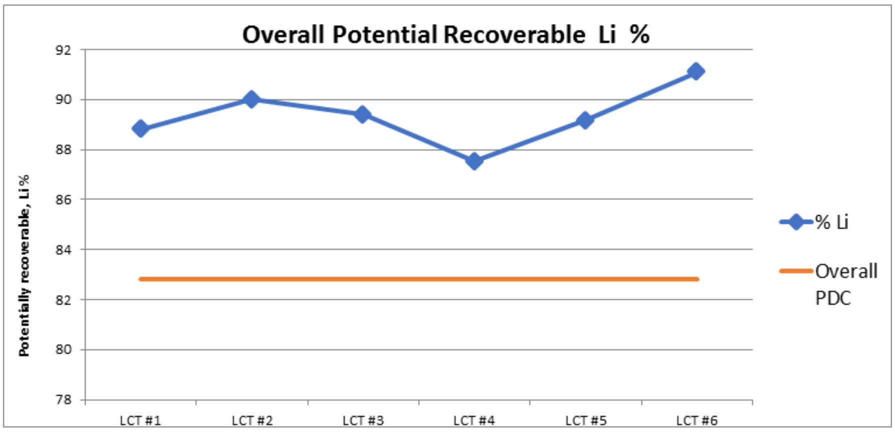


Figure 13.5.3: Potential Lithium Recovery to Lithium Carbonate

The sources of lithium losses used to calculate the lithium recovery were as follows:

1. Solid lithium after Water Leach
2. Co-precipitated lithium in Purification 1 (calcium carbonate precipitate)
3. Lithium co-precipitated in Glaserite.

The values quoted in the Figure 13.5.3 refer to the potential lithium that can be recovered, which assumes, for example, complete (or close to) recovery of soluble lithium from filter cakes. This may not be the case in practice as filter cake washing is not perfect and the amount of wash water available will be limited by the circuit water balance.

Even if there were lithium losses, such as soluble lithium remaining in the water leach filter cake or lithium extracted onto resin and not recovered, which would reduce the overall lithium recovered, the actual lithium recovered is highly likely to be significantly higher than the 82.8% extraction plant process design.

13.5.6.5 Roasting

The lithium extraction was reasonably consistent and varied from 87.9 to 91.1%, averaging 89.7% (calculated from feed and residue solid masses and analyses).

13.5.6.6 Purificication

Due to the low concentration of magnesium in solution <5 mg/L, there was no need to add sodium hydroxide.

The addition of Na_2CO_3 effectively reduced the calcium from ~700 to ~30 mg/L at 150% stoichiometric addition, the majority of the strontium also precipitated.

13.5.6.7 Glauber's Salt

Glauber's salt readily formed on rapid cooling of the solution to ≤10°C. However, on controlled cooling to just above 10°C, no crystals formed, and when the solution was then cooled to ≤10°C the glauber's salt formed contained significantly higher concentrations of other alkali metal sulfates;

The glauber's salt process averaged 70% removal of sodium from solution, with a maximum of ~89% in cycle 3.

13.5.6.8 Evaporation

The lithium after evaporation varied between 9.2 and 11 g/L as Li. It is unlikely that higher lithium concentrations can be achieved by evaporation due to the concentration of other ions present.

The volume of water evaporated varied between 27 and 41%.

13.5.6.9 Impurity Removal – F, Ca and B Removal

Both fluorine and boron were removed to below the detection limits of 1 mg/L.

Calcium was removed to between 1 and 5 mg/L.

Silica was removed to below the detection limit of 5 mg/L.

No aluminium was detected in solution after fluoride removal.

There did appear to be a reduction in lithium concentration during the purification steps, which may be because some lithium co-loaded onto the ion exchange resins.

13.5.6.10 Lithium Carbonate Precipitation

The lithium carbonate was at least 99.29% pure and the impurities are in line with what could be expected from the feed solution composition. The lithium purity was calculated by conducting a speciation from the anionic and cationic impurities and then subtracting this from 100%.

13.5.6.11 Refined Lithium Carbonate

On re-precipitation of lithium carbonate a battery grade product was produced.

13.5.6.12 Glaserite

The maximum potassium precipitated was ~75% in LCT3.

In three cycles a K/Na molar ratio of ~2 was achieved. This is below the theoretical maximum of 3.

The volume of solution evaporated varied from ~41 to 56%.

Overall the best result was LTC3 with the highest percentage potassium precipitated and the second highest K/Na molar ratio of 2.01.

13.5.6.13 Glauber's Salt Melt

The product assay indicates that it was >95% anhydrous sodium sulfate.

Approximately 38% of the sodium reported to the anhydrous solid product.

The solid product was relatively pure and contained <0.02% chloride.

13.5.6.14 Cs and Rb Removal

Approximately 48% of the caesium and 22% of the rubidium precipitated overall, which was well below the 96% Cs removal achieved by SGS, Malaga.

It is likely that the caesium precipitation was much higher, but some dissolved on raising the pH to 7.5 when caustic was added.

13.5.6.15 Glaserite Decomposition

Glaserite was decomposed and recrystallised to produce potassium sulphate, and XRD confirmed that postassium sulphate was the only phase detected.

14 MINERAL RESOURCE ESTIMATION

14.1 Introduction

This MRE update was completed by Oliver Jones (Consultant - Resource Geology) and Ben Lepley (Senior Consultant - Resource Geology) under the supervision of Martin Pittuck, CEng, MIMMM, FGS (Corporate Consultant - Mining Geology) who has some 20 years' experience in generating and reviewing Mineral Resource estimates for a wide variety of deposit styles; meeting the definition of an "independent Qualified Person" as this term is defined in National Instrument 43-101.

This section describes the Mineral Resource estimation methodology and parameters. The Mineral Resources have been reported in accordance with generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practices" guidelines and National Instrument 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted to Mineral Reserves.

The database used to estimate the Mineral Resources was audited by SRK and SRK is of the opinion that the current drilling information is sufficiently reliable to support a Mineral Resource.

Leapfrog Geo Software ("Leapfrog") was used to construct the geological model. Microsoft Excel was used to audit the drillhole database, and prepare assay data for geostatistical analysis. Supervisor Software ("Supervisor") was used for geostatistical analysis and variography. Datamine Studio Version 3 ("Datamine") was used to construct the block model, estimate grades and tabulate the resultant Mineral Resources.

14.2 Resource Estimation Procedure

The estimation methodology comprised:

- Database verification and preparation for geological modelling (including compositing)
- Discussions with client regarding geology and mineralisation
- Construction of geological model and wireframes
- Definition of fault blocks and resource domains
- Preparation of database for geostatistical analysis and variography
- 2D and 3D Block modelling and grade interpolation
- Resource validation and classification
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cutoff grade
- Preparation of a Mineral Resource Statement.

14.3 Resource Database

SRK was provided with a package of electronic and paper based data by the Company. This included:

- Raw drillhole data sheets in Microsoft Excel format covering the drillhole collars, associated assay results and geology for each of the La Ventana and El Sauz / Fleur concessions independently

- MapInfo data files relating to:
  - topography
  - licence tenure
  - geological and structural interpretation.

14.4 Topographic Survey

A detailed 1 m resolution topographic survey has been undertaken, covering the extent of the known lithium deposit included in this study. Topographic data was collected using LiDAR simultaneously with high resolution aerial photography.

Figure 14.4.1 show the LiDAR imagery and aerial photography draped over the LiDAR Digital Elevation Model ("DEM", Figure 14.4.2) which has allowed verification of the drillhole collars as well as adding increased definition to the mapped geological contacts between the clay and various other units.

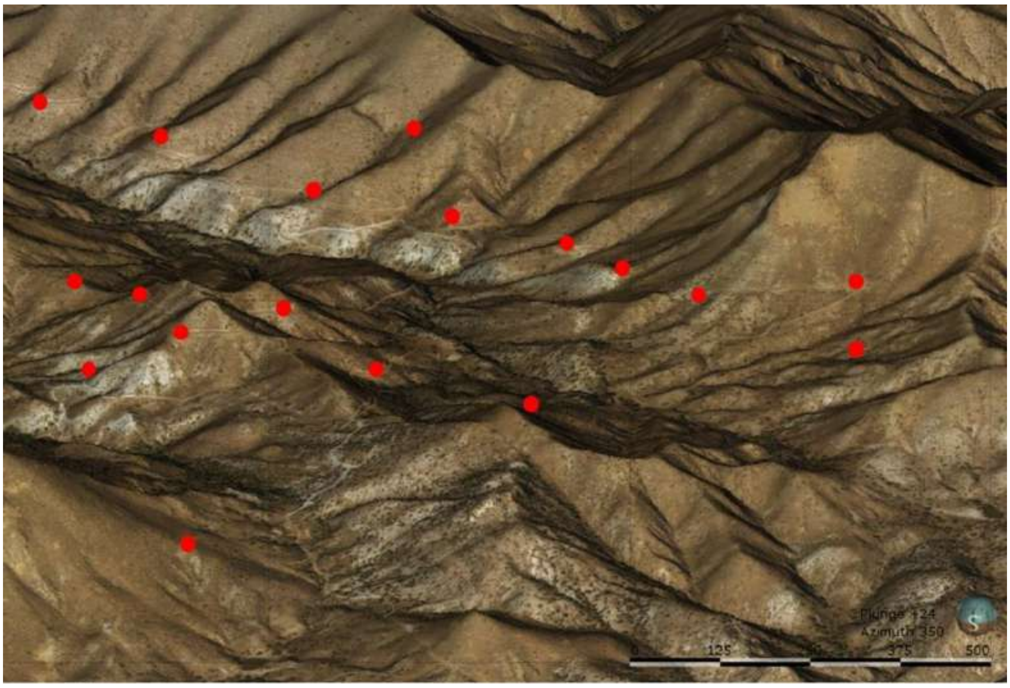


Figure 14.4.1: Aerial Imagery Draped Over Topographic Mesh to Validate Drillhole Locations (Red)

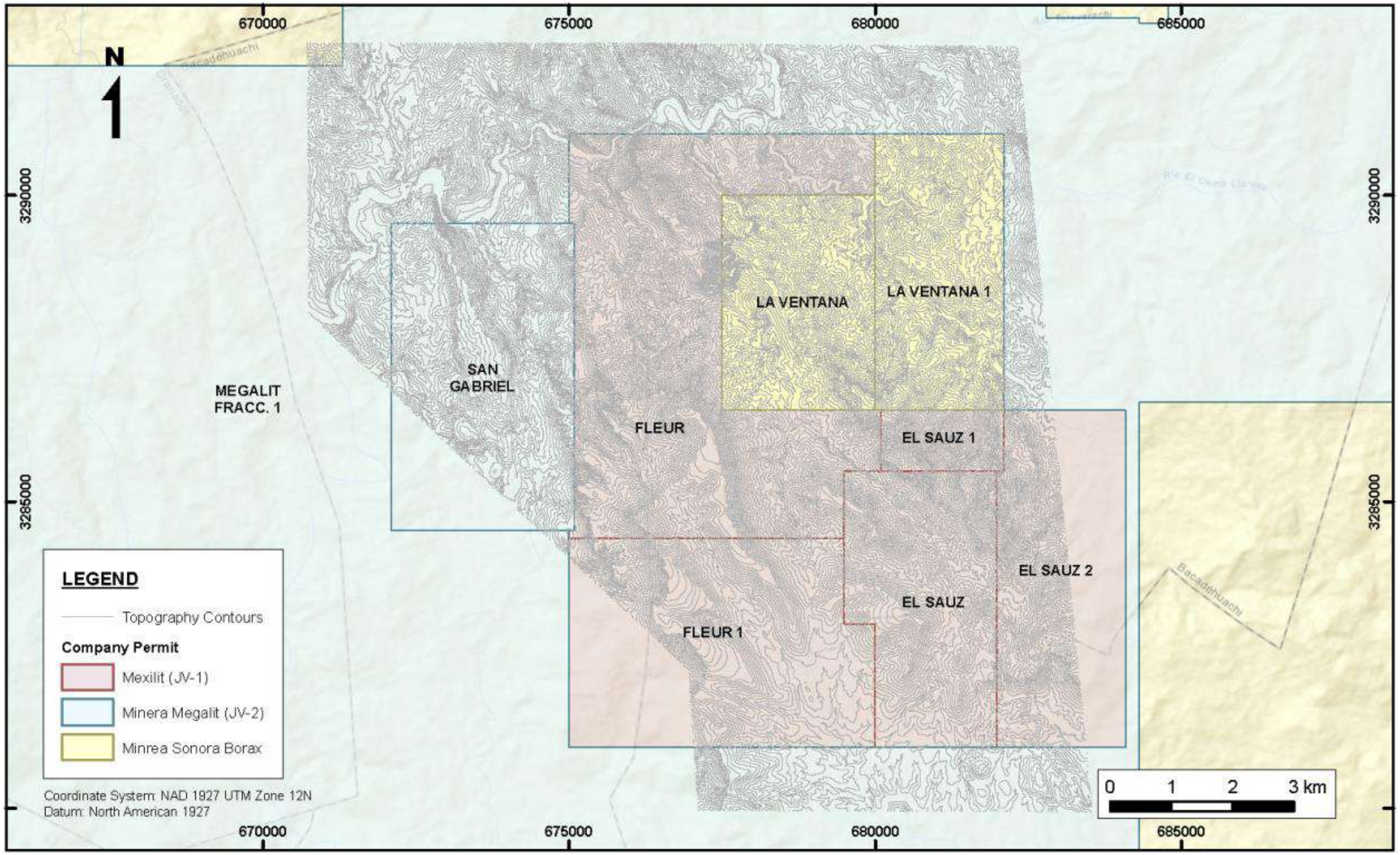


Figure 14.4.2: Area Covered by Available LiDAR Imagery

14.5 Geological Modelling

The MRE is based on a 7.2 km portion of a northwest-southeast regional trending lithium enriched clay sequence. SRK has created a geological model constrained by the licence holdings of the company and based on the lithological logging, assay data, structural and interpretive sections provided by the company. The deposit has been modelled as three main geological domains. At the stratigraphic base of the clay bearing units is the “Lower Clay Unit”, this is typically well mineralised and up to 20 m thick, this is overlain by a weakly mineralised Ignimbrite sheet. At the top of the sequence is the “Upper Clay” which has been subdivided into a “High Grade Upper Clay” and an “Upper Clay” unit in the well drilled Fault Block 4 area of the deposit. The deposit has also been subdivided into five fault blocks, described in further detail in Section 14.6.2.

14.6 2D Modelling and Interpretation

In developing a 3D model, SRK has created a series of 2D representation to assess the deposit geometry and grade distribution for each clay unit, which has identified several features material to the estimation process; these are described in the following sections.

14.6.1 Thickness

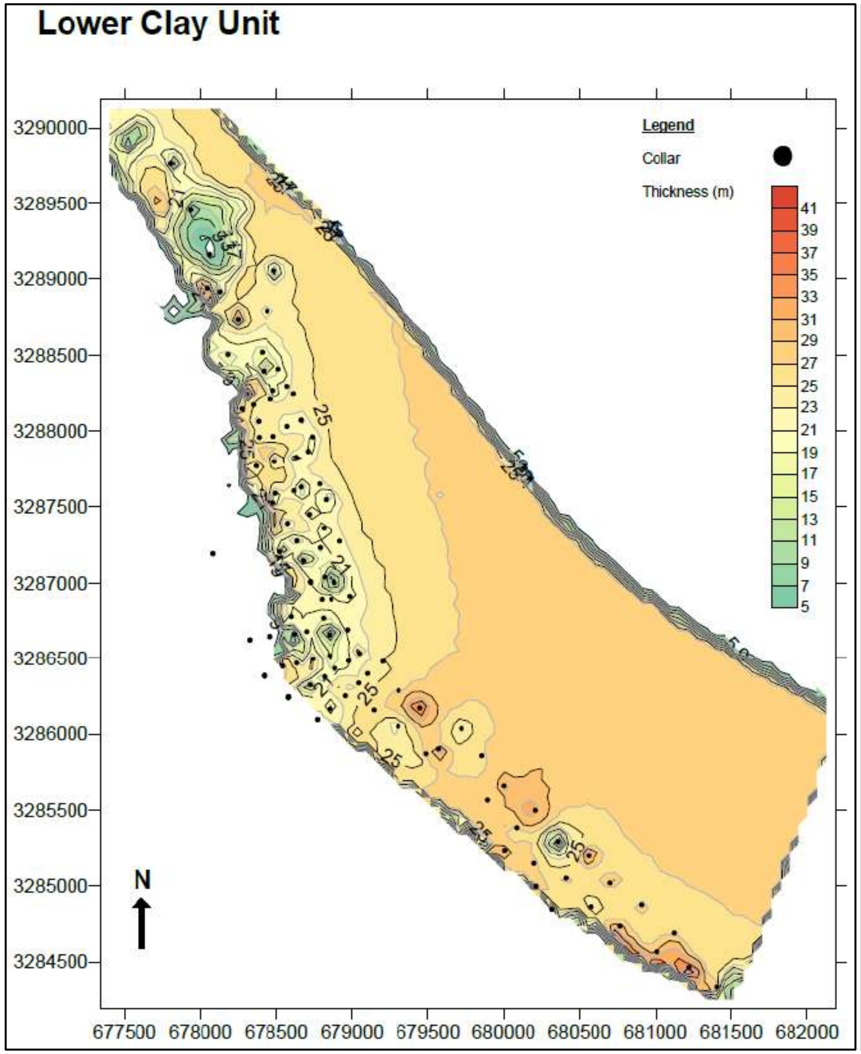
Figure 14.6.1 and Figure 14.6.2 show the wireframed thickness of the Lower Clay, Ignimbrite, Upper Clay (high-grade) and Upper Clay (low-grade) within the main northern fault block. In the Lower Clay Unit, the thickness is greatest in the southeast where it reaches 50 m; this reduces gradually to 20 m at the centre of the zone and towards the northern extents of the data. The Upper Clay (low-grade) and Upper Clay (high-grade) unit thicknesses are greatest at the northern end of the drilled area where it reaches 50 m and 20 m respectively; this reduces southwards varying gradually between 10 m and 30 m thick at the southern extent of the data.

14.6.2 Structures

A 3D assessment of lithological drillhole logging and surface structural maps identify the presence of several faults which offset the mineralised horizons. These structures have been used in the subsequent 3D geometry and grade modelling processes as fault block domain boundaries. Other faults may also be present, but due to the vertical nature of the majority of the holes (sensible given the dip of the mineralisation), these structures cannot currently be confirmed.

14.6.3 Grade

Section 14.12.2 provides plan maps of the grade variation across the deposit. Although these trends are visible in the raw data, they are best visualised in the resultant estimated block model. The figures demonstrate a strong trend towards grade zoning, resulting in a “bulls-eye” grade pattern with highest grades seen in the centre of the domains, gradually transitioning to towards lower grades at the margins. This effect is best observed in the northern fault block where the majority of the drilling has been undertaken.



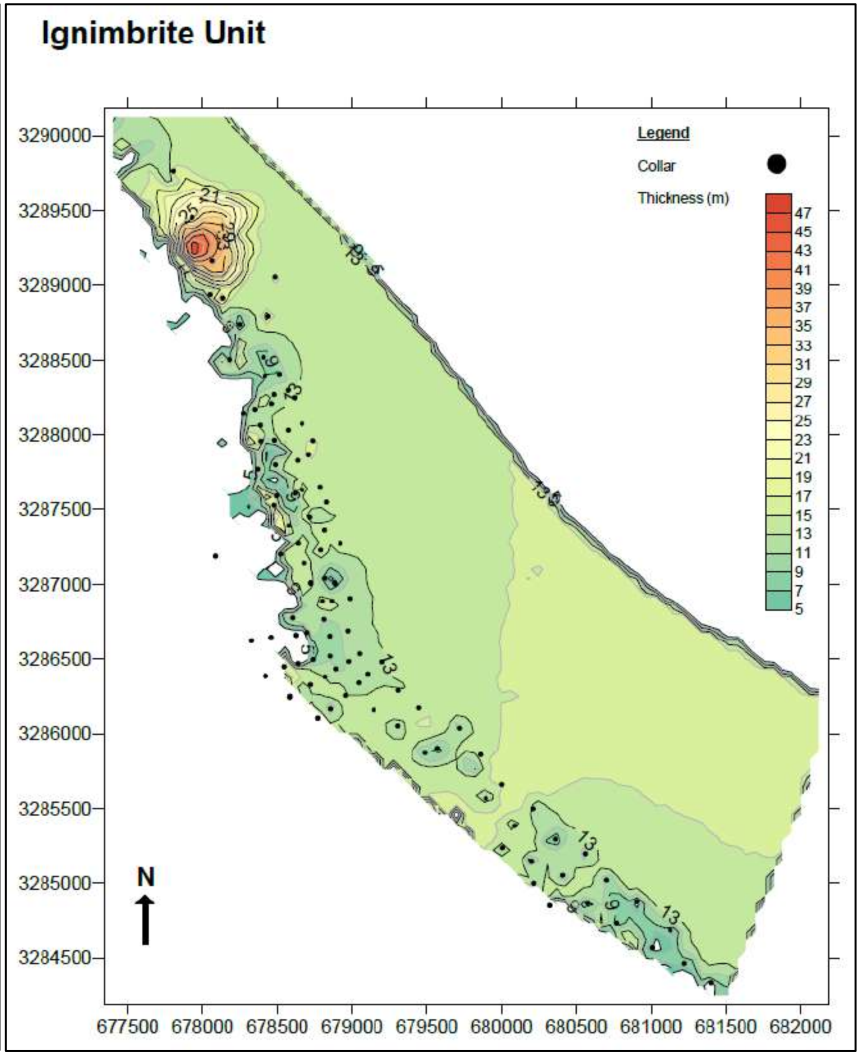


Figure 14.6.1: Lower Clay Thickness Contour Map (Left) and Ignimbrite Thickness Contour Map (Right)

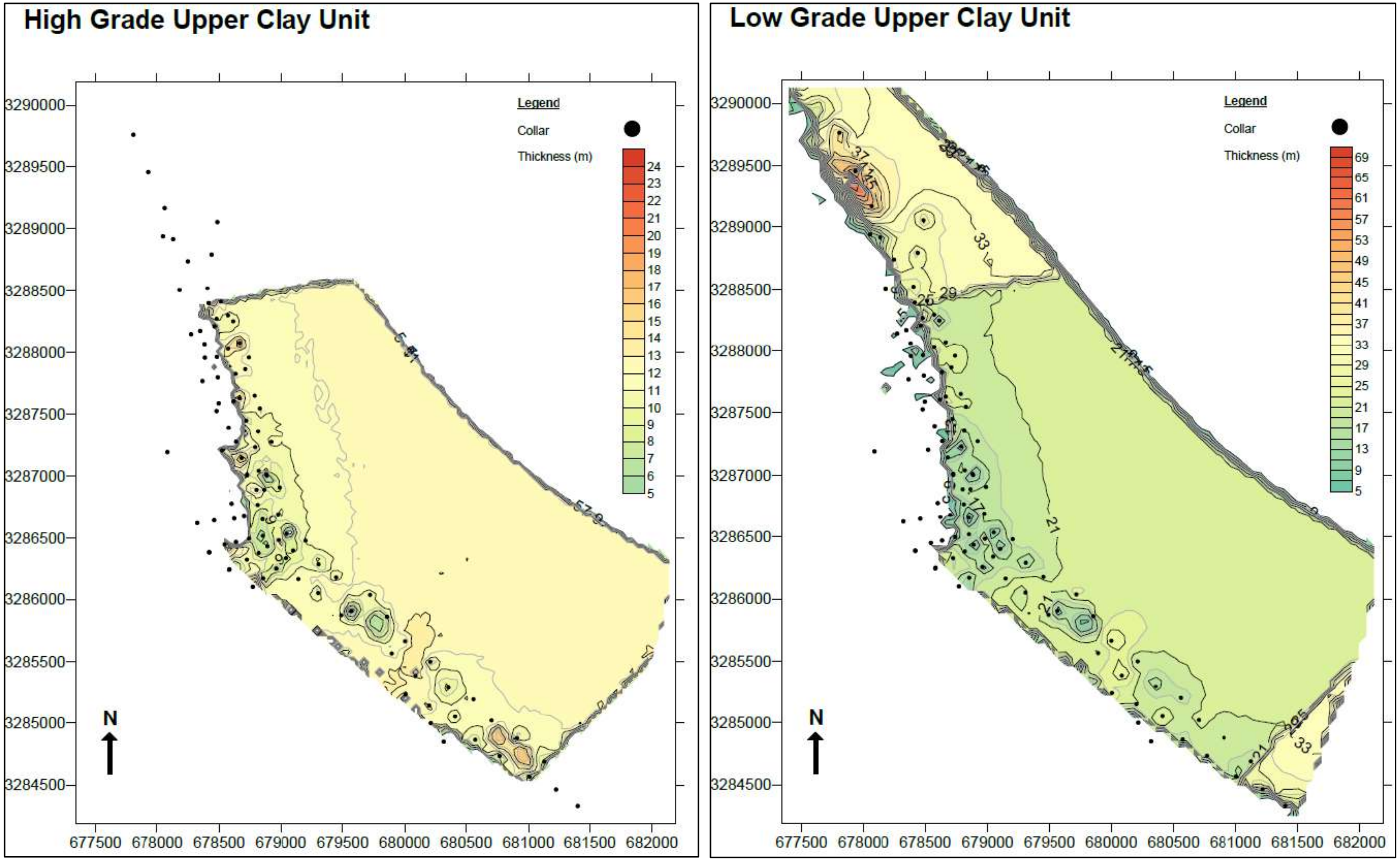


Figure 14.6.2: Upper Clay (High-Grade) Thickness Contour Map (Left) and Upper Clay (Low-Grade) Thickness Contour Map (Right)

14.7 3D Geological Modelling

SRK has undertaken geological modelling to provide geological constraints for the MRE. These constraints are provided as wireframe models into which the final block models were created and domained. The geological model constructed for the Project has been used to differentiate between fault blocks and the Upper and Lower Clay Units, as well as the high and low grade sub domain within the northern Upper Clay Unit.

14.8 Deposit Modelling

The following section describes the methodology undertaken for modelling of the Project. All modelling was undertaken using Leapfrog Geo software into which cross sections from previous interpretations were imported for reference.

14.8.1 Geological Zone Modelling

The deposit modelling comprised the following:

- Importing the collar, survey, assay, geology, and magnetic susceptibility data into Leapfrog to create a de-surveyed drillhole file)
- Importing the topography data file
- Importing site generated interpretations, plan maps and cross sections
- Creating the mineralisation wireframes based on the domain.

A number of fault surface wireframes were first modelled based on mapped traces, dip-strike field data and interpreted occurrence in drillholes. This process resulted in five fault blocks which materially impact the strike continuity of the lithium bearing clay units.

Geological zones were created by grouping the logged lithology codes then generating wireframes for each lithological unit linking between drillholes and outcrop, ensuring the stratigraphic sequence continued through the Project area. Each lithological wireframe has been clipped against the fault domain boundaries and topography.

Figure 14.8.1 shows the mineralisation wireframes produced by SRK in combination with interpretive cross sections provided by the client. Figure 14.8.2 provides a cross section showing all stratigraphic units which have been offset and controlled by generating differing fault blocks independently referenced to structural data collected on site.

Figure 14.8.3 shows the wireframes that were used to constrained the raw data and define the zone coding implemented during the creation of the block model. Table 14.8.1 references each of the domain codes applied representing both the clay unit and the respective fault domain.

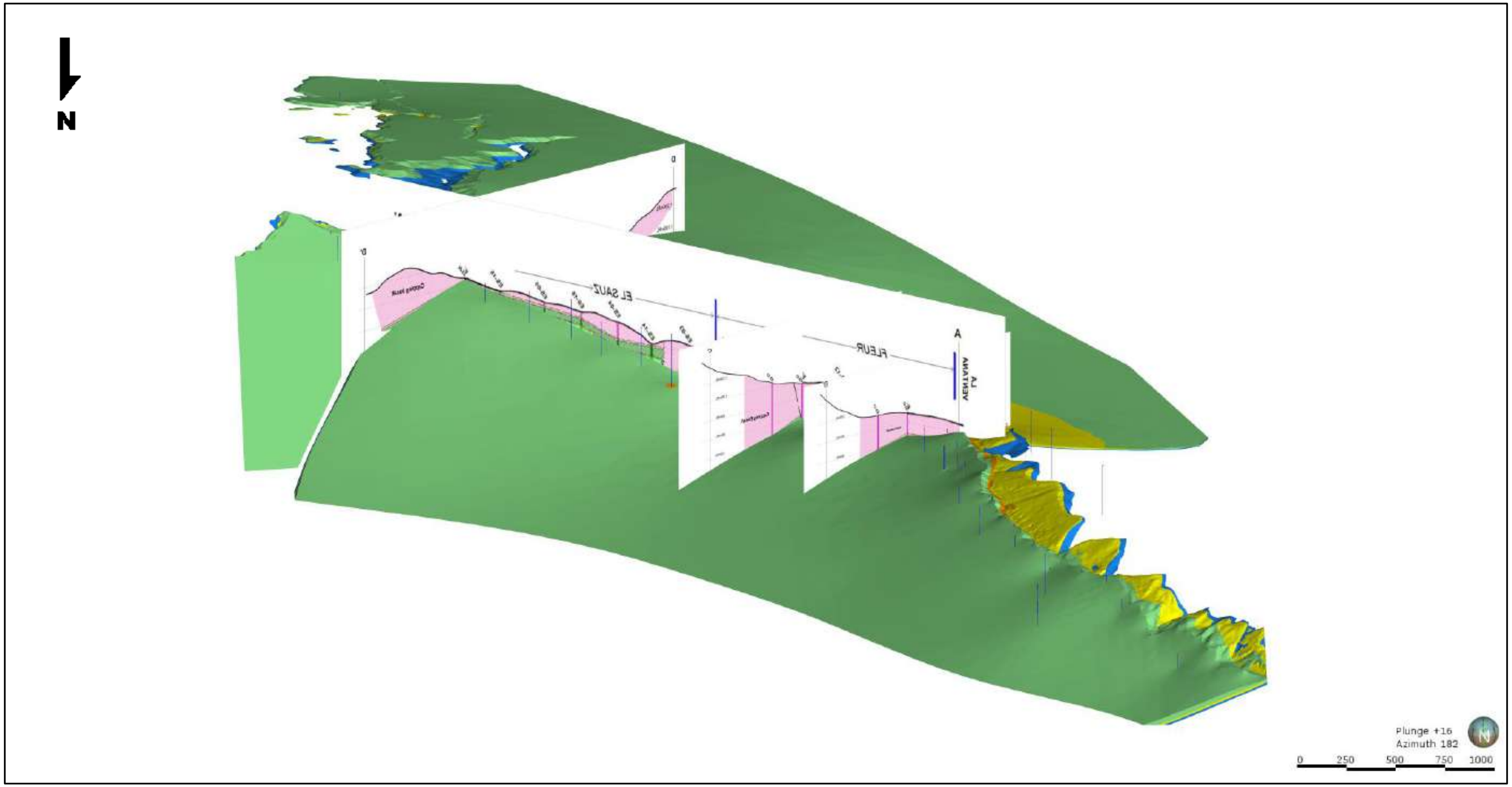


Figure 14.8.1: South Facing Isometric View of Cross Sections Provided by the Company Registered in 3D Space

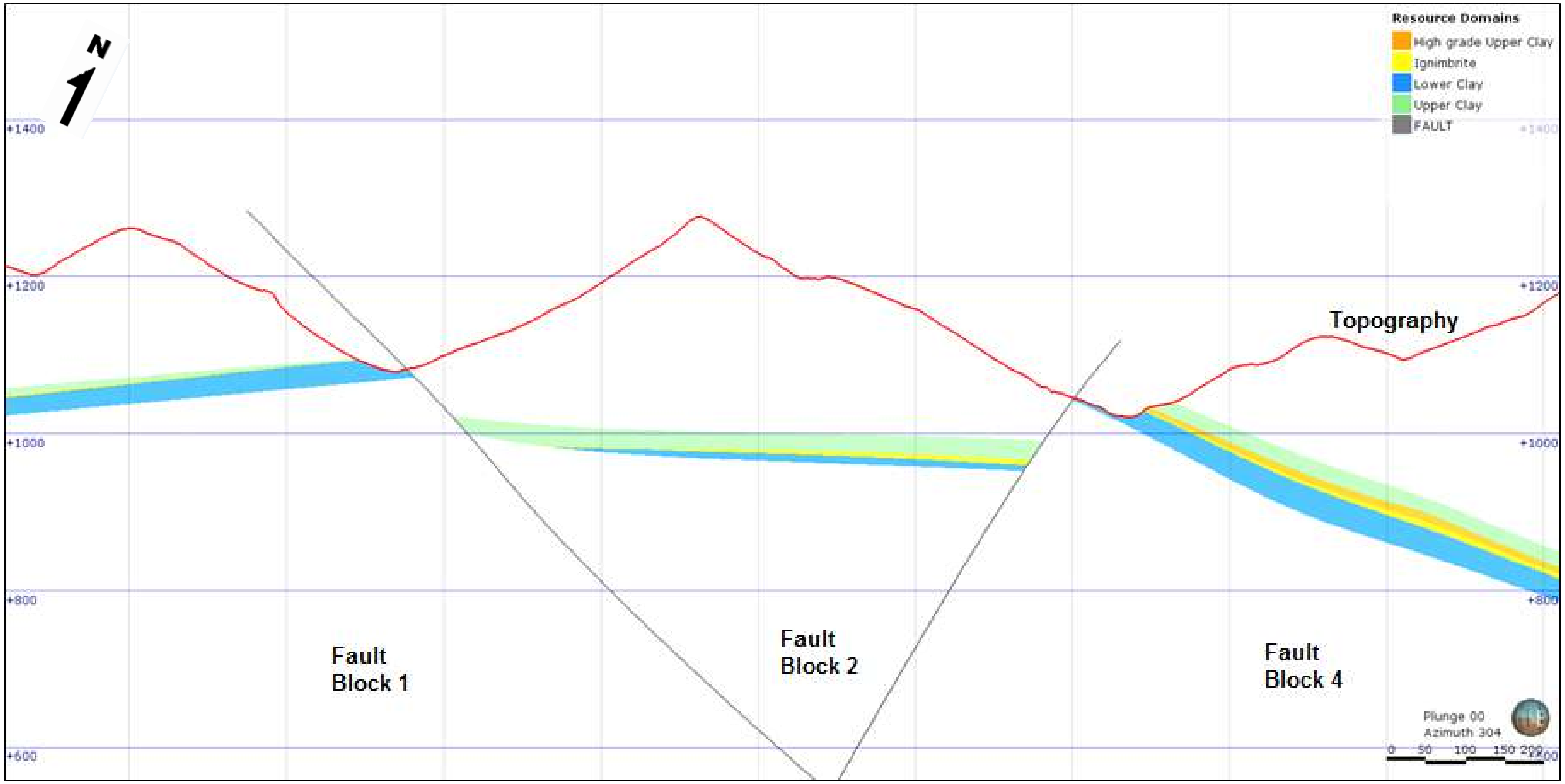


Figure 14.8.2: Northwest-Looking Cross Section Showing Stratigraphic Units and Related Fault Structures

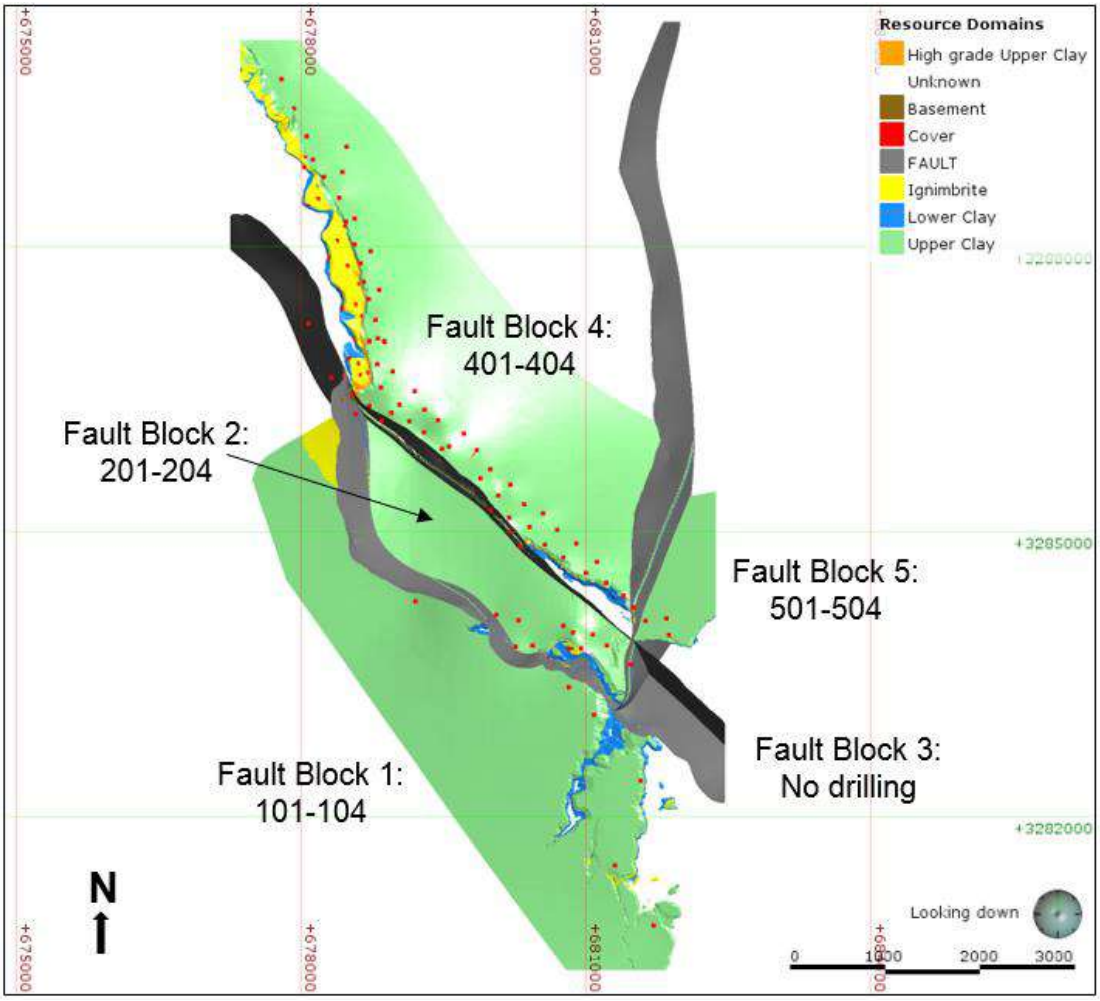


Figure 14.8.3: Wireframes in Plan Showing the Fault Blocks and Domain Codes

Table 14.8.1: Domain / Kriging Zone Codes (KZONES)

Domain / KZONE	Description
101	Lower Clay (Fault Block 1)
102	Ignimbrite (Fault Block 1)
103	Upper Clay (Fault Block 1)
201	Lower Clay (Fault Block 2)
202	Ignimbrite (Fault Block 2)
203	Upper Clay (Fault Block 2)
401	Lower Clay (Fault Block 4)
402	Ignimbrite (Fault Block 4)
403	Upper Clay High Grade domain (Fault Block 4)
404	Upper Clay Low Grade domain (Fault Block 4)
501	Lower Clay (Fault Block 5)
502	Ignimbrite (Fault Block 5)
503	Upper Clay (Fault Block 5)

14.8.2 Block Model Creation

An empty block model was generated in Datamine Studio 3 software ("Datamine"). The block model includes zone codes for each of the mineralised clay units and ignimbrite wireframes in each of the fault blocks.

The mineralisation modelled has a strike length of some 7.2 km. Deep drilling has demonstrated the existence of mineralisation some 500 m down dip from outcrop and SRK has extended the block modelled mineralisation a further 300 to 400 m down dip to ensure any potentially economic material below that already defined can be included in the Mineral Resource or identified as a drilling target. A waste model was also generated below the topography and outside of the mineralisation zones.

14.9 Classical Statistical Study

This section presents the results of the statistical studies undertaken on all the available assay and density data sets to determine their suitability for the estimation process and to derive appropriate estimation constraints.

14.9.1 Introduction

The samples analysed typically comprise an approximate 1.5 m sample interval. A total of 4,993 raw drillhole assays are available for use in the modelling and MRE process.

14.9.2 Raw Statistics

The domains described above have been used to distinguish the differing horizons and spatial relationships, based principally on the lithological logging and geological interpretation supported by Li grade.

Figure 14.9.1 shows a positive skew for Li data in histograms (negative skew in lognormal histograms) for both the Lower Clay and Upper Clay Low Grade domains. This distribution is related to the gradual transition in grade over the entire strike length of the deposit, resulting in a mixture of high and low grade samples rather than a specific grade population.

SRK also notes that the maximum value of 10,000 ppm Li that can be returned by the laboratory and method employed terminates the distribution curve of the Lower Clay Unit unnaturally. This suggests that all samples currently in the database with a value of 10,000 ppm would have higher grades if they were submitted for assay using a different method with a higher detection limit. There are a total of twenty samples in the raw sample database that have been returned with the upper analytical detection limit of 10,000 ppm Li. All of these samples fall within the high grade core of the Lower Clay Unit in Block 4. It is not considered to be a material issue, however, local grade estimates in this area will be under-reporting true grades.

Figure 14.9.2 shows generally normal to sub-normal populations for K% data in each of the primary domains.

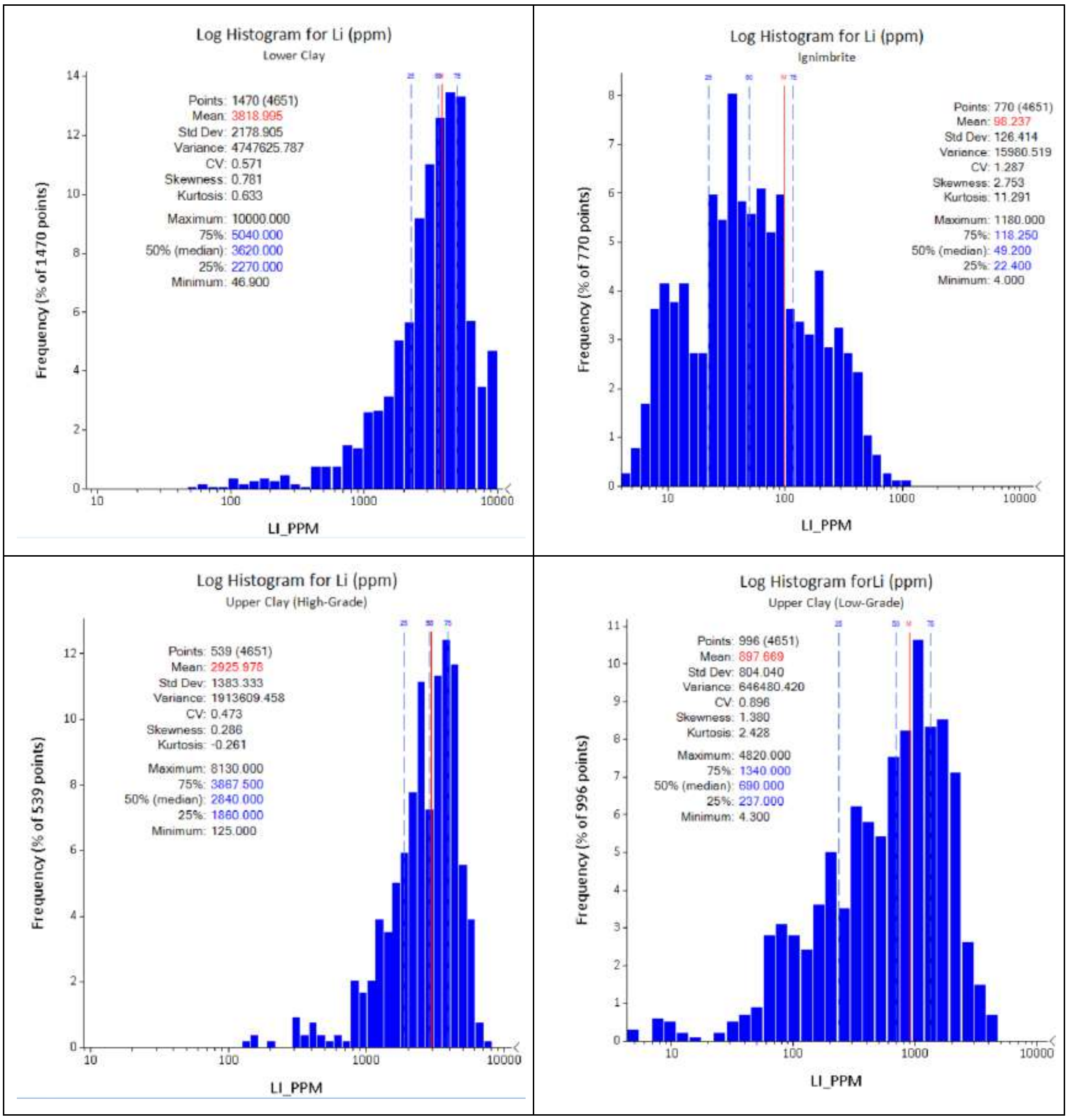


Figure 14.9.1: Li (ppm) Lognormal Histograms for Upper and Lower Clay Units as Well as the Upper Clay High Grade and Low Grade Subdivisions

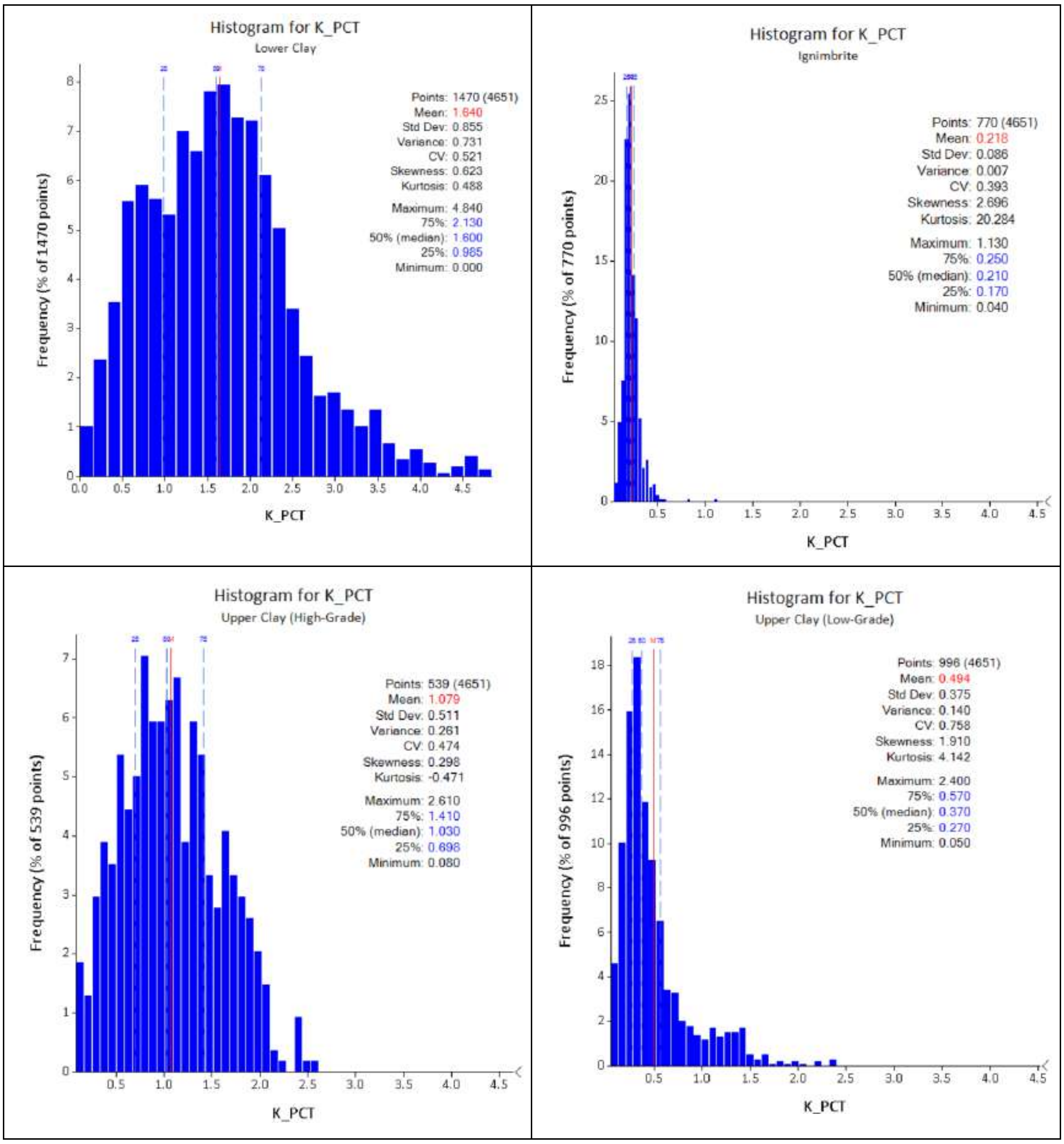


Figure 14.9.2: K% Histograms for Upper and Lower Clay Units as Well as the Upper Clay High Grade and Low Grade Subdivisions

14.9.3 Domain Boundary Analysis

In order to check whether 'hard' or 'soft' boundaries should be used between the modelled geological units during estimation, a domain boundary (or contact analysis) was undertaken. A hard boundary implies that only data within the unit is used for estimation, whereas a soft boundary allows for samples close to the boundary to be shared between domains. The average grades of samples at 1 m intervals adjacent to each boundary were analysed and plotted to show the nature of the boundary. If a gradational boundary is observed, a soft domain boundary may be necessary, whereas a sharp boundary would require a hard domain boundary.

The analysis plots for Li are shown in Figure 14.9.3. All of the plots show sharp boundaries between the units with generally large steps in average grades at the contacts. Following this analysis, SRK opted to use hard boundaries for all domains.

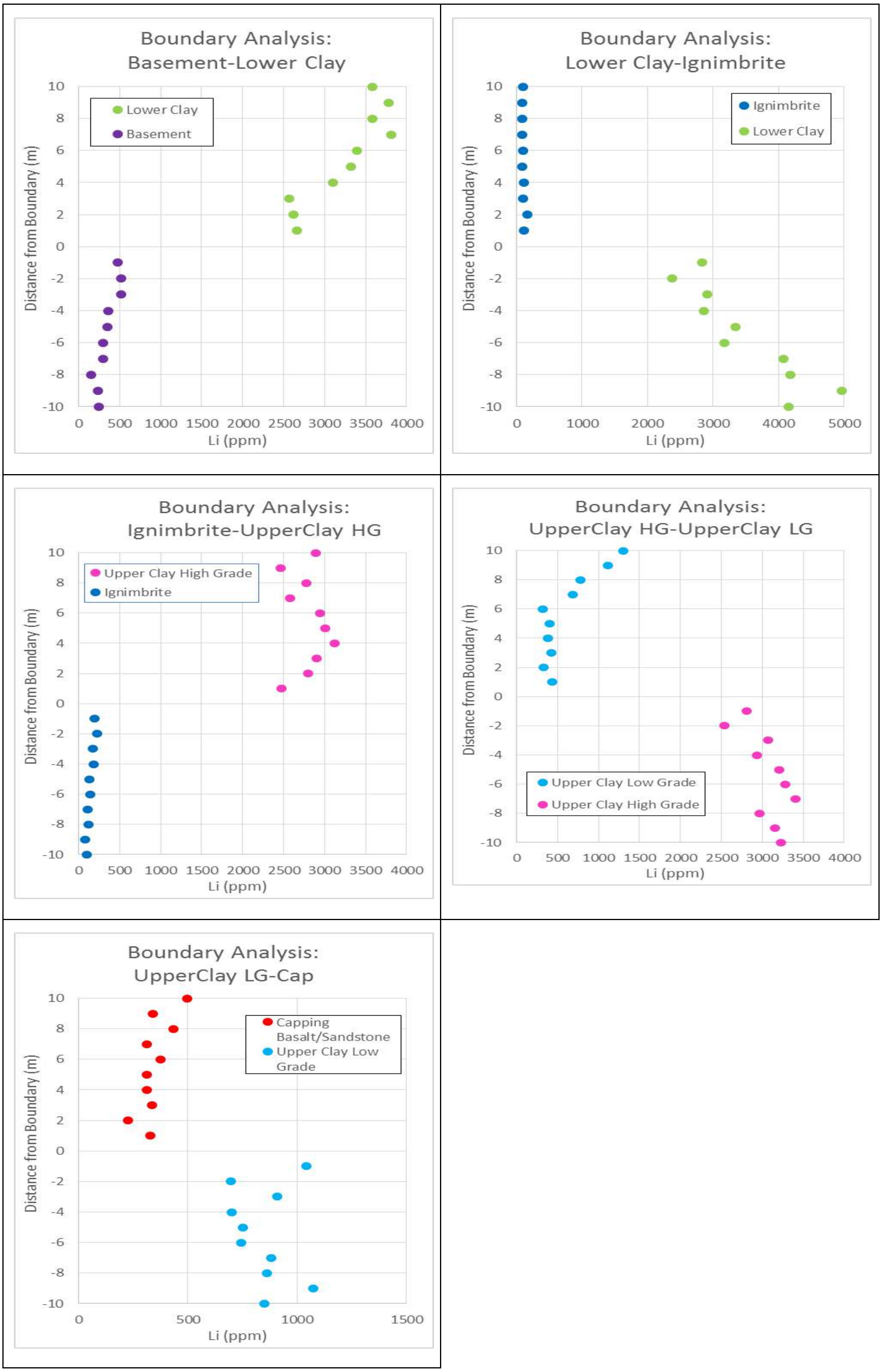


Figure 14.9.3: Domain Boundary Analysis Plots (Li)

14.9.4 Data Compositing

Due to the relatively flat lying nature of the mineralisation and the large lateral extent compared with the vertical extent of each domain, a decision was made to undertake a 2D grade estimate. Vertical grade variation is noted in places, but it has not been identified with sufficient continuity between drillholes to have been modelled as further subdomains or to have been reflected in the estimation process. The samples in each drillhole have therefore been composited to create one sample per unit as described below.

