Preliminary Economic Assessment Technical Report
Clayton Valley Lithium Project
Esmeralda County, Nevada

Effective Date: September 4, 2018
Issue Date: October 1, 2018

Prepared for:

Cypress Development Corp.

Prepared by:

Global Resource Engineering, Ltd.

Qualified Persons
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Date and Signature Page

This Technical Report on the Clayton Valley Lithium Project is submitted to Cypress Development Corp. and is effective September 4, 2018.

The Qualified Persons and Responsible Report Sections follow:

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TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF PHOTOS

APPENDICES

Appendix A - Claims Lists

ABBREVIATIONS AND ACRONYMS

µm	microns
2-D	2-dimensional
3-D	3-dimensional
AAS	atomic absorption spectroscopy
BLM	Bureau of Land Management
CH_3COOH	acetic acid
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimeter
CMS	Continental Metallurgical Services, LLC
Cypress	Cypress Development Corp.
GRE	Global Resource Engineering Ltd.
H_2SO_4	sulfuric acid
Hazen	Hazen Research Inc.
HCl	hydrochloric acid
HNO_3	nitric acid
ICP-AES	inductively coupled plasma atomic emission spectroscopy
ICP-MS	inductively coupled plasma mass spectrometry
kg	kilogram
km^2	square kilometers
km^3	cubic kilometers
kWhr/t	kilowatt-hours/tonne
LCE	lithium carbonate equivalent
Li	lithium
$LiCO_3$	lithium carbonate
MMSA	Mining and Metallurgical Society of America
NAA	neutron activation analysis
NaOH	sodium hydroxide
NI	National Instrument
NSR	Net Smelter Return
PEA	Preliminary Economic Assessment
PLS	pregnant leach solution
ppm	parts per million
QA/QC	quality assurance/quality control
QP	qualified person
SG	specific gravity
SME	Society of Mining, Metallurgy & Exploration
USGS	United States Geological Survey

XRD	X-ray Diffraction

1.0 SUMMARY

Global Resource Engineering was retained by Cypress Development Corp. (Cypress) to prepare a National Instrument (NI) 43-101 compliant Preliminary Economic Assessment Technical Report for the Clayton Valley Lithium Project, Nevada.

1.1 Location and Property

The Clayton Valley Lithium Project (the project) is centered near 452800 m East, 4178200 m North, UTM NAD 83, Zone 11 North datum, in Esmeralda County, Nevada. The project's location is 220 miles south of Reno, Nevada. The regional gold mining town of Tonopah is 40 miles northeast of the project and the small community of Silver Peak lies 10 miles west of the project. The project lies entirely within T2S, R40E, Mt. Diablo Meridian. The project is accessed from Tonopah, Nevada, by traveling south on US Highway 95, then west on Silver Peak Road.

The project consists of 139 placer mining claims and 178 overlapping lode mining claims as listed in Table 4-1 and shown in Figure 4-1. The claims cover 4,780 acres and provide Cypress with the rights to lithium-bearing brines and mudstones on the property. The claims lie within portions of surveyed sections 14, 15, 16, 17, 20, 21, 22, 23, 27, 28 and 33 of T2S, R40E in the central and eastern portions of the Clayton Valley, Nevada.

The property is held 100% by Cypress, with all claims subject to a 3% NSR. The royalty can be brought down to a 1% NSR in return for $2 million in payments to the original property vendor. The claims require annual filing of Intent to Hold and cash payments to the BLM and Esmeralda County totaling $167 per 20 acres. All claims are all in good standing with the BLM and Esmeralda County.

The terrain is dominated by mound-like outcrops of mineralized mudstones, which are cut by dry, gravel wash bottoms. Access on the property is excellent due to the overall low relief of the terrain.

The project is in a region of active extraction of lithium brines and open pit gold mining. The immediately adjacent Silver Peak Lithium Production Complex has been in production since the 1960s. The project lies near power lines and regional towns that service the mining industry.

1.2 History

Cypress issued a Mineral Resource Estimate in June 2018 (GRE, 2018). This PEA updates that Mineral Resource Estimate.

1.3 Geology and Mineralization

The Clayton Valley is a closed basin near the southwestern margin of the Basin and Range geo-physiographic province of western Nevada. Horst and graben normal faulting is a dominant structural element of the Basin and Range and is thought to have occurred in conjunction with deformation due to lateral shear stress, resulting in disruption of large-scale topographic features.

Significant lithium concentrations are encountered in the sedimentary units of the Esmeralda within the project area at ground surface and to depths of up to 124 meters. The lithium bearing sediments primarily occur as calcareous and salty interbedded tuffaceous mudstones and claystones. The overall mineralized sedimentary package is a laterally and vertically extensive, roughly tabular zone of interbedded mudstone and claystone with at least two prominent oxidation horizons in the subsurface. The mineralized zone consists of three primary units: an "upper" olive-colored mudstone, "middle" blue mudstone/claystone, and "lower" olive-colored mudstone. The middle (reduced) portion of the mineralized zone represents most of the overall mineralized sedimentary package. The upper and lower mudstone units are oxidized to an olive-green color, while the middle mudstone/claystone is reduced and blue, black, or grey in color in fresh drill core. The three primary units are generally overlain by tuffaceous mudstone and underlain by increasingly sandy mudstones. Elevated lithium concentrations occur in all the uplifted lacustrine strata encountered, but lithium concentrations are notably higher and more persistent in the three primary units.

1.4 Drilling

Cypress drilled a total of 23 NQ-core holes within the project area from 2017 to early 2018. Drill hole depths range from 33 to 129.5 meters and totaled 1,904 meters drilled.

The drilling results generally indicate a particularly favorable section of ash-rich mudstones that extend to depths of up to approximately 120 meters, within which exists a strong, apparently planar, oxidation/reduction front. While the drill holes are widely spaced, averaging 650 to 700 meters between holes, the lithium profile with depth is consistent from hole to hole. Unweighted lithium content averages 929.8 ppm for all 665 samples assayed, with a range of 116 to 2,240 ppm.

1.5 Mineral Processing and Metallurgical Testing

The preliminary process design for the Clayton Valley Lithium project is based on laboratory tests conducted by SGS Canada in 2017 (DCH-5 Oxide and DCH-5 Reduced), Hazen Research Inc in 2017 and 2018 (DCH-16 Oxide and DCH-16 Reduced) and Continental Metallurgical Services, LLC in 2018 (DCH-2 Oxide and DCH-2 Reduced). These tests indicate the claystone minerals can be digested in dilute sulfuric acid, liberating the lithium as lithium sulfate.

The deposit is classified into two categories that include Oxidized and Reduced materials. Dilute sulfuric acid reached extractions as high as 78% from the oxidized material and 83.5% from the reduced sample. Although the test work is preliminary in nature, it suggests that a dilute sulfuric acid leach is a viable method of extracting the lithium found at the project. Test results indicate that lithium extractions greater than 80% are achievable with acid dosages of 5% at 75C-80C with 4 to 6 hours leaching. More detailed test work is required to examine individual lithologic units.

Continental Metallurgical Services, LLC developed and conducted a series of acid leaching diagnostic tests on a variety of samples from the deposit. The results indicate that the deposit, as a whole, is amenable to dilute sulfuric acid leaching.

Bond work index testing indicate the oxide and reduced samples would be categorized as very soft with a work index of 1 to 1.5 kilowatt-hours/tonne (kWhr/t). At this stage no grinding has not been included in the process design as the samples digested easily in water with minimal coarse solids present.

Preliminary tests were conducted related to the production of a final lithium product as lithium carbonate. Initial indications are that conventional sequential precipitation processes are able to effectively remove elements such as iron, aluminum, magnesium, and calcium prior to the precipitation of the final lithium carbonate. Lithium hydroxide and lithium carbonate production from sulfate leach solutions are well-defined commercial processes.

The test work by Hazen and CMS indicated the presence of significant levels of rare earth elements in the samples analyzed. Further, during dilute sulfuric acid leaching of the lithium a significant portion of the rare earths elements was also solubilized. Indications are that rare earth elements could contribute to the project economics, but additional test work is needed.

1.6 Mineral Resource Estimation

Cypress has staked additional placer and lode claims since GRE's June 5, 2018 Mineral Resource Estimate. GRE has updated the Clayton Valley Mineral Resource to include mineralization contained on those new claims. The economic break-even cut-off grade is 300 ppm Li, and is calculated based upon an operating cost of $17.50/t, recovery of 81.5% and product price of $13,000/tonne of LCE. The updated Mineral Resource results at cutoffs from 300 ppm to 1,200 ppm are summarized in Table 1-1.

This Mineral Resource estimation includes data from 23 drill holes. At a cutoff of 300 ppm, the results of the estimate were an Indicated Mineral Resource of 720.3 million kilograms (kg) of lithium within 831.0 million tonnes and an Inferred Mineral Resource of 963.0 million kg lithium within 1.12 billion tonnes. Within an initial pit area, at a cutoff of 300 ppm, there are 344.2 million kg lithium within 365.3 million tonnes in the Indicated category and 159.2 million kg lithium within 160.5 million tonnes of Inferred material (Table 1-2). The initial pit area contains resources sufficient to supply a 15,000 tonne per day operation for over 40 years.

Five to 10 additional holes are recommended in the initial pit area for resource conversion and development, with a goal of converting some of the Indicated Mineral Resource to the Measured category and most of the Inferred Mineral Resource to the Indicated or Measured categories.

Cautionary statements regarding Mineral Resource estimates:

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 into Mineral Reserves. Inferred Mineral Resources are that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Table 1-1: Summary of Clayton Valley Lithium Project Preliminary Mineral Resource Estimate (1000s)

Lithology	Tonne	Li-kg	Grade - ppm	Tonne	Li-kg	Grade - ppm	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm
	Indicated Mineral Resource @ 300 ppm Cutoff			Indicated Mineral Resource @ 600 ppm Cutoff			Indicated Mineral Resource @ 900 ppm Cutoff			Indicated Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**50,020**	**35,280**	**705**	43,198	31,500	729	1,827	1,776	973	0	0	-
Upper Olive	**151,438**	**135,340**	**894**	151,438	135,340	894	65,102	67,735	1,040	0	0	-
Main Blue	**248,394**	**270,850**	**1,090**	248,394	270,850	1,090	221,207	248,073	1,121	23,477	29,190	1,243
Lower Olive	**138,773**	**115,265**	**831**	138,773	115,265	831	28,475	28,409	998	942	1,159	1,231
Hard Bottom	**242,418**	**163,567**	**675**	186,661	132,527	710	3,089	2,860	926	0	0	-
Total	**831,042**	**720,303**	**867**	768,464	685,482	892	319,700	348,853	1,091	24,418	30,349	1,243
	Inferred Mineral Resource @ 300 ppm Cutoff			Inferred Mineral Resource @ 600 ppm Cutoff			Inferred Mineral Resource @ 900 ppm Cutoff			Inferred Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**50,307**	**34,352**	**683**	43,956	30,668	698	670	629	939	0	0	-
Upper Olive	**189,650**	**161,042**	**849**	189,650	161,042	849	56,531	57,362	1,015	0	0	-
Main Blue	**357,362**	**391,098**	**1,094**	357,362	391,098	1,094	343,370	379,114	1,104	10,668	13,000	1,219
Lower Olive	**176,530**	**145,886**	**826**	176,530	145,886	826	29,752	28,382	954	0	0	-
Hard Bottom	**346,461**	**230,584**	**666**	254,698	178,830	702	0	0	-	0	0	-
Total	**1,120,310**	**962,962**	**860**	1,022,195	907,524	888	430,323	465,486	1,082	10,668	13,000	1,219

Table 1-2: Classified Mineral Resources in Initial Pit Area (1000s)

Lithology	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm
	Indicated Mineral Resource @ 300 ppm Cutoff			Indicated Mineral Resource @ 600 ppm Cutoff			Indicated Mineral Resource @ 900 ppm Cutoff			Indicated Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**26,520**	**18,575**	**700**	23,004	16,623	723	0	0	-	0	0	-
Upper Olive	**74,964**	**72,186**	**963**	74,964	72,186	963	44,644	46,339	1,038	0	0	-
Main Blue	**140,873**	**160,389**	**1,139**	140,873	160,389	1,139	140,457	160,032	1,139	0	0	-
Lower Olive	**53,316**	**45,079**	**846**	53,316	45,079	846	12,843	12,326	960	0	0	-
Hard Bottom	**69,643**	**47,947**	**688**	69,155	47,670	689	33	30	911	0	0	-
Total	**365,316**	**344,176**	**942**	361,311	341,946	946	197,977	218,726	1,105	0	0	-

Lithology	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm
	Inferred Mineral Resource @ 300 ppm Cutoff			Inferred Mineral Resource @ 600 ppm Cutoff			Inferred Mineral Resource @ 900 ppm Cutoff			Inferred Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**11,776**	**8,125**	**690**	11,776	8,125	690	0	0	-	0	0	-
Upper Olive	**30,839**	**28,761**	**933**	30,839	28,761	933	15,306	15,436	1,008	0	0	-
Main Blue	**83,602**	**96,730**	**1,157**	83,602	96,730	1,157	83,423	96,570	1,158	15,712	19,618	1,249
Lower Olive	**8,066**	**7,525**	**933**	8,066	7,525	933	8,066	7,525	933	0	0	-
Hard Bottom	**26,174**	**18,067**	**690**	24,244	16,925	698	0	0	-	0	0	-
Total	**160,457**	**159,208**	**992**	158,527	158,066	997	106,795	119,531	1,119	15,712	19,618	1,249

1.7 Pit Design, Schedule and Mining

The intial pit for the project encompasses most of the minable land within the property boundaries and results in more than 95 years of mining capacity. Eight phases were developed to mine higher-grade material and a preliminary mining schedule was generated for the base case scenario based on a nominal daily production rate of 15,000 tonnes per day (tpd) of mill feed.

Several types of surface mining methods and equipment are potentially suitable for the Clayton Valley Lithium Project, including dozer and scraper, surface planer - continuous miner with conveyor and haul trucks, truck and loader, and in-pit semi-mobile feeder-breaker and repulper. No drilling and blasting is anticipated for the operation. GRE selected the in-pit feeder-breaker and slurry pumping for the base case because it has the lowest operating cost. The production equipment for this case includes a 22 cubic meter front end loader, a D10 class bull dozer, and one 90 tonne class haul truck to haul lower grade claystone to a waste dump. The stripping ratio is 0.10:1. The mine operates on a two 10 hour shift, 7 days per week schedule.

1.8 Processing

The process has been developed based on industry-standard operations and commercially-proven leaching and recovery circuits. The designed throughput for the process is 15,000 tonnes per day or 5,475,000 tonnes per year averaging 1,012 ppm lithium. The estimated lithium recovery is 81.5% producing 4,516 tonnes per year of lithium or approximately 24,042 tonnes of lithium carbonate.

The flowsheet developed represents a typical lithium production pathway producing a high-grade lithium carbonate product. The process has been divided into basic unit operations, including: feed preparation, lithium extraction, primary impurity removal, secondary impurity removal, solution polishing, lithium carbonate production, tailings, and utilities – acid production, water recycle, reagents.

Lithium extraction is achieved through agitated tank leaching with sulfuric acid, heated by introduction of live steam delivered from an acid plant heat recovery system. The leach solution impurities are removed in a series of stages of Primary Impurity Removal (PIR), Secondary Impurity Removal (SIR) and solution polishing. An evaporation stage in included to maintain solution tenors for higher efficiency impurity removal and product precipitation. The lithium carbonate product is formed through the addition of soda ash to the leach solution after impurity removal, then filtered and dried for shipment.

The filtered and washed primary leach residue, PIR residue, and SIR residue are combined and placed in a dry-stack tailing impoundment and, later, mined-out portions of the pit. Water will be recovered from spent leach solution via a reverse osmosis system with the retentate being pumped to an evaporation pond to allow potassium and other salts to crystalize.

The sulfuric acid plant is a Double Contact Double Absorption (DCDA) sulfur burning acid plant with an energy recovery system, capable of producing 2,000 tonnes per day of sulfuric acid (100% purity basis) by combusting elemental sulfur. The plant has the ability to produce up to 25 MW of electricity, but only enough generation is assumed to allow the acid plant to be electrically self-sufficient.

1.9 Costs and Economics

The base case mining and processing scenario (in-pit semi-mobile slurry pumping with dilute acid processing) results in initial capital costs, occurring in years -2 and -1, of $481 million and total capital costs for the 40-year schedule of $600 million.

Annual operating costs with contingency vary from $3.5 million to $88.9 million. Total operating costs for the 40-year schedule are $3.56 billion.

Recovery was set at 81.5% of the lithium tonnes processed, with production of 5.323 kg of lithium carbonate per tonne of contained lithium. Over the course of the 40-year schedule, there are 209.4 million kg of contained lithium, resulting in 170.7 million kg of recovered lithium and 909.2 million kg of recovered lithium carbonate.

Economic analysis of the Clayton Valley Lithium project, at a lithium carbonate price of $13,000/tonne of lithium carbonate, over the 40-year schedule, projects an after-tax Net Present Value @ 6% (NPV@6%) of $1.97 billion, NPV@8% of $1.45 billion, and NPV@10% of $773 million, and Internal Rate of Return (IRR) of 32.7%. The expected maximum negative cash flow is $488 million.

An allowance for state property and income taxes of 7% was included, and Federal taxes were included at 21% for this evaluation. Depreciation and amortization, depletion, and loss carry forward were included.

Salient results for the project base case are shown below.

- Mining operating cost per process tonne of $1.73, including the strip ratio of 0.1:1.
- Process operating cost per process tonne of $15.09. Sulfuric acid accounts for 65% of the processing costs.
- G&A operating cost per process tonne of $0.68.
- Total operating cost plus contingency per process tonne of $17.50, which equates to a cost of $3,983/tonne of LCE.
- Total cash cost (with capital included) per tonne of lithium carbonate is $4,609/tonne of LCE.
- Average annual production of 24.0 million kg of lithium carbonate.
- $6.2 billion after-tax cumulative cash flow for the 40-year schedule.
- Payback period of 2.7 years and Payback multiple of 12.8.
- After-tax NPV of 1.45 billion @ 8% discount rate and IRR of 32.7%.

GRE evaluated the after-tax NPV@8% sensitivity to changes in lithium carbonate price, capital costs, and operating costs. The base price used for lithium carbonate is $13,000/tonne LCE based on Benchmark's market study.

The after-tax NPV@8% is most sensitive to changes in lithium carbonate price, ranging from $390 million at 60% of the base case lithium carbonate price ($7,800/tonne) to $2.8 billion at 150% of the base case lithium carbonate price ($19,500/tonne), or approximately $263 million per 10% change in lithium carbonate price. The after-tax NPV@8% stays positive for the full range of lithium carbonate prices examined. The project has a breakeven IRR at a lithium carbonate price of $4,800/tonne.

The after-tax NPV@8% is least sensitive to changes in capital costs, ranging from $1.6 billion at 60% of the base case capital costs to $1.3 billion at 150% of the base case capital costs, or approximately $4.5 million per 10% change in capital costs. The after-tax NPV@8% stays positive for the full range of capital costs examined.

The after-tax NPV@8% is moderately sensitive to changes in operating costs, ranging from $1.8 billion at 60% of the base case operating costs to $1.1 billion at 150% of the base case operating costs, or approximately $7.9 million per 10% change in operating costs. The after-tax NPV@8% says positive for the full range of operating costs examined.

1.10 Conclusions and Recommendations

The project is a large lithium-bearing claystone deposit. The estimated resources in this report are open to depth and laterally in some areas. The lithium occurs as discreet mineralization that is readily available for direct acid leaching. The PEA limits the mine life to 40 years, but still indicates the project has good economics. The estimated initial capital cost is $482 million, with a Net Present Value @8% of $1.45 billion after tax and an internal rate of return of 32.7 percent. Relatively low acid consumption, combined with soft rock and low mining costs contribute to an average $3,983 / tonne LCE operating cost. The project has the potential to be a major supplier of lithium products in the world, and additional work is warranted.

GRE recommends the following activities be conducted for the Cypress Clayton Valley lithium project:

- Infill drilling to upgrade resource categories and optimize production schedule within the initial pit area
- Further testing for determination of acid concentration, consumption, temperature, and leach times for the individual units
- Determine optimum leaching configuration for process plant with respect to acid consumption and lithium extraction
- Bench-top testing to demonstrate production of lithium carbonate suitable for battery usage
- Detailed capital and operating cost estimates
- Investigate rare earth elements and other byproducts; quantify those elements in resources if appropriate
- Investigate alternative processing methods, including membranes and ion exchange resins for the concentration of lithium and other elements
- Investigate trade-offs between additional capital vs. saleable electrical generation for acid plant
- Initiate baseline data collection, hydrology and geotechnical studies
- Complete a Pre-Feasibility Study based upon the above results, with an estimated budget of $800,000.

2.0 INTRODUCTION

As requested by Cypress Development Corp. (Cypress), Global Resource Engineering (GRE) has prepared this National Instrument (NI) 43-101 Preliminary Economic Assessment (PEA) Technical Report for the Clayton Valley Lithium Project, Nevada, based on data collected from 2015 to present. This NI 43-101 Technical Report includes resources on the contiguous Dean and Glory claim blocks, which are referred to in this Technical Report as the "Clayton Valley Lithium Project."

Cypress previously published a NI 43-101 Technical Report summarizing exploration drilling results and other relevant data (Cypress Development Corp., 2018) for the Dean claim blocks only and a NI43-101 Technical Report Mineral Resource Estimate for the project (GRE, 2018).

The Qualified Persons for this report are Terre A. Lane, J. Todd Harvey, Hamid Samari, and J. J. Brown of GRE and Todd S. Fayram of Continental Metallurgical Services.

2.1 Scope of Work

The scope of work undertaken by GRE is to prepare a PEA for the Clayton Valley Lithium Project (the project) and prepare recommendations on further work required to advance the project to the Prefeasibilty Study (PFS) stage.

2.2 Qualified Persons

The Qualified Persons (QP) responsible for this report are:

- Terre A. Lane, Mining and Metallurgical Society of America (MMSA) 01407QP, Society for Mining, Metallurgy & Exploration (SME) Registered Member 4053005, Principal Mining Engineer, GRE
- J. Todd Harvey, PhD, QP, Member SME Registered Member 4144120, Director of Process Engineering, GRE
- Todd S. Fayram, QP, Member of SME MMSA #01300QP and owner of Continental Metallurgical Services, LLC.
- Hamid Samari, PhD, QP, MMSA #01519QP
- J. J. Brown, QP, SME Registered Member 4168244, PG

Practices consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2010) were applied to the generation of this PEA.

Ms. Lane, Dr. Harvey, Mr. Fayram, Dr. Samari, and Ms. Brown are collectively referred to as the "authors" of this PEA. Ms. Brown visited the project during February 6-9, 2018. In addition to their own work, the authors have made use of information from other sources and have listed these sources in this document under "References."

Table 2-1 identifies QP responsibility for each section of this report.

Table 2-1 List of Contributing Authors

Section	Section Name	Qualified Person
1	Summary	ALL
2	Introduction	ALL
3	Reliance on Other Experts	ALL
4	Property Description and Location	Terre Lane
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Terre Lane
6	History	Terre Lane
7	Geological Setting and Mineralization	J. J. Brown
8	Deposit Types	J. J. Brown
9	Exploration	J. J. Brown
10	Drilling	J. J. Brown
11	Sample Preparation, Analyses and Security	J. J. Brown
12	Data Verification	J. J. Brown
13	Mineral Processing and Metallurgical Testing	J. Todd Harvey and Todd S. Fayram
14	Mineral Resource Estimates	Terre Lane, Hamid Samari
15	Mineral Reserve Estimates	Terre Lane
16	Mining Methods	Terre Lane
17	Recovery Methods	J. Todd Harvey
18	Project Infrastructure	Terre Lane and J. Todd Harvey
19	Market Studies and Contracts	Terre Lane and J. Todd Harvey
20	Environmental Studies, Permitting and Social or Community Impact	Terre Lane
21	Capital and Operating Costs	Terre Lane, J. Todd Harvey,
22	Economic Analysis	Terre Lane
23	Adjacent Properties	Terre Lane
24	Other Relevant Data and Information	ALL
25	Interpretation and Conclusions	ALL
26	Recommendations	ALL
27	References	ALL

Note: Where multiple authors are cited, refer to author certificate for specific responsibilities.

2.3 Sources of Information

Information provided by Cypress included:

- Drill hole records
- Project history details
- Sampling protocol details
- Geological and mineralization setting
- Data, reports, and opinions from third-party entities
- Lithium assays from original records and reports

- Metallurgical reports
- Claim information and land position
- Royalty agreements

2.4 Units

All measurements used for the project are metric units unless otherwise stated. Tonnages are in metric tonnes, and grade is reported as parts per million (ppm) unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

The authors relied on statements by Cypress concerning geological and exploration matters in Sections 7.0, 8.0, and 9.0, mineral rights ownership data and legal and environmental matters included in Sections 4.0 and 5.0 of this report. All mineral rights owned by Cypress are the result of the Mining Law of 1872 and are on public lands administered by the BLM out of the Tonopah Field Office.

The authors have not independently conducted any title or other searches, but have relied on Cypress for information on the status of claims, property title, royalties, agreements, permit status, and other pertinent conditions.

The authors have reviewed and incorporated reports and studies as described within this Report, and have adjusted information that required amending.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The project is centered near 452800m East, 4178200m North, UTM NAD 83, Zone 11 North datum, in central Esmeralda County, Nevada. The location is 220 miles southeast of Reno, Nevada (Figure 4-1). The regional gold mining town of Tonopah is about 40 miles northeast of the project and the small community of Silver Peak lies 10 miles west of the project. The project lies entirely within T2S, R40E, Mt. Diablo Meridian. The project is accessed from Tonopah, Nevada, by traveling 22 miles south on US Highway 95, then 20 miles west on Silver Peak Road.

Figure 4-1: Project Location Map

4.2 Mineral Rights and Tenure

The project consists of 139 placer mining claims and 178 overlapping lode mining claims as listed in Table 4-1 and shown in Figure 4-1. The claims cover 4,780 acres and provide Cypress with the rights to lithium-bearing brines and mudstones on the property. The claims lie within portions of surveyed sections 14, 15, 16, 17, 20, 21, 22, 23, 27, 28 and 33 of T2S, R40E in the central and eastern portions of the Clayton Valley, Nevada.

The placer claims cover the entire project area. Lode claims were staked over the placer claims to insure control of all mineral rights related to either brine or mudstones. The placer claims vary in size from 20 to 80 acres and were staked as even divisions of a legal section, as required under placer mine claim regulations. The lode claims are a maximum of 600 x 1,500 feet in size or about 20.5 acres each and together cover an area of 3,587 acres.

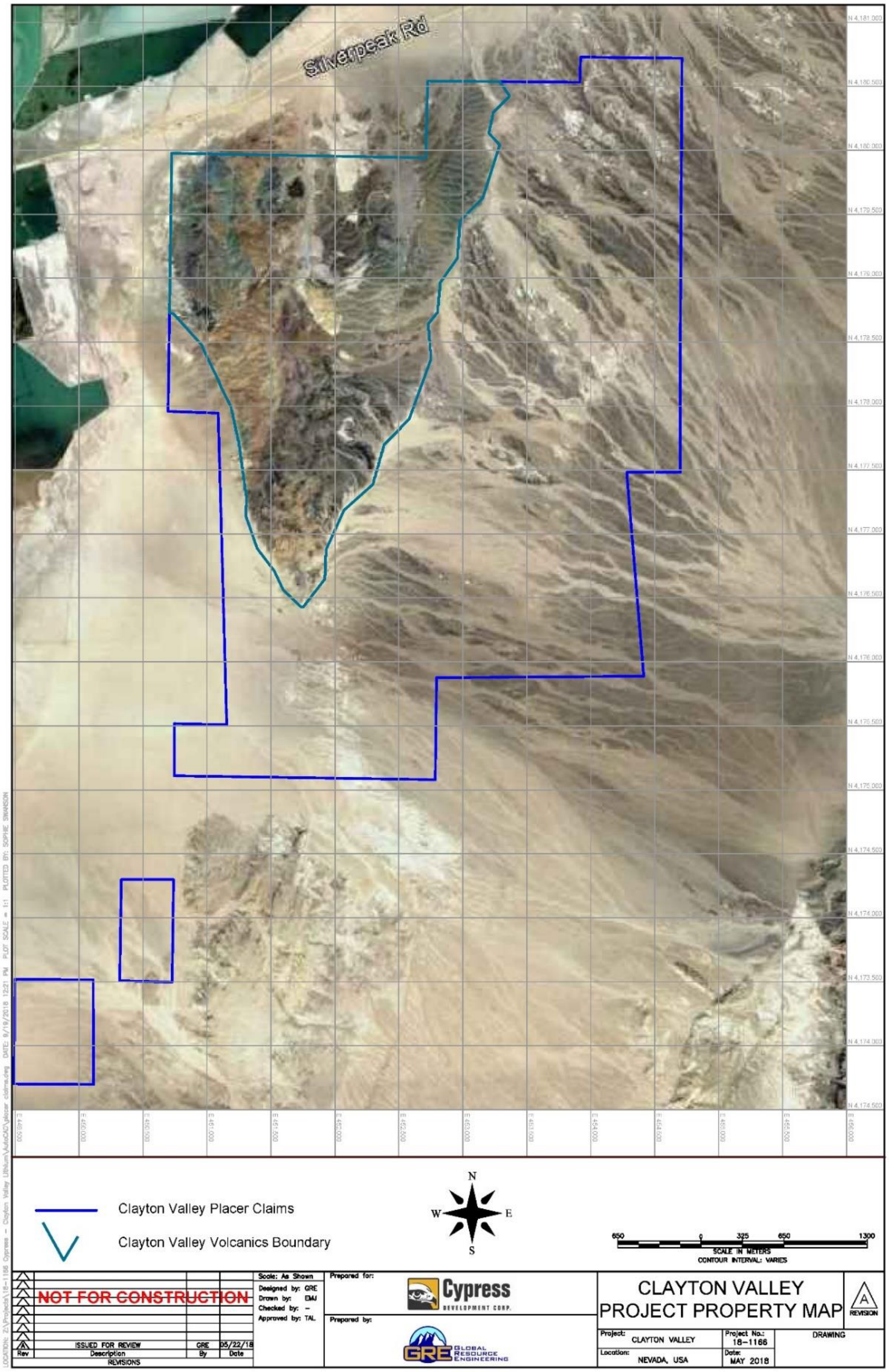
The property is held 100% by Cypress, with all claims subject to a 3% NSR. The royalty can be brought down to a 1% NSR in return for $2 million in payments to the original property vendor. The claims require annual filing of Intent to Hold and cash payments to the BLM and Esmeralda County totaling $167 per 20 acres. All claims are all in good standing with the BLM and Esmeralda County.

Table 4-1 Clayton Valley Property Mineral Claims

Placer Mining Claims		
NMC From	NMC To	Claims
NMC1119079	NMC1119089	11
NMC1119046	NMC1119078	33
NMC1120318	NMC1120352	35
NMC1121389	NMC1121394	6
NMC1121397	NMC1121400	4
NMC1124933	NMC1124952	20
NMC1129564	NMC1129565	2
NMC1177632	NMC1177643	12
NMC1177672	NMC1177687	16
Total Placer Claims		139

Lode Mining Claims		
NMC From	NMC To	Claims
NMC1136414	NMC1136484	71
NMC1162324	NMC1162402	79
NMC1177644	NMC1177655	12
NMC1177656	NMC1177671	16
Total Lode Claims		178

Figure 4-2: Clayton Valley Lithium Project Property Map



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The project is accessed from Tonopah, Nevada, by traveling 22 miles south on US Highway 95, then 20 miles west on Silver Peak Road, a paved and well-maintained gravel road.

5.2 Climate

The climate of the Clayton Valley is hot in summer, with average high temperatures around 100 °F and cool in the winter with average daily lows of 15 to 30 °F. Precipitation is dominantly in the form of thunderstorms in late summer. Snow cover in winter is rare.

Year-round low humidity aids in evaporation. Wind storms occur in the fall, winter, and spring.

5.3 Physiography

The project is in the Great Basin physiographic region and, more precisely, within the Walker Lane province of the western Great Basin. The Clayton Valley is a flat-bottomed salt basin that is surrounded by a complete pattern of mountain ranges. Broad, low passes lead into the basin from the north and east.

On the project itself, the terrain is dominated by mound-like outcrops of mineralized mudstones, which are cut by dry, gravel wash bottoms. Access at the project is excellent due to the overall low relief of the terrain (see Photo 5-1 Photo 5-2, and Photo 5-3).

Photo 5-1: Northern Half of Clayton Valley Lithium Project Looking East

Clayton Ridge is 2 miles in background, where basement rocks are exposed to the east of a major normal fault.

Photo 5-2: Clayton Valley Lithium Project, Dry Wash Channels and Mounds of Mineralized Mudstone

Photo 5-3: Typical Outcrop at Clayton Valley Lithium Project

Note tuffaceous unit overlying olive green mudstone. This interbedding is typical of the Upper Tuffaceous Mudstone unit.

5.4 Local Resources and Infrastructure

The project is in a region of active extraction of lithium brines and open pit gold mining. The immediately adjacent Silver Peak Lithium Production Complex has been in production since the 1960s. The project lies near paved roads, power lines, and regional towns that service the mining industry.

6.0 HISTORY

6.1 Project History

The project area shows signs of limited past exploration in the form of old weathered pits and trenches, and rare old piled stone rock mound claim corners. The area is roughly mapped and is shown as Esmeralda Formation sedimentary rocks and volcanic rocks on 1960s era geologic maps. The mapping mentioned here is the only known written evidence of geologic work in the project area. The DB placer claims were staked as part of the Rodina effort; these claims covered the entire project but were dropped.

The United States Geological Survey (USGS) has reportedly worked in the mudstones on several occasions. Limited sampling was completed as part of the USGS traverses. An assay of >2,000 ppm Li was noted on the west side of Angel Island from work done in the 1970s. The majority of USGS work in the basin was focused on lithium brine investigations.

The Nevada Bureau of Mines and Geology did work with mineralized mudstones on the Glory claims. The ongoing work involves XRD work on thin pumice layers within the exposed mudstone package.

There is no indication of any drilling occurring on the project prior to Cypress' efforts in 2017. Drilling by Noram Ventures in an area near the northeast corner of the project was done in winter 2016-2017, and again in 2018. Spearmint Resources drilled three holes south of the property in 2018.

A series of bench like open cuts into mudstone units has occurred along the west flank of Angel Island. The cuts and quarries are of recent age and may still be used. These operations have occurred in the recent past on Cypress placer claims in the southwest portion of the project, but are largely located on private lands owned by Albemarle Corp.

There is very little past surface exploration work. A small number of surface samples of mineralized mudstone were collected, and a significant lithium anomaly was noted by the USGS.

6.2 Compilation of Reports on Exploration Programs

The February 2018 Technical Report (Cypress Development Corp., 2018) was the first report to document exploration of the project. Other descriptions of the mineralized mudstones at the project are contained within Cypress news releases of 2016 and 2017 as well as within well-organized maps and other documents which are available on the Cypress website.

Numerous USGS reports are available detailing drill results and other activities in the adjacent salt playa.

Additionally, both Pure Energy Resources and Noram Ventures have produced a series of NI 43-101 compliant reports of nearby properties. The Pure Energy reports detail investigation of commercial grade brine resources immediately west of the project, while the Noram reports outline significant lithium exploration results to the northast of the project.

Reports from both the private and public sectors were read by the authors.

6.3 Historical Mineral Resource Estimate

GRE reported the Mineral Resource for Clayton Valley June 5, 2018 (GRE, 2018). Cypress staked additional claims since that time, resulting in changes to property boundaries and re-interpretation of the deposit model and pit-constrained mineral resources. **Table 6-1** shows the June 5, 2018 Mineral Resource.

Table 6-1: June 5, 2018 Clayton Valley Lithium Project Mineral Resource Estimate (1000s)

Lithology	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm
	Indicated Mineral Resource @ 300 ppm Cutoff			Indicated Mineral Resource @ 600 ppm Cutoff			Indicated Mineral Resource @ 900 ppm Cutoff			Indicated Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**58,700**	**41,500**	**707**	51,700	37,600	727	2,000	1,900	950	-	-	-
Upper Olive	**148,300**	**133,000**	**897**	148,300	133,000	897	64,700	67,700	1,046	-	-	-
Main Blue	**220,500**	**238,400**	**1,081**	220,500	238,400	1,081	190,300	213,100	1,120	22,500	28,000	1,244
Lower Olive	**132,200**	**112,500**	**851**	132,200	112,500	851	33,700	33,300	988	900	1,100	1,222
Hard Bottom	**136,900**	**92,100**	**673**	102,300	72,700	711	2,000	1,800	900	-	-	-
Total	**696,600**	**617,500**	**886**	655,000	594,200	907	292,700	317,800	1,086	23,400	29,100	1,244
	Inferred Mineral Resource @ 300 ppm Cutoff			Inferred Mineral Resource @ 600 ppm Cutoff			Inferred Mineral Resource @ 900 ppm Cutoff			Inferred Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**65,300**	**45,000**	**689**	62,200	43,200	695	500	500	1,000	-	-	-
Upper Olive	**112,400**	**99,300**	**883**	112,400	99,300	883	43,200	44,600	1,032	-	-	-
Main Blue	**190,700**	**196,800**	**1,032**	190,700	196,800	1,032	150,200	163,200	1,087	5,600	6,800	1,214
Lower Olive	**149,400**	**124,400**	**833**	149,400	124,400	833	35,000	33,400	954	-	-	-
Hard Bottom	**125,000**	**82,100**	**657**	80,300	56,800	707	-	-	-	-	-	-
Total	**642,800**	**547,600**	**852**	595,000	520,500	875	228,900	241,700	1,056	5,600	6,800	1,214

7.0 GEOLOGIC SETTING AND MINERALIZATION

The following descriptions of the regional and local geologic setting of the Clayton Valley are largely based on work completed by Davis and Vine (1979), Davis et. al (1986), Munk (2011) and Bradley et. al (2013), and much of the following text is modified and/or excerpted from these reports. The author has reviewed this information and available supporting documentation in detail, and finds the discussion and interpretations presented herein to be reasonable and suitable for use in this report.

7.1 Regional Geology

The Clayton Valley Lithium Project is part of a closed basin near the southwestern margin of the Basin and Range geo-physiographic province of western Nevada. Horst and graben normal faulting is a dominant structural element of the Basin and Range and is thought to have occurred in conjunction with deformation due to lateral shear stress, resulting in disruption of large-scale topographic features. The Walker lane, a zone of disrupted topography (Locke, et al., 1940) perhaps related to right-lateral shearing (Stewart, 1967), may pass within a few kilometers of the northern and eastern boundaries of Clayton Valley. The Walker lane is not well defined in this area and may be disrupted by the east-trending Warm Springs lineament (Ekren, et al., 1976), which could be a left-lateral fault conjugate to the Walker lane (Shawe, 1965). To the west of Clayton Valley, the Death Valley-Furnace Creek fault zone is a right-lateral fault zone that may die out against the Walker lane northwest of the valley. South of Clayton Valley, the arcuate form of the Palmetto Mountains is thought to represent tectonic “bending,” a mechanism taking up movement in shear zones at the end of major right lateral faults (Albers, 1967).

In the mountains bordering the valley to the east and west, faults in Cenozoic rocks generally trend about N20° to 40°E. Near the margins of the playa surface, fault scarps having two distinct trends have been studied in detail (Davis, et al., 1979). At the eastern margin, a set of moderately dissected scarps in Quaternary alluvial gravels strike about N20°E. In the east central portion of the valley, a more highly dissected set of scarps in alluvium and upper Cenozoic lacustrine sediments strikes about N65°E. If the modification of these fault scarps is similar to fault-scarp modification elsewhere in Nevada and Utah (Wallace, 1977; Bucknam, et al., 1979) the most recent movement on the N20°E set of scarps probably occurred less than 10,000 years ago, while the last movement on the N65°E set is probably closer to 20,000 years in age (Davis, et al., 1979).

Regional basement rocks consist of Precambrian (late Neoproterozoic) to Paleozoic (Ordovician) carbonate and clastic rocks deposited along the ancient western passive margin of North America. Regional shortening and low-grade metamorphism occurred during late Paleozoic and Mesozoic orogenies, along with granitic emplacement during the mid to late Mesozoic (ca. 155 and 85 Ma). Tectonic extension began in the late Cenozoic (~16 Ma) and has continued to the present.

