NI 43-101 Technical Report
Prefeasibility Study Clayton Valley Lithium Project

Esmeralda County, Nevada

Effective Date: May 19, 2020

Prepared for:

Cypress Development Corp.

Prepared by:

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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 May 19, 2020. The authors are Qualified Persons and their respective responsibilities in the Report's Sections are listed below.

Qualified Person	Responsible for Report Sections
Todd Fayram	Parts of 1-3, and 18, 21, 22, and 24–27 All of 13, 17 and 19
Terre Lane	Parts of 1-3, and 18, 21, 22, 24-27 All of 14-16
Daniel Kalmbach	Parts of 1-3, and 24-27 All of 4-12, 20 and 23

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ABBREVIATIONS AND ACRONYMS

µm	microns
2-D	2-dimensional
3-D	3-dimensional
AAS	atomic absorption spectroscopy
AIPG	American Institute of Professional Geologists
asl	above sea level
BFA	bench face angle
BLM	Bureau of Land Management
bsg	below surface grade
CH_3COOH	acetic acid
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CMS	Continental Metallurgical Services, LLC
CP	Centipoise
CVLP	Clayton Valley Lithium Project
Cypress	Cypress Development Corp.
FOS	Factor of Safety

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
IRR	internal rate of return in a cash flow analysis
IX	ion exchange
mg	milligram
kg	kilogram
t	tonne
kWh	kilowatt-hour
kWh/t	kilowatt-hours/tonne
Lilac	Lilac Solutions
LCE	lithium carbonate equivalent
Li	lithium
$LiCO_3$	lithium carbonate
ml	milliliter
L	liter
mm	millimeter
cm	centimeter
km	kilometer
km^2	square kilometer
km^3	cubic kilometer
LL	Liquid Limit
MMSA	Mining and Metallurgical Society of America
MW	megawatt
Mya	million years ago
NI	National Instrument
NORAM	NORAM Engineering and Constructors Ltd.
NPV	net present value of a discounted cash flow
NSR	net smelter return
PEA	preliminary economic assessment
PFS	prefeasibility study
PI	Platicity Index
PL	Plastic Limit
PLS	pregnant leach solution
ppm	parts/million
QA/QC	quality assurance/quality control
QP	qualified person
REE	rare earth element
SG	specific gravity
SME	Society of Mining, Metallurgy & Exploration

USGS	United States Geological Survey
ya	years ago
XRD	x-ray diffraction

REPORT NOTES

Pages 114 through 116 are intended to print in Landscape on Tabloid or 11 x 17-inch paper.

1.0 SUMMARY

Cypress Development Corp. (Cypress) commissioned this Prefeasibility Study of the Clayton Valley Lithium Project (project or CVLP). The project is in Esmeralda County, Nevada, six miles east of the community of Silver Peak, and is located within township 2 south, range 40 east, and township 3 south, range 40 east, Mt. Diablo Meridian. Cypress' property consists of 5,430 acres (2,197 hectares) of U.S. Federal mining claims. The claims are held 100% by Cypress and subject to an underlying net smelter return (NSR) agreement.

Cypress issued a Mineral Resource Estimate in June 2018 (GRE, 2018a) and a Preliminary Economic Assessment (PEA) in September 2018 (GRE, 2018b). This PFS updates the PEA Mineral Resource Estimate and economic assessment.

1.1 Geology & 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 likely occurred in conjunction with deformation due to lateral shear stress, resulting in disruption of large-scale topographic features. Clayton Valley is the lowest in elevation of a series of regional 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 western portion of the project area is dominated by the uplifted basement rocks of Angel Island which consist of metavolcanic and clastic rocks, and colluvium. 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. Elevated lithium concentrations, generally greater than 600 ppm, are encountered in the local sedimentary units of the Esmeralda Formation from surface to at least 142 meters below surface grade. The lithium-bearing sediments primarily occur as silica-rich, moderately calcareous, interbedded tuffaceous mudstone, claystone and siltstone.