The average grade of the entire composite interval per domain is a length-weighted average of the sample grades. The drillholes are domained using wireframes based on lithological contacts prior to compositing. There is a separate composite for each drillhole intersection within each of the major lithological units:

- Lower-grade upper part of the upper clay
- Higher-grade lower part of upper clay
- Barren ignimbrite
- Lower clay.

This method assumes that there will be limited vertical selectivity in the mining method other than mining to lithological contacts, which is currently considered valid.

The statistics of the composited point data by KZONE are presented in Table 14.9.1. Only grades for the area updated during this MRE (Fault Block 4) are displayed.

Table 14.9.1: Composite Statistics by KZONE (Weighted by Clay Unit Thickness)

Domain	Field	No Samples	Minimum	Maximum	Mean	Stand Dev	CoV
Upper Clay (LG)	Li (ppm)	75	103	1,734	892	342	0.4
Upper Clay (HG)		65	593	4,535	2,937	813	0.3
Ignimbrite		94	17	605	97	73	0.8
Lower Clay		90	107	6,283	3,817	1219	0.3
Upper Clay (LG)	K (%))	75	0.1	0.9	0.5	0.2	0.3
Upper Clay (HG)		65	0.4	1.6	1.1	0.2	0.2
Ignimbrite		94	0.1	0.7	0.2	0.1	0.3
Lower Clay		90	0.3	2.5	1.6	0.4	0.3

14.9.5 Density Analysis

Bulk density measurements have been undertaken for all material types. In total, 3,804 samples have been analysed for bulk density (sun-dried) from the identified stratigraphic horizons. No relationship is observed between lithium grade and density. Due to the large quantity of density measurements available, SRK estimated density (sun-dried values) into the block model for tonnage calculations.

Table 14.9.2 shows the average density values determined for each material type. These values are applied to the waste zones and where the density was not estimated into mineralised units.

Table 14.9.2: Average Dry Density per Unit

Unit	Average Dry Density (g/cm^3)
Capping Basalt	2.42
Capping Sandstone	2.17
Upper Clay (low-grade)	2.23
Upper Clay (low-grade)	2.32
Ignimbrite	2.19
Lower Clay	2.32
Basement	2.23

14.10 Geostatistical Analysis (Variography)

14.10.1 Introduction

A geostatistical analysis (variography) of the drilling data was undertaken to understand the spatial variability of the data. Variography was undertaken for Li and K in the zone 400 fault block for the 401, 402, 403 and 404 domains where sufficient data to undertake a geostatistical study are present. The drillhole database, flagged by modelled zones, was imported into Snowden Supervisor software for the geostatistical analysis.

14.10.2 Drift

Grade in all Zone 400 domains shows zonation through a bulls-eye pattern with the high-grade at the centre in the fold hinge where La Ventana and Fleur licences split, with lower grades towards the northern and southern extents. This is considered to be a geological phenomenon and may be related to the primary source of lithium into the basins.

As a result of this zonation, all variograms show a parabolic behaviour, which is due to a statistical trend known as ‘drift’. This is demonstrated in Figure 14.10.1, where after the variance has stabilised at a partial sill (variance inflection point) it then rises sharply again in the case of most domains above the total population variance. The occurrence of drift demonstrates that the domained data is non-stationary (i.e. the histogram of sample points is not consistent across the domain but rather changes with distance). As one of the assumptions of the Kriging process is that the data is stationary, modelling the variogram to this trend is not considered correct.

SRK therefore attempted to model drift into variograms using Isatis software. Variograms showed improvements, however, the resulting grade interpolation showed large-scale smoothing of grade. SRK considered it more appropriate to model normal spherical model variograms to first structure before the drift is evident and not to the entire population variance, which is heavily skewed by the drift. These ranges still much longer than general drill spacing and it is therefore not thought to be having a material on local or global estimates of grade.

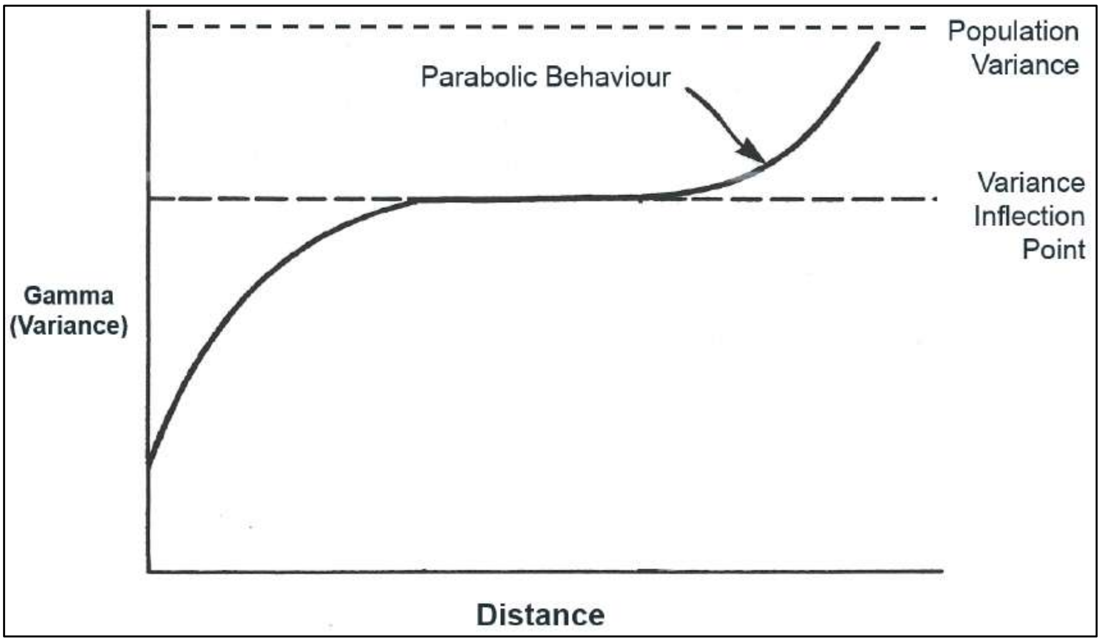


Figure 14.10.1: Variogram Model Displaying Parabolic Behaviour (Drift)

14.10.3 Variogram Fitting

Experimental semi-variograms were produced for using a sensible lag to define the nugget effect, sill (variance) structures and ranges. Due to the lack of number of samples down-dip, omni-directional variograms were produced using the 2D sample data (one composite point for each domain per drillhole), which provided the most robust variogram structures.

Figure 14.10.2 shows the modelled variograms produced for the three clay units in Block 4 for Li. Variograms produced for K showed similar ranges and structures to Li.

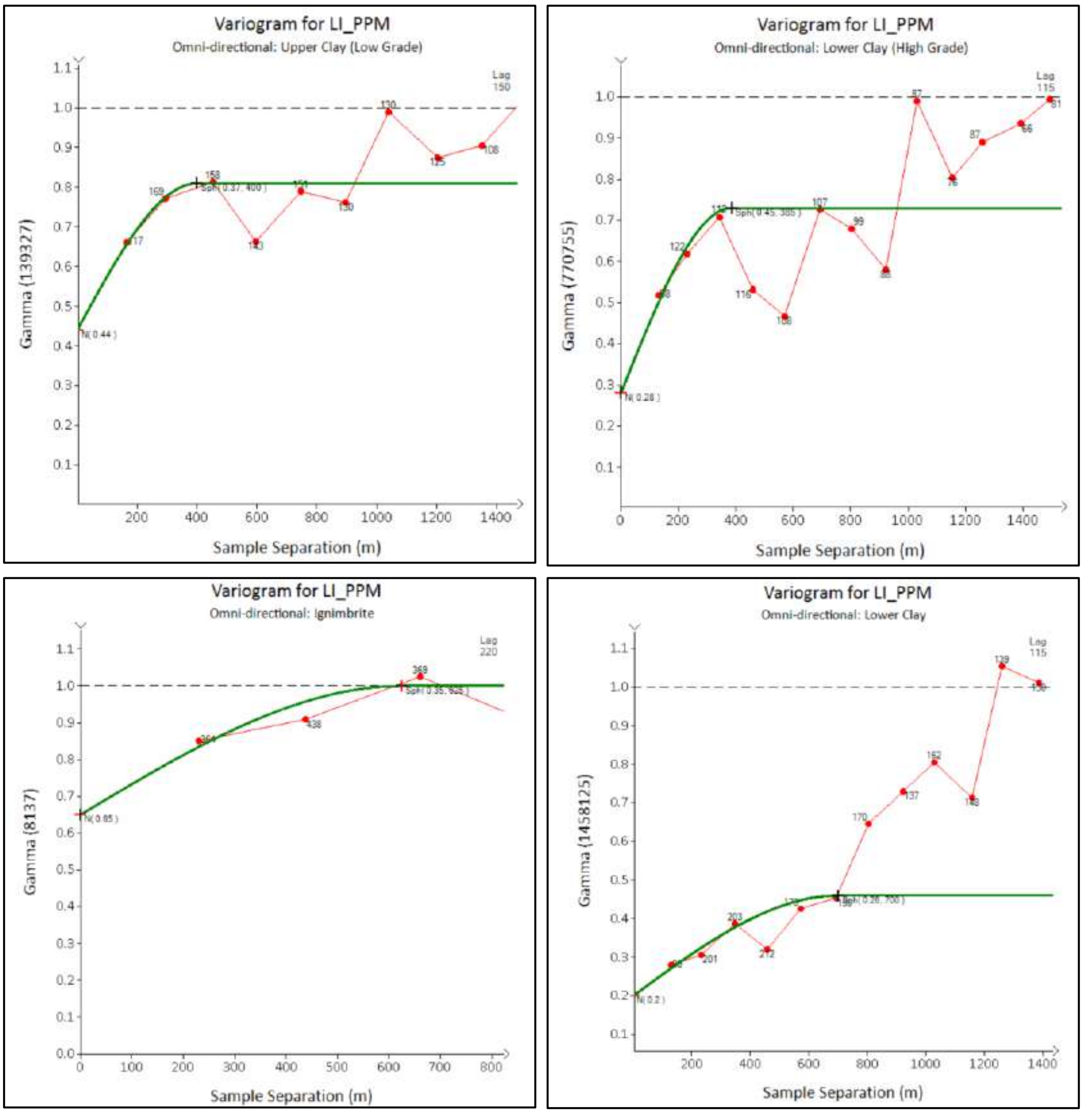


Figure 14.10.2: Lithium Variography for Zone 400 Domains Based on the 2D Composites

14.10.3.1 Summary

Due to the volume of data available in fault block 4 relative to the other fault domains, the variogram models produced for fault block 4 were applied to all other fault blocks to generate suitably reliable interpolation parameters. The results of the variography were used in the interpolation to assign the appropriate weighting to the sample points utilised to calculate the block model grades.

The total ranges modelled are also incorporated to help define the optimum search parameters and the search ellipse radii dimensions used in the interpolation. Ideally, sample pairs that fall within the range of the variogram (where a strong covariance exists between the sample pairs) should be utilised if the data allows.

Table 14.10.1 shows the rounded total ranges of the Li variograms for the differing domains. As shown, the modelled ranges are greatly in excess of the drill spacing. The variograms for K showed similar ranges and sills to Li.

Table 14.10.1: Summary of Lithium 2D Semi-Variogram Parameters (Normalised)

Domain	Rotation (X)	Rotation (Y)	Rotation (Z)	Nugget (%)	Range Strike (m)	Range Dip (m)	Sill (%)
Upper Clay (LG))	0	0	0	54	400	400	46
Upper Clay (HG)	0	0	0	38	385	385	62
Ignimbrite	0	0	0	65	625	625	35
Lower Clay	0	0	0	43	700	700	57

14.11 Block Model and Grade Estimation

14.11.1 Block Model Set-Up

The geological wireframes were used to create a rotated 2D block model with origins and dimensions described in Table 14.11.1. The 2D block model was used for grade interpolation. A rotated 3D block model with origins and dimensions described in Table 14.11.2 was also created. The 2D interpolated block model was then converted into the 3D block model. Both the 2D and 3D block models were rotated -45°. Unique codes were developed for use in coding the block model and during estimation, as summarized in Table 14.11.3.

Table 14.11.1: 2D Block Model Origins and Dimensions

Dimension	Origin	Block Size	Number of Blocks
X	673,970	50	200
Y	3,287,560	50	105
Z	0	1700	1

Table 14.11.2: 3D Block Model Origins and Dimensions

Dimension	Origin	Block Size	Number of Blocks
X	673,970	50	200
Y	3,287,560	50	105
Z	400	10	105

Table 14.11.3: Summary of Fields Used During Estimation

Field Name	Code	Description
Domain (KZONE)	101	Lower clay zone fault block 1
	103	Upper clay zone fault block 1
	201	Lower clay zone fault block 2
	203	Upper clay zone fault block 2
	401	Lower clay zone fault block 4
	403	Upper clay zone (high grade) fault block 4
	404	Upper clay zone (low grade) fault block 4
	501	Lower clay zone fault block 5
	502	Upper clay zone fault block 5
Grade	LI_PPM	Ordinary kriged lithium (ppm) Grade
	K_PCT	Ordinary kriged potassium (%) grade
	MG_PCT	Inverse distance cubed magnesium (%) grade
	CA_PCT	Inverse distance cubed calcium (%) grade
	B_PPM	Inverse distance cubed boron (ppm) grade
	SR_PPM	Inverse distance cubed strontium (ppm) grade
	RB_PPM	Inverse distance cubed rubidium (ppm) grade
	CS_PPM	Inverse distance cubed caesium (ppm) grade
Search Parameters	LI_SV	Search volume number
	LI_KV	Kriging variance
	LI_NS	Number of samples used for block estimate
Licence (LICENCE)	La Ventana	La Ventana licence
	La Ventana 1	La Ventana 1 licence
	El Sauz	El Sauz licence
	Fleur	Fleur licence
	El Sauz 1	El Sauz 1 licence
	El Sauz 2	El Sauz 2 licence
	Fleur 2	Fleur 2 licence
Classification (CLASS)	1	Measured
	2	Indicated
	3	Inferred
	4	Unclassified

14.11.2 Grade Interpolation

Ordinary kriging ("OK") was used for grade interpolation into the 2D block model for Li and K grades and inverse-distance weighted interpolation for Ca, Mg, Cs, Rb, Sr and B grades. All grades were interpolated into the 2D block model honouring the geological contacts defined by the geological modelling process, and using the domains (KZONES) previously assigned. The same search parameters were used for all KZONES; these are summarized in Table 14.11.4. A 300 m search radius is well-within the modelled variogram ranges and ensures an adequate number of samples is used for each block estimate. The second and third searches were expanded by a multiplier factor of 2 and 15, respectively; the latter ensured all blocks in the model were estimated. Following the interpolation of the 2D block model, SRK converted the 2D grade interpolation into the 3D block model.

Table 14.11.4: Search Parameters for Interpolation

Domain	Search Dist (X, Y and Z)	Min Samp 1	Max Samp 1	Search Volume Factor 2	Min Samp 2	Max Samp 2	Search Volume Factor 3	Min Samp 3	Max Samp 3
All domains	300	4	6	2	4	6	15	2	8

14.12 Block Model Validation

14.12.1 Introduction

SRK has undertaken a number of validation checks to confirm that the modelled estimates of Li and K grades represent the input sample data on both local and global scales and to check that the estimate is not biased. Methods of validation used include:

- Visual inspection of block grades in comparison with drillhole data (in plan and cross section)
- Estimating Li (ppm) grades using an inverse-distance weighted algorithm ("IDW")
- Swath/validation plots
- Comparison of block model statistics.

Based on the visual and statistical validation, SRK has accepted the grades in the 2D and 3D block models. The resultant block grade distribution is considered appropriate for the mineralisation style. In areas of limited sampling, the block grade estimates have been produced using expanded search ellipses. Localised comparisons of block grades to block estimates will be less accurate in these areas.

14.12.2 Visual Validation

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection of cross-sections, and bench plans, comparing the sample grades with the block grades has been undertaken. This demonstrates a good comparison between local block estimates and nearby samples without excessive smoothing in the block model.

Figure 14.12.1 to Figure 14.12.3 show the visual validation checks for Li for the Upper Clay (low-grade), Upper Clay (high-grade) and Lower Clay zones. Validation of K grades produced similar results showing a good comparison between the sample and block grades.

Figure 14.12.4 and Figure 14.12.5 show two cross-sections through the block model in the La Ventana area (all units combined). The model grades are shown in comparison to the original (un-composited) drillhole grades to show the loss of resolution when compositing across the units. Given the bulk-scale of the mining operation, this is not considered to be an issue.

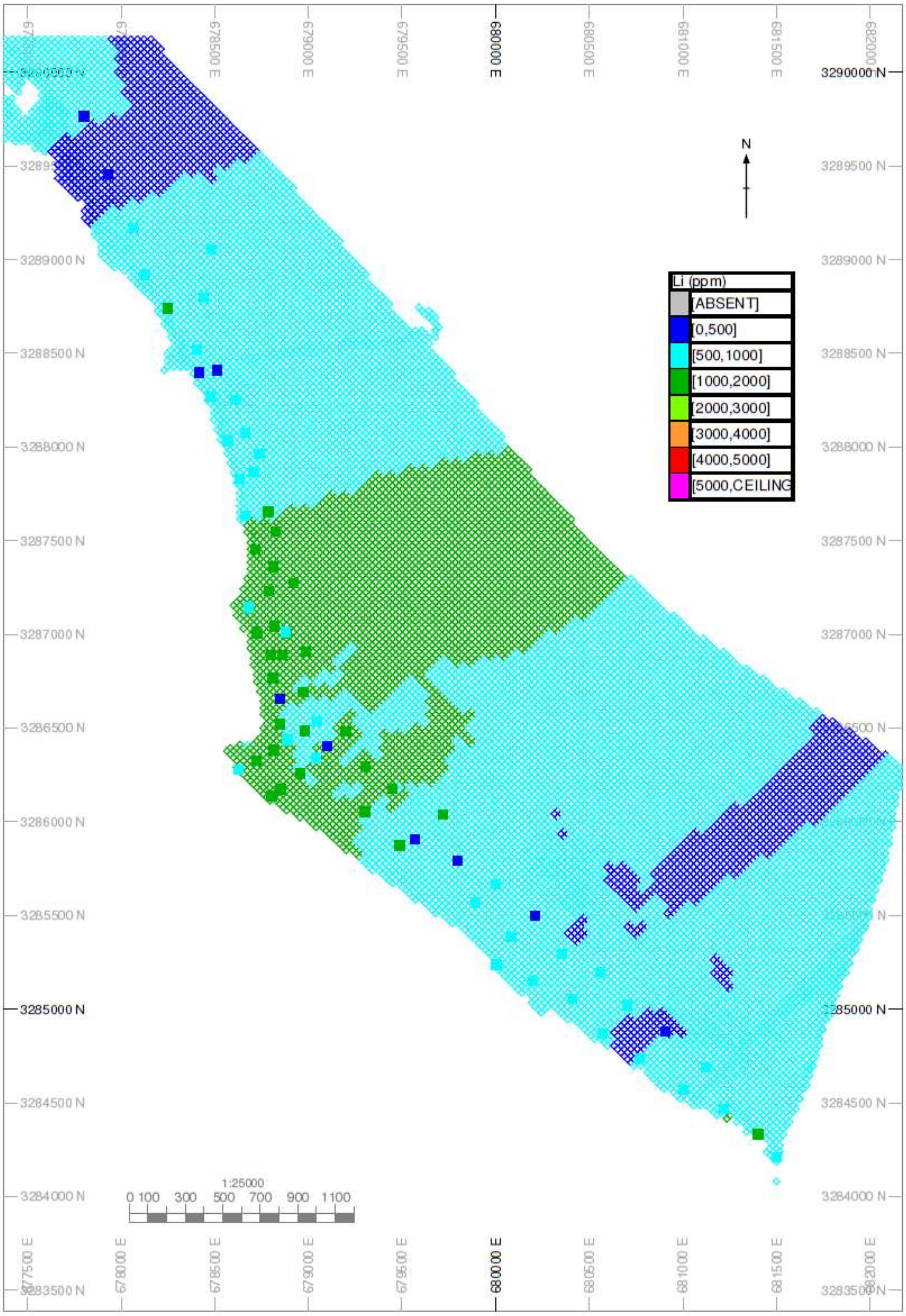


Figure 14.12.1: Upper Clay (Low-Grade) Li Block Model Validated Against Composited Drillhole Data (Squares)

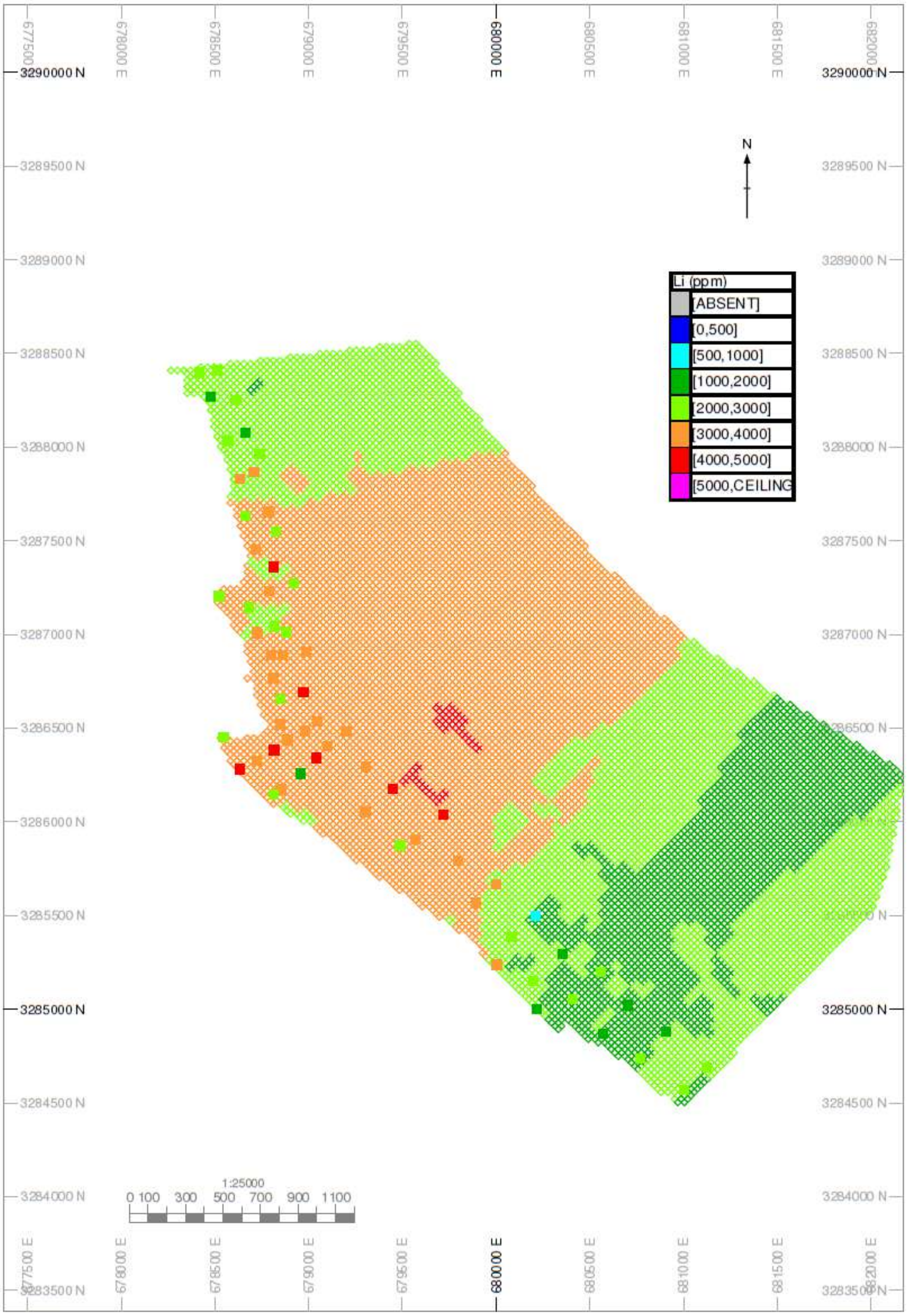


Figure 14.12.2: Upper Clay (High-Grade) Li Block Model Validated Against Composited Drillhole Data (Squares)

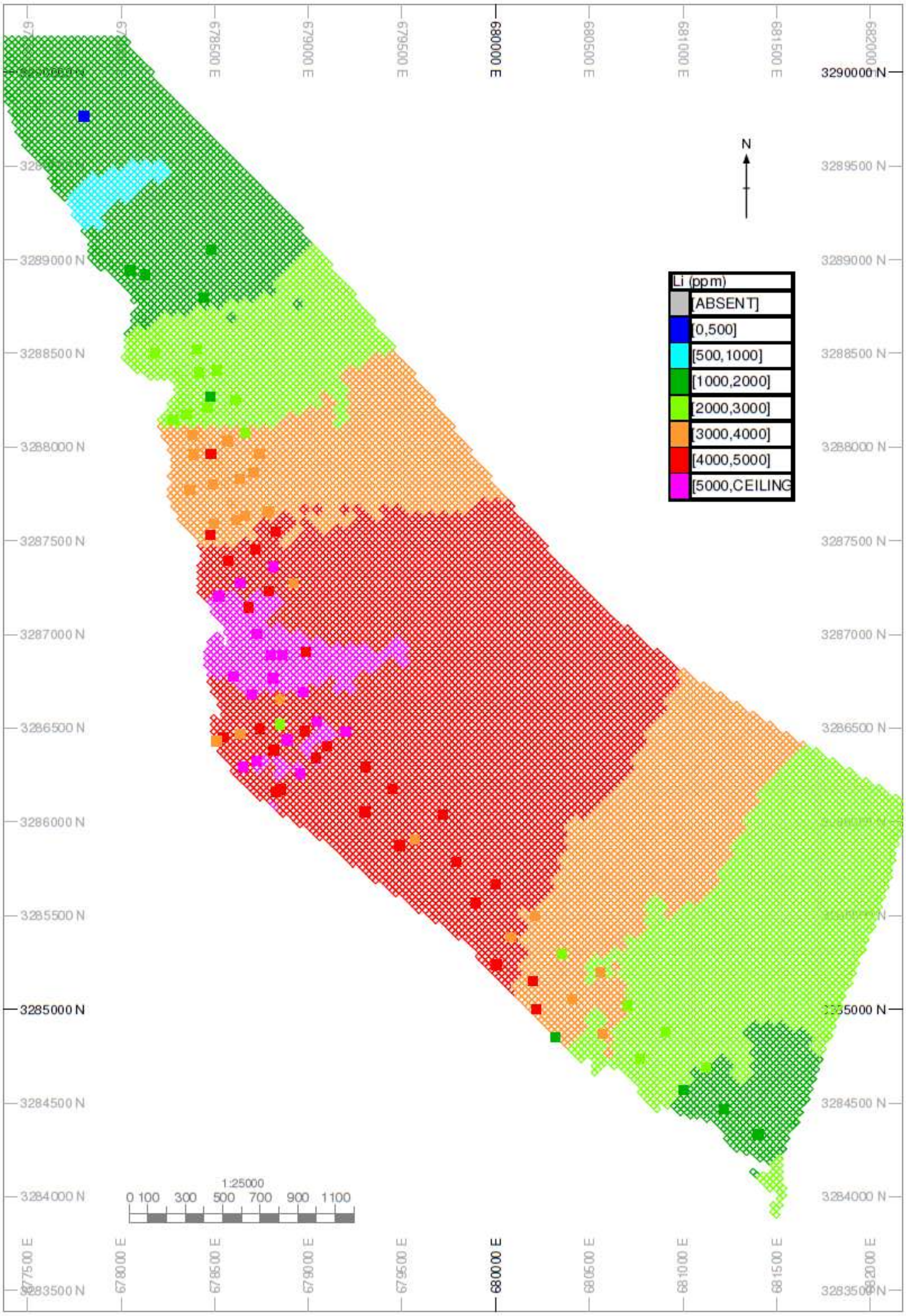


Figure 14.12.3: Lower Clay Li Block Model Validated Against Composited Drillhole Data (Squares)

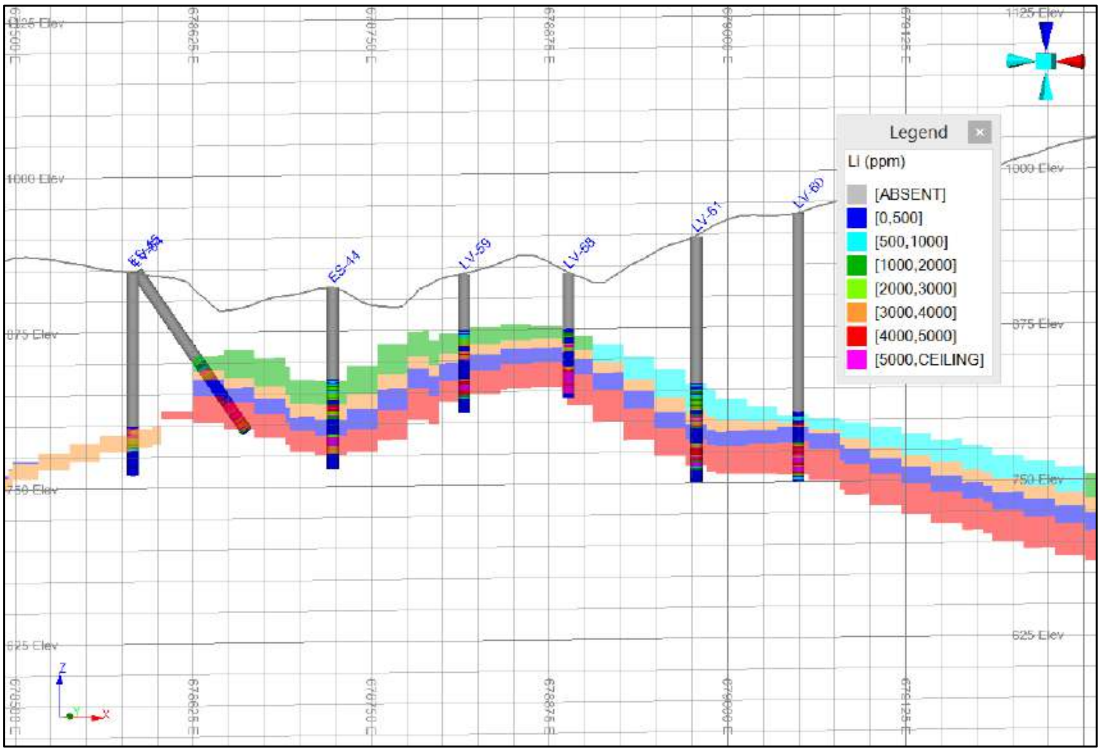


Figure 14.12.4: W-E Cross-Section Through Southern La Ventana Area Showing Li Block Model Grades and Original (Un-Composited) Drillhole Grades

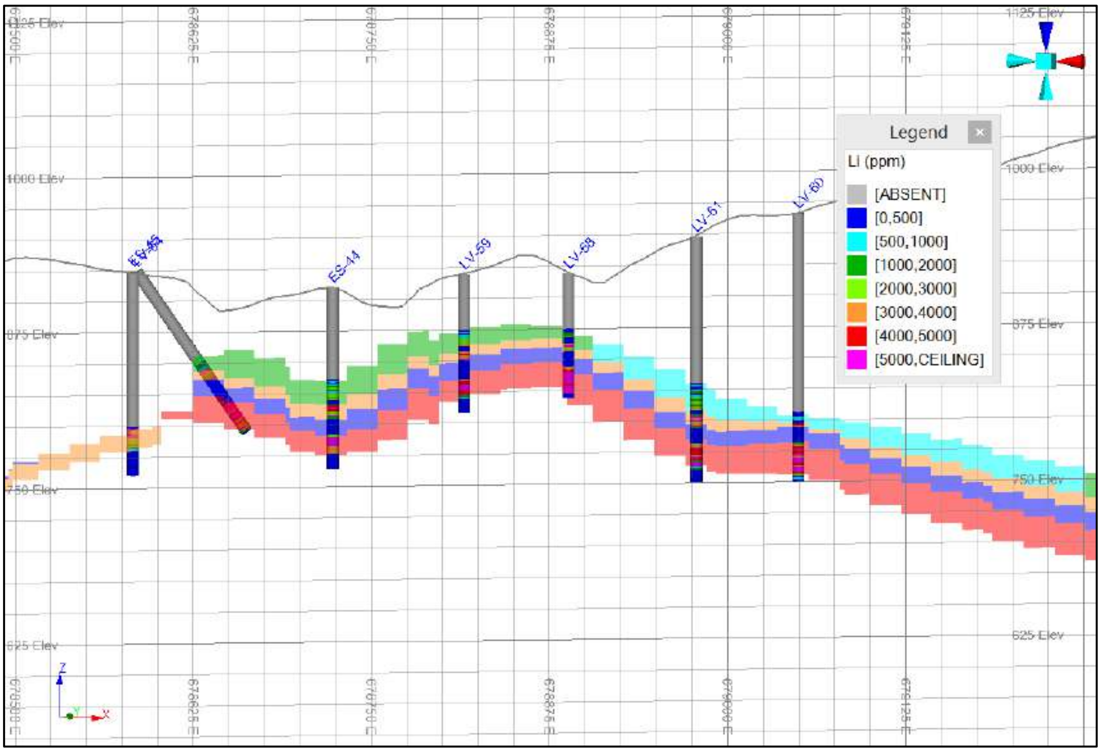


Figure 14.12.5: W-E Cross-Section Through Northern La Ventana Area Showing Li Block Model Grades and Original (Un-Composited) Drillhole Grades

14.12.3 Swath Plots

Visual validation of composite samples grades against the interpolated 2D block grades was undertaken to assess the performance of the estimation in the main fault block were sufficient data exists to conduct a useful assessment of estimation quality. The resultant swath plots for Li are presented in Figure 14.12.6 to Figure 14.12.9. Swath plots have been created using data from the rotated block model.

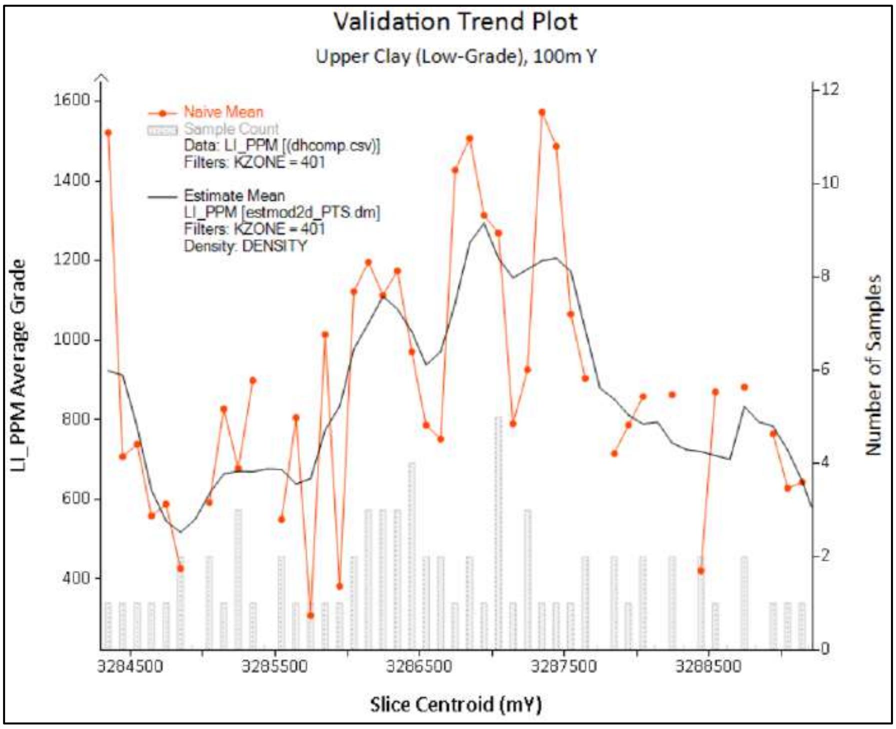


Figure 14.12.6: Northing (Y) Swath Plot for Upper Clay (Low-Grade)

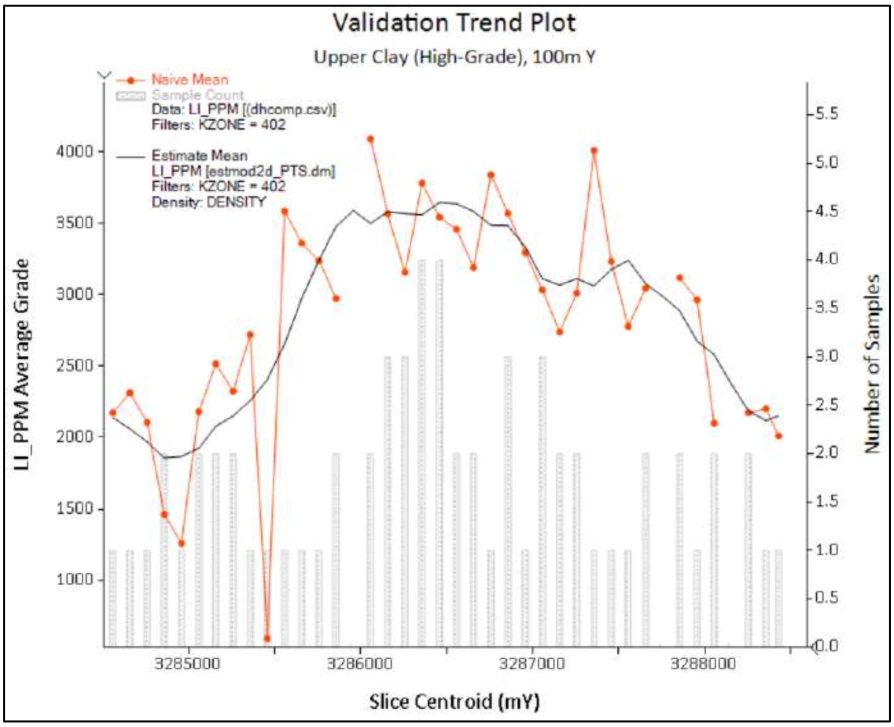


Figure 14.12.7: Northing (Y) Swath Plot for Upper Clay (High-Grade)

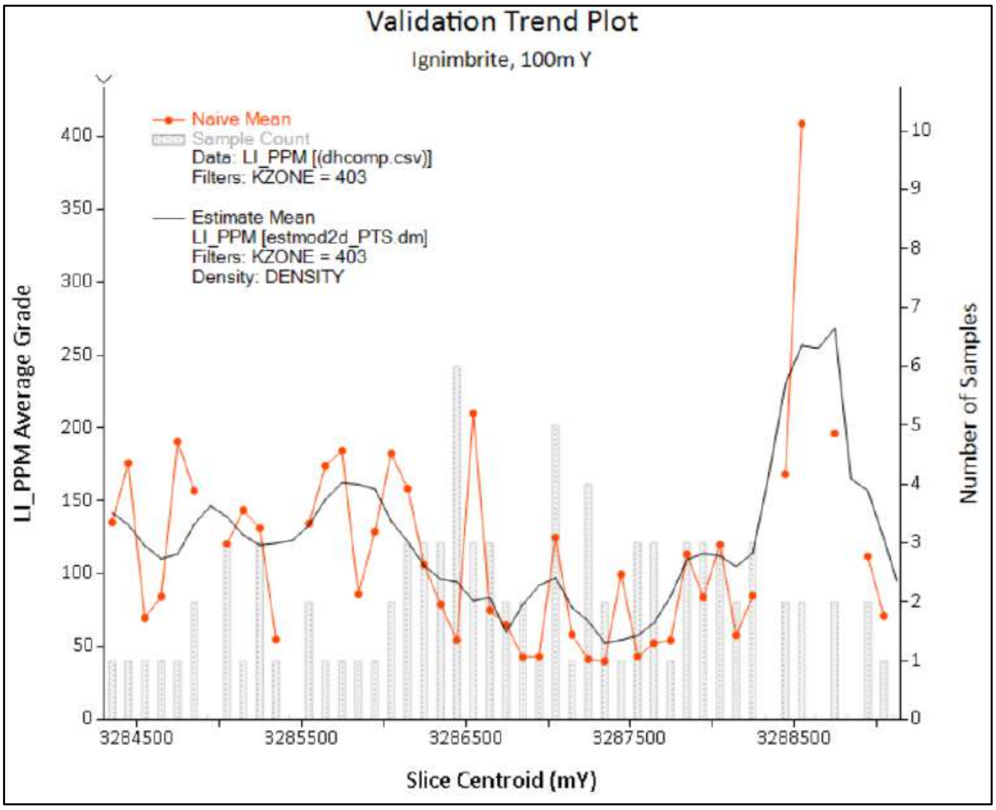


Figure 14.12.8: Northing (Y) Swath Plot for Ignimbrite

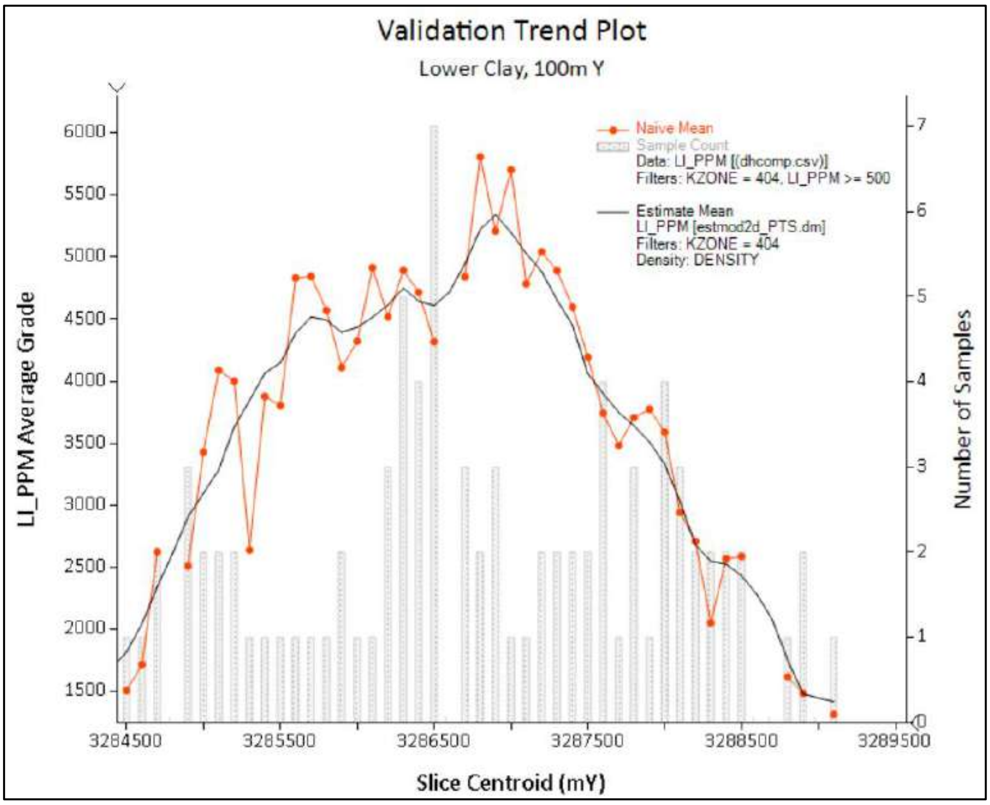


Figure 14.12.9: Northing (Y) Swath Plot for Lower Clay

14.12.4 Statistical Validation

Classical statistics were calculated for the estimated 2D and 3D block grades and compared with the composited drillhole statistics used in the estimation process. The absolute difference in the composite and block model means was considered immaterial for all mineralised domains. The comparison between the composites and OK and IDW^2 interpolated 3D block model statistics is shown in Table 14.12.1 for Li and Table 14.12.2 for K.

The difference in mean block grade between the OK and IDW interpolations is typically <10% and shows that the deposit is not significantly sensitive to estimation technique and that OK has not introduced a bias compared to the input composite sample data.

Table 14.12.1: Comparison Statistics for Li Composites Versus 3D Block Model Grades

Domain	Mean Li (ppm) composite grade	Mean Li (ppm) Block model grade (OK)	Mean Absolute Difference (%)	Mean Li (ppm) Block model grade (IDW)	Mean Absolute Difference (%)
Upper Clay (LG)	892	844	-6%	847	-5%
Upper Clay (LG)	2,937	2,882	-2%	2,882	-2%
Ignimbrite	97	121	20%	121	20%
Lower Clay	3,817	3,765	-1%	3,769	-1%

Table 14.12.2: Comparison Statistics for K Composites Versus 3D Block Model Grade

Domain	Mean K (%) composite grade	Mean K (%) Block model grade (OK)	Mean Absolute Difference (%)
Upper Clay (LG)	0.5	0.5	-8%
Upper Clay (LG)	1.1	1.1	-2%
Ignimbrite	0.2	0.2	6%
Lower Clay	1.6	1.6	-1%

14.13 Mineral Resource Classification

14.13.1 Introduction

Block model tonnage and grade estimates for the Project have been classified according to the terminology and definitions given in the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Martin Pittuck, CEng, MIMMM, FGS, who is a Qualified Person as defined by the Canadian National Instrument 43-101 and the companion policy 43-101CP.

Mineral Resource classification is a subjective concept, which considers the geological confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the grade estimates.

SRK is satisfied that the geological modelling honours the current geological information and knowledge and extrapolates this reasonably. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired by diamond core drilling on sections spaced at approximately 200 m, and associated drill core samples on 1.5 m intervals. In many places, the drilling combined with satellite imagery and mapped outcrop gives high confidence in the geometry of the geological features controlling grade and the grade trends themselves.

SRK has also considered sampling quality, representivity and accuracy of historical and recent assaying and density determinations. The QA/QC results suggest an acceptable level of quality for the assays; in particular, the results from the quarter core submissions to an umpire laboratory support the accuracy of the assays at the primary laboratory based on numerous batches representing the major drill phases undertaken. The standards used to date have demonstrated reasonable consistency at the primary laboratory although the grade levels were too low or too high to represent the majority of samples in the model.

SRK considers that the number of density determinations and the method used gives an accurate estimate of dry in situ bulk density.

Overall, it is SRK's view that the recent data is of a sufficient quality for the reporting of Measured, Indicated and Inferred category of Mineral Resources. The parts of the model that have been excluded from the Mineral Resource are characterised by one or more of poor or no sample coverage and being too thin, deeply buried or low grade to be realistically mined by open pit.

14.13.2 Geological and Grade Continuity

The deposit has been modelled consistently throughout the Project area as a single stratigraphic package containing two units of lithium enriched clays separated by an ignimbrite unit. Within the eastern portion of the deposit in fault block 4, the Upper Clay Unit is observed to have a stratification of Li grade, with high grades at the base and lower grades in the upper portion. This grade distribution has been accounted for during the wireframing and estimation process. The clay units have also been offset in places by faults, dividing the deposit into five fault blocks, with majority of the modelled deposit falling in a strike extensive fault block tending northwest-southeast. The remaining fault blocks are less extensive on strike and are based on limited drilling at present, thus reducing the confidence in the modelling in these areas.

SRK considers there may be greater geological complexity than has been currently been interpreted particularly in less well drilled or/ mapped areas, specifically:

- There may be more faults than currently modelled
- There is lower confidence in the geometry of faults in the southern area
- Thickness is thinner and more variable towards the north and south extents
- Dip and orientation of the deposit in the western fault blocks is less well defined.

Grades have been composited across the thickness of each clay unit domain which has resulted in very good grade continuity in the data used for the block model estimate.

Overall, it appears that the clay zones identified at the project are of a reasonably low geological complexity and the hanging wall and footwall contacts are easily defined. Localised complexities in the geology however arise in the narrow internal banding, as such, and, based on the current level of data supporting the geological model, the associated risk relating to the internal continuity of layers is considered to be low.

SRK is aware that the lithium deportment in the clay units is such that an initial screening beneficiation process is likely to be used to produce an upgraded product by removing relatively coarse boulders and cobbles of chert and calcite. These lumps and nodules have very low lithium grades other than the clay coating they may carry. The proportion of such coarse barren material in the clay units has not been studied in the drillhole data and it is an important variable that may be less continuous than the composited grades modelled to date.

14.13.3 Data Quality

SRK considers the QA/QC protocols that have been put in place to monitor sample preparation quality and laboratory accuracy and precision are sufficiently robust to be confident in the underlying data used for grade estimation.

There is a systematic process of sample preparation at the facilities on site. Regular submission of standards into the sample stream has tracked the performance of the primary laboratory over time albeit using grades which do not fully represent the clay units. Samples sent to an umpire laboratory have confirmed the accuracy of primary laboratory assays but this has not happened consistently through the duration of the program to date.

Validation checks of standards are within acceptable reporting limits and duplicate field samples show a strong correlation to the original sample. Minor periodic drift has been recorded within the reference standard and SRK would recommend this is reported to the certified laboratory and monitored closely.

With respect to the density determinations, SRK considers that the current procedure provides a robust measure of the dry density.

14.13.4 Results of the Geostatistical Analysis

The data used in the geostatistical analysis resulted in suitably reliable variograms for all zones in Block 4 that allowed the nugget effects, sills and ranges to the determined. The variography allowed the determination of reasonable search distances to be used through the estimation process.

14.13.5 Quality of the Estimation

The validation tools utilised for the Project show that the input data used to estimate the model is replicated in the estimation. The block model grades are smoothed around the input composites and the mean grades of the block model and composites are comparable for all modelled zones.

14.13.6 SRK Classification Approach

The block model has been classified into Measured, Indicated and Inferred Mineral Resource categories in the Upper and Lower clay units.

The Measured Mineral Resources have been limited to the area delineated during the 2016 infill drilling program. This program produced relatively close-spaced drilling data to confirm the geological and grade continuity within the area previously outlined as Mineral Reserves in the PFS. The Measured blocks estimated in run one of the grade estimation routine and where on cross section, there are at least three to four points of geological evidence from mapping, drilling and trenching. The approximate drillhole spacing in areas classified as Measured is 100 - 200 m.

The Indicated Mineral Resources have been limited to one broad area which was estimated in run one of the grade estimation routine, there are at least three points of geological evidence from mapping and drilling. The approximate drillhole spacing in areas classified as Indicated Mineral Resources is 200 - 300 m.

Inferred Mineral Resources have been limited to areas where there is a wider spacing of drilling and outcrop; these areas extend some 200 m beyond the deepest drillhole intersection.

There are large areas of SRK's 3D geological model that have been extrapolated beyond the Mineral Resource that remain unclassified, the intention being to facilitate drillhole planning should that be desirable in the future. Figure 14.13.1 shows the full classified model in terms of Measured, Indicated, Inferred and unclassified material.

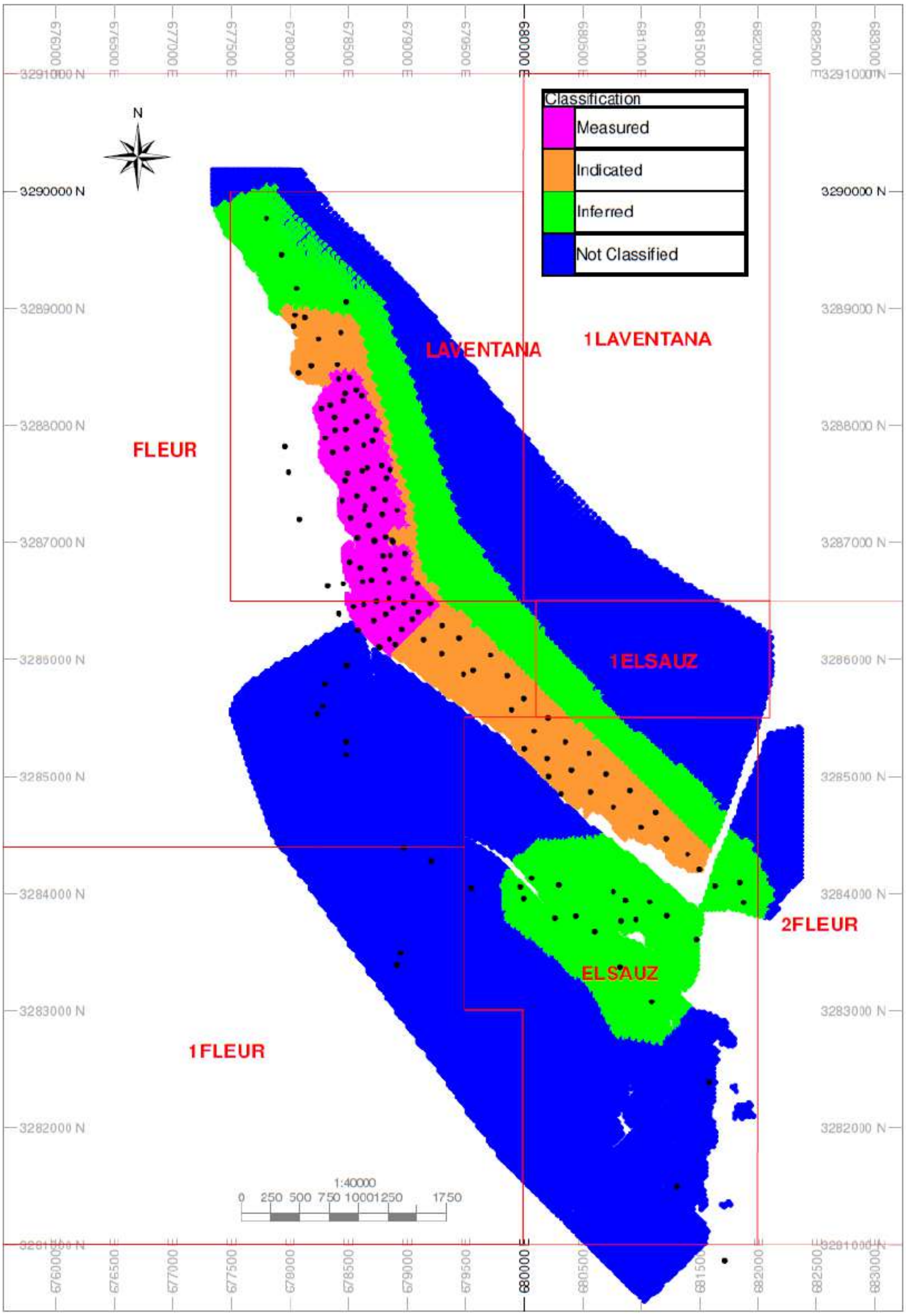


Figure 14.13.1: Plan View Showing Classification of the Sonora Lithium Project

14.14 Mineral Resource Cutoff Grade and Practical Limits

A Mineral Resource, according to the CIM Guidelines, should show 'reasonable prospects for economic extraction' which generally implies that the tonnage and grade estimates meet certain economic thresholds by reporting using an appropriate cutoff grade and to a practical depth below surface taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that portions of the Project are amenable for open pit extraction.