East of Clayton Valley, more than 100 cubic kilometers (km^3) of Cenozoic ash-flow and air-fall tuff is exposed at Clayton Ridge and as far east as Montezuma Peak. These predominantly flat-lying, pumiceous rocks are interbedded with tuffaceous sediments between Clayton Ridge and Montezuma Peak; but at Montezuma Peak these rocks are altered considerably and dip at angles of as much as 30°. In the Montezuma Range, they are unconformably overlain by rhyolitic agglomerates. Davis et al. (1986) speculate that the source of these tuff sheets may have been a volcanic center to the east near

Montezuma Peak, to the south in the Montezuma Range, the Palmetto Mountains, Mount Jackson, or perhaps even the Silver Peak center to the west.

Cenozoic sedimentary rocks are exposed in the Silver Peak Range, in the Weepah Hills, and in the low hills east of the Clayton Valley playa. These rocks all are included in the Esmeralda Formation (Turner, 1900). The Esmeralda Formation consists of sandstone, shale, marl, breccia, and conglomerate, and is intercalated with volcanic rocks, although Turner (1900) excluded the major ash-flow units and other volcanic rocks in defining the formation. The rocks of the Esmeralda Formation in and around Clayton Valley apparently represent sedimentation in several discrete Miocene basins. The age of the lower part of the Esmeralda Formation in Clayton Valley is not known, but an air-fall tuff in the uppermost unit of the Esmeralda Formation has a K-Ar age of 6.9 ± 0.3 Ma (Robinson, et al., 1968).

The regional geology is illustrated in Figure 7-1.

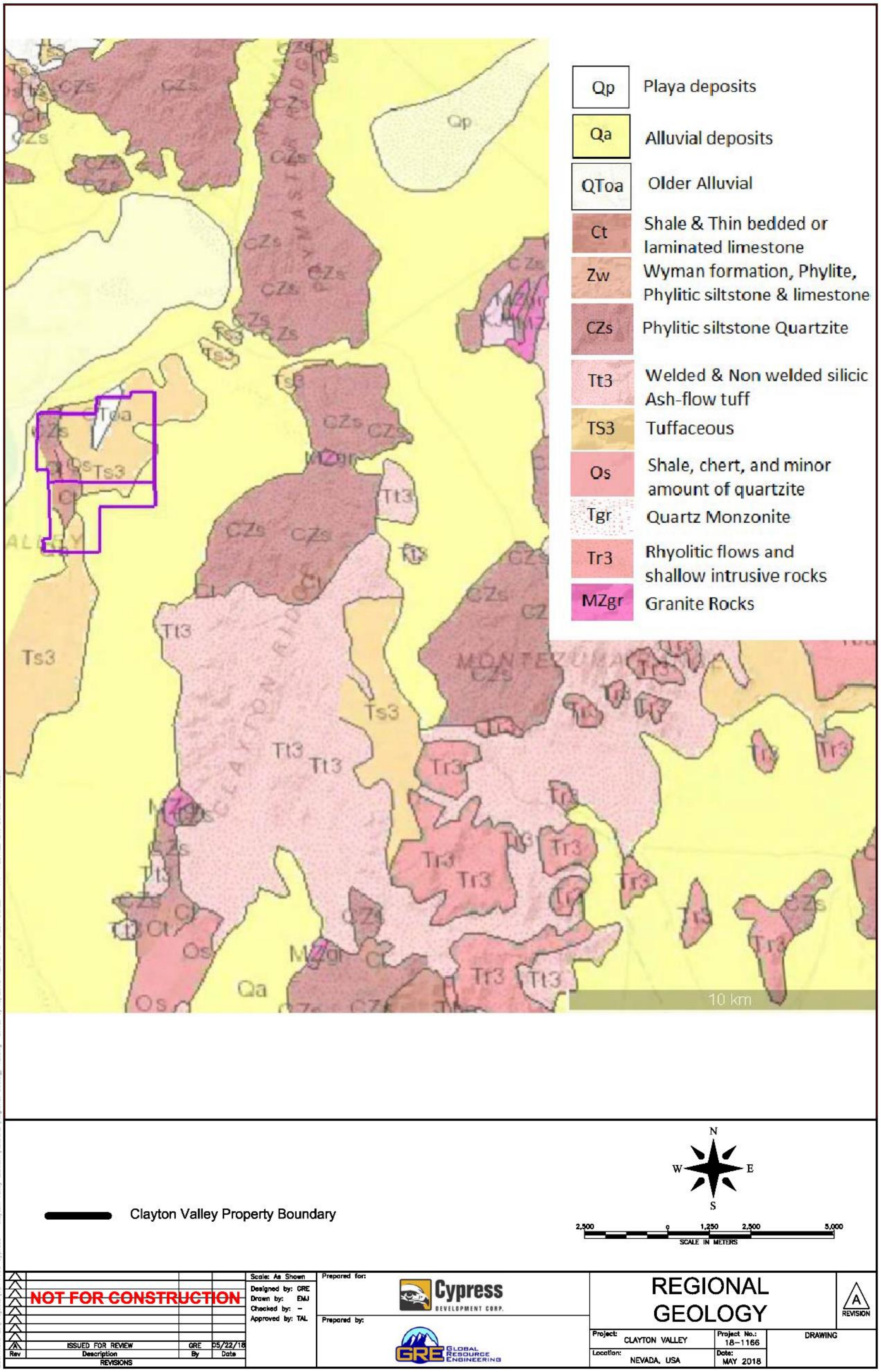
7.2 Local Geologic Setting

Clayton Valley is the lowest in elevation of a series of intermediate size playa filled valleys, with a playa floor of about 100 square kilometers (km^2) that receives surface drainage from an area of about 1,300 km^2. The valley is fault-bounded on all sides, delineated by the Silver Peak Range to the west, Clayton Ridge and the Montezuma Range to the east, the Palmetto Mountains and Silver Peak Range to the south, and Big Smokey Valley, Alkali Flat, Paymaster Ridge, and the Weepah Hills to the north.

The valley lies within an extensional half-graben system between a young metamorphic core complex and its breakaway zone (Oldow, et al., 2009). The general structure of the north part of the Clayton Valley basin is known from geophysical surveys and drilling to be a graben structure with its most down-dropped part on the east-northeast side of the basin along the extension of the Paymaster Canyon Fault and Angel Island Fault (Zampirro, 2005). A similar graben structure has been identified in the south part of the Clayton Valley basin through gravity and seismic survey.

Multiple wetting and drying periods during the Pleistocene resulted in the formation of lacustrine deposits, salt beds, and lithium-rich brines in the Clayton Valley basin. Extensive diagenetic alteration of vitric material to zeolites and clay minerals has taken place in the tuffaceous sandstone and shale of the Esmeralda Formation, and anomalously high lithium concentrations accompany the alteration. The lacustrine sediment near the center of pluvial lakes in Clayton Valley is generally green to black calcareous mud. According to (Davis, et al., 1986), about half of the mud, by weight, is smectite and illite, which are present in nearly equal amounts, with the remaining half composed of calcium carbonate (10-20%), kaolinite, chlorite, volcaniclastic detritus, traces of woody organic material, and diatoms. These tuffaceous lacustrine facies of the Esmeralda Formation contain up to 1,300 parts per million (ppm) lithium and an average of 100 ppm lithium (Kunasz, 1974; Davis, et al., 1979). Lithium bearing clays in the surface playa sediments contain from 350 to 1,171 ppm lithium (Kunasz, 1974). More recent work by Morissette (2012) confirms elevated lithium concentrations in the range of 160-910 ppm from samples collected on the northeast side of Clayton Valley. Miocene silicic tuffs and rhyolites along the basin's eastern flank have lithium concentrations up to 228 ppm (Price, et al., 2000).

Figure 7-1: Regional Geology



7.3 Project Geology and Mineralization

7.3.1 Lithology

The western portion of the project area is dominated by the uplifted basement rocks of Angel Island, while the southern and eastern portions are dominated by uplifted, lacustrine sedimentary units of the Esmeralda Formation. Within the project area, the Esmeralda Formation is comprised of fine grained sedimentary and tuffaceous units, with some occasionally pronounced local undulation and minor faulting (Figure 7-2). The resulting topography consists of elongate, rounded ridges of exposed Esmeralda Formation separated by washes and gullies filled with alluvial and colluvial gravels and fine sediment. The ridge tops are commonly mantled weathered remnants of rock washed down from the surrounding highlands.

Cypress provides the following description of the individual stratigraphic units of the Esmeralda within the project area, which together form a laterally and vertically continuous stratigraphic section which underlies the eastern 60% of the project area (Figure 7-3):

Recent Gravel Cover - a thin veneer of polylithic cobble, boulder and sand cover exists over portions of the project. This cover unit varies from 0 to 3 meters in thickness. The gravel is being shed out of steep canyons cutting Clayton ridge to the east.

Upper Tuffaceous Mudstone Cap Rock - this is the highest unit in the mineralized sequence and consists of interbedded silty mudstones and harder tuffaceous beds. The unit is approximately 70% mudstone and 30% hard tuff layers. This layer is generally 3 to 10 meters thick. Grades average 600 to 700 ppm Li.

The Upper Olive Mudstone Unit - this unit starts the main ash rich mudstone sequence which contains much of the lithium mineralization found to date. The unit is oxidized and contains locally abundant iron oxide staining and partial layer replacement. Below an interbedded top section, this unit becomes massive with uniform texture, color, and grain size. This layer is generally 20 to 30 meters thick. Average grade is 800 ppm Li.

Main Blue Mudstone Unit - (aka the Black and Blue), this is a continuation of the Upper Olive unit above but below an oxidation-reduction boundary. A sharp color change from robust olive to blue occurs at the redox, or several times as the redox is locally complex and interbedded. This layer is generally 10 to 20 meters thick. Average grade is 1,100 ppm Li.

Lower Olive Mudstone Unit - this unit underlies a second, locally complex oxidation-reduction boundary, where the blue and black unit above change gradationally back to olive colored mudstone. Fully olive colored mudstone sections occur within this unit that contain completely black, reduced mudstone interbeds. The uppermost 9 to 12 meters are well mineralized. After about 12 meters, the unit starts to turn tan and to contain increasing percentages of hard, sandy or other silica layers. Pumice fragments are common in this unit. This layer is generally 15 to 20 meters thick. Average grade is 800 ppm Li.

Figure 7-2 Geologic Map of the Clayton Valley Project

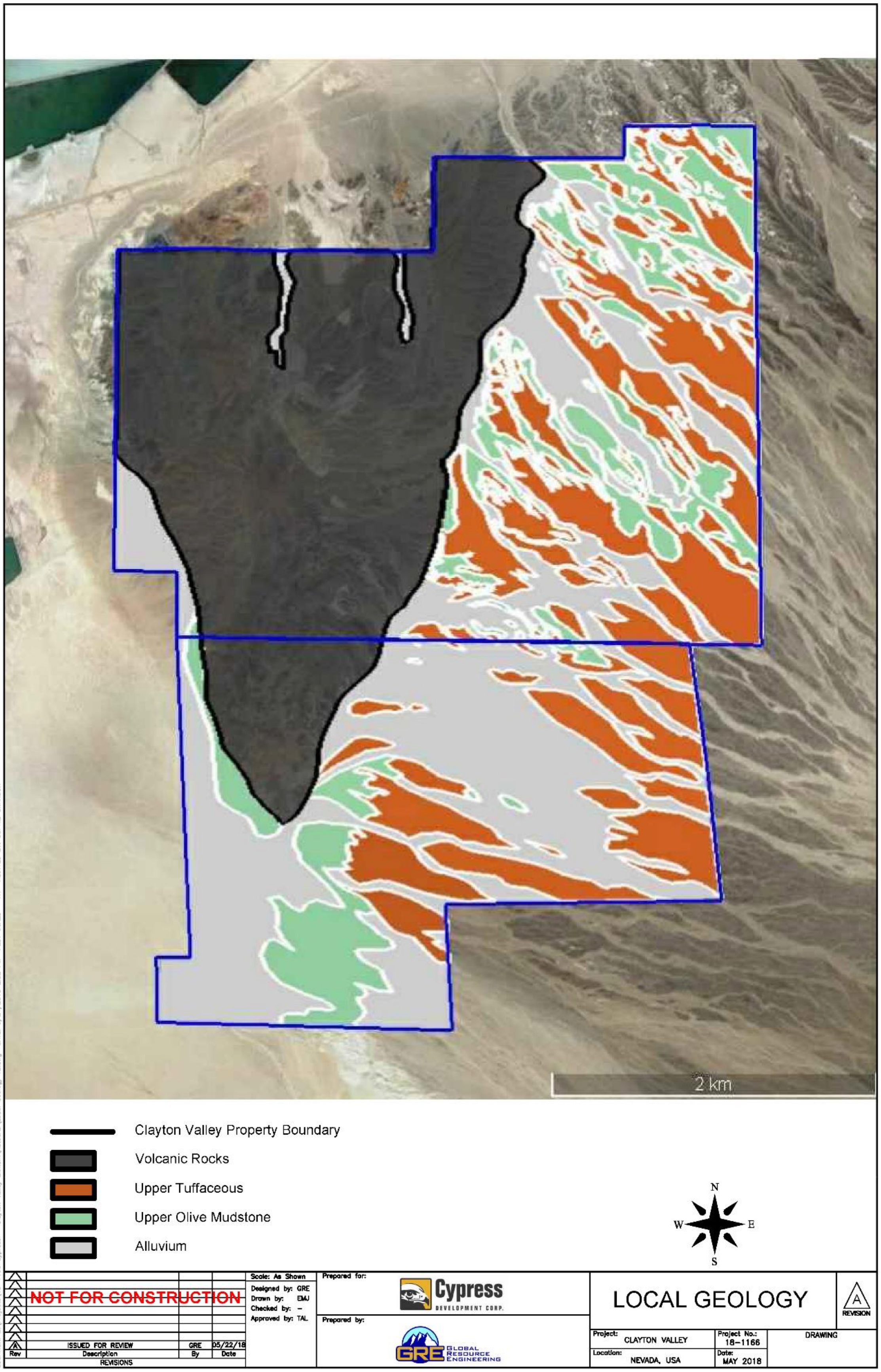
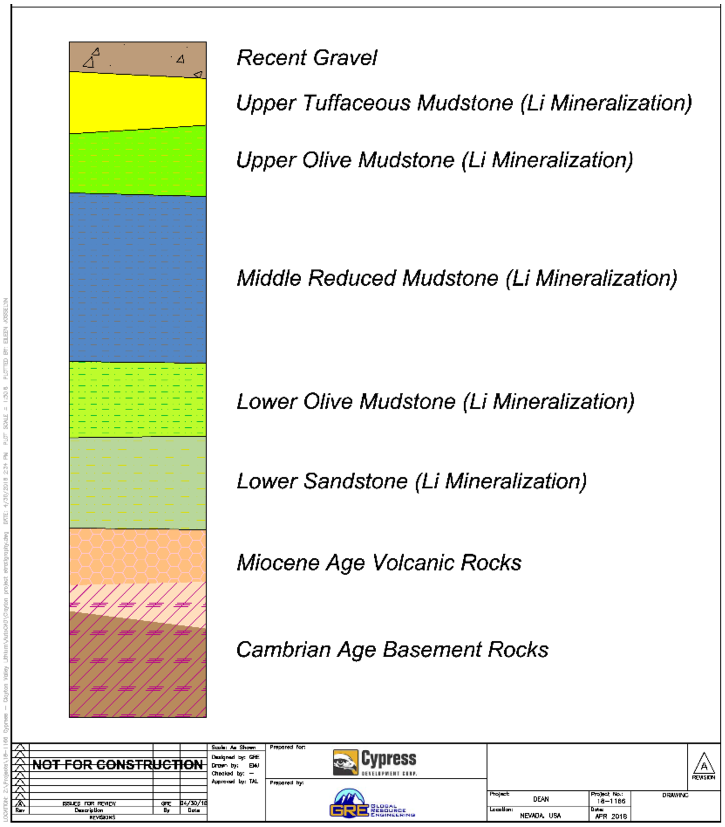


Figure 7-3: Clayton Valley Lithium Project Stratigraphy



The Hard Bottom – this unit has a gradational upper contact and represents a unit where the olive color is totally changed to tan and in which the percentage of sand is 20% to 40%. Lithium values are lower than in the strongly mineralized zones above and range from 400 to 700 ppm Li. Cypress has not drilled through this unit, and its thickness and the underlying structure remain unknown.

7.3.2 Mineralization

Significant lithium concentrations are encountered in the sedimentary units of the Esmeralda within the project area at ground surface and to depths of up to 124 meters. The lithium bearing sediments primarily occur as calcareous and salty interbedded tuffaceous mudstones and claystones. The overall mineralized sedimentary package is a laterally and vertically extensive, roughly tabular zone of interbedded mudstone and claystone with at least two prominent oxidation horizons in the subsurface. The mineralized zone consists of three primary units: an “upper” olive-colored mudstone, “middle” blue mudstone/claystone, and “lower” olive-colored mudstone. The middle (reduced) portion of the mineralized zone represents much of the overall mineralized sedimentary package. The upper and lower mudstone units are oxidized to an olive-green color, while the middle mudstone/claystone is reduced and blue, black, or grey in color in fresh drill core. The three primary units are generally overlain by tuffaceous mudstone and underlain by increasingly sandy mudstones. Elevated lithium concentrations occur in all the uplifted lacustrine strata encountered, but lithium concentrations are notably higher and more persistent in the three primary units. These units are 20 to 80 meters thick, with the middle units, referred to as Upper Olive, Main Blue, and Lower Olive, respectively, having average grades of 850 to 1,100 ppm. Portions of these units could be selectively mined at grades exceeding 1,100 ppm lithium.

A series of longitudinal and cross sections showing logged geology and hole to hole geologic interpretation and assay results from split core intervals was prepared. Representative sections are presented in Section 14 as Figure 14-17 and Figure 14-18.

Cypress splits 100% of drill core from surface and through the entire mudstone section and into the underlying hard sandstone units seen in the bottom of many of the holes. Ten-foot interval samples taken between core footage marker blocks make up over 90% of the assay data. These individual sample assay results are plotted on the sections and are also available in the compiled drill exploration database for the project.

8.0 DEPOSIT TYPE

Lithium is known to occur in potentially economic concentrations in three types of deposits: pegmatites, continental brines, and clays. While lithium is produced from both pegmatites and continental brines, with brines the most important source of lithium worldwide, economic extraction of lithium from clays has yet to be proven.

In clay deposits, lithium is most often associated with the smectite (montmorillonite) group mineral hectorite, which is rich in both magnesium and lithium. The USGS presents a preliminary descriptive model of lithium in smectites of closed basins (Asher-Bolinder, 1991), Model 251.3(T), which postulates three forms of genesis for clay lithium deposits: alteration of volcanic glass to lithium-rich smectite; precipitation from lacustrine waters; and incorporation of lithium into existing smectites. In each case, the depositional/diagenetic model is characterized by abundant magnesium, silicic volcanics, and an arid environment. The project appears to have a higher portion of illite and kaolinite than some other claystone deposits. This appears to differentiate the project from other claystone deposits.

Regional geologic attributes of lithium clay deposits, as presented by Asher-Bolinder (1991), include a Basin-and-Range or other rift tectonostratigraphic setting characterized by bimodal volcanism, crustal extension, and high rates of sedimentation. The depositional environment is limited to arid, closed basins of tectonic or caldera origin, with an age of deposition ranging from Paleocene to Holocene. Host rocks include volcanic ashes, pre-existing smectites, and lacustrine beds rich in calcium and magnesium.

The project is reasonably well represented by the USGS preliminary deposit model, which describes the most readily ascertainable attributes of such deposits as light-colored, ash-rich, lacustrine rocks containing swelling clays, occurring within hydrologically closed basins with some abundance of proximal silicic volcanic rocks. The geometry of the deposit at the project is roughly tabular, with the lithium concentrated in gently dipping, locally undulating, sedimentary strata of the Esmeralda Formation. The sedimentary units consist of interbedded calcareous, ash-rich mudstones and claystones, with interbeds of sandy and tuffaceous mudstone/siltstone and occasional poorly cemented sandstone. The lithium is largely concentrated within the mudstones and claystones, but elevated concentrations were recorded in a sandstone unit that underlies the clays.

Cypress geologists suggest deposition of the lithium-rich sediments late in the history of the associated paleo brine lake, based largely on the stratigraphic position of the mudstones and claystones above the thick overall sandstone- and siltstone-dominated basin fill. Such a setting would be ideal for concentration of lithium from ash and groundwater inputs over an extensive period. As a result, the lithium-rich strata may well represent several million years of lithium input and concentration within the basin.

The lithium-bearing sediments of the deposit surround an oxidation/reduction horizon that is readily recognizable in core samples. Based on drilling results to date, the higher lithium concentrations occur largely within a thick (up to 80-meter) central reduced zone and in oxidized zones that both overlie and underlie the zone of reduction. Cypress geologists suggest that this distribution of mineralization results from modern, oxidizing surface waters penetrating down dip within more permeable facies of the sedimentary package to create a series of oxidation-reduction fronts. Based on this interpretation, significantly elevated lithium concentrations within the deposit may represent redistribution of lithium in a tabular roll front reduction environment.

9.0 EXPLORATION

Cypress began exploring the project in late 2015. Exploration activities carried out by Cypress to date include surface sampling, detailed geological mapping, and drilling.

9.1 Surface Sampling

Surface samples of friable outcropping mudstone were collected by Cypress geologists over a 10-month period ending in October 2016. The samples were largely located in the eastern and southern portions of the project area.

In total, Cypress has submitted 634 soil and rock chip samples (28 of which were duplicate samples) for laboratory analysis by 33 element 4-acid inductively coupled plasma atomic emission spectroscopy (ICP-AES) and 35-element aqua regia atomic absorption spectroscopy (AAS). Analytical results indicate elevated lithium concentrations at ground surface over nearly the full extent of the area sampled. Assay values exceeding 2,000 ppm Li were returned for samples collected in the northern portion of the Glory property and from just west of the Angel Island fault, in the central portion of the Project area.

9.2 Mapping

Cypress has conducted general geologic surface mapping over most of project area. The total mapped surface is roughly 8-10 km^2. The surficial geologic maps are used as a general guide for exploration planning in conjunction with soil sampling and drilling results. The author knows of no other exploration activities carried out by Cypress, except for drilling, that warrant discussion in this report.

10.0 DRILLING

10.1 Cypress Drilling Exploration

Cypress conducted drilling exploration at the project in 2017 and early 2018. A total of 23 vertical, NQ-size (1.87-in core diameter) core holes were drilled, all by Morning Star Drilling of Three Forks, Montana, using an Acker track-mounted drill rig. Drill hole depths range from 33 to 129.5 meters (108 to 425 feet), totaling 1,905 meters (6,250 feet) drilled. Given the shallow depth of the holes, no downhole surveys were completed. Drill hole locations are presented in plan view on Figure 10-1, and drill hole collar details are summarized in Table 10-1.

Figure 10-1: Clayton Valley Lithium Project Drill Hole Locations

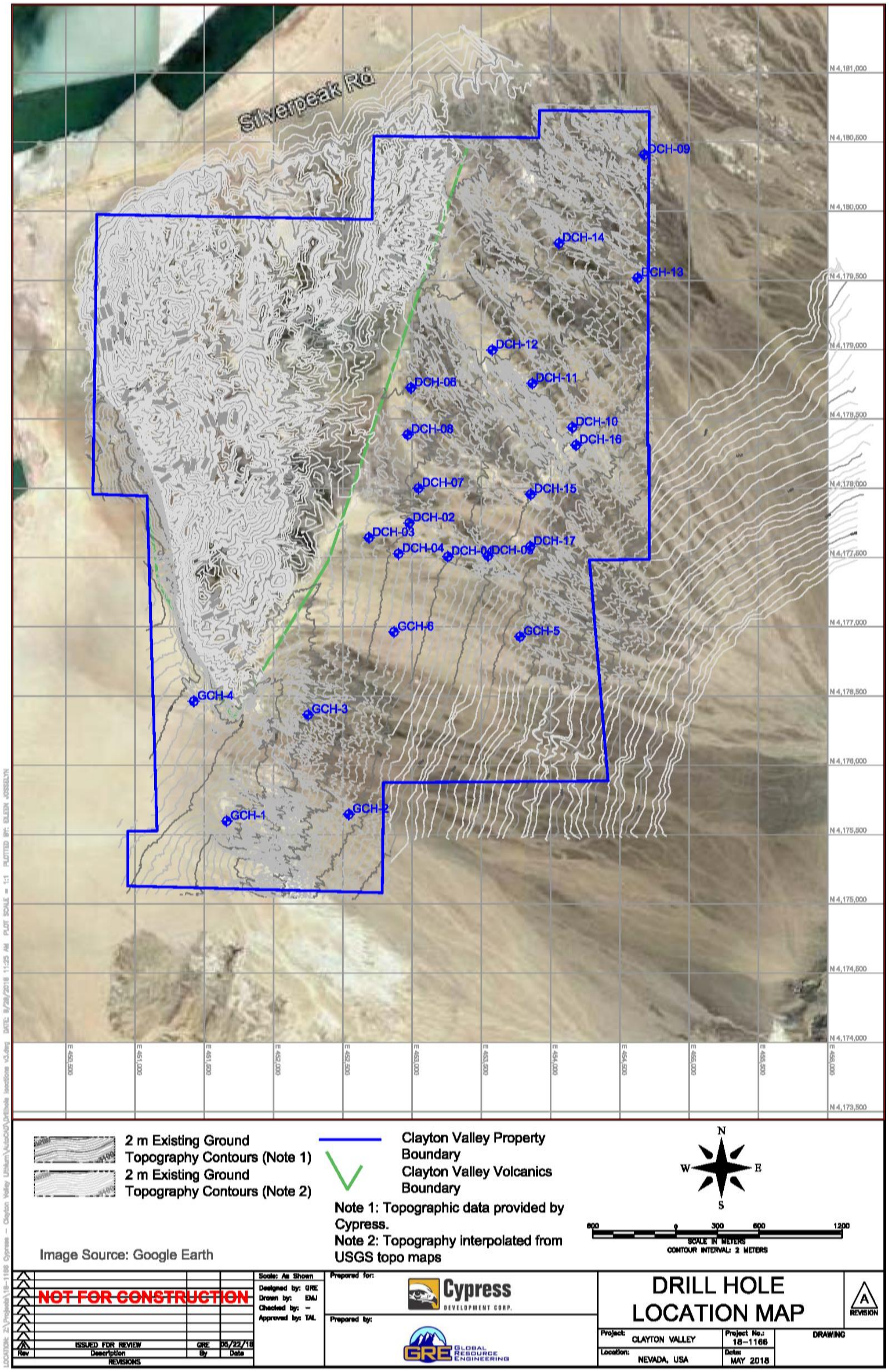


Table 10-1: Clayton Valley Lithium Project Drill Hole Summary

Drill hole ID	Easting	Northing	Elevation	Depth (m)	Azimuth	Dip
DCH-01	4,177,532.44	453,237.16	1,362.07	36.0	0	-90
DCH-02	4,177,756.49	453,060.06	1,355.47	112.2	0	-90
DCH-03	4,177,621.83	452,693.52	1,352.95	76.8	0	-90
DCH-04	4,177,602.95	452,957.86	1,354.87	72.5	0	-90
DCH-05	4,177,475.73	453,583.74	1,366.18	79.9	0	-90
DCH-06	4,178,517.61	452,910.54	1,351.24	49.4	0	-90
DCH-07	4,178,003.29	453,065.24	1,362.15	78.6	0	-90
DCH-08	4,178,312.60	453,010.23	1,354.02	78.6	0	-90
DCH-09	4,180,419.62	454,674.65	1,345.25	106.1	0	-90
DCH-10	4,178,378.40	454,162.54	1,366.54	64.3	0	-90
DCH-11	4,178,663.73	453,915.50	1,353.65	103.0	0	-90
DCH-12	4,178,972.27	453,590.83	1,344.67	66.4	0	-90
DCH-13	4,179,497.61	454,640.67	1,359.41	112.2	0	-90
DCH-14	4,179,743.73	454,066.14	1,341.47	81.7	0	-90
DCH-15	4,177,956.58	453,856.77	1,375.84	127.4	0	-90
DCH-16	4,178,312.14	454,184.29	1,367.52	122.5	0	-90
DCH-17	4,177,579.38	453,852.80	1,380.57	124.4	0	-90
GCH-01	4,175,597.19	451,662.30	1,330.77	32.9	0	-90
GCH-02	4,175,646.24	452,543.58	1,362.20	39.0	0	-90
GCH-03	4,176,365.47	452,249.45	1,345.67	60.4	0	-90
GCH-04	4,176,462.17	451,424.50	1,319.92	51.2	0	-90
GCH-05	4,176,929.28	453,778.86	1,390.20	129.5	0	-90
GCH-06	4,176,962.81	452,869.53	1,359.33	100.0	0	-90

Table 10-2: 2017 Clayton Valley Lithium Project Significant Drill Intervals

Drill Hole ID	Depth (m)		Length (m)	Ave Li (ppm)
	From	To		
DCH-01	4.4	36.0	31.5	1,140
DCH-02	0.5	54.3	53.8	1,036.4
DCH-03	8.5	36.0	27.4	999
DCH-04	1.5	51.2	49.7	1,126.7
DCH-05	8.5	75.6	67.1	1,129.1
DCH-06	14.6	31.4	16.8	1,012.9
DCH-07	32.2	51.2	19.0	974.3
DCH-09	11.3	69.5	58.2	1,092.5
DCH-10	8.5	64.3	55.8	1,107.5
DCH-11	8.2	63.4	55.2	1,208.6
DCH-13	23.8	106.1	82.3	1,221.2
DCH-15	20.1	124.4	104.2	1,106.4
DCH-16	14.6	122.5	107.9	1,198.6
DCH-17	14.6	109.1	94.5	1,049.9
GCH-04	3.7	29.9	26.2	1,076.7
GCH-05	84.7	109.7	25.0	1,017.5
GCH-06	3.0	100.0	96.9	1,141.6

10.2 Drilling Results

Based on drilling exploration to date, shallow (<130 meter) subsurface stratigraphy consists of variably interbedded lakebed deposits of calcareous and ash-rich mudstone and claystone, ashfall tuff, and occasional tuffaceous sandstone, all dipping gently to the east. These sediments are underlain by a distinct, poorly to very well indurated sandstone unit in at least 11 of the 23 drill hole locations. Lithium values in the sandstone are significantly lower than those within the overlying sediments, and this unit represents the "bottom" of drilling exploration carried out to date.

The drilling results generally indicate a particularly favorable section of ash-rich mudstones that extend to depths of up to 100 meters, within which exists a strong, apparently planar, oxidation/reduction front. The color change in freshly drilled core is dramatic, with olive green mudstones changing to blue and black mudstones. The change is sharp, but frequently olive and blue mudstones are interbedded over several meters before continuous blue to blue black mudstones are intersected. Lithium content is often, but not always, slightly higher within the oxidized sediments, though any specific significance of the oxidation horizon regarding lithium mineralization is not yet well understood.

While the drill holes are widely spaced, averaging 650 to 700 meters between holes, the lithium profile with depth is consistent from hole to hole. Unweighted lithium content averages 929.8 ppm for all 665 samples assayed, with an overall range of 116 to 2,240 ppm. Average sample interval length is 2.7 meters (9 feet). Significant drill hole intervals are presented in Table 10-2. The length of the mineralized intervals presented in Table 10-2 should closely represent the true thickness of mineralization, given the apparent tabular (or horizontal) occurrence of the lithium deposit and the very shallow dip of the sedimentary strata.

Cypress reports that core recoveries are generally excellent, and this was verified by visual examination of the core during the site visit. While on site, the QP carefully reviewed the drilling and sampling procedures employed by Cypress with Cypress staff. Based on that review, the QP finds no drilling, sampling, or recovery factors that might materially impact the accuracy or reliability of the drilling results. The QP recommends that Cypress produce annual (or seasonal) exploration reports to describe the drilling and sampling carried out during each given year or drilling campaign. The exploration report should contain adequate detail concerning the drill rig, drilling contractor, number of holes, total meters, recovery rates, drill targets, and rationale for drill hole distribution, etc., to ensure that all pertinent information is captured and catalogued in a practical and efficient manner for ease of future use.

11.0 SAMPLE PRESERVATION, ANALYSES AND SECURITY

11.1 Sample Preparation

Samples collected at the project to date consist of bulk surface samples and NQ-size (1.87-inch diameter) drill core. Drill core samples are collected at the drill rig and placed into plastic-coated cardboard boxes by the drill crew and are transported to the core storage and logging facility in Silver Peak by Cypress personnel. Cypress geologists photograph the core as it is received from the drill rig and collect core recovery information prior to preparing sample intervals for assay. Cypress currently splits and assays 100% of the recovered core. Assay samples, generally 10 feet in length, are split using a meat cleaver. One half of the sampled core is returned to the box for geologic logging, and the other half is bagged and tagged with a blind sample number assigned by Cypress.

Surface samples of outcropping mudstone and soil are collected by Cypress geologists using standard hand tools. These samples typically consist of roughly 5 kg of rock or soil, which is placed directly into a cloth sample bag and marked with a blind sample number.

All core and surface samples are delivered to one of two ISL-certified, independent laboratories, ALS Chemex or Bureau Veritas, both located in Reno, by Cypress personnel. Retained core and samples prepped for shipment are stored in the secure core storage facility in Silver Peak (Figure 11-2).

Photo 11-1: Core Storage

11.2 Analytical Procedures

Samples are crushed, split, and pulverized at the laboratory in preparation for analysis. After pulverizing, two subsamples are selected by the lab for duplicate analysis. While Cypress has submitted at least eight pulp duplicates to a secondary laboratory as check samples, the pulp duplicates are principally used by the primary lab for internal quality control and are not relied on by Cypress to evaluate the overall quality of the sampling program.

Drill core samples are analyzed by 33-element, 4-acid ICP-AES (or ICP-mass spectrometry (MS), depending on the laboratory), and soil and rock chip samples are analyzed by 33-element 4-acid ICP-AES and/or 35-element aqua regia AAS. Select drill core samples have been submitted for iodine by neutron activation analysis (NAA), and a small number of soil samples have been submitted for deionized water leach.

11.3 Quality Assurance and Quality Control

Cypress' in-house Quality Assurance and Quality Control (QA/QC) procedures are currently limited to insertion of a certified standard reference at a rate of one standard sample per 30 core samples. These standards are purchased in durable, pre-sealed aluminum packets. The standard sample assay results are routinely reviewed by Cypress geologists, and to date these results fall within the anticipated range of variability as described by the manufacturer of the standards. The assay results in total, including standard, core, and surface sample data, provide no indication of systematic errors that might be due to sample collection or assay procedures.

11.4 Sample Security

Cypress maintains formal chain-of-custody procedures during all segments of sample transport. Samples prepared for transport to the laboratory are bagged and labeled in a manner which prevents tampering and remain in Cypress's control until released to the laboratory. Upon receipt by the laboratory, samples are tracked by a blind sample number assigned and recorded by Cypress. Retained core and soil samples are securely stored in the core storage facility in Silver Peak, while duplicate and reject materials are stored at the assay laboratory.

11.5 QP Opinion on Adequacy

The QP finds the sample preparation, analytical procedures, and security measures employed by Cypress to be reasonable and adequate to ensure the validity and integrity of the data derived from Cypress' sampling programs to date, with some room for improvement. Based on observations and conversation with Cypress personnel during the QP site visit, and in conjunction with the results of GRE's review and evaluation of Cypress' QA/QC program, the QP makes the following recommendations:

- Formal, written procedures for data collection and handling should be developed and made available to Cypress field personnel. These should include procedures and protocols for field work, geological mapping and logging, database construction, sample chain of custody, and documentation trail. These procedures should also include detailed and specific QA/QC procedures for analytical work, including acceptance/rejection criteria for batches of samples.
- A detailed review of field practices and sample collection procedures should be performed on a regular basis to ensure that the correct procedures and protocols are being followed.
- Review and evaluation of laboratory work should be an on-going process, including occasional visits to the laboratories involved.
- Cypress' existing QA/QC program should be expanded to include at least standards, blanks, and duplicates. All QA/QC control samples sent for analysis should be blind, meaning that the laboratory should not be able to differentiate a check sample from the regular sample stream. The minimum control unit with regard to check sample insertion rate should be the batch of

samples originally sent to the laboratory. Samples should be controlled on a batch by batch basis, and rejection criteria should be enforced. Ideally, assuming a 40-sample batch, the following control samples should be sent to the primary laboratory:

- Two blanks (5% of the total number of samples). Of these, one coarse blank should be inserted for every 4th blank inserted (25% of the total number of blanks inserted)
- Two pulp duplicates (5% of the total number of samples)
- Two coarse duplicates (5% of the total number of samples)
- Two standards appropriate to the expected grade of the batch of samples (5% of the total number of samples).

- For drill hole samples, the control samples sent to a second (check) laboratory should be from pulp duplicates in all cases and should include one blank, two sample pulps, and one standard for every 40-sample batch.
- The purpose of the coarse duplicates is to quantify the variances introduced into the assay grade by errors at different sample preparation stages. Coarse duplicates are inserted into the primary sample stream to provide an estimate of the sum of the assay variance plus the sample preparation variance, up to the primary crushing stage. An alternative to the coarse duplicate is the field duplicate, which in the case of core samples, is a duplicate from the core box (i.e., a quarter core or the other half core). Because sample preparation is currently carried out by the laboratory (and not by Cypress), if coarse duplicates are preferred (to preserve drill core), the coarse duplicates should be sent for preparation and assaying by the second laboratory.
- QA/QC analysis should be conducted on an on-going basis and should include consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.
- In general, AAS should provide better accuracy for Li analysis than ICP-AES, and comparisons should occasionally be performed.

12.0 DATA VERIFICATION

Data verification efforts included an on-site inspection of the project site and core storage facility, check sampling, and manual auditing of the project database.

12.1 Site Inspection

GRE representative and QP J.J. Brown, P.G., conducted an on-site inspection of the project and Silver Peak core storage facility on February 7 and 8, 2018, accompanied by Cypress geology staff Bob Marvin and Daniel Kalmbach and Cypress CEO Bill Willoughby. While on site, Ms. Brown conducted general geologic field reconnaissance, including inspection of surficial geologic features and ground-truthing of reported drill collar and soil sample locations. The majority of the first day of the site visit was spent at the core storage facility in Silver Peak, where drill core samples were visually inspected and duplicate (half-core) samples were selected to submit for check assay.

Field observations during the site visit generally confirm previous reports on the geology of the project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting (as described in Section 7 of this report).

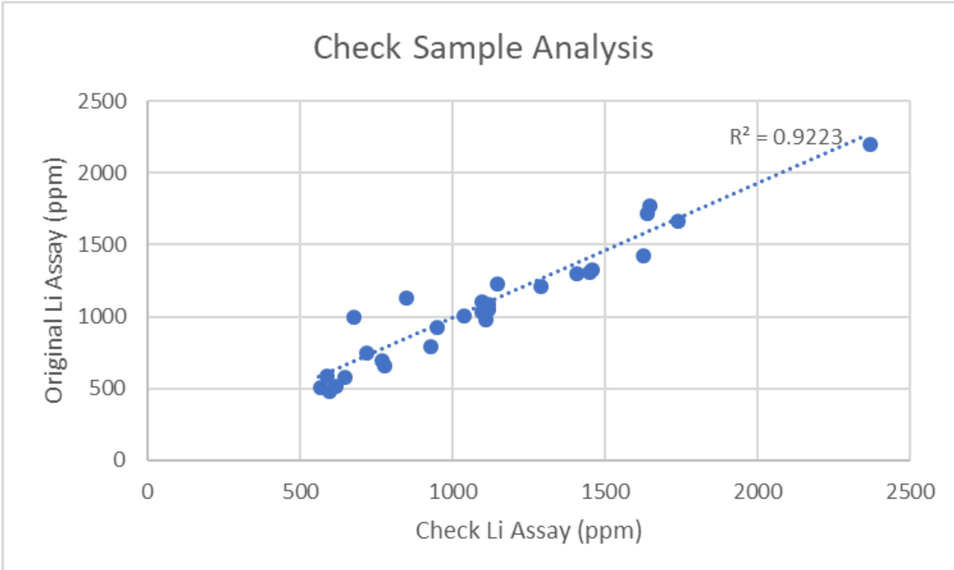
Geographic coordinates for seven of the 14 existing drill hole collar locations were recorded in the field using a hand-held GPS unit. The average variance between field collar coordinates and collar coordinates contained in the project database is roughly 43 meters, which is well outside of the expected margin of error. The drill hole collars are not well marked in the field, and some have no marker at all. The QP recommends that Cypress clearly identify all existing drill holes in the field by installing semi-permanent markers, such as labeled and grouted-in lathe, at each collar location. The existing drill collars should then be professionally surveyed and tied in to the digital topographic surface used for geologic and resource modeling. Future drill holes can be located using survey-grade GPS instrumentation, provided that the GPS coordinates are reasonably similar to those reported for the same locations within the digital topographic surface.