1.2 Drilling

Cypress drilled at the project in 2017, 2018, and 2019. A total of 29 vertical, NQ-size core holes. Drill hole depths from 33 to 142.3 meters, totaling 2,574.9 meters drilled. The drilling results indicate a favorable section of claystone extending to depths of approximately 120 meters, where a strong, apparently planar, alternating oxidation/unaltered zone exists. The lithium content through these zones appears consistent, as do other geochemical factors and any specific significance of the oxidized and unaltered zones regarding lithium mineralization is not apparent.

In 2019, six in-fill holes were drilled to confirm and upgrade a portion of the 2018 PEA mineral resource. All six holes intersected lithium mineralization.

1.3 Mineral Processing & Metallurgical Testing

Lithium in the deposit is associated with illite and smectite clays. The lithium is amenable to leaching with dilute sulfuric acid leach followed by filtration, solution purification, concentration and electrolysis to produce lithium hydroxide.

Leaching tests were conducted by Continental Metallurgical Services in Butte, Montana. Tests on solid-liquid separation, tailings handling, and lithium recovery from solution were performed at several laboratories in the US and Canada. All analytical work was supported by ALS Minerals in Reno, Nevada and Vancouver, B.C.

Physical property testing shows the clay is soft, has negligible abrasion and work indices, and readily disaggregates with agitation in water. Testing has shown that leaching must be done at less than 30% solids for the slurry to mix, pump, and flow properly.

Leach tests were conducted on various samples under varying conditions to determine optimum acid concentrations and temperatures in leaching, and whether variability exists by material type. Tests on composite samples from four drill holes in 2019 showed only minor differences with respect to sample depth, oxidation or weathering state of the clay.

Large leach tests were performed on samples to provide slurry for rheology, filtration, and lithium recovery testing. The tests yielded average results of 86.5% extraction of lithium into solution and 126.5 kilograms per tonne (kg/t) for acid consumption.

Testing was conducted to determine a commercial means of solid-liquid separation. Specific conditions and equipment were identified. Solids from filtration tests simulating the final circuit were generated. The solids following single stage washing are suitable for handling by conveyor to a conventional dry-stack tailings facility.

CMS and NORAM designed and tested critical key elements of the flowsheet for recovering the lithium from solution. The flowsheet uses several stages to remove impurities and recycle 85% of the inflow back to leaching. The remaining 15% is treated by evaporation, followed by crystallization of salts and recovery of free sulfuric acid. Sulfuric acid is returned to the leach circuit along with the water recovered from evaporation. The NORAM-CMS test program was successful in yielding a concentrated lithium solution containing 1.85% lithium (Li) with low impurities and suitable for direct production of lithium hydroxide after additional treatment.

1.4 Mineral Resources

The Mineral Resource Estimate is based on all drilling results from the project, including six holes drilled in 2019.

The reported Mineral Resource is pit constrained by an “ultimate” pit that extends to the property boundaries as shown in Figure 14-17 and uses slope angles determined from geotechnical study described in Section 16.0.

The pit-constrained Mineral Resource (Table 1-1) totals 432.4 million tonnes averaging 1,088 parts per million (ppm) Li in the Measured Resource and 160.9 million tonnes at 1,032 ppm Li in the Indicated Resource, for a total of 593.3 million tonnes at 1,073 ppm Li in Measured and

Indicated Resource. Lithium contained in the pit-constrained Measured and Indicated Resource totals 636.4 million kg of Li, or 3.387 million tonnes of lithium carbonate equivalent (LCE).