14.14.1 Lithium Price

The basis of the lithium price used for this Mineral Resource estimate is outlined in Section 19 SRK believes it is reasonable to expect prices, technology and costs in the future to be different from what they are today, more so in the long term than in the short term. The Mineral Resource is a long term / strategic assessment of a mineral asset and we believe a different approach to deriving cutoff grade for Mineral Resources (compared with that used for Ore Reserves) is justified given that conditions may become more favourable in the long term at which point it may make sense to develop the asset further.

There is additional merit in this case given the price increases forecast by SignumBox in the medium to long term and the potential to add a credit from potassium sulfate.

In order to effect a lower cutoff grade for the Mineral Resource, SRK has used a battery grade lithium carbonate price of $ 14,300 / t lithium carbonate (which represents a premium of 30% above the $ 11,000 / t used for the Mineral Reserve). SRK's cutoff grade, when combined with cost and recovery information being considered in the Feasibility Study is 1,000 ppm (0.1%) Li.

14.14.2 SRK Mineral Resource Pit Optimisation and Cutoff Grade Analysis

SRK used the lithium price assumptions described above in addition to mining and processing costs and efficiencies provided by Bacanora's FS team, to evaluate the Measured, Indicated and Inferred parts of the model that could be "reasonably expected" to be mined from an open pit. Revenue from potassium was not specifically taken into account but this opportunity is one of the long term assessment factors on which SRK's cutoff grade has been based.

The reader is cautioned that the results from the pit optimisation are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent Mineral Reserves; the Mineral Resource is inclusive of the Mineral Reserves reported in the Feasibility Study.

The optimisation parameters are given in Table 14.14.1. The resultant pit shell used to limit the resource is shown in green in Figure 14.14.1. A minor increase in processing cost was used for the Mineral Reserve estimate but not incoporated into the Mineral Resource due to an immaterial change in the resulting pit shell and cutoff grade analysis.

In addition, SRK calculated an updated marginal cutoff grade using the updated prices, costs and efficiencies provided by the FS team. This resulted in the same cutoff grade (1,000 ppm Li) as was used for the PFS.

Table 14.14.1: Pit Optimisation and Cutoff Grade Parameters

Parameters	Units	Value
Pit Slope		
Footwall	(Deg)	42
Hangingwall	(Deg)	42
Mining Factors		
Dilution	(%)	0.0
Recovery	(%)	100
Processing		
Recovery Li	(%)	75
Operating Costs		
Processing	($/$t_{LCE}$)	3,297
General & Administration (including re-handling)	($/$t_{LCE}$)	210
Selling Cost (Royalty)	(%)	3
Metal Price*		
Lithium Carbonate	($/$t_{LCE}$)	14,300
Cutoff Grade		
Marginal Cutoff Grade (in situ)	(ppm Li, rounded)	1,000

* Every 1 unit of lithium metal is equivalent to 5.323 units of Li_2CO_3 (lithium carbonate)

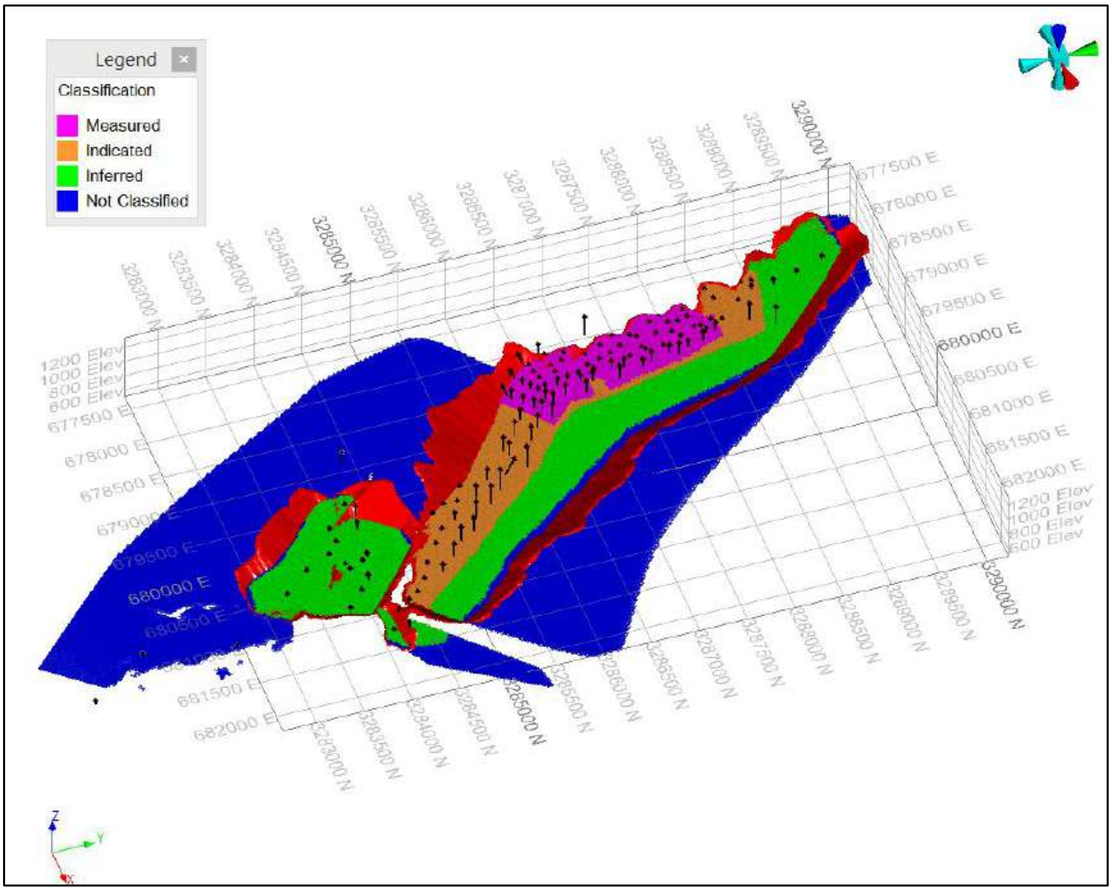


Figure 14.14.1: Oblique View Showing Classified Material Within the Resource Pit Shell (Red)

14.15 Mineral Resource Statement

The Mineral Resource is based on exploration results from mapping drilling and trenching made available to SRK on the 05 September 2016 and technical economic inputs received from the Bacanora team on 13 December 2017.

The Mineral Resource is the total for the Project; with 93%, 75% and 85% of metal in the Measured, Indicated and Inferred Mineral Resource categories attributable to Bacanora, respectively.

The Mineral Resource statement represents the material which SRK considers has reasonable prospects for eventual economic extraction taking into account cutoff grade and stripping ratio by means of a pit optimisation. Table 14.15.1 shows the resulting Mineral Resource Statement for the Sonora project, with increased detail in Table 14.15.2. The statement has been classified in accordance with the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves (May, 2014) and has been reported in accordance with NI 43-101, by the Qualified Person, Mr Martin Pittuck (CEng., MIMMM, FGS). Mr Pittuck is a consultant who is independent of Bacanora.

SRK is not aware of any additional factors (environmental, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource estimate.

The tonnage and grade of Inferred Mineral Resources are uncertain and there has been insufficient exploration to define these Inferred Mineral Resources as an Indicated or Measured Mineral Resource. It is reasonable to expect that the majority of Inferred Resources could be upgraded to Indicated with continued exploration.

Table 14.15.1: SRK Mineral Resource Statement as of 05 December 2017

Classification	Tonnes (Mt)	Grade		Contained Metal		
		Li (ppm)	K (%)	kt Li	kt LCE	kt K
Measured	103	3,480	1.5	359	1,910	1,532
Indicated	188	3,120	1.3	588	3,130	2,460
Meas + Ind	**291**	**3,250**	**1.4**	**947**	**5,038**	**3,993**
Inferred	268	2,650	1.2	710	3,779	3,101

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.
2. Mineral Resources are reported inclusive of Mineral Reserves.
3. The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
4. Mineral Resources are reported on 100 percent basis for all Project areas.
5. SRK assumes the Sonora Lithium deposit to be amenable to surface mining methods. Using results from initial metallurgical test work, suitable surface mining and processing costs, and forecast LCE price SRK has reported the Mineral Resource contained within an optimistic open pit shell and above a cutoff grade of 1,000 ppm Li.
6. SRK completed a site inspection of the deposit by Mr. Martin Pittuck, CEng, MIMMM, FGS, an appropriate "independent qualified person" as such term is defined in NI 43-101.
7. LCE is the industry standard terminology for, and is equivalent to, Li_2CO_3. 1 ppm Li metal is equivalent to 5.323 ppm LCE / Li_2CO_3. Use of LCE is to provide data comparable with industry reports and assumes complete conversion of lithium in clays with no recovery or process losses.
8. Mt = million tonnes (metric).
9. kt = thousand tonnes (metric).

Table 14.15.2: Detailed SRK Mineral Resource Statement

Classification	Concession	Owner	Geological Unit	Tonnes (Mt)	Grade		Contained Metal		
					Li (ppm)	K (%)	kt Li	kt LCE	kt K
Measured	La Ventana	Minera Sonora Borax (99.9% Bacanora)	Lower Clay	53	4,110	1.8	217	1,153	950
			Upper Clay	27	2,210	0.9	59	316	240
	El Sauz	Mexilit (JV-1) (70% Bacanora)	Lower Clay						
			Upper Clay						
	Fleur		Lower Clay	12	4,750	2.0	58	311	241
			Upper Clay	11	2,190	0.9	24	130	101
	El Sauz1		Lower Clay						
			Upper Clay						
Measured Total			**Combined**	**103**	**3,480**	**1.5**	**359**	**1,910**	**1,532**
Indicated	La Ventana	Minera Sonora Borax (99.9% Bacanora)	Lower Clay	29	2,720	1.3	78	417	377
			Upper Clay	8	2,180	0.9	18	95	74
	El Sauz	Mexilit (JV-1) (70% Bacanora)	Lower Clay	57	2,950	1.3	167	888	714
			Upper Clay	19	2,050	0.8	38	202	152
	Fleur		Lower Clay	44	4,490	1.8	195	1,041	803
			Upper Clay	27	2,670	1.0	73	389	265
	El Sauz1		Lower Clay	4	4,080	1.7	15	79	61
			Upper Clay	2	2,280	0.9	4	19	14
Indicated Total			**Combined**	**188**	**3,120**	**1.3**	**588**	**3,130**	**2,460**
Meas+Ind	La Ventana	Minera Sonora Borax (99.9% Bacanora)	Lower Clay	81	3,620	1.6	295	1,570	1,327
			Upper Clay	35	2,200	0.9	77	411	315
	El Sauz	Mexilit (JV-1) (70% Bacanora)	Lower Clay	57	2,950	1.3	167	887	714
			Upper Clay	19	2,050	0.8	38	202	152
	Fleur		Lower Clay	56	4,550	1.9	254	1,352	1,044
			Upper Clay	39	2,530	1.0	98	519	367
	El Sauz1		Lower Clay	4	4,080	1.7	15	79	61
			Upper Clay	2	2,280	0.9	4	19	14
Measured+Indicated Total			**Combined**	**291**	**3,250**	**1.4**	**947**	**5,038**	**3,993**
Inferred	La Ventana	Minera Sonora Borax(99.9%Bacanora)	Lower Clay	81	3,220	1.5	260	1,382	1,184
			Upper Clay	46	2,200	0.9	102	541	417
	El Sauz	Mexilit (JV-1) (70%Bacanora)	Lower Clay	75	1,720	0.8	130	690	624
			Upper Clay	9	1,850	0.7	16	87	66
	Fleur		Lower Clay	18	4,230	1.8	78	414	327
			Upper Clay	12	2,750	1.0	34	181	122
	El Sauz1		Lower Clay	18	3,990	1.6	70	374	285
			Upper Clay	8	2,510	0.9	21	110	75
Inferred Total			**Combined**	**268**	**2,650**	**1.2**	**710**	**3,779**	**3,101**

14.16 Comparison with Previous Estimate

The previous MRE undertaken by SRK in April 2016 is detailed in Section 6.3.3.

14.16.1 Geological Modelling Update

The infill drilling has enabled Measured Mineral Resources to be declared for the first time. The drilling generally verified the previous interpretation although some drilling designed to improve fault location accuracy resulted in an increase in the mineralised area in a particularly thick and high grade part of La Ventana, overall increasing the modelled volume by 10%.

Two examples of the change in geological modelling are presented below. Figure 14.16.1 shows the 2017 interpretation of the lithological units, this compares to Figure 14.16.2, which shows the April 2016 interpretations but also the new drilling data not used in the previous MRE. This section shows the most pronounced changes in interpretation, with a more folded structure than originally anticipated but similar overall tonnage. Figure 14.16.3 and Figure 14.16.4 show another comparison of geological modelling, with a similar overall simple structure but with higher tonnage in the 2017 update.

14.16.2 Pit Optimisation Parameters

The updated Mineral Resource statement was reported using an open pit shell above a marginal cutoff grade of 1,000 ppm Li, which replicates the reporting procedure for the PFS. The pit shell has been driven deeper by the increased Li selling price, which has allowed for additional Lower Clay material to be included down dip up to the Inferred classification limit, which has not changed position in this update. The overall contained metal reported has increased from 7.2 Mt to 8.8 Mt (+22%) LCE, which is a combination of the increased tonnage (+33%) and decreased grade (-8%) resulting mainly due to Li selling price.

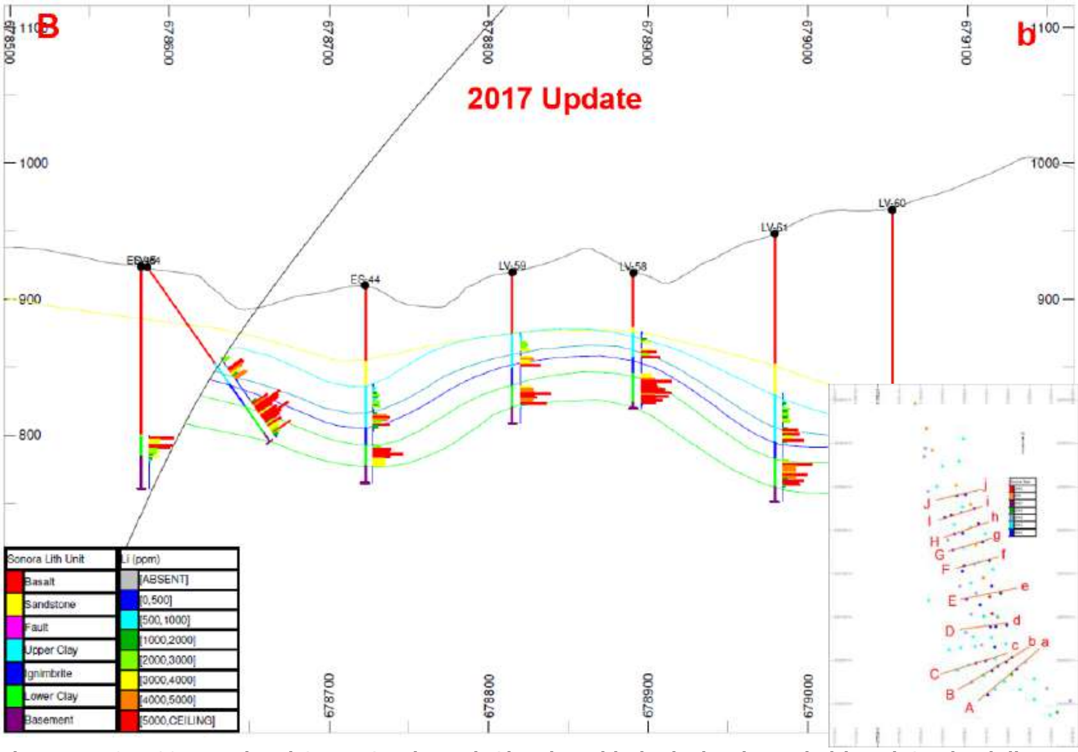


Figure 14.16.1: 2017 Updated Cross-Section B-b Showing Lithological Units and Li (ppm) Grades (All Holes Except ES-44 and ES-45 [Far Left] Drilled in 2016)

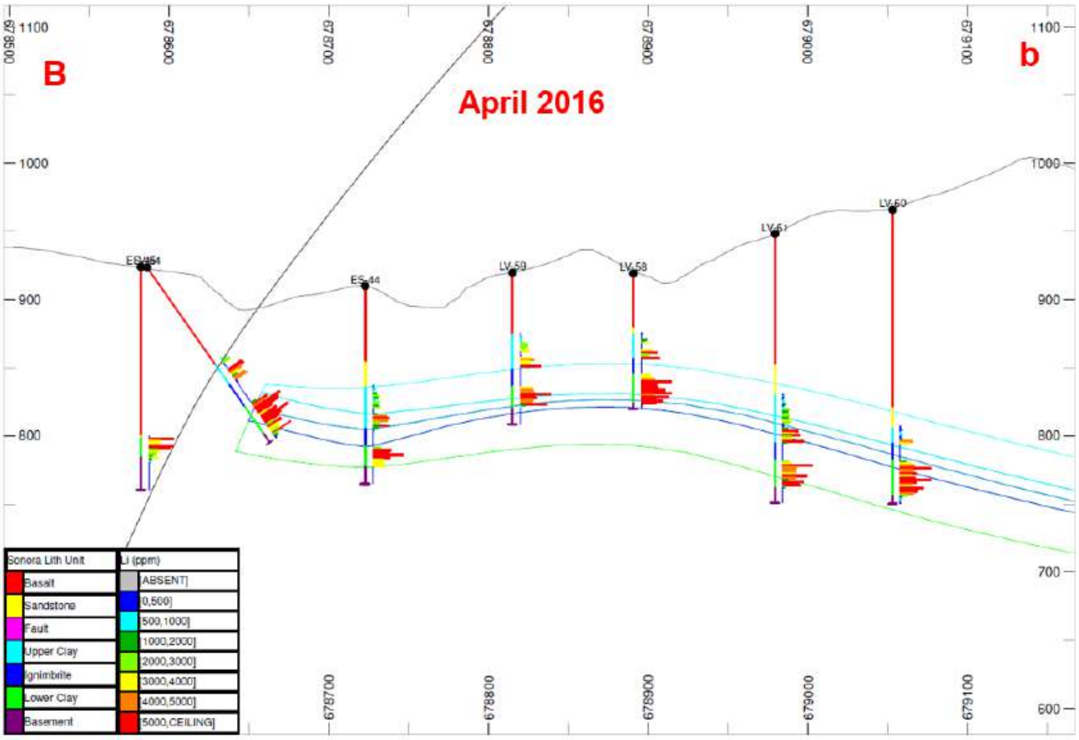


Figure 14.16.2: 2016 Cross-Section B-b Showing Lithological Units and Li (ppm) Grades

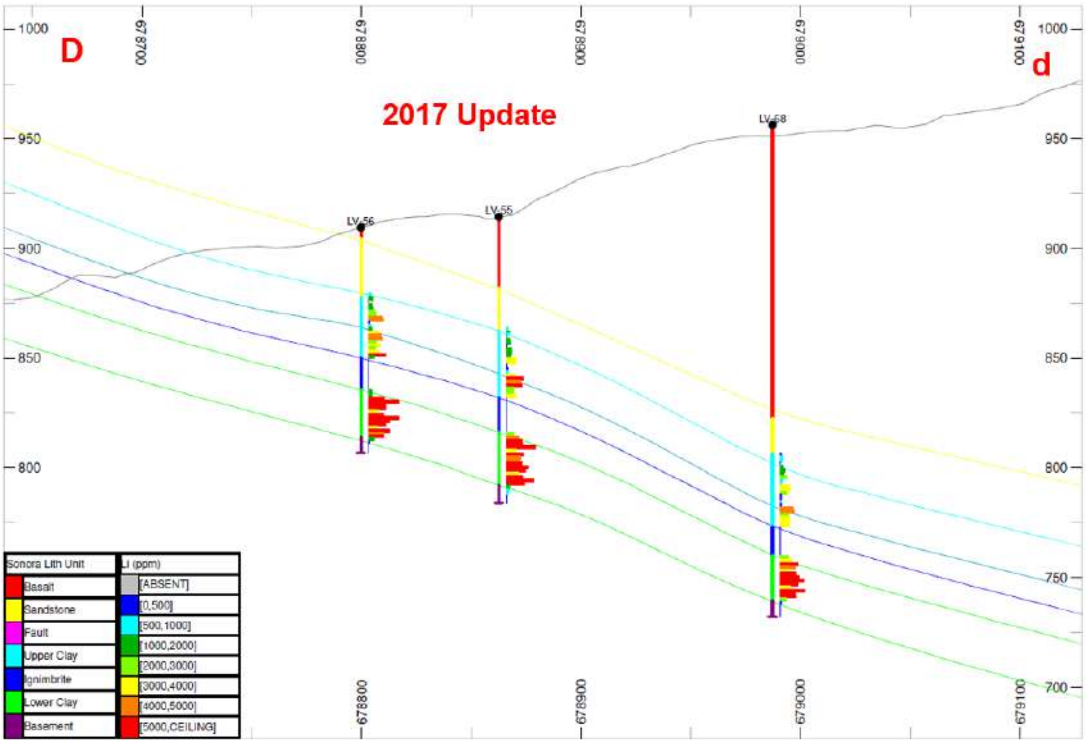


Figure 14.16.3: 2017 Updated Cross-Section D-d Showing Lithological Units and Li (ppm) Grades (All Holes on Section From 2016 Drilling)

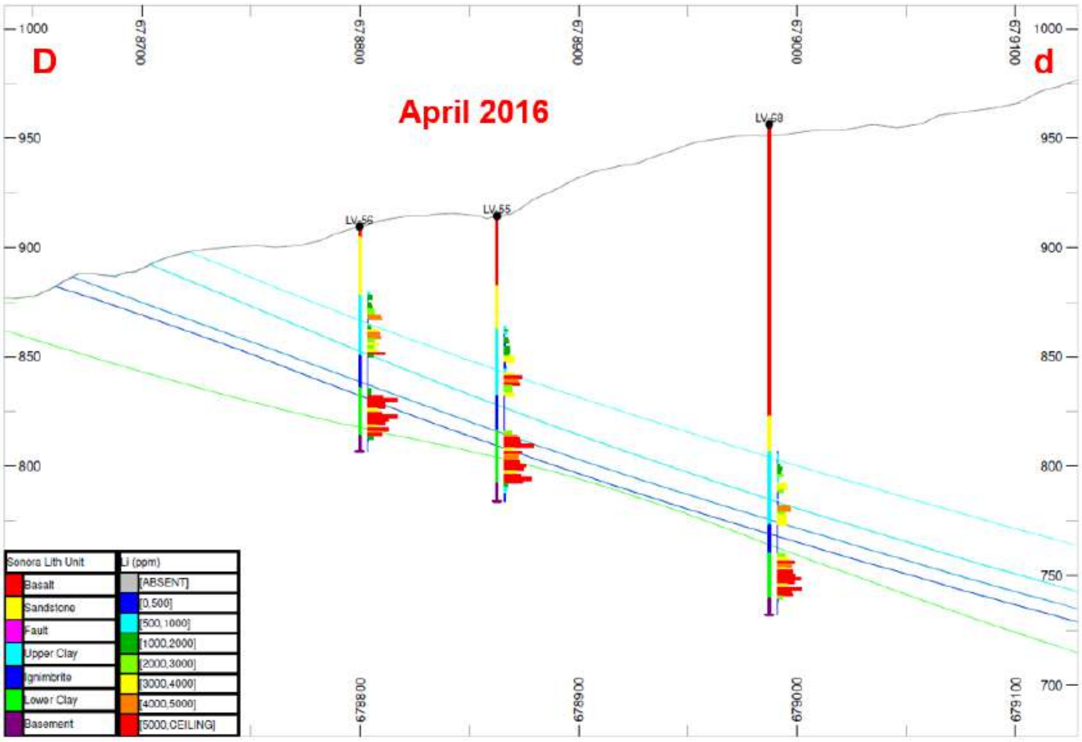


Figure 14.16.4: 2016 Cross-Section D-d Showing Lithological Units and Li (ppm) Grades (All Holes on Section From 2016 Drilling)

14.17 Grade Sensitivity Analysis

SRK has completed a number of check block model estimates on the deposit using a variety of parameters and the resultant models produced similar estimates.

The Mineral Resources stated in this report is sensitive to the selection of the reporting cutoff grade. To illustrate this sensitivity, the block model quantities and grade estimates within the conceptual pit used to constrain the Mineral Resources are presented in Figure 14.17.1 (M+I+I or MII = Measured, Indicated and Inferred combined).

These figures are only presented to show the sensitivity of the block model within the optimised pit to the selection of cutoff grade.

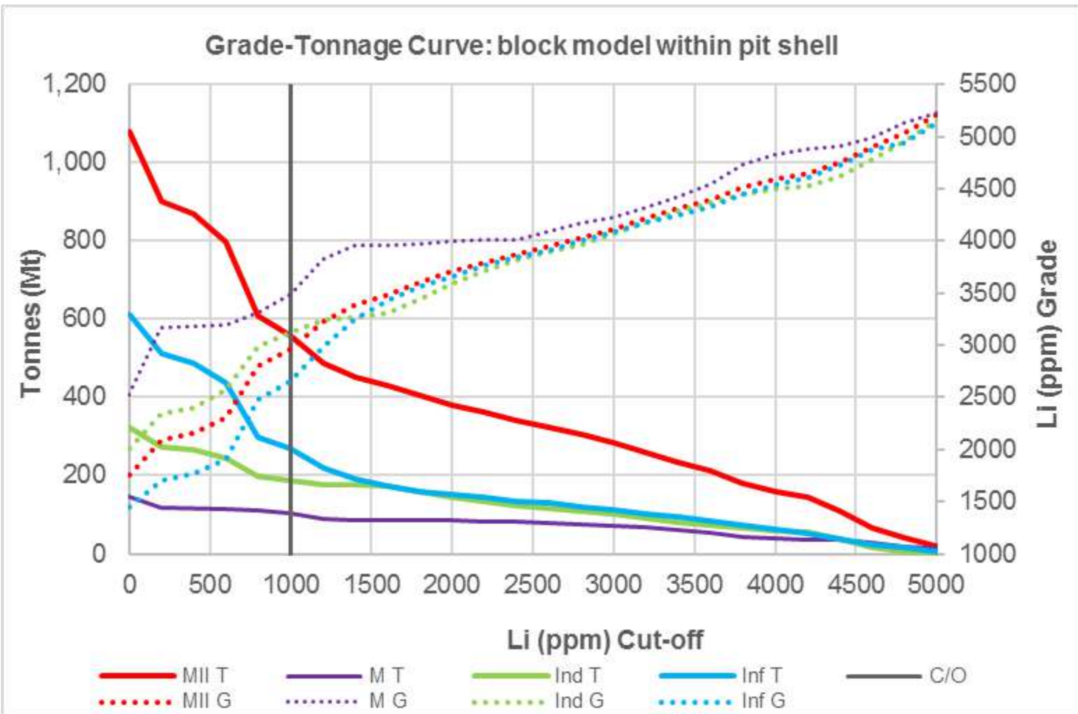


Figure 14.17.1: Grade-Tonnage Curve for All Classified Material Within Pit (Black Line = 1,000 ppm Li Cutoff Grade Used for Reporting Mineral Resources)

15 MINERAL RESERVE ESTIMATION

15.1 Introduction

The mineral reserves for the Sonora Lithium project are contained within an open pit design based on the current knowledge of the deposit, geotechnical information, operating costs, recoveries and the selling price of the lithium carbonate (Li_2CO_3). The mineral reserves are the sum of the Proven and Probable classification based on the classifications assigned to the resource model described in previous sections of this report. Table 15.1.1 is a summary of the mineral reserve at a 1500 ppm lithium cutoff grade. The mineral reserves are the diluted mineral reserves based on a dilution of 100 cm at the contacts between the lithium clays and the adjacent lithologies with the grade of the adjacent lithology. The mineral reserves use the terminology, definitions and guidelines given in the CIM Standards on Mineral Resource and Mineral Reserves (May 2014). Herb Welhener, vice president of Independent Mining Consultants, Inc. (IMC) is the qualified person for the mineral reserve reporting.

Table 15.1.1: Open Pit Mineral Reserve

Category	Ore > = 1,500 ppm Li				Waste (kt)	Total (kt)	Waste : Ore Ratio	% LCE to Bacanora
	Ore (kt)	Li (ppm)	LCE (kt)	K (%)				
Proven	80,146	3,905	1,666	1.64				93.03%
Probable	163,662	3,271	2,849	1.36				74.63%
Total	**243,808**	**3,480**	**4,515**	**1.45**	**2,298,701**	**2,542,509**	**9.43**	**81.42%**

Notes

1. kt = tonnes x 1000
2. LCE = lithium carbonate equivalent

15.2 Inputs to Mineral Reserve

The inputs for defining the mineral reserve pits are shown in Table 15.2.1 and were provided by SRK, Ausenco, Bacanora Minerals and IMC. The process and G&A operating costs along with the total recovery of lithium were provided early in the project based on previous work and are not the final results of the Feasibility Study. The resource block model was developed by SRK and is described in previous report sections. The process and G&A costs, plant recovery and selling price of lithium carbonate were provided by Ausenco and Bacanora Minerals. The process and G&A costs were given as cost per tonne of lithium carbonate assuming a production rate of 35,000 t/y of lithium carbonate. The mining royalty by the Mexican government of 7.5% was not included in the economics for the pit definition algorithm used for the mineral reserve pit design. As a check, a subsequent pit definition run was made which included the 7.5% royalty and a one half of one percent difference was noted in the two pit shells. The final mineral reserve pit design has less tonnage above cutoff than either of the pit shells used to guide the final pit design. The geotechnical study and the recommended pit wall slope angles were completed by Ausenco (La Ventana Pit Slope Design Report, December 2016). The mining costs were provided by IMC based on the cost estimates developed for this FS.

Table 15.2.1: Inputs for Definition of Mineral Reserve Pits

Parameter	Units	Amount
Process Operating Cost	$/t LCE	$3,297
G&A Operating Cost	$/t LCE	$198
Total Operating Cost	$/t LCE	$3,495
Sales Price	$/t LCE	$11,000
Li Total Recovery:	%	75.0
Lithium % of Li_2CO_3	%	18.79
Royalty	%	3
Mining Recovery Factor	%	100
Mining Cost	$/t	1.75
Additional Mining Cost Below 900 Elevation	$/t per 10 m bench	0.02
Ore Stockpile Re-handle Cost	$/t ore	$0.25
Discount Rate	% per 10 m bench	0.5
Overall Slope Angle	Degrees	42

15.3 Net Value Calculation

A net value per tonne of mill feed for each block with a lithium grade in the resource mode was calculated based on the costs and recoveries shown Table 15.2.1. The higher the lithium grade is, the more product per tonne of mill feed can be made and a lower cost or higher net value per tonne of mill feed is achieved. . A net value approach was used for the pit limit determination as the cost per tonne of mill feed is dependent on both the beneficiation stage (feed tonnage driven) and the hydrometallurgical stage (Li grade dependent). The split between these two costs was not completely determined at this time in the project. The average process plus G&A cost ($3,495/t Li2CO3) compares well to the resultant Feasibility Study combined cost of $3,475/t Li2CO3. The net value per tonne of mill feed was assigned to the blocks in the model and used with a floating cone algorithm to determine the mineral reserve pit limits. Table 15.3.1 shows the net value calculation for a range of lithium grades.

Table 15.3.1: Net Value Calculation

Parameter	4,000 ppm Li	3,000 ppm Li	2,000 ppm Li	1,800 ppm Li	1,500 ppm Li
Sale Price Lithium Carbonate	**$11,000**	**$11,000**	**$11,000**	**$11,000**	**$11,000**
Royalty	3%	3%	3%	3%	3%
Sale Price after royalty deduction	$10,670	$10,670	$10,670	10670	$10,670
Process + G&A cost/t Li_2CO_3	$3,495	$3,495	$3,495	$3,495	$3,495
Realized price/t Li_2CO_3 sold	$7,175	$7,175	$7,175	$7,175	$7,175

Parameter	4,000 ppm Li	3,000 ppm Li	2,000 ppm Li	1,800 ppm Li	1,500 ppm Li
Tonnes Li_2CO_3 recovered /t mill feed ((Li x 0.75)/1000000)/0.1879	0.0.01597	0.01197	0.00798	0.00718	0.00599
Net value / mill feed (before mining costs), $/t	$114.56	$85.92	$57.28	$51.55	$42.96

15.4 Pit Design

The open pit designs are based on 10 m mining benches, 20 m wide haul roads (includes allowance for berms and ditches) and 42 degree inter-ramp slope angle on the hanging wall (east) side of the pits. The lithium clay beds dip to the east and there are no haul ramps on the east wall so the inter-ramp slope angle and overall slope angle are the same at 42° based on the recommendations in the Ausenco report.

Table 15.4.1 presents the open pit diluted mineral reserves by the major lithology units. The dilution is from the adjacent units, for example the lower clay dilution would come from 100 cm of the block grade of the ignimbrite above and the basement unit below. Table 15.4.2 presents the open pit diluted mineral reserves by the three claim groups. Bacanora owns 99.9% of the La Ventana claim group and 70% interest in each of the El Sauz and Fleur claim groups. Figure 15.4.1 illustrates the reserve pit geometry. The red outline in the north central area of the reserve pit represents the portion of the reserve that is mined during the first 19 years of the project and is described in Section 16.

Table 15.4.1: Diluted Mineral Reserve by Clay Unit

Lithology	Ore > = 1,500 ppm Li				Waste (kt)	Total (kt)
	Ore (kt)	Li (ppm)	LCE (kt)	K (%)		
Basalt	0				1,977,803	1,977,803
	0				65,742	65,742
Upper Clay, low grade	0				136,759	136,759
Upper Clay, high grade	62,351	2,844	944	1.04	5,550	67,901
Ignimbrite	0				91,431	91,431
Lower Clay	181,457	3,698	3,571	1.59	4,339	185,796
Basement & Undefined	0				17,077	17,077
Total	**243,808**	**3,479**	**4,515**	**1.45**	**2,298,701**	**2,542,509**

Table 15.4.2: South Pit Diluted Mineral Reserve by Claim Group

Lithology	Ore > = 1,200 ppm Li				Waste (kt)	Total (kt)
	Ore (kt)	Li (ppm)	LCE (kt)	K (%)		
La Ventana	92,557	3,488	1,718	1.51	789,464	882,021
El Sauz	75,366	2,765	1,109	1.16	783,154	858,520
Fleur	75,885	4,179	1,688	1.65	726,083	801,968
Total	**243,808**	**3,479**	**4,515**	**1.45**	**2,298,701**	**2,542,509**

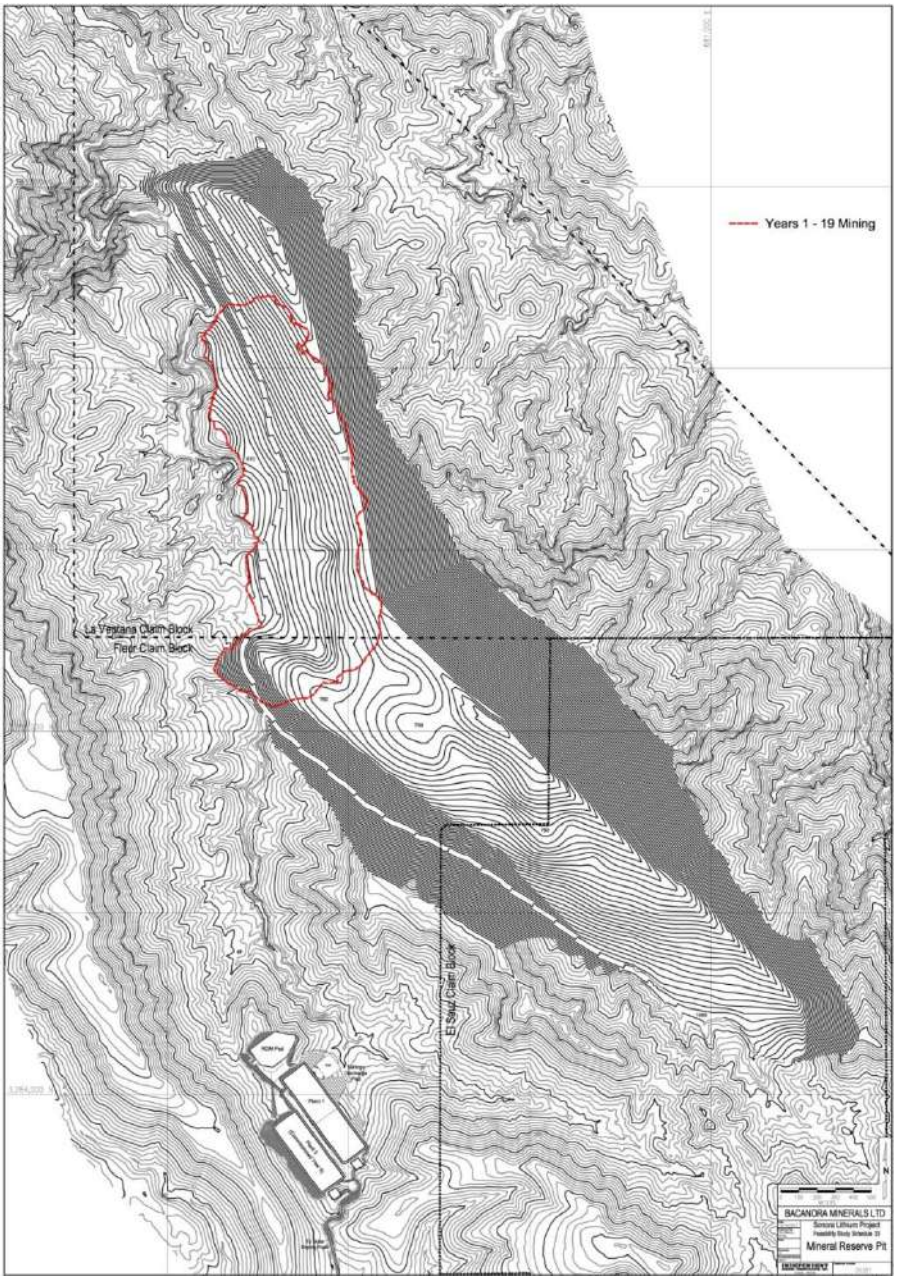


Figure 15.4.1: Mineral Reserve Pit

16 MINING METHODS

16.1 Introduction

The mine production for a targeted 19 year schedule comes from four mining phases in the north end of the reserve pit and the sum of thee phases is smaller than the reserve pit design. A summary of the tonnage and grade for the phases at 1,500 ppm Li cutoff (minimum cutoff for the production schedule) is shown in Table 16.1.1. The cutoff is applied to the model diluted block grades and the associated grades for all elements stored in the block model are tabulated.

Phase 1 is located on the west side of the pit and the ore comes primarily from the higher grade lower clay seam. Phase 1 is split into a south and north part for grade blending during the mine production scheduling. Phase 2 begins the stripping of the basalt waste on the east side of the pit and deepens the pit bottom. A stream diversion is included in the southwest wall of Phase 2 which intercepts the arroyo at the south end of the pit and connects to an existing drainage on the west side. The north end of the pit stays south of this drainage as it crosses over the clay seams north of the pit design. Phase 3 expands the pit to the east for the southern portion of the pit. Phase 4 expands the northeast side of the pit and completes the 19 year production pit. Access ramps are left in the east wall of Phases 3 and 4 for future mining beyond the 19 year schedule. The mining phases for the 19 year production schedule target the higher grade, lower waste:ore ratio portion of the reserve pit design. Figure 16.1.1 through Figure 16.1.4 illustrates the pit phase designs.

Table 16.1.1: Pit Phases for Production Schedule

Mining Phase	Ore (kt)	Li (ppm)	LCE (kt)	K (%)	Waste (kt)	Total (kt)	Waste/Ore Ratio
	Ore > = 1,500 ppm Li						
Phase 1 south	1,566	5,115	43	2.07	885	2,451	0.57
Phase 1 north	11,026	3,716	218	1.71	8,051	19,077	0.73
Phase 2	12,205	4,249	276	1.78	14,646	26,851	1.20
Phase 3	10,970	4,300	251	1.72	63,891	74,861	5.82
Phase 4	10,464	3,334	186	1.44	44,150	54,614	4.22
Total	**46,231**	**3,956**	**973**	**1.68**	**131,623**	**177,854**	**2.85**

The phase designs have 10 m benches which have been sub-divided into 2 m bench slices for the production scheduling. The 2 m bench slices have been tabulated from the 1 m bench resource model. The resource block model uses a minimum sub-block 10 m by 10 m block in plan and a 1 m in height. The contacts between the geologic units are represented in the block model in a stairstep fashion whereas the contacts are smoother, dipping planes in space. The volume of each unit is respected in the block model, but the geometry will be slightly different when being mined. In order to assure all of the lower clay is captured in the mine plan, the phase designs take some of the footwall volcanics (basement waste rock) as part of the pit design. A surface miner machine is planned for mining the clays and it has an adjustable digging depth between 0.0 and 1.1 meters, along with the ability to work on sloped surfaces. Thus it will be able to mine closer to the contacts of the ore horizons than represented by the phase tonnages tabulated from the block model. The current mine production schedule includes more basement waste tonnage than will actually be mined.

During mining, the upper waste will be mined with a bench between 5 and 10 m depending on the equipment fleet used by the mining contractor or Bacanora. The clay ore zones will be mined with a surface miner along with the internal waste zones. This approach will allow for ore grade control and blending of plant feed head grades. The surface miner will generate a windrow of broken ore which can be sampled and flagged by grade ranges. The windrow will be loaded into mine haul trucks for transport to the plant with blending either in the pit, at the run of mine (ROM) pad at the plant or both in order to provide a uniform Li head grade to the process plant.

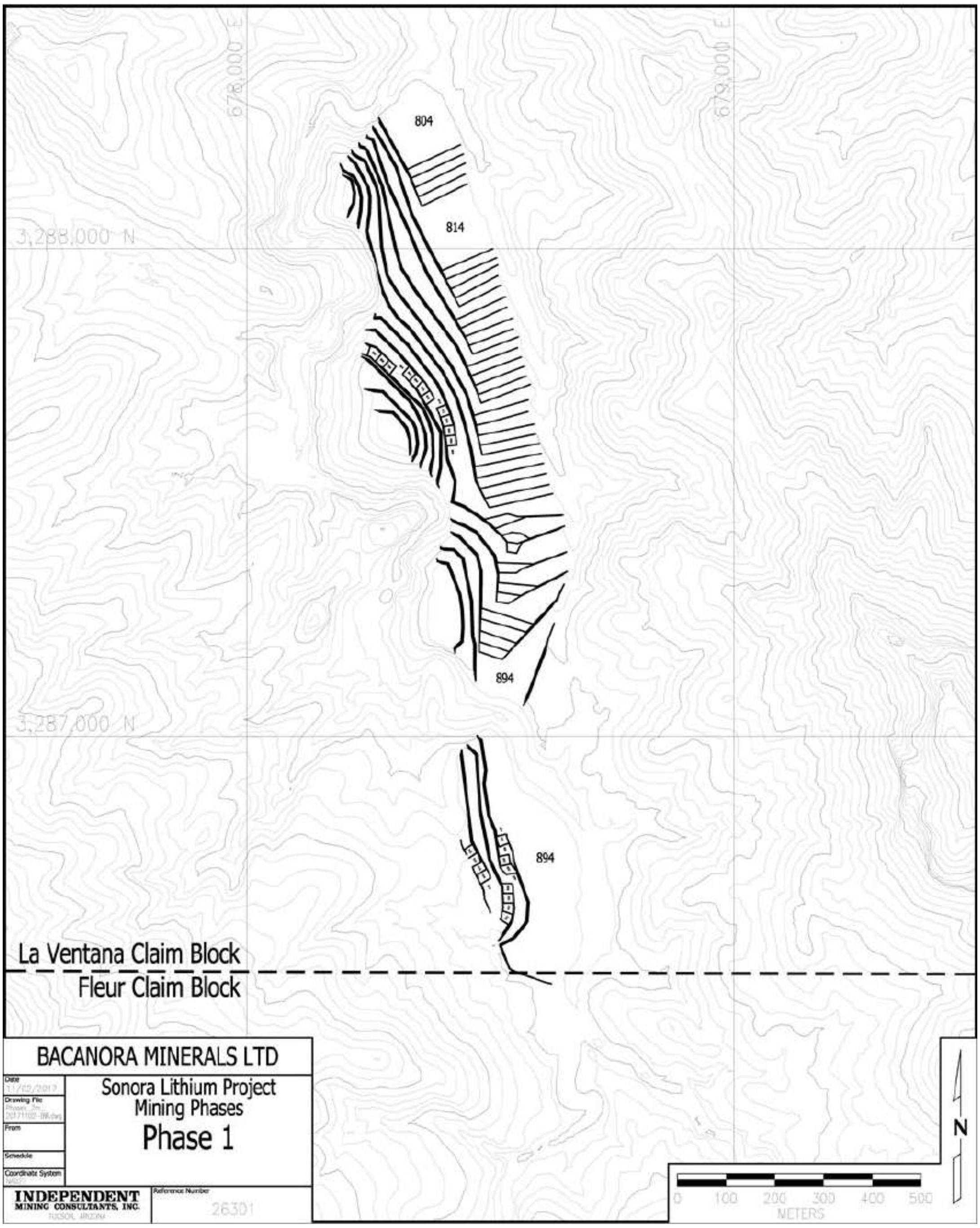


Figure 16.1.1: Pit Design – Phase 1

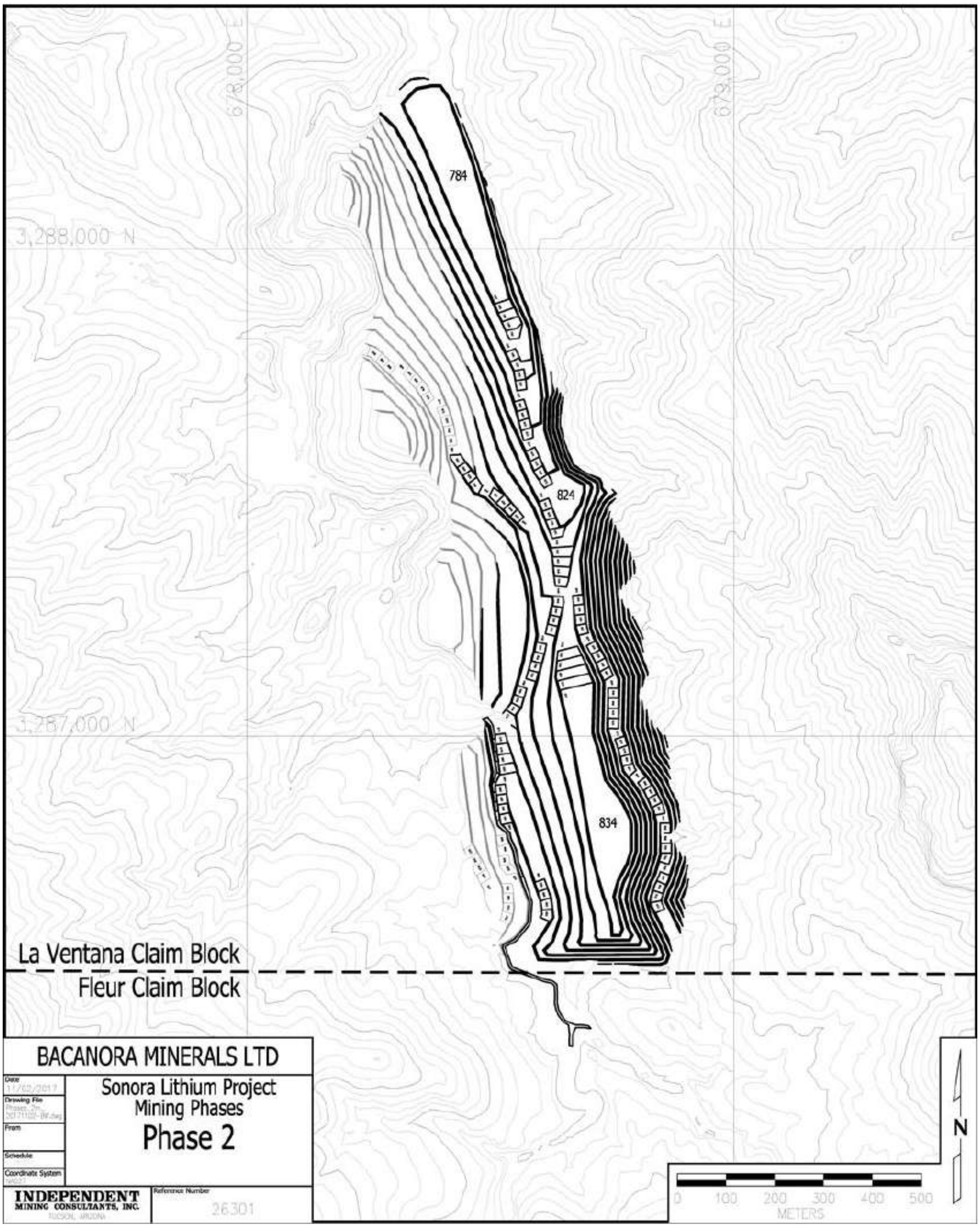


Figure 16.1.2: Pit Design – Phase 2

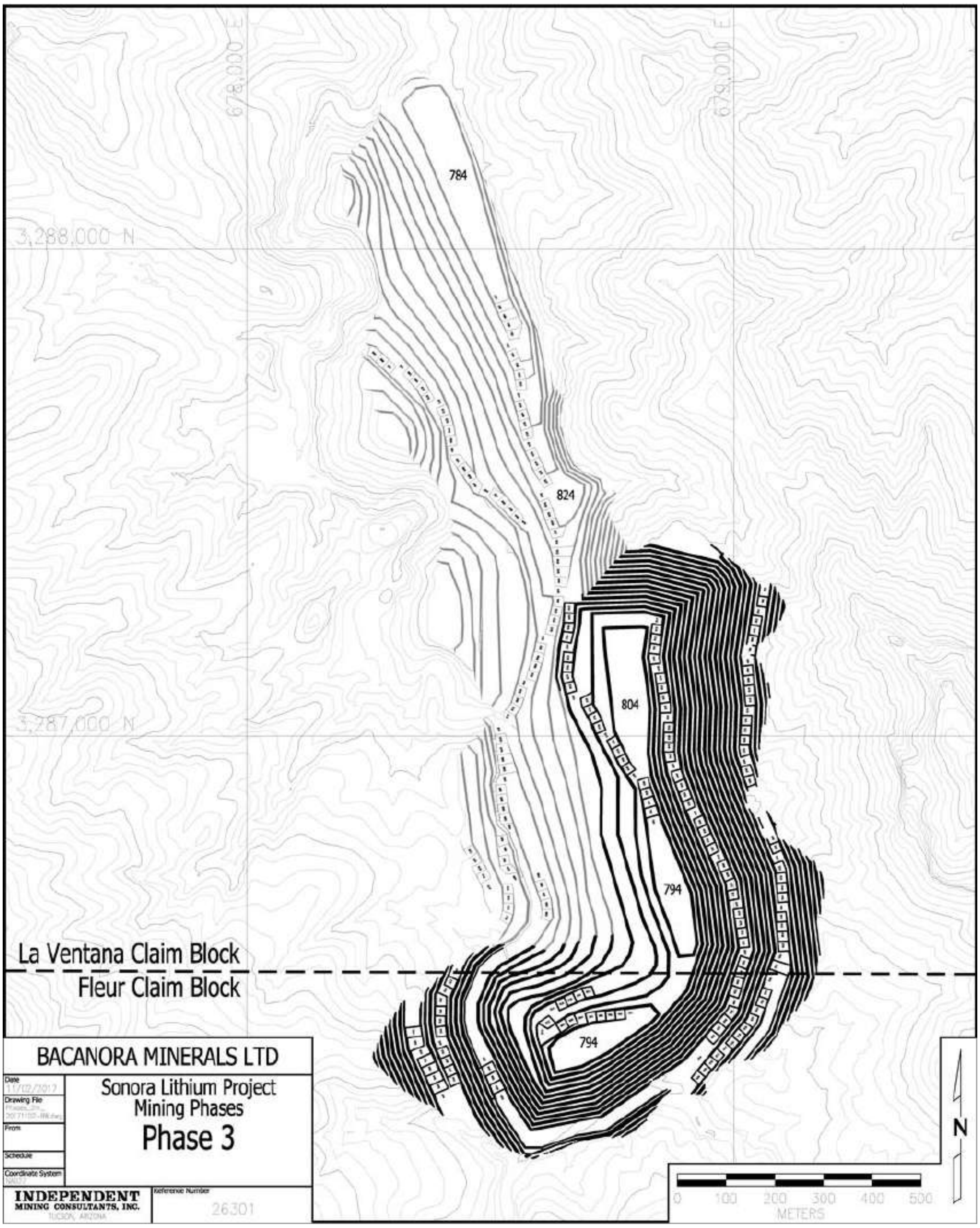


Figure 16.1.3: Pit Design – Phase 3

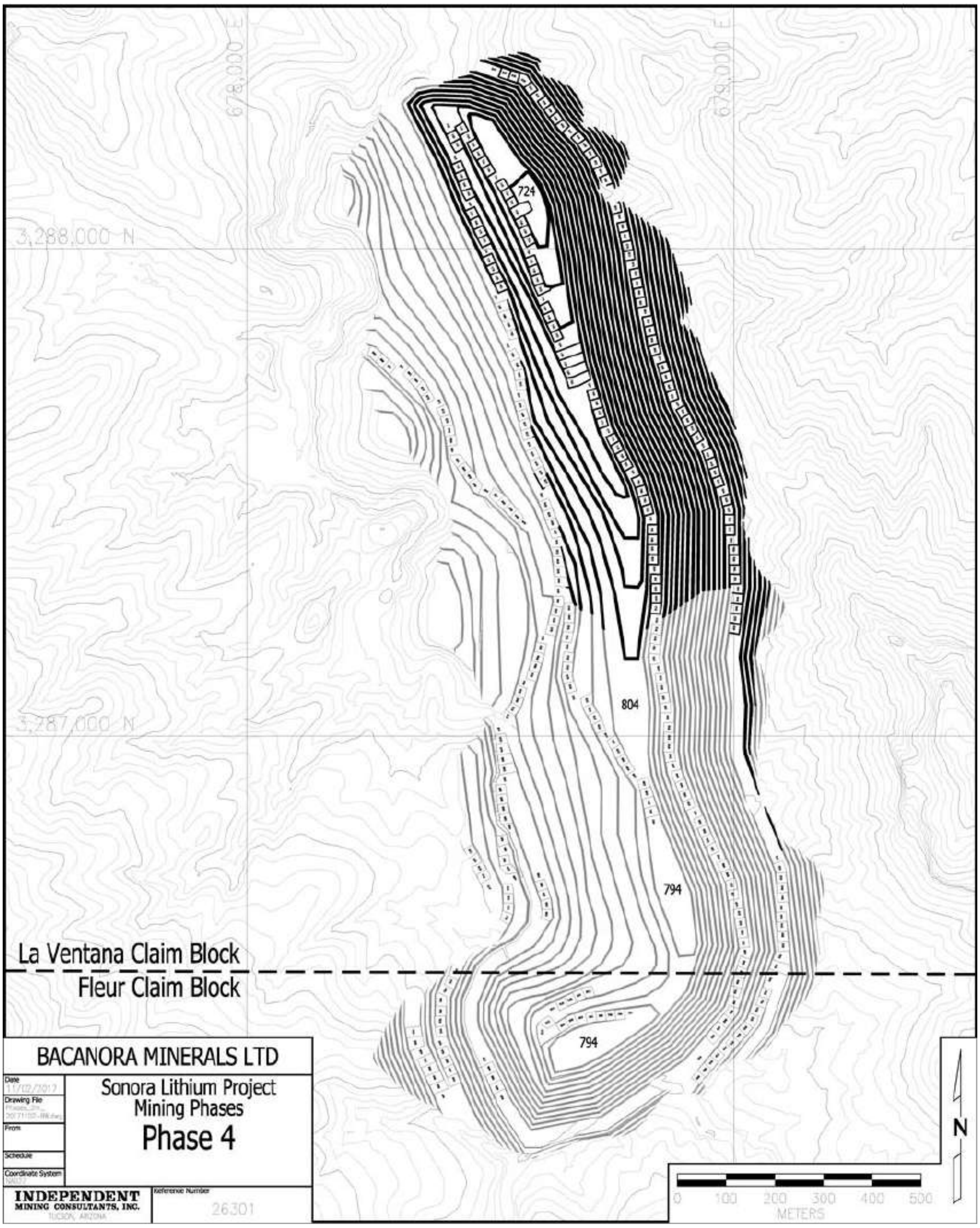


Figure 16.1.4: Pit Design – Phase 4

16.2 Mine Production Schedule

The mine production schedule is linked to the ramp up and expansion of the lithium carbonate (Li_2CO_3) process plant which will be located to the south of the open pit and southwest of the waste rock and tailings storage areas. The plant starts with a single production line with a target Li_2CO_3 tonnage of 17,500 tonnes per year. In Year 5, a second plant line is commissioned and the total Li_2CO_3 production capacity increases to a minimum of 35,000 t Li_2CO_3 per year. The production schedule is shown for 19 years and there is reserve to extend it beyond that time period.