12.2 Check Sampling

During the site visit, 26 core sample intervals from eight separate drill holes were selected for visual inspection and check sampling based on a review of the drill hole logs and original assay results. The sample intervals selected were gradational regarding both assay value and oxidation (i.e., high, moderate, and low original assay values; and above, within, and below the apparent oxidation horizons). Without exception, the core samples inspected accurately reflect the lithologies and sample descriptions recorded on the associated drill hole logs and within the project database.

A total of 29 check samples (26 core intervals and three surface samples) were delivered to ALS (Elko) for analysis using the same sample preparation and analytical procedures as were used for the original samples. A comparison of the original versus check assay values for 24 of the 26 samples shows good correlation between the results, with an R^2 of 0.9223 (Figure 12-1). Two samples were removed from the sample population: one core sample based on a discrepancy in sample length, and one surface sample for which an original assay value was unavailable.

Figure 12-1: Check Sample Analysis



12.3 Database Audit

The author completed a manual audit of the digital project database by comparing original assay certificates and drill hole logs to corresponding information contained in the database. The manual audit revealed no discrepancies between the hard-copy information and digital data, and only a single data entry error. The data entry error was easily rectified, and the overall error rate is considered negligible regarding potential impact to the mineral resource estimate and PEA.

12.4 QP Opinion on Adequacy

Based on the results of the QP's check sampling effort, visual examination of selected core intervals, and the results of the database audit, the QP considers the lithology and assay data contained in the project database to be reasonably accurate and suitable for use in estimating mineral resources and reserves.

Results of the comparison between field and database collar coordinates indicates that additional or improved ground survey may be necessary to increase confidence in the accuracy of the drill hole collar data contained within the database. The QP recommends that Cypress clearly identify all existing drill holes in the field. The existing drill collars should then be professionally re-surveyed and tied in to the digital topographic surface used for geologic and resource modelling.

The database audit work completed to date indicates that occasional inconsistencies and/or erroneous entries are likely inherent or inevitable in the data entry process. The QP recommends that Cypress establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Lithium can occur in a wide variety of lithium bearing deposits, including brines, pegmatites, hectorite clays, and claystones. The pegmatite deposits host the lithium-bearing mineral spodumene, while the lithium in clay or claystone deposits may be contained in the minerals illite, smectite, hectorite, and lipidiolite. The optimum extraction method depends heavily on the lithium mineral associations. The Clayton Valley project is a claystone hosted lithium that is amenable to a conventional dilute sulfuric acid leach followed by solution purification to produce a high-grade final lithium product. The selection of the final product pathway is dependent on the intended market, with lithium carbonate and lithium hydroxide being the two most common product classes, and with lithium carbonate typically being the easiest to produce.

This report includes metallurgical test work conducted by or under the direction of SGS (Lakefield, Ontario), Continental Metallurgical Services (CMS) (Butte, Montana), and Hazen (Golden, Colorado) from 2017 to 2018.

For this report, GRE and CMS reviewed the metallurgical test work with the goal of developing a viable process flowsheet for the production of lithium carbonate.

13.1.1 Metallurgical Reports

Project metallurgical test work consisted of the following:

- Eagle Engineering – ASICS mineralogical study on oxide and reduced claystone material.
- ALS Metallurgical Laboratories – Kamloops – crushing work index and abrasion testing.
- CMS – physical property tests, agitated leaching tests, and development of proprietary leaching techniques. Core samples from DCH-2. Diagnostic leaching tests on 29 samples from other core holes and surface samples. Technology development and review, including membrane, precipitation, ion exchange (IX) recovery and work by 3rd party vendors.
- ALS – analytical work for CMS including rare earth metal analyses.
- SGS – minerology, agitated leaching tests on surface and core samples with different acids, temperatures, acid concentrations, solids. Core samples from hole DCH-5.
- Hazen – agitated leaching tests on core samples with different temperatures and concentrations using sulfuric acid. Core samples from hole DCH-16. Rare earth element analyses.

13.1.2 Sample Selection

Samples for metallurgical testing were collected from surface and core samples as follows:

- A set of surface samples in 2017, that were collected from the Glory property in 2017 by the Pure Energy/Cypress JV and tested in a series of leach scoping tests by SGS. The samples were designated NVBL-002 through NVBL-007.

- A set of composite core samples tested by SGS, CMS and Hazen as shown in Table 13-1. The samples were from three core holes, DCH-5, DCH-2 and DCH-16, and divided into two composites to each respective laboratory. The composites were designated Oxide and Reduced.
- Diagnostic leach samples at CMS were conducted on the 29 check-assay samples taken by GRE, as described in Section 12-2. The sample material tested was obtained from the assay rejects provided by ALS. The samples consist of 26 core samples from various intervals across the property and 3 surface samples.
- ALS Kamloops was provided material from selected intervals in DCH-10.

Table 13-1: Leach Test Samples

Drill Hole ID	Li Grade (ppm)	Laboratory
DCH-5 Oxide	900	SGS
DCH-5 Reduced	1,100	SGS
DHC-2 Oxide	810	CMS
DHC-2 Reduced	720	CMS
DCH-16 Oxide	1,020	Hazen
DCH-16 Reduced	620	Hazen

The designation of Oxide and Reduced material occurred prior to the resource classification of GRE into various claystone/mudstone units. In general, the composites were obtained from intervals of 0-75 feet down-hole for Oxide and 75 feet to the end of hole for Reduced, varying based upon the location of the redox boundary within each hole. For the diagnostic leach tests, samples were identified by one of four lithologic units, Upper Olive, Main Blue, Lower Olive and Hard Bottom. In general, by depth of sample, Oxide material corresponds to the Upper Olive, and Reduced material corresponds to the latter three units.

13.2 Mineralogy

Two separate x-ray diffraction samples were sent in for mineral identification.

The first review was completed in Sept 2017 by SGS. The first sample was sent to SGS for clay mineral identification. The SGS testing was completed on the following material: CYPDEV Master DDCV-137, 144, 153,158,241,247,253, and 254.

SGS reported that a majority of the material was an illite - montmorillonite mixture, with silicates in general being a majority of the material. The only lithium bearing material found in the review was lepidolite – K(Li,Al)3Si4O10(OH)2.

The second test was completed by Eagle Engineering in March 2018. The second sample was sent to Eagle Engineering for AMICS Review and mineral identification. Eagle Engineering testing was completed on the following material: DHC-2 Oxide and Reduced Material.

Eagle Engineering identified the following:

"For lithium oxide sample, AMICS analysis was completed and according to the data, two major phases were identified, illite and smectite. Minor phases include quartz, biotite, wollastonite, chlorite, orthoclase, oligoclase, calcite, dolomite, ilmenite and apatite. Two lithium phases, glaucophane at 4.37% and lepidolite at 2.29%, were identified."

"For lithium reduced sample, AMICS analysis was completed and according to the data, two major phases were identified, illite and smectite. Minor phases include quartz, biotite, wollastonite, chlorite, orthoclase, oligoclase, calcite, dolomite, ilmenite and apatite. Two lithium phases, glaucophane at 4.08% and lepidolite at 1.49%, were identified."

Table 13-2 provides the total oxide sample results. Table 13-3 provides the total reduced sample results.

Table 13-2 Oxide Mineral AMICS Results

Mineral	Chemistry	Percentage
Illite	$K(Al,Mg,Fe)Si_4O_{10}$	60.91
Smectite	$(Na,Ca)_3(Al,Mg)_2Si_4O_{10}(OH)_2$	17.42
Quartz	SiO	4.68
Glaucophane	$LiNa_2Mg_3Al_2Si_8O_{22}(OH)_2$	4.37
Biotite	$K(Mg,Fe)_3AlSi_3O_{10}(OH)_2$	4.11
Omphacite	$(Ca,Na,Mg,Fe_{2,}Al)Si_2O_6$	3.05
Lepidolite	$K(Li,Al)_3Si_4O_{10}(OH)_2$	2.29
Wollastonite	$CaSiO_3$	1.64
Chlorite	$(Fe,Mg,Al)_6Si_4O_{10}(OH)_8$	0.83
Orthoclase	$KAlSi_3O_8$	0.22
Oligoclase	$(Na,Ca)AlSi_3O_8$	0.20
Calcite	$CaCO_3$	0.17
Dolomite	$Ca,Mg(CO_3)_2$	0.10
Illmenite	$FeTiO_3$	0.03
Apatite	$Ca(PO_4)_3OH$	< 0.01

Table 13-3 Reduced Mineral AMICS Results

Mineral	Chemistry	Percentage
Illite	$K(Al,Mg,Fe)Si_4O_{10}$	57.07
Smectite	$(Na,Ca)_3(Al,Mg)_2Si_4O_{10}(OH)_2$	25.94
Glaucophane	$LiNa_2Mg_3Al_2Si_8O_{22}(OH)_2$	4.08
Biotite	$K(Mg,Fe)_3AlSi_3O_{10}(OH)_2$	3.74
Omphacite	$(Ca,Na,Mg,Fe_{2,}Al)Si_2O_6$	3.30
Oligoclase	$(Na,Ca)AlSi_3O_8$	1.62
Lepidolite	$K(Li,Al)_3Si_4O_{10}(OH)_2$	1.49
Wollastonite	$CaSiO_3$	1.23
Quartz	SiO	1.19
Calcite	$CaCO_3$	0.21
Chlorite	$(Fe,Mg,Al)_6Si_4O_{10}(OH)_8$	0.03
Illmenite	$FeTiO_3$	< 0.01
Dolomite	$Ca,Mg(CO_3)_2$	< 0.01

Lithium-bearing minerals included glaucophane and lepidolite. Also noted was significantly higher amounts of illite clay compared to smectite.

13.3 Physical Property Testing

13.3.1 Crusher Work Index and Abrasion Index

Core samples were taken from DCH-10 core to complete Crusher Work Index and Abrasion Index testing. The Crusher Work Index or Bond Impact Work Index is identified to calculate the net power requirements for crushing equipment. The Bond Abrasion Test determines the Abrasion Index, which is used to determine steel media and liner wear in crushers, mills, and other abrasive areas. There are correlations of wear rate developed in kilograms/kWhr of energy used for different comminution processes.

Most of the claystones are friable and break very easily. Figure 13-1 shows hole DCH-10 and the friability of the core.

Figure 13-1 Picture of DCH-10 Core

Crusher Work Index testing and Abrasion Index was completed by ALS Kamloops. ALS uses a Bond Low Energy Impact test for the crusher work index and a Bond Abrasion Testing apparatus for the Abrasion Index.

The following are the results of the test work:

- Crusher Work Index – 2.5 kWh/tonne
- Abrasion Index – 0.0001

Both the work index and abrasion index are very low.

13.3.2 Grind Work Index

A single scoping level comparative grind test was completed using a 1,000-gram charge of oxide material in a 6-inch diameter rod mill. Grinding was completed using a standard 6-inch diameter by 10-inch tall rod mill. The rods were filled to 35% volumetric fraction with a mixture of large and small rods. The sample was 100% minus 10 mesh in 15-30 seconds of grinding. Based on comparative experience, this equates to a work index less than 2 kWhr/tonne.

13.3.3 Material Competence

Particles of up to ½ inch diameter were combined with water at various pulp densities to define the ease with which the minerals were separated into core particles. The majority of the particles readily decomposed in water under agitation in under to 5 minutes. Small particles (<10 mesh) broke down into their core grains in less than one minute in water under agitation. Size analysis was completed using a standard 25, 100, 200, 270 mesh sieve and dry screening, and demonstrated the material can be readily reduced to grain-size particles with water addition and agitation.

13.3.4 Density Determination

Density was completed by using the "Instantaneous Water Immersion" density determination. The instantaneous water immersion method results in a bulk density measurement. The volume of the sample is calculated comparing the difference between the submerged weight of the sample and the dry weight of the sample. The most common fluid to suspend the sample is water; however, any incompressible fluid with a known density can be used. Water is typically used because the specific gravity of water at room temperature is 1.0. The density was completed on both a dry and in-situ oxidized and reduced material. The densities of the material were identified as follows:

The in-situ density:

- Reduced Material – 1.68 gm/cc (Range 1.49-1.93) - US Units (104.9 #/ft3) (Range 93.0-120.5)
- Oxide Material – 1.76 gm/cc (Range 1.58-1.90) – US Units (109.9 #/ft3) (Range 98.6-118.6)

The dry density:

- Reduced Material – 1.60 gm/cc (Range 1.42-1.84) – US Units (99.9 #/ft3) Range (88.7-114.9)
- Oxide Material – 1.69 gm/cc (Range 1.53-1.83) – US Units (105.5 #/ft3) (Range 95.5-114.2)

13.3.5 Other Physical Property Tests

Other physical tests performed by CMS included flotation, desliming, leach solution viscosity, filtration and settling tests.

All settling tests performed showed no issues with filtration when under acidic conditions.

13.4 Leach Extraction Tests

Leaching is the primary processing step to remove lithium from the claystone. Scoping tests were performed and concluded the most effective means of leaching material from the project is by using dilute sulfuric acid at elevated temperatures. Leaching conditions impact the overall processing recovery and cost. Significant work was performed to identify leaching conditions on composite samples from three representative drill holes. A diagnostic leach test program was then conducted to extrapolate leach conditions and results across the deposit.

13.4.1 Surface Samples

Initial testing was conducted by SGS in 2017 on a set of surface samples. The tests were conducted in an agitated leach condition using either distilled water, dilute sulfuric acid, and a saline brine. Samples were leached at room temperature over 4-hour periods in solution mixtures of 10% solids.

The tests are summarized in Table 13-5. Samples leached with distilled water-only yielded lithium extractions from a low of 1% to high of 56%. Lithium extractions improved to a range of 42% to 72% when the tests were repeated with dilute sulfuric acid. Leaching in saline brine had no discernable effect.

The results indicated: 1) a portion of the lithium in some surface samples is water soluble, and 2) dilute sulfuric acid is a potential lixiviant. Acid consumptions were not determined in the tests.

Table 13-4 SGS Scoping Leach Tests

	Li Extraction %		
Sample ID	DI Water	5% H_2SO_4	Brine
NVBL-002	8	30	6
NVBL-003	48	72	56
NVBL-004	14	57	16
NVBL-005	1	42	1
NVBL-007	56	71	45

Source: SGS Report.

13.4.2 Core Samples – SGS

Subsequent tests in 2017-2018 were done at SGS on core from hole DCH-2. These tests were completed by SGS at different temperatures, lixiviants, and pH in agitated conditions over 4-hour time period. The results are summarized in Table 13-5.

Unlike the surface samples previously tested by SGS, the two composite core samples resulted in zero lithium extractions when leached with distilled water-only. This, along with sodium levels in the surface samples, indicate that the surface samples have some component of surface enrichment in the form of water-soluble salts, and this enrichment is absent in the subsurface clays.

As with the surface samples, leaching the two composite core samples with dilute sulfuric acid at room temperature yielded positive results, with 40.5% extraction on the Oxide sample and 57.8% extraction on the Reduced sample. Heating the mixtures to 80C increased the lithium extractions further to 76% on the Oxidized sample and 76.5% on the Reduced sample. Increasing the acid concentration to 10% was tested on the Reduced sample and increased the extraction further to 83.5%.

Leaching was tested with other lixiviants: ammonium sulfate, acetic acid, hydrochloric acid and nitric acid. Of these, only heated solutions of ammonium sulfate and hydrochloric acid were effective at extracting lithium, with 5% HCl yielding 77.7% and 1.5M ammonium sulfate yielding 48%.

Table 13-5 SGS Leaching Test Work

Test ID	Sample ID	Lixiviant	Temp (°C)	Acid Cons (kg/t)	Extractions (%)			Solutions Tenors (mg/L)						
					Li	Ca	Mg	Li	Al	Fe	Mg	Ca	Na	K
OL-01	Oxidized	DI Water	room	-	0.0	0.0	0.0	< 2	0.6	0.4	0.9	2.1	78	8.0
OL-02	**Oxidized**	**5% H_2SO_4**	**room**	**142.1**	**40.5**	**15.5**	**35.6**	**30**	**518**	**335**	**713**	**716**	**423**	**550**
OL-03	Oxidized	2% $(NH_4)_2SO_4$, pH 4	room	86.4	5.5	17.1	8.4	3	< 0.9	0.2	120	531	410	605
OL-04	Oxidized	2% $(NH_4)_2SO_4$, 7% NaCl, pH 4	room	-	4.8	23.1	7.4	3	< 0.9	0.2	124	872	30,700	732
OL-05R	Oxidized	1.5M $(NH_4)_2SO_4$, pH 2	room	94.8	8.8	24.3	10.7	6	38	46.8	194	984	454	854
OL-06	**Oxidized**	**5% H_2SO_4**	**80**	**176.9**	**76.0**	**15.8**	**67.9**	**67**	**1,150**	**1,090**	**1,480**	**633**	**449**	**954**
RL-01	Reduced	DI Water	room	-	0.0	0.0	0.1	< 2	4.1	3.1	3.3	1.7	102	13
RL-02	**Reduced**	**5% H_2SO_4**	**room**	**124.0**	**57.8**	**18.5**	**48.3**	**68**	**388**	**807**	**1,100**	**719**	**358**	**403**
RL-03	Reduced	2% $(NH_4)_2SO_4$, pH 4	room	87.7	9.0	18.9	7.2	7	< 0.9	1	113	519	345	240
RL-04	Reduced	2% $(NH_4)_2SO_4$, 7% NaCl, pH 4	room	88.3	8.3	26.1	6.4	7	< 0.9	2	109	805	29,300	250
RL-05R	Reduced	1.5M $(NH_4)_2SO_4$, pH 2	room	15.8	12.8	26.9	11.6	13	47	210	229	964	362	279
RL-06	**Reduced**	**5% H_2SO_4**	**80**	**171.2**	**76.5**	**18.1**	**68.0**	**99**	**797**	**1,460**	**1,660**	**647**	**370**	**734**
RL-07	Reduced	1.5M (NH4)2SO4, pH 2	80	117.3	48.0	33.8	42.6	47	220	662	795	1,040	385	417
RL-08 *	**Reduced**	**5% H_2SO_4**	**80**	**155.4**	**56.0**	**8.4**	**50.5**	**146**	**986**	**1,850**	**2,500**	**648**	**788**	**874**
RL-09	Reduced	5% CH_3COOH	room	76.3	6.1	85.5	5.3	7	7	8	117	3,760	324	105
RL-10	Reduced	5% CH_3COOH	80	96.9	8.6	85.2	6.7	10	6	42	159	3,930	309	148
RL-11	**Reduced**	**10% H_2SO_4**	**50**	**-85.3**	**83.5**	**29.0**	**71.6**	**132**	**857**	**1,630**	**1,800**	**1,100**	**370**	**811**
RL-12	Reduced	5% HCl	50	45.2	77.7	90.4	68.6	100	779	1,480	1,660	3,690	367	738
RL-13	Reduced	5% HNO_3	50	136.3	3.7	71.2	3.1	< 2	768	2,190	39	1,000	2730	< 10

* RL-08 conducted at 20% solids, all other tests at 10% solids.

13.4.3 Core Samples - CMS

CMS conducted leaching tests on two composite samples of drill core from DCH-5, as summarized in Table 13-7. Analytical work on solids and liquid leach materials from the CMS tests was performed by ALS.

As in the previous test work by SGS, CMS found that leaching with dilute sulfuric acid yielded the highest extractions of lithium. Other lixiviants, nitric acid, hydrochloric acid, acetic acid and aqua regia, were also tested.

The following tests were completed towards further improvement in lithium extraction:

- Sodium Hydroxide Pretreatment ("Crack") followed by Sulfuric Acid Leaching (Tests Li-A3, Li-O2) – The Reduced sample material was leached using a sodium hydroxide pretreatment followed by normal sulfuric acid leaching. Cracking is used in many operations to open the mineral or material to allowing for better leaching. Recoveries with the sodium hydroxide crack improved to over 70% for the base leaching parameters.
- Pug Mill Leaching – Pug mill leach testing was completed using parameters similar to those used by the US Bureau of Mines in the early 1970s on McDermitt claystones (Crocker, Lien, and others, 1988). The leaching kinetics were slow at ambient temperature but produced higher lithium leach grades (Tests Li-P1, Li-P2)
- Split Leaching Tests – These tests (Li-B1 – B3) split the acid addition into two equal quantities and each quantity was added on 30-minute intervals. The test was done to see if the first acid addition would open the material and the second acid would leach the material. The recoveries were identified in the low 40% range.
- Sodium Hydroxide Leaching – This test (Li-OH) was completed to see if a base could leach the material with significantly less contaminates in the leach solutions. Minimal or no recovery of lithium was identified.
- Varying time and temperature – These tests (Li-O2 -O3 series) varied time frame and temperatures to identify leaching kinetics and optimize leach conditions. Lithium extraction improved with increased time. Most leaching is effectively completed in 1.5 hours with a slow increase thereafter through 3 hours. Tests at room temperature to 75^{o} C showed an increase in lithium extraction with elevated temperature.

Table 13-6 CMS Leaching Tests

Test	Sample ID	Lixiviants	Time (min)	Solids (%)	Temp (C)	Li Grade (ppm)	Extraction Li (%)	Mg Grade (ppm)	Ca Grade (ppm)	K Grade (ppm)
Li-A1	**Reduced**	**Sulfuric**	**60**	**17%**	**50**	**120**	**64**	**2740**	**560**	**800**
Li-A2	**Oxide**	**Sulfuric**	**60**	**17%**	**50**	**130**	**62**	**2710**	**630**	**679**
Li-A3	**Reduced**	**NaOH, sulfuric**	**60**	**17%**	**50**	**120**	**71**	**2920**	**580**	**879**
Li-A4	Reduced	Nitric	60	17%	50	70	35	1470	8050	700
Li-A5	Oxide	Nitric	60	17%	50	70	35	1320	8320	467
Li-A6	Reduced	Hydrochloric	60	17%	50	40	20	789	8190	381
Li-A7	Oxide	Hydrochloric	60	17%	50	40	23	789	5110	381
Li-A8	Reduced	Acetic acid	60	17%	50	20	9	172	7720	161
Li-A9	Reduced	Aqua regia	60	17%	50	70	37	1500	7880	600
Li-A10	Oxide	Aqua regia	60	17%	50	60	34	1255	630	416
Li-P1	Oxide	Sulfuric	60	36%	room	180	24	4040	660	900
Li-P2	Reduced	Sulfuric	60	36%	room	220	32	5420	640	1600
Li-B1	Reduced	Sulfuric	30	17%	50	110	48	2660	640	900
Li-B2	Reduced	Sulfuric	90	17%	50	110	63	3260	640	1000
Li-B3	Reduced	Sulfuric	60	17%	50	91	41	2160	660	700
Li-OH	Reduced	NaOH	60	25%	80	1	0	1	8	500
LiO2-C	Reduced	Sulfuric	120	17%	50	129	67	3342	597	930
LiO2-D	**Reduced**	**Sulfuric**	**240**	**17%**	**50**	**197**	**83**	**5431**	**931**	**900**
LiO2-E	Reduced	Sulfuric	60	17%	75	201	79	5575	793	1021
LiO2-F	Reduced	Sulfuric	60	17%	63	167	76	4617	725	785
LiO2-G	Reduced	Sulfuric	60	25%	50	165	68	3565	830	830
LiO2-H	Reduced	Sulfuric	60	30%	50	267	74	7410	742	1217
LiO2-I	Reduced	NaOH, sulfuric	120	17%	50	160	77	4368	883	759
LiO3-A	Oxide	Sulfuric	60	17%	75	201	66	5575	793	1021

Sulfuric acid concentration at 5% by weight in all tests. Other acids at 10% by weight.

13.4.4 Core Samples - Hazen

Hazen conducted additional leach testing to further define the leaching kinetics.

The head assay split samples were analyzed for the following:

- Lithium, sodium, potassium, calcium, magnesium, aluminum, and silicon by flame atomic absorption spectroscopy
- Chloride by titration
- Fluoride by ion selective electrode
- Specific gravity by nitrogen pycnometry
- Carbon, acid insoluble carbon, and sulfur by LECO combustion analysis
- Rare earth analysis by ICP

A summary of the head sample analyses is shown in Table 13-7.

Table 13-7 Hazen Head Sample Assays

HRI Number	Sample ID	Density (g/cm3)	Assays (wt%)											
			Li	Na	K	Ca	Mg	Al	Si	Cl	F	C	Acid Insoluble C	S
54985-01	Reduced	2.841	0.102	0.934	5.49	3.80	2.31	7.25	24.1	0.009	0.271	1.29	0.170	0.407
54986-01	Oxide	2.620	0.062	1.22	4.06	4.73	1.72	6.39	21.9	0.036	0.178	1.29	0.064	0.170

Leaching experiments were performed using feed samples that had been stage crushed to a P_{100} of 1.7 mm, blended, and riffle split into 1 kg charges. Leaching experiments were conducted in a resin kettle equipped with a lid, overhead mixer, baffle cage insert, pH probe, thermometer, and water-cooled condenser. Heat was provided with an electric heating mantle. Leaching was performed using a 5% H_2SO_4 solution (by weight). Tests were performed using the Reduced sample with a two hour residence time, at 10% solids and temperatures of 25, 50, and 75°C. A kinetic experiment was also performed with the Reduced sample with a four hour residence time, at 5% solids and 50°C. Samples were taken at 15, 30, 45, 60, 90, 120, 180, 240, 360, and 480 min. A kinetic experiment was performed using the Oxide sample with a four-hour residence time, at 10% solids and 75°C. Kinetic samples were taken at the same intervals as during the Lower Reduced Zone experiment. Table 13-8 summarizes the leaching experiment conditions and results.

Table 13-8 Summary of Leaching Experiments

HRI Number	Sample ID	Temp (°C)	Time (min)	Kinetic Samples, yes or no	Solids Concentration (wt%)	Lithium Extraction[a] (%)	H_2SO_4 Consumption (kg/t ore)
54985-01	Reduced	25	240	No	10	39	166
54985-01	Reduced	50	240	No	10	63	204
54985-01	Reduced	75	240	No	10	74	238
54985-01	Reduced	50	480	Yes	5	73	222
54986-01	Oxide	75	480	Yes	10	78	271

[a] Lithium extraction was determined by calculated head.

Lithium extraction was observed to increase with increasing temperature, as shown in Figure 13-2. The data indicated that increasing the leaching temperature to greater than 75°C would result in improved lithium extraction. Figure 13-3 shows the extraction of impurities with respect to temperature in the Reduced sample. The extraction of sodium, potassium, calcium, and aluminum impurities did not appear to be influenced by temperature to the same degree as lithium extraction. The magnesium extraction showed a similar relationship to temperature as lithium.

The effect of reaction time on lithium extraction from the Reduced and Oxide samples is shown in Figure 13-4. In both ore samples, lithium extraction increased sharply over time in the first 120 min and then increased more slowly between 120 and 480 min. The lithium extraction slowly increased at 480 min, indicating that slight increases in lithium extraction would be observed with longer leach residence times. The effects of leach residence time on impurity extraction from the Reduced and Oxide samples are shown in Figure 13-5 and Figure 13-6, respectively. In both samples, aluminum and magnesium extractions increased over time, whereas sodium and potassium extractions were not affected by reaction time. The calcium extraction decreased with increasing reaction time, likely due to the precipitation of gypsum during the reaction.

Figure 13-2 Temperature v. Li Extraction (240 min, 10% Solids, 5% H_2SO_4)

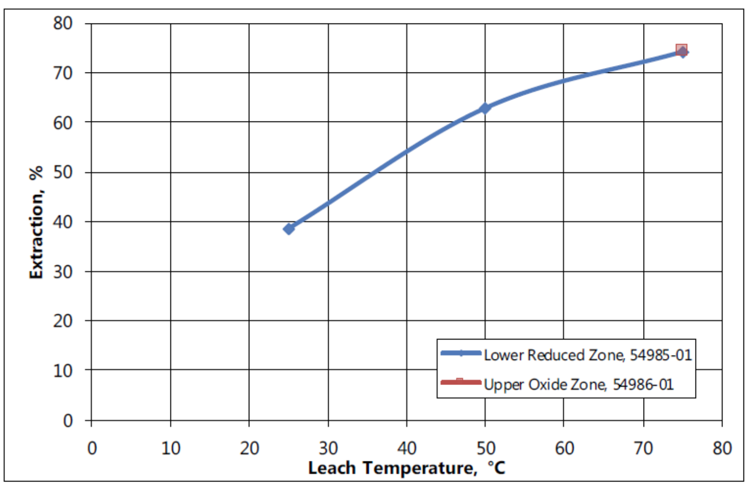


Figure 13-3 Effect of Temperature on Impurity Extraction, Reduced Sample (HRI 54985-01) (120 min, 10% Solids, 5% H_2SO_4)

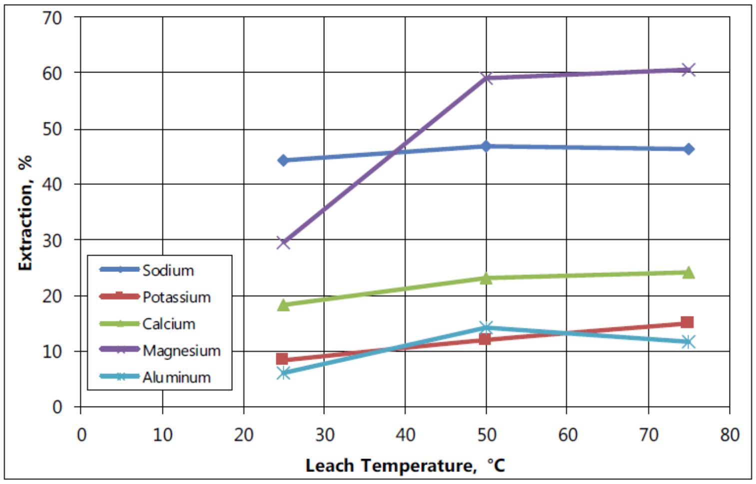


Figure 13-4 Effect of Leach Time on Lithium Extraction

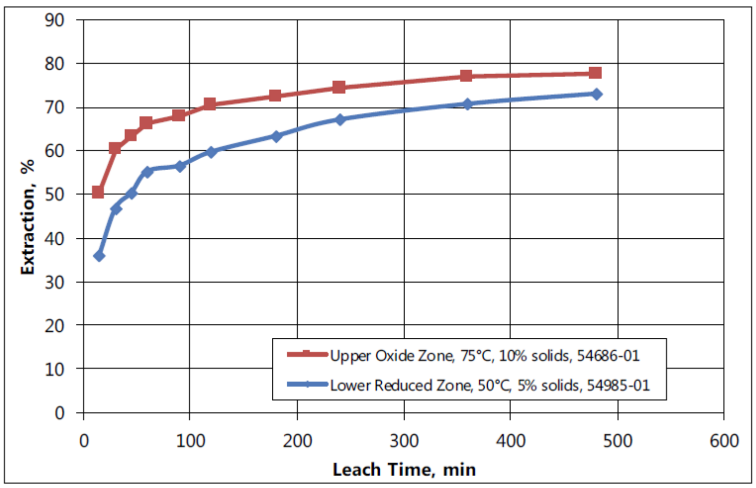


Figure 13-5 Effect of Leach Time on Impurity Extraction, Reduced Sample (HRI 54985-01) (50°C, 5% Solids, 5% H_2SO_4)

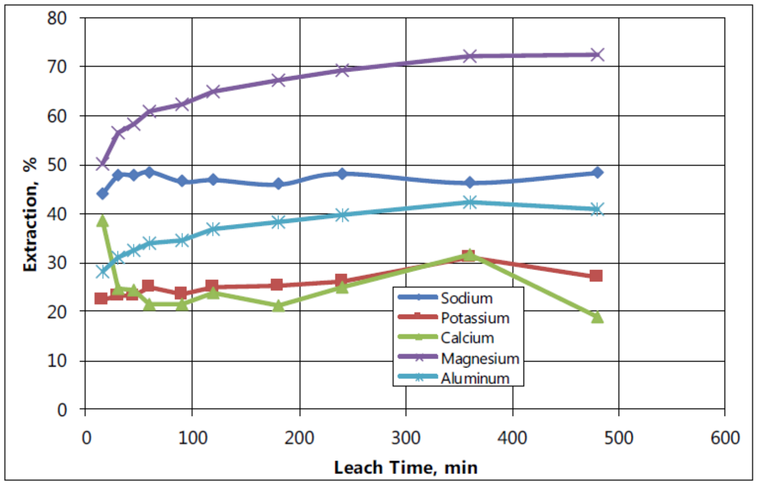
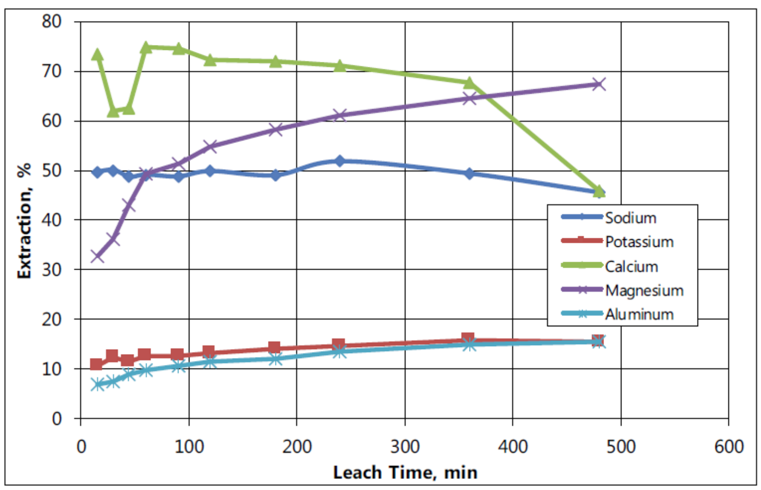


Figure 13-6 Effect of Leach Time on Impurity Extraction, Oxide Sample (HRI 54986-01) (75°C, 10% Solids, 5% H_2SO_4)



13.4.5 CMS Diagnostic 1-hour Leach Tests

CMS developed a test procedure to quickly examine leach characteristics of samples from the property. The test procedure consisted of the following steps:

- Sample is pulverized to 100% passing 20 mesh. Total lithium is determined on a sample split using four-acid digestion followed by ICP/AAS analysis.
- A 100-gram split is then leached for one hour in 5% sulfuric acid solution at 50C.
- Leached sample is filtered, washed, dried, and weighed. Content of lithium tails is determined using four-acid digestion followed by ICP/AAS analysis and leach solution is assayed using ICP/AAS analysis.

A set of diagnostic tests were conducted on the 29 samples collected by GRE for check-assays earlier in 2018 (see Section 12.2). These included 26 core samples at varying intervals from 8 holes, as listed in Table 13-8, as well as 3 surface samples.

The tests were devised as a diagnostic test to indicate the leach response within the first hour of leaching. The extractions percentages listed are not projections of ultimate lithium extractions. Calculated head grades (solution assays plus leached tail residue assays) from the tests corresponded well with the original and check assays obtained by GRE (average 1004 ppm Li for CMS calculated head versus 1025 ppm Li for GRE check assays).

Conclusions from the diagnostic tests are:

- None of the core samples tested were indicative of refractory clay, i.e. hectorite, for which a lithium extraction of less than 5% would be expected.
- All samples yielded greater than 31% lithium extractions and are consistent with the time - extraction date for the first hour of leaching.
- Lithium extractions appear to increase with depth, which may indicate faster leaching and shorter residence times for deeper material.
- Average lithium grades and extractions for the respective resource units in the 26 samples are:
    - Upper Olive 7 samples 877 ppm Li 39.3%
    - Main Blue 13 samples 1162 ppm Li 46.9%
    - Lower Olive 2 samples 1125 ppm Li 52.0%
    - Hard Bottom 4 samples 667 ppm Li 59.2%

  (Note: no samples of Upper Tuff were included in the GRE check-assay.)

Table 13-9 Diagnostic 1-hour Test Results

Drillhole ID	Interval (ft)	Lith Unit	Calc Li Head PPM	Extraction %
DCH-03	138-148	Main Blue	680	51.8%
DCH-04	21-28*	Upper Olive	680	55.7%
DCH-04	91.25-98	Main Blue	1000	57.2%
DCH-04	148-158	Main Blue	1060	48.9%
DCH-06	95-103	Hard Bottom	840	69.5%
DCH-07	105.8-112	Main Blue	1010	47.9%
DCH-07	158-168	Main Blue	1050	50.5%
DCH-09	57-67	Upper Olive	1180	36.0%
DCH-09	77-88	Upper Olive	720	43.8%
DCH-09	88-98	Main Blue	1100	42.2%
DCH-09	128-135	Main Blue	1590	49.9%
DCH-09	198-208	Lower Olive	1190	55.5%
DCH-09	248-258	Hard Bottom	550	45.8%

Drillhole ID	Interval (ft)	Lith Unit	Calc Li Head PPM	Extraction %
DCH-09	338-348	Hard Bottom	580	57.2%
DCH-11	48-58	Upper Olive	1260	34.1%
DCH-11	78-89	Main Blue	1301	42.6%
DCH-11	116-119	Main Blue	1690	40.8%
DCH-11	148-153	Main Blue	710	50.7%
DCH-13	28-38	Upper Olive	940	34.5%
DCH-13	68-78	Upper Olive	800	39.4%
DCH-13	78-88	Main Blue	1380	49.2%
DCH-13	148-158	Main Blue	1630	31.0%
DCH-14	28-38	Upper Olive	560	31.4%
DCH-14	78-88	Main Blue	910	47.1%
DCH-14	138-148	Lower Olive	1060	48.5%
DCH-14	258-268	Hard Bottom	630	64.5%
Avg		**26 samples**	**1004**	**47.1%**

The three surface samples collected by GRE were also tested using the diagnostic procedure for sulfuric acid, and were also tested in a 1-hour leach at 50C using distilled water-only. The results are shown in Table 13-8 and are consistent with those seen in Section 13.4.1 which showed the presence of water-soluble (surface-enriched?) lithium and increased extraction when leached with dilute sulfuric acid.

Table 13-X Diagnostic 1-hour Tests on Surface Samples

Sample ID	Calc Li Head PPM	Li Extraction %	
		DI Water	5% H_2SO_4
DN-27	1480	18	51
DN-28	2180	25	55
DN-29	2230	39	73

13.4.6 Lithium Extraction Plots

Basic lithium extraction results were plotted for all data from the tests using sulfuric acid on the composite core samples from SGS, CMS and Hazen. Figures 13-7, 13-8 and 13-9, show a scatter graph of the lithium extraction versus temperature, leach time and acid consumption for all tests.

Comments from test results:

- The oxide material extraction was approximately the same as the reduced material. The oxide material requires slightly more leach time to achieve the same extraction.
- Leach extractions are a function of temperature, acid dosage and leach times. Extractions in excess of 85% appear to be achievable with acid dosages of 5% at 75C - 80C with 4 to 6 hours leaching.

- Acid consumption varies depending on the sample tested, the acid dosage and the inclusion of the residual free acid in the leach solution. The average consumption was in the range of 125 kg/tonne.
- Higher agitation rates tended to produce higher extractions mainly due to gas evolution.
- The hydroxide pretreatment resulted in a 4 to 7% increase in lithium recovery under the same leach conditions.