Table 1-1: Summary Mineral Resource

Domain	Tonnes above 900 ppm Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)
Measured			
Tuffaceous mudstone	19.6	1,062	20.8
Claystone all zones	412.0	1,089	448.7
Siltstone	0.9	974	0.9
Total	**432.4**	**1,088**	**470.4**
Indicated			
Tuffaceous mudstone	14.5	1,043	15.1
Claystone all zones	146.2	1,031	150.7
Siltstone	0.20	963	0.2
Total	**160.9**	**1,032**	**166.0**
Measured + Indicated			
Tuffaceous mudstone	34.1	1,054	35.9
Claystone all zones	558.2	1,074	599.4
Siltstone	1.1	972	1.1
Total	**593.3**	**1,073**	**636.4**
Inferred			
Tuffaceous mudstone	0.1	933	0.1
Claystone all zones	2.2	1,009	2.2
Siltstone	0.0	0	0.0
Total	**2.3**	**1,005**	**2.3**

1. The effective date of the Mineral Resource Estimate is May 19, 2020. The QP for the estimate is Ms. Terre Lane of Global Resource Engineering Ltd. and is independent of Cypress.
2. The Mineral Resource estimate was prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the generally accepted Canadian Institute of Mining's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. (November 29, 2019).
3. 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 Resource will be converted into Mineral Reserves. Inferred Mineral Resources are the 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.

1.5 Mineral Reserves

The Measured and Indicated Resources were used to determine the Mineral Reserves as described in Sections 14.0 and 15.0.

Within the ultimate pit shell, 16 pit phases were constructed, expanding from initial mining in the southwest to the northeast. For the production schedule and analysis, only the first eight phases are used to produce a mine life of approximately 40 years. The cumulative result for all eight phases forms the Mineral Reserves in Table 1-2.

Table 1-2: Summary Mineral Reserve

Domain	Tonnes above Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)
Probable Reserve			
Total	**222.8**	**1,141**	**254.3**

1. The effective date of the Mineral Reserve Estimate is May 19, 2020. The QP for the estimate is Ms. Terre Lane of Global Resource Engineering Ltd. and is independent of Cypress.
2. The Mineral Reserve estimate was prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the with generally accepted Canadian Institute of Mining's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019).
3. Mineral Reserves are reported within the pit design at a mining cutoff of 900 ppm.
4. The cutoff of 900 ppm is an optimized cutoff selected for the mine production schedule.
5. The Mineral Reserves are derived from and not separate from the Mineral Resources.
6. Mineral Reserves are estimated based on delivery to the mill stockpile.
7. No Inferred Resources are included in the Mineral Reserves or given value in the economic analysis

The Mineral Reserve is classified as a Probable Reserve based on Modifying Factors as described in Section 15.0. The Probable Reserve contains 254.3 million kg of Li, or 1.353 million tonnes LCE.

1.6 Mining

The initial pit, is based on the first eight phases of the ultimate pit (Table 1-3) and 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/day (tpd) of mill feed. No drilling or blasting will be required.

Table 1-3: Initial Pit Mineral Reserve by Phase

Pit Phase	Ore Tonnes (millions)	Low Grade Ore Tonnes (millions)	Waste Tonnes (millions)	Li Contained (millions Kg)	Li Grade (ppm)	Stripping Ratio
1	33.93	0.41	0.79	40.67	1,199	0.02
2	20.04	0.00	1.18	23.00	1,148	0.06
3	28.62	0.59	2.92	32.28	1,128	0.10
4	14.31	1.62	1.83	16.68	1,165	0.11
5	36.47	4.55	7.07	40.45	1,109	0.17
6	33.50	3.80	9.27	38.31	1,144	0.25
7	16.19	0.81	2.42	18.39	1,136	0.14
8	38.40	0.64	10.59	43.24	1,126	0.27
Total	**221.46**	**12.42**	**36.07**	**253.03**	**1,143**	**0.15**

The processable material will be removed from the pit using in-pit semi-mobile feeder-breaker with conveyors. The production equipment includes a 12 m³ hydraulic excavator and scrapers to haul lower grade claystone to a waste dump. The stripping ratio is 0.15:1. The mine operates on a two 10-hour shift, 7 days/week schedule.

1.7 Infrastructure

Access to the project is via Silver Peak Road. The east side of Angel Island was identified for the plant location based upon proximity to the road, power, mine area, and favorable topography.