Ausenco provided a plant ramp up schedule (to be used for the mine production schedule) for both tonnage and Li recovery in Stage 1 and Stage 2 which is commissioned at the start of Year 5. Table 16.2.1 shows this ramp of schedule by quarters through Year 6 when both plants are at full capacity of tonnage rate and maximum Li recovery of 75%.

Table 16.2.1: Process Rate and Recovery Ramp Up for Mine Plan

Year	Quarter	Stage 1			Stage 2		
		Throughput Rate		Li recovery	Throughput Rate		Li recovery
		kt	% of capacity	%	kt	% of capacity	%
1	1	129	46.7%	28.4%			
	2	212	76.8%	43.2%			
	3	222	80.4%	57.5%			
	4	239	86.6%	63.9%			
2	1	249	90.2%	68.3%			
	2	257	93.1%	70.7%			
	3	261	94.6%	71.9%			
	4	266	96.4%	72.0%			
3	1	276	100.0%	73.7%			
	2	276	100.0%	75.0%			
	3	276	100.0%	75.0%			
	4	276	100.0%	75.0%			
4	1	276	100.0%	75.0%			
	2	276	100.0%	75.0%			
	3	276	100.0%	75.0%			
	4	276	100.0%	75.0%			
5	1	276	100.0%	75.0%	185	67.0%	24.0%
	2	276	100.0%	75.0%	262	94.9%	49.7%
	3	276	100.0%	75.0%	276	100.0%	63.0%
	4	276	100.0%	75.0%	276	100.0%	71.0%
6	1	276	100.0%	75.0%	276	100.0%	73.0%
	2	276	100.0%	75.0%	276	100.0%	74.7%
	3	276	100.0%	75.0%	276	100.0%	75.0%
	4	276	100.0%	75.0%	276	100.0%	75.0%
Years 7 - 19		per Qrt.	276	100.0%	75.0%	276	100.0%

Table 16.2.2 is a summary of the mine production schedule by year and includes the percent of the ore coming from the La Ventana claim block which is 99.9% owned by Bacanora Minerals. The balance of the mine production comes from the Fleur claim block of which Bacanora has a 70% interest.

The 1,500 ppm cutoff grade removes the upper clay low grade material from the plant feed stream and is treated as waste for this production schedule. As well, during years 1 through 10, the upper clay high grade material is excluded from the plant feed tonnage as requested by Ausenco. Tonnage and grade of the upper clay high grade mined during years 1 – 10 is 1,961 ktonnes at 3134 ppm Li (diluted) and 1.11% K (diluted). This material is above cutoff but of lower value than the production schedule plant feed grade and could be stockpiled for later processing or blending, but is currently not part of the plant feed schedule and is considered waste.

The production of Li_2CO_3 tonnage in most years exceeds the design of 17,500 tonnes for Plant 1 and 35,000 tonnes for combined plants 1 and 2 by a few percent. It is within the design capacity of the plants.

The mine production schedule was developed in detail by months for Years 1 through 7, then by quarters for years 8 through 10 and Years 11 through 19 on an annual basis. Table 16.2.2 through Table 16.2.5 are annual summaries of the mine schedule. Table 16.2.2 shows the mine schedule of ore and waste by year; Table 16.2.3 shows the ore feed by clay seam (lower clay for Years 1 through 10, then a mix of lower clay and upper high grade clay); Table 16.3.2 shows the ore and waste tonnage mined for each of the mining phases by year and Table 16.2.5 shows the elevations mined by year within each mining phase (tabulated in 10 m intervals).

Table 16.2.2: Mine Production Schedule

Year	Li Cutoff Grade ppm	Ore, kt	Li, ppm	Li_2CO_3 tonnes		K, %	Waste, kt	Total, kt	Waste/Ore ratio	% of feed from La Ventana Claims
				contained	recovered					
1	1,500	802	5,142	21,947	11,153	2.08	390	1,192	0.49	100.00%
2	1,500	1,033	4,776	26,257	18,578	2.00	911	1,944	0.88	100.00%
3	1,500	1,104	4,355	25,588	19,097	1.90	2,013	3,117	1.82	99.82%
4	1,500	1,104	4,218	24,783	18,588	1.86	3,252	4,356	2.95	100.00%
5	1,500	2,103	4,131	46,235	30,424	1.82	7,497	9,600	3.56	100.00%
6	1,500	2,208	4,157	48,849	36,494	1.83	8,592	10,800	3.89	100.00%
7	1,500	2,208	4,152	48,790	36,592	1.84	8,592	10,800	3.89	100.00%
8	1,500	2,208	4,128	48,508	36,376	1.84	8,992	11,200	4.07	100.00%
9	1,500	2,208	4,193	49,272	36,949	1.85	8,992	11,200	4.07	99.68%
10	1,500	2,208	4,141	48,661	36,493	1.83	8,992	11,200	4.07	97.96%
11	1,500	2,208	4,201	49,366	37,024	1.73	9,892	12,100	4.48	68.80%
12	1,500	2,208	4,174	49,048	36,786	1.67	9,892	12,100	4.48	56.30%
13	1,500	2,208	4,012	47,145	35,359	1.60	9,892	12,100	4.48	66.26%

Year	Li Cutoff Grade ppm	Ore, kt	Li, ppm	Li_2CO_3 tonnes		K, %	Waste, kt	Total, kt	Waste/Ore ratio	% of feed from La Ventana Claims
				contained	recovered					
14	1,500	2,208	4,087	48,026	36,020	1.68	9,892	12,100	4.48	71.20%
15	1,500	2,208	4,069	47,815	35,861	1.71	9,892	12,100	4.48	83.15%
16	1,500	2,208	4,021	47,250	35,438	1.64	7,820	10,028	3.54	89.54%
17	1,500	2,208	4,001	47,015	35,262	1.69	3,575	5,783	1.62	87.95%
18	1,500	2,208	4,007	47,086	35,314	1.67	3,499	5,707	1.58	89.22%
19	2,000	2,208	3,999	46,992	35,244	1.63	4,095	6,303	1.85	89.40%
Total		**37,058**	**4,151**	**818,631**	**603,052**	**1.76**	**126,672**	**163,730**	**3.42**	**88.05%**

Table 16.2.3: Annual Plant Feed Schedule by Clay Zone

Year	Li Cutoff Grade, ppm	Lower Clay		Upper Clay		Total Plant Feed	
		kt	Li, ppm	kt	Li, ppm	kt	Li, ppm
1	1,500	802	5,142			802	5,142
2	1,500	1,033	4,776			1,033	4,776
3	1,500	1,104	4,355			1,104	4,355
4	1,500	1,104	4,218			1,104	4,218
5	1,500	2,103	4,131			2,103	4,131
6	1,500	2,208	4,157			2,208	4,157
7	1,500	2,208	4,152			2,208	4,152
8	1,500	2,208	4,128			2,208	4,128
9	1,500	2,208	4,193			2,208	4,193
10	1,500	2,208	4,141			2,208	4,141
11	1,500	1,756	4,421	452	3,347	2,208	4,201
12	1,500	1,467	4,573	741	3,383	2,208	4,174
13	1,500	1,349	4,463	859	3,305	2,208	4,012
14	1,500	1,627	4,376	581	3,276	2,208	4,087
15	1,500	1,714	4,327	494	3,175	2,208	4,069
16	1,500	1,528	4,431	680	3,101	2,208	4,021
17	1,500	1,743	4,249	465	3,073	2,208	4,001
18	1,500	1,740	4,287	468	2,965	2,208	4,007
19	2,000	1,600	4,410	608	2,919	2,208	3,999
Total		**31,710**	**4,314**	**5,348**	**3,184**	**37,058**	**4,151**

Mining begins in phase 1 after a short pre-production period during which the haul roads (including the haul road to the plant and mine maintenance facilities) are constructed and any clearing of the phase 1 pit area is completed. It is assumed that these activities will take about 3 months to complete. No pre-stripping of Phase 1 is required as the lower clay unit is exposed just below the ignimbrite layer. A small amount of mining off of the ignimbrite layer will expose the lower clay for mining. Mining is confined to phase 1 (south and north sub-phases) during years 1 and 2 and phase 1 stays out of the arroyo bottom so no water diversion is required during the early years. Ore production from Phase 1 continues through Year 11.

The mining begins during Year 3 in Phase 2 with stripping of the basalt cap and ore production begins in Year 5 and continues through Year 19. By Year 5, the total material rate has increased to 7,497 kt/y with the waste stripping in Phases 2 and 3. A water course diversion is included in the southwest wall of Phase 2 to divert water during the west season around the pit to the west and return it to a natural drainage on the west side of the pit. The haul road exiting the pit will have to cross this diversion.

Stripping in Phase 3 begins in Year 4 with the peak being during Years 6 through 12 when the total tonnage rate steps up to 9,892 kt/y. Steady ore production from Phase 3 is during Years 12 through 19. The west and southwest portions of the water diversion remain the same as Phase 3 mines down through the 910 bench elevation, but the south side is moved further south as part of Phase 3 in Year 10.

Phase 4 is north of Phase 3 on the east side of the pit and mining begins in Phase 4 in Year 12 with ore production starting in Year 15. The mine schedule stops in Year 19, but there is still ore in the bottom of Phase 4, but at a declining Li grade.

If mining continues past Year 19, then the stripping of the next mining phase to the east of the mine plan pit needs to begin around Year 15. Each phase as the pit expands east has a higher waste:ore ratio and takes longer to mine down to the ore, which is at a lower elevation as the pit expands eastward. An alternative to mining eastward would be to start a new mining area to the south which would have a lower waste:ore ratio, but also lower Li head grades. To maintain the 35,000 tonnes of Li_2CO_3 production rate, the plant feed tonnage rate would need to increase from the 2,208 kt/year rate.

Table 16.2.4 shows the ore and waste tonnage by year by mining phase. Table 16.2.3 details the mining elevation in each phase by year. The grey shaded years and elevations in Phases 2, 3 and 4 indicate predominately waste stripping with minimal ore production.

Table 16.2.4: Mine Production by Mining Phase

Year	Cutoff Grade Li (ppm)	Phase 1 south			Phase 1 north			Phase 2			Phase 3			Phase 4			Total		
		Ore (kt)	Li (ppm)	Waste (kt)	Ore (kt)	Li (ppm)	Waste (kt)	Ore (kt)	Li (ppm)	Waste (kt)	Ore (kt)	Li (ppm)	Waste (kt)	Ore (kt)	Li (ppm)	Waste (kt)	Ore (kt)	Waste (kt)	Total (kt)
1	1,500	796	5,152	366	6	3,764	24										802	5,142	390
2	1,500	653	5,085	422	380	4,240	489										1,033	4,774	911
3	1,500	117	5,026	97	987	4,277	1,158	0		759							1,104	4,356	2,014
4	1,500				1,104	4,218	912	0		1,733	0		607				1,104	4,218	3,252
5	1,500				1,907	4,034	1,104	196	5,070	2,607	0		3,786				2,103	4,131	7,497
6	1,500				1,575	3,803	1,082	633	5,038	2,228	0		5,282				2,208	4,157	8,592
7	1,500				1,335	3,582	1,003	873	5,024	1,965	0		5,624				2,208	4,152	8,592
8	1,500				1,270	3,449	953	938	5,047	1,419	0		6,620				2,208	4,128	8,992
9	1,500				1,040	3,293	757	1,149	5,000	1,317	19	4,568	6,918				2,208	4,192	8,992
10	1,500				910	3,059	703	1,181	4,938	1,097	117	4,508	7,192				2,208	4,141	8,992
11	1,500				243	2,530	139	993	4,788	640	972	4,020	9,113				2,208	4,201	9,892
12	1,500							785	4,692	465	1,423	3,887	7,483	0		1,944	2,208	4,173	9,892
13	1,500							657	4,382	506	1,551	3,856	3,821	0		5,565	2,208	4,013	9,892
14	1,500							599	3,748	400	1,609	4,213	2,400	0		7,092	2,208	4,087	9,892
15	1,500							742	3,560	509	1,269	4,506	1,355	197	3,166	8,028	2,208	4,069	9,892
16	1,500							417	3,280	262	981	4,560	1,062	810	3,750	6,496	2,208	4,021	7,820
17	1,500							605	3,058	391	1,124	4,616	927	479	3,748	2,256	2,208	4,001	3,574
18	1,500							452	2,905	166	996	4,783	902	760	3,646	2,431	2,208	4,007	3,499
19	2,000							105	2,871	60	775	4,830	932	1,328	3,604	3,103	2,208	3,999	4,095
Total		**1,566**	**5,115**	**885**	**10,757**	**3,735**	**8,324**	**10,325**	**4,447**	**16,524**	**10,836**	**4,310**	**64,024**	**3,574**	**3,641**	**36,915**	**37,058**	**4,151**	**126,672**

Table 16.2.5: Mining Years by Phase and Elevation

Elevation Range		Phase				
From	**To**	**1-south**	**1-north**	**2**	**3**	**4**
1030	1020				4	
1020	1010				4	
1010	1000				4	
1000	990				4 - 5	12
990	980				5	12
980	970				5	12
970	960				5 - 6	12
960	950			3	6	12
950	940		1	3	6 - 7	12 - 13
940	930	1	1 – 2	3	7	13
930	920	1	2	3	8	13
920	910	1	2 – 3	4	8 - 9	13
910	900	1 - 2	3 – 4	4 - 5	9 - 10	13 - 14
900	890	2 - 3	4 – 5	5	10	14
890	880		5 – 6	6 - 7	11	14
880	870		6	7 - 8	11 - 12	14 - 15
870	860		6 – 7	8 - 9	12	15
860	850		7 – 8	10 - 11	12 - 13	15
850	840		8 – 9	11 - 12	13 - 14	15 - 16
840	830		9	13 - 14	14 - 15	16
830	820		10	14	15-16-17	16 - 17
820	810		10 – 11	15	17 - 18	17 - 18
810	800		11	15-16-17	18 - 19	19
800	790			17 - 18	19	
790	780			18 - 19		

16.3 Waste Storage Facilities

Waste Storage Facilities ("WSF") are designed to hold the 126.7 Mt of waste presented in Table 16.2.2. The WSF is located in the valley south of the pit and west and north of the process plant and mine maintenance facilities (see figures at the end of this report section). The WSF is comprised of the tailings storage facility (TSF) and the waste rock only storage (WRD). The tailings generated from the process plant are a filtered tailing which will be co-migled with the waste rock (60% rock and 40% tailings by volume) for permanent storage.

The volume of tailings is calculated by ore tonnage times 1.023 divided by a desity of 1.5, thus 1 tonne of ore generates 0.682 cubic meters of tailings to be stored.

During years 1 and 2 there is insufficient waste rock tonnage to blend with the tailings. During these two years, a small rock dam is built in one of the side valley southwest of the plant and the tailings are stored behind it at a slope of 3:1 (horizontal:vertical). This area is later covered with the rock-tailings mix.

Starting in Year 3, there is sufficient waste rock volume to mixed with the tailings for storage and have excess waste rock for placement in the WRD. The TSF is located at the south end of the WSF (close to the plant) and WRD is located at the north end of the WSF (closer to the pit). The WRD is designed with a 2.5 (horizontal) to 1.0 (vertical) slope angle for any open faces of the WRD. This will allow for concurrent reclamation. The WRD and TSF are built from the bottom up with access ramps on the open face. A 30% swell is assumed for the waste rock volume calculations: 40% swell from the pit in place volume to the trucks and then 10% compaction in the WSF for a final 30% swell volume. The waste rock volume storage requirements are calculated from the insitu densites of the various waste rock units (shown below) times the final 30% swell volume.

Table 16.3.1: Waste Rock Average Densities

Rock Type	Average Density
Basalt	2.42
Sandstone	2.17
Upper Clay, low grade	2.23
Upper Clay, high grade	2.39
Ignimbrite	2.20
Lower Clay	2.36
Basement rock	2.23

The placement of the tailings – rock mix starting in year 3 assumes the following approach as outlined in the report “Design of the Residue Waste Disposal and Waste Rock Dump – November 2016” authored by Ausenco:

- Waste rock is dumped and dozed in a +- 5 m layer within the TSF boundary
- Tailing is loaded into a truck at the plant tailings discharge pile and hauled to the active disposal area, dumped and spread (by a dozer) over the waste rock during which approximately 40% of the tailings will occupy the void space within the rock
- A new layer of rock and tailings is placed on top of the previous one.

Table 16.3.2 summarizes the volume of waste rock and tailings stored in the WSF.

Table 16.3.2: Waste Tonnage to Storage Facilities

Year	Waste Rock Dump (m^3 x 1,000)	Tailings Storage Facility (m^3 x 1,000)	
		Waste Rock	Tailings
1	229	0	545
2	534	0	704
3	191	950	758
4	649	1,140	758
5	2,000	2,150	1,430
6	2,500	2,256	1,506
7	2,486	2,256	1,506
8	2,675	2,256	1,506
9	2,666	2,256	1,506
10	2,675	2,256	1,506
11	3,182	2,256	1,506
12	3,271	2,259	1,506
13	3,245	2,259	1,506
14	3,199	2,259	1,506
15	3,200	2,259	1,506
16	2,111	2,259	1,506
17	0	2,037	1,506
18	0	1,997	1,506
19	90	2,259	1,506
Total	**34,903**	**35,364**	**25,279**

Figure 16.3.1 through to Figure 16.3.9 illustrate the pit progress and WSF advances at the end of Years 1, 2, 3, 4, 5, 7, 10, 15, and Year 19 (end of mine schedule).

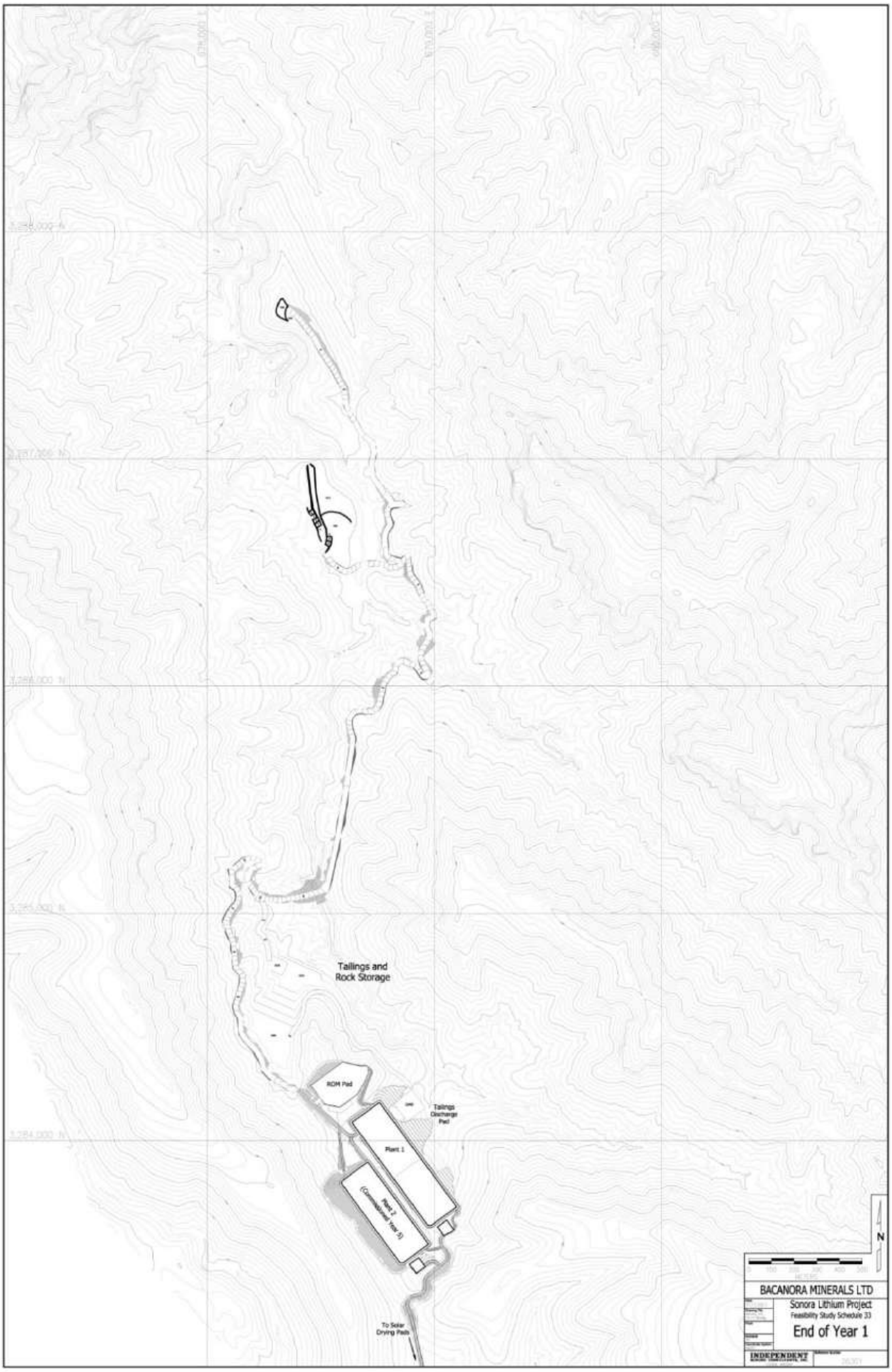


Figure 16.3.1: Mine Plan at End of Year 1

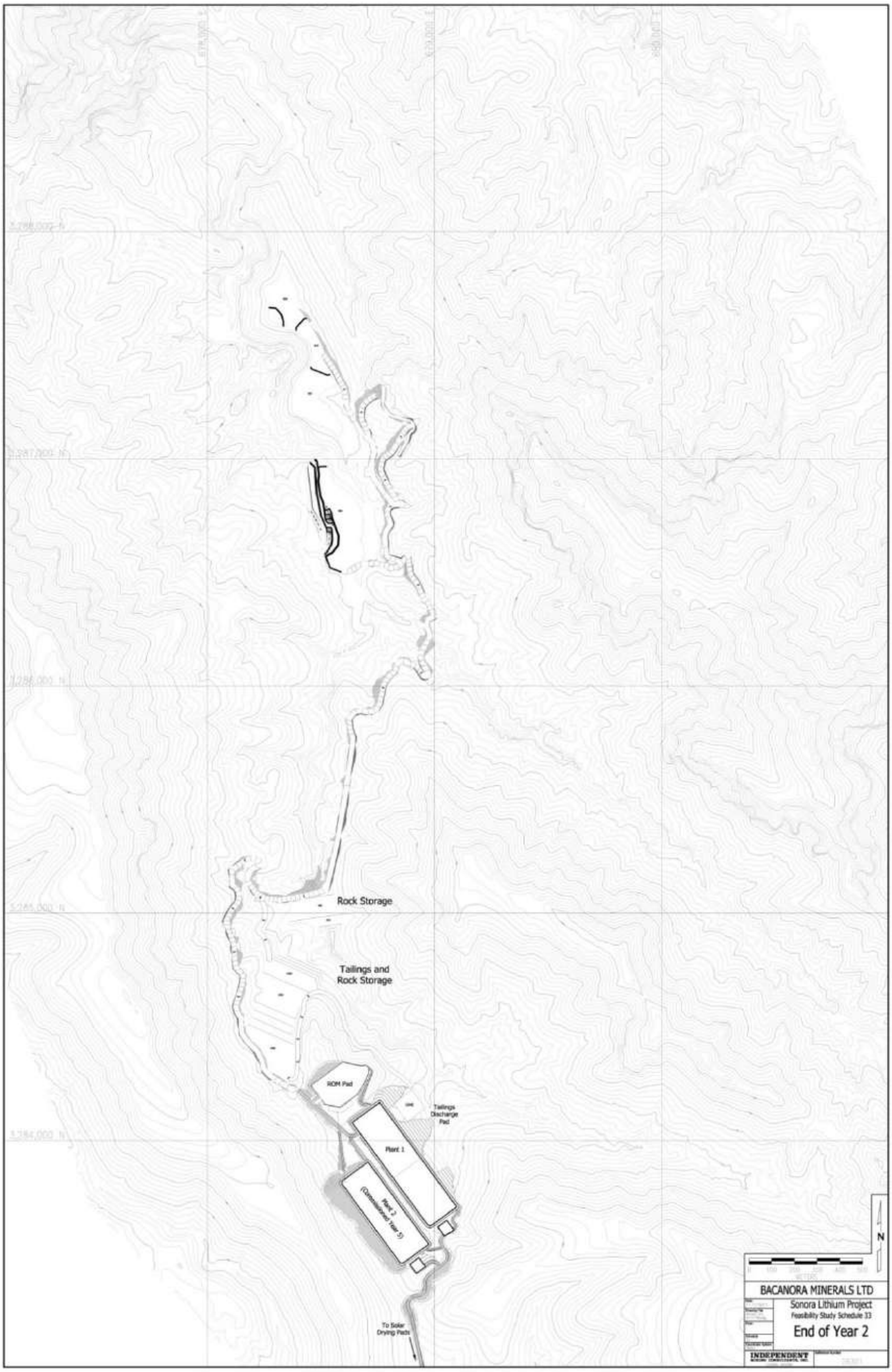


Figure 16.3.2: Mine Plan at End of Year 2

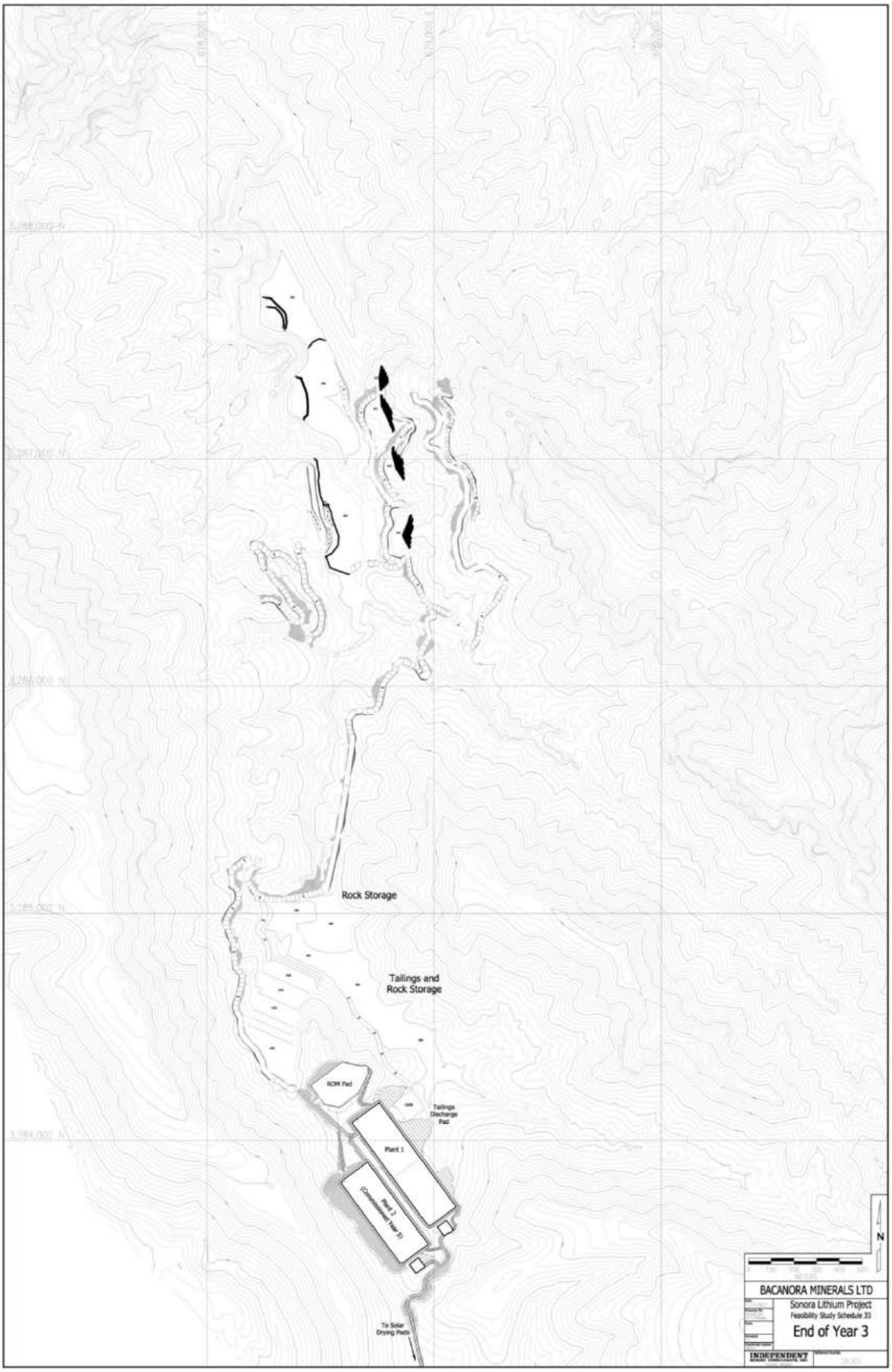


Figure 16.3.3: Mine Plan at End of Year 3

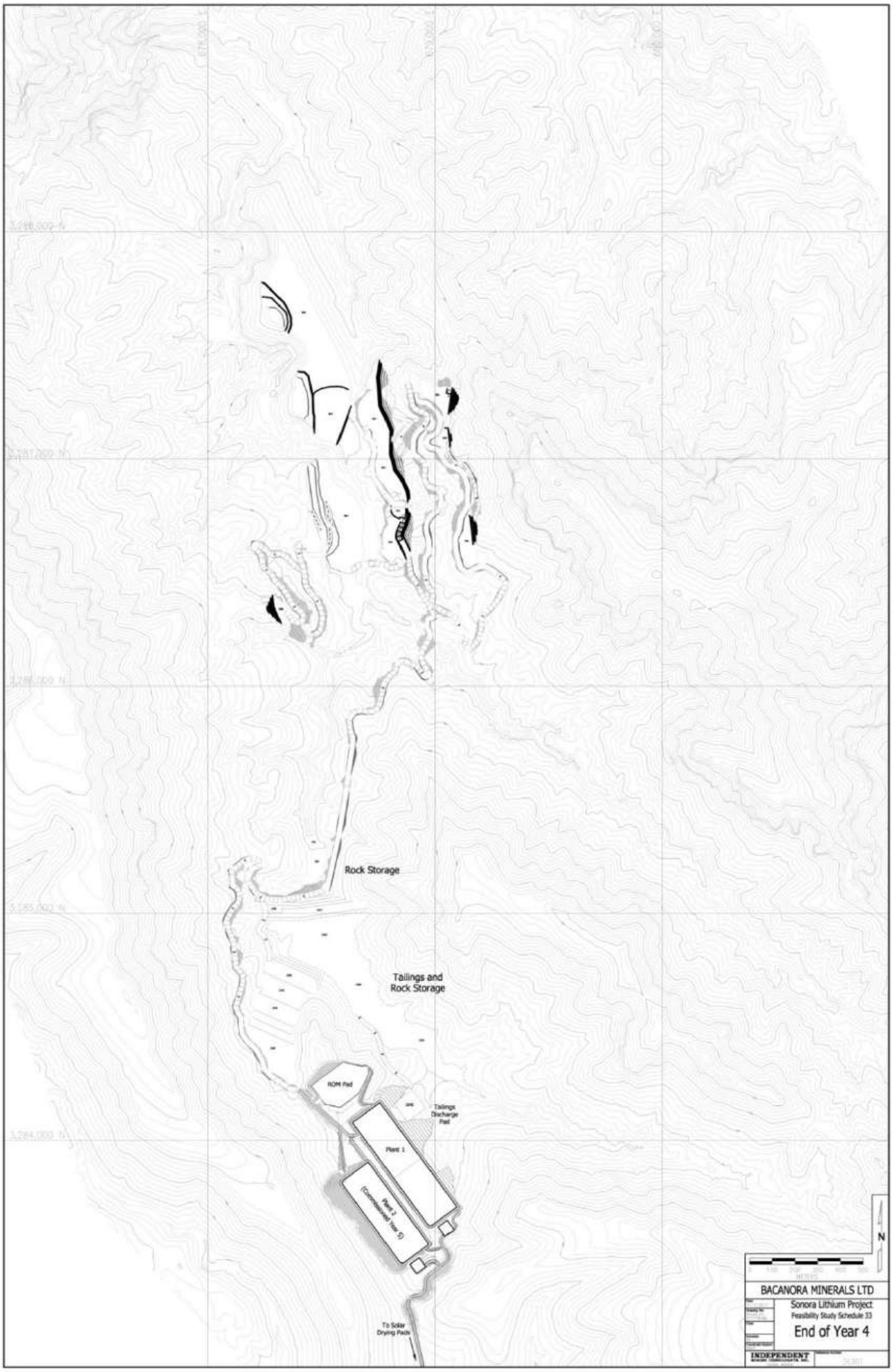


Figure 16.3.4: Mine Plan at End of Year 4

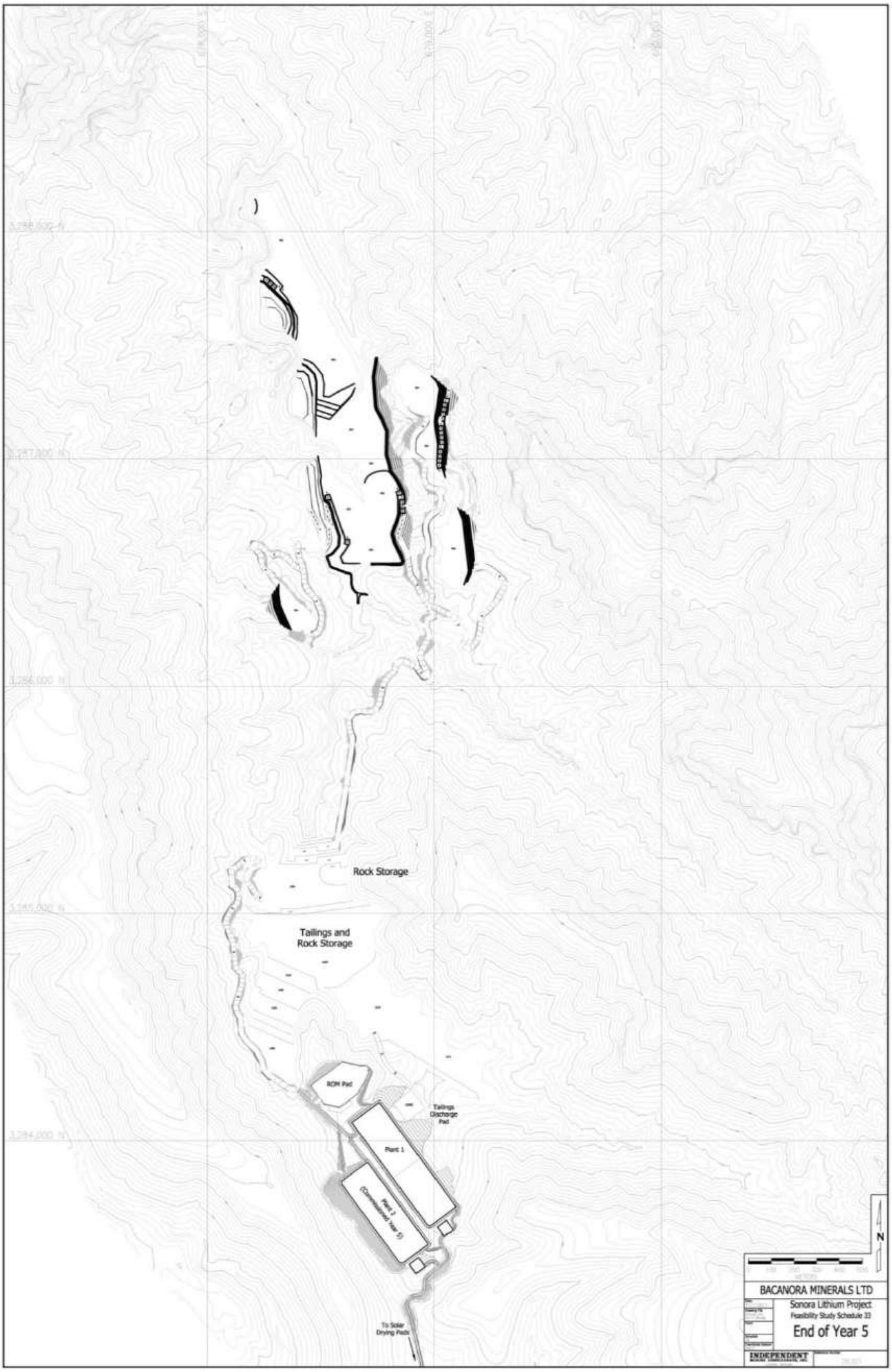


Figure 16.3.5: Mine Plan at End of Year 5

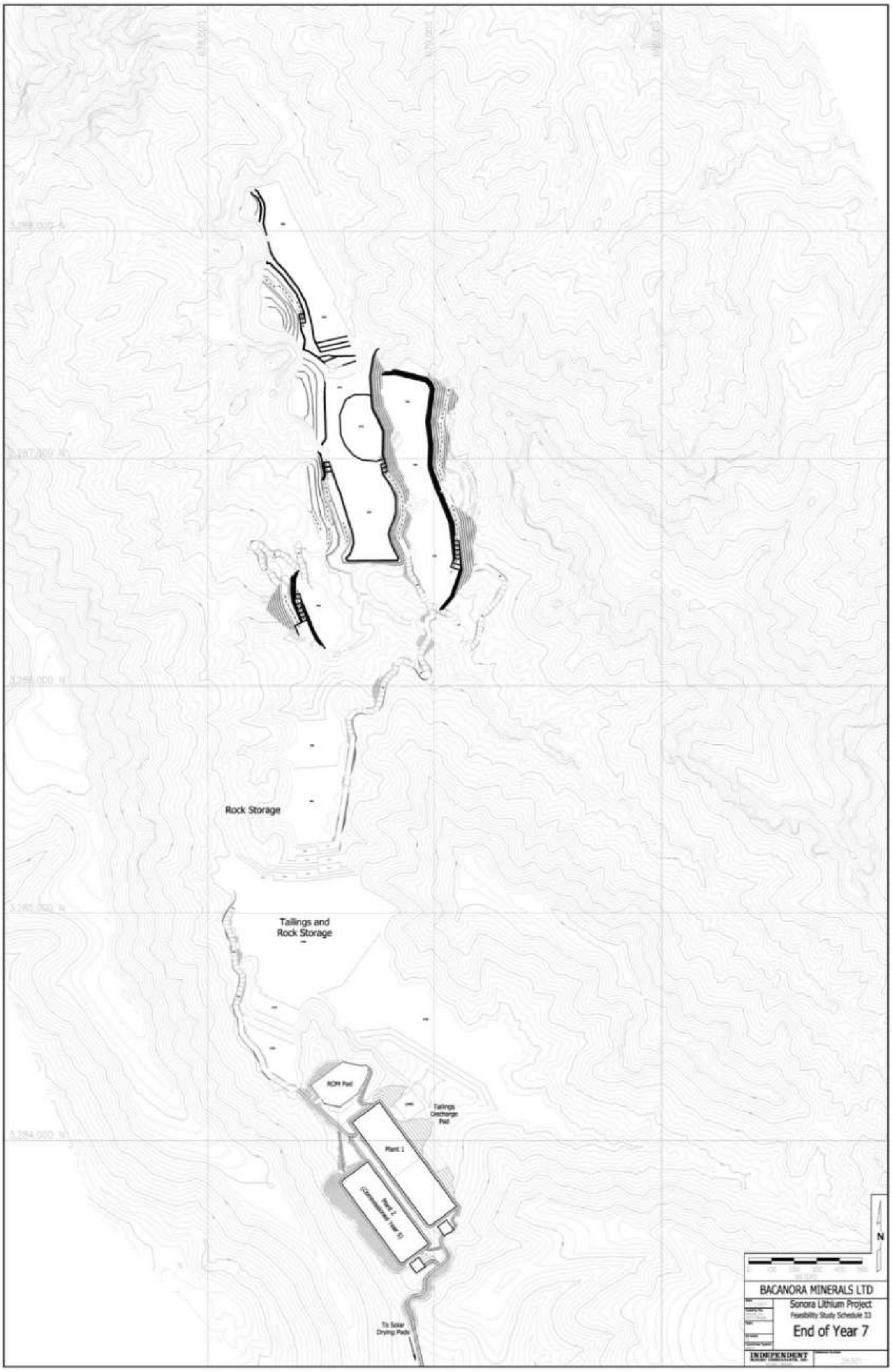


Figure 16.3.6: Mine Plan at End of Year 7

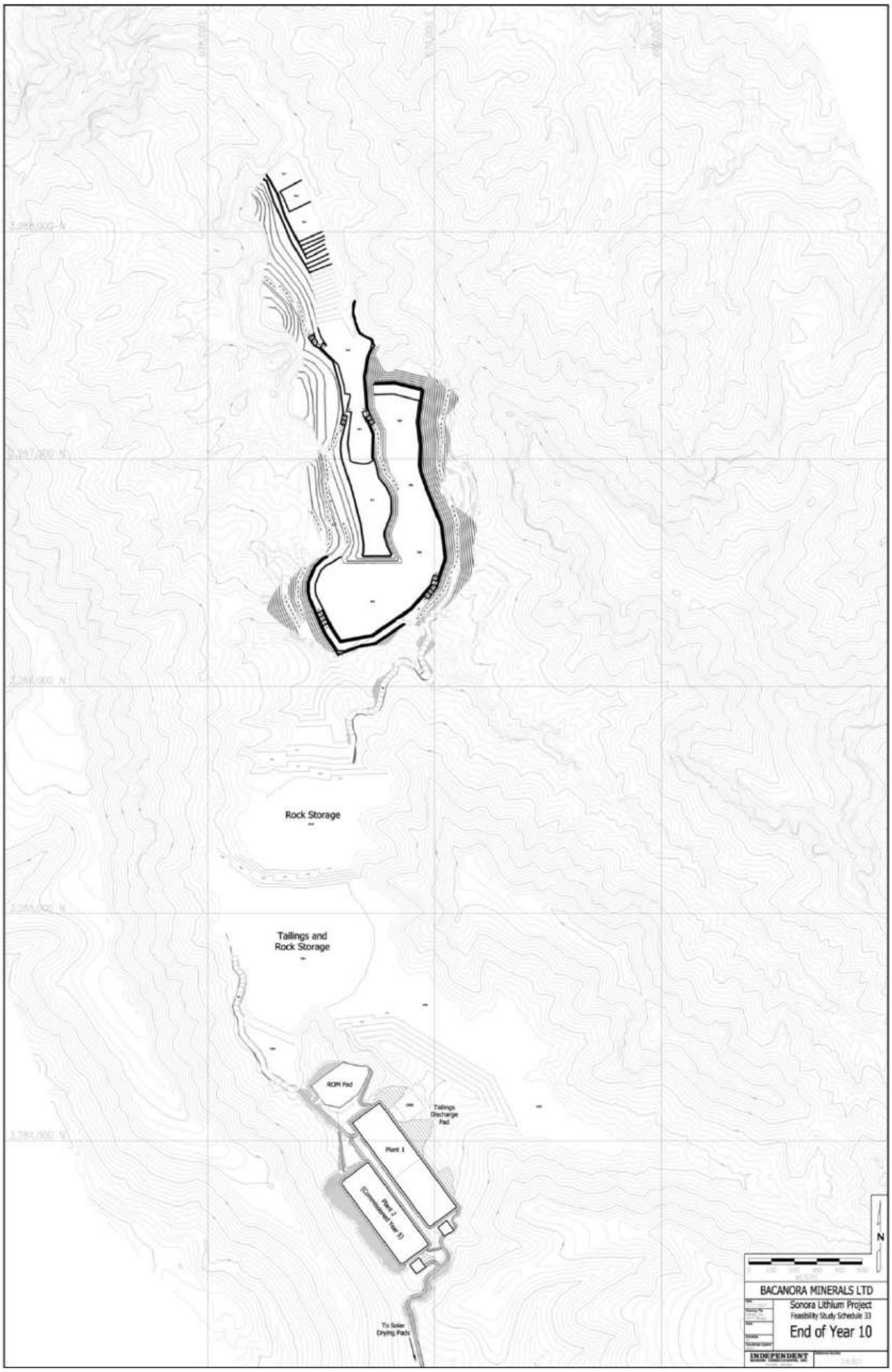


Figure 16.3.7: Mine Plan at End of Year 10

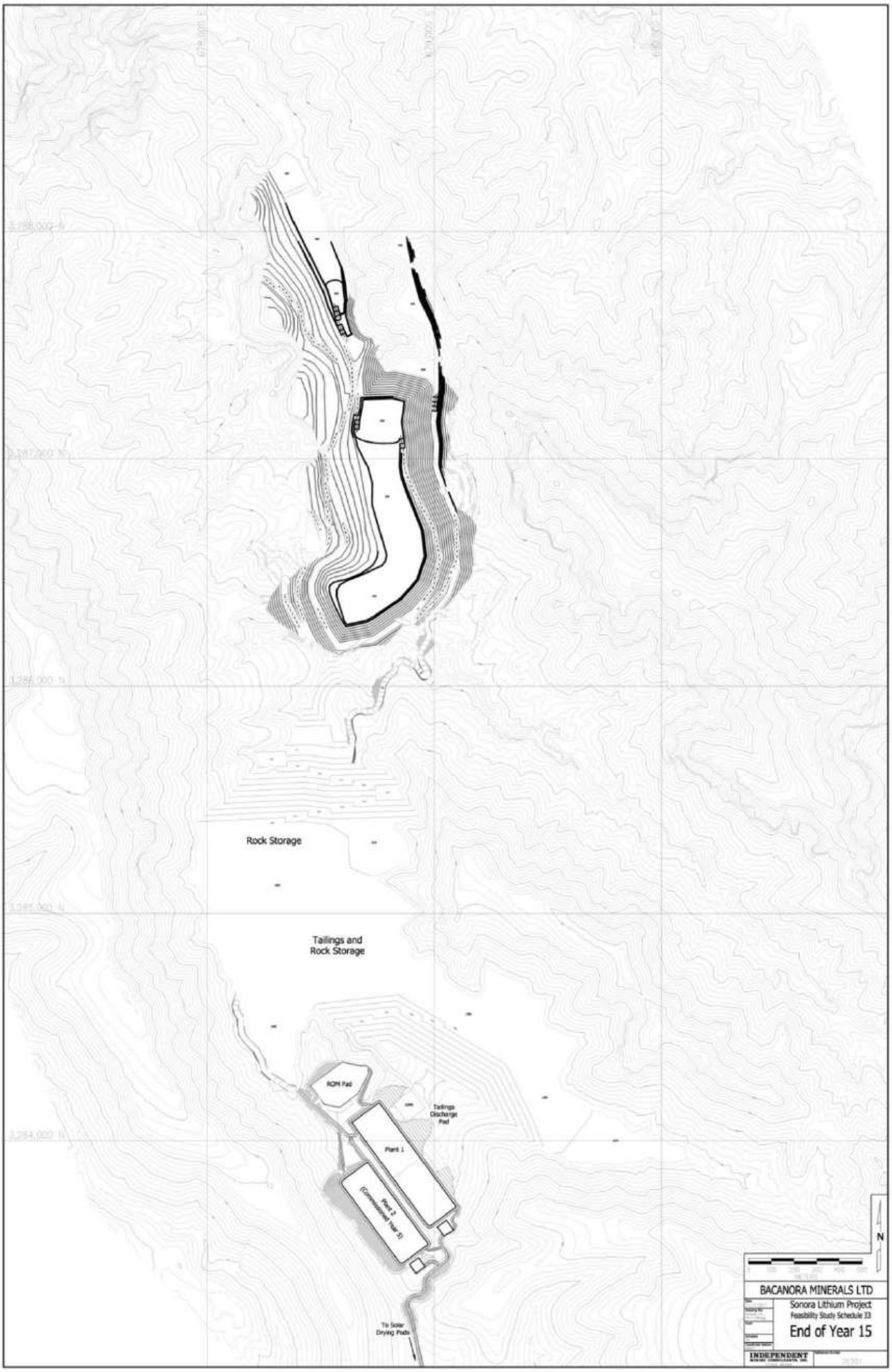


Figure 16.3.8: Mine Plan at End of Year 15

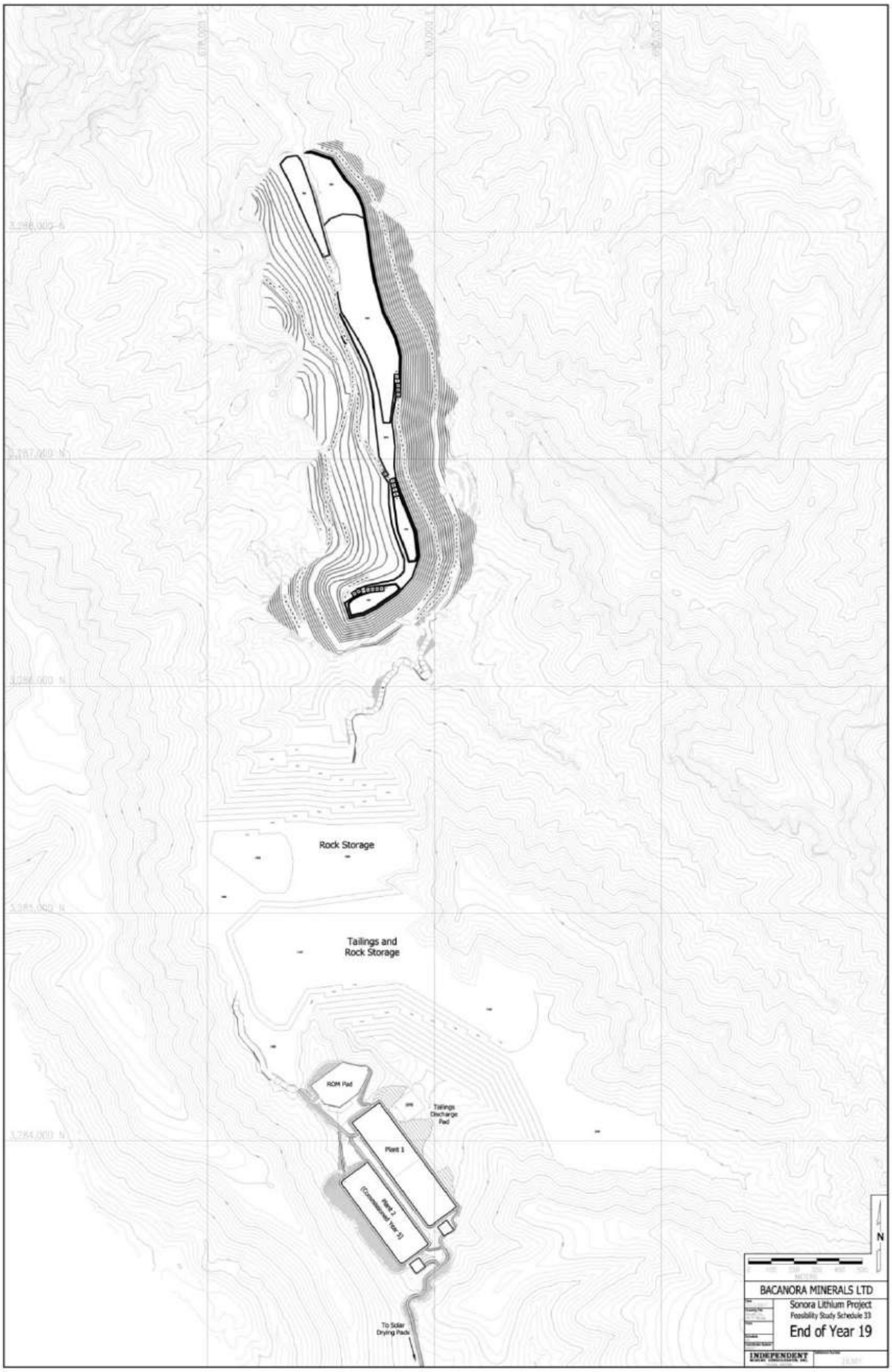


Figure 16.3.9: Mine Plan at End of Year 19 (End of Production Schedule)

16.4 Mining Equipment

16.4.1 Production Schedule Parameters

The mine production schedule is based on a 7 day per week schedule, with two 12 hour shifts per day once Stage 2 is operational in Year 5. There are four crews planned to cover the rotating schedule. During Years 1 through 4, there is one shift per day covered by 2 alternating crews. Each 12 hour shift has a one hour allowance for lunch, blasting shutdowns, fueling, equipment inspections, and the start and ending of the shift for a total of 11 effective working hours. A job efficiency factor of 50 minutes of work per 60 minutes of scheduled work is included to calculate the net productive operating hours per shift that equipment will be doing work. The job efficiency factor is an allowance for unscheduled delays throughout the shift which impede production. Table 16.4.1 shows typical shift and annual schedule parameters.