Figure 13-7 Summary of Leach Temperature Results (3 different core samples (laboratories) containing both oxidized and reduced sections, varying leach times 30 to 480 minutes, varying solids concentrations 5-20%, varying acid dosages 5 and 10%)

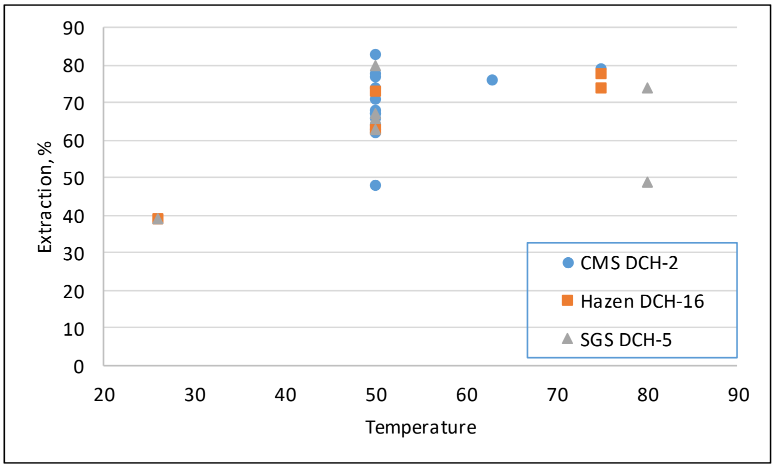


Figure 13-8 Summary of Leach Time Results (3 different drill core samples (laboratories) containing both oxidized and reduced sections, varying leach times 30 to 480 minutes, varying solids concentrations 5-20%, varying acid dosages 5 and 10%)

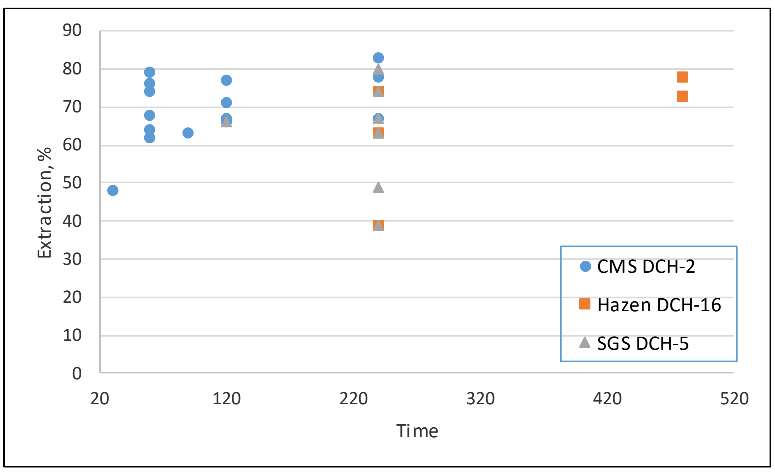
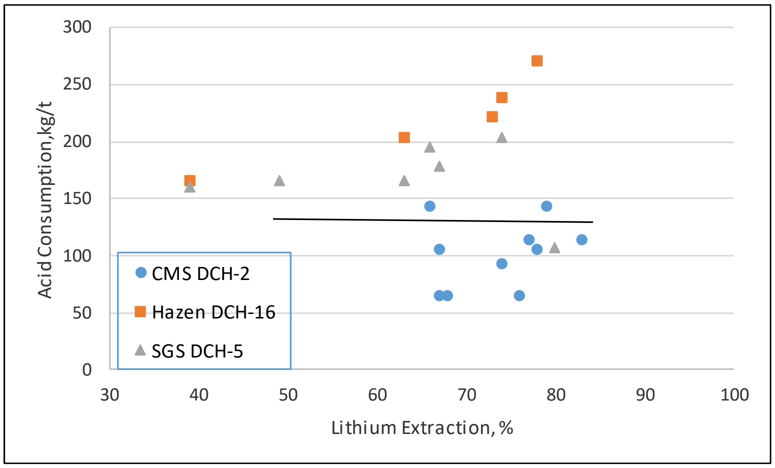


Figure 13-9 Summary of Leach Acid Consumption Results

(3 different drill core samples (laboratories) containing both oxidized and reduced sections, varying leach times 30 to 480 minutes, varying solids concentrations 5-20%, varying acid dosages 5 and 10%)



13.5 Rare Earth Metals

Rare earth metals (REEs) were detected in leach solutions initially from DCH-2 at CMS. Subsequent assays confirmed the presence of REEs in the Oxide and Reduced samples at Hazen, and in samples from DCH-17 and GCH-6 that were analyzed at Bureau Veritas. The leach solutions and residues from the 29 diagnostic leach tests were next analyzed at ALS for REEs.

The results show rare earth metals are present in all samples, in the range of 110 to 200 ppm total REEs. These assays are summarized in Table 13-12, and include scandium, dysprosium, and neodymium, in order of potential economic importance.

Using the solution and tail assays from the diagnostic leach tests, average extractions were also calculated for the check-assay samples of GRE. These results show, within a 1-hour leach at 50C with 5% sulfuric acid, the average extraction for all 29 samples ranges from 16% for lanthanum to 49% for yyitrium.

Based on a review of the REE assays and solubilities shown in the diagnostic leach tests, there is a potential to recover these elements. Using just the 1-hour leach test extractions and an annual feed rate of 5.475 million tonnes, the project, for example, could generate 10 to 15 tonnes of scandium, 25 to 40 tonnes of neodymium, and 5 to 10 tonnes of dysprosium in solution as potentially recoverable oxides. Additional test work is warranted.

Table 13-10 Rare Earth Concentrations

Sample ID	Assay (ppm)																Total
	Sc	Y	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	ppm
Hazen – DCH-16 Oxide	<5	15	24	60	10	9	<5	<5	6	<5	<5	<5	<5	<5	<5	<2.5	124
Hazen – DCH 16 Reduced	5	15	25	57	11	10	<5	<5	5	<5	<5	<5	<5	<5	<5	<2.5	128
DCH-17	4	11	21	42	5	16			2								110
GCH-6	6	13	28	55	6	20			3								141
DN1-DN-29 avg	10	22	34	68	8	28	6		4		4						200
Extraction -%	25	49	16	20	23	24	30		40		40						

13.6 Potassium, Magnesium and other Salts

The tests from SGS, CMS and Hazen indicate potassium, magnesium and other salts are readily solubilized in a sulfuric acid leach. Additional test work into the potential recovery of these elements as salable byproducts is warranted. Cost for their removal as non-saleable products is calculated based stoichiometric balances and included in the capital and operating costs.

13.7 Lithium Recovery

13.7.1 Conventional Lithium Recovery Process – GRE Base Case

Preliminary test work has been conducted for lithium recovery as a lithium carbonate using conventional downstream process methods. The process for lithium recovery from an acid leach solution has been adopted from commercial lithium leaching circuits, and consists of a series of purification processes which are applied before final product generation. The process is broken down into primary impurity removal (PIR), secondary impurity removal (SIR), solution polishing, and product formation.

The PIR process uses lime and/or self-neutralizing properties of the feed to raise the pH and cause iron and aluminum impurities to precipitate. The SIR process raises the pH further to cause calcium, magnesium, and manganese impurities to precipitate. A further ion exchange solution polishing circuit is often employed to remove residual impurities. Final product as lithium carbonate is precipitated using soda ash. A series of solid-liquid separation stages are involved between purification stages along with changing temperatures to facilitate the chemical reactions. The final lithium carbonate product is dried at low temperature for final product delivery, purities typically exceed 99.5%.

Table 13-11 shows the results of the purification test work conducted by CMS. In the purification test, lime, soda ash and sodium oxylate, are added in steps to neutralize the leach solution. The tests almost complete removal of magnesium, calcium, manganese and iron with increasing pH, and with minimal loss of lithium from solution.

Table 13-11 CMS Purification Test Work Results

Brine pH	Li	Mg	K	Ca	Na	Mn	Fe
1	173	4225	1087	593	1997	98	183
6	157	3172	984	473	1807	5	0
10	172	16	1063	756	1978	ND	ND
12	193	2	1223	2	7097	ND	ND

13.7.2 Counter Current Leach Test Work

Based on the evaluation of previous leach tests, a counter-current leaching (CCL) scenario was developed to use the excess acid in the leach solutions to leach fresh incoming feed, thereby increasing solution grade and reducing downstream neutralization requirements. The CCL system uses multiple leach vessels operating counter-current flows of the leach solution and feed input. Overall leaching is completed at higher pH, reducing acid consumption and increasing solution tenors.

Tests conducted by CMS indicated that a counter-current leaching approach could be a successful method to enhance solution tenors and reduce acid consumption. Acid consumption was calculated at 65 to 75 kg/tonne for the Reduced and Oxide samples from DCH-2.

13.7.3 Membrane Recovery Processes

Recently, a significant interest has been directed towards the use of membranes to enhance the recovery of lithium from acid leach solutions. Membranes are designed to pump solutions at high pressure and allow the selective passage of elements across the membrane surface to enrich the permeate or the retentate depending on the system. Membranes have yet to be commercially applied for the lithium industry, but there appears to be significant upside in terms of solution enrichment and potential acid recovery. Cypress engaged a third party vendor to outline a basic process for the recovery of lithium from lithium leach solutions using membrane technology, including developing a basic flowsheet and estimation of capital and operating costs. The costs appear competitive with the base case processing flowsheet, and further consideration is warranted given the flowsheet includes potential recovery of rare earth elements and magnesium.

13.8 Conclusions and Interpretation

The following are conclusions and interpretations of the metallurgical work:

- Clayton Valley claystones are very soft and decompose to constituent grains in water.
- The lithium in the claystones is readily soluble in a weak sulfuric acid solution.
- Test work has identified that lithium extractions in excess of 80% can be obtained using leach temperatures from 50 to 80^{o} C, in leach times of less than 480 minutes at atmospheric pressures.
- Acid consumption is variable and dependent on the sample type, dosage, and leach times. On average, acid consumption was approximately 125 kilograms per tonne of claystone. Acid consumptions may be reduced via alternative processing methods such as counter-current leaching or membrane processing.
- The lithium can be readily recovered from the leach solutions using conventional commercial processes.
- Membrane technology shows promise as a method of increasing leach solution grades and recovering excess free acid.
- The rare earth metals may be recoverable as a salable by-product.
- Caustic pretreatment prior to sulfuric acid leaching has shown potential to increase lithium extraction and should be further reviewed.

13.9 Recommendations

On the basis of the information available at the date of this Technical Report, the following recommendations are made:

- Bench top pilot scale test work should be conducted using conventional lithium recovery processes to confirm process parameters such as retention time, temperature, and reagent consumptions. Locked-cycle testing will allow equilibriums to be established and provide a higher level of accuracy for all process parameters. Full scale pilot testing of the conventional process routes should not be required as these are well proven conventional processes.
- Acid consumption by deposit area needs to be better defined via a geometallurgical investigation.
- Alternative processes, such as membrane separation, should continue to be investigated. Known issues, including fouling, maintenance, and power consumptions, should be investigated to ensure that process risks are well understood.

The potential for further recovery of valuable elements, including rare earths and potassium, should be investigated.

14.0 MINERAL RESOURCE ESTIMATE

GRE has updated the June 5, 2018 Mineral Resource including additional land staked by Cypress following the June Report. No other data changed from the prior estimate.

The Mineral Resource Estimate reported for the project was completed under the direction of Terre Lane, Principal of GRE and a NI 43-101 Qualified Person. Resource modeling and resource estimation was done with Techbase® software.

14.1 Definitions

Mineral resources stated for the project conform to the definitions adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) as amended May 10, 2014, and meet criteria of those definitions, where:

> A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
>
> A "Measured Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.
>
> An "Indicated Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
>
> An "Inferred Mineral Resource" is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

14.2 Estimation Model

Resource estimation was done using Techbase® software. The mineral resource estimate includes all sedimentary units located in the eastern and southern part of the volcanic units. As there is no drilling in the volcanic areas, they were excluded from the mineral resource estimate (see Figure 14-1). The attributes for the area included in the Mineral Resource Estimate are shown in Table 14-1.

Figure 14-1: Included and Excluded Areas in the Mineral Resource Estimate

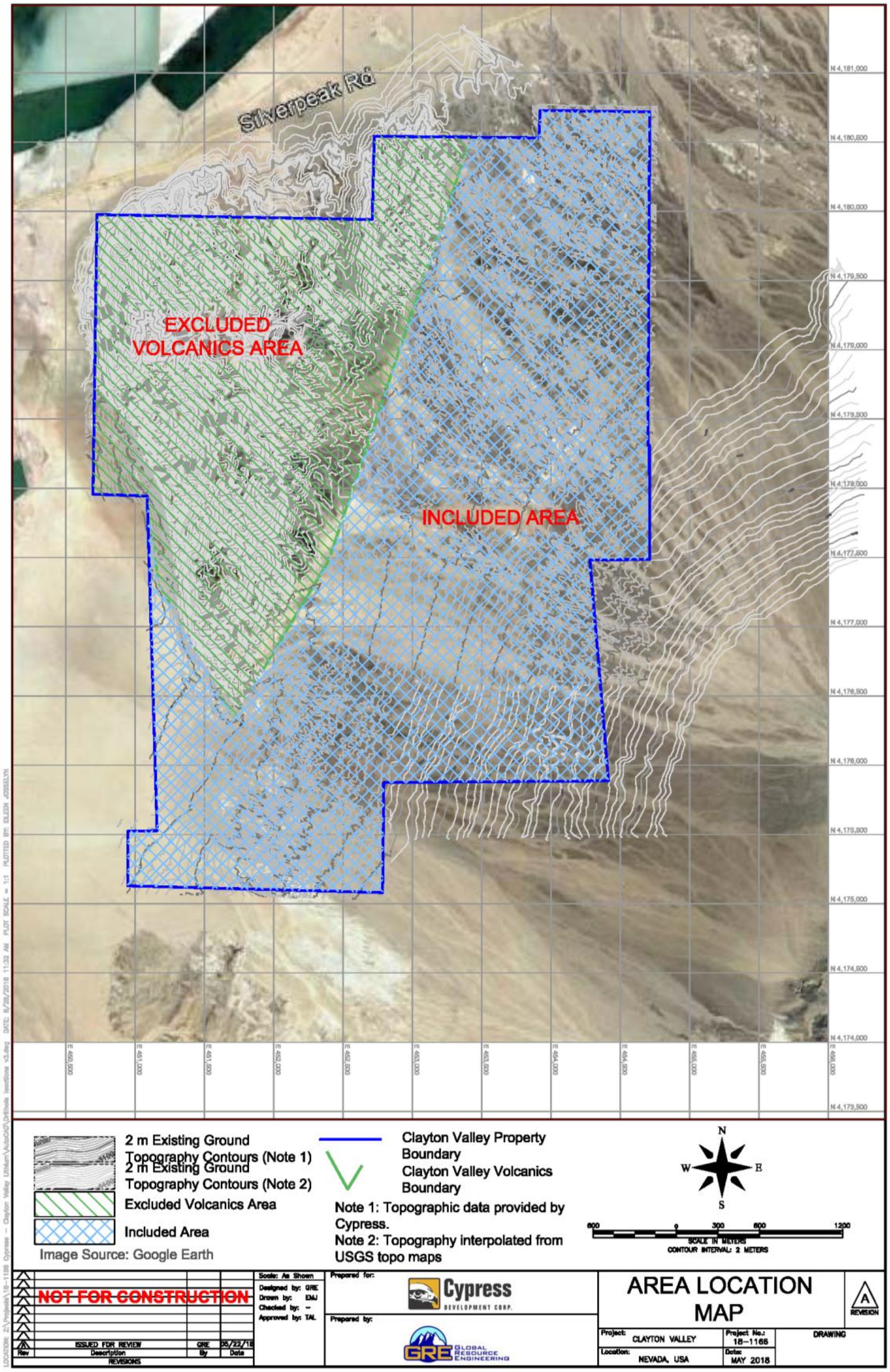


Table 14-1: Area Attributes

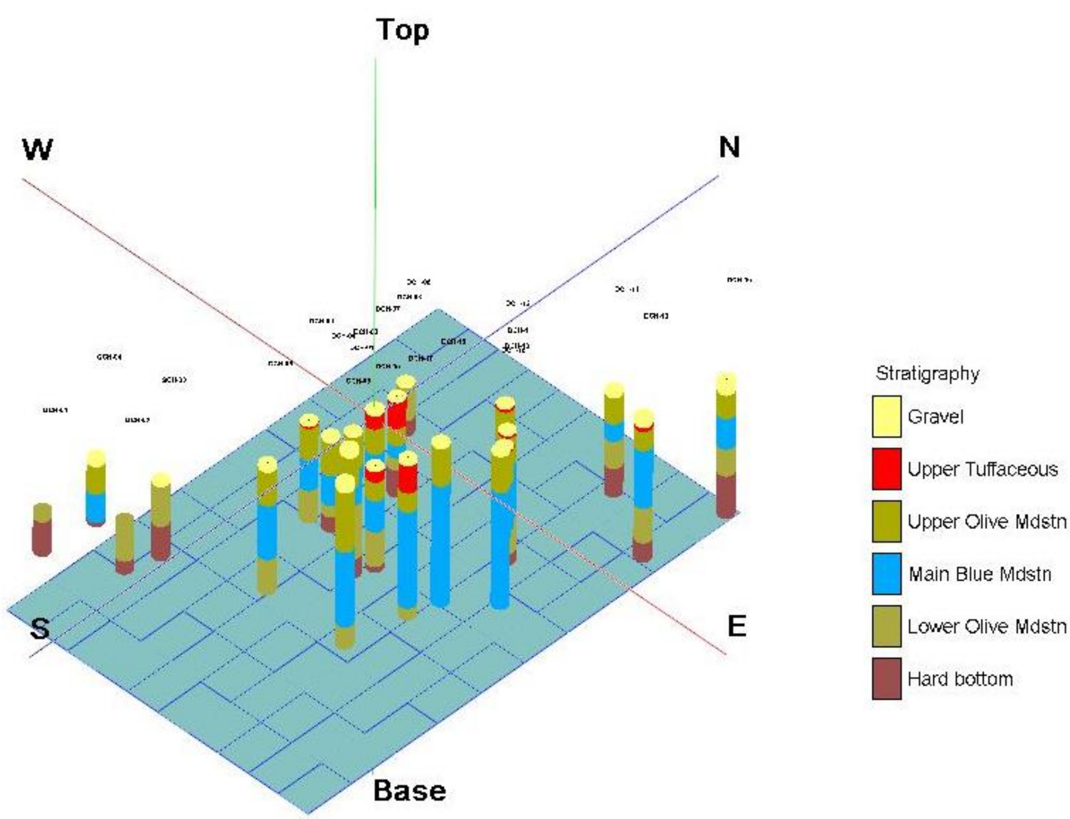
	Easting	Northing	Elevation	Azimuth
Minimum (m)	450,666.67	4,175,300.00	1,310.00	
Maximum (m)	454,666.00	4,180,733.30	1,435.00	
Baseline				90 degrees

14.3 Data Used for the Lithium Estimation

14.3.1 Drill Holes

The mineral resource estimate incorporates geologic and assay results from drilling on the project, including 17 drill holes on the Dean claim blocks and six drill holes on the Glory claim blocks (Figure 14-2). Data provided by Cypress and verified by J.J. Brown, included drill hole data for all drill holes, collar coordinates, drill hole direction (azimuth and dip) (Table 10-1), lithology, sampling, and assay data. This study uses 23 drill holes, totaling 1,905 meters, with an average depth of 82.8 meters per hole. Topography was derived from land survey. Drilling was limited to the sedimentary areas.

Figure 14-2: Clayton Valley 3D View of Drill Hole Logs



14.3.2 Assay Data

The assay data included hole ID, sample weight, lithium in ppm, rock code, lithology code, recovery percentage, and lithology description. The majority of 666 assays for % Li analysis were done on five to ten-foot assay intervals.

14.3.3 Specific Gravity

GRE used a specific gravity (SG) of 1.7 g/cm^3 for all lithological units. This SG is comparable to other similar lithium deposits. GRE recommends additional test work to determine lithology-specific SGs.

14.4 High Grade Capping

GRE produced histograms and cumulative frequency plots of the assay data. If the cumulative frequency plots form a relatively straight line, and the histograms show a nearly normal distribution. Capping is not needed.

14.4.1 Assay

The assay data (excluding gravel) contains a total of 660 Lithium assays, ranging from 165.7 to 2,240 ppm. A histogram of the project's assay data is provided as Figure 14-3.

A cumulative frequency plot of the assay data is shown in Figure 14-4 The cumulative frequency plot indicates a log normal distribution with very few outliers. One assay value over 2,000 ppm occurs in the data. The data approximates a straight line, which is consistent with a nearly normal distribution and one population.

Figure 14-3: Clayton Valley Lithium Project Assay Data Histogram

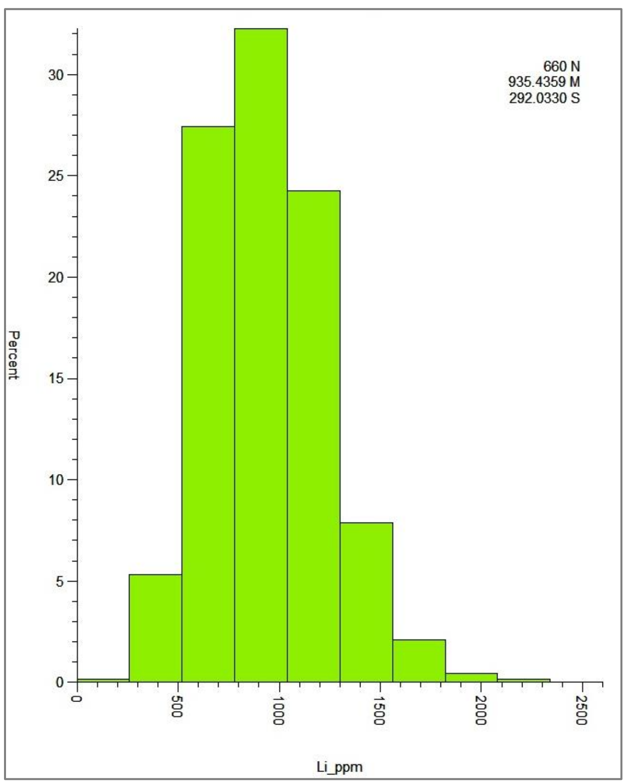
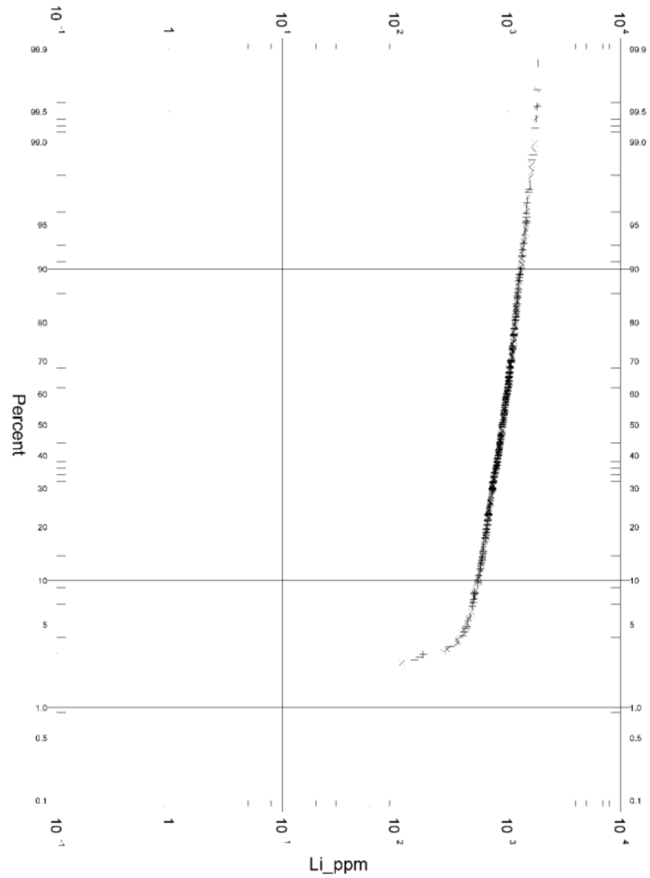


Figure 14-4: Cumulative Frequency Plot, Clayton Valley Lithium Project Assay Data



14.4.2 Composite

The project's assaying was performed almost exclusively using 1.52- or 3.048-meter-long (or 5.0- or 10.0-foot-long) sample intervals. GRE created a single composite for each lithologic unit in each drill hole. The composite intervals are shown in Table 14-2. Examples of the deposit stratigraphy are illustrated in Figure 14-5 and Figure 14-6.

Table 14-2: Composite Intervals

Hole_ID	Lithology	from(ft)	to(ft)	length(ft)	Li Average grade (ppm)
DCH-01	Gravel	0	14.5	14.5	0
DCH-01	Upper Olive Mdstn	14.5	95	80.5	1156.58
DCH-01	Main Blue Mdstn	95	118	23	1108.7
DCH-02	Gravel	0	1.5	1.5	0
DCH-02	Upper Olive Mdstn	1.5	84.9	83.4	953.85
DCH-02	Main Blue Mdstn	84.9	178	93.1	1071.37
DCH-02	Lower Olive Mdstn	178	318	140	777.14
DCH-02	Hard bottom	318	368	50	480
DCH-03	Gravel	0	1	1	0
DCH-03	Upper Tuffaceous	1	6	5	708
DCH-03	Upper Olive Mdstn	6	88	82	997.93

Hole_ID	Lithology	from(ft)	to(ft)	length(ft)	Li Average grade (ppm)
DCH-03	Main Blue Mdstn	88	168	80	789.13
DCH-03	Lower Olive Mdstn	168	252	84	803.1
DCH-04	Gravel	0	5	5	0
DCH-04	Upper Olive Mdstn	5	91.25	86.25	1059.77
DCH-04	Main Blue Mdstn	91.25	168	76.75	1186.81
DCH-04	Lower Olive Mdstn	168	198	30	816.67
DCH-04	Hard bottom	198	238	40	945
DCH-05	Gravel	0	0.25	0.25	0
DCH-05	Upper Tuffaceous	0.25	28	27.75	743.96
DCH-05	Upper Olive Mdstn	28	77	49	893.88
DCH-05	Main Blue Mdstn	77	158	81	1206.17
DCH-05	Lower Olive Mdstn	158	248	90	1186.67
DCH-05	Hard bottom	248	262	14	658.57
DCH-06	Gravel	0	2	2	0
DCH-06	Lower Olive Mdstn	2	88	86	877.97
DCH-06	Hard bottom	88	128	40	955.5
DCH-07	Gravel	0	6	6	0
DCH-07	Upper Tuffaceous	6	41	35	768
DCH-07	Upper Olive Mdstn	41	105.8	64.8	801.88
DCH-07	Main Blue Mdstn	105.8	168	62.2	968.49
DCH-07	Lower Olive Mdstn	168	258	90	631.11
DCH-08	Gravel	0	1.5	1.5	0
DCH-08	Upper Tuffaceous	1.5	69	67.5	700.07
DCH-08	Upper Olive Mdstn	69	114	45	808.44
DCH-08	Main Blue Mdstn	114	146	32	801.88
DCH-08	Lower Olive Mdstn	146	178	32	800.63
DCH-08	Hard bottom	178	248	70	586.57
DCH-09	Gravel	0	27	27	185.56
DCH-09	Upper Olive Mdstn	28	88	61	1156.72
DCH-09	Main Blue Mdstn	88	168	80	1097.75
DCH-09	Lower Olive Mdstn	168	238	70	920
DCH-09	Hard bottom	238	348	110	785.73
DCH-10	Gravel	0	0.25	2.5	0
DCH-10	Upper Tuffaceous	0.25	5	2.5	481.6
DCH-10	Upper Olive Mdstn	5	88	83	900.33
DCH-10	Main Blue Mdstn	88	211	123	1102.37
DCH-11	Gravel	0	1	1	0
DCH-11	Upper Tuffaceous	1	8	7	829.43
DCH-11	Upper Olive Mdstn	8	78	70	1136.99
DCH-11	Main Blue Mdstn	78	218	140	1182.53
DCH-11	Lower Olive Mdstn	218	308	90	827.34
DCH-11	Hard bottom	308	338	30	710.67
DCH-12	Gravel	0	2	2	0
DCH-12	Upper Tuffaceous	2	10	8	496.8
DCH-12	Upper Olive Mdstn	10	88	78	663.62

Hole_ID	Lithology	from(ft)	to(ft)	length(ft)	Li Average grade (ppm)
DCH-12	Main Blue Mdstn	88	168	80	759.82
DCH-12	Lower Olive Mdstn	168	198	30	609.97
DCH-12	Hard bottom	198	218	20	581.2
DCH-13	Gravel	0	18	18	178
DCH-13	Upper Tuffaceous	18	28	10	1008
DCH-13	Upper Olive Mdstn	28	78	50	748.2
DCH-13	Main Blue Mdstn	78	228	150	1219.71
DCH-13	Lower Olive Mdstn	228	318	90	1305.08
DCH-13	Hard bottom	318	368	50	985.02
DCH-14	Gravel	0	9.5	9.5	0
DCH-14	Upper Olive Mdstn	9.5	78	68.5	670.05
DCH-14	Main Blue Mdstn	78	123	45	775.93
DCH-14	Lower Olive Mdstn	123	194	71	764.54
DCH-14	Hard bottom	194	268	74	702.54
DCH-15	Gravel	0	5	5	0
DCH-15	Upper Olive Mdstn	5	104	99	881.85
DCH-15	Main Blue Mdstn	104	418	314	1127.93
DCH-16	Gravel	0	4	4	359.2
DCH-16	Upper Olive Mdstn	4	98	94	833.76
DCH-16	Main Blue Mdstn	98	402	304	1242.67
DCH-17	Gravel	0	6.5	6.5	0
DCH-17	Upper Tuffaceous	6.5	78	71.5	727.2
DCH-17	Upper Olive Mdstn	78	128	50	765.66
DCH-17	Main Blue Mdstn	128	378	250	1114.27
DCH-17	Lower Olive Mdstn	378	408	30	779.47
GCH-01	Lower Olive Mdstn	0	18	18	675.1
GCH-01	Hard bottom	18	108	90	592.49
GCH-02	Lower Olive Mdstn	0	94	94	724.87
GCH-02	Hard bottom	94	128	34	638.41
GCH-03	Gravel	0	5	5	0
GCH-03	Lower Olive Mdstn	5	108	103	762.49
GCH-03	Hard bottom	108	198	90	541.67
GCH-04	Gravel	0	12	12	0
GCH-04	Upper Olive Mdstn	12	84.5	72.5	1076.54
GCH-04	Main Blue Mdstn	84.5	158	73.5	837.26
GCH-04	Hard bottom	158	168	10	497.9
GCH-05	Gravel	0	18	18	410.6
GCH-05	Upper Olive Mdstn	18	172	154	680.5
GCH-05	Main Blue Mdstn	172	368	196	876.75
GCH-05	Lower Olive Mdstn	368	425	57	748.58
GCH-06	Gravel	0	10	10	115.7
GCH-06	Upper Olive Mdstn	10	98	88	1145.38
GCH-06	Main Blue Mdstn	98	238	140	1308.76
GCH-06	Lower Olive Mdstn	238	328	90	885.33

Figure 14-5: Drill Hole DCH-01 Stratigraphy

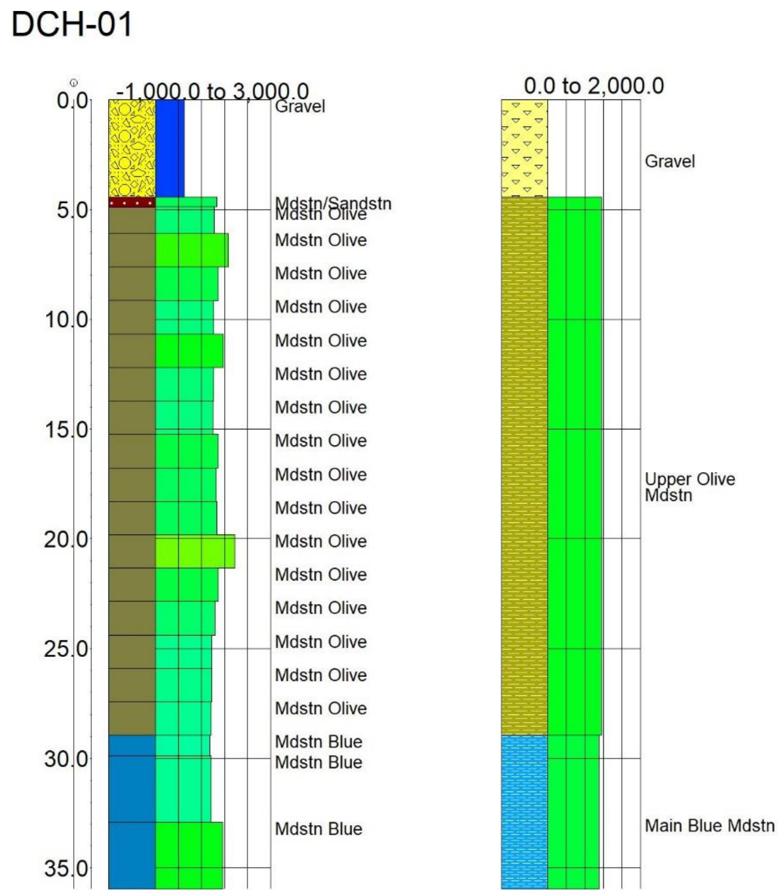
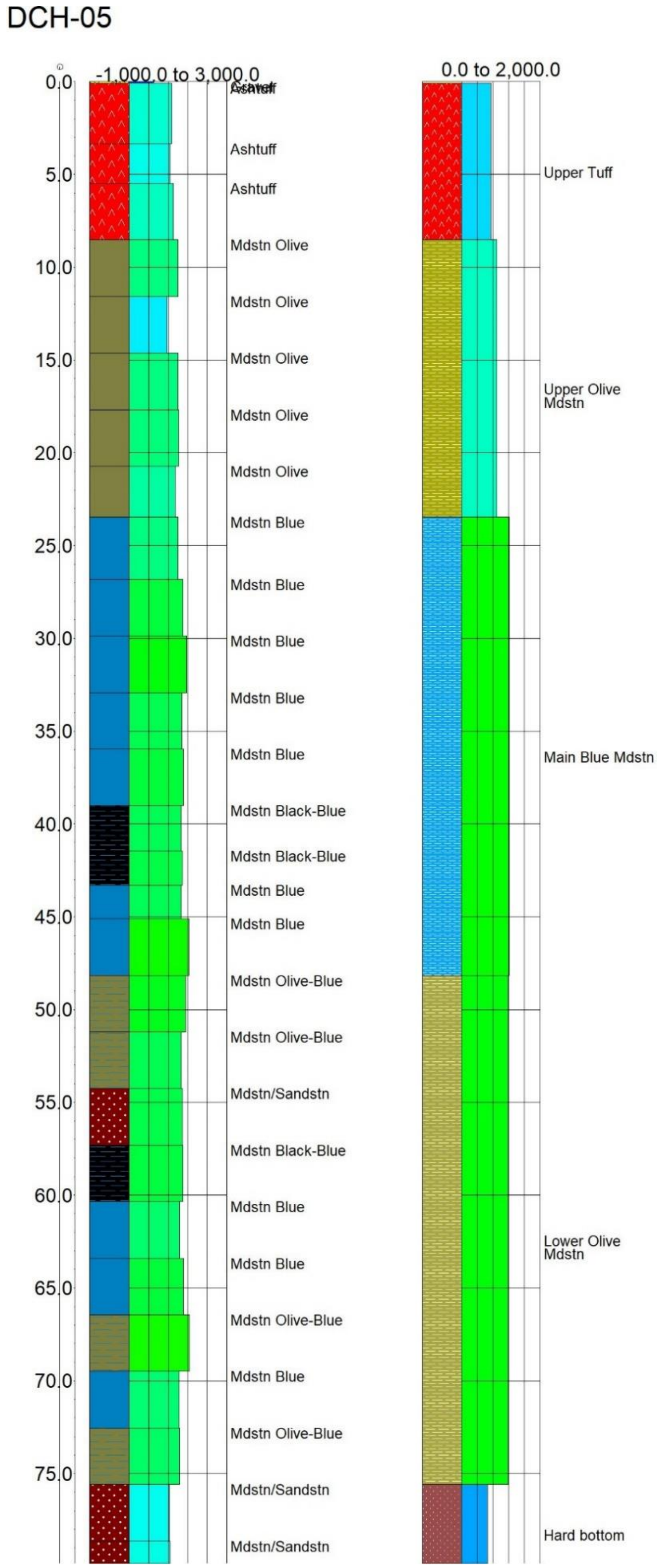


Figure 14-6: Drill Hole DCH-05 Stratigraphy



The statistics for the raw assay data and composited data are shown in Table 14-3.

Table 14-3: Sample and Composite Summary Statistics

Statistic	Sample Values Li (ppm)	Composite Values Li (ppm)
Number	681	79
Mean	908.16	862.41
Standard Deviation	326.15	211.31
Variance	106,372.52	44,653.06
Maximum	2,240.0	1,308.76
Minimum	0	480
Range	2,240.0	828.76
Coefficient of Variance	35.91	24.50

The composite data contains a total of 100 Lithium average grade results, ranging from 0 to 1,308.76 ppm. A histogram of the composite data is provided as Figure 14-7. A cumulative frequency plot of the composite lithium average grade values above 0 ppm is shown in Figure 14-8. The data approximates a straight line, which is consistent with a log-normal distribution and one population.

Figure 14-7: Clayton Valley Lithium Project Composite Data Histogram

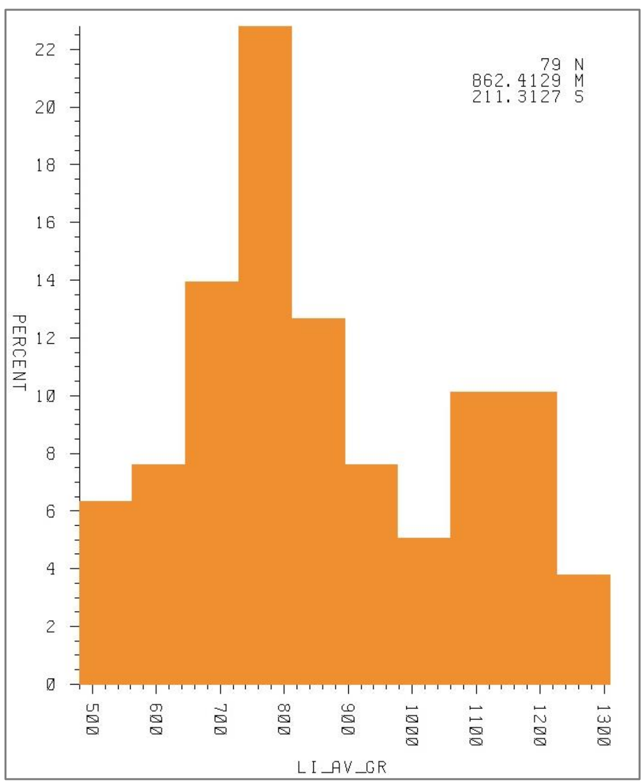
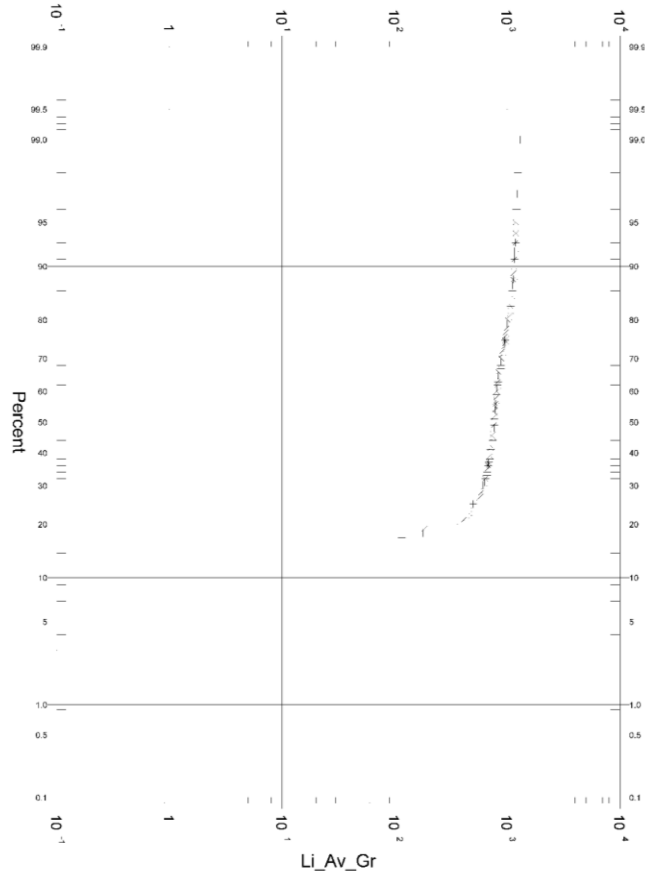


Figure 14-8: Cumulative Frequency Plot, Clayton Valley Lithium Project Composite Data



14.5 Estimation Methodology

The project's lithium claystone deposit is typical of other types of sedimentary deposits, like limestone, potash, soda ash and coal. There is very high lateral continuity of the sedimentary beds with relatively low variability of grade within each of the beds. All drill holes intersected the mineralized beds. The southern end of the Glory claim block appears to be in an uplifted fault block.