Facilities on-site include administration, laboratory, warehouse, reagent storage, sulfuric acid plant, crushing, leaching and lithium recovery areas, mine shop, and fuel and reagent storage areas.

An acid plant, with 2,500 tpd of acid capacity, is a key item of infrastructure. The plant will burn elemental sulfur to create sulfuric acid and, in the process, generate steam to heat leach tanks. The plant will also be equipped for power generation.

Tailings will be conveyed from the filtration area and stacked in tailings facility south of the plant by conveyor. Dozers will be used for final spreading and contouring.

Cypress has evaluated options for securing makeup water estimated at 2,000 gallons per minute (gpm). A specific source and related costs are excluded from the study. Allowances are included in the estimates for constructing supply wells, pipeline, and power.

1.8 Permitting & Environmental

Environmental permitting requirements for the Project are expected to be like other mines in Nevada. The permitting process consists of submitting a Plan of Operations to the Bureau of Land Management, who will act as lead agency, conducting environmental baseline studies, and preparing an Environmental Impact Statement along with other permit applications prior to site development and operations. The time frame for permitting the project is estimated at 18 to 24 months.

A Phase I Environmental Site Assessment of the project was conducted in 2019 and found no existing environmental liabilities. A Threatened and Endangered Species Preliminary Study was also completed. Initiation of field studies is included in the recommendations.

1.9 Capital & Operating Costs

Capital Costs

The capital and operating costs are estimated according to accepted methods for prefeasibility studies. The estimates constitute a Class 4 estimate, as defined by the AACE International, and have an accuracy of +30%/-15%. All costs are presented in Q1 2020 US$. The initial capital costs total $493 million, which includes $95 million in contingency plus working capital. Vendor quotes, internal data and public information were used along with construction factors to estimate Direct Costs. Indirect costs allow for EPCM, freight, sales tax and Owners Costs. Contingency at 20% is applied to the Direct and Indirect Costs.

Table 1-4: Capital Cost Summary

Area	$ x 1000
Facilities	5,891
Mine	34,757
Plant	306,855
Infrastructure	25,907
Owners Costs	24,992
Contingency & Working Capital	94,883
Total CAPEX	**493,284**

Operating Costs

The operating costs were developed for the operation sized to at the nominal mill rate of 15,000 tpd. The estimated operating costs total an average of $91.2 million/year, or $16.78/t.

Table 1-5: Operating Cost Summary

Area	Avg Annual $ x 1000	Mill Feed $/t
Mining	9,932	1.83
Processing	77,735	14.30
G&A	3,550	0.65
Total OPEX	**91,218**	**16.78**

The operating costs are developed from estimates of labor, operating and maintenance supplies, and power. The total labor force required for the operation is estimated at 183 on-site employees.

Acid plant operations are a major component in the operating costs and account for one third of the total operating cost based on a delivered cost of $145 per tonne for sulfur. The acid plant has capacity to generate 93% of the power required by the operation and will have surplus power available when the operation is running. No allowances are made in the operating cost estimates for potential power sales or offsets.

1.10 Economic Analysis

An after-tax discounted cash flow model was prepared using the information and estimates in the report. The model includes federal, state and local taxes.

The nominal production rate at full operation is set at 15,000 tpd, or 5.475 million tonnes/year (tpy). The production schedule uses the material from the first eight pit phases, which results in a 40-year mine life, and 223 million tonnes of mill feed at an average grade of 1,141 ppm Li. Recovery of lithium is estimated at 83%. The resulting annual output averages 27,400 tpy of LCE.

The economic evaluation is reported in terms of LCE using an average price of $9,500 per tonne. The price assumption reflects variations expected over time due to start-up and type of lithium product.

The only revenue stream considered is from the sale of lithium products. No revenues are included for any other by-products. Such revenues remain to be determined.

No credit is taken for power sales or offsets on purchased electricity.