Table 16.4.1: Mine Schedule Parameters

Mine Schedule	
Crews	4
Shifts/Day	2
Hours/Shift	12 (720 minutes)
Lunch, Breaks, etc.	30 minutes
Equipment Inspection	10 minutes
Shift Change and Blasting	10 minutes
Fueling, Lube, and Service	10 minutes
Scheduled Productive Time	660 minutes
Job Efficiency (50 minutes/hour)	83.3%
Net Productive Minutes/Shift	550
Days/Year	360
Scheduled Shifts/Year, Years 1-4	360
Scheduled Shifts/Year, Years 5 - 19	720

The mine maintenance personnel work the same 12 hour shifts, two shifts per day during Years 5 through 19 and one shift per day during Years 1 through 4. The scheduled productive time for them is 680 minutes (no fueling or equipment inspection time) resulting in the net productive minutes per shift of 567 minutes.

16.4.2 Equipment Requirements

The amount of equipment required to meet the scheduled tonnages is calculated based on the mine schedule, equipment availabilities, usages and haul and loading times for the equipment. The equipment requirements to accomplish the mine schedule are based on new equipment. The reference to a specific vendor or equipment model is for reference only and is not a recommendation to purchase by IMC. The list of major mine mobile equipment is shown in Table 16.4.2, including the initial units and maximum units required to accomplish the mine production schedule.

Table 16.4.2: Major Mine Equipment and Fleet Size

Equipment Type	Initial Units	Maximum Units
Primary Production Fleet		
Rotary Drill (22.9 cm) PV 271 or equivalent	1	1
Surface Miner, Trencor T1460SM	1	2
Front End Loader (13.8 m^3) 993K or equivalent	1	2
Haul Truck (90 t) 777G or equivalent	3	14
Auxiliary Support Fleet		
Track Dozer D9 or equivalent	1	3
Motor Grader, 16M or equivalent	1	2
Water Truck, 777G or equivalent	1	2
Auxiliary Loader (4.4 m^3), 980M or equivalent	2	3
Auxiliary Truck (40 t), 740 ADT or equivalent	2	4
Rock Drill	1	1
Excavator	1	1

The rotary drill will be used in the basalt and other waste material to drill holes for blasting. Waste rock will be mined on 10 m benches using the front end loader and 90 t trucks. Waste rock that is associated with the ore on smaller benches will be mined with the front end loader.

Ore material will not require blasting. Ore will be mined with the surface miner machine on one meter or less cut depth, thus a minimum of two passes per 2 m bench slices used for the production schedule. The surface miner makes a cut 3.8 m wide and between 0.0 and 1.1 m deep and deposits the milled ore in windrows. The cut can be adjusted by the operator as ore control requirements dictate. Ore windrows will be picked up by the front end loader and loaded into 90 t trucks for delivery to the process plant. Any waste windrows will also be picked up by the front end loader, loaded into the 90 t trucks, and hauled to the waste storage facility.

The auxiliary loader and trucks will be used to haul the tails away from the process plant and deliver it to the combined waste rock and tailings storage facility (TSF). This will be a dedicated fleet to the tails haul with some additional use for miscellaneous work in the mine. A second auxiliary loader will be dedicated to feeding ore from the ROM pad to the process plant.

Table 16.4.3 lists the equipment productivities. Productivity was calculated based on machine size, operating schedule, and material characteristics.

Table 16.4.3: Mine Equipment Productivities

Material	Shift Productivity Table (tonnes per shift)			
	Drill	Production Loader	Surface Miner	Aux Loader
Ore Mined		11,648	9,406	
Ore to Feed				3,523
Basalt	16,241	16,854		
Other Rock	15,863	13,566		
Tailings				3,573
Mined Waste Rock		11,580		

The haulage equipment requirements have been developed based on the tonnage moved each year and the destinations of the material hauled. All of the haul routes have been measured and the travel times simulated. The inputs to the truck simulation runs include:

- Fixed times for loading and dumping when loaded with the front end loader:
    - ore, 5.3 minutes
    - basalt, 4.1 minutes
    - other waste rock, 4.7 minutes
- Maximum speeds:
    - downhill at 10% is 25 km/h
    - flats is 57 km/h
    - switchbacks is 15 km/h.

Table 16.4.4 shows the truck requirements by year. The required truck fleet is the total number of trucks necessary to ensure enough trucks are ready and available for service after taking into account mechanical availability and utilization.

Table 16.4.4: Truck Fleet Requirements

Year	Required Fleet	Average Max. Utilisation
1	3	0.59
2	4	0.69
3	6	0.70
4	7	0.81
5	8	0.74
6	9	0.73
7	9	0.73
8	9	0.77

Year	Required Fleet	Average Max. Utilisation
9	10	0.76
10	11	0.75
11	11	0.80
12	12	0.79
13	13	0.76
14	14	0.76
15	14	0.80
16	13	0.80
17	8	0.75
18	8	0.75
19	9	0.75

The auxiliary support equipment fleet is sized to handle all of the mine road construction and maintenance, dump maintenance, and clean up around the loading areas. Smaller support equipment is included in the fleet and a complete list is included in the mine capital cost section of this report. This equipment includes a fuel/lube truck, tire handler, 10 t service truck, mechanics truck, welders' truck, pick-up trucks, forklifts, and additional support equipment.

16.5 Mine Personnel

The mine personnel requirements are based on the annual shift schedule, the tonnages of material mined and moved and the number of pieces of equipment in operation. The equipment operator requirements assume that the operators are trained on multiple types of equipment and can move between types of equipment as needed to achieve the mine production schedule.

Maintenance personnel are low in the first three years during Stage 1. A maintenance and repair contract (MARC) is assumed to be in effect during the first three years. A large maintenance staff will not be necessary during this period. After the MARC expires the maintenance staff will increase. The additional mechanics will come from the ranks of the MARC personnel.

There is no blasting crew. Blasting operations will be conducted by a contractor.

Table 16.5.1 lists the supervisory and support staff personnel and

Table 16.5.2 shows the mine operating and maintenance crews along with an estimate of the number of additional personnel to cover vacations, sick leave and absences.

Table 16.5.1: Mine Supervisory and Support Staff Personnel

	Mine Production Years																		
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**
MINE OPERATIONS:																			
Mine Division Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
FL Supervisors	2	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Drilling/Blasting Supervisor	2	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Clerk																			
Mine Trainer	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Mine Operations Total	**8**	**8**	**7**	**7**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**10**
MINE MAINTENANCE:																			
Mine Maintenance Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
FL Supervisors Mnt	1	1	1	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Planner/Clerk				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Trainer				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Maintenance Clerk																			
Mine Maintenance Total	**2**	**2**	**2**	**5**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**6**
MINE ENGINEERING:																			
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Junior Mining Engineer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Draftsman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor Helper																			
Mine Engineering Total	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
MINE GEOLOGY:																			
Senior Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Sr Geotechnical Engineer																			
Geotechnical Engineer																			
Sampler	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Clerk																			
Mine Geology Total	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
TOTAL PERSONNEL	**20**	**20**	**19**	**22**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**26**

Table 16.5.2: Mine Operating and Maintenance Crews

	Mine Production Years																		
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**
MINE OPERATIONS:																			
Drill Operator	0	0	0	0	3	3	3	3	3	3	4	3	4	4	4	3	1	1	1
Surface Miner Operator	1	1	2	2	3	3	3	3	3	3	3	4	3	3	3	3	3	3	3
Loader Operator	1	1	2	2	5	5	5	6	6	6	6	6	6	6	6	5	3	3	4
Haul Truck Driver	5	7	10	14	29	32	32	34	37	40	43	46	48	52	55	50	29	30	33
Track Dozer Operator	1	1	1	4	7	7	7	7	7	7	7	7	7	7	7	7	7	6	6
Wheel Dozer Operator	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Grader Operator	1	1	1	2	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4
Service Crew	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Blasting Crew																			
Water Truck Operator	1	1	1	2	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4
Tailings Loader Operator	2	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Tailings Haul Truck Operator	3	4	5	4	8	8	9	9	9	10	10	10	11	10	11	12	12	13	12
Operations Total	**27**	**30**	**36**	**44**	**81**	**84**	**85**	**88**	**91**	**95**	**99**	**102**	**105**	**108**	**112**	**106**	**81**	**80**	**83**
MINE MAINTENANCE:																			
Mechanic	1	1	2	6	12	13	13	14	14	15	16	17	17	18	19	18	10	10	10
Mechanic's Helper	1	1	1	3	6	7	7	7	7	8	8	9	9	9	10	9	5	5	5
Welder	1	1	1	3	5	5	5	5	5	6	6	6	7	7	7	7	4	4	4
Electrician	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Fuel & Lube Man	4	4	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Tire Man	0	0	0	0	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Laborer	0	0	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Total	**7**	**7**	**8**	**16**	**43**	**45**	**45**	**46**	**46**	**49**	**50**	**52**	**53**	**54**	**56**	**54**	**39**	**39**	**39**
VS&A at 10.0	3	4	4	6	12	13	13	13	14	14	15	15	16	16	17	16	12	12	12
TOTAL LABOR REQUIREMENT	**37**	**41**	**48**	**66**	**136**	**142**	**143**	**147**	**151**	**158**	**164**	**169**	**174**	**178**	**185**	**176**	**132**	**131**	134
Maint/Operations Ratio	0.26	0.23	0.22	0.36	0.53	0.54	0.53	0.52	0.51	0.52	0.51	0.51	0.50	0.50	0.50	0.51	0.48	0.49	0.47

Notes:

1. Service Crew operates 980 Loader, Rock Drill, Excavators, etc.
2. VSA Basis: 10%

17 RECOVERY METHODS

17.1 Summary Flowsheet

During the FS different flowsheet options were investigated for the recovery of lithium from the Sonora ore. Following testwork and economic evaluations, the flowsheet was based on sodium sulfate roasting.

The Sonora Lithium Plant is proposed to be constructed in two stages. Stage 1 is designed to process 1.10 Mt/y of ROM feed, at 0.46% Li, to produce a minimum 17,500 t/y BG Li_2CO_3 and 17,000 t/y K_2SO_4. The potassium sulfate produced is expected to be sold as a Sulfate of Potash fertiliser. About 42,000 t/y of Na_2SO_4 is produced in Stage 1. This is not expected to be saleable and is therefore stored in a lined tailings storage facility or gifted.

Stage 2 involves adding a duplicate 1.10 Mt/y train, to be constructed for production in Year 5, to treat a combined total of 2.21 Mt/y of ROM feed, at 0.41% Li, to produce marginally more than 35,000 t/y Li_2CO_3, 28,000 t/y K_2SO_4 and 73,000 t/y Na_2SO_4.

Ausenco's SysCAD modelling of the final design shows that Stage 1 is can produce up to 21,113 t/y of battery grade Li_2CO_3 and 17,808 t/y K_2SO_4. Similarly, modelling shows for Stage 2 the plant indicates 35,918 t/y of battery grade Li_2CO_3 and 28,805 t/y K_2SO_4. These potential production volumes are higher than the design minima that were established as targets during early stages of the feasibility study.

Whilst the Stage 1 plant is largely duplicated for Stage 2 the modelled Li_2CO_3 and K_2SO_4 production for Stage 1does not simply double for Stage 2 due to a change in feed grade in Stage 2 when compared to the feed grade in Stage 1.

The flowsheet consists of the following unit processes:

- Beneficiation to recover lithium into a fine ground stream while rejecting coarse gangue using grinding, screening and hydrocyclone classification
- Sodium sulfate roasting, which converts the lithium to water soluble Li_2SO_4 at 900°C, in the presence of gypsum, sodium sulfate and limestone
- A hydrometallurgical section where the roast product is repulped in water to form an impure Li_2SO_4 PLS. Impurities are then removed from the PLS using precipitation and ion exchange prior to the evaporation and precipitation of battery grade lithium carbonate
- Potassium sulfate is recovered from the barren liquor using crystallisation and selective dissolution. The filtrate is returned to the sodium sulfate circuit
- Sodium sulfate is produced from the PLS via crystallization and stockpiled for either reclaim for reuse in the roasting circuit or for disposal or gifting.

A simplified illustration of the flowsheet is contained in Figure 13.5.2.

17.2 Process Design Criteria

The operating schedule for the plant is a continuous 24 h/d operation, using two 12 h shifts per day, 365 d/y. Design plant availabilities are typical at 90% (7,884 h/y) for the beneficiation plant and 83% (7,270 h/y) for the extraction plant.

The key PDC were used in developing the mass balance that forms the basis for the sizing of process plant equipment. The key elements were derived from the metallurgical testwork program. Section 13 provides a summary of the testwork programs.

Selected aspects of the PDC for Stage 1 and Stage 2 are summarized in Table 17.2.1. Statements of ore grade, concentrate grade and metals recovery relate to figures used for design purposes, rather than being a prediction of production, grade or recovery.

Table 17.2.1: Process Design Criteria Summary

Description	Units	Stage 1 Value	Stage 2 Value
Overall Lithium Recovery	%	78.0	74.2
Beneficiation			
ROM feed rate	t/h	140	280
Design ROM feed grade	% Li	0.46	0.41
Mass recovery to concentrate	%	82.5	75.2
Lithium recovery to concentrate	%	94.2	89.5
Extraction			
Leach lithium extraction	%	85.3	85.3
Glauber Salt Removal			
Discharge sodium concentration	g/L	91	91
PLS Evaporation			
Target lithium concentration	g/L	11	11
Lithium Carbonate Precipitation			
Target product grade	%	>99.5	>99.5
Potassium Recovery			
Evaporation temperature	°C	75	75
Bicarbonation			
Target product grade	%	>99.6	>99.6
Sodium Sulfate Crystalliser			
Evaporator discharge temperature	°C	100	100

17.3 Beneficiation Circuit

The purpose of beneficiation is to reject non-lithium bearing minerals (gangue) while maximising lithium recovery. In the process ore is ground to suitable size for the following extraction. In addition, limestone is added as required and the slurry dewatered to suit the following briquetting and roast processes.

17.3.1 ROM Feed

Run-of-mine (ROM) ore is delivered from the ROM pad by Front End Loader (FEL) to the ROM Bin. A static grizzly above the ROM Bin screens out oversize ore. The ROM Bin discharges via a feeder and conveyor to a single SAG Mill.

17.3.2 Primary Grinding

The 2.6 MW SAG Mill discharges onto a double-deck vibrating screen. The combined oversize material is discarded into a bunker where it is reclaimed by FEL and transported by truck to the dry tailings stockpile.

The SAG Mill Discharge Screen undersize is pumped to hydrocyclones. The overflow gravitates to the Concentrate Thickener, whilst the coarse underflow recycles back to the SAG Mill. Limestone is added to the SAG Mill via a conveyor to maintain a specific mass fraction of calcium in the beneficiated ore product (SAG Mill cyclone overflow).

The SAG Mill cyclone overflow is sampled and analysed by an On Stream Analyser for multiple elements including calcium grade, which is used to control the rate of limestone addition to the SAG Mill.

17.3.3 Thickening and Filtration

The SAG Mill cyclone overflow feeds high-rate Concentrate Thickener. The flocculated solids settle to underflow and are pumped to the Concentrate Filter Feed Tank. The thickener overflow is collected in the Beneficiation Process Water Tank for reuse in the beneficiation process.

Eight automated plate and frame filters treat the Concentrate Thickener underflow. Filtrate is recycled back to the Concentrate Thickener. The filter cake is reclaimed from beneath the filters by FEL and transferred to the extraction plant Filter Cake Feed Bin.

17.4 Extraction and Precipitation Circuit

The processes of the extraction circuit aim to produce battery grade lithium carbonate (Li_2CO_3). The Li is extracted by roasting with sulfate salts, gypsum and sodium sulfate, as well as $CaCO_3$. It has been found that using sodium sulfate minimises cost by decreasing gypsum requirement, lowers the roast temperature, and also improves lithium conversion. Roast is followed by water leaching of sulfates. The solution impurities F, B, Ca, Mg, Cs and Rb are removed and the K from ore and Na from reagents are also separated and removed as sulfate salt by products. Some of the Na_2SO_4 is recycled as a reagent in roasting.

17.4.1 Briquetting and Drying

The beneficiation concentrate is reclaimed by FEL and loaded onto the Pugmill Feed Conveyor via a dedicated bin, feeder and conveyor. The Pugmill Feed Conveyor and reagent conveyors are fitted with weightometers which are used to proportion the amount of gypsum and sodium sulfate to add to the Pugmixers.

Dry gypsum is milled and added onto the Pugmill Feed Conveyor via a bin and rotary valve. Sodium sulfate is reclaimed from a storage stockpile by FEL into a feed bin then then a feeder and conveyor to the Pugmill Feed Conveyor. Pugmixers blend the beneficiated ore, gypsum and sodium sulfate before transferring the blend to Briquetting Machines.

The blended feed is fed into the top of the Briquetting Machines which utilise oppositely positioned rollers to compress the feed into briquettes. The formed briquettes are conveyed to a screen, which separates formed briquettes and recycle the remainder back to the Pugmill Feed Conveyor. Correctly formed briquettes are transported from the screen as oversize.

The briquettes are conveyed to load onto a truck which transports the briquettes to a pad for solar drying.

The briquettes are deposited onto a solar drying pad. After drying in the sun, the briquettes are reclaimed and transferred to the Dried Briquette Feed Bin. The briquettes are then fed to roasting.

17.4.2 Roasting

The briquettes from solar drying are stored in the kiln feed bin and loaded via a belt feeder into the kiln. The material proceeds through each kiln to undergo the roasting reactions.

The briquettes are progressively heated up to temperature and then held at that temperature for 1 hour before being cooled to near ambient temperature.

The briquettes are discharged into bins. A static grizzly above each bin captures oversize briquettes which may have clumped together. The roasted briquettes are transferred from the discharge bins by apron feeders to the Leach Mill Feed Conveyor.

Gasses from the kilns are directed to a dry scrubber which utilises hydrated lime ($Ca(OH)_2$) as a neutralising agent. The unloading of the roasted briquettes generates dust which is similarly collected in a dedicated baghouse unit.

17.4.3 Milling and Leaching and Residual Removal

The roasted briquettes are conveyed to a 1.6 MW Ball Mill. The milled roast product discharges through a trommel. Recycled washings from the PLS Filter is added to the mill to achieve the target solids density.

The mill discharge reports to a hopper and is pumped to classification hydrocyclones. The cyclone overflow proceeds to the Roast Leach Tanks, whilst the coarse underflow recycles back to the Leach Mill. Mill trommel oversize material is collected in a bunker where it is reclaimed by FEL and transported by truck to the dry tailings stockpile.

During roast product leaching, lithium sulfate (Li_2SO_4), along with water soluble metal-sulfate impurities of iron, magnesium, calcium, aluminium, sodium, and potassium leach into solution. Following leaching, the slurry is then pumped to the Leach Thickener.

The Leach Thickener overflow (PLS stream) overflows to a tank which is then pumped to Purification.

The Leach Thickener underflow is dewatered and washed on a vacuum belt Leach Filter to recover entrained liquor. The filter cake is transferred by conveyors to the dry tailings disposal area where it is dry stacked.

The recovered filter wash water is re-used in the Leach Mill to maintain the mill discharge slurry density. Unused filtrate recycles back to the Leach Thickener.

17.4.4 Purification

The Purification process treats the PLS Leach Thickener overflow to remove impurities. Sodium carbonate (Na_2CO_3) is added which converts metal-sulfates to metal-carbonates, while also producing sodium sulfate. To maximise the amount of calcium carbonate ($CaCO_3$) precipitated, the temperature is maintained at 85°C by indirect steam heating via immersed heating coils.

The purified solution is then pumped to the PLS Filter Feed Tank before being filtered in plate and frame filters. Due to the fine particle size of the feed slurry, diatomaceous earth ('DE') is added as filtration aid to assist in the formation of filter cake.

The filter cake is periodically discharged into a bunker from where it is reclaimed by FEL and transported by truck to the dry tailings stockpile. The filtrate is collected in the PLS Filter Filtrate Tank. The adjacent Glauber Salt Feed Tank combines the PLS filtrate with centrate from the Glaserite and Sodium Sulfate Centrifuges. This solution is then pumped to the Glauber Salt Extraction circuit.

17.4.5 Glauber's Salt Extraction

The feed to Glauber's Salt Extraction is cooled to 10°C by applying a vacuum to facilitate flash cooling. The cooling of the liquor results in the formation of glauber's salt ($Na_2SO_4.10H_20$).

The solid Glauber's Salt crystals are recovered in a centrifuge, which yields a solid cake with a moisture content of 9% w/w. The centrate is collected in a tank, in which a portion of the stream is diverted to the Caesium/Rubidium Removal circuit. The remaining volume of centrate, containing lithium, passes through the secondary Glauber's Salt Heater to absorb heat before proceeding to PLS Evaporation.

The Glauber's Salt crystals discharging the centrifuge feed into a melt tank where they are melted to form a sodium sulfate solution. The liquor then proceeds to Sodium Sulfate Crystallisation.

17.4.6 Sodium Sulfate Crystallisation

The Sodium Sulfate Crystalliser is a forced circulation evaporative crystalliser, which is powered by two-stage mechanical vapour recompression. The crystalliser evaporates off water to produce an anhydrous sodium sulfate slurry.

The slurry discharging the crystalliser is pumped to the Sodium Sulfate Centrifuge to produce a filter cake containing 5% w/w moisture. A bulkof the stockpiled filter cake is recycled back to the briquetting process, the balance is periodically reclaimed and disposed of as waste or gifted.

The bulk of the centrate from the Sodium Sulfate Centrifuge is recycled back to the Glauber's Salt Feed Tank. There is also the ability to bleed of a protion of the centrate to the Effluent Pond to control the build up of impurities in the circuit.

17.4.7 Caesium/Rubidium Removal

To prevent the build-up of caesium and rubidium within the circuit, a portion of the centrate from the Glauber's Salt Centrifuge is separately treated.

The centrate feeds a crystalliser unit to promote the downstream crystallisation of alum-sulfate species. The solid Glauber's Salt crystals produced are separated from the liquor in a centrifuge, then transferred to the Glauber's Salt Melter Tank. The centrate is collected in a Holding Tank before being pumped to an Acidification Tank.

The centrate is dosed with sulfuric acid to lower the pH. The Acidification Tank overflows to the Precipitation Tank where caesium, rubidium and potassium form alums.

The slurry discharging the Precipitation Tank is pumped to a candle filter. The caesium and rubidium rich filter cake is discarded into a bunker where it is reclaimed by FEL and transported by truck to its own dedicated waste pond. The filtrate stream gravitates to a Neutralisation Tank where caustic is added to convert excess aluminium to aluminium hydroxide ($Al(OH)_3$). This neutralised slurry is then pumped to a Clarifier to separate the solid aluminium hydroxide from the liquor. The overflow stream is collected in a tank then pumped through the secondary Glauber's Salt Pre-Heater with the Glauber's Salt centrate.

The clarifier underflow is pumped to Purification in order to remove the solids using the PLS filter.

17.4.8 PLS Evaporation

The PLS Evaporator drives off water to increase the concentration of lithium in solution. This maximises the amount of lithium carbonate precipitated in the precipitation stage downstream. The evaporator is a falling film type utilising mechanical vapour recompression. The concentrated PLS discharging the evaporator is cooled and proceeds to Activated Alumina IX.

17.4.9 Activated Alumina Ion Exchange

The concentrated PLS is pumped through a number of AA columns, configured in series, for the removal of fluorine. When a column is fully loaded the absorption media is removed and replaced with fresh absorption media. The newest column is then placed in the last position.

17.4.10 Boron Removal Ion Exchange

The liquor from the calcium and fluoride removal step is the feed to the boron removal ion exchange. The ion exchange circuit consists of three columns in a lead–lag–regeneration configuration to enable continuous and automated operation to remove boron from solution.

17.4.11 Lithium Carbonate Precipitation

The Lithium Carbonate Precipitation circuit consists of four agitated tanks operating in batch mode. Three tanks are in operation at any time with the fourth tank either being descaled or ready to be brought into operation.

At the end of the batch cycle, the slurry is batch pumped to centrifuges where the lithium carbonate solids are separated from the liquorThe lithium carbonate solids are then repulped and recentrifuged to remove additional impurities and then transferred to Bicarbonation.

17.4.12 Bicarbonation

Lithium carbonate solids from the Lithium Carbonate Precipitation are batch fed into the Bicarb Dissolution Tanks. Carbon dioxide (CO_2) gas is bubbled into the Dissolution Tanks to convert lithium carbonate into soluble lithium bicarbonate ($LiHCO_3$).

Off-gas from the Dissolution Tanks is recirculated with recovered carbon dioxide gas from the Bicarb Crystallisation Tanks through a blower unit.

The solution discharging the Dissolution Tanks passes through a polishing filter to remove any residual insoluble impurities. The liquor discharging the Bicarb Filter is directed to an ion exchange circuit which utilises Purolite S950 resin to target removal of residual calcium and magnesium.

The ion exchange product solution passes to the Bicarb Crystallisation Tanks where lithium carbonate is recrystallised at a temperature of 95°C. The CO_2 evolved in the Crystallisation Tanks is recovered by removing the water vapour via a condenser and returning the remaining carbon dioxide to the blower unit.

The lithium carbonate slurry discharges to the Bicarb Surge Tank where it is batch pumped to the Bicarb Centrifuges to recover the solid lithium carbonate product.

The centrate and wash water from the centrifuges is predominantly recycled back to the Bicarb Dissolution Tank. The balance is recycled to the Pregnant Liquor Tank feeding the PLS Evaporator.

The Bicarb Centrifuge cake, with a moisture content of 20% w/w, proceeds via a feeder to the Lithium Dryer Feed Bin. From here, it is dried and packaged as battery grade lithium carbonate product.

17.4.13 Lithium Drying and Packaging

The lithium carbonate cake is transferred to the product dryer using a screw feeder. The rotary dryer uses compressed natural gas (CNG) and indirect heating to remove moisture.

Off-gas from the dryer is exhausted through the Lithium Dryer Baghouse by an induced draft fan. Entrained solids are returned to the dryer solids discharge stream. The captured dust and the dryer discharge are transported to the Lithium Silo via a bucket elevator.

A rotary valve on the Lithium Silo transfers the product to a feeder and then to the Lithium Product Microniser. The microniser reduces particle size to the buyers specifications.

Lithium carbonate exhibits slight hygroscopic properties which can lead to particle agglomeration over time. To minimise the absorption of moisture and prevent agglomeration, dry air is used in the Microniser.

A baghouse and induced fan is utilised to capture dust from the microniser. A feeder transfers the captured dust to a Bucket Elevator which stores the product in the Lithium Bagging Feed Bin. A rotary valve controls the amount of product going to the Lithium Packaging Plant via the Lithium Bagging Feeder.

One tonne bulk-a-bags are semi-automatically filled and placed onto pallets for storage. A forklift then transfers the loaded bags into a shipping container.

17.4.14 Glaserite Crystallisation

The feed to Glaserite Crystallisation is barren liquor (mother liquor) from Lithium Carbonate Precipitation. The glaserite crystallisation system is a multi-effect unit utilising flash evaporation and flash crystallisation.

The glaserite slurry discharging the crystalliser is then pumped to the Glaserite Centrifuge. The cake proceeds to the SOP Crystalliser whilst the centrate is recycled back to the Glauber’s Salt Feed Tank. A bleed of centrate is directed to the Effluent Pond to prevent a build-up of chlorides in the Glaserite Crystalliser.

17.4.15 Glaserite Decomposition

The solid cake discharging the Glaserite Centrifuge feeds into the SOP Crystalliser where it undergoes partial dissolution in filtered water. A gravity settling section on the perimeter of the tank allows for the continuous removal of solids-free brine.

The dense slurry of glaserite and SOP discharging into the Crystalliser is transferred to the agitated SOP Leach Tank. The slurry from this tank is continuously transferred to the SOP Centrifuge where it undergoes dewatering to yield a solid cake. This potassium sulfate cake then proceeds to the Potassium Sulfate Dryer Feed Bin, from where it will be dried and packaged.

17.4.16 Potassium Sulfate Drying and Packaging

The potassium sulfate cake is transferred to the dryer via a screw feeder. The dryer uses CNG and indirect heating to reduce the moisture content.

Off-gas from the dryer is extracted through the Potassium Sulfate Dryer Baghouse by an induced draft fan. Entrained solids are returned to the dryer solids discharge stream. The captured dust and the dryer discharge are transported to the Potassium Sulfate Product Bin via a Bucket Elevator. A rotary valve controls the amount of product going to the Potassium Sulfate Packaging Plant via a feeder.

One tonne bulk-a-bags are semi-automatically filled and placed onto pallets for storage. A forklift then transfers the loaded bags into a shipping container.

17.5 Services

17.5.1 Reagents

Reagents used in the process have been described above and include the following:

- Sulfuric acid is received by bulk road tanker. It is stored in the Sulfuric Acid Storage Tank and is distributed around the plant via a ring main.
- Sodium carbonate in solid form is received by bulk road tanker and stored in the Sodium Carbonate Silo.
- Agricultural gypsum is supplied in bulk by truck with a top size of 2 inches. It is stored on a stockpile and is reclaimed by FEL and dumped into a feed bin.
- Caustic soda solution is delivered to site by bulk road tanker, suitable for direct use in the process. It is stored in the Caustic Storage Tank and is distributed around the plant via a ring main.
- Fresh limestone is delivered to site in bulk and stored on a stockpile. A FEL reclaims the limestone from the stockpile and loads it into a bin which feeds a variable speed feeder and onto a conveyor where it is transported to the SAG Mill Feed Conveyor.
- Hydrated lime is used in the Kiln Gas Scrubber to neutralise off-gasses discharging the kilns. It is supplied in bulk and stored in a silo adjacent to the scrubber unit.
- Aluminium sulfate is delivered to site in 1,150 kg bulk bags which are emptied into an agitated mixing tank where process water is added to create a solution for distribution.
- Carbon dioxide is supplied in liquid form in cryogenic tank containers. The liquid is passed through a vaporiser and refrigeration system (which is part of the tank container unit), to produce CO_2 gas which is used in the Bicarbonation circuit. Excess CO_2 from the Bicarb Crystallisation Tanks is discharged into condensers to remove any residual water vapour prior to recompression and feed back into the Dissolution Tanks by the CO_2 Blowers.
- Activated Alumina is delivered to site in 800 kg bulk-bags.
- Magnafloc 5250 is used in both the Concentrate and Leach Thickeners. It is supplied as a powder in 907 kg bulk bags and initially stored in a feed bin.
- Diatomaceous earth (DE) is used as a filtration aid in the PLS Filter. It is delivered to site in 450 kg bulk bags which are emptied into a bin.
- Biocide, algaecide and hypochlorite are supplied in barrels and are dosed to the Cooling Tower using dedicated dosing pumps as required.

- Both the SAG Mill and Leach Mill utilises forged steel grinding balls. The SAG Mill uses 75 and 100 mm and the Leach Mill 50 and 65 mm. All grinding media is supplied in 2,000 kg bags.

17.5.2 Water Services

Water services include:

- Raw water supplied by bore fields
- Filtered water for make-up to the extraction plant process water and cooling tower, reagent mixing and feed to potable water plant, demineralised water plant and fire water
- Demineralised water (via a reverse osmosis plant) that is heated to wash battery grade lithium carbonate centrifuges and cooled to dilute the sulfuric acid and caustic streams
- Cooling water
- Potable water
- Gland water
- Chilled water used in the Bicarb Dissolution Tanks and Glauber Salt Crystalliser.

17.5.3 Other Services

Other services include:

- Compressed air
- Natural gas supplied to the plant via a main pipeline
- Steam produced by a natural gas fired Steam Boiler Package.

17.6 Equipment Selection

Equipment selection was undertaken via budgetary enquiries to multiple vendors for all major packages. Scopes of work and process data sheets were prepared for each equipment package to allow budgetary quotation preparation by vendors. These budget quotations were technically and commercially evaluated in order to determine the suitable selection of equipment for the Sonora project.

17.6.1 Kiln

Roasting kilns are required to convert lithium compounds within the ore to water soluble lithium sulfate. The kiln processes a bulk material which is also susceptible to glassing from temperatures in excess of 1,000 degrees Celsius. With the high mass flow rate the required heat is considerable; however, accurate control of temperature is also required in order to prevent glassing. This requires temperature monitoring and control down the length of the kiln.

The handling of the kiln waste gas is via a dry, hydrated system followed by a bag house. A dry system is advantageous due to the high cost of water on site and the ease of handling of the discarded waste for disposal.

The current solution selection offers significant capital and operating cost advantages. While a conventional kiln is selected, modifications are required. The technology of the modifications, however, has only currently been proven at laboratory scale. As the kiln modifications are still at a conceptual design stage there is a requirement for further development within the early detailed design phase to mitigate the technical risks together with the Kiln Vendors.

17.6.2 Evaporator / Crystallisers

The evaporator / crystalliser package is split into three main processing steps, the Sodium Sulfate System, the Pregnant Leach Solution Evaporator and the Potassium Sulfate System. The Sodium Sulfate System consist of a glauber salt crystalliser which utilises a multi stage flash crystallisation process, a melt tank and a sodium sulfate forced circulation crystalliser with mechanical vapour recompression. The Pregnant Solution Evaporator consists of a falling film evaporator. The Potassium Sulfate System consists of a Glasserite Evaporator and Crystalliser followed by decomposition process to extract potassium sulfate from the glaserite solids.

For the feasibility study the supply of the Evaporator / Crystalliser package components have been split between multiple Vendors. This was based from a technical evaluation on each system, test work outcomes and commercial pricing.

17.6.3 Mills

The mills are trunnion supported mills with variable voltage, variable frequency (VVVF) drive for the Semi-Autogenous Grinding 'SAG' and Ball Mill. Due to the low abrasive nature of the clay feed, steel mill liners have been used.

17.6.4 Briquette Machine

The briquette machines selected in the feasibility study, feature a twin auger feed system ideally suited to handling clay ores. The vendor that had been selected in the feasibility study had been chosen due to their previous experience in handling clay material. Whilst the selected units are smaller than other alternatives, multiple units increases operational flexibility.

17.6.5 Filters

The Leach Filter is a vacuum belt filter suitable for slurry containing chlorides at elevated temperatures. The belt filter selected in the feasibility study, whilst not the technically preferred offer, was technically compliant and provided commercial advantages.

The Concentrate and PLS Filters are plate and frame filters. The vendor which performed the test work provided a technically preferred solution. However, the filters offered were commercially cost prohibitive. The plate and frame filters selected in the feasibility study were technically fit for purpose and is from a vendor located in Asia. Whilst the Asian vendor is attractive in cost, they are not technically preferred due to control system limitations and size restrictions available in their range which results in a great number of units required.

The Precipitation Filter is a candle filter suitable for low solids content. The filter used in the feasibility study had been sourced from a vendor with previous experience in lithium applications.

17.6.6 Ion Exchange

The ion exchange in the feasibility study is a complete in line system with the vendor selected in the feasibility study chosen on commercial grounds. The system nominated in the feasibility study utilises sulfuric acid (H_2SO_4) and caustic (NaOH) for regeneration, both readily available on site. However, there is a concern that the gypsum formation may lead to plugging and possible scale build up which will need to be investigated further in the early design stage.

17.6.7 Centrifuges

The Precipitation and Bicarb centrifuges are peeler type centrifuges while the glauber's salt centrifuge in the caesium / rubidium removal area is a worm screen centrifuge. In the feasibility study an Asian vendor experienced in supplying equipment for similar lithium applications had been selected. This was due to the commercial advantage of the units heavily outweighing the technical advantage of the technically preferred Vendor. Whilst the centrifuges are capable of fulfilling the process requirements they have a lower capacity which results in a greater number of units, less efficient wash capabilities and subsequently a higher retention time.

17.6.8 Thickener

The Leach and Concentrate Thickeners are high-rate above-ground thickeners complete with a full truss bridge, multi-pinion system of planetary gearboxes, hydraulic rake drive system and control panel. Due to the temperature of the slurry there is a preference for a welded carbon steel shell.

Test work had been performed by one of the vendors with most of the equipment offerings similar. Vendor selection was based upon the most commercially attractive, technically compliant offer.

17.6.9 Conveyors

The conveyor package is a complete turnkey system inclusive of design and supply. As local Mexican and US vendors declined to provide a submission in the feasibility study an Australian Vendor has been selected. Given the large capital cost involved with this package, further layout work and conveyors' profile development are required in the next phase of the works. Additional quotations should be sought from Mexican vendors as total project costs of conveyors are highly dependent upon transportation cost and dependent on in country rates.

17.6.10 Microniser

Ultrafine grinding of the lithium carbonate product may be achieved via either a jet mill or a pulveriser. Whilst both alternatives have comparable costs and foot print, the pulveriser uses substantially less electrical power with no need for a dedicated compressor system. In the feasibility study only one vendor was capable of providing a pulveriser capable of achieving the required product size.

17.6.11 Pumps

As the clay slurry duties are non-abrasive selection has been made in the feasibility study based on a commercial preference. Similarly for the solution, fire and water pumps the most economically attractive vendor has been selected.

18 PROJECT INFRASTRUCTURE

Figure 18.3.1 shows the plant layout that was developed for Stage 1 and Stage 2 of the Project.

18.1 Site Access Roads

The access road from Bacadéhuachi to the site requires upgrading to allow the expected construction and operational traffic to use the road. The upgraded road will remain as an unpaved gravel road, approximately follow the existing track and includes widening the existing track and modifying maximum grades. The length of the access road is 14.7 km. The access road from Bacadéhuachi to the site includes the upgrade and/or installation of six (6) concrete floodway crossings and 29 culvert crossings.

18.2 Secondary Roads

The project will require the construction of secondary roads on-site. Typically, these roads are shorter in length (less than 2 kilometers), unidirectional and developed to a lesser standard than the main site access road. The roads should safely cater for a lower volume of traffic. The roads include access to the following areas:

- Bore field access roads
- Tailings storage facility access road
- Open pit access roads
- Mine waste dump access roads
- Pit dewatering bores access roads
- Mine administration and workshops access road.

18.3 Transport Roads

A road survey has been commissioned as part of the feasibility study by Bacanora on the highways and roads shown in Figure 18.3.2 available to transport cargo to Bacadéhuachi, Sonora, from four points of origin

- Guaymas, Sonora (TPP – Port)
- Nogales, Sonora (Mariposa Intl Bridge)
- Caanea, Sonora (Rail Spur)
- Nacozari, Sonora (Rail Spur).

It has been ascertained that all road routes are feasible to transport cargo of standard gauge.

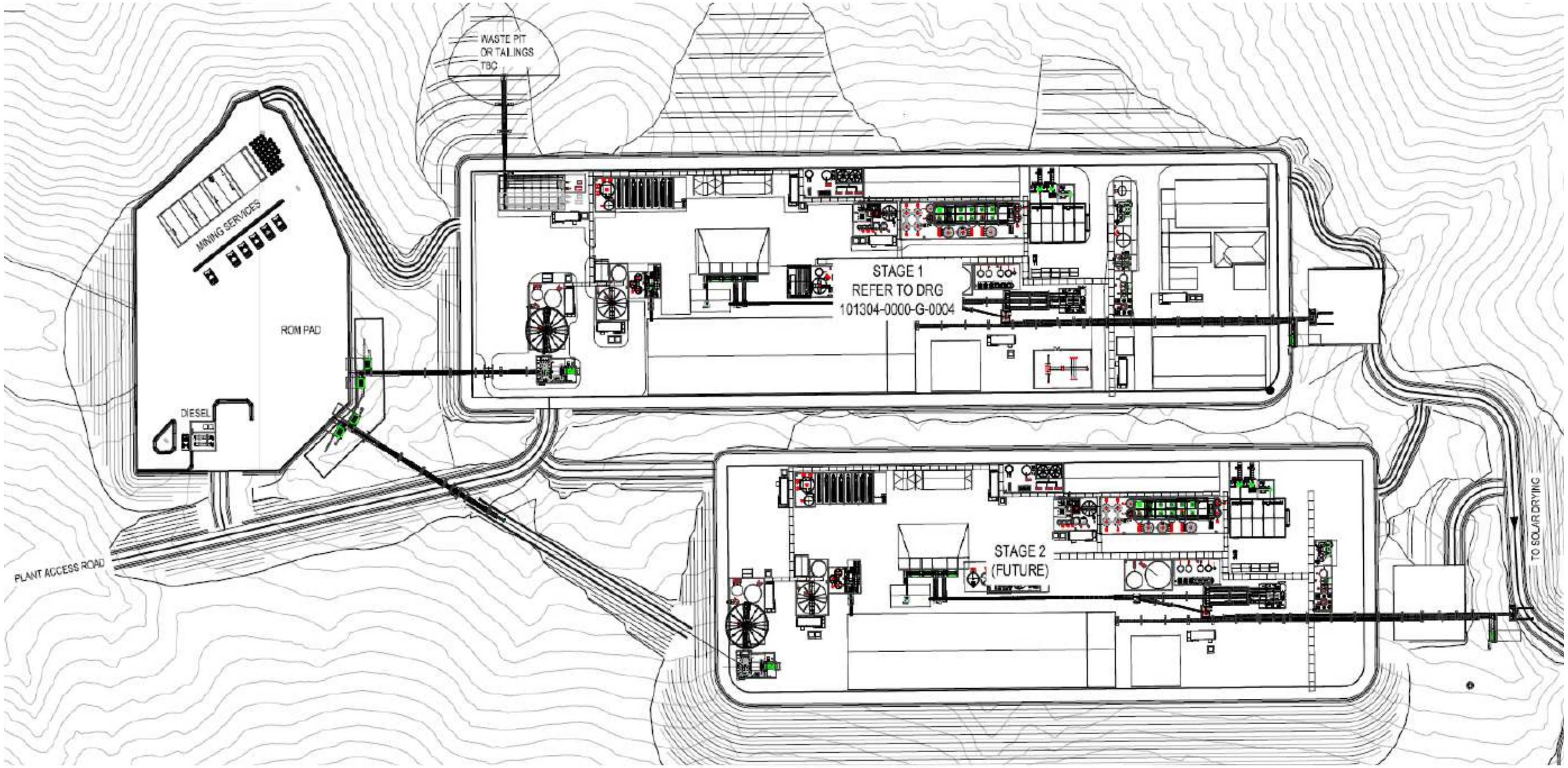


Figure 18.3.1: Plant Layout

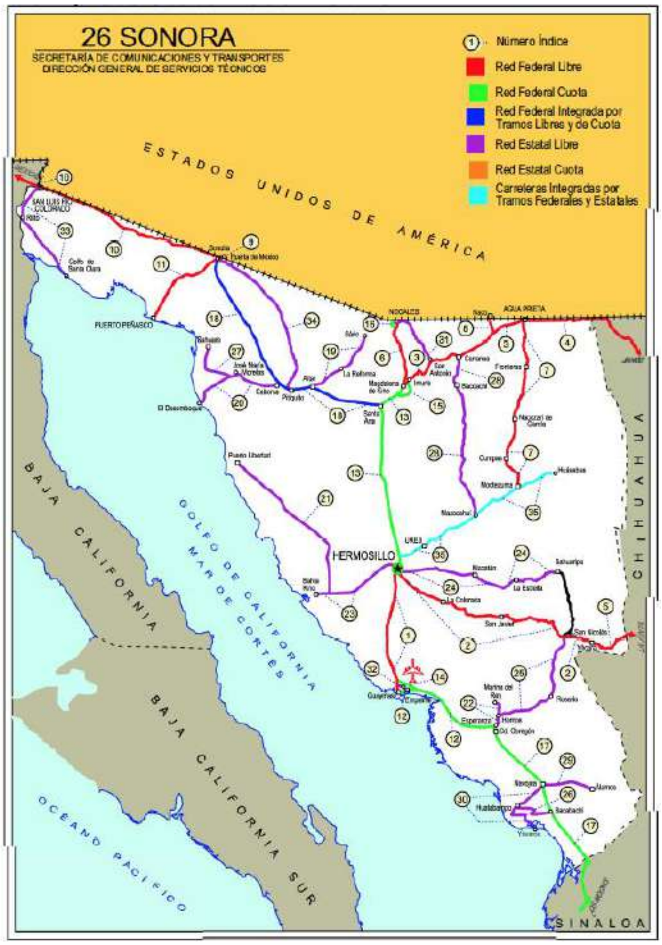


Figure 18.3.2: Complete State Road Network (Source: M3 177013_R0 Report)

18.4 Port Access and Product Delivery

Guaymas is a city located in the southwest part of the state of Sonora in northwestern Mexico. The city is located 117 km south of the state capital of Hermosillo, and 390 km from the U.S. border. It is the principal port for the state. Figure 18.4.1 shows the location of Guaymas in relation to the site, and Figure 18.4.2 shows the port. The port has road and rail access and container and bulk handling capabilities.

The Port of Guaymas will be utilised for the export of lithium carbonate to Asia. Trucked product in containers will be taken from site to El Coyote, which is situated on Federal Highway 14, then south west to Hermosillo and then south to the Port of Guaymas via Federal Highway 15.

Product being delivered to North America is expected to be trucked in containers to Hermosillo using Federal Highway 14 where they will be loaded onto trains and transported to the USA and Canada.

There is no foreseeable capital expenditure required at the port or rail loading site.

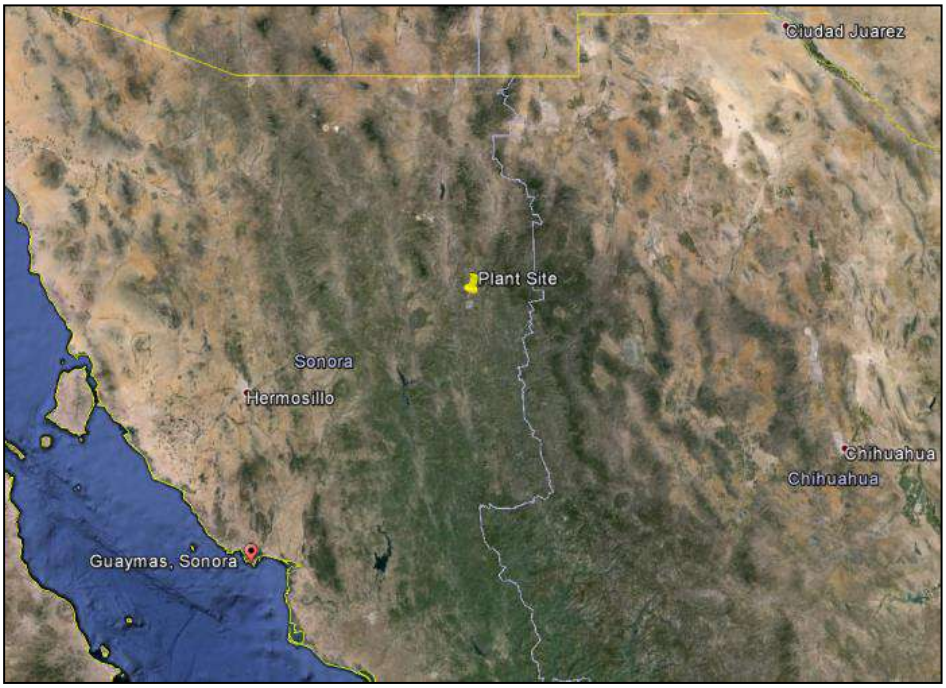


Figure 18.4.1: Location of Port Guaymas (Source: GoogleEarth)

Figure 18.4.2: Satellite Photo of Port of Guaymas (Source: GoogleEarth)

18.5 Traffic Management

The access roads have been designed in order to prevent interaction between the mining fleet and the light vehicle fleet. The heavy vehicle haul roads and their access to the mining services area is located on the eastern side of the light vehicle traffic. This side readily provides access to the open pit and the tailings storage facility.

The light vehicle fleet and the solar drying fleet share a common road between the plant site and the camp / solar drying. The solar drying fleet however is truck with live bottom trailer and does not present the same risk to light vehicles as the mining fleet. In addition the light vehicles in this region will be driven by site personnel, whom will be aware of the presence of the solar drying fleet.

18.6 Accommodation

The Stage 1 construction camp will be retained into operations and provide accommodation for operations staff. As operations progress through Stage 1 it is assumed that operations staff (local) will be progressively housed in Bacadéhuachi. It is anticipated that the project will stimulate development of housing and accommodation in Bacadéhuachi by increased economic activity in the area so that when Stage 2 construction commences the camp can be re-tasked once more, after minor refurbishment, as a construction camp for Stage 2, with the majority of the Stage 1 operations staff commuting from Bacadéhuachi.

18.7 Natural Gas

Bacanora has received proposals from potential build-own-operate (BOO) partners for the construction and operation of a natural gas pipeline. The pipeline would begin near Agua Prieta at the United States/Mexico border and continue to the Project site south of Bacadéhuachi. The pipeline is designed to transport the Project natural gas design requirements provided by Ausenco for Stage 2, approximately 1,000 GJ/h. The BOO partner will bear the entire capital cost of the pipeline project and be reimbursed through a transport surcharge in addition to the base cost of natural gas.

18.8 Electrical Power

The PFS indicated that electrical power would be provided from the existing electrical grid operated in Mexico by Comisión Federal de Electricidad (CFE). Further cost studies concluded that the installation of a natural gas turbine is a more cost effective solution resulting in minimal additional capex and a significant reduction in electrical costs. Bacanora has received proposals from potential BOO partners to construct and operate a natural gas combined heat and power (CHP) electrical generation facility to provide site electrical needs per requirements provided by Ausenco for Stage 2; one 25 MW unit per stage is estimated to be required. Heat is recovered from the CHP exhaust gas and utilized in the process to increase overall efficiency. The BOO partner will bear the entire capital cost of the electric power station project and be reimbursed through a combination of fixed and variable kWh rates for electricity.

18.9 Water Supply

Source of adequate water sources to meet the needs of the project is a critital component in the success of the Sonora Lithium Project. Solum Consuting Group was engaged to investigate potential water sources and prepare the documentation needed to present a water rights application to the regulators to purchase the water rights. The estimated annual raw water demand is 1.1 Mm^3 (40 l/s) for stage 1 and 1.9 Mm^3 (70 l/s) for Stage 2.

A Radial Collector Well (RCW) system of three (3) wells are located 7 km north of the plant site, will pump raw water to the raw water tank in the process plant. The raw water is then distributed throughout the process plant and to the mining and administration departments.