GRE used Techbase to create a 2-dimensional (2-D) gridded model of the thickness and grade of each of the sedimentary beds of the project. The thickness and grade of six lithologic units was modeled: gravel, upper tuff, upper olive, main blue, lower olive, and hard bottom. The units are visually distinguished and logged by color and physical characteristics like grain size.

GRE created a single composite of assay data for each sedimentary unit in each drill hole. Control points were added for holes that did not intersect all the units, due to the drill capabilities or erosion, to control unit thickness.

The bottom elevation of each sedimentary layer was then modeled, creating a gridded surface elevation model. Thickness was calculated as the difference in elevation from the bottom of one unit to the bottom of the underlying unit. No drill hole past through the lowest (hard bottom) unit, all ended in above cutoff grade material. GRE therefore extended the depth of the hard bottom 10 meters below the actual drill hole depth.

Table 14-4 provides search parameters used in the modeling.

Table 14-4: Search Parameters

Lithology	Ellipsoid Distance	Major Axis Azimuth
Upper Tuff	1,500 x 750	20
Upper Olive	1,500 x 750	20
Main Blue	1,500 x 1,000	20
Lower Olive	1,500 x 1,000	20
Hard Bottom	1,500 x 1,000	20

14.5.1 Variography

GRE generated variograms on the composites values using Techbase software. The analysis was used to determine the size and orientation of the search ellipsoid for the ID2 grade estimate. First, an omnidirectional analysis was performed for each lithologic unit to obtain the maximum search distance for the grade estimate. Afterwards, each lithologic unit was analyzed to determine the orientation and relative length of the search ellipsoid axes using the maximum search distance. The analysis indicates a nugget of 8,000, a sill of 30,000, and ranges of 1,000 to the east and 1,500 to the north, and 1,500 globally. Figure 14-9 through Figure 14-11 show the variograms for the Main Blue unit.

Figure 14-9: Main Blue Variogram East

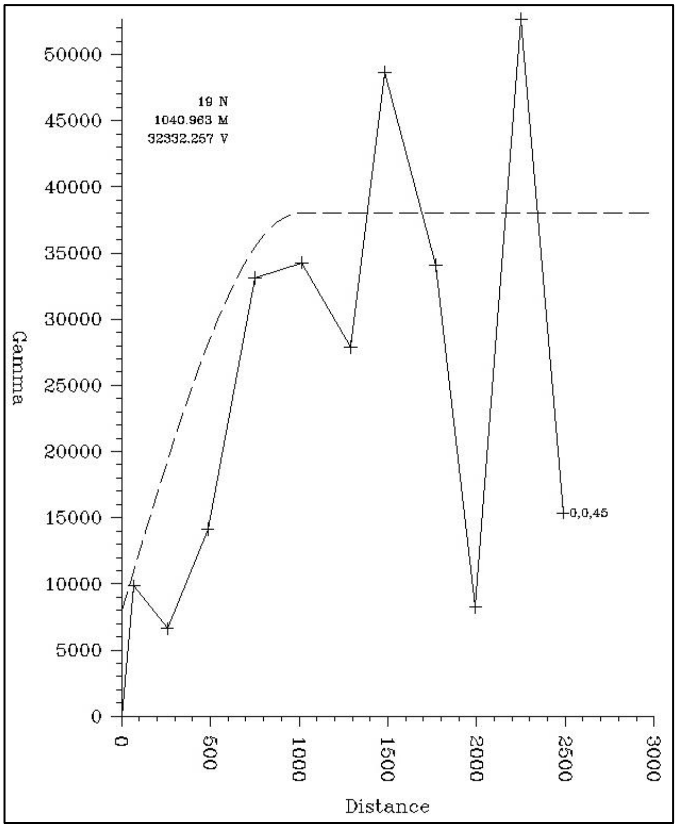


Figure 14-10: Main Blue Variogram Global

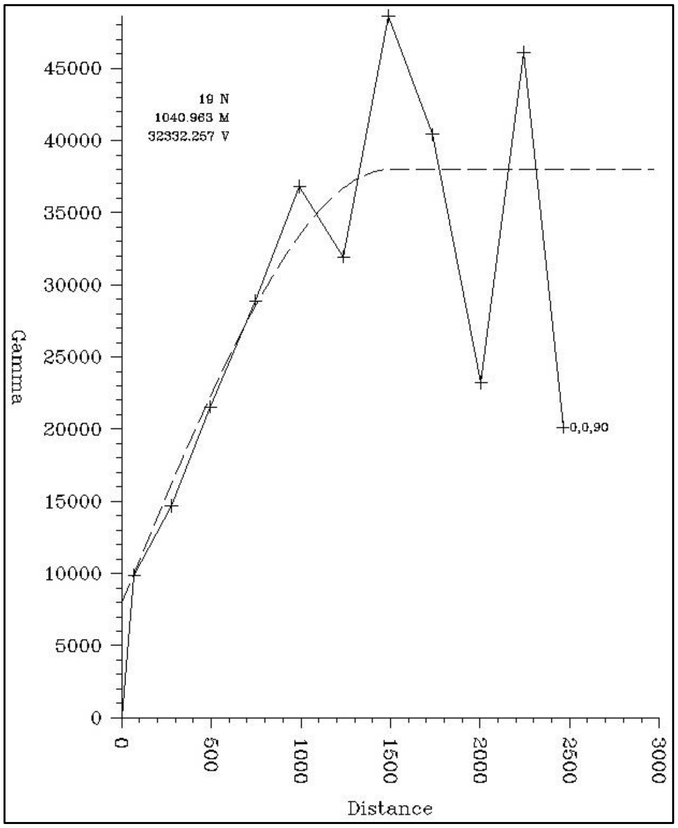
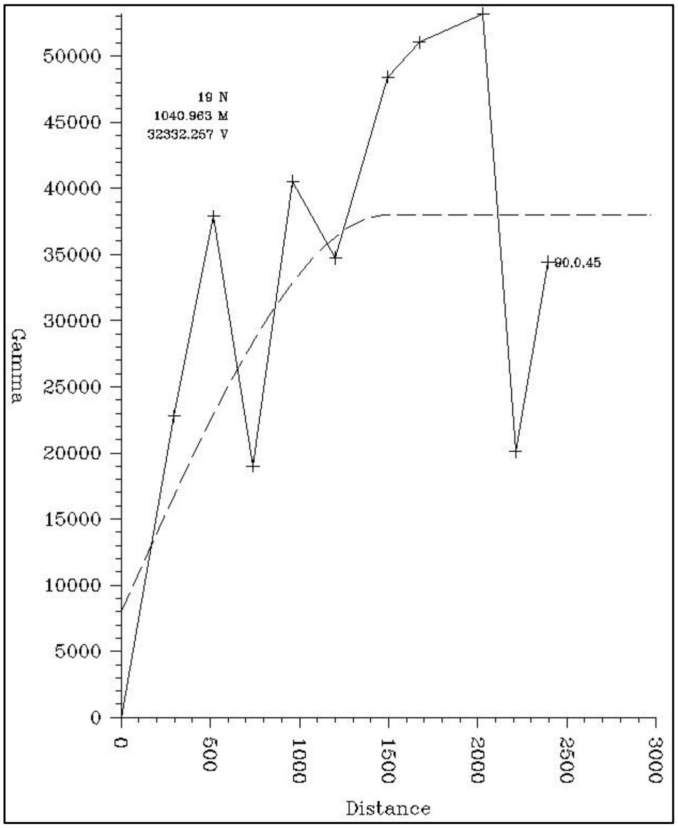


Figure 14-11: Main Blue Variogram North



14.5.2 Grade Modeling and Resource Categories

Grade was estimated using an inversed distance squared algorithm from a minimum of two composites and a maximum of 4 composites. The Mineral Resource was categorized as indicated within 300 meters of a drill hole, which represents 1/5 of the overall variogram range, and inferred farther than 300 meters out to a maximum of the variogram range. These parameters are more conservative than typical industry practice.

Plan views for all six major lithology units were prepared with cell dimensions of 10 m x 10 m showing cells color coded by resource category – green for inferred and blue for indicated. Plan views also were made with cells color coded for different ranges of lithium grade. The two plan views for Upper Olive Mudstone are presented in Figure 14-12 and Figure 14-13, respectively.

For the five main lithology units (not including Gravel), contours of lithium average grade were prepared. The lithium grade contours for the Main Blue mudstone unit are presented in Figure 14-14.

Figure 14-12: Plan View of Resource Categories for Main Blue Mudstone Unit

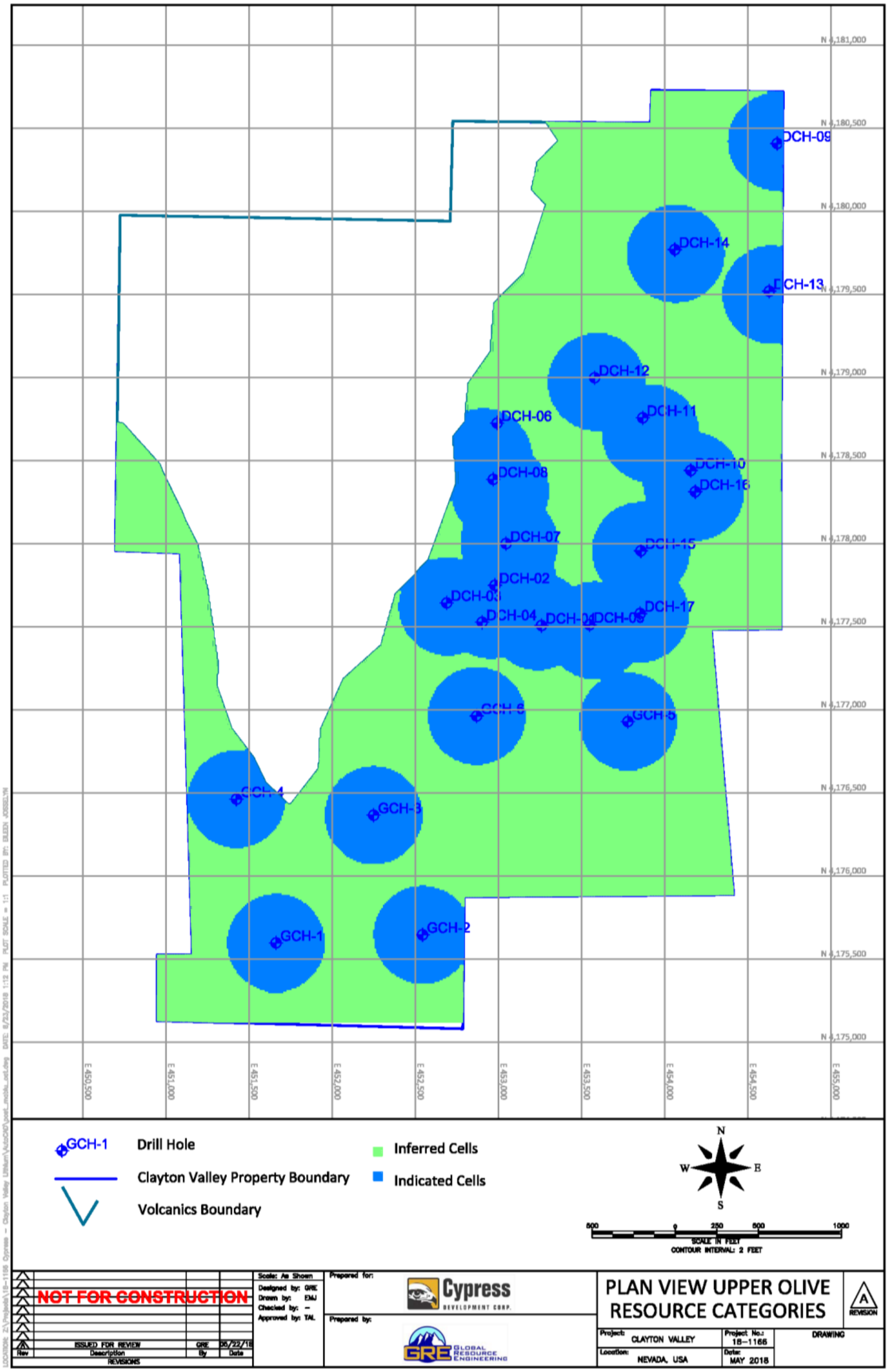


Figure 14-13: Plan View of Lithium Grades for Main Blue Mudstone Unit

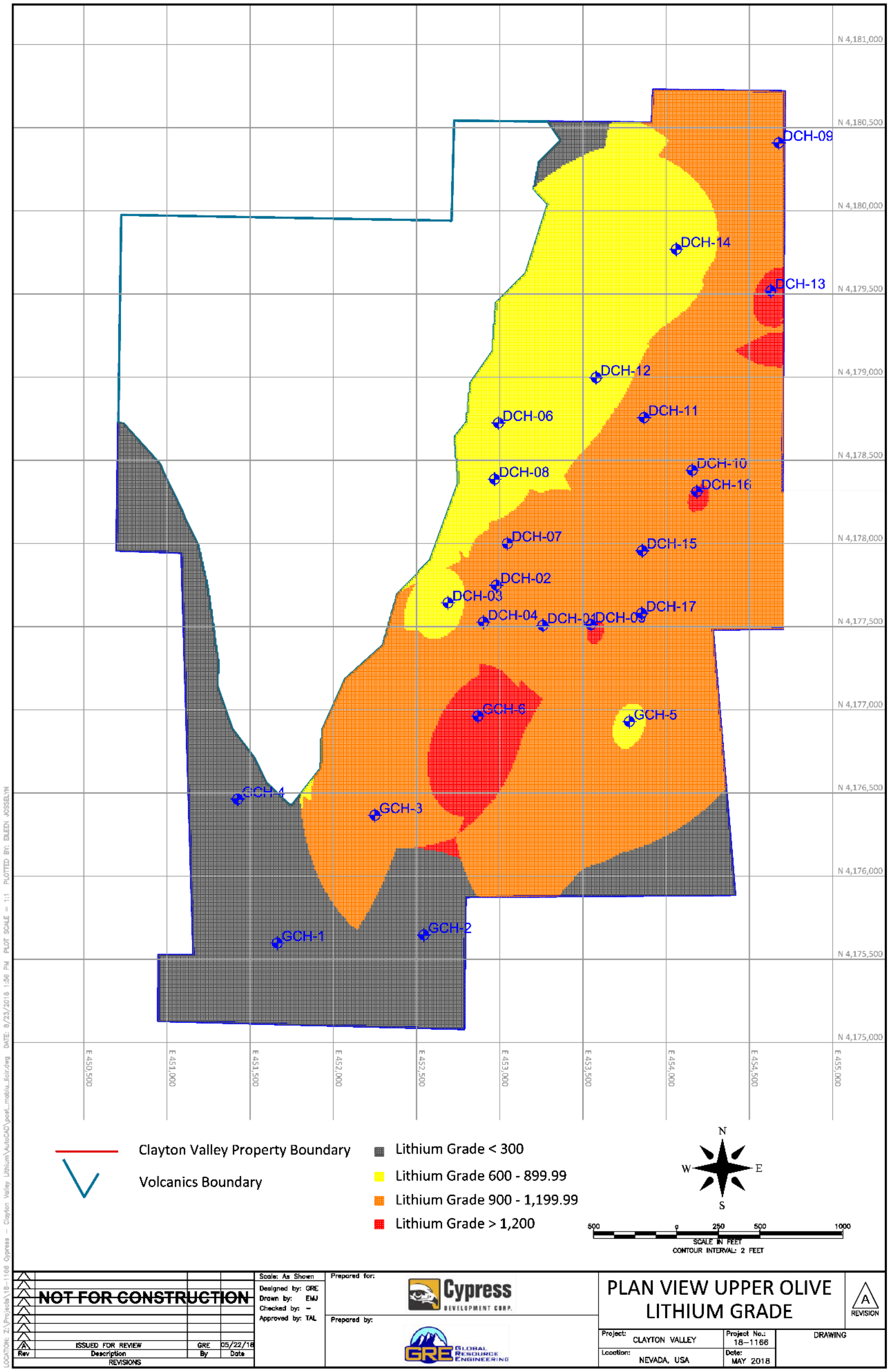
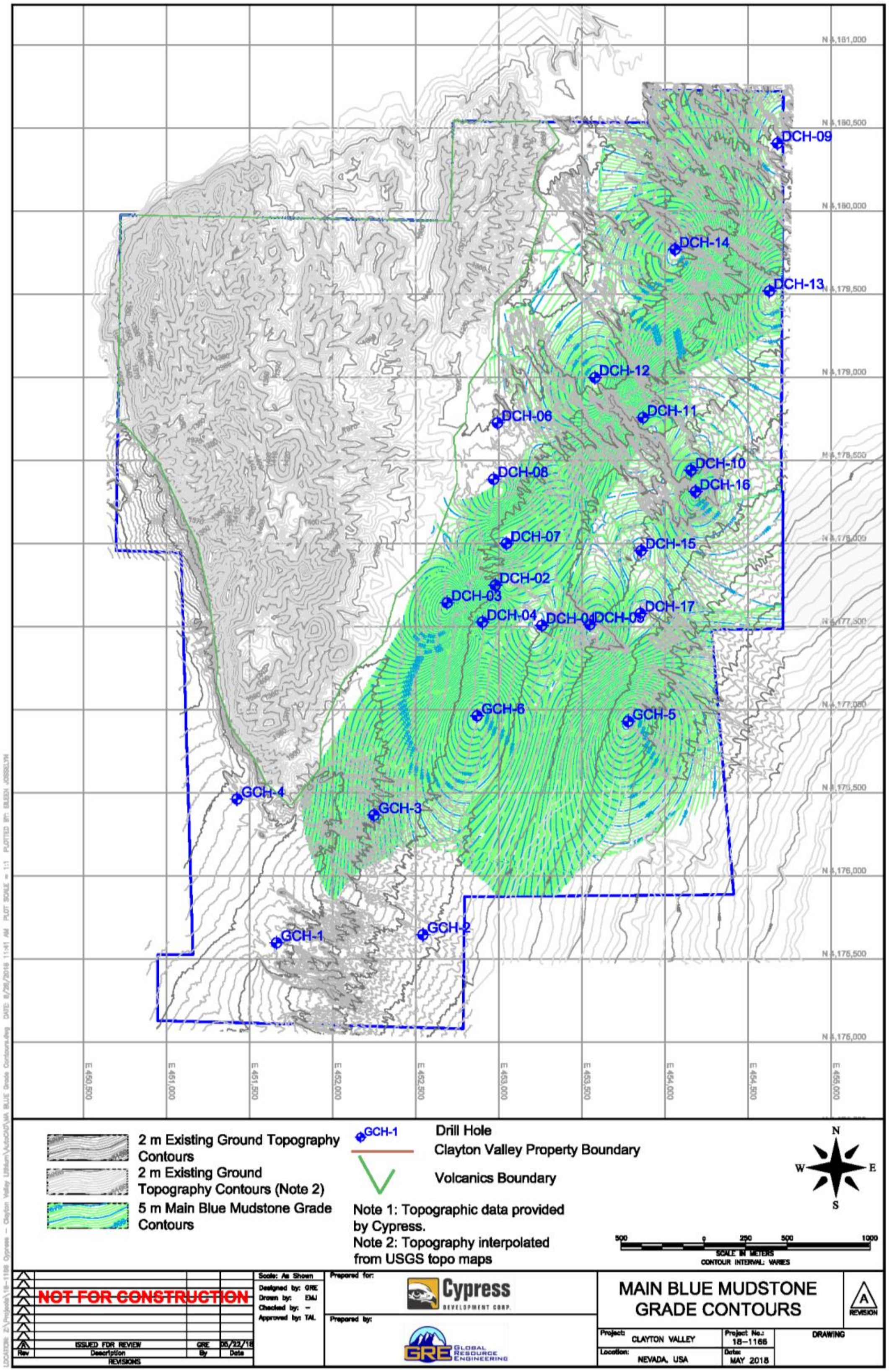


Figure 14-14: Lithium Average Composite Grade Grid-Contour Map for Main Blue Mudstone Unit



14.6 Economic Parameters

The following parameters were input into the model to generate a pit so that pit-constrained Mineral Resources could be calculated:

- Mining cost: $1.00/tonne
- Processing cost: $13.00/tonne processed
    - 100 kg acid/tonne @ $80/tonne delivered
    - $1.25 labor/tonne
    - $1.50 power/tonne
    - $2.25/tonne other leach reagents
- G&A cost: $1.00/tonne
- Lithium recovery: 80%
- Lithium price: $10,000/ tonne of lithium carbonate ($LiCO_3$) (5.323 kg $LiCO_3$ / kg Li)

These costs reflect a 10,000 to 15,000 tonne per day mining operation in soft sedimentary material that does not require blasting.

Half of the processing costs are acid costs. Further study should be conducted on construction of an acid plant and producing sulfuric acid on site.

GRE used these economic parameters to design a preliminary pit, as shown in Figure 14-15.

14.7 Cutoff Grade

GRE calculated the cutoff grade as follows:

Mining	$1.00/tonne
Process	$13.00/tonne
G&A	$1.00/tonne
Total	$15.00/tonne

With 80% recovery, the cost is $18.75/tonne, and with production of 5.323 kg $LiCO_3$ per kg of Li contained and a price of $13,000/tonne $LiCO_3$, the calculated cutoff grade is:

$$\frac{\$18.75}{tonne\ Li} \times \frac{1\ kg\ Li}{5.323\ kg\ LiCO3} \times \frac{tonne\ LiCO3}{\$13{,}000} = 271 \text{ ppm or} \sim 300 \text{ ppm.}$$

The 300 ppm cutoff is the reported Mineral Resource and is bolded in the Mineral Resource tables.

14.8 Estimate Results

Mineral Resource estimate results at cutoffs of 300, 600, 900, and 1,200 ppm are summarized in Table 14-5. This resource estimation includes data from all 23 drill holes. At a cutoff of 300 ppm, the results of the estimation were 720.3 million kg Indicated lithium (831.0 million tonnes) and 963.0 million kg Inferred lithium (1,120.3 million tonnes). Within an initial pit area, at a cutoff of 300 ppm, there are 344.2 million kg Indicated lithium (365.3 million tonnes) and 159.2 million kg Inferred lithium (160.5 million tonnes)

Figure 14-15: Plan View of Preliminary Pit

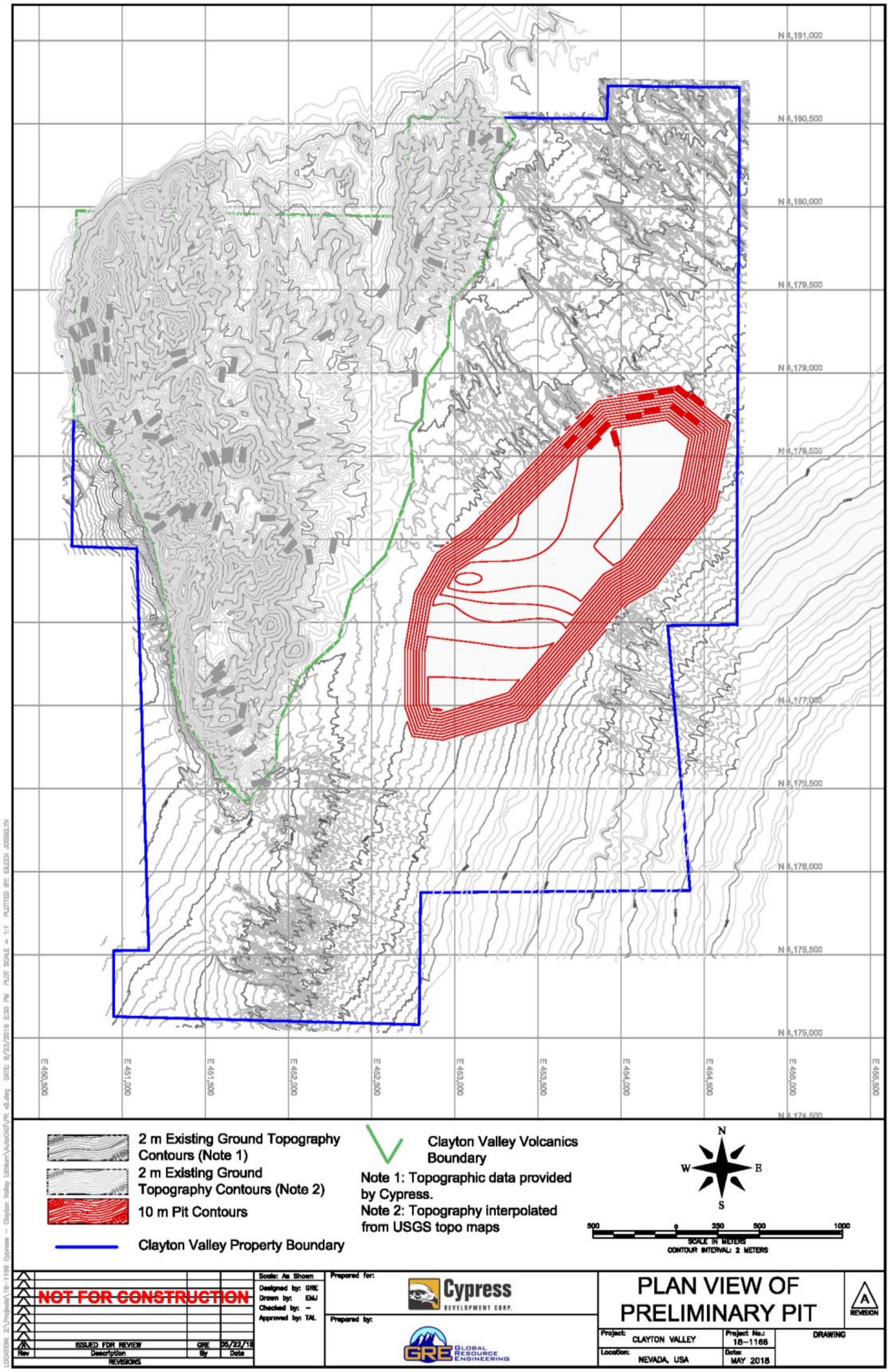


Table 14-5: Summary of Clayton Valley Lithium Project Preliminary Mineral Resource Estimate (1000s)

Lithology	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	Tonne	Li-kg	Grade - ppm
	Indicated Mineral Resource @ 300 ppm Cutoff			Indicated Mineral Resource @ 600 ppm Cutoff			Indicated Mineral Resource @ 900 ppm Cutoff			Indicated Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**50,020**	**35,280**	**705**	43,198	31,500	729	1,827	1,776	973	0	0	-
Upper Olive	**151,438**	**135,340**	**894**	151,438	135,340	894	65,102	67,735	1,040	0	0	-
Main Blue	**248,394**	**270,850**	**1,090**	248,394	270,850	1,090	221,207	248,073	1,121	23,477	29,190	1,243
Lower Olive	**138,773**	**115,265**	**831**	138,773	115,265	831	28,475	28,409	998	942	1,159	1,231
Hard Bottom	**242,418**	**163,567**	**675**	186,661	132,527	710	3,089	2,860	926	0	0	-
Sum	**831,042**	**720,303**	**867**	768,464	685,482	892	319,700	348,853	1,091	24,418	30,349	1,243
	Inferred Mineral Resource @ 300 ppm Cutoff			Inferred Mineral Resource @ 600 ppm Cutoff			Inferred Mineral Resource @ 900 ppm Cutoff			Inferred Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**50,307**	**34,352**	**683**	43,956	30,668	698	670	629	939	0	0	-
Upper Olive	**189,650**	**161,042**	**849**	189,650	161,042	849	56,531	57,362	1,015	0	0	-
Main Blue	**357,362**	**391,098**	**1,094**	357,362	391,098	1,094	343,370	379,114	1,104	10,668	13,000	1,219
Lower Olive	**176,530**	**145,886**	**826**	176,530	145,886	826	29,752	28,382	954	0	0	-
Hard Bottom	**346,461**	**230,584**	**666**	254,698	178,830	702	0	0	-	0	0	-
Sum	**1,120,310**	**962,962**	**860**	1,022,195	907,524	888	430,323	465,486	1,082	10,668	13,000	1,219

Table 14-6: Summary of Clayton Valley Lithium Project Mineral Resource Estimate in Initial Pit Area (1000s)

Lithology	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm
	Indicated Mineral Resource @ 300 ppm Cutoff			Indicated Mineral Resource @ 600 ppm Cutoff			Indicated Mineral Resource @ 900 ppm Cutoff			Indicated Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**26,520**	**18,575**	**700**	23,004	16,623	723	0	0	-	0	0	-
Upper Olive	**74,964**	**72,186**	**963**	74,964	72,186	963	44,644	46,339	1,038	0	0	-
Main Blue	**140,873**	**160,389**	**1,139**	140,873	160,389	1,139	140,457	160,032	1,139	0	0	-
Lower Olive	**53,316**	**45,079**	**846**	53,316	45,079	846	12,843	12,326	960	0	0	-
Hard Bottom	**69,643**	**47,947**	**688**	69,155	47,670	689	33	30	911	0	0	-
Sum	**365,316**	**344,176**	**942**	361,311	341,946	946	197,977	218,726	1,105	0	0	-

Lithology	Tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm	tonne	Li-kg	Grade - ppm
	Inferred Mineral Resource @ 300 ppm Cutoff			Inferred Mineral Resource @ 600 ppm Cutoff			Inferred Mineral Resource @ 900 ppm Cutoff			Inferred Mineral Resource @ 1200 ppm Cutoff		
Upper Tuff	**11,776**	**8,125**	**690**	11,776	8,125	690	0	0	-	0	0	-
Upper Olive	**30,839**	**28,761**	**933**	30,839	28,761	933	15,306	15,436	1,008	0	0	-
Main Blue	**83,602**	**96,730**	**1,157**	83,602	96,730	1,157	83,423	96,570	1,158	15,712	19,618	1,249
Lower Olive	**8,066**	**7,525**	**933**	8,066	7,525	933	8,066	7,525	933	0	0	-
Hard Bottom	**26,174**	**18,067**	**690**	24,244	16,925	698	0	0	-	0	0	-
Sum	**160,457**	**159,208**	**992**	158,527	158,066	997	106,795	119,531	1,119	15,712	19,618	1,249

(Table 14-6). The initial pit area contains resources sufficient to supply a 15,000 tonne per day operation for more than 40 years.

Five to 10 additional holes are recommended in the initial pit area for resource conversion and development, with a goal of converting some of the Indicated resource to the Measured category and most of the Inferred resource to the Indicated or Measured categories.

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under National Instrument 43-101. This Mineral Resource Estimate is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under National Instrument 43-101.

14.9 Estimate Validation

GRE also constructed a 3-dimensional (3-D) block model using the geologic horizons from the 2-D model and 5-meter downhole composites. The 3-D block model used inverse distance squared with a maximum of 10 composites and a minimum of 4 composites using the similar 1500x750x50 meter search parameters as were used for the 2-D model (see Section 14.5.1). Results from the 3-D method are consistent with and verify the 2-D modeling.

GRE also generated cross-sections and longitudinal sections of the deposit to examine the results of the modeling and confirm that the results agree with the geology. Figure 14-16 shows the locations of the sections. The azimuth of the sections is consistent with the apparent strike of the deposit, which is southwest-northeast.

The sections indicate relatively horizontal depositional layers for each of the units. Dips of layers generally follow topographic dips that are generally very gentle from south-east to the north-west. In section S9, deeper layers such as Lower Olive Mudstone and Hard Bottom Sandstone show a gentle dip to the south-east that represents a very open syncline form. The obvious upper contact of Lower-Olive Mudstone in these parts make an angular unconformity with younger lithologic units.

Figure 14-17 presents a representative longitudinal section; Figure 14-18 presents a representative cross section.

Figure 14-16: Clayton Valley Lithium Project Section Locations

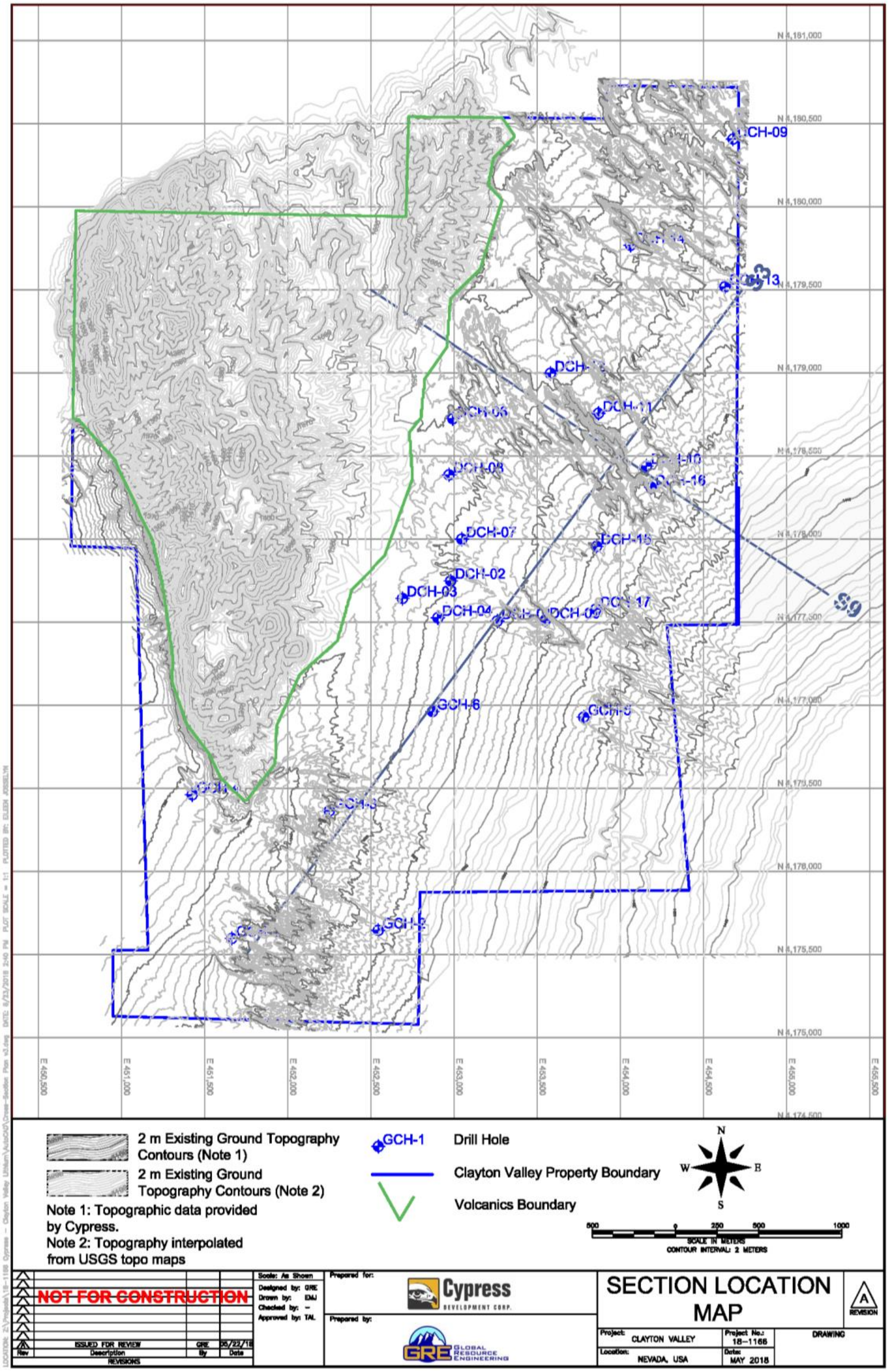


Figure 14-17: Longitudinal Section S3

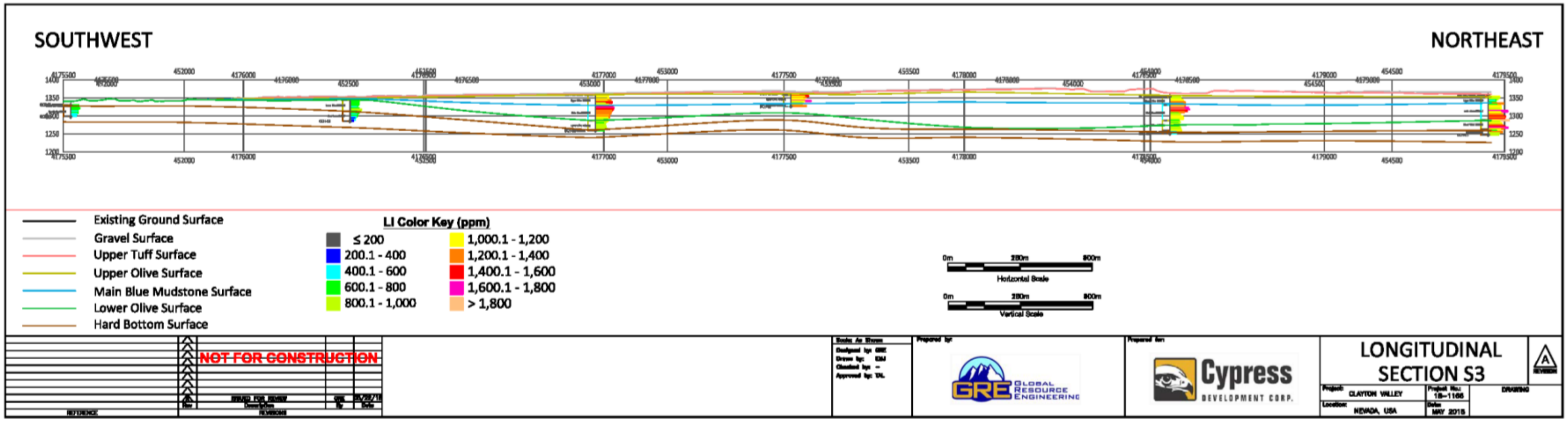
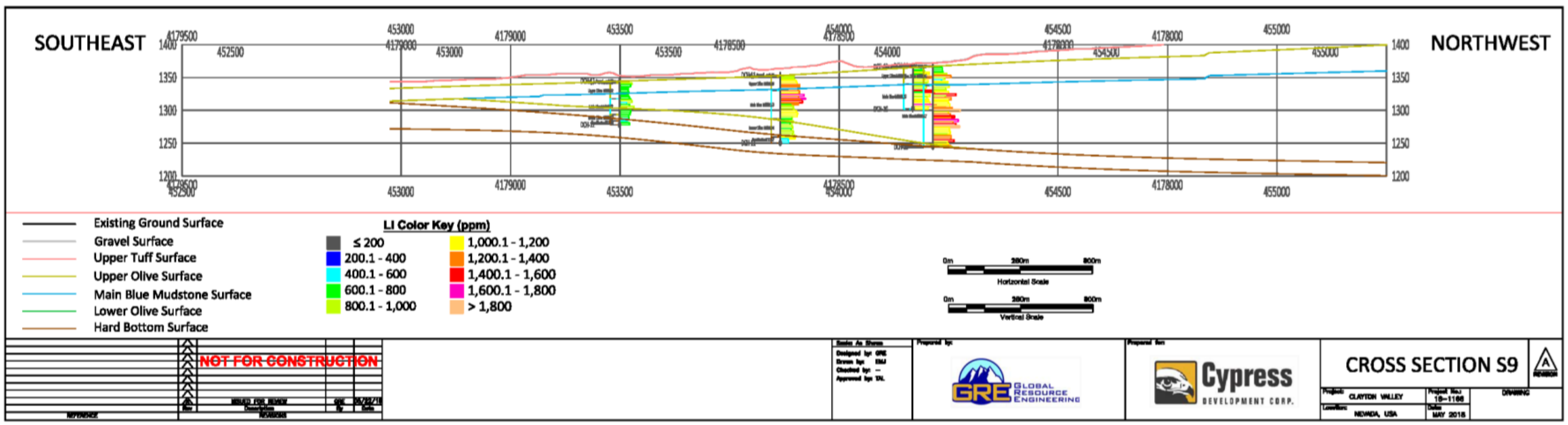


Figure 14-18: Cross Section No. S9



15.0 MINERAL RESERVE ESTIMATES

There are no Mineral Reserves for the project. The project is at a preliminary phase of project development. As defined by NI 43-101, a Prefeasibility Study or Feasibility Study is required to state Mineral Reserves.