Results for the project base case are:

- Average annual production of 27.4 million kg of LCE.
- Cash operating cost of $3,329/tonne LCE
- An after-tax $1.052 billion NPV at 8% discount rate
- An after-tax IRR of 25.8%
- Payback period of 4.4 years
- Break-even price (0% IRR) of $4025/t LCE

The cash flow model is most sensitive to changes in lithium price. Sensitivities to lithium price, capital and operating cost are shown in Table 1-6.

Table 1-6: Economic Sensitivity

Variation	60%	100% Base Case	150%
Lithium Price $/t LCE	$5,700	$9,500	$14,250
NPV-8%	$130 million	$1.052 billion	$2.173 billion
IRR	10.5%	25.8%	41.1%
Capital Cost	$296 million	$493 million	$740 million
NPV-8%	$1.352 billion	$1.052 billion	$673 million
IRR	30.1%	25.8%	20.0%
Operating Cost	$1,997/t LCE	$3,329/t LCE	$4,993/t LCE
NPV-8%	$1.229 billion	$1.052 billion	$828 million
IRR	39.6%	25.8%	17.9%

Note: IRR (internal rate of return) and NPV (net present value) are both shown after-tax

1.11 Interpretation & Conclusions

The project has Mineral Resources and Mineral Reserves to support a mine life in excess of 40 years at a production rate at 27,400 tpy LCE and an average estimated operating cost of $3,329/tonne LCE. The project risks are typical of a mining project at a prefeasibility level of study and further work with respect to processing and permitting are needed to advance the project to the feasibility level. A pilot plant program and environmental studies are needed to advance the project to the feasibility stage.

1.12 Recommendations & Risks

The recommendations to advance the project are:

- Processing—Additional test work is needed to confirm the process flowsheet and determine recoveries and reagent consumptions at the pilot stage. Critical information includes,
  - confirm steps and equipment in leaching and filtration
  - conduct further work to enhance solid-liquid separation and reduce acid consumption
  - determine lithium and acid losses in the processing plant, if any
  - optimize solution handling in the plant and determine if bleed streams or additional treatment are needed to recycle solutions
  - determine whether potassium, magnesium, rare earth elements and other elements have commercial value
- Mining—Drilling or limited test mining is required to obtain material for metallurgical testing.
- Permitting—A field program is required to determine if any species of concern are present and to gather data to prepare a Plan of Operations.
- Infrastructure—Feasibility-level designs for the mine, plant and tailings storage areas can begin. Further determination of project power and water supply are needed.

Cost of the programs is estimated at $6.75 million.

Table 1-7: Estimated Pilot Plant Costs

Area	$ x 1000
Pre-program studies	150
Sample procurement	500
Equipment	
Leaching	650
Lithium Recovery	2,600
Operating expenses	1,500
Contingency	1,350
Total Program	**6,750**

The potential risks at this stage of the project are:

- Recovery of lithium from the project was not proven at a commercial scale. Further testing in a pilot plant is needed.
- Production is potentially limited by the availability and cost of sulfur and its transportation.
- The project is most sensitive to lithium market prices which are currently dependent on the demand for lithium batteries in electric vehicles and energy storage.
- A source of makeup water has not been secured. Options to obtain water through rights acquisition, purchase or other agreements should be pursued.
- Environmental permitting is subject to presence of flora, fauna or other conditions which are yet to be determined.

2.0 INTRODUCTION

This National Instrument (NI) 43-101 Report titled Prefeasibility Study of the Clayton Valley Lithium Project (the PFS or report) was prepared for Cypress Development Corp. (Cypress).

Cypress has three published reports on the Clayton Valley Lithium Project (CVLP or project): a NI 43-101 Technical Report (Cypress Development Corp., 2018), a NI 43-101 Technical Report Mineral Resource Estimate (GRE, 2018a), and a NI 43-101 Technical Report Preliminary Economic Assessment (GRE, 2018b).