The RCW is located in an alluvial corridor associated with Rio Bacadehuachi. The RCW system can provide the water requirements for Stage 1 of the project for normal conditions. The uncertainty remains in surface flows through the dry season and sustained droughts to meet any increase in demands for the project.

Another source of water has been identified just south of the town of Bacadehuachi approximalty 12 km north of the plant site along the proposed site access road. Several wells are located in this area including the Bacadéhuachi town well. Field pumping test of these wells show this area can provide a backup system and if required addtional water to the project. It is strongly recommended that a supplemental water supply source be investigated, proven and developed (this cost is not allowed for in the capital cost estimate).

Potable water is processed using a water treatment plant utilising raw water drawn from the raw water tank. The water treatment plant includes filtration, reverse osmosis, ultra violet treatment and chemical dosing. The potable water is then stored in a plastic-lined and roofed tank. Pumps and pipelines distribute potable water to all demand points, including the mining department's facilities.

18.10 Sewage Treatment

The process plant and administration facility is fed to a central sewage treatment plant utilising pits and macerator pumps. The bioreactor sewage treatment plant concentrates solids material as sewage sludge, with the liquid effluent stream following chlorine dosing suitable for irrigation. The sewage sludge is stored within the sewage treatment plant to be periodically removed by a sewage contractor. This facility is suitable for the process plant only and an additional larger sewage plant will be required for the camp facility.

18.11 Buildings

18.11.1 Administration and Process Plant Building

The Administration and Process Plant Building provides offices and workstations for the Administration and Process Plant. The building includes a reception area, enclosed offices, a conference/training room, open plan office area for junior staff, photocopy and printer area, first aid and recovery room, kitchen and ablutions.

18.11.2 Mining Office

The Mining Office is intended to house mining staff with enclosed offices for senior staff and an open area for junior staff. The office includes a meeting/training room and a photocopy/printer area.

18.11.3 Process Plant Workshop-Warehouse

Steel-framed and cladded type construction was included for the plant workshop–warehouse. The workshop includes a 10 t overhead gantry crane and air compressors, workshop tools, workshop equipment, warehouse racking and shelving. The plant workshop–warehouse includes provision of offices for maintenance supervisors, planners and warehouse staff.

18.11.4 Laboratory

A fully-functioning sample preparation and assay laboratory is provided with a nominal capacity of 200 samples per day. The number of process plant samples is estimated to be 136 per day which allows 64 samples per day for mine grade control purposes. Environmental samples will be sent offsite for analysis.

18.11.5 Gatehouse

The gatehouse has an office, turnstile and boom gate, a drug/alcohol testing area, kitchen/meals area and two toilets.

18.12 Mobile Equipment

A list of mobile equipment for the process and administration for the operations around the process plant and the site is included in Table 18.12.1.

Table 18.12.1: Mobile Equipment List

Department	Vehicle	Number
Process General	FEL – Cat 988	1
	Forklift – 2.5 t	1
	Forklift – 10 t	1
	Forklift – Laden containers stacked 3 high	1
	Skid Steer Loader	1
	Trailer Mounted Pumps	1
	Yard Crane, 25 t Frana	1
	Crane Truck, 10 t HIAB	1
	HDPE Welding Machine – McElroy 8/24 machine	1
	Mobile Generating Set - 50 kVA	1
	Portable Light Plant	1
Finance & Admin HR & HSEC	45 Seater Bus	2
	Ambulance	1
	Fire Truck – Hino GT500	1
	Mine Rescue Vehicle	1
	Sewage Pump Out Truck	1
	Tip Truck – 10 t	1
	Pallet Jacks – 2.5 t	1
	Warehouse Forklift – 5 t	1
	Forklift – 10 t	1
TOTAL		**21**

18.13 Communications

An external communications link will be supplied via a fibre optic cable that is part of the BOO gas pipeline.

Mine site radio communications will be established to provide dedicated radio channels for the respective departments (e.g. mining, process plant, emergency response).

18.14 Tailings Storage Facility

Table 18.14.1 summarizes the (wet) quantities of tailings which will be produced for Stage 1 and Stage 2 when the plant is ramped up to full production.

Table 18.14.1: Stage 1 and Stage 2 Tailings – Design Production (Mt/y)

Description	Stage 1	Stage 2
Scrubber Oversize	0.19	0.38
Tailings Filter Cake	0.61	1.22
Leach Filter Cake	1.03	2.06
Sub-total	**1.83**	**3.67**
Sodium sulfate	0.08	0.15
Impurity removal precipitate	0.00	0.00
Sub-total	**0.08**	**0.15**
TOTAL	**1.92**	**3.82**

The impurity removal precipitate and sodium sulfate are expected to be water soluble and will therefore be stored in 50,000 m^3 double HDPE lined ponds with leak detection. It is proposed that additional cells (ponds) are installed each year.

The scrubber oversize, tailings filter cake (filtered beneficiation oversize and flotation tailings) and leach filter cake comprise 95% of the tailings to be produced and are currently expected to be benign. Bacanora personnel are currently proposed to load, haul, dump and spread these filtered tailings in the tailings storage facility. Further evaluation of overland conveying and stacking is recommended.

A geotechnical and geological investigation was carried out of the site from September to October 2016 that was located in the El Capulín stream gorge, where the infrastructure for the residue waste stockpile and waste rock dump will be located along with the inital process plant. The geotechnical investigation included drilling, test pits, geological survey and geophysical work with P and S wave velocity measurements. The plant location since has moved and a geotechnical reconisSance of the new site was carried out which included 10 shallow boreholes. A detailed geotechnical of the new plant location should be carried out.

During the 2016 program, thirteen (13) geotechnical holes were drilled and they were geotechnical and geomechanical logged. Permeability tests were carried out in selected boreholes at a variety of depths to measure soil and rock permeability. Fourteen (14) test pits were completed along with identification of potential construction materials.

The selected location of the tailings storage facility is located in the same watershed as the proposed mine infrastructure which reduces environmental impact and is located nearest to the plant site, reducing operating costs. The basic concept and management stratergy allow for the co-disposal of waste rock and dewatered tailings. Further detailed investigations and laboratory tests are proposed before the completion of detailed design.

A geochemical study was conducted of residue waste and waste rock. The test results indicate that both materials may have the potential to generate acid and the ability to leach metals. Selected samples of waste rock were randomly selected from both shallow and deep zones within the pit to provide a representative spatial distribution of materials. Samples were analyzed using "acid-base accounting (ABA) and paste pH using the Modified Sobek Method (NOM-141-SEMARNAT- 2003)". These samples were analyzed for total metals to evaluate the mobility of antimony, arsenic and chromium. These tests were performed by extracting water in equilibrium with CO_2 (NOM-157-SEMARNAT-2009).

Some of the results from the Upper Clay and volcanic base ignimbrite samples indicated that concentrations of some constituents exceed the norms listed in NOM-157-SEMARNAT-

2009. All the samples contained very low levels of sulfides, sulfates and total sulfur. Consequently, the samples had very low acid potential (AP). Due to the high neutralization potential of the rocks, the resulting value of net neutralization potential (NPR) indicate that the samples are not acid generating. The results of leachable metals from the representative waste rock samples contained relatively low levels of antimony, arsenic and chromium. The concentrations of leachable metals are below their respective allowable limits for antimony, arsenic from NOM-157-SEMARNAT-2009. The samples are not considered hazardous with regards to potential leaching of metals.

As shown in Figure 4.3.4 the tailings storage facility is located in a valley south of the ore body, adjacent to the plant.

The filtered residue waste disposal facility includes a compacted rockfill buttress. This buttress wil have a crest width of 8 m and upstream slope of 2.5H:1V and downstream slope of 1.5H:1.0V. Based on geochemical characteristics of the filtered residue, which are not acid generating does not require a geomembrane liner system.

During the first years of operation, this disposal facility will be independent of the waste rock placement downstream of the reservoir. Depending on the production rate of the waste rock, the placement of waste rock may be placed closer to the filtered residue waste facility until it meets the outer slope of the buttress to provide additional buttressing. In the later years of operation, the waste rock will provide a cover to encapsulate the residue waste.

It has been assumed that the porosity of waste rock is about 40%. For this level of study, it has been assumed that only 62% of the total porosity will be filled with residual waste. The porosity shall be verified when a more detailed blasting plan and operations plan has been completed. The placement of the residue waste into the spaces between waste rock will need to be verified in a pilot test program.

The overall north facing exterior slopes of the co-disposal facility is 2.5H:1V and will have a final height of 160 m after 19 years. The slope stability of final configuration shall be verified in further studies based on material properties tests and geotechnical testing results.

The interior of the co-disposal facility will include filtered residue waste with waste rock. There are several methods for co-disposal:

- Waste rock and the filtered residue are disposed simultaneously
- Filtered residue waste are added in the stacking of the waste rock
- Filtered residue waste are placed on top waste rock
- Filtered residue waste are placed in cells built with waste rock (paddock)
- Alternating layers of waste rock and filtered residue waste.

The method used (or combination of methods) will depend on the physical characteristics of each material and its behavior during co-disposal. For the next level of engineering it is recommended that tests with specific mixing ratio be examined to include strategies on the transportation of these materials.

Based on the current mine operational parameters, the total filtered tailings production during the 19 year mine life is estimated to be 38.0 Mt or 25.3 Mm^3 of tailings at an assumed tailings dry density of 1.5 t/m^3.

18.15 Mine Infrastructure

The mine infrastructure will include:

- Hardstand area (unsealed)
- Tyre change pad
- Vehicle wash down area
- Electric power supply
- Potable water supply
- Diesel fuel supply with day tank and high volume bowser
- Workshop shed with crib room, ablution, and offices
- Explosives magazine (remote location).

19 MARKET STUDIES AND CONTRACTS

19.1 Lithium Carbonate Market

The following information has been provided by Bacanora and SignumBox, a Chilean based natural resources research and consulting company with a specific focus on the lithium industry.

Lithium is used in a variety of industrial applications, the most relevant of which is energy storage (via lithium batteries). This is also the fastest growing sector for lithium due to rising demand in both the automotive industry and the portable consumer electronics industry. For 2017 the split of end use for the broader lithium market was as per Figure 19.1.1.

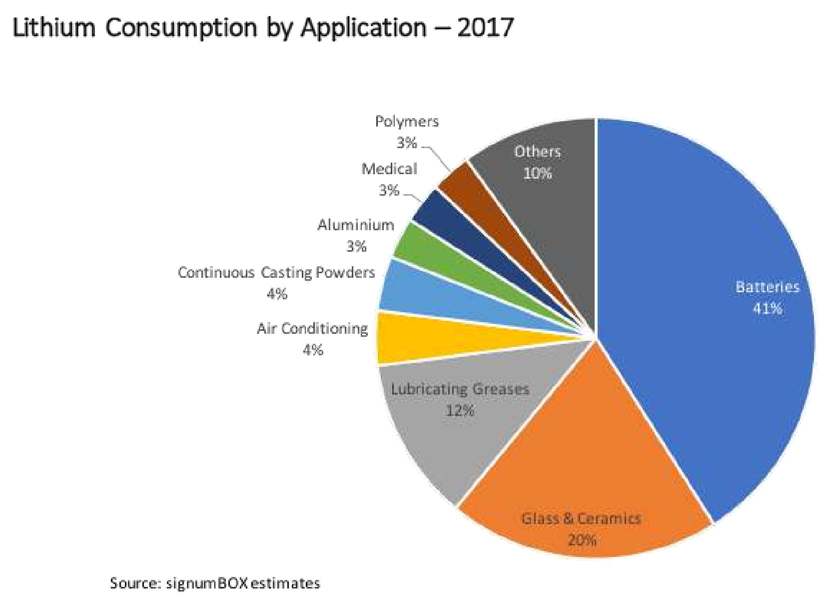


Figure 19.1.1: Lithium Consumption by Application – 2017

One of the most valuable uses of lithium is as a component of high energy-density rechargeable lithium-ion batteries. Lithium-ion batteries represent the fastest growing industrial demand for lithium. The use of lithium-ion batteries in electric powered forms of transport is expected to have a major influence on the lithium market. Because of concerns over carbon dioxide footprint and increasing hydrocarbon fuel cost, lithium is expected to become even more important in large batteries for powering all-electric and hybrid vehicles. Lithium batteries already enjoy a sizeable market, powering laptop computers, cordless heavy-duty power tools and hand-held electronic devices.

Due to the quantum of lithium used in electric vehicle batteries the electrification of the transport sector has the potential to effect a step change in demand for lithium. While electric vehicles currently represent only ~1% of global annual vehicle sales the number of electric vehicles sold has increased by [30 to 50%] per year in recent years. As economies of scale and, improvements in energy density and increased competition combine to lower the cost of electric vehicles it can be expected that high levels of growth could continue. In addition the regulatory environment in many key markets is shifting in favour of electrification of transport. Several countries including France and the UK have announced the future banning of vehicles powered by fossil fuels and China, currently the largest automotive market in the world, has put in place requirements for manufacturers to significantly increase the volumes of electrified vehicles sold.

Lithium supply is currently dominated by four producers: Tianqi, SQM, Albermale and FMC who jointly accounted for an estimated 73% of global lithium production in 2017. SignumBOX estimate total lithium supply in 2017 as 224,000 tons LCE. The following chart sets out the estimated market share by producer.

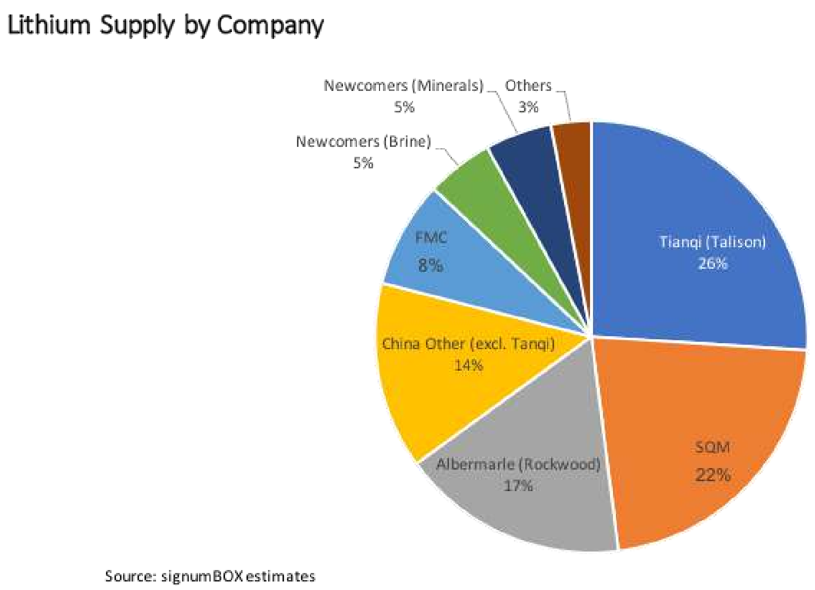


Figure 19.1.2: Lithium Supply by Company

Historically the majority of lithium produced has been from the brine producers but the share coming from minerals has increased significantly in recent years from ~34% in 2009 to ~44% in 2017. The largest producer of lithium minerals is Australia which is the source for ~30% of contained LCE produced globally. However, all of the lithium minerals produced in Australia are shipped to China as Li_2O spodumene concentrates where they are converted into lithium carbonate or lithium (ultimately) hydroxide. Given this, in terms of production of lithium chemicals, China is the largest lithium chemical producer with a ~44% share, followed by Chile with a ~37% share.

In terms of costs of production the Chilean brine producers enjoy a significant advantage given the fact that there is no mining and crushing involved and their location in arid regions enables them to utilise evaporative drying. This allows them to occupy the bottom quartile of the cost curve. Mineral producers on the other hand, have the costs associated with hard rock mining and also do not benefit from integration with chemical conversion.

19.1.1 Demand Forecast

The lithium market (as expressed in terms of volume of LCE) is currently growing in excess of 15% p.a. and in value terms has more than doubled since 2014/15 to an estimated $2.2 – 2.5 billion. Signum Box forecast that annual growth over the next 20 years will average 11.6% in their Base Case scenario. The bulk of the growth is anticipated to occur due to increasing demand from the battery sector implying continued strong growth for battery grade lithium carbonate and lithium hydroxide.

SignumBOX has performed a bottom up demand forecast for lithium in which they have estimated the use of lithium in each of the applications in which it is used. They have estimated three different demand scenarios broadly varying based on different potential outcomes for general economic growth and, most importantly, the development of the electric vehicle (“EV”) market which is anticipated to be the primary driver of battery demand for lithium.

In deriving the Base Scenario SignumBOX estimate the demand for lithium for EV batteries based on announcements made by each automaker combined with available information regarding sales and technical specifications for each type of vehicle. In 2017 SignumBOX estimate lithium consumption from EVs to be ~29,000 t LCE representing a 22% increase from 2016. They anticipate that this rate of growth will be sustained and supplemented by additional end uses such as stationary storage resulting in overall annual growth in lithium consumption from battery applications of 16.5%. Non-battery uses are anticipated to grow more in line with the general economy.

The Low Scenario still assumes robust growth in demand from battery applications but at less than half the current rate of growth over the 20 year forecast period. It also assumes a weak global economic outlook over the period and a concomitantly lower level of demand growth for non-battery demand. This scenario still sees a tripling of demand in terms of LCE over the next 20 years.

The High Scenario assumes a robust general economic development coupled with an extremely rapid adoption of EVs and a concomitant impact on lithium demand from this sector.

Table 19.1.1 summarizes the three demand scenarios.

Table 19.1.1: Summary of the Three Demand Scenarios

Scenario/Application	2017	2020	2025	2030	2035	2037	CAGR
Base Scenario (t/y)	190,500	270,700	521,300	926,100	1,473,200	1,708,200	11.6%
Batteries	67,500	129,000	331,700	676,800	1,205,200	1,436,400	16.5%
Rest	123,000	141,700	189,600	249,300	268,000	271,800	4.0%
Low Scenario (t/y)	192,000	232,800	326,800	432,600	536,700	580,400	5.7%
Batteries	67,300	100,400	173,300	260,600	366,100	409,700	9.5%
Rest	124,700	132,400	153,500	172,000	170,600	170,700	1.6%
High Scenario (t/y)	193,700	293,700	693,600	1,481,100	2,732,700	3,305,400	15.2%
Batteries	67,500	147,600	477,900	1,159,300	2,361,400	2,924,700	20.7%
Rest	126,200	146,100	215,700	321,800	371,300	380,700	5.7%

19.1.2 Supply Forecast

SignumBOX estimate that total lithium production in 2017 will be ~224,000 t LCE with the top four players responsible for 73% of this production. In terms of forecast growth in supply SignumBOX make the following points about sources of supply growth.

The expectation of robust demand growth is stimulating increases in production capacity for lithium. This is coming from traditional sources / players and also from new players. New players include brine projects in South America, particularly Argentina, and also hard rock pegmatite projects predominantly in Australia. Brine projects typically involve a material delay in ramping up production due to the two to three year evaporation ramp up and this delay has increased the focus on rock and clay projects.

SignumBOX have derived a view of how supply will develop going forward that encapsulates these new and expansion projects. The key uncertainties that they identify when developing their view are (i) the possible outcomes for SQM with regard to their license/quota in Chile, and (ii) the extent of new supply from Argentina where they see risks from both a technical standpoint to get projects built and ramped up and also from a political risk perspective.

19.2 Lithium Carbonate Price Forecast

Total lithium chemical supply has narrowly exceeded demand in recent years. The lack of timely capacity additions by brine producers in South America coupled with an increasing rate of demand growth will put pressure on the supply/demand balance over the next several years. Despite capacity being slightly higher than demand, due to long, complex supply chains, some of the capacity producing at a quality level that is unacceptable for use in the high growth battery market and the monopolistic behaviour of certain producers, prices have increased dramatically since the third quarter of 2015 from a global average price of lithium carbonate in the $6,000 /t range to over $12,000 /t in Q3 2017.

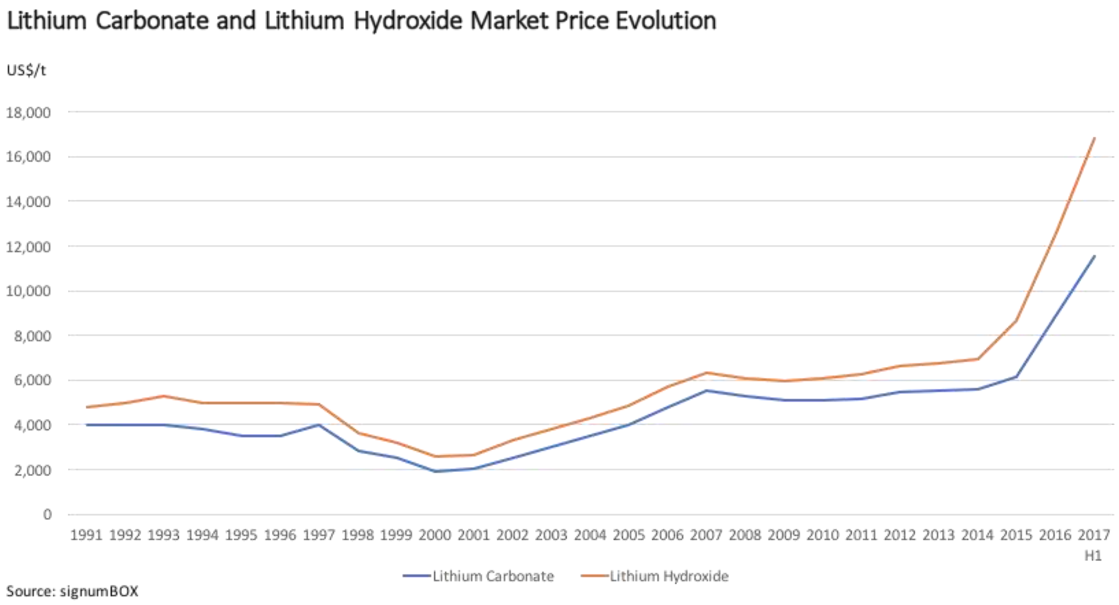


Figure 19.2.1: Lithium Carbonate and Lithium Hydroxide Market Price Evolution

While there is a spot market for lithium carbonate for which a market price can be observed, much of the industry's production is sold on negotiated contract terms.

Another important element of lithium pricing is the pricing differential for different grades. Much of the supply is for technical grade lithium carbonate which is ~99% pure. Battery grade lithium carbonate by contrast is >99.5% purity and commands a price premium. In recent history this premium has been between $1,200 and $3,500 /t of LCE. This reflects the segment demand within the lithium sector and also the relative cost of achieving the higher battery grade specification from input raw materials. Given that demand will be driven by the battery sector, SignumBOX anticipate that the premium will increase to over $5,000 /t over the next 20 years. The Sonora Project will produce battery grade lithium carbonate.

The following chart shows the Signum Box battery grade Li_2CO_3 price forecast under their Base Case and Low Demand scenarios. The Base Case envisages a battery grade pricing of ~$13,500 /t for the next five years, rising thereafter as battery demand primarily for electric vehicles increases and outstrips production increases. Even under the Low Demand scenario prices remain above $13,250 /t for the entire forecast period.

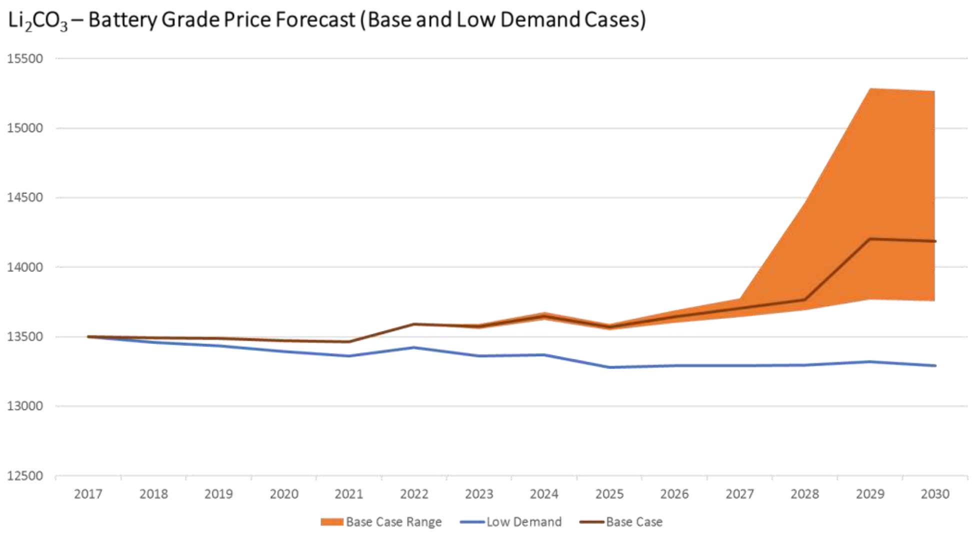


Figure 19.2.2: Li_2CO_3 – Battery Grade Price Forecast (Base and Low Demand Cases)

For the purposes of the DFS Bacanora has chosen to apply a price of $11,000 /t lithium carbonate which represents a material discount to the spot prices in Q4,2017 of $12,000 to $20,000 /t.

19.3 Potassium Sulfate

The primary by-product produced at the Project is potassium sulfate (K_2SO_4 or sulfate of potassium "SOP"). SOP is a high value fertilizer with particular application for producers of fruits, vegetables and nuts. The global demand for SOP is currently ~ 8 million t and is in deficit. The North American market is currently ~500,000 t. Mexico is an important market with annual demand of 50,000 to 90,000 t. California is also a large market with annual demand typically greater than 125,000 t.

Global production capacity is predominantly located in China. Within North America there is currently only one supplier, Compass Minerals, with operations in Utah. There are no existing Mexican producers. Mexican supply is typically sourced from Chile, China and Belgium.

Bacanora has commissioned a market study on the SOP market from Green Markets (Bloomberg). Green Markets forecast North American SOP pricing of $550/t for the next ten years.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Introduction

Environmental and social studies, carried out by Solum, are based upon the Sonora Lithium Project being located within the La Ventana basin which is a sub-basin of the Rio Bavispe Bajo. Investigations conducted include protected natural areas, flora, fauna, surface water, ground water and social-economic activities.

Environmental baseline data collection and reporting has been initiated for the mine site and processing area with no significant environmental issues being identified. The environmental baseline work includes a survey of biological, cultural, socio-economic resources and water quality.

The collection of environmental baseline data is required to support permitting efforts and project design. Baseline collection activities follow guidelines and study plans established by the authorities in Mexico and "International Lending Institution Standards' to satisfy potential financing interests and requirements for the project.

Ausenco and Solum established six (6) stand pipe monitoring wells, two (2) vibrating wire wells, four (4) piezometers and two (2) river stilling wells. These standpipe wells are monitored at least twice a year and the river stilling gaging stations are continuously monitored for water level and streamflow. These wells are used to monitor both water availability and water quality, establishing a baseline against which impacts can be quantified.

20.2 Permits

SEMARNAT is the chief agency regulating environmental matters in Mexico. The Comisión Nacional Del Agua (CONAGUA) has authority over water rights and activities that affect ground and surface water, including diversion of floodwaters.

Separate permit applications will be submitted for the three areas of activity, La Ventana site works (mine, tailings, process plant area, roads), site access (traffic and utility corridors), and the borefield. This strategy will allow for development work to commence in any area that is permitted without having to wait for the entire project area to be permitted.

SEMARNAT permits are mandatory to begin construction. The Environmental Impact Manifest (MIA), the Land Use (ETJ), and the Risk Analysis (RA) are only required if the project meets certain criteria. A land use license from the local municipality and an archaeological release letter from National Institute of Anthropology and History (INAH) are also required before starting construction.

It is a requirement of the Mexican government that before a Change of Land Use permit is issued, the Company has secured a federally approved occupation agreement (sale or temporary occupation agreement) with the land owner or owners (ejidos or communities) covered under the land use permit.

The natural gas pipeline permit applications are handled external to the project by the BOO partner. These permits (MIA and Land Use) are anticipated to take one year to complete and will include surface water, flora, fauna, archaeological, risk, and social impact studies.

20.3 Environmental Impact Manifest (MIA)

Environmental liabilities associated with mines typically fall within the following categories:

- Acid Mine Drainage
- Heavy Metal Contamination
- Processing Pollution

- Erosion and Sedimentation.

The MIA identifies specific environmental liabilities that will need to be addressed during the final design of the facilities. One important component to address the liabilities above is the mine closure. A properly closed mine with contain a specific plan for each liability for long term management.

The areas of the Project that have potential environmental liabilities in order of rank:

- Tailings Storage Facility surface water management components to reduce any erosion and sedimentation
- Plant Site containment systems within to avoid any pollution into the environment and contingency plans.

The Manifestacion de Impacto Ambiental (MIA), was submitted to the appropriate local authorities in Q2 2017. An approved resolution for the MIA was received in Q3 2017. The general requirements for a MIA include general description of the project and the responsibilities for the environmental studies. The MIA incorporates monitoring systems for the regulated facilities and closure requirements. Methodologies for monitoring and reporting requirements are detailed in the MIA along with the responsible parties. Preventive measures and mitigation of environmental impacts for each facility are also incorporated in the MIA.

MIAs for site access and the borefield are scheduled for submission to local authorities in Q1 2018 and resolutions expected in early Q2 2018.

20.4 Land-Use Change Application (ETJ)

All land in Mexico has a designated use, and any project areas not currently designated for mining must undergo a procedure to change the designation. The Estudio Tecnico Justifiactivo (ETJ) application is the formal instrument for changing the designation.

The ETJ requires an evaluation of the existing conditions of the land, including a plant and wildlife study, an evaluation of the current and proposed use of the land and impacts on natural resources, and an evaluation of plans to protect and save topsoil and certain plants and animals.

An ETJ will not be granted without a positive demonstration of agreements with all affected surface land owners. An ETJ was prepared for the process plant areas and submitted Q1 2018 for review and approval by SEMARNAT. The total duration typically is 60 working days and is required before any construction activity can begin.

ETJs for site access and the borefield are scheduled for submission to local authorities in Q1 2018 and resolutions expected in early Q2 2018.

20.5 Social and Community Impacts

The towns in the area (Bacadéhuachi, Nacori, Chico, Huasabas and Granados) have similar characteristics in terms of economic activity. Main activities are agriculture and livestock production. The average education is to high school level. In every town there are one or two medical centres and at least one doctor. Due to minimal employment sources and low economic development, perception towards new job employment is positive. The cultural features of these communities have been self-defined as supportive, enemies of conflict, mediators and seekers of common wellbeing. This can be observed in the high index of trust and social participation, which is mainly channelled to improve the community conditions.

During the visits performed to the Project site and the surrounding towns, contact has been made with the inhabitants and the feedback about the project has been positive. This is due to the perception the inhabitants have about the project as an employment source and therefore an improvement of the living conditions in these towns.

As with the environment baseline, a social baseline was developed for the Project and the surrounding towns. The general objective of this investigation is to develop a Baseline Study which analyses the social-demographic, political and cultural aspects, as well as the main problems and challenges that the communities in the area of Project influence face. The Baseline Study determines elements that guide the decision making in terms of social and relationship management and identify the possible social risks and measures to prevent them. A mixed study contemplating several interest groups was carried out to obtain quantitative and qualitative results.

In terms of social life, the surrounding towns are living peacefully, and are characterized by a calm and peaceful social environment. This environment is highly valued, almost comparable to the appreciation for their beliefs and respect to nature and it can be said that the main three axes of these communities are: safety/tranquillity, natural environment and religion.

When it comes to indigenous communities in this region, is important to note that the CDI reports the presence of two indigenous habitants in Nácori Chico. However, during the study field work no indigenous habitants were identified nor an indigenous community identified by self-definition, indigenous language spoken, worldview or own body representation. Nonetheless, some vestiges of the Yaqui and Optaean ethnicities are observed such as arts and crafts activities, the use of herbal medicine and gastronomy.

Bacanora Minerals should consider developing programs oriented to promoting economic autonomy in its social management plan through strengthening job capabilities and boost productive projects that do not depend on mining, in order to avoid negative effects at the end of the project. It is important that these programs are designed in conjunction with the community to include their expectations and potential insights.

20.6 Surface Water Management

Surface water management for Sonora Lithium project is defined based on the requirements of process water and stormwater volumes and surface water in the site area. There are no storage surface water bodies which have been identified in the influence project area which can be affected by contaminant discharges derived during project construction or operations.

The management of on-site stormwater will be by diversion channels, dikes and stormwater containment and sediment control structures. The sub-basin within the mine site watershed includes Las Huatas and El Capulin that converge to form La Ventana stream that empties into the Bacadehuachi River. Surface water runoff volume was calculated based on rainfall data according to weather stations close to the site and then generated based on probabilistic storm events.

The water balance for the metallurgical process includes the consumption of water for general mining and process operations. The rain and groundwater inflows accumulated in the pit may be used for dust suppression for roads and mine operations, with the remaining portion being evaporated. Rainwater accumulated in the contact water dam may also be used for dust suppression on roads and mine operations as required.

Surface water investigations have not identified any impediments for construction of the Project. The guidelines required by Secretaría del Medio Ambiente y Recursos Naturales ("SEMARNAT") will be met in terms of alteration to the riverbeds and streams. Corresponding permits for use, channelling, and/or storage of surface waters must be requested. Downstream ecological flow calculations of planned works must be performed and approved by SEMARNAT before the construction of such structures.

Surface water harvesting is not currently proposed excluding any inflows generated from the open pit during operations.

Solum has been collecting baseline water flows and water quality from 2015 through 2017. Prior to the collection of laboratory samples, surface water flow measurements and field water quality parameters were collected. Water quality results obtained as part of this study were tabulated and compared to applicable Mexican water quality standards and regulations. All samples taken to date are within the limits identified in the Mexican water quality standards.

The gaging stations on the Bacadehuachi River have been operating from Q3 2017 to present. These gaging stations were installed by Solum for estimating streamflows and are installed at stable stream cross-sections and consist of a stilling well and pressure transducer. Gaging station rating curves used to translate pressure to streamflow have been updated sporadically since installation.

Estimates of streamflow were made at two gaging locations during a site visit in September 2017, late into the wet season, baseflow was an estimated 100 liters per second (L/s).

20.7 Groundwater

The site is located in the area of the Bacadéhuachi aquifer, which is included within the Hydrological Region 9 South Sonora. This has a relief of distinct elevations, where most of the flow is born in the Sierra Madre Occidental that belongs to the Yaqui River Basin (B) and Bavispe-La Angostura River Sub-basin. The most important surface flow for the area is the Bacadéhuachi River. It starts in the Sierra los Ciriales to the east of the project area and heads north near the town of Santo Domingo, then change its direction southwest passing through the community Bacadéhuach and then through the town of San Gabriel where it joins the Bavispe River, which is southwest of the Bacadéhuachi aquifer.

In the site area composed of granular deposits, where all the exploitation was carried out, there is a heterogeneous aquifer, which is mainly in the zone of influence of the Bacadéhuachi River that has a high permeability floodplain since it was formed by a polymeric conglomerate of Quaternary.

Recharge zones are in the topographically elevated portions of the aquifer, where there are adequate permeability zones for infiltration of rainwater, such as the Los Ciriales mountains to the east and Huasabas mountains to the west that consists of fractured medium, mainly consisting of materials of volcanic origin such as: rhyolites, rhyolitic tuffs, basalts and andesites. These units represent important recharge zones. Springs located near the project area are related to these fractured systems. A smaller proportion of the aquifer is recharged by rainwater in the valley along with a small volume of recharge from agricultural irrigation.

The site is in the sub-basin of Rio Bavispe and the El Capulín, Las Huatas and La Ventana streams, whose flow regimes are perennial and crosses normal and inverse faults. The secondary permeability is through faults and fracturing that provide a conduit for groundwater. In terms of groundwater, as indicated by Acuerdo General (published in the Diario Oficial de la Federacioón, April5 2013), request mechanisms for grants, allowances and authorizations from CONAGUA must be followed in case drilling for groundwater use is required. The availability of groundwater annually from the Bacadéhuachi Aquifer is 10.7 Mm^3 of which 1.9 Mm^3 are used based on Public Registre from Conagua dated 30 June 2014. Of the total annual recharge of 10.7 Mm^3, 69,730 m^3 per annum are purchased water rights which equates to 9.93 Mm^3 available for purchase.

20.8 Protected Areas

The Project is located outside federal, state and municipality protected natural areas. The closest protected natural area “Campo Verde” is located 47.5 km from the Project.

The Project is also not within the limits of Importance Areas for Birds (Areas de Importancia para las aves – AICAS). The closest AICAS is called “Sistema de Sierras de las Sierra Madre Occidental” and is located 17 km west from the project polygon. The next closest AICA is called “Bacerac – Sierra Tabaco – Río Bavispe”, located 22 km east of the Project.

20.9 Flora

Three catalogued species of flora were found within the property limits; in two different categories in the NOM-059-SEMARNAT-2010. Non-endemic coniferous treelike species Sabal uresana Trel (white palm) and Cupressus lusitanica Mill (cypress) were found in the higher parts of the property and both are under the category of Special protection. Elements of Agave parviflora Torr. (Sóbari), a non-endemic species categorized as endangered, were also found spread out within different environments in the property.

The presence of these species in the project area will require performing a flora recovery of the specimens that are under any endangered category according to NOM-059-SEMARNAT-2010 prior to clearing and once the project has obtained all the permitting. With regards to the vegetation types that are within the area of interest, the following were found: oak forest, thorn scrub and subtropical grasslands. These types of vegetation are widespread throughout the state of Sonora and are not endangered. Flora species that are under protected categories according to the Mexican regulations and the ones with slow growth like cacti must be rescued before initiating any project works. Rescue works must be included in the mitigation measures of the project.

During sampling performed in October of 2015, 12 species were found in the Project area to be under the category of protected. These species must be relocated before operation. The existence of these species is compatible with the Project due to the fact that the habitat surrounding the area can be used for relocation. Further vegetation density studies were performed in Q3 and Q4 2017 and form the basis for the ETJ permit applications. A phased approach to ETJ permitting is planned such that deforestation activities occur concurrent with mine pit and tailings storage facility expansion.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The overall capital cost estimate is summarized by Stage and area in Table 21.1.1. The estimate has a base date of the fourth quarter 2017 (Q4 2017) and an accuracy range of ±15%. All costs are in United States Dollars.

The capital cost estimate has been compiled by Ausenco, with input from IMC for mining capital costs and Bacanora for Owner's costs.

Table 21.1.1: Estimated Capital Cost - Summary for the Two Stages

Area	Stage 1 $M	Stage 2 $M
Mining Equipment	14.2	17.6
Mining Infrastructure	3.4	0
Beneficiation Plant	18.5	18.5
Lithium Processing (Extraction) Plant	158.3	158.3
Common Plant Services	55.3	55.3
On-Site Infrastructure	37.8	20.5
Off-Site Infrastructure	21.0	3.1
EPCM/Owner's Costs/Indirects	72.9	72.4
Contingency	38.1	34.6
Total	**420**	**380**

21.1.1 Mining Capital Costs

The mining capital cost estimate was developed by Independent Mining Consultants, Inc. ('IMC') based on the purchase of new equipment.

The initial mining capital costs at $14.2M include:

- an initial fleet comprising a surface mining machine, a 13.8 cubic metre front end loader and three 90 t haul trucks. In addition, there is an ancillary mobile fleet including dozers, graders and front end loaders. The total capital cost of the equipment for stage 1 is estimated to be $14.2M.
- mining infrastructure capital costs were estimated by Ausenco at $3.4M.

The Stage 2 capital cost estimate of $17.6M is to purchase additional mobile equipment required for the increase in production associated with the process plant expansion in Year 4.

21.1.2 Direct Capital Costs (Process Plant and Infrastructure)

21.1.2.1 Process Plant Capital Costs

The capital cost estimates for process plant is shown in Table 21.1.2. The process plant capital cost estimate is based on an on-site processing plant comprising all new equipment, to produce battery-grade lithium carbonate.

Table 21.1.2: Plant Capital Cost Estimate (Excl. Reagents, Utilities, Services)

Description	Stage 1 ($M) rounded	Stage 2 ($M) rounded
PROCESS PLANT	**232**	**232**
Beneficiation	18.5	18.5
Roasting and Leaching	80.4	80.4
Solution Preparation	42.7	42.7
Lithium Carbonate	15.6	15.6
Sulfates	19.6	19.6
Reagents	8.6	8.6
Utilities and Services	18.4	18.4
Plant Wide Common	28.4	28.4
FIELD INDIRECTS	**10.6**	**10.6**
Temporary Construction Facilities & Utilities	2.0	2.0
Construction Support	3.9	3.9
Contractor Commissioning Assistance (by Client)	0.0	0.0
Accommodation and Messing	4.6	4.6
OTHER	**13.5**	**13.5**
First Fills	1.7	1.7
Spares	6.9	6.9
Mobile Equipment	5.0	5.0
ENGINEERING	**36.3**	**35.8**
EPCM Services	36.3	**35.8**
EPCM Commissioning Services (Inc in EPCM)	0.0	0.0
OWNER'S COSTS	**12.5**	**12.5**
Owner's Costs	12.5	12.5
PROVISIONS	**38.1**	**34.6**
Escalation (Excluded)	0.0	0.0
Contingency	38.1	**34.6**
FOREX (Excluded)	0.0	0.0
Total Cost:	**343**	**339**

21.1.2.2 On-Site Infrastructure Capital Costs

Onsite infrastructure includes power distribution, buildings, mobile equipment and a weighbridge and is shown below in Table 21.1.3.

Table 21.1.3: On-Site Infrastructure Capital Cost Estimate

Cost Area	Stage 1 ($M)	Stage 2 ($M)
Bulk Earthworks	34.3	20.5
Infrastructure Buildings	3.3	0.0
Fuel Storage	0.3	0.0
TOTAL ON-SITE INFRASTRUCTURE	**37.9**	**20.5**

21.1.2.3 Off-Site Infrastructure Capital Costs

Offsite infrastructure includes the site access road, borefield and water supply, tailings storage facility and the accommodation camp and is shown in Table 21.1.4.

Table 21.1.4: Off-Site Infrastructure Capital Cost Estimate

Cost Area	Stage 1 ($M)	Stage 2 ($M)
Main Access Road	7.4	0.0
Water Supply	6.8	0.9
Power Supply (Inc in OPEX)	0.0	0.0
Tailings Storage Facility	2.4	0.0
Accommodation Village	4.5	2.2
TOTAL OFF-SITE INFRASTRUCTURE	**21.1**	**3.1**

21.1.2.4 Direct Cost Development

Direct Costs include:

- Labour to undertake and manage the construction activities. This includes wages and salaries, with loadings for site labour, supervision and management, including associated expenses such as home and/or satellite office management expenses
- Supply of permanent materials and fixed equipment
- Contractors' and suppliers' mark-up and profit
- Transport expenses for permanent and temporary equipment and materials.

The direct costs were calculated based on the quantities listed in Table 21.1.5.

Table 21.1.5: Material Quantities

STAGE 1

Area Name	Clear and Grub ha	Strip Topsoil BCM	Cut BCM	Fill m^3	Concrete m^3	Platework (Non Liners) t	Platework (Liners) m^2	Platework (Rubber) m^2	Structural Steel t	Floor Grate m^2	Handrailing m	Cladding m^2	Pipework m	Electrical Only Cable m
PROCESS PLANT					**13,063**	**941**	**586**	**120**	**2,645**	**11,119**	**6,904**	**31,825**	**36,082**	**309,114**
Beneficiation					2,109	62	110	81	406	2,410	840	3,500	2,596	11,050
Roasting and Leaching					5,123	157	323	39	842	2,111	1,620	24,350	2,926	42,170
Solution Preparation					1,704	218			504	3,993	2,679	1,500	5,407	30,220
Lithium Carbonate					913	183			184	1,435	530	2,475	3,130	30,420
Sulfates					472	57			160	1,020	1,115		911	10,120
Reagents					894	112	153		75	150	120		6,388	17,470
Utilities and Services					622	152							14,724	12,150
Plant Wide Common					1,227				474					155,514
ON-SITE INFRASTRUCTURE	**0**	**23,684**	**1,682,654**	**784,045**	**21**									**1,000**
Bulk Earthworks	0	23,684	1,682,654	784,045										
Fuel Storage					21									1,000
OFF-SITE INFRASTRUCTURE						**11**							**9,924**	**1,400**
Water Supply						11							9,924	1,400
Total:	**0**	**23,684**	**1,682,654**	**784,045**	**13,084**	**952**	**586**	**120**	**2,645**	**11,119**	**6,904**	**31,825**	**46,006**	**311,514**

STAGE 2

Area Name	Clear and Grub (ha)	Strip Topsoil (BCM)	Cut (BCM)	Fill (m^3)	Concrete (m^3)	Platework (Non Liners) (t)	Platework (Liners) (m^2)	Platework (Rubber) (m^2)	Structural Steel (t)	Floor Grate (m^2)	Handrailing (m)	Cladding (m^2)	Pipework (m)	Electrical Only Cable (m)
PROCESS PLANT					**13,063**	**941**	**586**	**120**	**2,645**	**11,119**	**6,904**	**31,825**	**36,082**	**309,114**
Beneficiation					2,109	62	110	81	406	2,410	840	3,500	2,596	11,050
Roasting and Leaching					5,123	157	323	39	842	2,111	1,620	24,350	2,926	42,170
Solution Preparation					1,704	218			504	3,993	2,679	1,500	5,407	30,220
Lithium Carbonate					913	183			184	1,435	530	2,475	3,130	30,420
Sulfates					472	57			160	1,020	1,115		911	10,120
Reagents					894	112	153		75	150	120		6,388	17,470
Utilities and Services					622	152							14,724	12,150
Plant Wide Common					1,227				474					155,514
ON-SITE INFRASTRUCTURE	**0**	**14,000**	**1,682,654**	**107,320**	**21**									
Bulk Earthworks	0	14,000	1,682,654	107,320										
Fuel Storage														
OFF-SITE INFRASTRUCTURE						**11**							**9,924**	**1,400**
Water Supply						11							9,924	1,400
Total:	**0**	**14,000**	**1,682,654**	**107,320**	**13,063**	**952**	**586**	**120**	**2,645**	**11,119**	**6,904**	**31,825**	**46,006**	**310,514**

21.1.3 Indirect Capital Costs

Indirect costs include temporary construction facilities, spares, first fills, EPCM, commissioning, owner's costs and contingency as shown in Table 21.1.6.

Table 21.1.6: Indirect Capital Cost Estimate

Cost Element	Stage 1 ($M)	Stage 2 ($M)
FIELD INDIRECTS	**10.6**	**10.6**
Temporary Construction Facilities & Utilities	2.0	2.0
Construction Support	3.9	3.9
Contractor Commissioning Assistance (by Client)	0.0	0.0
Accommodation and Messing	4.6	4.6
OTHER	**13.5**	**13.5**
First Fills	1.7	1.7
Spares	6.9	6.9
Mobile Equipment	5.0	5.0
ENGINEERING	**36.3**	**35.8**
EPCM Services	36.3	**35.8**
EPCM Commissioning Services (Inc in EPCM)	0.0	0.0
OWNER'S COSTS	**12.5**	**12.5**
Owner's Costs	12.5	12.5
PROVISIONS	**38.1**	**34.6**
Escalation (Excluded)	0.0	0.0
Contingency	38.1	**34.6**
FOREX (Excluded)	0.0	0.0
FIELD INDIRECTS	**10.6**	**10.6**
Temporary Construction Facilities & Utilities	2.0	2.0
Construction Support	3.9	3.9
Contractor Commissioning Assistance (by Client)	0.0	0.0
Accommodation and Messing	4.6	4.6
OTHER	**13.5**	**13.5**
First Fills	1.7	1.7
Spares	6.9	6.9
Mobile Equipment	5.0	5.0
ENGINEERING	**36.3**	**35.8**

Cost Element	Stage 1 ($M)	Stage 2 ($M)
EPCM Services	36.3	**35.8**
EPCM Commissioning Services (Inc in EPCM)	0.0	0.0
OWNER'S COSTS	**12.5**	**12.5**
Owner's Costs	12.5	12.5
PROVISIONS	**38.1**	**34.6**
Escalation (Excluded)	0.0	0.0
Contingency	38.1	**34.6**
FOREX (Excluded)	0.0	0.0
TOTAL INDIRECT COSTS	**222**	**214**

The cost for all capital/operating spares was factored using a percentage established from previous experience, representing approximately 3% of the overall ex-works mechanical cost. Likewise an allowance of a further 3% was included to cover insurance/strategic spares.

An allowance for all commissioning spares were also included using a percentage established from previous experience, representing approximately 0.6% of the overall ex-works mechanical cost.

First-fill reagents were included in the estimate and were developed from the quantities and costs in the operating cost estimate.

Engineering, project management, project controls, procurement and contracting, and site construction management (EPCM) labour costs have been developed from first principles based on the developed schedule and expected engineering deliverables.

Engineering support labour costs for commissioning has been developed from first principles based on the developed schedule. Support from Bacanora operations and maintenance staff has been assumed.

Owner’s Costs were provided by Bacanora and include field staffing, travel, general expenses, basic office costs, and insurance. In addition allowances have been made for pre-production operating costs such as operational readiness labour, ERP software, termporary facilities, and visits to critical vendors.

Contingency refers to costs that will probably occur based on past experience, but with some uncertainty in regards to precisely how and where it will be spent. These uncertainties are risks to the project that are often referred to as "known-unknowns". A cost contingency of 10% has been applied.

21.1.4 Exclusions and Assumptions

The following items are specifically excluded from the estimate at this level of study:

- allowances for special incentives (schedule, safety or others)
- cost changes due to currency fluctuation and escalation

- force majeure issues
- Owner's Costs prior to project approval
- finance charges and interest during construction
- sunk costs
- future scope changes
- mine closure and rehabilitation costs
- costs for community relations and services
- relocation or preservation costs, delays and redesign work associated with any antiquities and sacred sites
- all costs associated with weather delays including flooding or resulting construction labour stand-down costs.

The following assumptions underlie this estimate:

- The design is as detailed in the relevant sections of this report
- Suitably qualified and experienced construction labour will be available at the time of execution of the project
- No extremes in weather will be experienced during the construction phase and as such no allowances are included for flooding or construction-labour stand-down costs
- All geotechnical design data was assumed due to the lack of geotechnical information at the proposed plant site and access road corridor
- Gas turbine power station and gas supply pipeline are BOO by the vendor.

21.1.5 Sustaining Capital Costs

Bacanora determined that the LOM sustaining mining and processing capital requirement is approximately $141 million.

21.2 Operating Costs

The operating cost estimate uses prices obtained in Q3 2017 and is considered to have an accuracy of ±15%. The estimate includes all site-related operating costs associated with the production of battery-grade lithium carbonate and potassium sulfate for sale as a Sulfate of Potash fertiliser.

The mining operating costs were developed by IMC while the process plant and administration operating costs were developed by Ausenco, in conjunction with Bacanora.

Table 21.2.1 summarizes the overall Stage 1 (Years 1 to 4) and Stage 2 (Years 5 onwards) operating costs assuming a steady state operation. Costs increase in Stage 2 due to the reduction in lithium feed grade resulting in less lithium carbonate production. Note that the operating cost is based upon a steady-state operation (hence, excludes ramp up).

This report details two battery-grade Li_2CO_3 annual production rates.

- The plant design data includes the use of SysCAD mass balance numbers based on steady state conditions. That is, the plant has been fully commissioned and ramped up to design parameters. The design steady state Li_2CO_3 annual production rates are 21,113 t/y for Stage 1 (based on a ROM Li grade of 0.46%) and 35,918 t/y for Stage 2. These production figures have been used for the operating cost calculations in Table 21.2.1.
- The costs in the financial model (Table 21.2.2) are based on the operating costs per unit process factored for key cost drivers per period and uses production rates during plant ramp up.

Table 21.2.1: Overall Operating Costs ($/t Li_2CO_3) – SysCAD

Category	$/t Li_2CO_3		
	Stage 1	Stage 2	LOM
Mining	295	499	471
Processing	3,093	3,266	3,243
General and Administration	263	209	216
Total	**3,651**	**3,974**	**3,930**

Table 21.2.2: Overall Operating Costs ($/t Li2CO3) – Financial Model

Category	$/t Li_2CO_3		
	Stage 1	Stage 2	LOM
Mining	325	511	490
Processing	3,418	3,169	3,198
General and Administration	296	212	222
Total	**4,039**	**3,893**	**3,910**

21.2.1 Mining Operating Costs

The mining operating costs are based on an owner operated fleet of newly purchased equipment to accomplish the mine production schedule, including maintaining haul roads and work areas, re-handle ore from the temporary stockpiles and maintaining the equipment. The mining costs are summarized below in Table 21.2.3.

Table 21.2.3: Mine Operating Costs

Year	Tonnes Mined (Mt)	$/t Mined									TOTAL ($M)
		Drill	Blast	Load	Haul	Auxiliary	General	Maint.	G&A	TOTAL	
1	1.19	0.03	0.04	0.38	0.74	0.74	0.11	0.12	0.46	3.07	3.12
2	1.94	0.05	0.05	0.35	0.7	0.45	0.09	0.1	0.28	2.43	4.02
3	3.12	0.08	0.09	0.32	0.66	0.28	0.07	0.08	0.17	2.01	5.47
4	4.36	0.14	0.14	0.23	0.55	0.39	0.06	0.08	0.14	1.87	7.56
5	9.6	0.09	0.15	0.22	0.52	0.32	0.05	0.07	0.09	1.63	14.58
6	10.8	0.1	0.16	0.21	0.52	0.29	0.05	0.07	0.08	1.57	15.83
7	10.8	0.09	0.16	0.21	0.52	0.29	0.05	0.07	0.08	1.57	15.79
8	11.2	0.1	0.16	0.21	0.52	0.28	0.05	0.06	0.08	1.57	16.39
9	11.2	0.1	0.17	0.2	0.57	0.28	0.05	0.06	0.08	1.62	16.94
10	11.2	0.1	0.17	0.2	0.62	0.28	0.05	0.07	0.08	1.68	17.51
11	12.1	0.11	0.18	0.19	0.62	0.27	0.05	0.06	0.07	1.66	18.74
12	12.1	0.1	0.16	0.21	0.66	0.26	0.05	0.06	0.07	1.69	19.04
13	12.1	0.1	0.16	0.21	0.69	0.26	0.05	0.06	0.08	1.73	19.44
14	12.1	0.1	0.17	0.2	0.74	0.26	0.05	0.06	0.08	1.78	20.1
15	12.1	0.1	0.17	0.2	0.79	0.26	0.05	0.06	0.08	1.83	20.69
16	10.03	0.09	0.15	0.22	0.87	0.31	0.05	0.07	0.09	2.01	18.61
17	5.78	0.06	0.1	0.28	0.87	0.54	0.06	0.08	0.15	2.39	12.33
18	5.71	0.06	0.1	0.28	0.89	0.5	0.06	0.09	0.15	2.4	12.06
19	6.3	0.06	0.1	0.28	0.9	0.45	0.05	0.08	0.12	2.29	12.89
TOTAL	163.7	0.1	0.15	0.22	0.67	0.31	0.05	0.07	0.09	1.66	271

Mining costs are summarized by category in Figure 21.2.1. Lube, repair, and wear parts are the largest mining operating cost followed by fuel.