16.0 MINING METHODS

Several types of surface mining methods and equipment are potentially suitable for the Clayton Valley Lithium Project, including, but not limited to:

- Dozer and scraper – a mining fleet consisting of five Caterpillar 657G (or equivalent 44 tonne) scrapers and one Caterpillar D10T (or equivalent) dozer
- Surface planer type continuous miner with conveyor and haul trucks – a mining fleet consisting of one PM620 (or equivalent 66 cubic meter per minute) cold planer and five Caterpillar 777G (or equivalent 90 tonne) trucks
- Truck and loader – a mining fleet consisting of five Caterpillar 777G (or equivalent 90 tonne) trucks and one Caterpillar 994 (or equivalent 12 cubic meter) loader
- In-pit semi-mobile feeder-breaker and repulper loaded by a single Caterpillar 994 (or equivalent) loader and pumped as slurry to plant

Drilling and blasting are not expected to be needed.

GRE created an ultimate pit of processable material that extends to the north, east, and south property boundaries, and is bounded by the volcanics boundary on the west, as shown in Figure 16-1. GRE divided the ultimate pit into nine phases, with higher grade material in the earlier phases, as shown in Figure 16-2.

The resources were reported by bench showing tonnes of processable material, waste, and tonnes of lithium. All processable material, whether indicated or inferred, was treated equally for the purposes of the PEA.

GRE varied the cut-off grade in order to create a production schedule with grades averaging above 1000 ppm Li. The cut-off grades used were 3-times the calculated break-even of 300 ppm Li during the first 5 years, and 2-times for the remainder of the mine life. Further optimization of grade and production schedule is warranted.

GRE's economic model evaluated each of the four mining methods mentioned above to optimize the mine planning and design. Based on the economic analysis of all four cases, GRE selected the in-pit feeder-breaker as the operating cost is significantly lower than other options. All further references to the "base case" in this document are referring to the in-pit feeder-breaker with slurry pumping and dilute acid leach processing case.

Table 16-1 displays the CAPEX and OPEX costs associated with each of the mine production equipment options.

Table 16-1 Mine Equipment Cost Comparison

Mine Production Option	Capital	Operating
Truck/Loader	$ 34,523,349	$ 569,183,552
Scraper/Dozer	$ 26,937,497	$ 514,992,963
Surface Planer/Truck	$ 25,993,786	$ 602,290,122
Feeder Breaker/Slurry Pump	$ 28,104,850	$ 345,272,836

16.1 Dozer and Scraper

A single D10 class bull dozer is used to rip the clayton valley claystone. The claystone is soft and friable, and ripping productivity is expected to be over 1000 bank cubic meters per hour. The fleet of 5 cat 657 (dual engine) scrapers will operate in a push-pull configuration to load and haul the broken material to the process plant. Five scrapers are estimated to move slightly more than 1,000 tonnes per hour.

16.2 Surface Planer Type Continuous Miner, Loading Trucks

The second option uses a surface planner, or "cold planer" to cut thin (<.5m) layers of claystone and load the claystone into 100 tonne class haul trucks. The surface planner is expected to have a capacity of about 66 bank cubic meters per minute when cutting. Since the claystone is relatively light, the trucks will need side boards to be fully loaded. A total of 5 trucks is needed to meet production.

16.3 Loader and Trucks

The third option is conventional front end loader loading 100 tonne class haul trucks. A dozer is provided to rip claystone. The 12 cubic meter loader weighs 100 tonnes and may dig claystone without ripping. Again five 100 tonne trucks are required.

16.4 Loader, Feeder Breaker, Repulp, and Pump

The last option GRE considered was a mining with a large 22 cubic meter bucket front end loader (200 tonne weight class) digging and tramming claystone to a feeder breaker where the claystone is crushed and repulped in water and then pumped to the process plant. The feeder breaker and repulp/pump system would be moved periodically when the tram distance exceeds 200 meters. A bull dozer is again provided to rip difficult areas. A single 100 haul truck is used to haul waste out of the pit and a second (back up) loader was provided. Photo 16-4 shows an example of a feeder breaker and photo 16-5 shows a loader loading a track mounted feeder breaker. This option has the lowest operating, and requires the least amount of support equipment as there is very little traffic on the haul road, which reduces road maintenance requirements, water usage, and related costs. GRE selected this option as the base case mining scenario.

Photo 16-1 Example of a Feeder Breaker

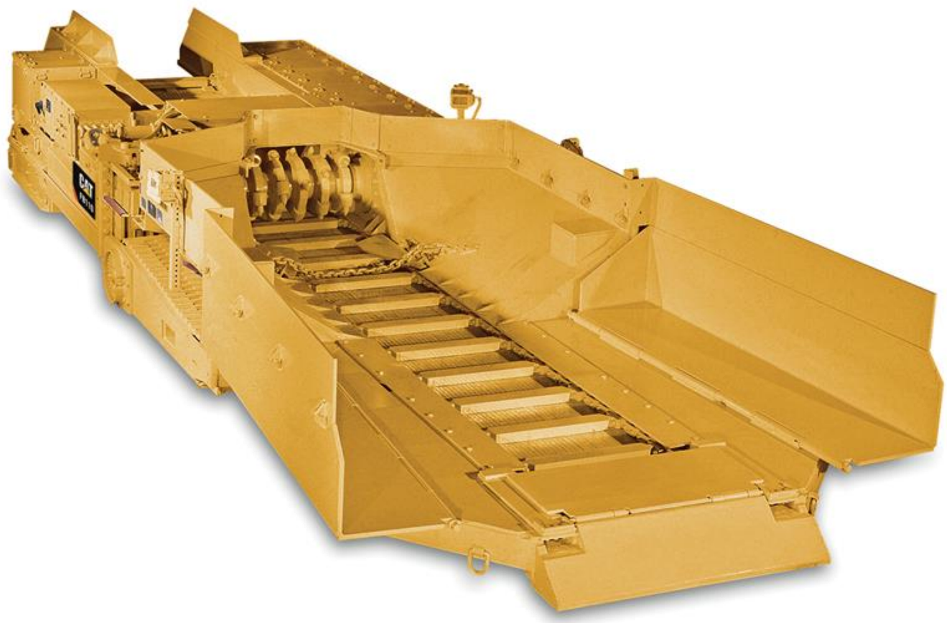


Photo 16-2 Example of a Loader loading a Track Mounted Feeder Breaker

16.5 Support Equipment

Each mining scenario included the following support equipment.

Dozer	D10T	1
Dozer	D8T	1
Dozer (rubber tired)	844k	1
Loader	992K	1
Grader		1
Water Truck		1
Service/Tire Truck		1
Light Plants		4
Pumps (submersible)		2
Pickup Truck		5

16.6 Manpower

GRE estimated the hourly and salary personnel requirements for each case. The mine was assumed to operate on a 2 shift per day, 10 hour shift, 7 days a week, 50 weeks per year. The number of production personnel varies with the mining method, however support staff remained the constant.

Hourly Personnel	Number
Truck/Scraper Operators	3-15
Loader Operators	3-6
Dozer/Grader Operators	3-6
Water Truck Operators	3
Mechanics	9-20
Laborers/Maintenance	6

Salary Personnel	Number
Mine Superintendent	1
Foreman	2
Maintenance Foreman	2
Engineer	2
Geologist	2
Surveyor/Technician	4

16.7 Mine Plan

GRE created an ultimate pit of processable material that extends to the north, east, and south property boundaries, and is bounded by the volcanics boundary on the west, as shown in Figure 16-1. GRE divided the ultimate pit into nine phases, with higher grade material in the earlier phases, as shown in Figure 16-2.

The resources were reported by bench showing tonnes of processable material, waste, and tonnes of lithium. All processable material, whether indicated or inferred, was treated equally for the purposes of the PEA.

GRE examined the project at an initial cutoff grade of 900 ppm for the first five years of operation, followed by a cutoff grade of 600 ppm the remaining years.

GRE's economic model evaluated each of the four mining methods mentioned above to optimize the mine planning and design. Based on the economic analysis of all four cases, GRE selected the in-pit feeder-breaker and slurry pumping case with evaporative processing (see Section 17). All further references to the "base case" in this document are referring to the in-pit feeder-breaker with slurry pumping and evaporative processing case.

16.8 Mine Scheduling

A preliminary mining schedule was generated from the base case pit mineral resource estimate. GRE used the following assumptions to generate the schedule:

- Process production rate: 15,000 tonnes per day (tpd)
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 10

Figure 16-1: Clayton Valley Lithium Project Ultimate Pit

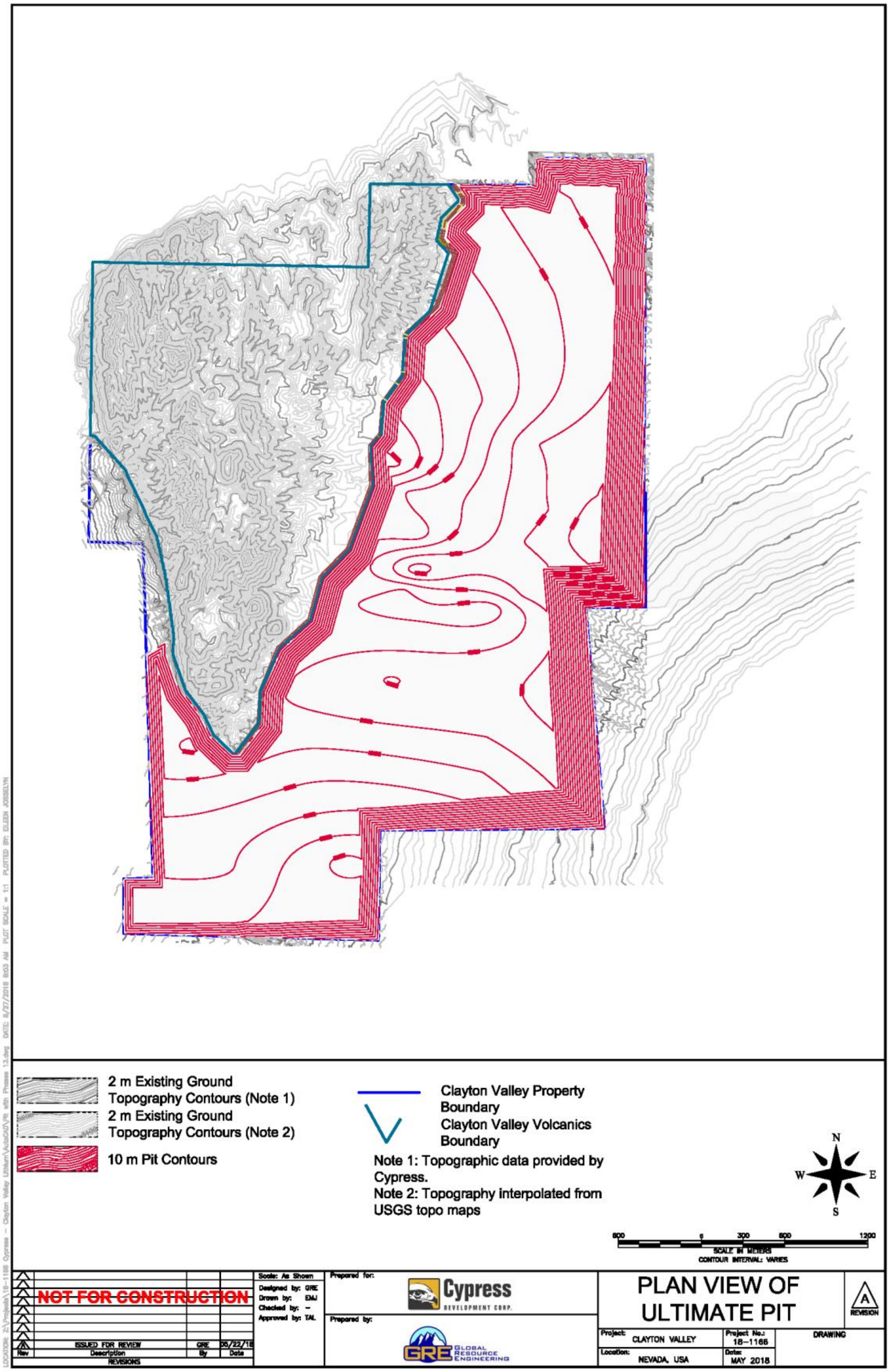
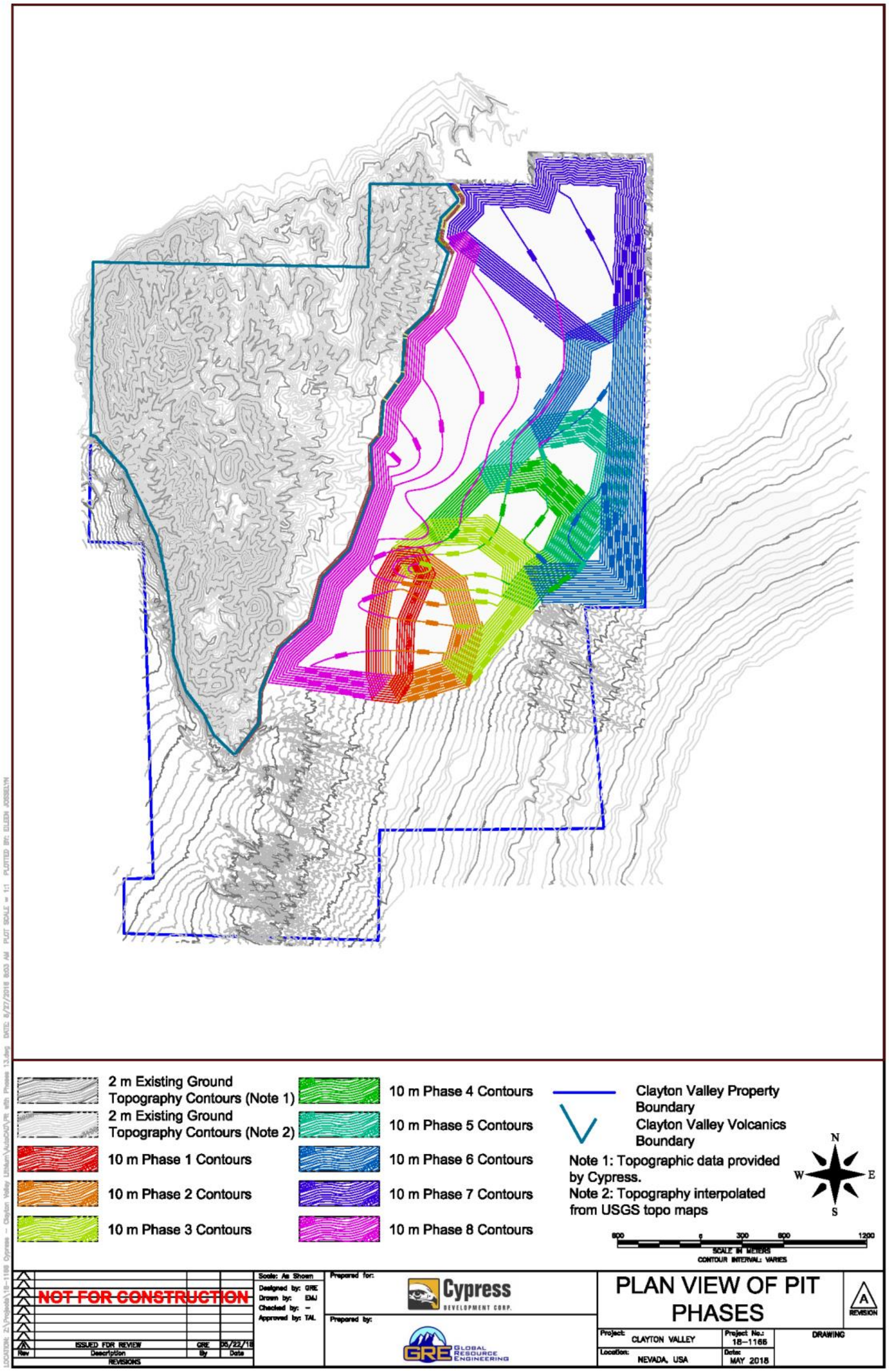


Figure 16-2: Clayton Valley Lithium Project Mining Phases



- The first five years use a cutoff grade of 900 ppm; the remaining years use a cutoff grade of 600 ppm. These grades are 3-times and 2-times the economic cut-off grade, and are used to produce a production schedule with grades averaging above 1000 ppm Li.

Pre-stripping of waste was included if either of the following criteria were true: 1) waste occurred on a bench that had no corresponding processable material or 2) the tonnage of waste on a bench exceeded nine times the tonnage of processable material on that bench. The production rate for pre-strip benches was set to the same rate as the processable material production rate, 15,000 tpd.

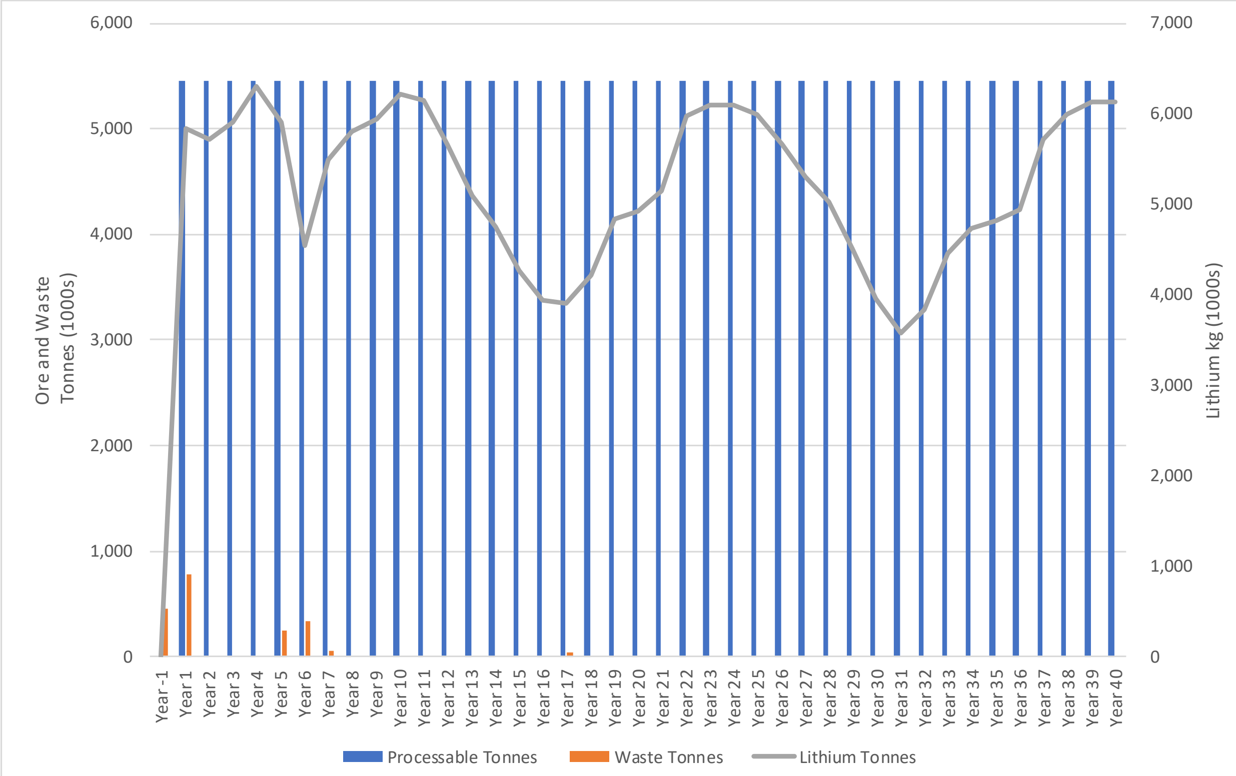
For all other benches, all waste on a bench was scheduled to be mined over the same duration as the processable material on that bench. This scheduling method resulted in some years with high waste quantities relative to the leach material quantity mined. GRE used pre-stripping and phasing, as described above, as much as possible to smooth out the production, but the limitations of the scheduling program resulted in some inefficiencies.

The ultimate (life of project) shell includes pit-constrained resources of 830 million tonnes of indicated material and 1.1 billion tonnes of inferred material. A sequence of pit shells was created based on a 15,000 tonne per day production rate. The nine stages shown in Figure 16-2 total total 1.5 billion tonnes of material and would result in a mine life of more than 200 years..

For this PEA, GRE scheduled only the first 40 years of production. At the break-even cut-off grade of 300 ppm, the pit shell for the first five expansion stages, which is referred to as the “initial pit”, contains 365 million tonnes of indicated resources averaging 942 ppm Li and 160 million tonnes of inferred resources averaging 992 ppm Li, as was detailed in Table 14-6.

The mining schedule derived from the initial pit is summarized in Figure 16-3. Future mine planning will be able to focus more on higher grade material and smooth the annual lithium production.

Figure 16-3: Clayton Valley Lithium Project Production Schedule (Years 1 – 40)



16.9 Mine Operation and Layout

Processable material will be transported from the open pit phase to the process plant, while waste rock will be transported to the waste dump.

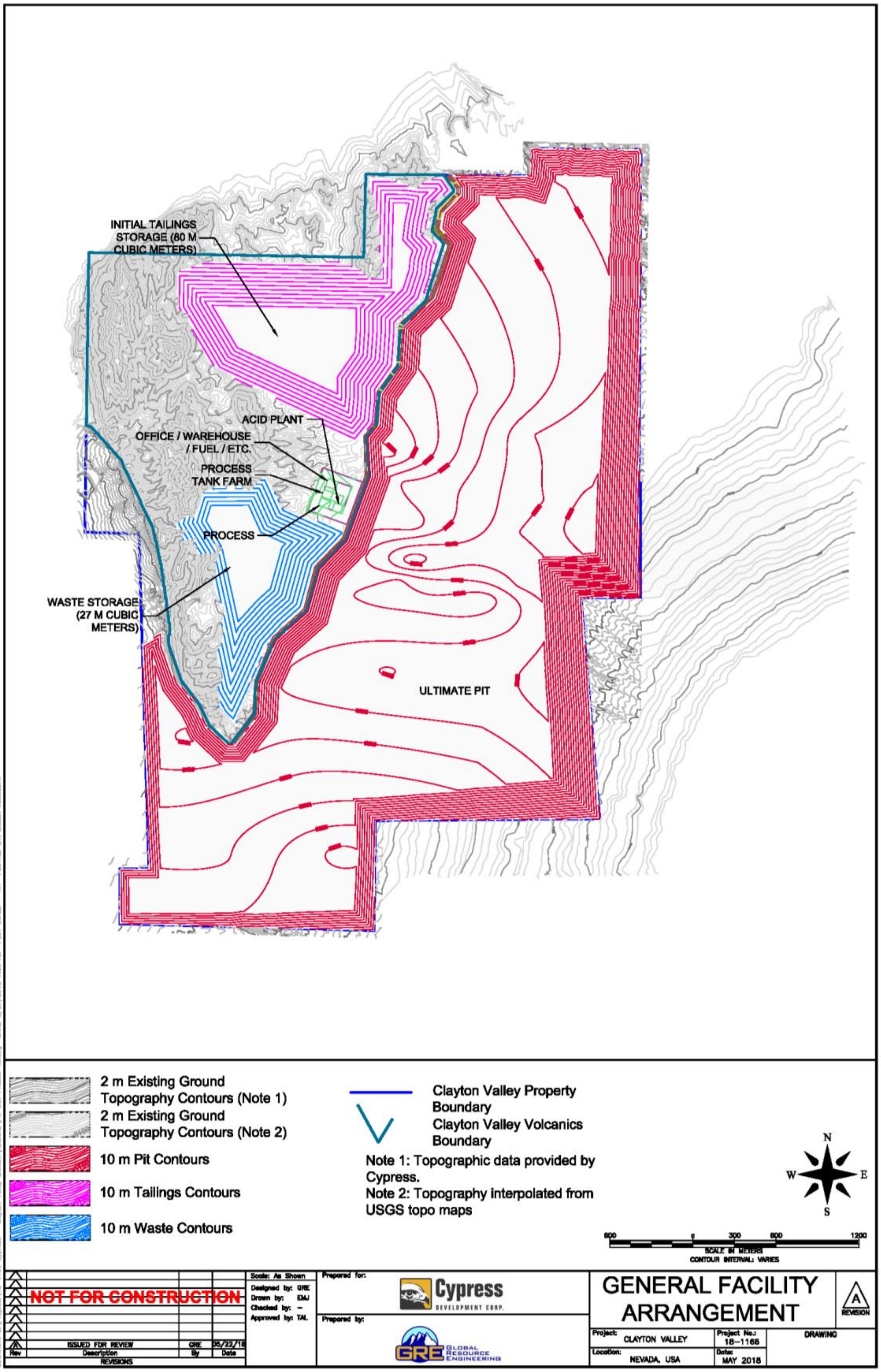
GRE developed a conceptual layout for the deposit area, including waste dump and tailings locations and sizes and processing facility locations. Figure 16-4 illustrates the conceptual Clayton Valley lithium project layout with pits, pads, and tailings and waste storage.

GRE used an overall inter-ramp pit slope of 30 degrees. The addition of haul roads would result in an overall slope of less than 30 degrees. The maximum road grade in pit would be 10%.

Access roads will be designed with a width of 50 feet to accommodate the proposed equipment fleet, including ditches and berms. The access road would be wide enough to accommodate two-way traffic. The maximum road gradient is 8%.

The overall pit slope parameters used in the pit shell were 30 degrees. No water was encountered in drill holes. A complete pit slope analysis needs to be completed to evaluate the project slope stability.

Figure 16-4: Clayton Valley Lithium Project Conceptual Site Layout



17.0 RECOVERY METHODS

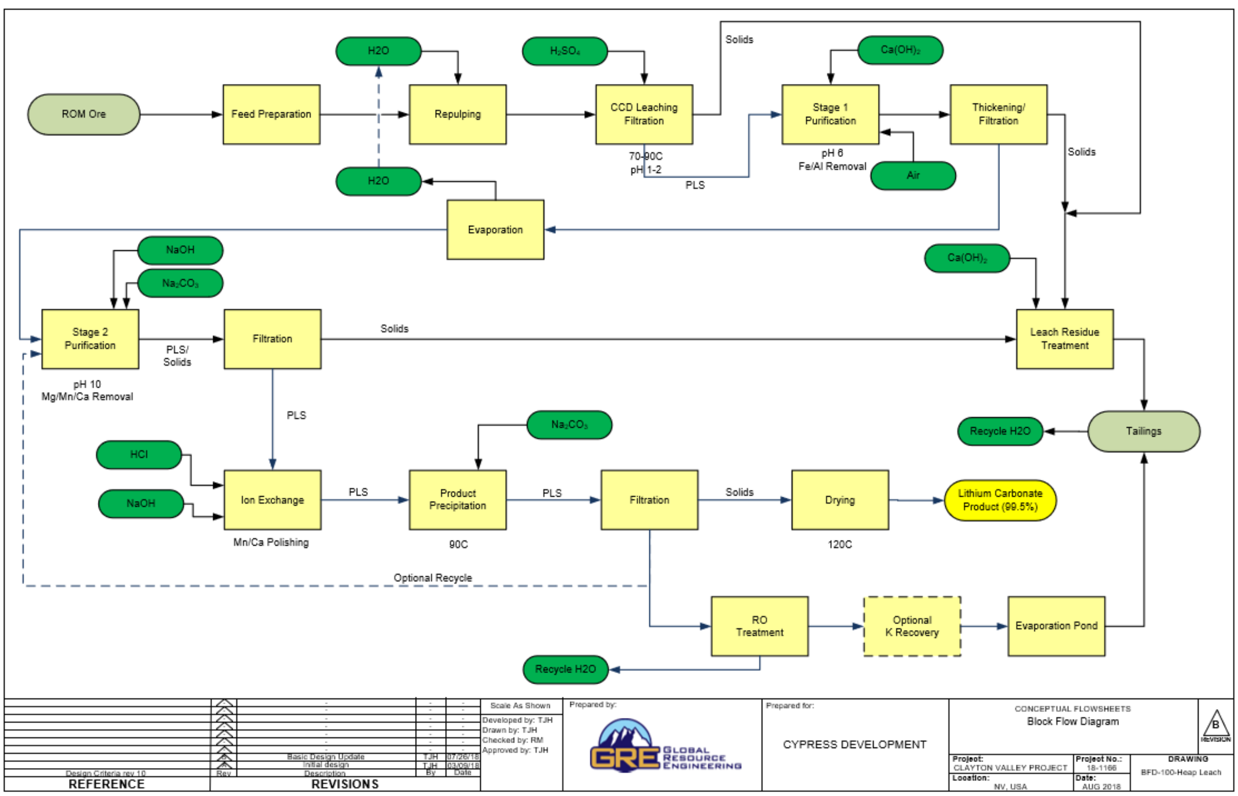
This section describes the processing pathway for the recovery of lithium as a lithium carbonate (99.5% purity) from the claystone material hosted within the Cypress Valley project. The flowsheet is based on test work outlined in Section 13.

The process has been developed based on industry-standard commercially proven unit operations derived from prevailing leaching and recovery circuits. This flowsheet is the basis of the capital and operating cost provide in subsequent sections of the document. An alternative lithium recovery circuit is discussed at the end of this section related to the use of membrane technologies for solution purification.

The designed throughput for the process is 15,000 tonnes per day or 5,475,000 tonnes per year averaging 1,012 ppm lithium. The anticipated lithium recovery is 81.5% producing 4,516 tonnes per year of lithium or approximately 24,042 tonnes of lithium carbonate.

Figure 17-1 shows the block flow diagram outlining the major processing unit operations.

Figure 17-1 Proposed Flowsheet



At this stage preliminary test work has been conducted related to final product production. This flowsheet represents a typical lithium production pathway producing lithium carbonate. The process has been divided into basic unit operations, including:

- Feed Preparation
- Lithium Extraction

- Primary Impurity Removal
- Secondary Impurity Removal
- Solution Polishing
- Lithium Carbonate Production
- Tailings
- Utilities – Acid production, water recycle, reagents

Each of the primary unit operation is described in detail in the following sections.

17.1 Feed Preparation

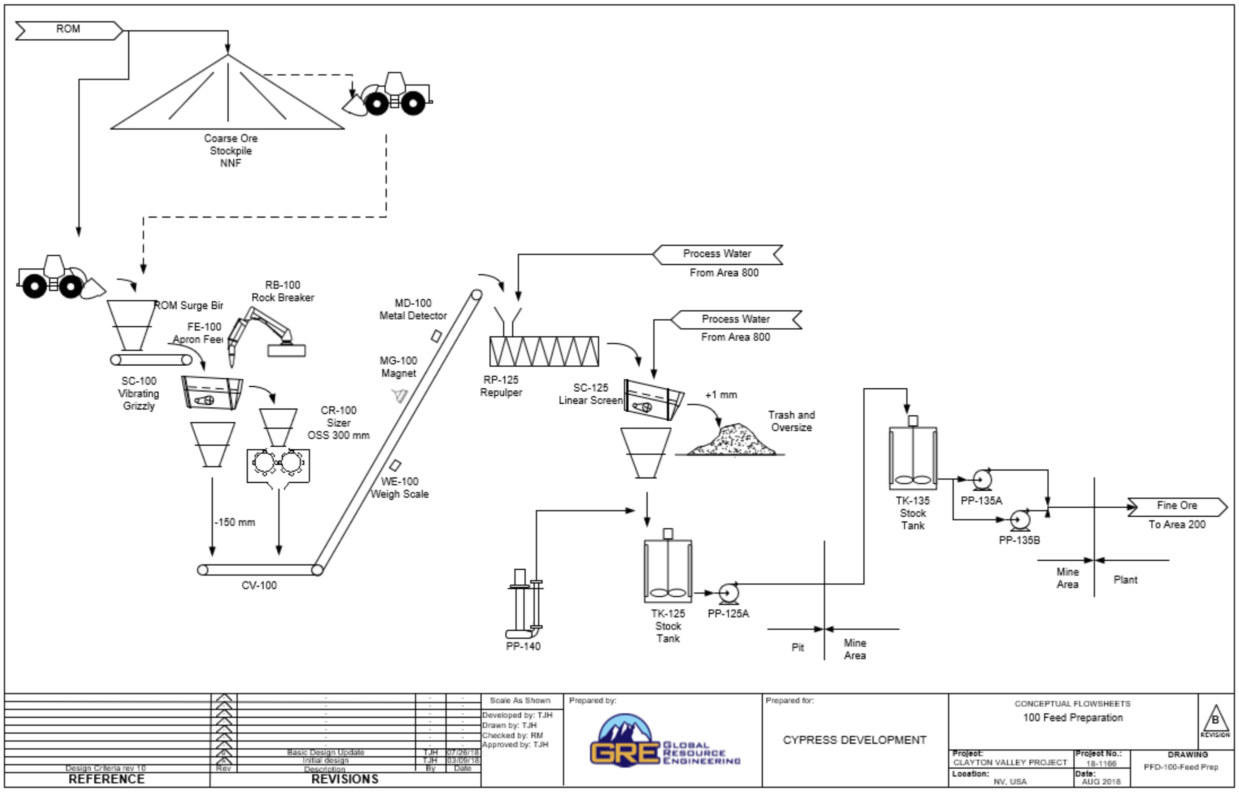
The feed preparation circuit is designed into two main components; a comminution/repulping circuit and a slurry transfer system. The objective is to utilize a semi-mobile system that allows ROM material to be processed in the active mining area and subsequently pumped to the processing facilities approximately 2 km away. The objective is to reduce the cost of material haulage. Figure 17-2 shows the feed preparation circuit.

The ROM material would be transported to a hopper equipped with an apron feeder coupled to a vibrating grizzly for preliminary material sizing. The oversize material, >300 mm, would report to a mineral sizer (toothed roll crusher with a compact footprint). Undersize from the grizzly and the sizer would combine on a common belt as feed to the repulper. A metal detection system and belt scale are included on this conveyor.

The repulper is designed to aggressively mix the ore with water and create a slurry of approximately 60% solids. This slurry is passed across a linear screen to remove trash and oversize grit (nominal 1 mm opening). The screen undersize reports to an agitated transfer tank.

The transfer tank slurry is pumped out of the pit to permanent agitated stock tank, additional water is added to achieve a solids density of 40% prior to the slurry being transferred by a series of pumps through a pipeline to the process plant stock tank. Process water is delivered to the mine facility from the plant via a pipeline.

Figure 17-2 Feed Preparation



17.2 Lithium Extraction

Lithium extraction is achieved through elevated temperature leaching (70-90°C) with sulfuric acid. The sulfuric acid concentration is targeted at 5-10% through the addition of concentrated acid delivered from the acid plant. Heating of the leach slurry is achieved through the introduction of live steam delivered from the acid plant heat recovery system.

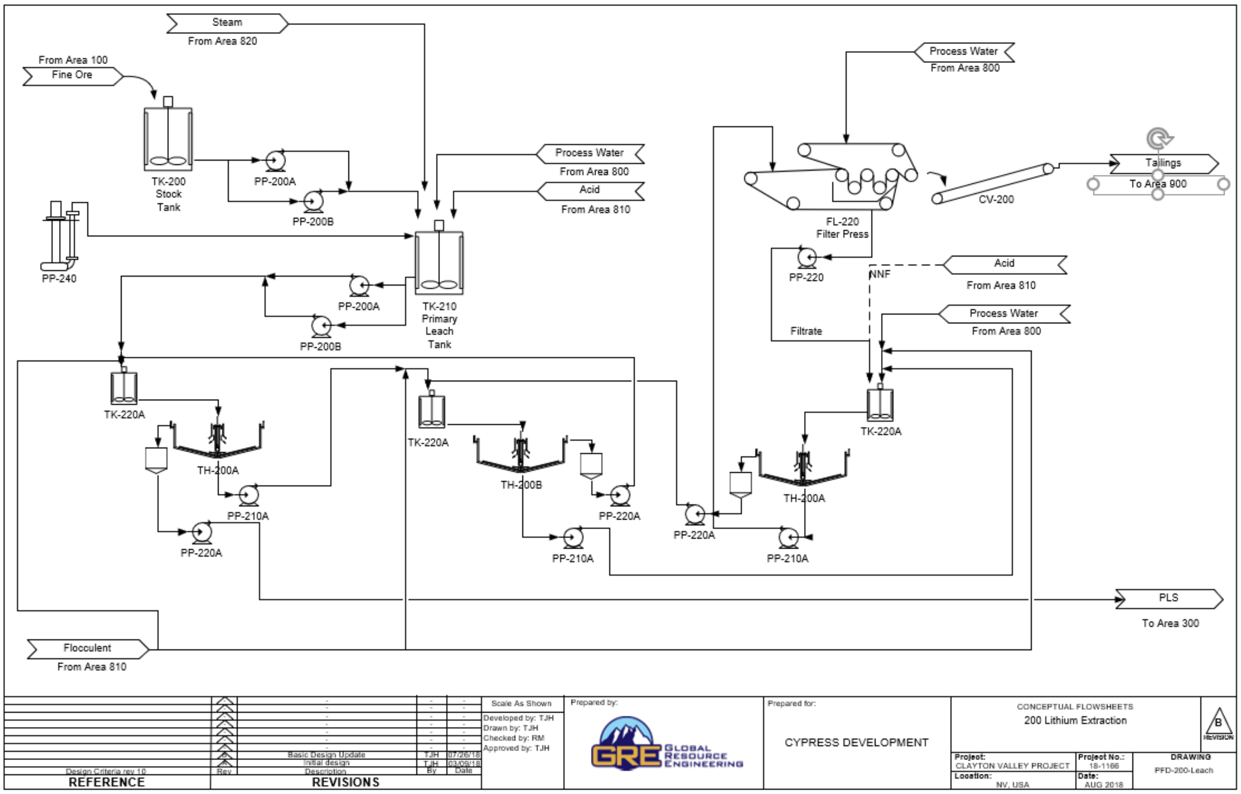
A single primary leach tank serves as the initial leach vessel equipped with a high shear agitator to assist the removal of the evolved carbon dioxide. Primary leaching is conducted at 35% solids. The retention time of the primary leach vessel is 2 hours designed to provide enough retention time to reduce gas evolution to an acceptable level prior to the slurry being transferred to a series of three counter-current decantation (CCD) thickeners each 43 meters in diameter. The solids from the leach circuit flow countercurrent to the leach solution to achieve efficient washing of the leach solids. The use of a CCD system maximizes the solution recovery from the leach circuit and increases the sulfuric acid utilization.

An additional total leach time of 4 hours is targeted in the CCD thickeners. Feed to the first thickener is combined with flocculant and the clear overflow solution from the second thickener and allowed to settle. The target underflow solids concentration is 45% solids. The clear overflow is pumped to the Primary Impurity Removal (PIR) process. The target discharge pH from the final thickener is 2.0.

The third and final thickener underflow is pumped to series of belt filters where additional washing occurs using fresh process water. The solids from filtration are discharged to a conveyor for delivery to the tailing impoundment area. The final solids density is targeted at 90% solids. Recovered filtrate is recycled back to the last thickener feed tank. Final leach recovery is estimated at approximately 82% with a 2.5% solution loss.

Additional acid and steam can be added in each of the thickener feed tanks as required to maintain temperature and free acid levels. Total acid consumption is estimated at 125 kg per tonne of ore. Figure 17-3 illustrates lithium extraction circuit.