This report includes the results from all drilling and metallurgical testing, updates to the capital and operating cost estimates, and addresses changes in the physical and economic conditions since the 2018 PEA was published.

2.1 Scope of Work

The scope of work assumed by the authors was to prepare a PFS for the Clayton Valley Lithium Project and prepare recommendations on further work required to advance the project to the feasibility study stage.

2.2 Qualified Persons

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

- Todd S. Fayram, QP, Member of SME MMSA #01300QP and owner of Continental Metallurgical Services, LLC.
- Terre A. Lane, QP, Mining and Metallurgical Society of America (MMSA) 01407QP, Society for Mining, Metallurgy & Exploration (SME) Registered Member 4053005, Principal Mining Engineer, GRE
- Daniel W. Kalmbach, QP, American Institute of Professional Geologists—Certified Professional Geologist (AIPG-CPG) CPG#11732, Independent Geologist

Practices consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2010) were applied to the generation of this PFS. Mr. Fayram, Ms. Lane and Mr. Kalmbach are collectively referred to as the "authors" of this PFS. In addition to their own work, the authors used information from other sources and 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	Kalmbach
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Kalmbach
6	History	Kalmbach
7	Geological Setting and Mineralization	Kalmbach
8	Deposit Types	Kalmbach
9	Exploration	Kalmbach
10	Drilling	Kalmbach
11	Sample Preparation, Analyses and Security	Kalmbach
12	Data Verification	Kalmbach
13	Mineral Processing and Metallurgical Testing	Fayram
14	Mineral Resource Estimates	Lane
15	Mineral Reserve Estimates	Lane
16	Mining Methods	Lane
17	Recovery Methods	Fayram
18	Project Infrastructure	Fayram Lane
19	Market Studies and Contracts	Fayram
20	Environmental Studies, Permitting and Social or Community Impact	Kalmbach
21	Capital and Operating Costs	Fayram Lane
22	Economic Analysis	Fayram Lane
23	Adjacent Properties	Kalmbach
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.

All currency amounts in this PFS are presented in US Dollars.

The project is planned to produce lithium hydroxide as its primary product, but the cost basis includes the provision for producing lithium carbonate. For reporting purposes, all production is quoted in terms of lithium carbonate equivalent (LCE).

2.5 Consultant Reviews

The PFS was reviewed by Dr. Corby G. Anderson, PE, QP, technical advisor to Cypress, and by Dr. Clive Brereton and Eric Mielke, M.A. Sc., P.Eng., of NORAM. Their comments and suggestions are included in the PFS.

3.0 RELIANCE ON OTHER EXPERTS

The authors relied on statements by Cypress concerning mineral rights ownership data and legal and environmental matters in Sections 4.0, 5.0, and 20.0 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 US Bureau of Land Management—Tonopah Field Office.