The operating consumables cost estimate was based on the following input parameters:

- Diesel fuel at $0.61/L based on the September 2017 wholesale (rack) price
- Equipment maintenance costs per hour from quoted MARC hourly costs. The MARC prices were provided by Equite Montevedeo Group LLC (EMG) equipment database which is considered the industry standard for aggregated equipment supply contracts.
- Tire pricing is based on quotations received in 2017 from US based vendors. Pricing for lubricants is based on recent costs on similar projects and InfoMine Mine and Mill Equipment Costs Handbook.

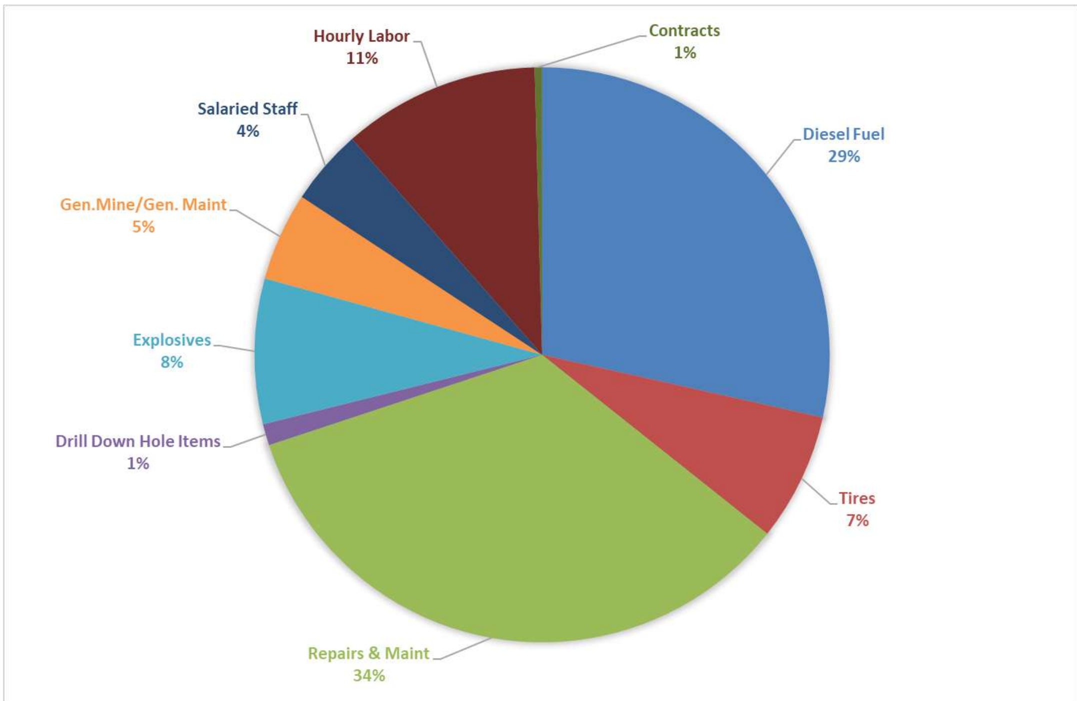


Figure 21.2.1: Mining Operating Costs by Category

The labour cost estimate is based on labour rates and rosters which were provided by Bacanora. Mining labour cost estimate is based on:

- Shift workers work 12 h shift, 14 day rotation
- Day workers work 10 h shifts, 5 days on 2 days off
- Burdens included at 35% of the base salary, which include coverage for overtime and leave, sick leave, annual leave, public holidays and payroll taxes
- All workers are based in Mexico. No allowances are included for expatriate staff and travel to and from their country of origin with the exception of mine management.

Table 21.2.4 summarizes the maximum number of mine workers in Stage 1 and Stage 2. Stage 1 maintenance is covered by a MARC agreement which reduces owner maintenance personnel and delays the capital expenditure for tooling, service trucks, and parts inventories. The MARC will end after Year 3, at which time the mine can hire the dealer maintenance technicians.

Table 21.2.4: Stage 1 and Stage 2 Mining Labour (No. People)

Labour Type	Stage 1	Stage 2
Management	22	28
Operations	42	107
Maintenance	18	62
Sub-total	**82**	**197**

21.2.2 Process Plant Operating Costs

Table 21.2.5 summarizes the Stage 1 and Stage 2 process plant operating costs based on the mass balance cost model. The costs summarised below also pertain to the mass balance cost model. This model was used as the basis for the costs for the financial model input.

Reagents and consumables are the key cost category representing 66% of the process plant costs in Stage 1 and 64% in Stage 2.

Table 21.2.5: Operating Cost Summary – Process Plant

Cost Centre	Stage 1		Stage 2	
	$M/y	$ /t Li_2CO_3	$M/y	$ /t Li_2CO_3
Labour	3.08	146	4.83	134
Power	12.52	593	25.04	697
Maintenance Materials	2.55	121	5.01	140
Reagents & Consumables	42.98	2,036	75.06	2,090
General & Administration	4.17	198	7.37	205
Total	**65.30**	**3,093**	**117.30**	**3,266**

The process plant operating costs increase from $3,093 /t Li_2CO_3 in Stage 1 to $3,266 /t Li_2CO_3 in Stage 2. This is largely due to the lower feed grade in Stage 2 (0.46% vs 0.41% Li).

21.2.3 Process Plant Labour Costs

The labour cost estimate was based on labour rates and rosters which were developed by Bacanora. The Process Plant labour cost estimate was based on:

- Shift workers work 12 h shift, 2 days, 2 nights, 4 off
- Day workers work 10 h shifts, 5 days on 2 days off
- Burdens included at 35% of the base salary, which include coverage for overtime and leave, sick leave, annual leave, public holidays and payroll taxes. Messing, bussing and accommodation are included in general and administration costs.

Table 21.2.6 summarizes the process plant labour cost estimate for Stage 1 and Stage 2.

Table 21.2.6: Process Plant Labour Summary

Labour Type	Stage 1		Stage 2	
	Numbers	Yearly Costs ($M/y)	Numbers	Yearly Costs ($M/y)
Management and Administration	35	1.01	43	1.14
Operations	102	2.27	182	3.47
Maintenance	32	0.81	57	1.36
Total	**169**	**4.09**	**282**	**5.97**

21.2.4 Process Plant Power Costs

The power consumption has been calculated for the beneficiation and extraction plants based on the installed equipment (i.e. excluding standby equipment) multiplied by the load factor in the mechanical equipment list.

The unit power cost used was $0.064/kWh as advised by Bacanora. Table 21.2.7 summarizes the power cost estimates for the Stage 1 and Stage 2.

Table 21.2.7: Process Plant Power Cost Summary

Area	Stage 1		Stage 2	
	Operating Power (MWh/y)	Yearly Costs ($M/y)	Operating Power (MWh/y)	Yearly Costs ($M/y)
Beneficiation Plant	23,324	1.49	46,650	2.99
Extraction Plant	140,899	9.02	281,798	18.04
Reagents	6,958	0.44	13,916	0.89
Utilities and Services	24,405	1.56	48,811	3.12
Total	**195,587**	**12.52**	**391,175**	**25.04**

21.2.5 Process Plant Maintenance Material Costs

The annual cost of maintenance materials for each plant area has been calculated by applying a factor to the area's installed mechanical costs. The factor is based on actual data from similar sized plants and is between 1 to 4%, depending on area.

21.2.5.1 Process Plant Reagent and Consumable Costs

Reagent consumption costs were based on testwork consumption rates, where available. Where reagent usage data was not available from testwork, consumption rates from Ausenco's database were used.

Table 21.2.8 summarizes the Stage 1 and Stage 2 reagents and consumables operating cost based on the design case (mass balance).

Table 21.2.8: Stage 1 Yearly Reagent and Consumables Operating Cost Estimate

Description	Unit Cost ($)	Units	Stage 1		Stage 2	
			Annual Usage	Yearly Cost ($M/y)	Annual Usage	Yearly Cost ($M/y)
Sodium Carbonate	248	t	48,917	12.11	82,420	20.40
CNG	5.60	GJ	1,574,985	8.82	3,149,971	17.64
Gypsum	25	t	182,092	4.55	331,951	8.30
Limestone	54	t	75,398	4.07	139,830	7.55
Sulfuric Acid	131	t	24,599	3.21	33,891	4.42
Caustic	374	t	7,371	2.75	8,844	3.30
Aluminium Sulfate	337	t	8,102	2.73	13,987	4.71
Miscellaneous				4.71		8.68
Total				**42.95**		**75.00**

21.2.6 Process Plant G&A Costs

The general and administration cost for the Process Plant covers items such as software licenses, training, consultants, mobile equipment, briqtte and oversize handling and light vehicles.

21.2.7 General and Administration Operating Costs

General and Administration include finance, human resources, health, safety and environment staff and general costs as itemized below.

Table 21.2.9 summarizes the Administration labour cost estimate.

Table 21.2.9: Stage 1 and Stage 2 Administration Labour Summary

Department	Stage 1		Stage 2	
	Labour Numbers	Yearly Costs ($M/y)	Labour Numbers	Yearly Costs ($M/y)
Finance, Administration and Management	25	0.72	32	0.83
Human Resources, Health, Safety and Environment	10	0.29	11	0.31
Sub-total	**35**	**1.01**	**43**	**1.14**

Table 21.2.10 summarizes the site General costs associated with operating the mine and process plant.

Table 21.2.10: General Cost Summary

Item	Stage 1		Stage 2	
	$M/y	$ /t Li_2CO_3	$M/y	$ /t Li_2CO_3
Rostered Travel, National – Admin	0.08	3.79	0.12	3.34
Training costs – Admin	0.01	0.47	0.02	0.56
Admin operating supplies – Admin	0.01	0.47	0.01	0.28
Computing Software – Admin	0.02	0.95	0.02	0.56
Medical Supplies	0.03	1.42	0.03	0.84
Recruitment	0.15	7.10	0.25	6.96
Government permits / relations	0.05	2.37	0.05	1.39
Legal and accounting fees	0.05	2.37	0.05	1.39
Licences, fees and permits	0.40	18.95	0.40	11.14
Operations and third party insurance	0.40	18.95	0.40	11.14
Employee health and environmental insurance	0.12	5.68	0.12	3.34
Entertainment	0.05	4.74	0.05	2.78
Communications	0.10	9.47	0.10	5.57
Community Support	0.20	2.84	0.20	1.67
Visitor Allowance	0.06	30.31	0.06	33.13
Camp hire costs	0.64	41.68	1.19	41.76
Li_2CO_3 product transport cost to port	0.88	35.05	1.50	33.41
K_2SO_4 product transport cost to port	0.74	9.95	1.20	7.52
Camp costs	0.21	15.63	0.27	9.19
Vehicles	0.33	3.79	0.33	3.34
TOTAL	**4.54**	**215.18**	**6.37**	**177.36**

22 ECONOMIC ANALYSIS

22.1 Outcome

An analysis of the projected capital expenditures, revenues net of royalties, operating expenses and corporate taxes was prepared on an annual basis to determine the estimated pre and post-tax cashflows from the project.

The economic analysis assumes the Project is 100% equity financed. The economic analysis includes the entire project life, comprising two years of detailed engineering and construction followed by approximately 20 years of operation.

Corporate sunk costs up to the project commencement, including costs for exploration, technical studies, and permitting are not included in this economic analysis.

The key inputs to the economic analysis are shown in Table 22.1.1.

Table 22.1.1: Key Inputs for Economic Analysis

Category	Units	Value
Li_2CO_3 Price	$/t	11,000
K_2SO_4 Price	$/t	550
Li_2CO_3 Process Recovery (Year 1)	%	53
Li_2CO_3 Process Recovery (Year 1 to 19)	%	77
Royalty – Colin Orr-Ewing	% of Li_2CO_3	3.0
Marketing	%	0
Mining Royalty Tax	%	7.5
Corporate Tax Rate	%	30

The Project annual cash flow is shown in Table 22.1.2.

The average annual revenue is $363M over the 19 years of operations. Average annual earnings before interest, taxes, depreciation and amortisation (“EBITDA”) estimated at $229M.

The Company intends to formally split its operations into Mining and Chemical Processing divisions and has taken advice from appropriate external advisers on the associated OECD compatible Transfer Pricing arrangements that must be entered into. The financial assumptions have been prepared according to this guidance. Mexican federal income tax depreciation and percentage depletion rules were applied to the appropriate capital assets and income categories to calculate the regular corporation tax burdens. A basic corporation tax rate of 30% has been assumed together with a 7.5% Mining Royalty tax due based solely on the mining parts of the operations.

In addition, there is a 3% royalty due on all product sales to Mr Colin Orr-Ewing, which has been included in the Life of Mine cashflows, with initial optimisation to assist in repayment schedules during initial funding and debt repayments. The Company has served notice in the Alberta Courts that it is challenging the validity of this royalty, but for the purposes of these financial projections, the royalty obligation is still included.

Table 22.1.2: Project Annual Cashflow Summary

Category	Units	Year -1, -2 Construction	Year 1 Stage 1	Year 2 Stage 1	Year 3 Stage 1	Year 4 Stage 1	Year 5 Stage 2	Year 6-19 Long Term	Total Life of Mine
Li_2CO_3	t	-	11,600	19,400	19,800	19,300	30,100	500,700	600,900
K_2SO_4	t	-	14,900	18,500	18,800	18,300	31,200	435,400	537,100
Net Revenue	$M	-	132.3	216.6	221.3	216.3	338.5	5,582.3	6,707.3
Operating Costs	$M	-	(59.9)	(72.2)	(74.8)	(76.1)	(132.7)	(1,934)	(2,349)
Capital Costs	$M	(398.7)	(20.9)	(2.3)	(161.3)	(209.5)	(18.3)	(122)	(933)
Pre-tax Cashflow	$M	(398.7)	51.5	142.2	(14.8)	(69.4)	187.5	3,527	3,425
Pre-tax NPV (8%)	$M	(398.7)	47.7	121.9	(11.8)	(51.0)	127.6	142	1,253
Post-tax Cashflow	$M	(398.7)	51.5	128.2	(50.0)	(102.4)	160.2	2,583	2,371
Post-tax NPV (8%)	$M	(398.7)	47.7	109.9	(39.7)	(75.3)	109.0	1,049	802

The project is currently estimated to have a payback period for Stage 1 of four years. The economic analysis indicates a pre-tax Net Present Value (NPV), discounted at 8%, of approximately $1,253M with an Internal Rate of Return (IRR) of approximately 26%. The post-tax NPV is approximately $802M and the post-tax IRR is 21%.

A sensitivity analysis on the base case NPV at different discount rates is shown in Table 22.1.3.

Table 22.1.3: Sensitivity Analysis – Discount Rate Impact

Discount Rate	Base Case Pre-Tax NPV	Base Case Post-Tax NPV
0%	3,425	2,371
2%	2,644	1,808
4%	2,054	1,382
6%	1,602	1,055
8%	**1,253**	**802**
10%	980	605

A sensitivity analysis has been conducted to determine the effect on post-tax NPV8% of $802 million and IRR of 21% from the base Li_2CO_3 price, operating cost and capital costs. Variations from +30% to -30% for each have been used in modelling. The analysis show the Project is most sensitive to the lithium price than it is to CAPEX or OPEX. As shown in Table 22.1.4 and Figure 22.1.1 an increase of 30% in the average lithium carbonate price, from $11,000 to $14,300, increases the Post-Tax NPV from $802M to $1,430M.

Table 22.1.4: Sensitivity Analysis – Post-Tax NPV8% ($ million)

Difference	Lithium Price	Operating Costs	Capital Costs
-30%	148	1,015	977
-20%	370	944	920
-10%	588	873	862
Base	802	802	802
10%	1,013	731	742
20%	1,222	659	680
30%	1,430	587	617

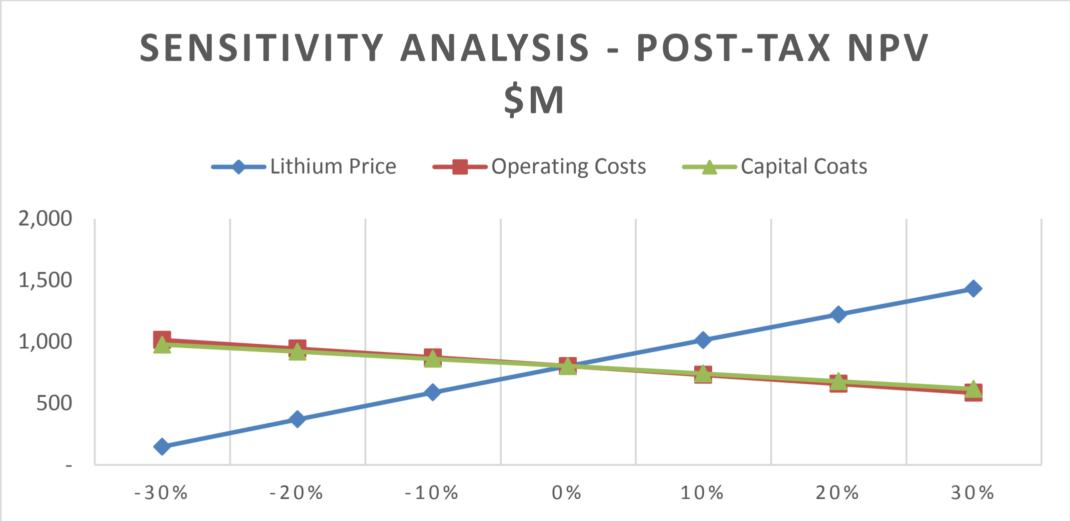


Figure 22.1.1: Sensitivity Analysis on Post-Tax NPV

A decrease of 30% in the average lithium carbonate price, from $11,000 to $7,700, decreases the Post-Tax NPV from $802M to $148M.

As shown in Table 22.1.5, an increase of 30% in the lithium carbonate price to $14,300, increases the Post-Tax IRR to 30%, while a decrease of 30% in the lithium carbonate price to $7,700 decreases the Post-Tax IRR to 11%.

Table 22.1.5: Sensivitity Analysis – Post-Tax IRR (%)

Difference	Lithium Price	Operating Costs	Capital Costs
-30%	11	24	30
-20%	14	23	26
-10%	18	22	23
Base	21	21	21
10%	24	20	19
20%	27	19	18
30%	30	18	16

23 ADJACENT PROPERTIES

No reference has been made to adjacent properties, the Sonora Lithium Project is the first such project to be developed in the area.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Implementation Schedule

Grupo Onza prepared the Manifestacion de Impacto Ambiental, which was approved and issued to Bacanora on Oct 11, 2017. In addition, the Company has designed an active program to engage with the local communities living within the project area.

Over the next six months, Bacanora will continue to progress the Sonora Lithium Project through the project development stages, with the intention of completing a Detailed Design (DD) in Q2 2018. The following preliminary indicative timetable is proposed:

- Q1 2018: finalise NI 43-101, FS
- Q1 2018: commence detailed design
- Q2 2018: commence long lead equipment procurement
- Q3 2018: commence site preparation works
- Q4 2019: commence commissioning.

A 'fast track' approach underpins the execution schedule which was developed during the FS assuming:

- Development work with critical vendors commences in Q1 2018
- An engineering company is appointed in January 2018, upon the conclusion of the FS, to support early works activities and critical vendor engagement
- An EPCM engineer is appointed in February 2018
- All final permits, approvals, and land access is granted in a timely manner.

Table 24.1.1 summarizes the manufacturing durations of the long lead items identified during this FS.

Table 24.1.1: Manufacturing Durations of the Long Lead Items

Equipment Package	Lead Time EXW (weeks)
Evaporator and Crystalliser	68
Briquette Machine	42
Plate and Frame Filter	38

Table 24.1.2 summarizes the manufacturing durations of the critical items identified during this FS.

Table 24.1.2: Manufacturing Durations of the Critical Items

Equipment Package	Lead Time EXW (weeks)
Evaporator and Crystalliser	68
Kiln	38

24.2 Ramp-up Schedule

24.2.1 Background

The Sonora ramp-up curve was based on evaluation of the McNulty curves as detailed in the paper – 'Minimisation of Delays in Plant Startups', Plant Operator's Forum 2004, Terry P. McNulty. There are four curves representing the plant throughput as a percentage of the annualised design over time. The different curves represent different metallurgical, process design and project execution considerations. The ramp-up curve types are:

- Type 1 – Mature technology, typical project execution. A Type 1 plant is typical of a concentrator with one stage crushing, SAG and ball mill and flotation
- Type 2 – New technology or equipment design, chemical plants. A Type 2 plant is typical of a concentrator with a complex crushing circuit (three stage crushing including HPGR), or a well-known chemical process
- Type 3 – As per Type 2, with limited pilot plant testwork. A Type 3 plant is typical of a hydrometallurgical sulphide leach plant
- Type 4 – As per type 3, with complex flowsheet. A Type 4 plant is typical of the nickel laterite pressure acid leach plants.

With an underperforming plant, it is usual to see additional capital invested to debottleneck and/or replace equipment not operating to design. In these cases the ramp-up curve will begin on one curve, for example Type 4, and then after 1 or more years of debottlenecking the ramp-up curve will shift to follow the next curve up, i.e. Type 3. This was observed in the Australian nickel laterite pressure acid leach plants.

Refer to Figure 24.2.1 for the four McNulty curves and the proposed ramp-up curves for Stage 1 and Stage 2.

24.2.2 Sonora Project Characteristics

To determine the ramp-up curve applicable to the Sonora Project, the future project characteristics including development of the technology, bench scale and piloting testwork, process design, flowsheet development and project development were reviewed against the four ramp-up curve types.

Whilst a demonstration plant has been constructed and operated the flowsheet varies from that proposed in the design and no detailed mass balance or significant operating data has been transferred into the design criteria.

Stage 2 is proposed to be a duplicate of Stage 1 and constructed and commissioned for production ramp up at the start of Year 5. The proposed Stage 2 ramp-up curve assumes that lessons learnt from Stage 1 are incorporated into the Stage 2 design.

Table 24.2.1 presents the project characteristics and their requirements to achieve a ramp-up curve type. The Sonora ranking is provided in the table for Stage 1 and Stage 2.

Table 24.2.1: Project Characteristic Rankings

Project Characteristic	Ramp-up Curve Type				Sonora Stage 1 Ranking	Sonora Stage 2 Ranking
	1	2	3	4		
Technology development	Existing technology, and multiple plants available for benchmarking	Existing technology, limited plants available for benchmarking	Existing or New technology, feed characteristics or mineralogy misunderstood	New technology, or process chemistry misunderstood	3	1
Testwork	Thorough pilot plant testwork on potentially risky unit operations, or thorough batch testwork on industry standard unit operations. Samples representative of the ore body.	Incomplete pilot plant testing (if required), or non-representative samples	Limited pilot plant testing (if required), or batch testwork steps neglected, Insufficient Attention to Product Quality During Development	Testwork to produce product only	1	1
Process Design	Standard equipment and sizes selected	Prototype equipment or sizes selected, or process conditions severe or corrosive	Prototype equipment or sizes selected, and process conditions severe or corrosive	Equipment Downsized and Process Design Criteria Less Conservative to Save Costs	3	1
Flowsheet	Standard gold plants, standard concentrators	Complex crushing, grinding, materials handling circuit, or complex multi-product hydrometallurgical circuits	Complex multi-product hydrometallurgical circuits with minimal plants in successful operation	New flowsheet design & technology with complex multi-product hydrometallurgical circuits and minimal plants in successful operation	3	2
Project Development	No shortcut in study work i.e. PFS and FS completed prior to detailed design	Non-process plant related constraints not understood (mining, power, water, local workforce)	Engineering, Design and Construction on “Fast Track”, serious design flaws		1	1

The following is a summary of the project characteristic ratings.

24.2.2.1 Technology Development

- Stage 1
  - Type 3:
    - The Sonora Lithium Process Plant consists mainly of existing technology, some of which has only limited use on an industrial scale in the lithium processing industry. However, the technology is established and used in other industries
    - The roasting technique is utilised in other industries but has not previously been used in a lithium carbonate plant.

            - Evaporators and crystallisers are typically used for the recovery of glaserite and sodium sulfate
            - Lithium carbonate precipitation is used in existing lithium plants.
- Stage 2
    - Type 1:
        - The technology will be established and benchmarked against Stage 1.

24.2.2.2 Test Work

- Stage 1
    - Type 1:
        - Bench scale and some bulk scale testwork has been carried out to optimise performance.
- Stage 2
    - Type 1:
        - Stage 1 will act as a complete full scale operating plant on representative ore.

24.2.2.3 Process Design

- Stage 1
    - Type 3:
        - The process incorporates mixed batch and continuous sequences, materials of construction are higher than average complexity, areas of the plant are susceptible to high scale growth, there is no resource pool to acquire experienced operations personnel and lithium plants historically have difficult ramp ups.
- Stage 2
    - Type 1:
        - Equipment sizing and selection will be well understood following Stage 1 operation.

24.2.2.4 Flowsheet

- Stage 1
    - Type 3:
        - The Sonora Lithium flow sheet is considered a complex hydrometallurgical plant
        - Lithium has not previously been extracted from lithium rich clays at commercial scale
        - Previous lithium (spodumene and brine) projects have not ramped up well.
- Stage 2
    - Type 2:
        - The Sonora Lithium flow sheet is considered a complex hydrometallurgical plant. The risks should be well understood following Stage 1 operation.

24.2.2.5 Project Development – Stages 1 and 2

The Sonora Lithium Project is proposed to be developed through these typical study phases to detail design, without shortcuts and therefore achieves the ranking of Type 2 for Stage 1 and Type 1 for Stage 2.

There is a series of other characteristics that if applicable can de-rate a projects ranking. At this stage of the project it is considered that these have or will be addressed in the future design and operation of the plant. These characteristics include:

- Corporate management had a promotional or overly aggressive attitude
- Driving forces underlying the project were ill-conceived
- Unanticipated increases occurred in costs of consumables
- Product prices declined unexpectedly
- The ore receiving and preparation areas received little attention
- Hands-on training of the workforce was inadequate
- Training manuals were inadequate or non-existent
- The supervisory staff was inexperienced
- Materials of construction were incorrectly specified
- Technical support during commissioning and start-up was inadequate
- There were serious engineering deficiencies
- Safety margins were inadequate
- Optimum mining technique is selected
- Reliable and consistent long term supply of raw water is proven.

24.2.3 Ramp-up Curve

Figure 24.2.1 shows the proposed Stage 1 and Stage 2 ramp-up schedules as compared to the four McNulty curves.

The proposed Stage 1 curve for the first 18 months is a Type 2 curve, with 100% capacity assumed to be achieved in month 36.

Stage 2 is a duplication of the Stage 1 process plant and therefore it is expected that the Stage 2 plant will ramp-up faster than Stage 1.

The ramp-up curve assumes that there is no interruption of feed ore to the Process Plant and there is no reduction in the plant operation due to a lower market product demand.

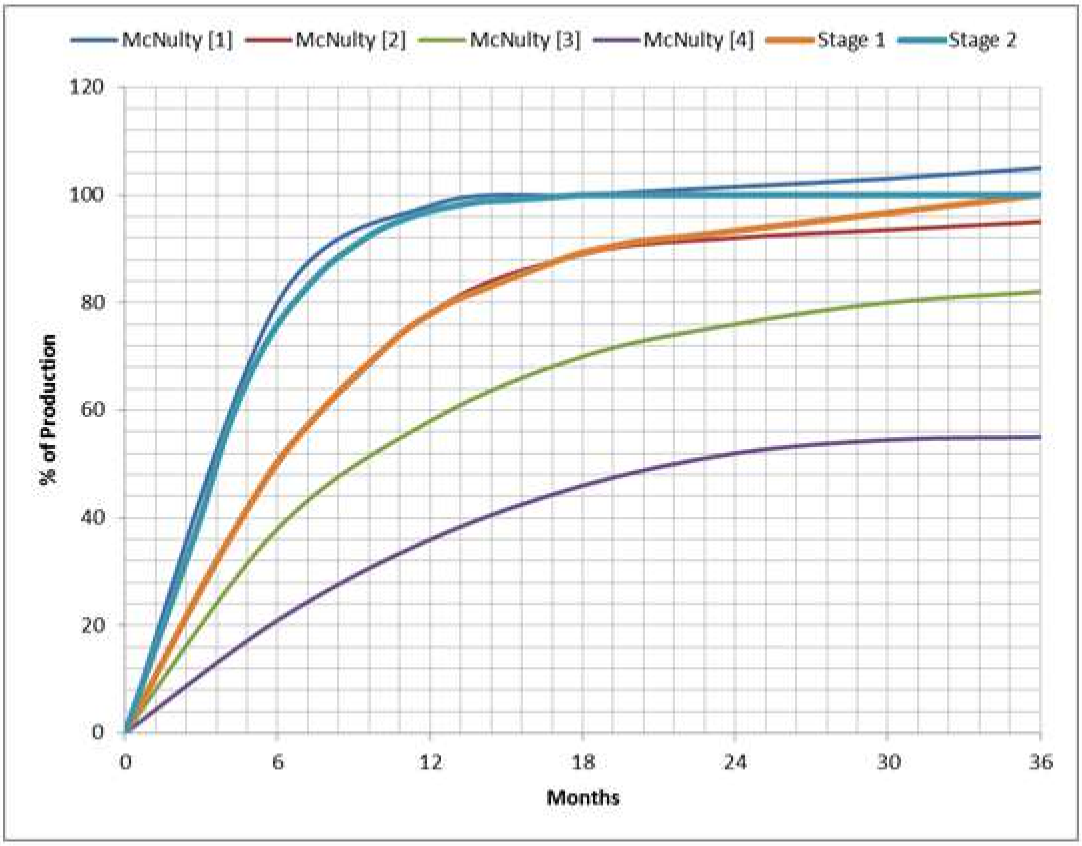


Figure 24.2.1: Stage 1 and Stage 2 Production Ramp-up Schedules

25 INTERPRETATION AND CONCLUSIONS

The following are the key interpretations and conclusions as well as risks and opportunities identified in the FS that need to be considered further during project execution.

25.1 Geology

The Sonora Lithium Project is substantial in size, with potential to produce several millions of tonnes of lithium carbonate product; it has a robust average grade compared with the cutoff grade which suggests there is potential to operate with a good profit margin. The Mineral Resource comprises 103 Mt of Measured Resource averaging 3,480 ppm Li for 1.9 Mt of LCE, 188 Mt of Indicated Resource averaging 3,120 ppm Li for 3.1 Mt of LCE and 268 Mt of Inferred Resource averaging 2,650 ppm Li for 3.8 Mt of LCE. The Mineral Resource is reported above a cutoff grade of 1,000 ppm lithium based on reasonably assumed technical and economic parameters and is constrained to an open pit shell which limits the resource to the near surface areas which have the best potential for economic extraction.

The 2017 Mineral Resource statement contains 33% increase in tonnage, 8% lower Li (ppm) grade and 22% higher contained metal when compared to the PFS. This reflects the updated geological modelling and the deeper resource pit constraint following updated costs and price assumptions used in the pit optimisation and cutoff grade analysis.

25.2 Mining

Since the completion of the PFS:

- Infill drilling has improved the confidence of the resource model by moving 99% of the reserve into the proven category
- Geotechnical work has improved the reliability of pit wall slope design
- More detailed engineering has improved the pit design
- Better road access from the pit to the ore and waste destinations
- A water diversion has been incorporated into the pit design
- 33 production schedule iterations have optimized the lithium carbonate production coupled with the plant ramp-up schedule on an annual basis.

25.3 Process Plant

Flow sheet development, locked cycle and variability testwork has shown the process flow sheet to be robust and stable at a feasibility level under laboratory conditions. The ability to produce battery grade lithium carbonate and by-products has been confirmed.

Although the project is considered viable, there have been risks identified that could impact delivery or economics and these need to be managed. The key aspects of the project presenting most execution risk are:

- On-time delivery of critical packages (kiln, crystalliser/evaporator) still requiring design development work.
- Adverse outcomes in the design development work for critical equipment packages may result in a detrimental cost impact, potentially linked to materials of construction for key components.

- Unexpected geotechnical data. The geotechnical assumptions underpinning the FS have been based on the limited available geotechnical data that was collected during the PFS and prior to the FS. This does not perfectly align with the proposed plant location but does give reasonable confidence in the assumptions that underpin the engineering work completed during the FS. Unexpected adverse results from the program currently in progress could have a detrimental capital cost implication.
- Stage 2 construction has limited lay down area in the immediate construction vicinity. There is sufficient space near the proposed solar drying area (stage 2 solar drying) but this will mean planning for traffic management is required to minimise interference with Stage 1 operations and the associated solar drying vehicle movements.
- Distribution and harvesting of briquettes for solar drying has been conceptually agreed but has not been field tested. The efficiency and success of this process operation may impact the feed consistency to the downstream operations.
- If NAFTA is cancelled it may have a detrimental impact on some equipment and reagent pricing that has been used in the FS due to increased taxes, duties, etc. There may also be a detrimental impact to schedule if there are new border and customs clearance requirements or delays that do not currently exist.

Other key risks to be noted include:

- The plant design assumes that all ROM ore is delivered after mining by a continuous surface miner which provides a -100 mm feed size to the ROM bin. No crusher has been provided so all material must be -100 mm for feeding to the SAG Mill. If this is not the case, additional capital expenditure may be incurred.
- Bicarbonation dissolution. The dissolution in the current design is at atmospheric pressure and the lithium carbonate dissolves within 2 hours. It is reported from the Demonstration Plant that dissolution is far slower. It is not known if this is due to the way carbon dioxide is sparged into the tank but data needs to be gathered and the current design refined, if required, as this may potentially be an issue with scale-up of the processes and equipment from laboratory to demonstration plant to full size process plant.
- Sodium carbonate is a significant contributor to the operating expense. The outcome of commercial pricing negotiations may have an impact on the forecast operating cost.
- Activated Alumina is an appreciable contributor to the operating expense. The outcome of commercial pricing negotiations may have an impact on the forecst operating cost.

25.4 Infrastructure

The feasibility study has shown that the infrastructure required for the project can be delivered. The key aspects of the project presenting most delivery risk for the infrastructure are:

- Successful finalisation of BOO contracts for the power station and gas pipeline. A delay in either of these will delay the completion of the project, particularly commissioning, or have a detrimental economic impact if alternate power and fuel sources need to be secured.
- Electrical loads need reconfirming prior to finalisation of BOO contracts being finalised as forecast loads leave negligible capacity in the event any change in electrical demand.

- Securing of right-of-way access for a yet to be determined gas pipeline route. If access cannot be secured it can reasonably be expected to delay the execution of the project unless a design change occurs (possibly trucking an on-site storage of gas instead of a pipeline). This may have a detrimental cost impact and may result in additional permitting requirements and/or bulk earthwork, subject to separation distances and storage volumes
- Securing of land access and permits for the borefield. An inability to secure access to water can be expected to cause significant delays
- The land for the proposed camp location lies outside the current mining lease. Failure to secure access to this land will have an impact on the earthworks cost associated with the camp establishment.

25.5 Environment

Work to-date has demonstrated that project can expect to receive all necessary environmental permits and licences. They key risks that may impact the project include:

- Successful approval of Land Use applications is required so as not to delay the start of construction activities.
- Delay of water rights approval to the project may delay the start of operations.
- The currently approved water permit is sufficient for Stage 1. A variation (increase) will be required to facilitate Stage 2. There is sufficient time for an application for an increase to be submitted, and approved, prior to Stage 2.

26 RECOMMENDATIONS

The following subsections summarise the recommendations and forward work plan for the project.

26.1 Geology

Bacanora has implemented all the recommendations suggested following the 2016 PFS report, including improving the quality control procedures, analysing all >10,000 ppm upper detection limit results and density determinations.

No further exploration is currently planned by the Company following completion of the Feasibility Study. Further work may be planned as part of the pre-production plan. In order to ensure small-scale features, such as minor faults, are understood fully prior to production, grade control drilling must be undertaken. The spacing of this drilling is yet to be confirmed, however, should be relative to the size of the mining panel.

26.2 Mining

Recommendations given in the PFS report and the status of those recommendations are:

- Infill drilling to ensure 5 years of Proven Mineral Reserves.
    - Infill drilling has been conducted resulting in all years of the 19-year mine plan having proven reserves except Year 17 which has less than 1% in the probable category.
- Geotechnical test work and update pit wall slope recommendations.
    - Ausenco has completed this work and issued a report “La Ventana Pit SlopeDesign Report, December 2016”.
- Detailed proposals for potential contract mining operations.
    - This work is yet to be done.

      NOTE: Contract mining may be used for removal of waste upper clay and capping basalt. Opex and Capex estimates already include the use of Bacanora personnel and equipment to perform this work but detailed proposals should be solicited and evaluated for potential cost savings.
- Evaluate the waste storage locations and plant locations to minimize haul distances.
    - The plant location has been evaluated.
    - There is opportunity for further optimization of the waste storage locations.

Additional recommendations from the FS work include:

- Develop standard operating procedures for:
    - Mining ore with the surface miner
    - Mining waste material adjacent to ore
    - Upper waste stripping.
- Develop a mine operations block model of the clay zones to include any vertical changes in grade within the clay ore zones.
- Develop sampling procedures to identify the Lithium grade trends in the host clay material to facilitate blending of different grade ranges in the plant feed.
- Prepare RFP’s for commercial procurement of mining equipment.

The preparation of the RFP's for mining equipment quotations can be done during the plant construction period, unless some of the mining equipment will be used for the earth works at the plant location. If so, then the RFP's for such equipment will be needed prior to project execution. All the other recommendations can be done during the plant construction period. If this work is completed by outside consultants or contractors, the estimated cost is $250,000 (cost allowance for this work by outside consultants or contractors is not included in the capex).

26.3 Process Plant

Recommendations include:

- There is an opportunity to optimise the kiln and leach feed size as well as the optimum reagent mix for the kiln. This could significantly reduce operating costs (improved economics) by reducing gypsum and limestone requirements. Alternatively, a specific roast recipe could be developed to reduce extraction of specific impurities, such as rubidium and boron, and reduce the associated downstream removal costs. It is recommended to conduct further testwork to optimise kiln and leach feed size and reagent mix ratio. This should include optimums for different lithlogies/mineralogies. This testwork can be done at anytime; however, the earlier it can be done the greater the possible reduction in capital expenditure. This work is expected to cost less than $10,000.
- A conventional kiln has been selected for use as a roaster. However, it is not used in a conventional manner. Modifications are required to work with a bulk product and alter how hot gases are contacted with the briquettes to effect forced convection. Development work is recommended to take place with the kiln vendor to engineer the modifications, test them at scale and verify that the kiln will perform as expected. Delays in this development work can be expected to adversely impact the project schedule. Costs for this development work have been included in the capital cost estimate and Bacanora have already starting to engage with the vendor(s) as appropriate.
- Additional testwork is required to quantify the breakthrough and determine the loading capacity of the activated alumina. The current FS design is considered conservative and is based on the number of columns in the Demonstration Plant and evidence from vendor supplied information and testwork regarding the activated alumina loading and number of bed volumes to achieve saturation. This additional testwork is testwork is required to facilitate an economic detailed design to validate the activated alumina inventory required. When undertaking the activated alumina testwork, F and Si should be checked for residual levels to ensure suitability for the overall process design. The testwok is expected to cost up to $12,000.
- Boron removal using IX (ion exchange). No testwork has been performed to date to confirm the design parameters for the boron removal columns. The column testwork will confirm the ion exchange resin selection and confirm the design which is based on an assumed loading. The testwork should also include gaining an understanding of the influence of potassium on the resin capacity as ANSTO reported that the loading achieved in the Locked Cycle was lower than what they expected based on using the same resin for boron removal in the past. They believe this might be due to the elevated levels of potassium in the liquor. This testwork is required to facilitate economic detailed design. The testwok and design is expected to cost up to $12,000 (testwork only is expected to cost up to $3,000).

- Glaserite Crystallisation and Decomposition. Vendors did glaserite crystallisation testwork and the subsequent decomposition. Based on this testwork they have presented proposals and are confident that the unit operations will work. It is recommended that additional vendor testwork be conducted prior to commercial award of the package as this step is considered to hold significant risk for the project. Delays in this development work can be expected to adversely impact the project schedule. Costs for this development work have been included in the capital cost estimate.

  The additional testwork should be focussed as follows:

  1. Increasing the potassium concentration of the glaserite crystal to as close to a molar ratio of 3:1 (potassium to sodium) as possible. Additonal testwork is required to improve the potassium to sodium ratio to improve potassium sulfate recovery and quality.
  2. Further testwork to better understand the combined decomposition of glaserite and simultaneous precipitation of potassium sulfate out of solution.
  3. Testwork has indicated that the glaserite dissolved completely within 1 hour. It is critical to understand how rapidly the glaserite dissolves to prevent undissolved glaserite in the potassium sulphate product.

- Scaling of the precipitation vessels should be further investigatd to minimise potential operational challenges. Significant scaling was observed in the LCT testwork at ANSTO whereas it is reported that at RB Energy scaling was not an issue. It is suggested that there is a workshop involving Bacanora, the appointed EPCM engineer and the crystalliser vendor/s specifically to address this issue and ensure a robust design is implemented with all possible learnings and knowledge incorporated.

- Bicarbonation dissolution. The dissolution in the current design is at atmospheric pressure and the lithium carbonate dissolves within 2 hours. It is reported from the Demonstration Plant that dissolution is far slower. It is not known if this is due to the way carbon dioxide is sparged into the tank but data needs to be gathered and the current design refined, if required, as this may potentially be an issue with scale-up of the processes and equipment from laboratory to demonstration plant to full size process plant.

- For the FS, the crystalliser package consisted of the Glauber Salt plant, anhydrous sodium sulfate crystalliser, PLS Evaporator and glaserite crystalliser. These items are all long lead delivery and Bacanora has indicated the intention to place order/s for these items at the start of 2Q2018 to achieve their proposed execution and delivery schedule. In order to be in a position to place order/s additional work is required:

  - Share the Locked Cycle Testwork data with the crystalliser vendor/s so they can better appreciate the technology. Bacanora have already started to share relevant testwork data and engage with the vendor(s) to advance this work.
  - The (Ausenco) SysCAD model should be updated with the revised operating conditions and design information from the vendor as it becomes available. This will help ensure an up-to-date SysCAD model is maintained and also serve to revalidate the process design and maintain confidence in the robustness and expected success of the flowsheet design.

  An allowance for this development work has been included in the capital cost estimate.

- It is strongly recommended that the ability to distribute and harvest briquettes for, and from, solar drying is physically demonstrated prior to the commencement of detailed design. This process involves equipment that is readily available and should thus be comparatively easy to demonstrate, so long as the Bacanora Demonstration Plant can provide enough briquettes for physical testing purposes

26.4 Infrastructure

The following infrastructure related work should be completed:

- Reconfirm electrical loads prior to finalisation of BOO contracts for the gas pipeline and power station.
- Continued development of market source LNG supply is recommended in the event natural gas pipeline delivery is delayed due to perceived finance risk, environmental permitting, easement negotiation, or construction (weather - material - labor). Proposals for medium and long-term dedicated LNG supply have been received but these require take or pay contracts, which may not prove advantageous depending upon the anticipated delay. Final evaluation will be required at the time orders for long lead item are placed, not later than Q3 2018.
- Further detailed geotechnical investigation is recommended for the final tailings storage facility (TSF) design to confirm proposed placement. Proposed placement has changed since the initial geotec work was performed for the PFS allowing the waste rock storage and tailings storage to be comingled a consolidated facility. While data is available from previous work in the surrounding area, additional testing is required at the TSF toe.

26.5 Environment

The following environment related work should be completed:

- A site-wide project water balance model is recommended for the project to help understand the management of water during construction and operations (a process water balance model currently exists). The model can quantify the total water needs for the entire project at any given point in time. This model would also estimate discharge quantities of water into the natural streams for permitting requirements.
- Bacanora Minerals should consider developing programs oriented to promoting economic autonomy in its social management plan through strengthening job capabilities and boost productive projects that do not depend on mining, in order to avoid negative effects at the end of the project. It is important that these programs are designed in conjunction with the community to include their expectations and potential insights.
- Test wells should be constructed and tested at other identified borefields to ensure project demands are meet for Stage 2. It is recommended a backup source of water be identified in an adjacent aquifer and tested.

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APPENDIX A

SUMMARY OF MAJOR LITHIUM AND POTASSIUM INTERCEPTS

Drillhole ID	Domain/Unit	From (m)	To (m)	Li (ppm)	K (%)
ES-01	Lower Clay	156.06	193.55	3966	1.7
	Upper Clay (High Grade)	135.33	143.41	4043	1.4
	Upper Clay (Low Grade)	116.13	135.33	950	0.5
ES-02	Lower Clay	203.55	244.45	3079	1.5
	Upper Clay (High Grade)	193.55	197.39	2984	1.2
	Upper Clay (Low Grade)	190.41	193.55	278	0.3
ES-03	Lower Clay	210.92	239.57	3901	1.6
	Upper Clay (High Grade)	183.34	199.85	2721	1.0
	Upper Clay (Low Grade)	158.5	183.34	899	0.3
ES-04	Lower Clay	140.42	171.75	3595	1.4
	Upper Clay (High Grade)	120.7	132.47	2336	0.9
	Upper Clay (Low Grade)	96.44	120.7	671	0.4
ES-05	Lower Clay	59.83	93.57	2948	1.2
	Upper Clay (High Grade)	47.55	54.56	2107	0.9
	Upper Clay (Low Grade)	23.16	47.55	558	0.3
ES-06	Lower Clay	33.48	75.9	1539	0.7
	Upper Clay (Low Grade)	9.75	27.74	708	0.4
ES-07	Lower Clay	36	69.49	808	0.9
	Upper Clay	0	32	842	0.4
ES-08	Lower Clay	49.38	73.76	1551	0.7
	Upper Clay	19.2	45.11	670	0.5
ES-09	Lower Clay	51.97	81.99	1163	0.6
	Upper Clay	14.94	46.79	602	0.5
ES-10	Lower Clay	3.96	28.35	1156	0.6
ES-11	Lower Clay	231.34	257.25	5206	2.2
	Upper Clay (High Grade)	207.47	218.69	3376	1.2
	Upper Clay (Low Grade)	183.74	207.47	1234	0.7
ES-12	Lower Clay	233.66	240.49	4052	2.0
	Upper Clay (High Grade)	211.76	221.77	4312	1.5
	Upper Clay (Low Grade)	188.06	211.76	971	0.5
ES-13	Lower Clay	322.48	349.61	4077	1.6
	Upper Clay (High Grade)	305.1	315.35	4523	1.3
	Upper Clay (Low Grade)	278.16	305.1	1017	0.4
ES-14	Lower Clay	65.53	95.1	4733	1.8
	Upper Clay (High Grade)	41.15	56.69	2549	1.0
	Upper Clay (Low Grade)	13.72	41.15	770	0.4
ES-15	Lower Clay	32.31	66.14	4087	1.6
	Upper Clay (High Grade)	18.59	21.95	1260	0.5
ES-16	Lower Clay	69.37	96.93	3312	1.3
	Upper Clay (High Grade)	52.65	62.18	1198	0.6
	Upper Clay (Low Grade)	34.23	52.65	584	0.3
ES-17	Lower Clay	190.07	221.59	4701	1.8
	Upper Clay (High Grade)	166.88	179.53	3585	1.2
	Upper Clay (Low Grade)	141.67	166.88	816	0.4
ES-18	Lower Clay	43.1	73.15	1720	0.8
	Upper Clay (High Grade)	31.7	38.71	2175	0.8
	Upper Clay (Low Grade)	13.41	31.7	637	0.3
ES-19	Lower Clay	129.33	157.58	2308	1.0
	Upper Clay (High Grade)	117.5	124.97	2314	0.8
	Upper Clay (Low Grade)	93.88	117.5	530	0.4
ES-20	Lower Clay	12.07	41.76	1521	0.8
	Upper Clay (Low Grade)	0	8.84	1428	0.6
ES-21	Upper Clay (Low Grade)	14.33	26.21	464	0.4
ES-22	Lower Clay	153.59	158.62	41	0.2
	Upper Clay	130.45	152	167	0.4
ES-23	Lower Clay	29.29	34.75	121	0.3
	Upper Clay	13.38	27.1	513	0.3
ES-24	Lower Clay	66.39	92.71	1593	0.8
	Upper Clay	48.46	61.14	820	0.5
ES-25	Lower Clay	168.37	177.39	555	0.5
	Upper Clay	156.67	168.35	157	0.4
ES-26	Lower Clay	48.23	66.14	745	0.4
	Upper Clay	16.43	44.81	482	0.4
ES-27	Lower Clay	24.38	49.48	1225	0.6
	Upper Clay	7.62	18.17	477	0.4
ES-28	Lower Clay	22.86	32.31	86	0.3
	Upper Clay	0	18.59	327	0.4
ES-29	Lower Clay	24.9	29.87	64	0.3

	Upper Clay	11.28	20.12	249	0.2
ES-30	Upper Clay	28.35	39.93	150	0.2
ES-31	Lower Clay	69.49	104.85	4864	1.9
	Upper Clay (High Grade)	43.89	59.13	3623	1.3
	Upper Clay (Low Grade)	15.12	43.89	760	0.4
ES-32	Lower Clay	32	35.36	1739	1.8
ES-33	Lower Clay	147.83	150.57	795	0.4
	Upper Clay	121.13	144.78	552	0.4
ES-35	Lower Clay	106.68	129.03	1446	0.6
	Upper Clay	78.33	100.89	808	0.4
ES-36	Lower Clay	23.26	44.68	1009	0.5
ES-37	Lower Clay	0	23.35	1668	0.7
ES-38	Upper Clay	109.42	141.12	937	0.6
ES-39	Lower Clay	40.23	44.81	10	0.2
	Upper Clay	35.6	40.23	129	0.3
ES-41	Lower Clay	70.1	95.83	774	0.5
	Upper Clay	34.14	64.31	529	0.4
ES-42	Lower Clay	39.32	64.6	4241	1.7
	Upper Clay (High Grade)	16.15	23.35	3069	1.1
ES-44	Lower Clay	118.11	133.2	5034	2.0
	Upper Clay (High Grade)	93.88	105.31	3575	1.3
	Upper Clay (Low Grade)	74.68	93.88	1252	0.7
ES-45	Lower Clay	125.73	140.51	4503	1.8
ES-46	Lower Clay	162.46	178.92	4604	1.8
	Upper Clay (High Grade)	147.22	154.38	3371	1.4
	Upper Clay (Low Grade)	133.2	147.22	1350	0.7
ES-47	Lower Clay	124.66	150.11	5146	2.1
	Upper Clay (High Grade)	105.77	111.56	1483	0.5
	Upper Clay (Low Grade)	94.79	105.77	1185	0.6
ES-48	Lower Clay	215.65	244.45	4523	1.9
	Upper Clay (High Grade)	195.38	203.25	3698	1.2
	Upper Clay (Low Grade)	182.58	195.38	1173	0.6
ES-50	Lower Clay	240.18	254.81	4916	2.1
	Upper Clay (High Grade)	218.39	228.6	3651	1.2
	Upper Clay (Low Grade)	193.85	218.39	863	0.5
ES-51	Lower Clay	238.66	267.3096	4400	1.7
	Upper Clay (High Grade)	218.39	230.124	2860	1.1
	Upper Clay (Low Grade)	197.0532	218.39	942	0.5
ES-52	Lower Clay	275.844	302.51	4572	1.7
	Upper Clay (High Grade)	263.0424	269.5956	3239	1.0
ES-53	Lower Clay	345.95	381.91	4844	1.9
	Upper Clay (High Grade)	318.8208	330.1	3362	1.1
	Upper Clay (Low Grade)	286.59	318.82	773	0.3
ES-54	Lower Clay	288.8	326.44	3802	1.7
	Upper Clay (High Grade)	274.78	280.87	804	0.4
ES-55	Lower Clay	236.68	243.6876	2639	1.2
	Upper Clay (High Grade)	221.1324	230.886	1026	0.6
	Upper Clay (Low Grade)	204.83	221.13	518	0.3
ES-56	Lower Clay	217.93	253.29	3140	1.3
	Upper Clay (High Grade)	197.21	209.4	2486	0.9
	Upper Clay (Low Grade)	179.53	197.21	669	0.4
ES-57	Lower Clay	251.03	284.07	2770	1.2
	Upper Clay (High Grade)	231.65	243.5352	1818	0.8
	Upper Clay (Low Grade)	206.96	231.648	522	0.4
ES-58	Lower Clay	195.38	227.99	2482	1.0
	Upper Clay (High Grade)	183.49	191.72	1727	0.6
	Upper Clay (Low Grade)	161.85	183.49	278	0.3
LV-01	Upper Clay (High Grade)	24.54	35.36	3508	1.1
	Upper Clay (Low Grade)	7.32	24.54	1658	0.8
LV-02	Upper Clay (High Grade)	98.45	108.51	2882	1.0
	Upper Clay (Low Grade)	78.94	98.45	1269	0.7
LV-03	Upper Clay (Low Grade)	126.49	141.73	921	0.5
LV-04	Lower Clay	126.49	150.88	4949	2.0
	Upper Clay (High Grade)	96.62	110.57	3059	1.0
	Upper Clay (Low Grade)	91.44	96.62	1221	0.6
LV-05	Lower Clay	60.35	80.47	4028	1.7
	Upper Clay (High Grade)	36.58	46.63	3234	1.0
	Upper Clay (Low Grade)	7.92	36.58	1102	0.6
LV-06	Lower Clay	46.18	67.97	3574	1.6