Figure 17-3 Lithium Extraction

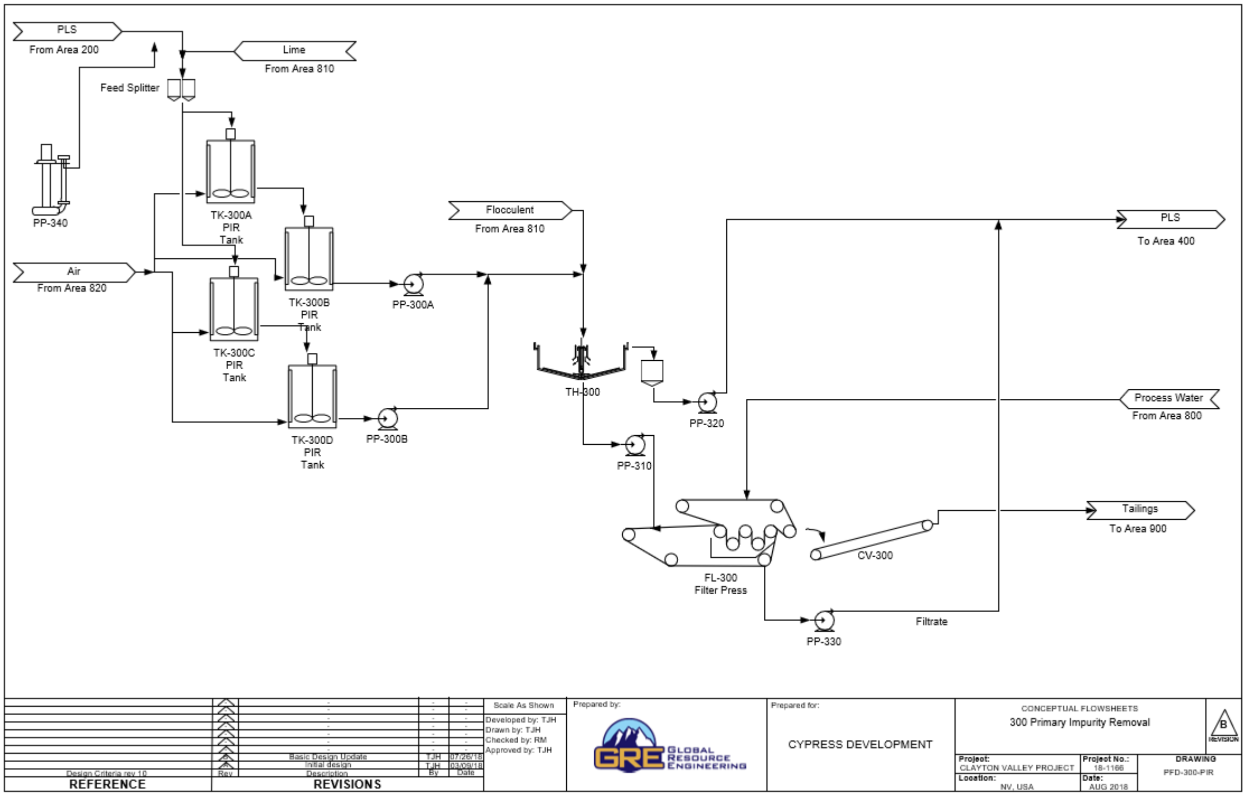


17.3 Primary Impurity Removal (PIR)

The clean overflow from the first thickener or the pregnant leach liquor (PLS) is fed to the PIR circuit. The circuit is comprised of two sets of tanks in series with a total retention time of four hours per train.

Calcium hydroxide (quick lime) is added to the first tank in each train to raise the pH to a target of 6 to facilitate the precipitation of iron and aluminum compounds. Air is added to tanks to enhance the precipitation efficiency and strip off the evolved carbon dioxide. The treated slurry is pumped to a single thickener (39 meters diameter) combined with flocculant and allowed to settle. The target underflow solids concentration is 45%. The underflow is pumped to a series of belt filters where the filter cake is washed with process water prior to being discharged at 90% solids to a conveyor. The filtered solids are combined with the primary leach tailings and delivered to the tailings impoundment. The filtrate is combined with the clear thickener overflow solution to form the PIR PLS and advanced to the second purification stage. Figure 17-4 shows the PIR circuit.

Figure 17-4 Primary Impurity Removal (PIR)



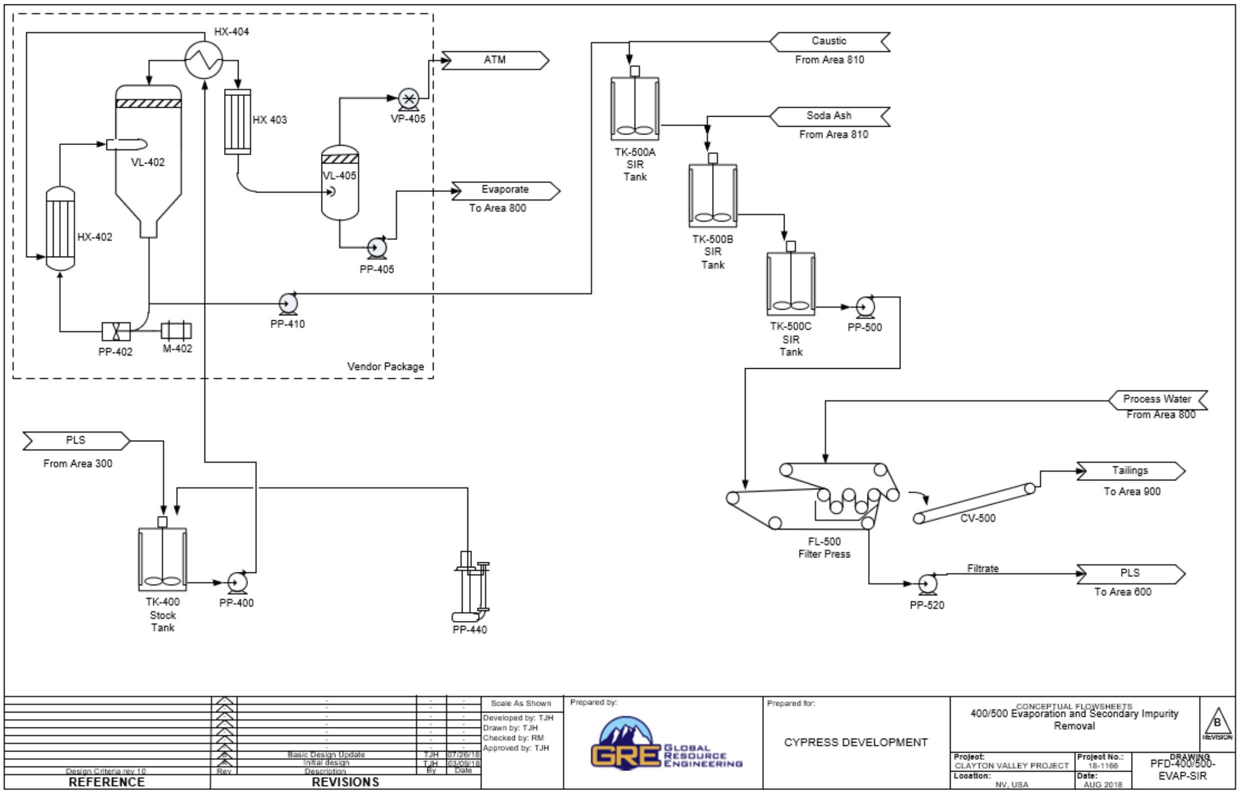
17.4 Secondary Impurity Removal (SIR)

The PIR PLS is pumped to a stock tank to provide buffer capacity within the circuit before being transferred to an evaporation circuit. Solution evaporation is achieved through the use of a Multi-Effect Evaporator. A 5-stage thermal-mechanical evaporation system is employed to provide a solution volume reduction of four times. This volume reduction is necessary to increase the PLS lithium grade to a concentration suitable for subsequent downstream treatment. The evaporate is collected and recycled as process water. The condensate is subsequently processed in the secondary impurity removal circuit.

The second purification stage is utilized to reduce the calcium, magnesium and manganese concentrations of the PLS through the stage-wise addition of sodium hydroxide and soda ash. The pH is first elevated to 9 and then to a final target of 10 in the second stage to facilitate precipitation of the impurities. The circuit consist of three tanks in series with a total retention time of four hours.

The resulting slurry is pumped to a series of filters to remove the precipitated impurities and the PLS advanced to a polishing circuit. The filtered solids are combined with the primary leach tailings and delivered to the tailings impoundment. The filtrate forms the SIR PLS and is advanced to the polishing circuit. Figure 17-5 shows the SIR circuit.

Figure 17-5 Secondary Impurity Removal (SIR)



17.5 Solution Polishing

The solution from SIR is pumped through a heat exchanger system to reduce the solution temperature to less than 60°C prior to being treated by an ion exchange system. The ion exchange system is designed to polish the PLS to remove additional calcium and manganese before final product production.

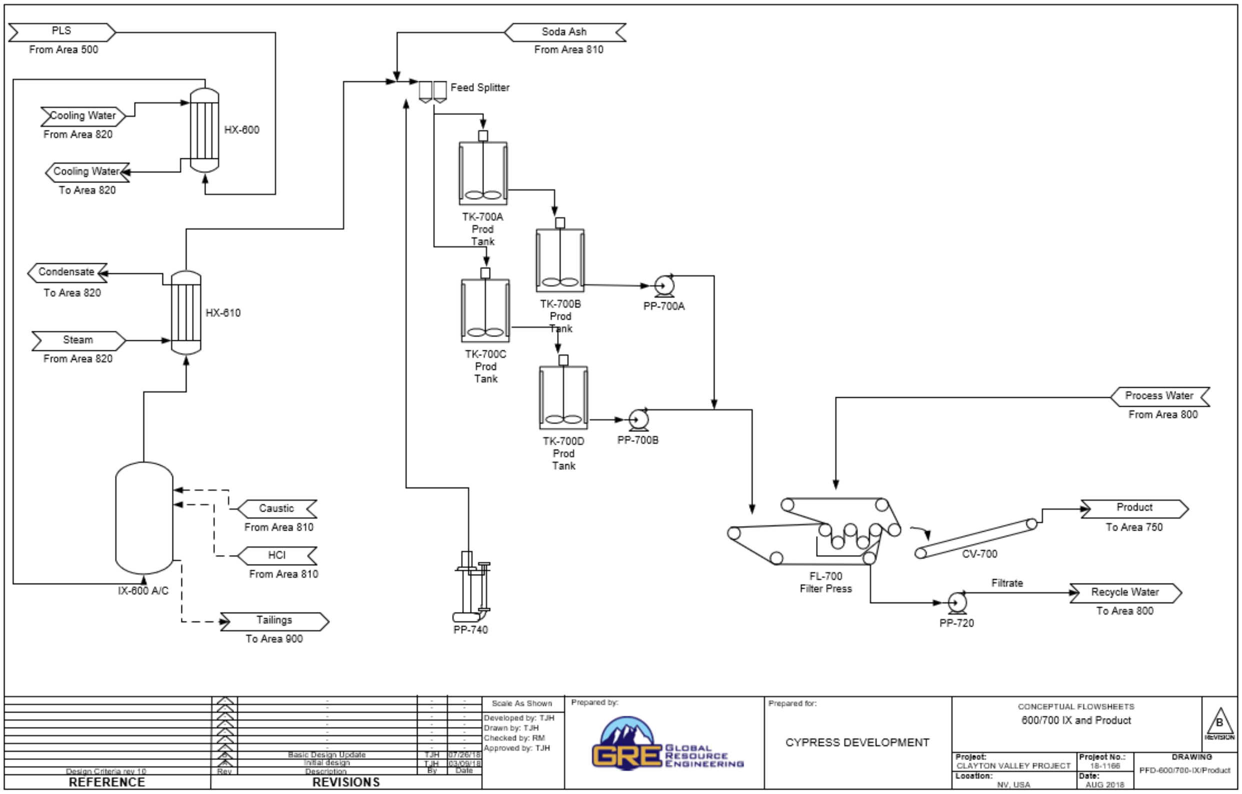
The ion exchange circuit consists of three columns to allow the adsorption/desorption/regeneration cycle to be conducted. Adsorbed impurities are stripped from the resin with dilute hydrochloric acid which is recycled to a circulating tank. The resin is rinsed with high purity water and then regenerated with dilute sodium hydroxide. The sodium hydroxide is recycled to a circulating tank. The resin is rinsed a final time with high purity water before being placed back in service. After a suitable number of cycles the spent reagents are combined and discharged to the evaporation pond.

17.6 Lithium Carbonate Production

The clarified and purified PLS would be pumped to the product precipitation circuit where the temperature would be increased to approximately 90-95°C and combined with soda ash. The precipitation circuit consists two-parallel trains of two tanks in series with a total of 6 hours of retention time per train. In this stage purified lithium carbonate precipitates from the PLS and is recovered by a final stage of filtration. The filtered and washed solids are conveyed to a drying circuit prior to being packaged for sale. The target is to produce a lithium carbonate product of 99.5% purity.

Drying of the lithium carbonate precipitate takes place in an indirect fired rotary dryer maintained at 120°C. The dried product is conveyed to a packing system where 1-tonne bulk bags are filled and sealed. Figure 17-6 shows the polishing and product production circuits.

Figure 17-6 Solution Polishing and Lithium Product



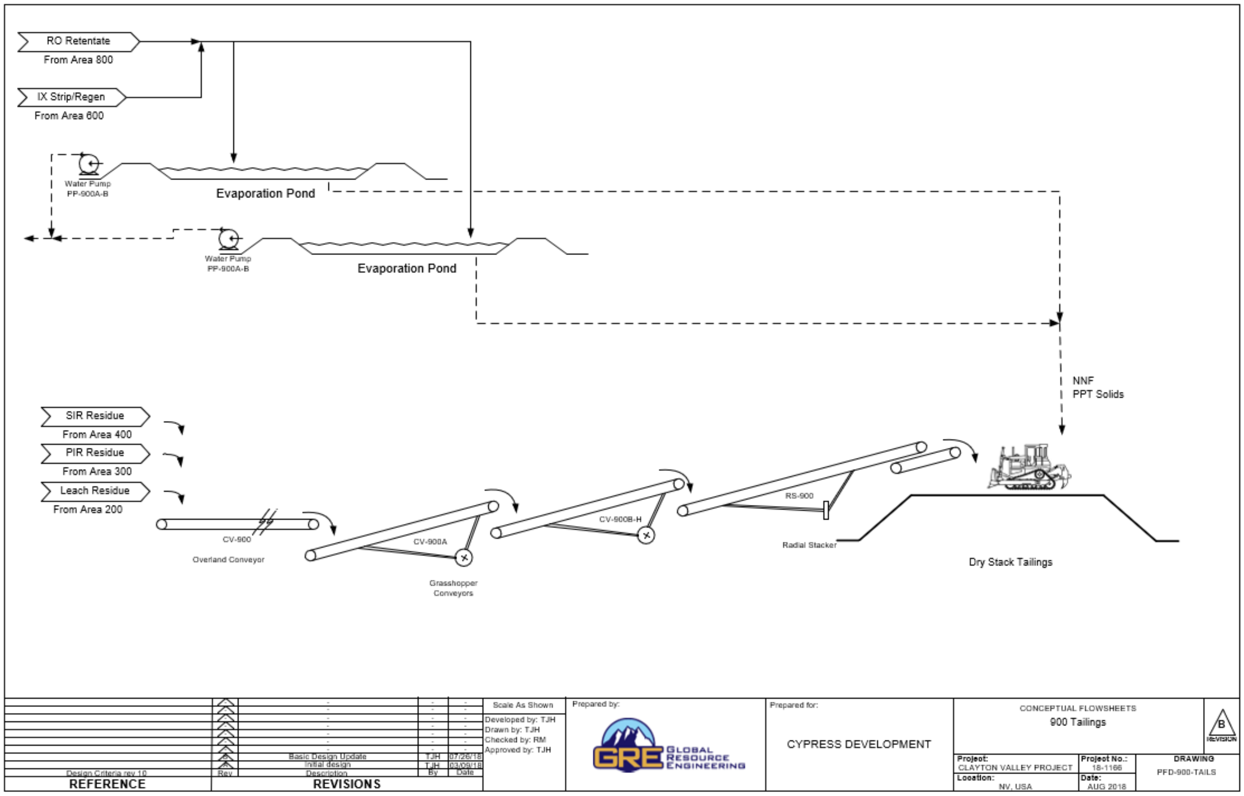
17.7 Tailings

The filtered and washed primary leach residue, PIR residue, and SIR residue are to be combined and placed in a dry-stack tailing impoundment. Additional lime may be added to ensure complete stability of any residual dissolved species before final stacking. After the initial tailings facility is filled to capacity, subsequent tailings will be placed in mined out portions of the pit.

The materials are conveyed via an overland conveyor to the impoundment area. A series of grasshopper conveyors transport the material to a slewing stacking conveyor for placement. A dozer will be utilized for final spreading and contouring.

Barren leach solution will be pumped to a reverse osmosis system for water recovery with the retentate being pumped to an evaporation pond to allow potassium and other salts to crystalize. These salts may be recovered for subsequent sale or combined with the other tailings products in the impoundment. Figure 17-7 shows the initial tailings handling.

Figure 17-7 Tailings Handling



17.8 Utilities

17.8.1 Acid Plant

The sulfuric acid plant envisioned for this project is Double Contact Double Absorption (DCDA) sulfur burning acid plant with an energy recovery system. The plant is capable of producing 2,000 tonnes per day of sulfuric acid (100% purity basis) by combusting elemental sulfur. The sulfur is designed to be delivered in the molten state by trucks.

The acid plant combusts elemental sulfur to produce sulfur dioxide gas, the gas is converted to the sulfur trioxide through catalysis and reacted with water to form concentrated sulfuric acid. The combustion of the sulfur produces significant heat that can be captured in a boiler to produce steam and electricity. The use of a backpressure turbine system optimizes the production of electricity and steam for the process. The acid plant has the ability to produce up to 25 MW of electricity but at additional capital expense so at this stage only enough electricity will be generated to allow the acid plant to be electrically self-sufficient.

Spent gas from the acid plant is piped to carbonate scrubbing where residual sulfur dioxide and acid mist are removed to less than US EPA Prevention of Significant Deterioration (PSD) emission limits before the gas is discharged to the atmosphere.

17.8.2 Water Treatment

Barren leach solution is to be treated in a reverse osmosis (RO) plant for water recovery. It is anticipated that approximately 60% of the water can be recovered through the RO system. This high purity water will be utilized for acid plant boiler water, reagent makeup, filter cake washing and ion exchange rinsing. Excess water will be combined with process water for general site usage.

Process water will be delivered via a dedicated pipeline from a well field approximately 10 km away. Process water will be stored in a process water tank and distributed to the required unit operations as required. Approximately 345 m^3/hr of fresh water is required by the project.

17.8.3 Reagents

The reagents area will be centralized to facilitate delivery, make up and storage. The reagent area consists of two thickener flocculation packages, hydrochloric acid, caustic, soda ash, lime and sulfuric acid systems.

18.0 PROJECT INFRASTRUCTURE

18.1 General Infrastructure

18.1.1 Existing Installations

There is currently no existing infrastructure at the site.

18.1.2 Access Road

The Clayton Valley project's location is 220 miles south of Reno, Nevada. The regional gold mining town of Tonopah is 40 miles northeast of the project and the small community of Silver Peak lies 10 miles west of the project. The project lies entirely within T2S, R40E, Mt. Diablo Meridian. The project is accessed from Tonopah, Nevada, by traveling south on US Highway 95, then west on Silver Peak Road. The existing access road is a two-lane road and is sufficient for current exploration and preliminary construction activities. For major construction and operations, road improvements, including road widening will be required.

18.1.3 Project Buildings

Buildings and facilities for the project and operations have been considered. Buildings required include the administration and mine offices buildings, process plant, laboratory, site gate house, reagent storage facility, mill shop and truck shop and warehouse building. A refueling and lubrication area will also be included.

18.1.4 Administration & Mine Offices Buildings

The administration and mine offices building will consist of a modular office complex 48' by 60' and is sized to accommodate approximately 30 persons, including private and open office spaces and a conference room.

18.1.5 Laboratory Building

The laboratory includes a finished steel building with a foot print of approximately 3,000 ft^2, dust collection system for the sample preparation area and a ventilation system with a wet scrubber for the wet lab area. The laboratory will include sample preparation, ICP spectroscopy, particle size distribution analyses, metallurgical testing (leach and precipitation testing) and personnel offices. Laboratory metallurgical chemical wastes will be stored temporarily on site. The laboratory is designed to process 200 solids samples daily.

18.1.6 Gate House

A gatehouse will be located at the entrance to the mine site. The gate house will include a reception desk and waiting area.

18.1.7 Reagent Storage Facility

The reagent storage area includes separate tanks for storing each liquid reagent. Each tank is installed within a concrete containment area sized for containment of 110% of the respective tank volumes. Additional space is provided for lime and soda ash silos.

18.1.8 Mill Workshop / Warehouse

The mill workshop and warehouse building will be will be approximately 6,500 ft^2. This building will be located adjacent to the process plant and will include a tool room, offices, a meeting and break room, and a bathroom.

18.1.9 Mine Truck Shop

A minimal truck shop has been included for each mine production scenario.

18.1.10 Fuel Storage & Dispensing

The diesel fuel storage and dispensing area will service heavy and light vehicles for the mine and process equipment. Diesel fuel will be delivered to the mine site via tanker trucks and stored in storage tanks. The fuel storage facility has a storage capacity of 10,000 gallons of fuel and is equipped with all necessary fuel dispensing equipment.

All fuel storage tanks will be placed in concrete containment with capacity to hold 110% for the fuel storage tank volume to assure no fuel is leaked to the environment.

18.1.11 Process Plant Building

A mill building will be constructed to cover the leach and purification. The main building will consist of a corrugated steel roof with open sides, includes a 10-ton overhead crane for maintenance activities and is sized at approximately 8,000 ft^2. The thickeners and evaporation systems are located outside with only minimal coverage of critical control elements. The acid plant has an enclosed control room with the balance of the plant being shielded from the elements only in critical areas.

18.1.12 Security and Fencing

Access to the site will be limited by fences around the process areas. A security gate and gate house are also included at the project site entrance and will be manned 24 hours per day.

18.2 Power Supply & Communication Systems

18.2.1 Power Supply

Electrical power for the Project will be supplied using line power. The plan is to receive retail service to the project by NV Energy.

The small emergency backup power generation for critical process equipment has been provided. The primary equipment requiring backup power are the thickeners, mine transfer pumps and acid plant.

18.2.2 Site Power Distribution & Consumption

Power distribution will be at 13.8 kV, 3 Ph, 60 Hz and will be further stepped down to 4,160 V, 480 V, 220 V and 120 V accordingly. Large operating motors will be supplied power at 4,160 V and smaller operating motors will use 460 V. Electrical outlets will be 120 V.

The estimated power attached power and consumption by area is presented in Table 18-1.

Table 18-1 Cypress Power Demand

Area	Installed KW	Load KW
Feed Prep	1,896	1,286
Leach/PIR/SIR	5,315	3,705
Acid Plant	1,492	1,194
Utilities	3,480	2,561
Total	12,183	8,747

18.2.3 Communication Systems

The site will be connected to the local phone and internet data network using a microwave or other through-the-air method.

18.3 Water Supply and Distribution

18.3.1 Water Balance

A water balance model was prepared by GRE and is discussed in greater detail in Section 17 of this Report. The water balance considers the Project's water demand, water collected from direct precipitation and seasonal evaporation. Additional water consumption allowances were included for road dust suppression (100 gpm), leach residue moisture loss (360 gpm), and miscellaneous uses (15 gpm). Based on the water balance model plus these allowances, makeup water requirements average 1000 gpm.

18.3.2 Site Water Management

Site water requirements will be met by a well field located 10-20 kilometers from the project.

18.3.3 Fire Water & Protection

A dedicated water system will be installed to provide fire protection to all areas of the project site.

18.4 Sewage & Waste

18.4.1 Effluents

Other than treated effluent from the site septic systems, the project will have no water discharge to the environment.

18.4.2 Sanitary Waste (Sewage)

Lavatory and wash facilities will be located throughout the project site. Sanitary waste from the lavatories will flow by gravity to multiple septic systems for treatment and disposal. Each septic tank and drainfield are sized for the building occupancy.

18.4.3 Solid Waste

Solid waste will be managed in dumpsters or other appropriate waste containers. All containers will be covered (or covered and weighted, if covers are not attached) to reduce the potential for blowing trash and to prevent access by wildlife. Containers used on site will be labeled. Trash from office and lunch areas will be bagged.

A licensed waste management company will transport collected waste to a dedicated offsite, third party-controlled landfill site. On-site burning of any waste materials, vegetation, domestic waste, etc. will not be allowed.

18.4.4 Hazardous Waste

Hazardous waste will be placed in drums, put on pallets, and stored in secure, impermeable, and appropriately sized containers, providing the required secondary containment, until being hauled offsite by a licensed contractor. Hazardous waste will be disposed of in a safe and environmentally sound manner using outside contractors.

19.0 MARKET STUDIES AND CONTRACTS

Benchmark Minerals Inc. (Benchmark) was contracted to prepare a report on lithium supply, demand, and independent price forecast.

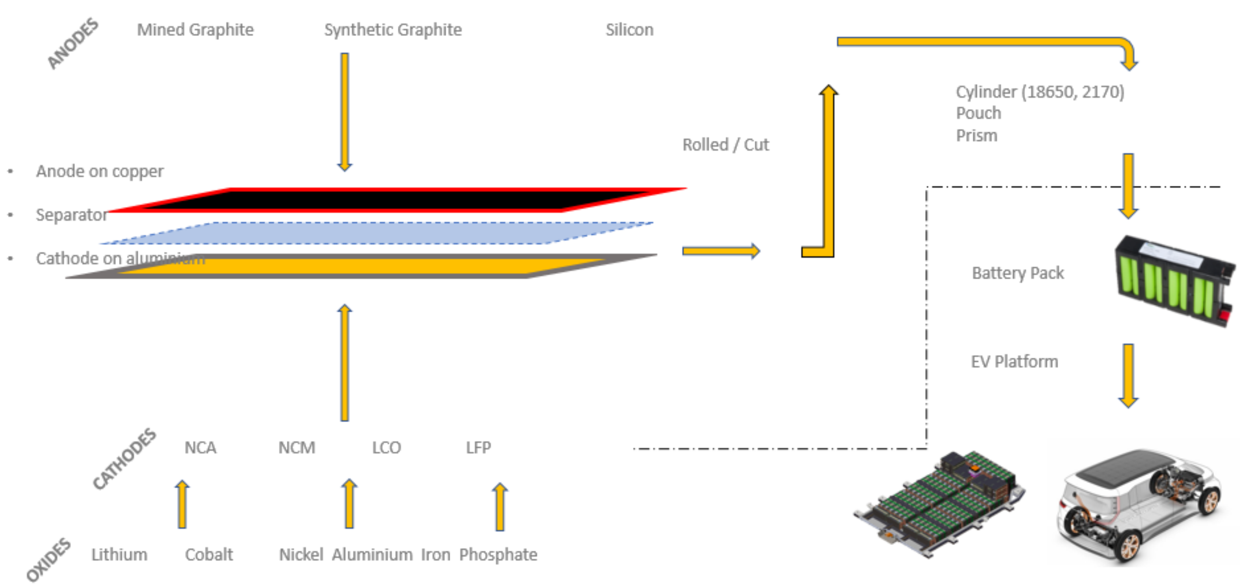
Benchmark is a leading provider of price assessments for the lithium industry and regularly produces bespoke forecasts for use in finance raising, as well as input into scoping studies, pre-feasibility and bankable feasibility studies.

19.1 Lithium-ion Supply Chain Overview

Benchmark examined the current and future supply chain for lithium, with a focus on the burgeoning li-ion battery market. The lithium market is set to grow sharply in the coming years as the mineral is critical for use in battery technologies employed in electric vehicles (EVs), grid storage and portable electronic equipment. As such there is a requirement for new supply to come online over the coming decade to meet this increased demand.

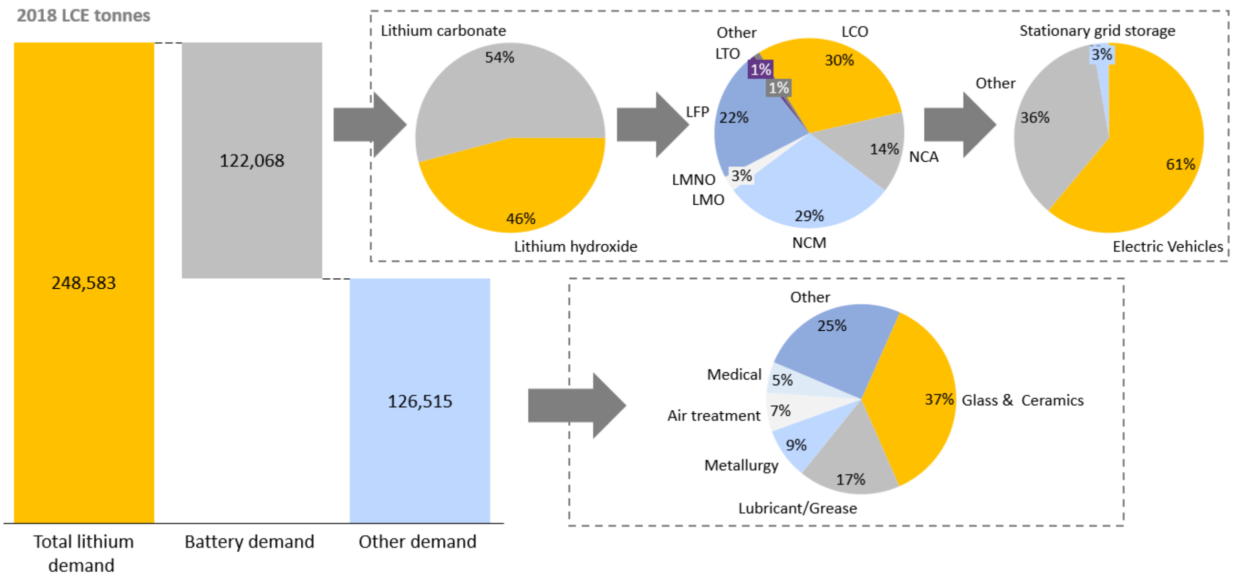
The graphic below (Figure 19.1) outlines the supply chain for lithium-ion batteries from mine to market. Lithium is an indispensable input for the development of lithium-ion batteries, unlike other constituent materials of the cathode which can be substituted or used in greater or lesser quantities based on the battery chemistry.

Figure 19-1 Lithium-ion Supply Chain



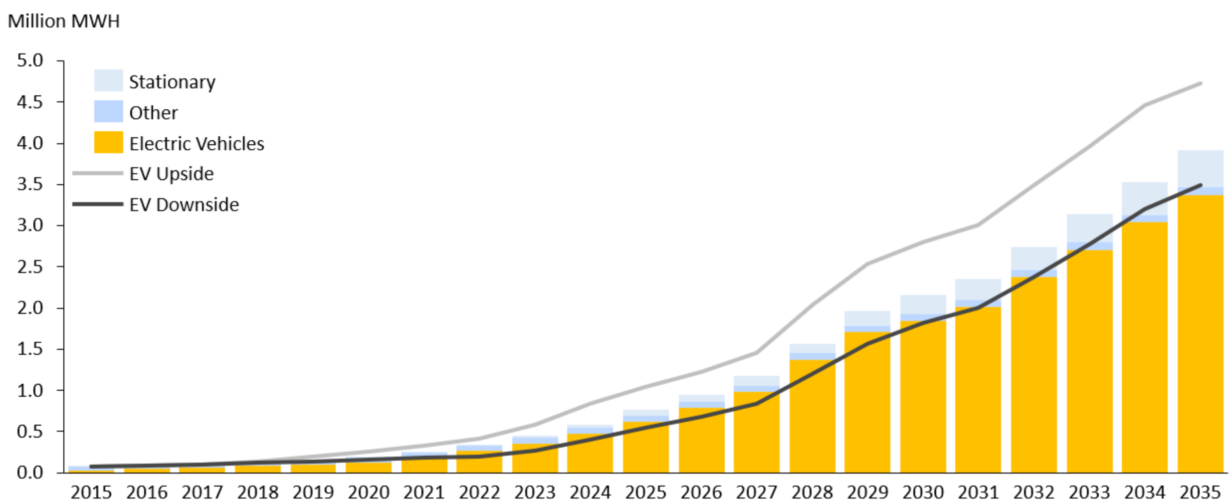
19.2 Lithium Demand

Figure 19-2 Current Lithium Supply and Breakdown of Demand by End Use



The chart below (Figure 19.3) outlines the key drivers of demand for lithium-ion batteries over the forecast period. As can be seen the major growth area is for EVs, followed by stationary (grid) applications. In Benchmark's base case scenario, demand of approximately 135,000 MWH is expected in 2018, reaching 760,000 MWH by 2025, and 4M MWH by 2035.

Figure 19-3 Lithium-ion Battery Demand by End Use Sector to 2035



19.3 Lithium Supply

Figure 19-4 Brownfield Lithium Capacity Forecast to 2035

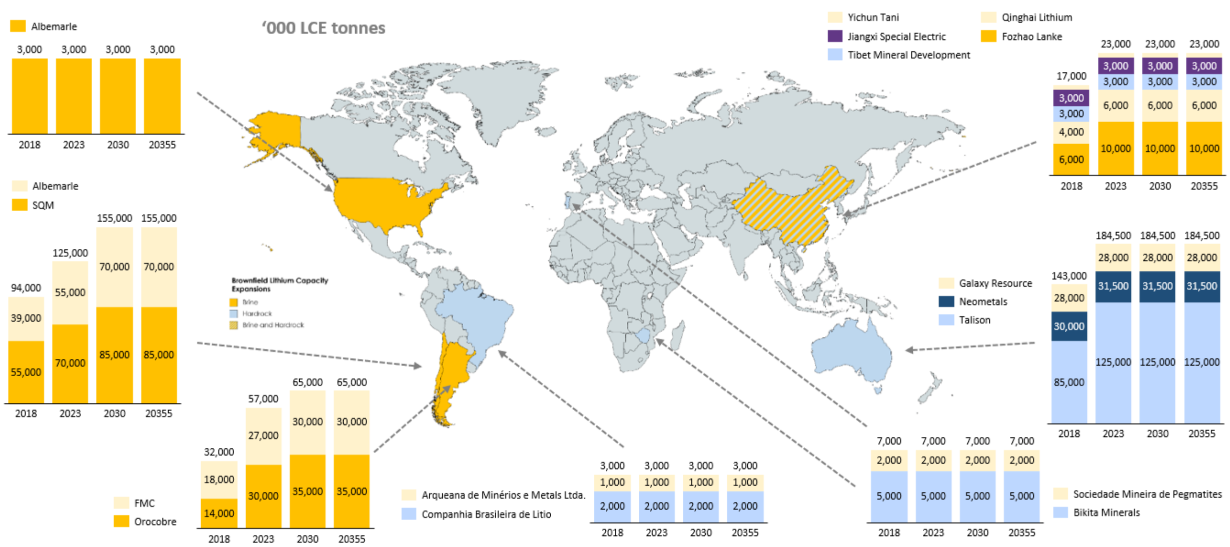


Figure 19-5 Greenfield Lithium Capacity Forecast to 2035

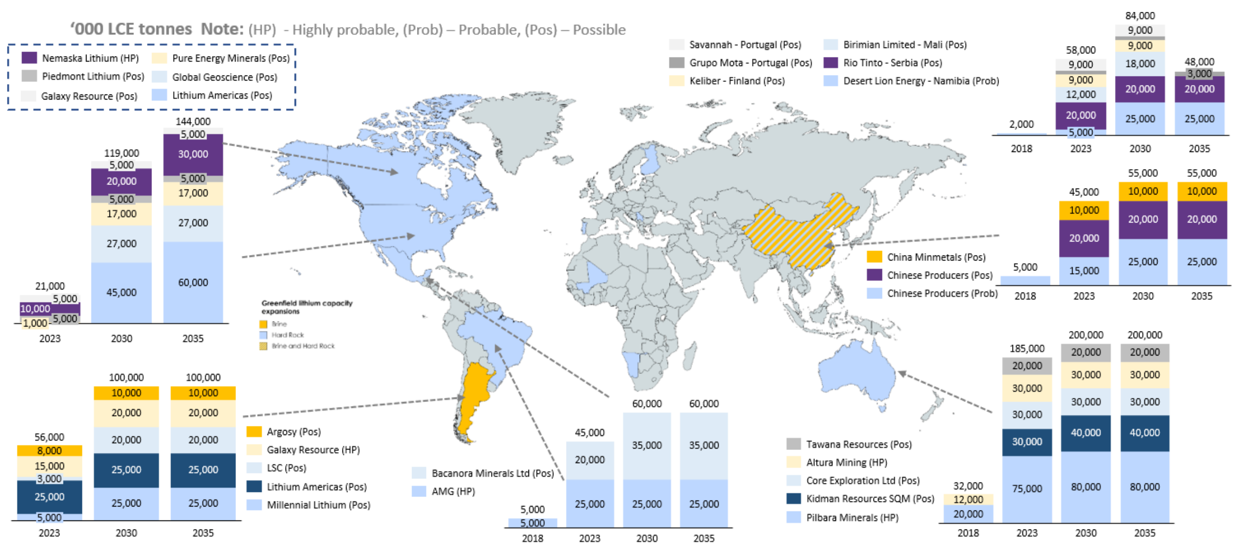
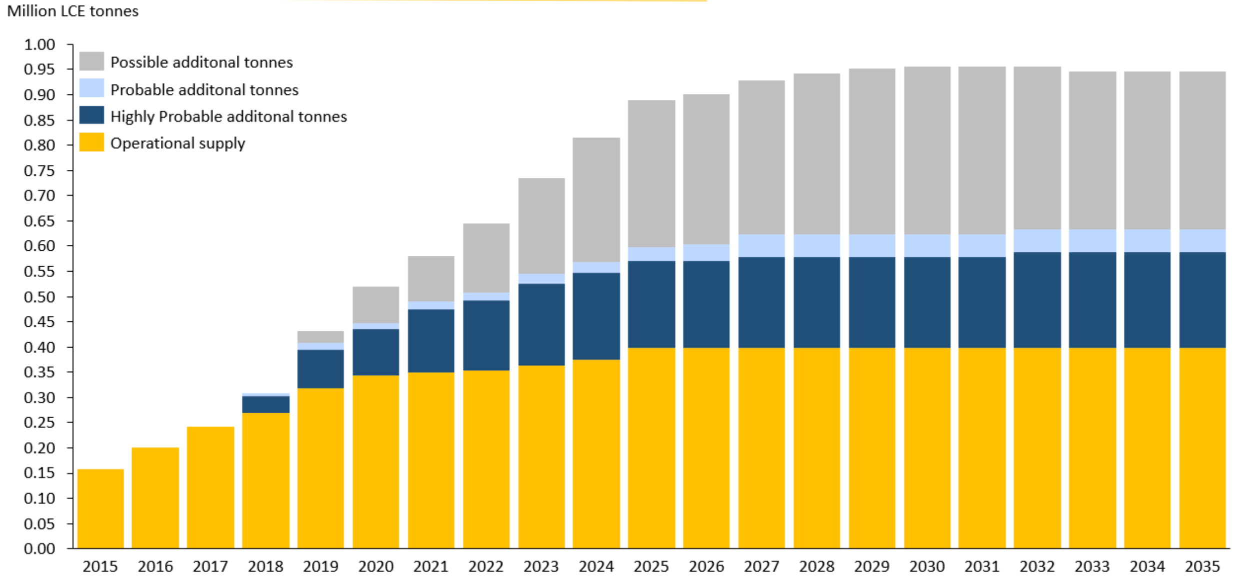


Figure 19-6 Lithium Capacity Forecast to 2035



19.4 Lithium Demand-Supply Balance to 2035

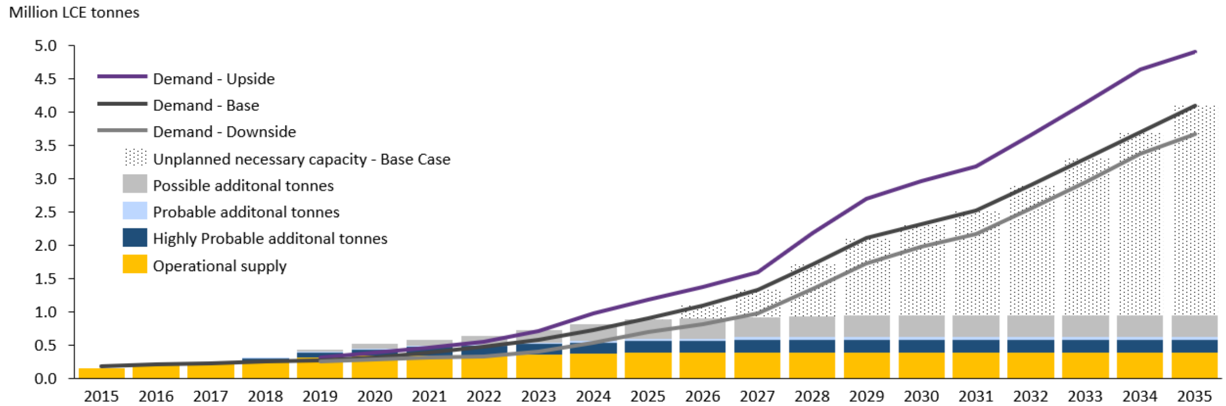
For the supply forecast Benchmark divided the forecast into three main phases, which reflect the development of the market over time.

Phase 1, 2015-2018: In this phase the supply-demand balance is very tight, with demand growing faster than new capacity expansions. New supply is largely from development of brownfield sites at operating producers

Phase 2, 2019-2025: New supply starts to come online from greenfield projects, as well as expansions at existing producers. The market moves into a period of relative oversupply by the end of the period

Phase 3, 2026-2035: Towards the latter part of the forecast period, there is a marked requirement for further as yet announced lithium capacity to come on-stream to meet rising demand. Prices are expected to remain in a range needed to stimulate this new investment, given that geological constraints are not an issue.