	Upper Clay (High Grade)	15.85	30.78	3161	1.1
	Upper Clay (Low Grade)	2.44	15.85	666	0.4
LV-08	Lower Clay	98.45	118.26	2623	1.1
	Upper Clay (Low Grade)	67.89	94.18	870	0.5
LV-09	Lower Clay	77.42	95.2	1329	0.7
	Upper Clay (Low Grade)	38.79	52.43	765	0.3
LV-10	Upper Clay (Low Grade)	55.17	118.26	689	0.5
LV-11	Upper Clay (Low Grade)	5.18	74.98	196	0.2
LV-12	Lower Clay	118.41	129.24	107	0.3
	Upper Clay (Low Grade)	71.32	98.6	103	0.2
LV-13	Lower Clay	13.26	34.59	5434	2.1
LV-14	Lower Clay	14.17	32	5809	2.4
LV-15	Lower Clay	18.29	42.11	3739	1.7
LV-16	Lower Clay	17.68	42.52	2844	1.4
LV-17	Lower Clay	23.16	41.76	1555	0.9
LV-18	Lower Clay	260.3	279.5	1143	0.8
	Upper Clay (Low Grade)	218.24	245.67	577	0.3
LV-19	Upper Clay (Low Grade)	11.89	48.77	1033	0.5
LV-20	Lower Clay	268.41	291.39	1622	0.9
	Upper Clay (Low Grade)	219.52	247.19	653	0.4
LV-21	Lower Clay	72.24	92.96	1759	1.0
	Upper Clay (Low Grade)	8.93	59.74	1194	0.6
LV-22	Lower Clay	75.86	96.35	2988	1.5
	Upper Clay (High Grade)	44.5	60.35	2457	1.0
	Upper Clay (Low Grade)	18.38	44.5	755	0.4
LV-23	Lower Clay	69.68	87.48	3547	1.6
	Upper Clay (High Grade)	38.56	56.69	2778	1.0
	Upper Clay (Low Grade)	15.97	38.56	722	0.6
LV-24	Lower Clay	145.27	158.88	4124	1.7
	Upper Clay (High Grade)	116.43	130.06	2771	0.9
	Upper Clay (Low Grade)	90.53	116.43	1012	0.5
LV-25	Upper Clay (High Grade)	143.66	155.75	2744	1.4
	Upper Clay (Low Grade)	127.71	143.66	695	0.3
LV-26	Lower Clay	53.95	76.05	2087	0.9
	Upper Clay (High Grade)	42.52	48.77	3233	1.2
	Upper Clay (Low Grade)	22.86	42.52	1042	0.5
LV-27	Lower Clay	78.03	98.33	5855	2.4
	Upper Clay (High Grade)	54.86	66.14	3842	1.4
	Upper Clay (Low Grade)	43.16	54.86	1428	0.8
LV-28	Lower Clay	179.83	203.3	5228	1.9
	Upper Clay (High Grade)	153.62	165.93	4309	1.4
	Upper Clay (Low Grade)	131.73	153.62	1037	0.5
LV-29	Lower Clay	51.82	74.68	5394	2.2
	Upper Clay (High Grade)	24.69	35.66	3297	1.1
	Upper Clay (Low Grade)	8.23	24.69	1609	0.7
LV-31	Lower Clay	203.7	226.04	3092	1.4
	Upper Clay (High Grade)	173.61	185.56	2956	1.1
	Upper Clay (Low Grade)	147.83	173.61	755	0.4
LV-34	Lower Clay	3.05	7.92	516	0.4
LV-35	Lower Clay	12.37	33.41	5786	2.3
LV-36	Lower Clay	15.33	35.36	4372	1.8
LV-37	Lower Clay	14.84	36.88	3942	1.9
LV-38	Lower Clay	13.96	37.49	3157	1.7
LV-39	Lower Clay	4.88	27.31	2188	1.3
LV-40	Lower Clay	148.25	166.70	2,716	1.4
	Upper Clay (High Grade)	123.20	139.50	2,013	1.0
	Upper Clay (Low Grade)	103.60	123.20	409	0.3
LV-41	Lower Clay	148.40	174.10	2,349	1.2
	Upper Clay (High Grade)	119.30	131.50	2,359	1.2
	Upper Clay (Low Grade)	106.00	119.30	738	0.3
LV-42	Lower Clay	68.50	78.40	2,425	1.4
	Upper Clay (High Grade)	50.20	60.40	2,203	1.1
	Upper Clay (Low Grade)	28.20	50.20	432	0.3
LV-43	Lower Clay	129.40	150.00	2,701	1.3
	Upper Clay (High Grade)	95.10	113.90	1,745	0.8
	Upper Clay (Low Grade)	73.93	95.10	981	0.5
LV-44	Lower Clay	144.60	165.85	3,775	2.0
	Upper Clay (High Grade)	112.60	126.80	3,080	1.1
	Upper Clay (Low Grade)	88.30	112.60	765	0.4

LV-45	Lower Clay	14.90	38.55	2,978	1.6
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-46	Lower Clay	13.17	39.30	2,576	1.4
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-47	Lower Clay	14.95	38.04	3,752	1.8
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-48	Lower Clay	13.00	38.10	4,192	1.9
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-49	Lower Clay	29.30	39.40	5,136	2.1
	Upper Clay (High Grade)	1.00	14.25	2,947	1.1
	Upper Clay (Low Grade)	0.00	1.00	470	0.3
LV-50	Lower Clay	62.15	71.00	4,784	2.2
	Upper Clay (High Grade)	26.95	45.20	2,534	1.0
	Upper Clay (Low Grade)	21.32	26.95	790	0.4
LV-51	Lower Clay	13.10	35.05	3,602	1.8
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-52	Lower Clay	12.00	31.15	3,545	1.7
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-53	Lower Clay	183.25	206.60	3,482	1.6
	Upper Clay (High Grade)	153.90	166.90	3,317	1.1
	Upper Clay (Low Grade)	132.45	153.90	1,113	0.6
LV-54	Lower Clay	165.55	185.40	5,016	2.1
	Upper Clay (High Grade)	137.20	148.90	4,012	1.4
	Upper Clay (Low Grade)	120.90	137.20	1,573	0.9
LV-55	Lower Clay	98.40	122.40	5,474	2.1
	Upper Clay (High Grade)	71.55	82.30	3,840	1.2
	Upper Clay (Low Grade)	52.00	71.55	1,280	0.6
LV-56	Lower Clay	74.00	97.10	5,183	2.2
	Upper Clay (High Grade)	45.50	59.45	3,379	1.3
	Upper Clay (Low Grade)	29.85	45.50	1,734	0.8
LV-57	Lower Clay	15.80	41.65	4,626	1.9
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-58	Lower Clay	74.40	95.85	6,283	2.5
	Upper Clay (High Grade)	55.60	62.90	3,954	1.3
	Upper Clay (Low Grade)	46.30	55.60	895	0.6
LV-59	Lower Clay	84.05	100.00	4,740	2.1
	Upper Clay (High Grade)	61.80	69.10	4,393	1.6
	Upper Clay (Low Grade)	46.25	61.80	1,324	0.7
LV-60	Lower Clay	183.85	210.10	5,049	2.1
	Upper Clay (High Grade)	169.00	171.95	3,688	1.5
	Upper Clay (Low Grade)	163.05	169.00	530	0.4
LV-61	Lower Clay	166.00	186.75	4,649	1.8
	Upper Clay (High Grade)	142.35	153.90	3,957	1.3
	Upper Clay (Low Grade)	119.00	142.35	1,281	0.7
LV-62	Lower Clay	110.45	115.45	3,857	1.9
	Upper Clay (High Grade)	91.60	99.70	2,072	0.7
	Upper Clay (Low Grade)	90.00	91.60	421	0.1
LV-63	Lower Clay	8.00	28.75	3,749	1.5
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-64	Lower Clay	125.00	155.70	5,315	2.1
	Upper Clay (High Grade)	95.00	105.35	4,293	1.5
	Upper Clay (Low Grade)	85.30	95.00	980	0.5
LV-65	Lower Clay	147.90	157.80	3,521	1.5
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				
LV-66	Lower Clay	171.15	199.30	4,902	2.0
	Upper Clay (High Grade)	127.30	155.75	2,608	0.9
	Upper Clay (Low Grade)	107.00	127.30	1,159	0.7
LV-67	Lower Clay	169.25	195.00	4,323	1.8
	Upper Clay (High Grade)	148.45	157.30	3,098	1.1
	Upper Clay (Low Grade)	143.80	148.45	332	0.3

LV-68	Lower Clay	196.60	217.10	4,973	1.9
	Upper Clay (High Grade)	174.00	183.50	3,494	1.3
	Upper Clay (Low Grade)	154.95	174.00	1,313	0.9
LV-69	Lower Clay	219.20	241.35	3,970	1.7
	Upper Clay (High Grade)	190.05	202.00	2,967	1.0
	Upper Clay (Low Grade)	170.40	190.05	1,086	0.6
LV-70	Lower Clay	15.70	36.75	4,361	1.9
	Upper Clay (High Grade)				
	Upper Clay (Low Grade)				

APPENDIX B

ANALYTICAL SOLUTIONS REPORT

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To: Martin Pittuck and Ben Lepley, SRK

CC:

From: Lynda Bloom

Date: November 4, 2016

Re: Bacanora Lithium Reference Materials

SRK's client, Bacanora, has a lithium project in Mexico and prepared in-house reference materials (RMs). This memo documents the certification process for these materials and recommends accepted values with tolerances for Li, K, Ca, Mg, and Sr.

Source Materials

Three different RMs were manufactured from bulk samples collected from the Sonora Lithium Project. The mineralisation primarily occurs as polylithionite and hectorite.

Bulk samples were collected from the pilot plant. The pilot plant processed material from a trench to test a pre-concentration (cleaning) process. Two products were obtained; a pre-concentrate and a reject.

- Standard SPRET-4 (~0.5% Li) was collected from a pulverized pre-concentrate at -75 microns that is produced at the plant and that is used for calcination. 15.9 kg was collected.
- Standard SREJT-4 (~0.1% Li) was collected from the rejects and then bulk milled/pulverized to -75 microns. 7.8 kg was collected.
- Standard SLUV-1 (~0.6% Li) was collected from the LUV standard (prepared as above previously). The 25g sachets were opened and all of the samples were mixed and followed the same homogenization subsample procedure. 13.6 kg was collected.

Comminution and Homogenisation Procedures

Bulk samples of 8 to 16 kg were homogenized in a single batch in a drum mixer for 24 hours at the third-party Sonora Sample Preparation Lab (ISO certified). Sub-samples of 25 grams were then sealed in HDPE sachets and submitted to 4 commercial laboratories and the company's Mexican laboratory.

Each laboratory received 5 packets of each RM for analysis.

Collaborative Study

Laboratories and Analytical Methods

Routine samples are submitted for sample preparation at ALS in Hermosillo, Mexico generating 250 gram pulps. Analysis for lithium has been performed at ALS, North Vancouver using an aqua regia digest (method code ME-ICP41). Lithium is reported with a 10 ppm detection limit and to a maximum of 1%.
All of the project samples and the results used for resource estimation have been reported using the same aqua regia digest. All laboratories participating in the collaborative study (also known as a round robin) were asked to analyze samples by aqua regia digest and ICP analysis.

The laboratories that received the RMs and the method codes for the analyses are summarized in Table 1.

Table 1: Summary of Laboratories and Analytical Method Codes

Laboratory	Method Code	Job Number
ALS (Vancouver)	ME-MS41 (Aqua regia)	HE16072252
ALS (Vancouver)	ME-MS81 (Metaborate fusion)	HE16093353
Bureau Veritas (Vancouver)	AQ270/AQ250-X (Aqua regia)	HMS16000200
Bacanora on-site lab (Hermosillo)	AQR (Aqua regia)	05-19-16 Explo
Skyline (Tuscon)	TE-3 (Aqua regia)	BZB001
Skyline (Tuscon)	TE-5 (4 acid digest)	BZB001 (revised)
SGS (Lakefield)	GE_ICM 14B	LK1600566

All of the Cs concentrations are greater than 500 ppm and exceed the upper limit for the ME-MS41 method. ALS reported Cs, Rb and Sr on a Li-metaborate fusion and not the aqua regia digest.

There were several apparent errors in the original Skyline report; for example lithium reported 3 orders of magnitude lower than the other laboratories. When asked to investigate, Skyline re-assayed the samples with a 4-acid digest and reported values within the expected range. No explanation was provided for the drastic change in results.

The data review identified biases between laboratories for the elements of interest. The differences are mostly likely a result of procedural differences in the aqua regia digestion. The effectiveness of the aqua regia digest can be impacted by temperature, time, acid strength and final volume. Both SGS and ALS provided more details for the aqua regia methods which are summarized in Table 2.

Table 2: Comparison of Aqua Regia Digest Procedures at SGS and ALS

	SGS	ALS
Sample Wt.	0.2 g	0.5 g
Final volume	20 ml	12.5 ml
HCl:HNO3	3:1	3:1
AR	4 ml	5 ml
%HCl	15	30

SGS generally reported higher values than ALS for most elements. Although the acid strength at ALS is greater than at SGS, the final solution is much more dilute at SGS. ALS quality control experts suggested that given the high Ca content of the samples that there could be re-precipitation that would cause values to be lower than reported by SGS.

ALS was not informed of the high lithium content of the samples and therefore did not include RMs of sufficient Li concentration to adequately monitor its' performance.

ALS repeated analysis of the collaborative study samples using the routine aqua regia digestion and 4-acid digest with results received at the end of October. All digests were analyzed by ICP. The results are discussed in the section "Control Charts". The results confirmed the original round robin data and in some cases lower values were reported. There is greater variability (RSD) for some elements, such as Li, in the repeat analyses. Both the original and repeat ALS analyses are included in the calculation of expected values.

The analytical methods used by the Bacaora on-site laboratory were not standardized in June when the test results were reported. Due to the variability in the results, it was decided not to include the results in determination of accepted values.

Within-Lab RSDs Test for Homogeneity

All laboratories reported a range of values for the elements of interest. The range of values, or uncertainty, is a product of both the homogeneity of the RM and the inherent error of the analytical method.

The Relative Standard Deviation was calculated for results reported from each laboratory individually excluding the repeat ALS data (Table 3).

Table 3: Relative Standard Deviation

	Ca RSD	K RSD	Li RSD	Mg RSD	Sr RSD
			SLUV		
ALS RSD	2%	2%	2%	2%	1%
BV RSD	1%	0.3%	1%	1%	3%
SGS RSD	1%	3%	4%	2%	1%
			SRJET-4		
ALS RSD	2%	2%	2%	2%	2%
BV RSD	1%	1%	2%	1.5%	1%
SGS RSD	2%	2%	2%	2%	2%
			SPRET-4		
ALS RSD	2%	2%	2%	2%	2%
BV RSD	1%	1%	2%	1.5%	1%
SGS RSD	2%	2%	2%	2%	2%

The relative standard deviation was calculated from

RSD =S*100/x

Where,
Mean = X/N
X - Summation of x value
N = The count of values
S = Standard Deviation value
x = Mean of the data.

The RSD was calculated separately for each laboratory, element and RM.

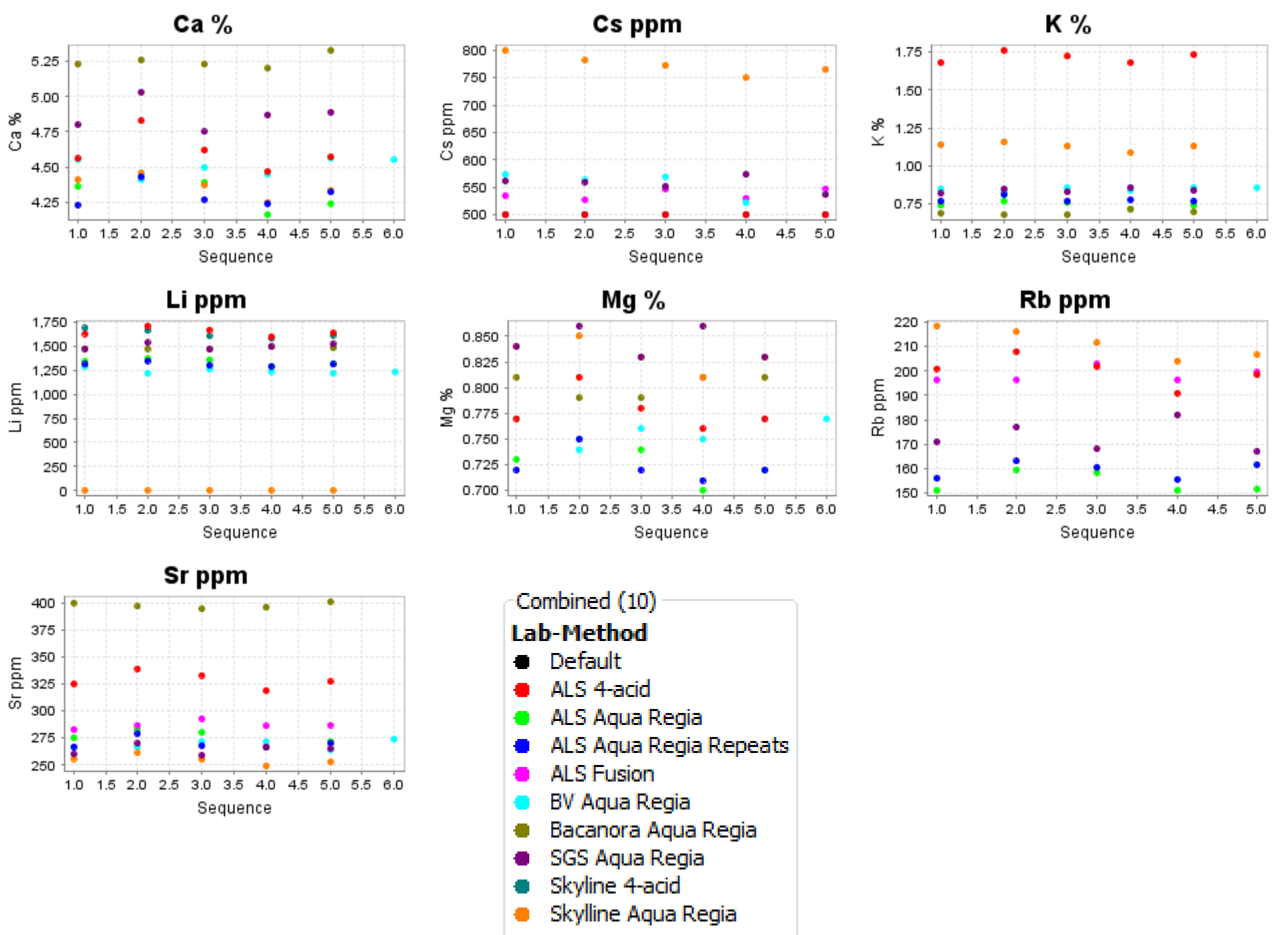
The RSDs range from 0.3 to 4% but are generally 1 to 2 %. Most quality control (QC) programs apply acceptance ranges based on ± 3 standard deviation. For the Bacanora in-house RMs this would translate to acceptance ranges up to ± 6%.

The Bacanora in-house RMs are suitably homogeneous for the purpose of monitoring laboratory performance for the elements in Table 3.

Control Charts

All results from the collaborative study were plotted on control charts. Figure 1 is a graph showing an overview of all results for SREJT-4. Similar graphs for SPRET-4 and SLUV-1 are provided in Appendix 1.

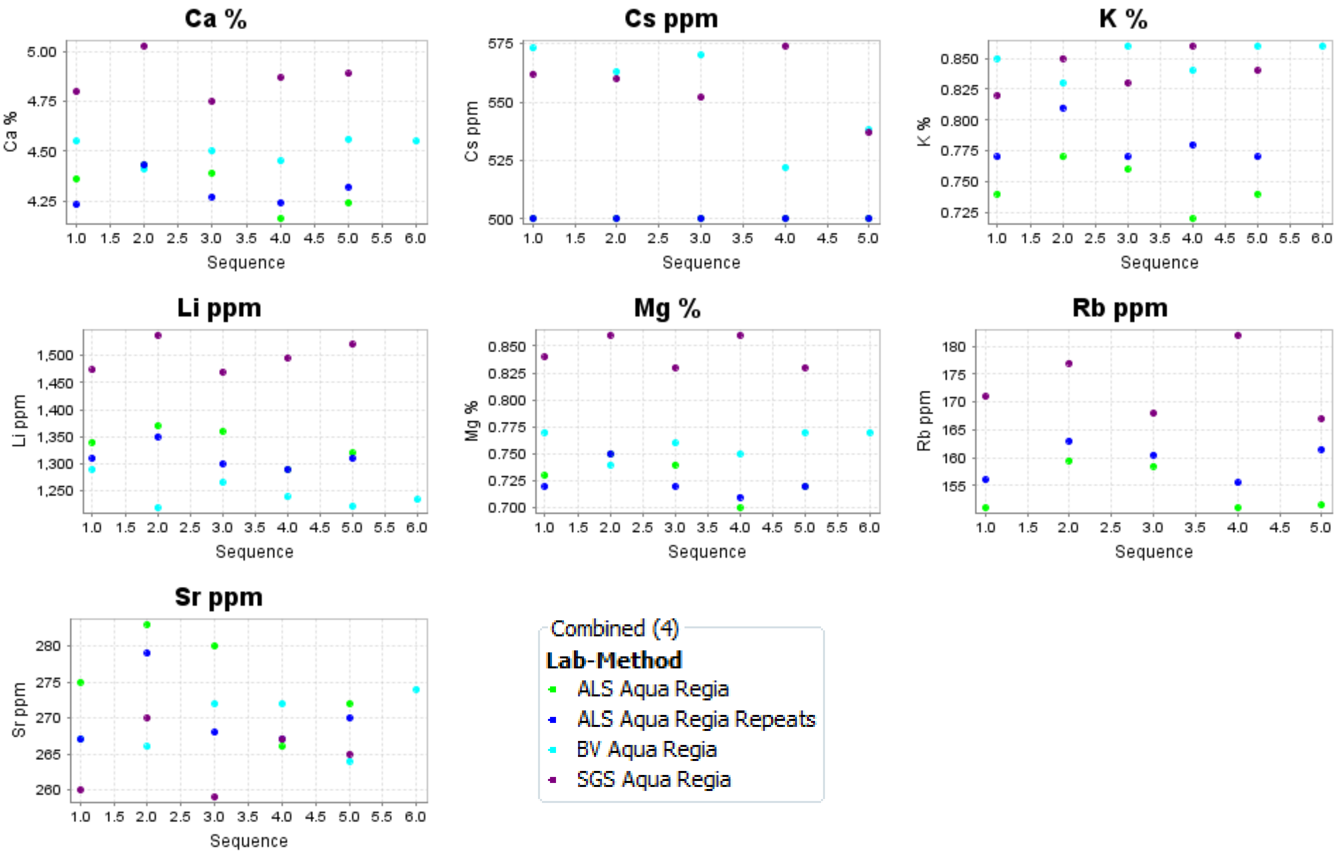
Figure 1: Summary of All Reported Round Robin Data for SREJT-4



It was determined that the on-site Bacanora laboratory and Skyline results should not be included in calculation of expected values for the in-house RMs. ALS data for the 4-acid digest were provided for information only.

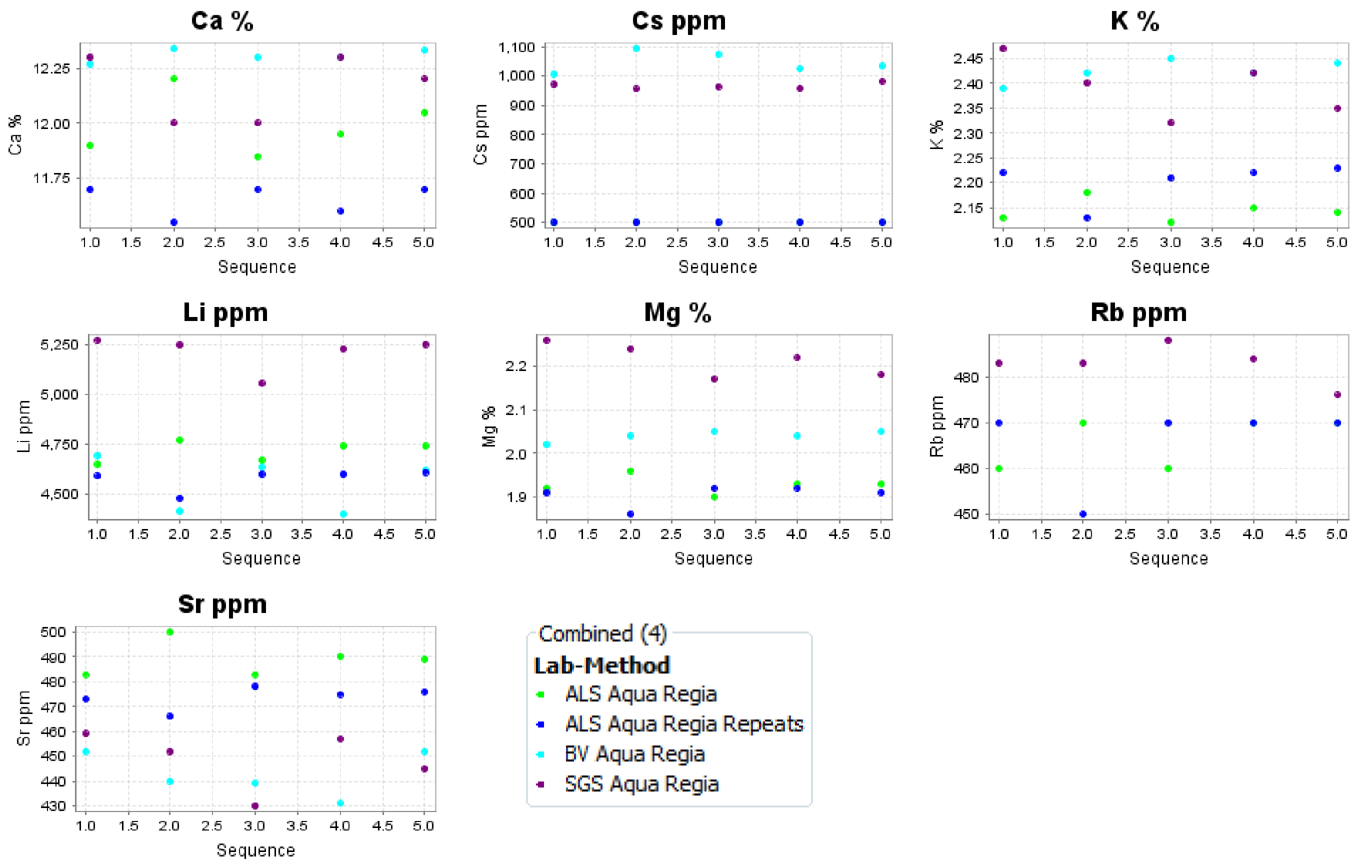
Control charts for the remaining data sets are shown in Figure 2a,b and c.

Figure 2a: Control Charts for SREJT-4



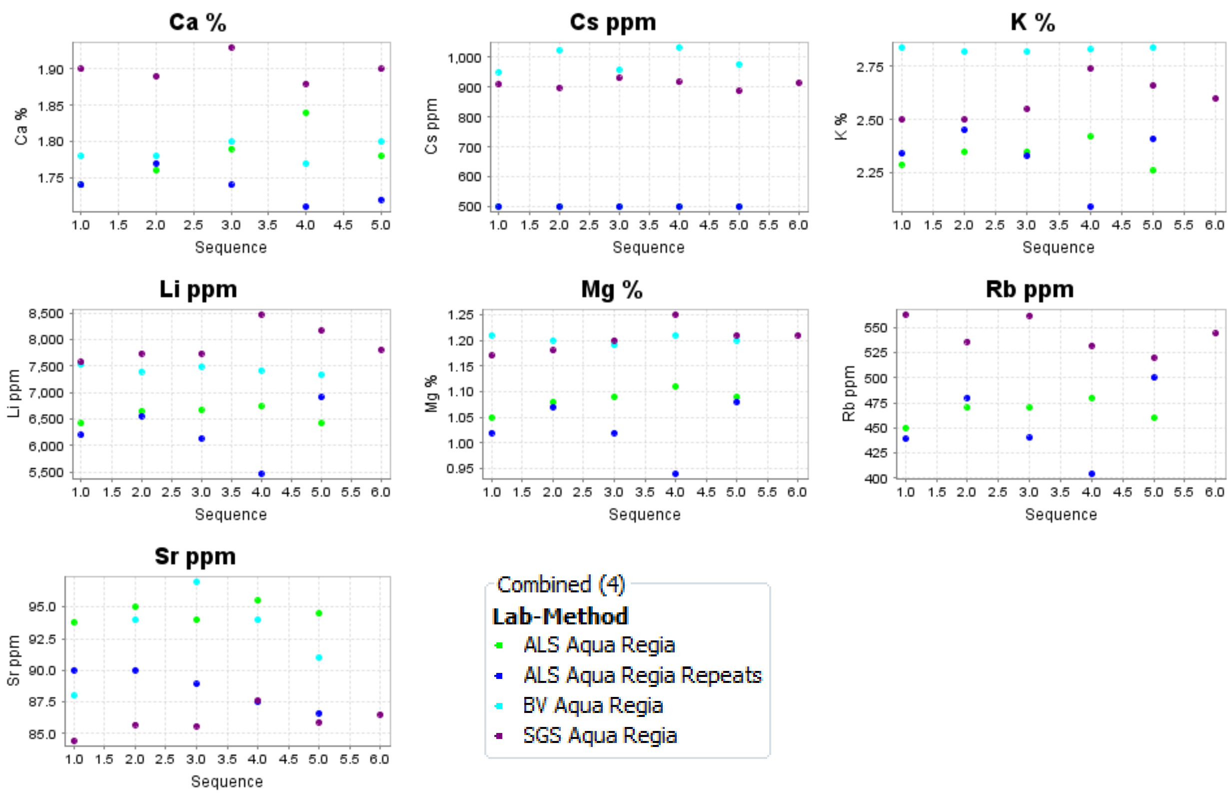
**ALS reports Cs > 500 ppm*

Figure 2b: Control Charts for SPRET-4



**ALS reports Cs > 500 ppm*

Figure 2c: Control Charts for SLUV-1



**ALS reports Cs > 500 ppm*

ALS generally reported lower values than BV or SGS for the elements of interest by aqua regia digest; SGS most commonly reported the highest values. As an example, the within-laboratory means are compared against the overall mean for ALS, BV and SGS for RM SLUV in Table 4. In some cases the highest and lowest means differ by almost 20%.

Table 4: Comparison of Means for Collaborative Study

	Mean per Lab Relative to Overall Mean				
Mean	**Ca**	**K**	**Li**	**Mg**	**Sr**
SLUV-ALS	-2%	-10%	-10%	-10%	2%
SLUV-BV	-2%	9%	1%	9%	0%
SLUV-SGS	4%	0%	8%	0%	-7%

ALS repeated all of the round-robin samples with both an aqua regia digest and 4-acid digest. The mean and standard deviations for Li data are presented in Table 5.

Table 5: Repeat ALS Lithium Analyses

	ALS Repeat Aqua Regia			ALS 4-acid			% Diff AR vs 4-acid
	Mean	**Std. Dev.**	**RSD %**	**Mean**	**Std. Dev.**	**RSD %**	
SRJET-4	1312	22	2%	1650	41	2%	26%
SPRET-4	4590	54	1%	5212	78	1%	14%
SLUV-1	6252	541	9%	8326	69	1%	33%

The mean and standard deviations are similar to those for the first round of analyses. An exception is the range of results for SLUV-1 from 5460 to 6910 ppm Li. This is an unacceptable range of results however it is difficult to determine if this is an issue with the homogeneity of the material or analytical results. Given that this is the only example of an unacceptable range of values, it is assumed that the issue is analytical.

It is also evident that the 4-acid digest reports considerably higher Li concentrations than the aqua regia digest (14 to 33% higher).

Summary Statistics

Based on a review of the control charts, it was determined that all aqua regia digest data from SGS, ALS and BV should be included in calculations of expected values. The summary statistics for all three standards are provided in Tables 5a, b and c. All of the statistics, for individual laboratories are provided in Appendix 2.

Table 5a: Summary Statistics for SREJT-4

SREJT-4							
	Ca %	Cs ppm	K %	Li ppm	Mg %	Rb ppm	Sr ppm
Count Numeric	21	10	21	21	21	15	21
Minimum	4.16	522	0.72	1219	0.7	151	259
Maximum	5.03	574	0.86	1537	0.86	182	283
Mean	4.50	555.1	0.81	1343	0.76	162	270
Standard Deviation	0.24	17.51	0.05	99.38	0.05	9.33	6.14
RPD (Std. Dev. / Mean) %	5%	3%	6%	7%	7%	6%	2%

Table 5b: Summary Statistics for SPRET-4

SPRET-4							
	Ca %	Cs ppm	K %	Li ppm	Mg %	Rb ppm	Sr ppm
Count Numeric	20	10	20	20	20	15	20
Minimum (Reported)	11.55	955	2.12	4398	1.86	450	430
Maximum (Reported)	12.34	1094	2.47	5274	2.26	488	500
Mean	12.03	1006.00	2.29	4763.70	2.02	471.6	463.5
Standard Deviation	0.27	49.13	0.13	285.57	0.13	10.20	20.74
RPD (Std. Dev. / Mean) %	2%	5%	6%	6%	6%	2%	4%

Table 5c: Summary Statistics for SLUV-1

SLUV-1							
47 rows - Univariate	Ca %	Cs ppm	K %	Li ppm	Mg %	Rb ppm	Sr ppm
Count Numeric	20	11	21	21	21	16	21
Minimum	1.71	887	2.09	5460	0.94	404	84.4
Maximum	1.93	1033	2.84	8478	1.25	563	97
Mean	1.80	945.3	2.52	7089	1.13	490.63	90.26
Standard Deviation	0.07	48.8	0.23	755.41	0.08	47.82	3.96
RPD (Std. Dev. / Mean) %	4%	5%	9%	11%	7%	10%	4%

The Relative Standard Deviation (RSD) generally exceeds 5%. The high RSD is because all of the data were used to calculate the standard deviation and there is significant bias between laboratory results. It is recommended that tolerances be applied based on the previous discussion of within-lab RSD. Allowed ranges of ± 8% should be applied when assessing data for quality control failures. Ranges can be increased to ± 10% for Rb and Sr which are reported at ppm levels and for elements that are not critical to project evaluation.

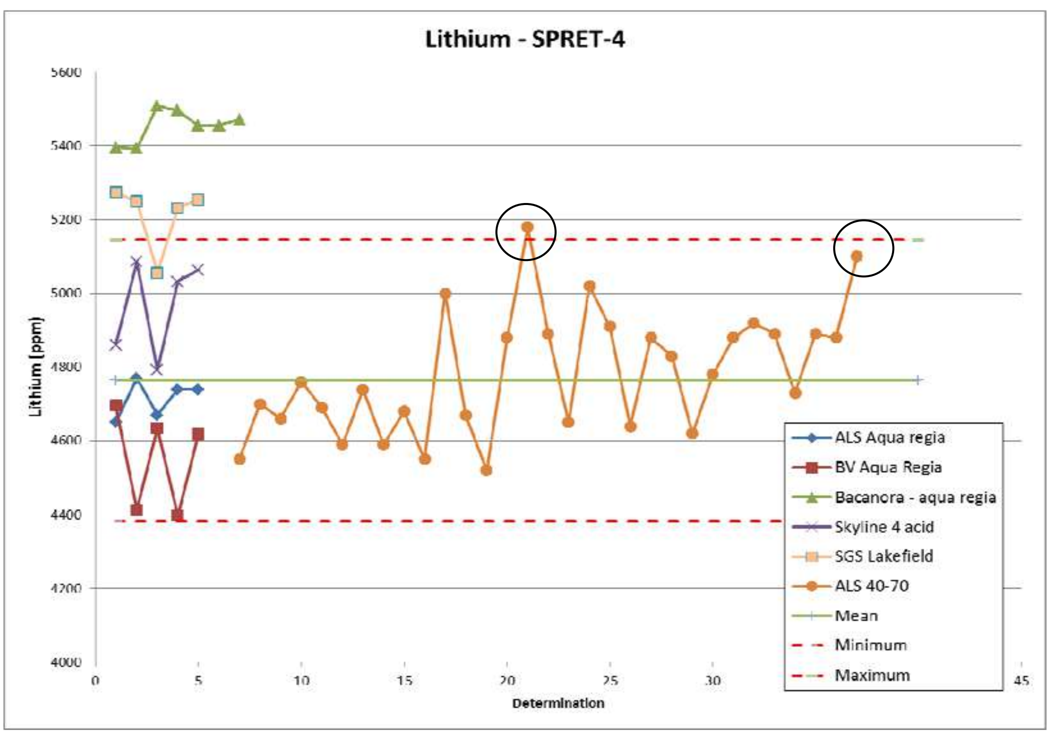
Conclusions

Data from ALS, BV and SGS have been included in calculated expected values for Ca, Cs, K, Li, Mg, Rb and Sr for three standards developed in-house by Bacanora.

The expected values as provided in Table 5 can be used to assess reported results for submitted standards. It is recommended that the accepted tolerances be set at ± 8% from the calculated mean.

As an example of ALS performance in 2016, Li data for SPRET-4 are plotted in Figure 3. This shows that most results are within the expected range. The two data points that are highlighted are considered quality control failures. An initial evaluation suggests that these failures are a result of solution carry-over at the ICP following very high grade samples.

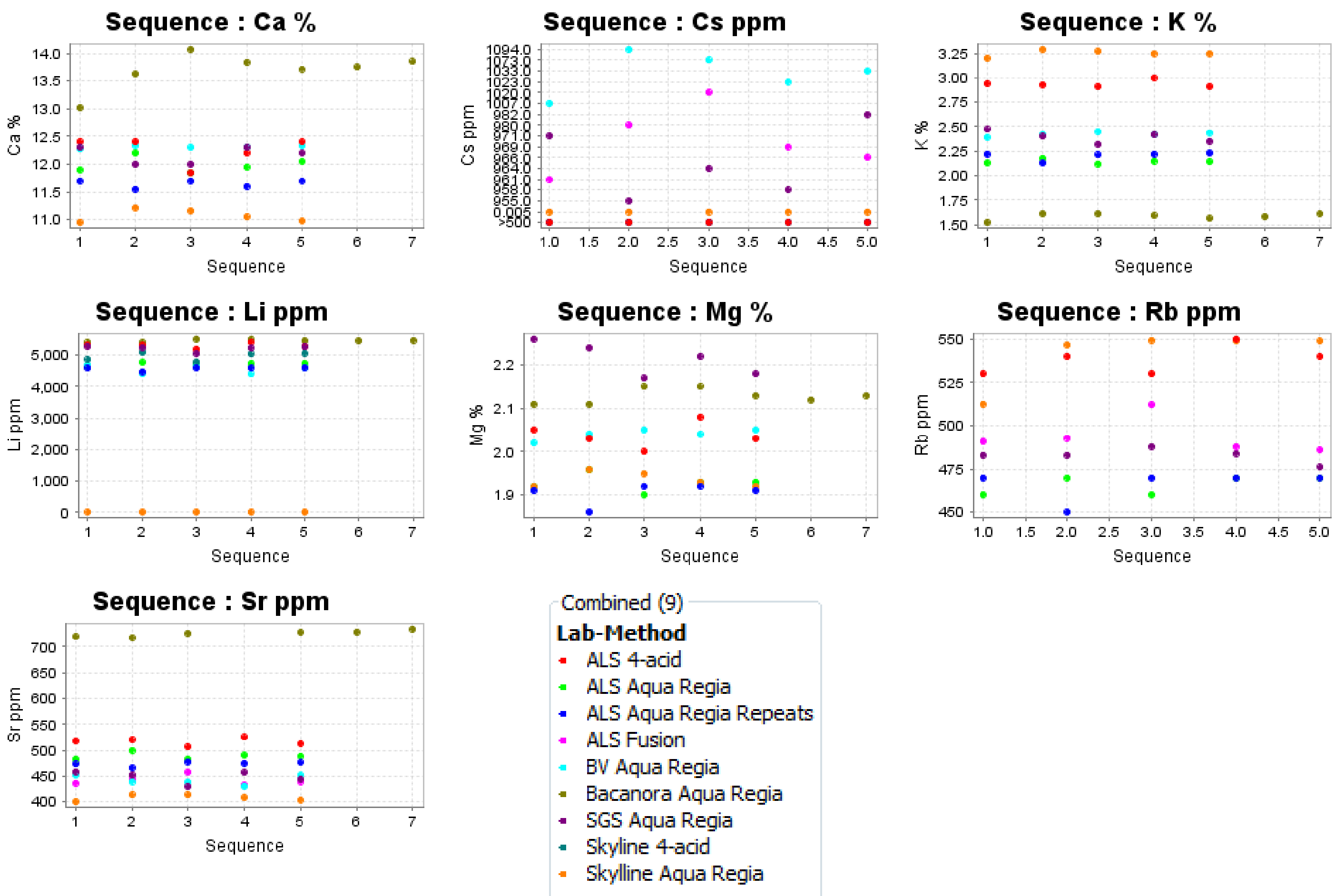
Figure 3: 2016 ALS Results for SPRET-4 with Holes LV40-70



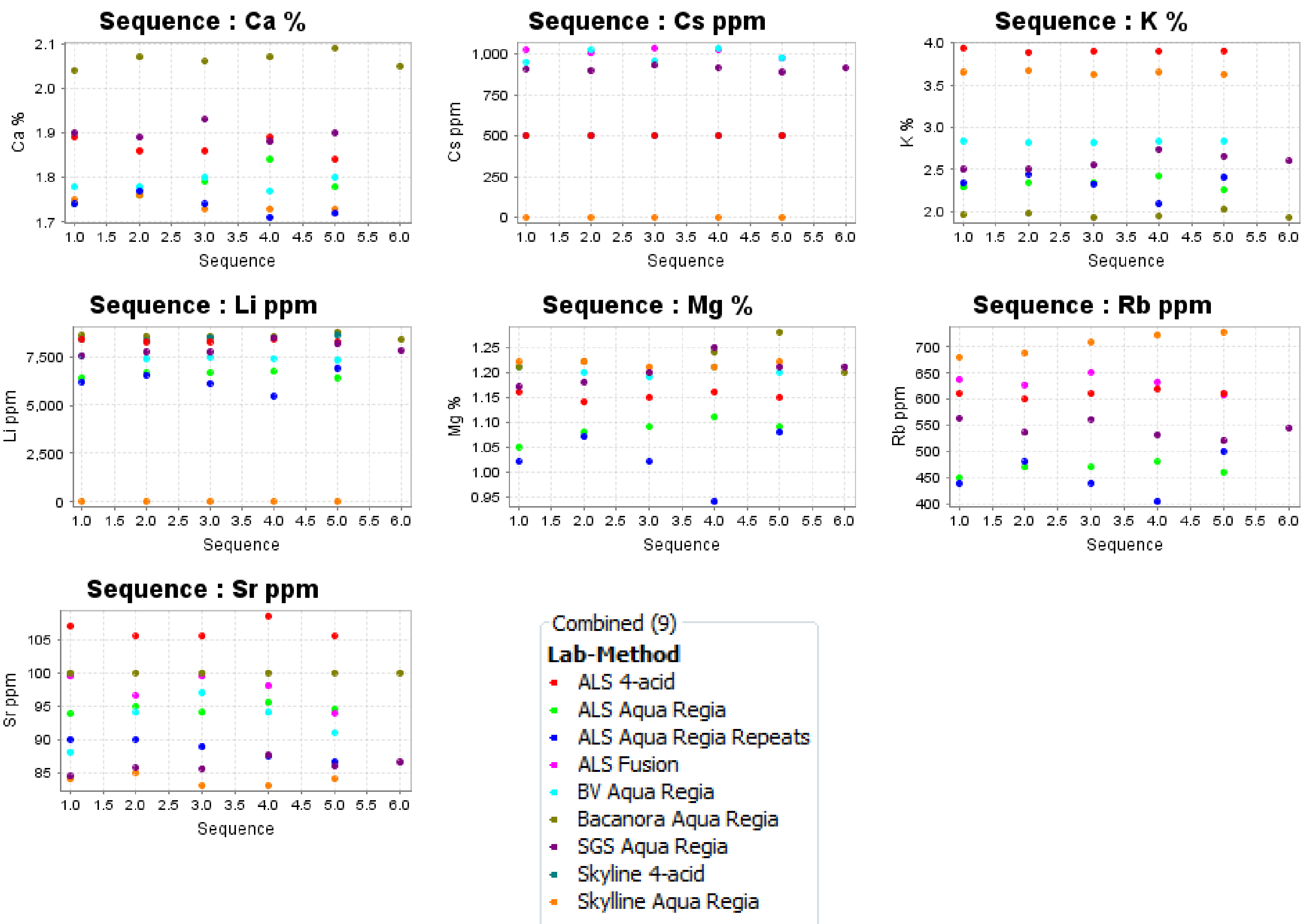
APPENDIX 1

Summary charts with all reported round robin data

SPRET-4



SLUV-1



Sequence : Ca %
Ca %
Sequence
Sequence : Cs ppm
Cs ppm
Sequence
Sequence : K %
K %
Sequence
Sequence : Li ppm
Li ppm
Sequence
Sequence : Mg %
Mg %
Sequence
Sequence : Rb ppm
Rb ppm
Sequence
Sequence : Sr ppm
Sr ppm
Sequence
Combined (9)
Lab-Method
ALS 4-acid
ALS Aqua Regia
ALS Aqua Regia Repeats
ALS Fusion
BV Aqua Regia
Bacanora Aqua Regia
SGS Aqua Regia
Skyline 4-acid
Skylline Aqua Regia

APPENDIX 2

Summary statistics for selected round robin data

SREJT-4							
	Ca %	Cs ppm	K %	Li ppm	Mg %	Rb ppm	Sr ppm
Count Numeric	21	10	21	21	21	15	21
Minimum	4.16	522	0.72	1219	0.7	151	259
Maximum	5.03	574	0.86	1537	0.86	182	283
Mean	4.50	555.1	0.81	1343	0.76	162	270
Standard Deviation	0.24	17.51	0.05	99.38	0.05	9.33	6.14
RPD (Std. Dev. / Mean) %	5%	3%	6%	7%	7%	6%	2%
ALS Aqua Regia : Count Numeric	5	5	5	5	5	5	5
ALS Aqua Regia : Minimum	4.16	>500	0.72	1290	0.7	151	266
ALS Aqua Regia : Maximum	4.43	>500	0.77	1370	0.75	159.5	283
ALS Aqua Regia : Mean	4.316	>500	0.746	1336	0.728	154.3	275.2
ALS Aqua Regia : Standard Deviation	0.11	0.00	0.02	32.09	0.02	4.31	6.69
ALS Aqua Regia Repeats : Count Numeric	5	5	5	5	5	5	5
ALS Aqua Regia Repeats : Minimum	4.23	>500	0.77	1290	0.71	155.5	267
ALS Aqua Regia Repeats : Maximum	4.43	>500	0.81	1350	0.75	163	279
ALS Aqua Regia Repeats : Mean	4.298	>500	0.78	1312	0.724	159.3	270.2
ALS Aqua Regia Repeats : Standard Deviation	0.08	0.00	0.02	22.80	0.02	3.37	5.07
BV Aqua Regia : Count Numeric	6	5	6	6	6	0	6
BV Aqua Regia : Minimum	4.41	522	0.83	1219	0.74		264
BV Aqua Regia : Maximum	4.56	573	0.86	1290	0.77		274
BV Aqua Regia : Mean	4.503333	553.2	0.85	1245	0.76		269.2
BV Aqua Regia : Standard Deviation	0.06	22.22	0.01	27.67	0.01		4.02
SGS Aqua Regia : Count Numeric	5	5	5	5	5	5	5
SGS Aqua Regia : Minimum	4.75	537	0.82	1469	0.83	167	259
SGS Aqua Regia : Maximum	5.03	574	0.86	1537	0.86	182	270
SGS Aqua Regia : Mean	4.868	557	0.84	1498.8	0.844	173	264.2
SGS Aqua Regia : Standard Deviation	0.11	13.67	0.02	29.48	0.02	6.36	4.66

SPRET-4							
	Ca %	Cs ppm	K %	Li ppm	Mg %	Rb ppm	Sr ppm
Count Numeric	20	10	20	20	20	15	20
Minimum (Reported)	11.55	955	2.12	4398	1.86	450	430
Maximum (Reported)	12.34	1094	2.47	5274	2.26	488	500
Mean	12.03	1006.00	2.29	4763.70	2.02	471.6	463.5
Standard Deviation	0.27	49.13	0.13	285.57	0.13	10.20	20.74
RPD (Std. Dev. / Mean) %	2%	5%	6%	6%	6%	2%	4%

ALS Aqua Regia : Count Numeric	5	5	5	5	5	5	5
ALS Aqua Regia : Minimum	11.85	>500	2.12	4650	1.9	460	483
ALS Aqua Regia : Maximum	12.2	>500	2.18	4770	1.96	470	500
ALS Aqua Regia : Mean	11.99	>500	2.144	4714	1.928	466	489
ALS Aqua Regia : Standard Deviation	0.14	0.00	0.02	51.28	0.02	5.48	6.96
ALS Aqua Regia Repeats : Count Numeric	5	5	5	5	5	5	5
ALS Aqua Regia Repeats : Minimum	11.55	500.1	2.13	4480	1.86	450	466
ALS Aqua Regia Repeats : Maximum	11.7	500.1	2.23	4610	1.92	470	478
ALS Aqua Regia Repeats : Mean	11.65	500.1	2.202	4576	1.904	466	473.6
ALS Aqua Regia Repeats : Standard Deviation	0.07	0.00	0.04	54.13	0.03	8.94	4.62
BV Aqua Regia : Count Numeric	5	5	5	5	5	0	5
BV Aqua Regia : Minimum	12.27	1007	2.39	4398	2.02		431
BV Aqua Regia : Maximum	12.34	1094	2.45	4695	2.05		452
BV Aqua Regia : Mean	12.308	1046	2.424	4552	2.04		442.8
BV Aqua Regia : Standard Deviation	0.03	36.24	0.02	136.43	0.01		9.09
SGS Aqua Regia : Count Numeric	5	5	5	5	5	5	5
SGS Aqua Regia : Minimum	12	955	2.32	5056	2.17	476	430
SGS Aqua Regia : Maximum	12.3	982	2.47	5274	2.26	488	459
SGS Aqua Regia : Mean	12.16	966	2.392	5212.8	2.214	482.8	448.6
SGS Aqua Regia : Standard Deviation	0.152	10.840	0.059	88.981	0.038	4.324	11.718

SLUV-1							
47 rows - Univariate	Ca %	Cs ppm	K %	Li ppm	Mg %	Rb ppm	Sr ppm
Count Numeric	20	11	21	21	21	16	21
Minimum	1.71	887	2.09	5460	0.94	404	84.4
Maximum	1.93	1033	2.84	8478	1.25	563	97
Mean	1.80	945.3	2.52	7089	1.13	490.63	90.26
Standard Deviation	0.07	48.8	0.23	755.41	0.08	47.82	3.96
RPD (Std. Dev. / Mean) %	4%	5%	9%	11%	7%	10%	4%
ALS Aqua Regia : Count Numeric	5	5	5	5	5	5	5
ALS Aqua Regia : Minimum	1.74	>500	2.26	6420	1.05	450	93.8
ALS Aqua Regia : Maximum	1.84	>500	2.42	6740	1.11	480	95.5
ALS Aqua Regia : Mean	1.782	>500	2.334	6580	1.084	466	94.56
ALS Aqua Regia : Standard Deviation	0.04	0.00	0.06	149.83	0.02	11.40	0.70
ALS Aqua Regia Repeats : Count Numeric	5	5	5	5	5	5	5
ALS Aqua Regia Repeats : Minimum	1.71	>500	2.09	5460	0.94	404	86.6
ALS Aqua Regia Repeats : Maximum	1.77	>500	2.45	6910	1.08	500	90
ALS Aqua Regia Repeats : Mean	1.736	>500	2.324	6252	1.026	452.6	88.6
ALS Aqua Regia Repeats : Standard Deviation	0.02	0.00	0.14	541.45	0.06	37.76	1.52
BV Aqua Regia : Count Numeric	5	5	5	5	5	0	5
BV Aqua Regia : Minimum	1.77	951	2.82	7339	1.19		88
BV Aqua Regia : Maximum	1.8	1033	2.84	7538	1.21		97
BV Aqua Regia : Mean	1.786	988.4	2.83	7439	1.202		92.8
BV Aqua Regia : Standard Deviation	0.01	37.55	0.01	78.60	0.01		3.42
SGS Aqua Regia : Count Numeric	5	6	6	6	6	6	6
SGS Aqua Regia : Minimum	1.88	887	2.5	7576	1.17	520	84.4
SGS Aqua Regia : Maximum	1.93	930	2.74	8478	1.25	563	87.6
SGS Aqua Regia : Mean	1.90	909.33	2.59	7920.17	1.20	542.83	85.95
SGS Aqua Regia : Standard Deviation	0.02	15.07	0.10	337.61	0.03	17.09	1.06