Figure 19-7 Lithium Demand-Supply Balance, 2015 - 2035



19.5 Lithium Price Forecast to 2035

19.5.1 Lithium Price Forecast Methodology

Benchmark's medium and long-term price forecast methodology for lithium considers the following factors:

Balance of supply and demand – Based on analysis of demand over time and understanding of the pipeline of new greenfield and brownfield capacity, Benchmark is able to make an assessment of the extent of over and under supply in the market over time, and how this is likely to impact prices.

Production costs for the marginal cost producer – Long run pricing in commodity markets is often determined by the level at which the highest cost producer needed to supply the market can continue to operate; for lithium this would be at a cash cost level of around USD8,000 per tonne LCE. For the forecast, Benchmark expects this will be less of a factor, and due to the ongoing need to incentivize new projects the price will be well above this level.

Incentive pricing for new greenfield and brownfield capacity investment – There will be an ongoing requirement for new greenfield capacity over the course of the forecast period. Benchmark conducted an Initial Rate of Return (IRR) analysis for a 'Typical' greenfield lithium project, which suggests that at a price level of USD13,000 per tonne LCE the IRR would be 35%. This is approximately the level that junior miners are using for their assessment of project economics, and reflects the fact that as the lower cost new supply comes online there will be a need for the development of higher capex projects over time.

Benchmark divided the forecast into three main phases, which reflect the development of the market over time. These phases are:

Phase 1, 2015 - 2018: Lithium prices have risen sharply in the period since 2015 on the back of rising demand for battery raw materials and a number of years of tight supply. There has also been some upward momentum in pricing from speculative buying on the back of a perceived ongoing supply shortage.

Phase 2, 2019 - 2025: Rapid price growth will lose momentum post 2018 as new capacity becomes available and market tightness eases. Nevertheless, prices in period 2019-2022 are expected to be maintained at high levels. By 2022 the market is expected to be in a pronounced over supply, and Benchmark forecasts that a price correction will begin at this point.

Phase 3, 2026 - 2040: the market is expected to begin to tighten again in the period to 2030, and that prices will rise in this period. Benchmark expects that a pipeline of new currently unannounced projects will begin to come through over the coming decade to meet this demand, and that ultimately prices will settle into long term average of $13,000 per tonne for lithium carbonate.

Figure 19-8 Lithium Carbonate Battery Grade Price Forecast

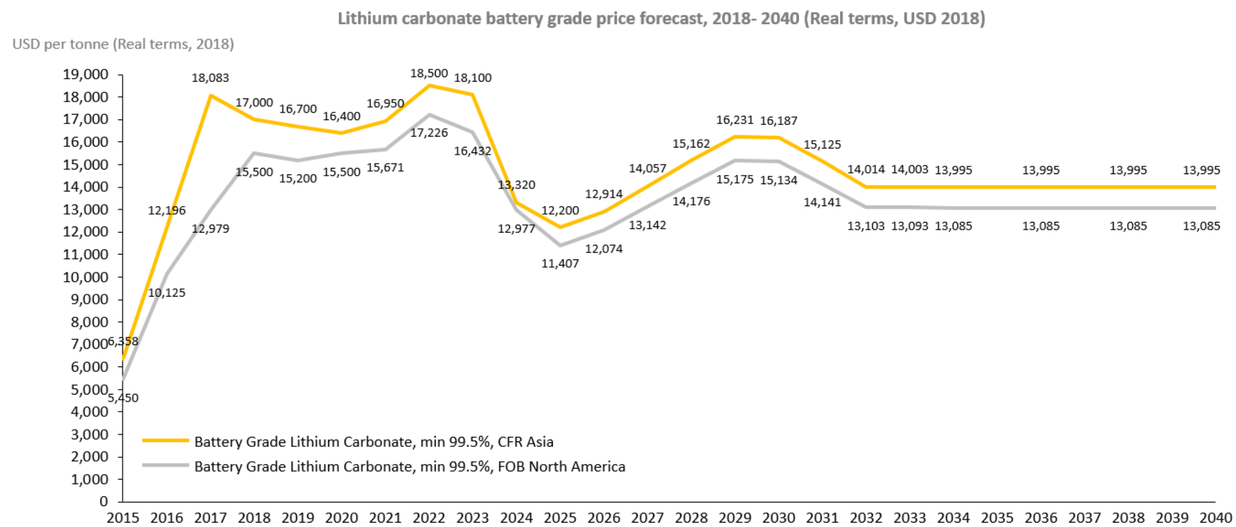
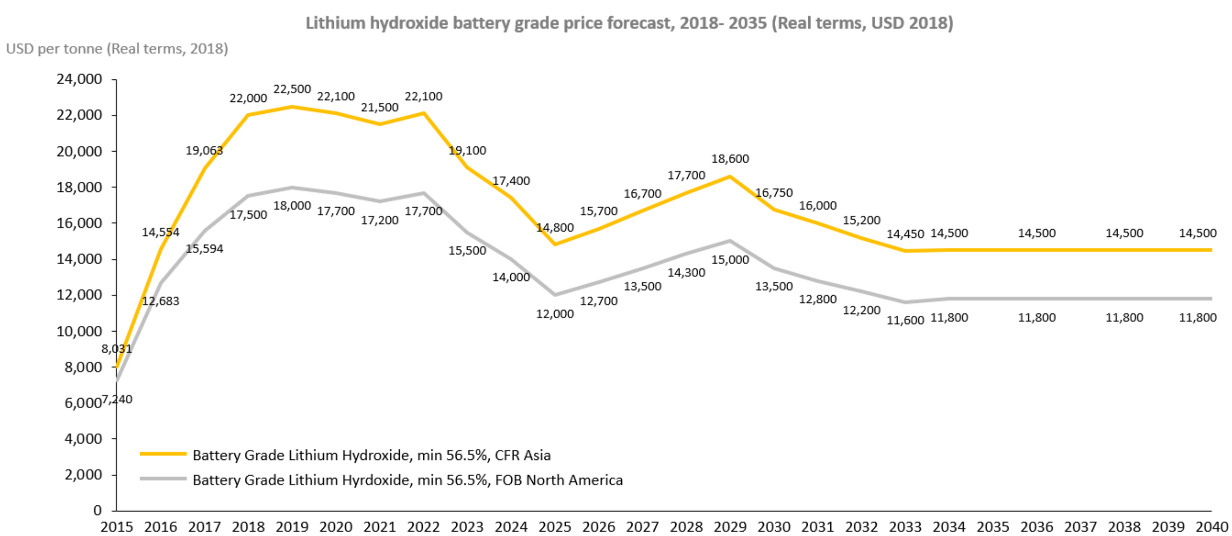


Figure 19-9 Lithium Hydroxide Battery Grade Price Forecast



The BMI price forecast has lithium carbonate prices of $15,500 to $17,200/tonne through 2023, followed by a decrease to $11,400/tonne in 2025. The price then begins to rebound to $15,200/tonne in 2030 and stabilize at a long term price of $13,085/tonne through 2040.

19.6 Base Case Lithium Price and Contracts

Based upon the Benchmark study, GRE has chosen a base case lithium carbonate price of $13,000/tonne for this PEA. As of the Effective Date of this PEA, Cypress has entered no contracts in the way of offtake agreements, streaming royalty obligations, materials or supplies, or other substantive agreements related to commodity sales, purchases or marketing.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Cypress has not completed any environmental studies relevant to this report's content nor has it undertaken any studies with respect to any social or community impacts that would relate to its past exploration at the project or to any further exploration it might carry out pursuant to recommendations contained in this Report.

Cypress has indicated that it does routinely apply for and receive notice-level permits from the BLM to carryout current activities on the project. Cypress is currently in compliance with all local and federal regulations and requirements relating to its activities on the project.

Under federal regulations and requirements, Cypress will need to carry out appropriate environmental, social, or community impact studies or acquire any related permits, permissions, or agreements to continue work on the project pursuant to recommendations contained in this Report. Cypress anticipates that the detailed study of multiple environmental aspects of the project will be necessary. This is normal for a project as it passes from initial exploration to more advanced stages.

Cypress has conducted all its activities at the project in accordance with environmental standards and compliance requirements and is not aware of any environmental issues related to its activities at the site. Cypress is also committed to conducting its project advancements with best management practices and to maintain an excellent reputation within the local communities the project may have an impact upon.

20.1 Permitting

The Project is subject to no known environmental liabilities. There are no mine workings, rock piles or tailings of significance within Cypress' claims.

Various permits and plans are required to meet and maintain regulatory compliance. Environmental permitting requirements for the Project are expected to be similar to other mines in Nevada. Permitting includes consideration of reclamation, surface water, groundwater and air pollution prevention plans, and other items common to mining operations in the State of Nevada. Permits and plans will include all applicable monitoring, reporting schedules, bonding and fees. Such plans and permits are expected to include the following in order of importance:

• Plan of Operations (POO), State of Nevada and U.S. National Environmental Policy Act (NEPA) compliance

• Use of BLM-Administered Land, Compliance with Title 43 Code of Federal Regulations (CFR) Subpart 3809 Surface Management

o Environmental Assessment (EA), or

o Environmental Impact Study (EIS)

• Mining Reclamation Permit

• Water Pollution Control Permit

• Stormwater NPDES General Permit

• Activities in Wetlands or Waters of the U.S.

• Air Quality Operating Permit

• Permit to Appropriate Public Waters

• Industrial Artificial Pond Permit

• Hazardous Materials Permit

• Fire and Life Safety

• General Local Permits

These permits are not obtained at this time and specific reporting and planning requirements will be identified through the permitting process.

20.2 Baseline Studies

Cypress has not yet initiated environmental studies with regard to the potential development of the Project. A variety of permits will be required from Federal, State, and county agencies for the Project as listed in Section 20.1 above. In order to secure these permits, data from numerous disciplines have been collected to assist with mine development, operations, and closure planning. This information will be included with ongoing studies. The following outlines the studies that will be needed for permitting.

- Vegetation Baseline Report
- Wildlife Baseline Survey and Threatened, Endangered, Sensitive, and Candidate Wildlife Species Survey
- Soils Literature Review
- Jurisdictional Waters Review and Seep and Spring Survey
- Monitoring wells and quarterly ground and surface water quality sampling
- Waste rock characterization and process leach residue characterization studies
- Archeological study

21.0 CAPITAL AND OPERATING COSTS

Capital and operating costs for the project were estimated based of a developed equipment list using InfoMine USA, Inc. (InfoMine) resources, including *Mining Cost Service 2017* (InfoMine, 2017a) and *2017 Mine and Mill Equipment Cost Guide* (InfoMine, 2017b), both copyright 2017 by InfoMine. For equipment outside of the scope of InfoMine, the cost estimation was conducted by vendor estimation or scaling costs for similar equipment from other projects, and GRE's inhouse data.

GRE has assumed the project is constructed over a two-year period, and has evaluated project economics at the time of a construction decision.

The project will likely require a year for additional drilling and metallurgical test work, project optimization and design, and three to four years for permitting. The timeline to production is five to six years with an active development program.

21.1 Capital Costs

The total initial capital cost estimate is $482 million distributed over two years of pre-production. An overall factor of 2.86 on equipment costs is used to allow for the necessary installation labor, construction materials, spares, first fill, buildings, and engineering and construction management. Infrastructure and G&A capital includes allowances for feasibility study, permitting, bonding, off-site electrical, and acquisition of process water. A summary of the mine capital costs are displayed below in Table 21-1.

Table 21-1 Mine Capital Costs Summary

Capital Cost	(USD Millions)
Mine development and equipment	24.3
Plant feed prep, leaching, purification and lithium recovery	319.2
Laboratory Equipment	0.5
Facilites	4.6
G&A capital	33.3
Direct Capital Costs	**381.9**
Working capital	23.9
Contingency (20% of Direct Costs)	76.4
Indirect Capital Costs	**100.3**
TOTAL CAPEX	**482.4**

The plant and G&A capital costs are split between years -2 and -1, and the mine capital costs occur during year -1. Recurring costs are scheduled to occur throughout the schedule as needed.

The following mining, processing, and G&A items were included in the capital cost estimate.

- Mining production equipment required includes two 994 loaders (one to be on standby), two D10 dozers (one to be on standby), and one 777G haul truck.
- Mining support equipment that will be needed at the mine includes the following:
    - one D8 size dozer

  - one 844 size rubber tired dozer
  - one 992 size loader
  - one 150-hp grader
  - one 5000-gallon water truck
  - service/tire trucks
  - light plants
  - submersible pumps
  - pickup trucks
- Mining facilities required include a heavy equipment shop, a fuel station, and a changing facility.
- The Process Plant includes the following unit operations:
  - Feed Preparation
  - Lithium Extraction
  - Primary Impurity Removal
  - Secondary Impurity Removal
  - Solution Polishing
  - Lithium Carbonate Production
  - Tailings Impoundment
  - Utilities – Acid production, water recycle, reagents
- The laboratory includes the following equipment items:
  - Jaw Crushers
  - Pulverizers
  - Dust Enclosures
  - Compressor
  - Dust Collector
  - Sample Splitters
  - Balances
  - ICP
  - Fume Hoods
  - Drying Ovens
  - Digestion Blocks
  - Miscellaneous – glass, titration, etc.
- Mobile equipment includes a tracked dozer for waste and tailings impoundment operations.
- G&A items include:

    - Survey
    - Guard house/security
    - Startup training
    - Emergency vehicle/supplies
    - Office
    - Warehouse
    - Fire Protection
    - Water supply
    - Power line to the site, which was assumed to be approximately 11 km
    - Electrical substation and switchgear
    - Reclamation bond
    - Permitting
    - Exploration and Met testing
    - Feasibility Study
    - Closure
- Development included pioneering, clearing, grubbing, access road improvements, and haul road construction, assumed to be 5,000 meters of new haul roads.
- First Fills: Materials and reagents needed for initiating mining and production were estimated based on the quantity needed during the first quarter of operation.

Working capital was estimated to be three months' operating costs, which would be recovered the year after production ends, but because the schedule was limited to the first 40 years of operation, the working capital recovery is not included. Sustaining capital was estimated as 10% of the mobile equipment cost per year Capital contingency was set at 20%.

The initial capital costs, occurring in year -2 and -1, total $481 million, and the total capital costs for the 40-year schedule are $600 million.

The capital costs are detailed in Table .

21.2 Operating Costs

Estimated operating costs are $17.50 per tonne of mill feed, or $96 million per year, including 10% contingency. Acid plant operations are the major component in the operating costs and account for more than half of the total. Project labor is estimated at 136 on-site employees. Connected power is estimated at 12 MW, with an all-in cost of $0.066 per KWH. The operating cost summary is displayed below in Table 21-2.

Table 21-2 Mine Operating Cost Summary

Operating Cost	$ per tonne of mill feed
Mining Op Cost	1.73
Plant Op Cost	15.09
G & A	0.68
TOTAL OPEX	**17.50**

Operating costs include manpower, mining equipment costs, processing equipment costs, reagents and consumables, and G&A operating costs.

- Mining production equipment hours were estimated from the equipment productivity estimates, the scheduled leach material and waste tonnage, and the number of pieces of equipment required.
- Mining support equipment hours were calculated from the number of pieces of equipment times the operating hours per day, assuming utilization of 90% and availability of 95%, times the operating days per year.
- Operating hours for the plant was assumed to be 24 hours per day, 7 days per week, for 52 weeks per year with an availability of 95%.
- Laboratory operating hours were set at 2 shifts per day, 8 hours per shift, and 260 operating days per year.
- Reagent usage was estimated from metallurgical test work and GRE experience.
- Manpower for the mine and processing facility includes hourly-rate employees and salaried employees, who are generally superintendents and professional personnel. The number of required equipment operators was estimated using the quantities of equipment required, the quantity of personnel per piece of equipment, and the number of shifts per day. Numbers of required processing and salaried personnel were estimated based on GRE's experience. A burden factor of 40% was added to all labor. The burden includes fringe benefits, holidays, vacation and sick leave, insurances, etc.
- Administrative operating costs include labor and services and supplies

Operating contingency was set at 10%.

Annual operating costs with contingency vary from $3.9 million to $95.7 million. Total operating costs for the 40-year schedule are $3.83 billion.

The operating costs for the 40-year production schedule are summarized in Table 21-3.

Table 21-3 Detail of Capital Costs (000's)

	Year -2	Year -1	Other Years	Total
Mine Capital				
Development				
Pioneering, Clearing, Grubbing, Access Road Improvements		$550		$550
Haul Road Construction		$1,800		$1,800
Mining Equipment				
Production Equipment				
Loader - 994k		$10,600		$10,600
Dozer - D10T		$1,670		$1,670
Truck - 777G		$1,350		$1,350
Support Equipment				
Dozer - D10T		$1,670		$1,670
Dozer - D8T		$918		$918
Dozer (rubber tired) - 844k		$1,790		$1,690
Loader - 992K		$0		$0
Grader		$1,944		$1,944
Water Truck		$631		$631
Service/Tire Truck		$170		$170
Light Plants		$74		$74
Pumps (submersible)		$46		$46
Pickup Truck		$231		$231
Other Mining Equipment				
Surveying Equipment		$42	$42k every 5 years	$378
Computers		$40	$40k every 4 years	$440
Operations Software		$50	$10k every year	$450
Planning Software		$80	$16k every year	$720
Geology Software		$50	$10k every year	$450
Maintenance Software		$100	$20k every year	$900
Dispatch System		$100	$20k every year	$900
Plotter		$5	$5k every 5 years	$45
Radios		$3	$3k every 5 years	$27
Total Mine Capital		**$25,265**	**$3,840**	**$28,104**

	Year -2	Year -1	Other Years	Total
Plant Capital				
Process Equipment				
Feed Preparation	$1,658	$1,658		$3,316
Lithium Extraction	$5,949	$5,949		$11,899
Purification	$21,290	$21,290		$42,581
Product Production	$1,187	$1,187		$2,374
Acid Plant	$18,333	$18,333		$36,667
Tailings	$4,420	$4,420		$8,839
Utilities	$3,017	$3,017		$6,033
Facilities				
Installation Labor	$32,461	$32,461		$64,921
Concrete	$4,149	$4,149		$8,297
Piping	$13,836	$13,836		$27,672
Structural Steel	$4,370	$4,370		$8,740
Instrumentation	$3,118	$3,118		$6,236
Insulation	$1,590	$1,590		$3,180
Electrical	$6,178	$6,178		$12,355
Coatings and Sealants	$543	$543		$1,087
Spares and First Fill	$5,000	$5,000		$10,000
Building	$11,710	$11,710		$23,420
Engineering/Management	$20,804	$20,804		$41,609
Total Process Equipment Capital	**$159,612**	**$159,612**		**$319,224**
Laboratory Equipment Capital Costs				
Jaw Crusher		$40		$40
Pulverizer		$80		$80
Dust Enclosure		$30		$30
Compressor		$5		$5
Dust Collector		$25		$25
Sample Splitter		$16		$16
Balance		$6		$6
ICP		$110		$110
Fume Hoods		$30		$30

	Year -2	Year -1	Other Years	Total
Drying Oven		$30		$30
Digestion Blocks		$30		$30
Misc - glass, titration, etc		$100		$100
Total Laboratory Equipment Capital		**$502**		**$502**
Facilities Capital				
Mine Facilities				
Heavy Equipment Shop w/tools		$1,500		$1,500
Dry		$400		$400
Fuel Station		$150		$150
Engineering /Management		$513		$513
Infrastructure				
Power to Buildings		$200		$200
Potable Water to Buildings		$200		$200
Earth Works for Buildings		$1,000		$1,000
Security System		$250		$250
Engineering/ Management		$413		$413
Total Facilities Capital		**$4,625**		**$4,625**
G&A Capital				
Diff. GPS – Survey	$28	$28		$55
Guard House / Security	$55	$55		$110
Startup Training	$419	$419		$838
Emergency Vehicle/Supplies	$55	$55		$110
Office	$175	$175		$350
Warehouse	$260	$260		$520
Fire Protection	$250	$250		$500
Water Rights	$2,500	$2,500		$5,000
Power line to site	$875	$875		$1,750
Substation (15 MW)	$1,000	$1,000		$2,000
Electrical Switch Gear	$150	$150		$300
Reclamation Bond	$1,875	$1,875	$300k every years for 39 years	$15,150
Permitting	$2,000	$2,000		$4,000
Exploration and Met Testing	$1,000	$1,000		$2,000

	Year -2	Year -1	Other Years	Total
Feasibility Study	$1,000	$1,000		$2,000
Closure	$5,000	$5,000		$10,000
Total G&A Capital	**$16,641**	**$16,641**	**$11,400**	**$44,683**
Working Capital		$23,915		$22,223
Sustaining Capital		$176	$4,886k every year	$84,354
Contingency	$35,251	$41,164	$55,134	$96,298
Total Capital Costs	**$211,504**	**$270,900**	**$119,301**	**$601,705**

Table 21-4 Summary of Operating Costs (000's)

	Year -1	Typical Year	Total
Mine Operating			
Production Equipment	$621	$4,708	$188,922
Support Equipment	$87	$1,048	$41,992
Mine Labor	$238	$2,853	$114,359
Total Mine Operating Costs	**$946**	**$8,608**	**$345,273**
Process Operating			
Plant Labor	$601	$7,201	$288,660
Power	$382	$4,575	$183,376
Reagents and Consumables	$1,333	$63,178	$2,528,466
Total Process Operating Costs	**$2,316**	**$74,955**	**$6,001,003**
Administrative Operating			
G&A Labor	$142	$1,700	$68,126
Services and Supplies	$142	$1,700	$68,158
Total Administrative Operating Costs	**$284**	**$3,400**	**$136,284**
Contingency	$355	$8,696	$348,206
Total Operating Costs	**$3,901**	**$95,659**	**$3,830,264**

22.0 ECONOMIC ANALYSIS

GRE has constructed a discounted cash flow economic model for the Clayton Valley lithium project. The model includes a two-year pre-production development and construction. The model evaluates the project on a standalone basis as if a decision to proceed were obtained following completion of a Feasibility Study. The time (3 or more years) and cost for exploration, engineering, and permitting and associated costs are not included in the model, but have been estimated separately and are included in the recommendation section of this report.

Recovery was set at 81.5% of the lithium tonnes processed, with production of 5.323 kg of lithium carbonate per tonne of contained lithium. Over the course of the 40-year schedule, there are 209.4 million kg of contained lithium, resulting in 170.7 million kg of recovered lithium and 909.2 million kg of recovered lithium carbonate.

22.1 Results

Economic analysis of the Clayton Valley Lithium project, at a lithium carbonate price of $13,000/tonne of lithium carbonate, over the 40-year schedule, projects an after-tax Net Present Value @ 6% (NPV@6%) of $1.97 billion, NPV@8% of $1.45 billion, and NPV@10% of $773 million, and Internal Rate of Return (IRR) of 32.7%. The expected maximum negative cash flow is $488 million.

The average estimated cash operating cost per tonne of lithium carbonate is $3,983.

An allowance for state property and income taxes of 7% was included, and Federal taxes were included at 21% for this evaluation. Depreciation and amortization, depletion, and loss carry forward were included.

Salient results for the project base case are shown below.

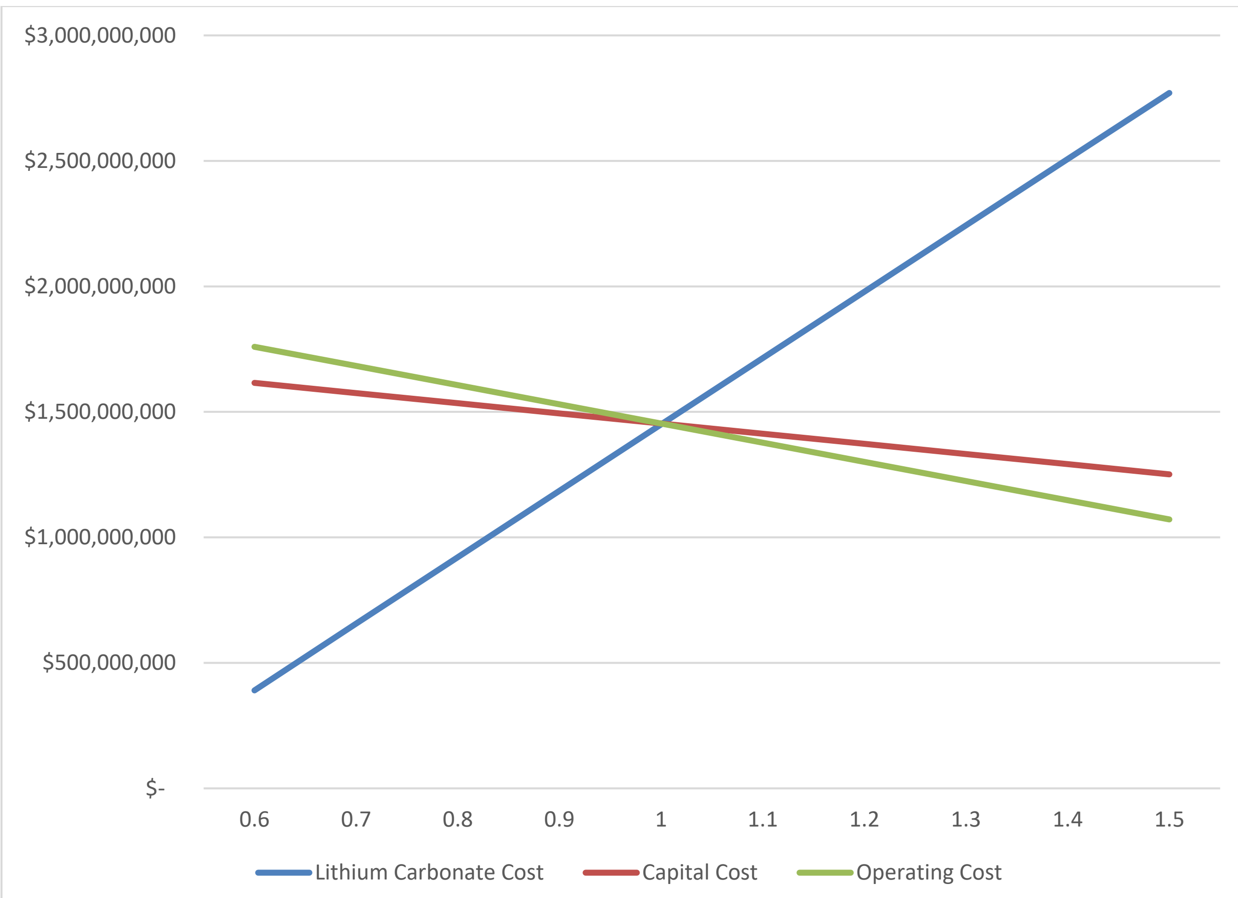
- Mining operating cost per process tonne of $1.73, including the strip ratio of 0.1:1.
- Process operating cost per process tonne of $15.09. Sulfuric acid accounts for 65% of the processing costs.
- G&A operating cost per process tonne of $0.68.
- Total operating cost plus contingency per process tonne of $17.50, which equates to a cost of $3,983/tonne of LCE.
- Total cash cost (with capital included) per tonne of lithium carbonate is $4,609/tonne of LCE.
- Average annual production of 24.0 million kg of lithium carbonate.
- $6.2 billion after-tax cumulative cash flow for the 40-year schedule.
- Payback period of 2.7 years and Payback multiple of 12.8.
- After-tax NPV of 1.45 billion @ 8% discount rate and IRR of 32.7%.

22.2 Sensitivity Analyses

GRE evaluated the after-tax NPV@8% sensitivity to changes in lithium carbonate price, capital costs, and operating costs. The results are shown in Figure 22-1. The base price used for lithium carbonate is $13,000/tonne LCE based on the Benchmark market study.

The after-tax NPV@8% is most sensitive to changes in lithium carbonate price, ranging from $390 million at 60% of the base case lithium carbonate price to $2.77 billion at 150% of the base case lithium carbonate price, or approximately $263 million per 10% change in lithium carbonate price. The after-tax NPV@8% stays positive for the full range of lithium carbonate prices examined. The after-tax NPV@8% is least sensitive to changes in capital costs, ranging from $1.62 billion at 60% of the base case capital costs to $1.3 billion at 150% of the base case capital costs, or approximately $4.5 million per 10% change in capital costs. The after-tax NPV@8% stays positive for the full range of capital costs examined. The after-tax NPV@8% is moderately sensitive to changes in operating costs, ranging from $1.8 billion at 60% of the base case operating costs to $1.07 billion at 150% of the base case operating costs, or approximately $7.9 million per 10% change in operating costs. The after-tax NPV@8% says positive for the full range of operating costs examined.

Figure 22-1 NPV@8% Sensitivity to Varying Lithium Carbonate Price, Capital Costs, and Operating Costs



23.0 ADJACENT PROPERTIES

The project is surrounded by valid mining claims held by several exploration and mineral production companies. The surrounding claims are 95% placer claims. A small group of valid lode claims exists on the northeast margin of the project. The project also directly adjoins fee simple patent private lands owned by Albemarle Corp., who is processing brines along the west boundary.

The property immediately to the south of the project, owned by Spearmint Resources, recently announced results of a first phase of exploration drilling, with lithium results as high as 1,670 ppm. Three holes were drilled, with lithium results ranging from 396 ppm to 1,670 ppm over 270 feet, averaging 835 ppm Li. Hole 2 had a range of 250 ppm to 1,570 ppm over 220 feet, averaging 642 ppm Li. Hole 3 had a range of 429 ppm to 1,280 ppm Li over 195 feet, averaging 772 ppm Li.

Noram Ventures Inc. has the property to the northeast and reported an inferred resource of 17 million tonnes grading 1,060 ppm lithium in a 43-101 Report dated July 24, 2017. In 2018, Noram reported five drill holes within its resource area had been deepened, encountering additional lithium mineralization.

Pure Energy Minerals has a brine resource to the west and southwest. Effective June 15, 2017, Pure Energy had a Mineral Resource Estimate of 5.24 million cubic meters inferred grading 123 mg/L containing 217,700 tonnes of LCE.

24.0 OTHER RELEVANT DATA AND INFORMATION

Section 27, References, provides a list of documents that were consulted in support of the PEA. No further data or information is necessary, in the opinion of the authors, to make the Report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

The project is a large lithium-bearing claystone deposit. The estimated resources in this report are open to depth and laterally in some areas. The lithium occurs in mineralization that is readily available for direct acid leaching.

A large higher-grade portion of the deposit is available for mine production over the first several decades of mine life. Many bulk tonnage mining methods appear to be applicable, and drilling and blasting is not anticipated to be required. Dozers, scrapers, surface planers, truck/loader, in-pit feeder-breaker/slurry pump are all viable methods. The base case selected for evaluation in this PEA uses the in-pit feeder-breaker/slurry pump method.

Preliminary metallurgical examinations indicate that the claystone responds well to conventional weak acid leaching with no upstream size reduction required. Initial results indicate that lithium extractions of greater than 80% can be achieved. Expected leach conditions of 2 – 8 hours of leaching with 5% sulfuric acid at temperatures ranging between 50 and 80 °C are anticipated. A conventional downstream lithium recovery circuit should be applicable to produce saleable lithium carbonate or lithium hydroxide.

The project has the potential to be a major supplier of lithium products in the world, and additional work is warranted.

The PEA limits the mine life to 40 years, but still indicates the project has good economics. The estimated initial capital cost is $482 million, with an after-tax Net Present Value at 8% discount rate of $1.45 billion and an internal rate of return of 32.7%. Relatively low acid consumption, combined with soft rock and low mining costs contribute to an estimated average operating cost of $3,983 per tonne of LCE.

The project has the potential to be a major supplier of lithium products in the world, and additional work is warranted.

26.0 RECOMMENDATIONS

GRE recommends the following activities be conducted for the Cypress Clayton Valley lithium project:

- Infill drilling to upgrade resource categories and optimize production schedule within the initial pit area
- Further testing for determination of acid concentration, consumption, temperature, and leach times for the individual units
- Determine optimum leaching configuration for process plant with respect to acid consumption and lithium extraction
- Bench-top testing to demonstrate production of lithium carbonate suitable for battery usage
- Detailed capital and operating cost estimates
- Investigate rare earth elements and other byproducts; quantify those elements in resources if appropriate
- Investigate alternative processing methods, including membranes and ion exchange resins for the concentration of lithium and other elements
- Investigate trade-offs between additional capital vs. saleable electrical generation for acid plant
- Initiate baseline data collection, hydrology and geotechnical studies
- Once the above are completed, GRE recommends completing a Pre-Feasibility Study. The estimated budget for the report and above items is $800,000.

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CERTIFICATE OF QUALIFIED PERSON

I, Terre A Lane, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "NI 43-101 Preliminary Economic Assessment Technical Report (PEA) of the Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of September 4, 2018 and an Issue date of October 1, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP and a Registered member of SME - 4053005.
2. I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University.
3. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this MRE is project management, mineral resource estimation, mine capital and operating costs estimation, and economic analysis with 25 or more years of experience in each area.
4. I have created or overseen the resource estimation, mine design, capital and operating cost estimation, and economic analysis of well over a hundred open pit projects.
5. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
6. I have been involved with the mine development, construction, startup, and operation of several mines.
7. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
8. I have not visited the property.
9. I am responsible for Sections 2, 3, 4, 5, 6, 14, 15, 16, 18-24, and corresponding sections of the Summary, Other Relevant Data and Information, Interpretation and Conclusions, Recommendations and References that are related to these sections.
10. I am independent of Cypress as described in section 1.5 by National Instrument 43-101.
11. I was an author in the prior Mineral Resource Estimate of the Clayton Valley Project issued June 5, 2018.
12. I have read National Instrument 43-101 and Form 43-101F1. The MRE has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
13. As of the effective date of the PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Terre A. Lane

"Terre A. Lane"

Principal Mining Mining Engineer

Date of Signing: October 1, 2018

CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Todd Harvey, PhD, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Preliminary Economic Assessment, NI43-101 Technical Report, Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of September 4, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a Society of Mining Engineers (SME) Registered Member Qualified Professional in Mining/Metallurgy/Mineral Processing, #04144120.
2. I hold a degree of Doctor of Philosophy (PhD) (1994) in Mining and Mineral Process Engineering from Queen's University at Kingston. As well as an MSc (1990) and BSc (1988) in Mining and Mineral Process Engineering from Queen's University at Kingston.
3. I have practiced my profession since 1988 in capacities from metallurgical engineer to senior management positions for production, engineering, mill design and construction, research and development, and mining companies. My relevant experience for the purpose of this PEA is as the test work reviewer, process designer, process cost estimator, and economic modeler with 25 or more years of experience in each area.
4. I have taken classes in mineral processing, mill design, cost estimation and mineral economics in university, and have taken several short courses in process development subsequently.
5. I have worked in mineral processing, managed production and worked in process optimization, and I have been involved in or conducted the test work analysis and flowsheet design for many projects at locations in North America, South America, Africa, Australia, India, Russia and Europe for a wide variety of minerals and processes.
6. I have supervised and analyzed test work, developed flowsheets and estimated costs for many projects including International Gold Resources Bibiani Mine, Aur Resources Quebrada Blanca Mine, Mineracao Caraiba S/A, Avocet Mining Taror Mine, Mina Punta del Cobre Pucobre Mine, and others, and have overseen the design and cost estimation of many other similar projects.
7. I have worked or overseen the development or optimization of mineral processing flowsheets for close to one hundred projects and operating mines, including copper flotation and acid heap leach SX/EW processes.
8. I have been involved in or managed many studies including scoping studies, prefeasibility studies, and feasibility studies.
9. I have been involved with the mine development, construction, startup, and operation of several mines.
10. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
11. I have not visited the project.
12. I am responsible for Section 17 of the PEA and have contributed to Sections 1, 2, 3, 13, 18, 19, 21, 24, 25, 26, and 27.
13. I am independent of Cypress Development Corp. as described in section 1.5 by National Instrument 43-101.
14. I have no prior experience with the Clayton Valley Lithium Project.
15. I have read National Instrument 43-101 and Form 43-101F1. The PEA has been prepared in

compliance with the National Instrument 43-101 and Form 43-101F1.

16. As of the effective date of the PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Jeffrey Todd Harvey, PhD

"Todd Harvey"

Director of Process Engineering

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: October 1, 2018

CERTIFICATE OF QUALIFIED PERSON

I, Todd S. Fayram, of 65 East Broadway Street, Suite 305, Butte, Montana 59701, the co-author of the report entitled "NI 43-101 Preliminary Economic Assessment Technical Report (PEA) of the Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of September 4, 2018 and an Issue date of October 1, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Metallurgy, #01300QP.
2. I hold a degree of Bachelor of Science (1984) in Mineral Processing Engineering and a Master's of Science in Metallurgical Engineering (2013) from Montana Tech of the University of Montana.
3. I have worked as a metallurgical engineer continuously for over 30 years since graduation from undergraduate university and have years of diversified experience in the consulting and operating fields for various mining and milling operations across the world.
4. My industrial experience includes: project and construction management; planning, design and engineering of precious and base metal heap leach and milling operations; industrial mineral development and operations, project evaluation for pre-feasibility, feasibility and bankable documents; and metallurgical testing and interpretation of numerous mineral deposits.
5. I have been involved with the mine and process development, construction, expansion, startup, and operation of numerous mines to include Minefinders-Dolores, American Bonanza-Copperstone, Americas Silver-Cosala, Middle Tennessee Zinc-Gordonsville, Getty Copper – Getty Project and others.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
7. I have visited the property several times. Most recently in June 2018.
8. I am responsible for parts of Sections 13, 17, and 21, and corresponding sections of the Summary, Other Relevant Data and Information, Interpretation and Conclusions, Recommendations and References that are related to these sections.
9. I am independent of Cypress Development Corp as described in section 1.5 by National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1. The PEA has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
11. As of the effective date of the PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Todd S. Fayram

Principal and Owner
Continental Metallurgical Services, LLC
Butte, Montana
Date of Signing: October 1, 2018

CERTIFICATE OF QUALIFIED PERSON

I, Jennifer J. Brown, P.G., of Hard Rock Consulting, LLC, 7114 W. Jefferson Ave., Ste. 313, Lakewood, Colorado, 80235, DO HEREBY CERTIFY THAT:

1. I am a graduate of the University of Montana and received a Bachelor of Arts degree in Geology in 1996.
2. I am a:
   - Licensed Professional Geologist in the State of Wyoming (PG-3719)
   - Registered Professional Geologist in the State of Idaho (PGL-1414)
   - Registered Member in good standing of the Society for Mining, Metallurgy, and Exploration, Inc. (4168244RM)
3. I have worked as a geologist for a total of 20 years since graduation from the University of Montana, as an employee of various engineering and consulting firms and the U.S.D.A. Forest Service. I have more than 10 collective years of experience directly related to mining and or economic and saleable minerals exploration and resource development, including geotechnical exploration, geologic analysis and interpretation, resource evaluation, and technical reporting.
4. 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 fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I am a co-author of the report titled “NI 43-101 Preliminary Economic Assessment Technical Report (PEA) of the Clayton Valley Lithium Project, Esmeralda County, Nevada, USA” with an effective date of September 4, 2018 and an Issue date of October 1, 2018, with specific responsibility for Sections 7 through 12 and corresponding sections of the Summary, Other Relevant Data and Information, Interpretation and Conclusions, Recommendations and References that are related to these sections. I visited the project on February 6 through February 9, 2018
6. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 1st day of October 2018.
“Signed” Jennifer J. (J.J.) Brown

JJBrown

Jennifer J. (J.J.) Brown, SME-RM

CERTIFICATE OF QUALIFIED PERSON

I, Hamid Samari, PhD, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Preliminary Economic Assessment, NI43-101 Technical Report, Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of September 4, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Geology, #01519QP.
2. I hold a degree of PhD of Science (2000) in geology (Tectonics - structural geology) from Tehran Azad University (Sciences & Research Branch).
3. I have practiced my profession since 1994 in capacities from expert of geology to senior geologist and project manager positions for geology, seismic hazard assessment and mining exploration.
4. I have been involved with many studies including scoping studies, prefeasibility studies, and feasibility studies.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
6. I have not visited the project.
7. I am responsible for parts of Section 14 the PEA.
8. I am independent of Cypress Development Corp. as described in section 1.5 by National Instrument 43-101.
9. I have no prior experience with the Clayton Valley Lithium Project.
10. I have read National Instrument 43-101 and Form 43-101F1. The PEA has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
11. As of the effective date of the PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Hamid Samari, PhD

"Hamid Samari"

Geologist

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: October 1, 2018