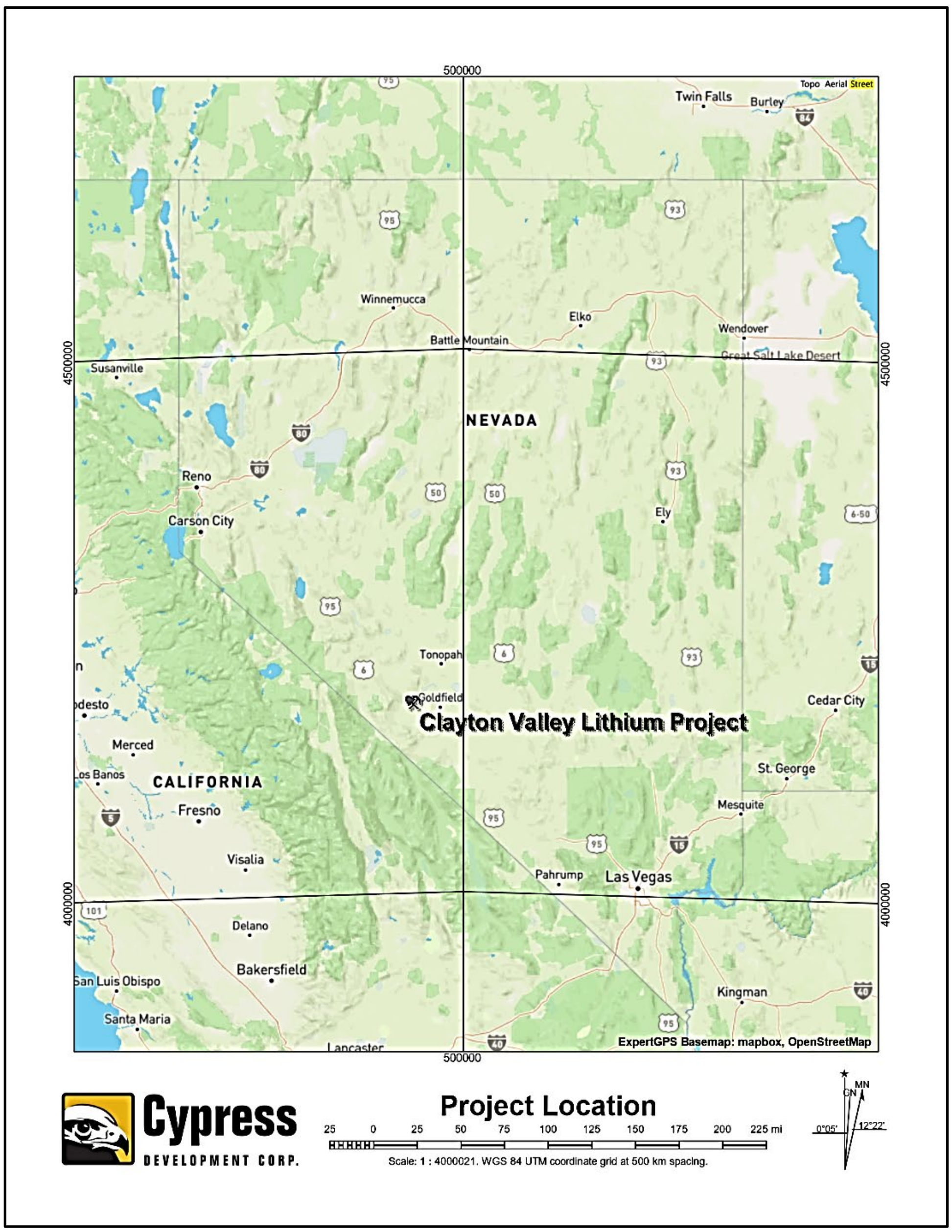
The authors reviewed and incorporated reports and studies as described within this Report.

4.0 PROPERTY DESCRIPTION & LOCATION

4.1 Location

The project is centered near 452,800 meters east, 4,177,750 meters north, WGS84, zone 11 north datum, in central Esmeralda County, Nevada. The project is located 220 miles southeast of Reno, Nevada (Figure 4-1). The regional town of Tonopah is 41 miles northeast of the project, and the small community of Silver Peak lies six miles west of the project. The project lies within township 2 south, range 40 east and township 3 south, range 40 east, Mt. Diablo Meridian. Access from Tonopah, Nevada, is by traveling 22 miles south on US Highway 95, then 19 miles west on Silver Peak Road.

Figure 4-1: Project Location Map



4.2 Mineral Rights & Tenure

The project comprises 129 unpatented placer mining claims and 212 unpatented lode mining claims listed in Table 4-1 and outlined in Figure 4-2. The claims cover 5,430 acres and provide Cypress with the rights to all brines, placer and lode minerals on the property. The claims lie within portions of sections 14-17, 20-23, 26-28, and 32-35 of township 2 south, range 40 east and section 5 of township 3 south, range 40 east, Mt. Diablo Meridian in the eastern portion of Clayton Valley, Nevada. All lode and placer claims are unpatented U.S. Federal claims administered by the BLM.

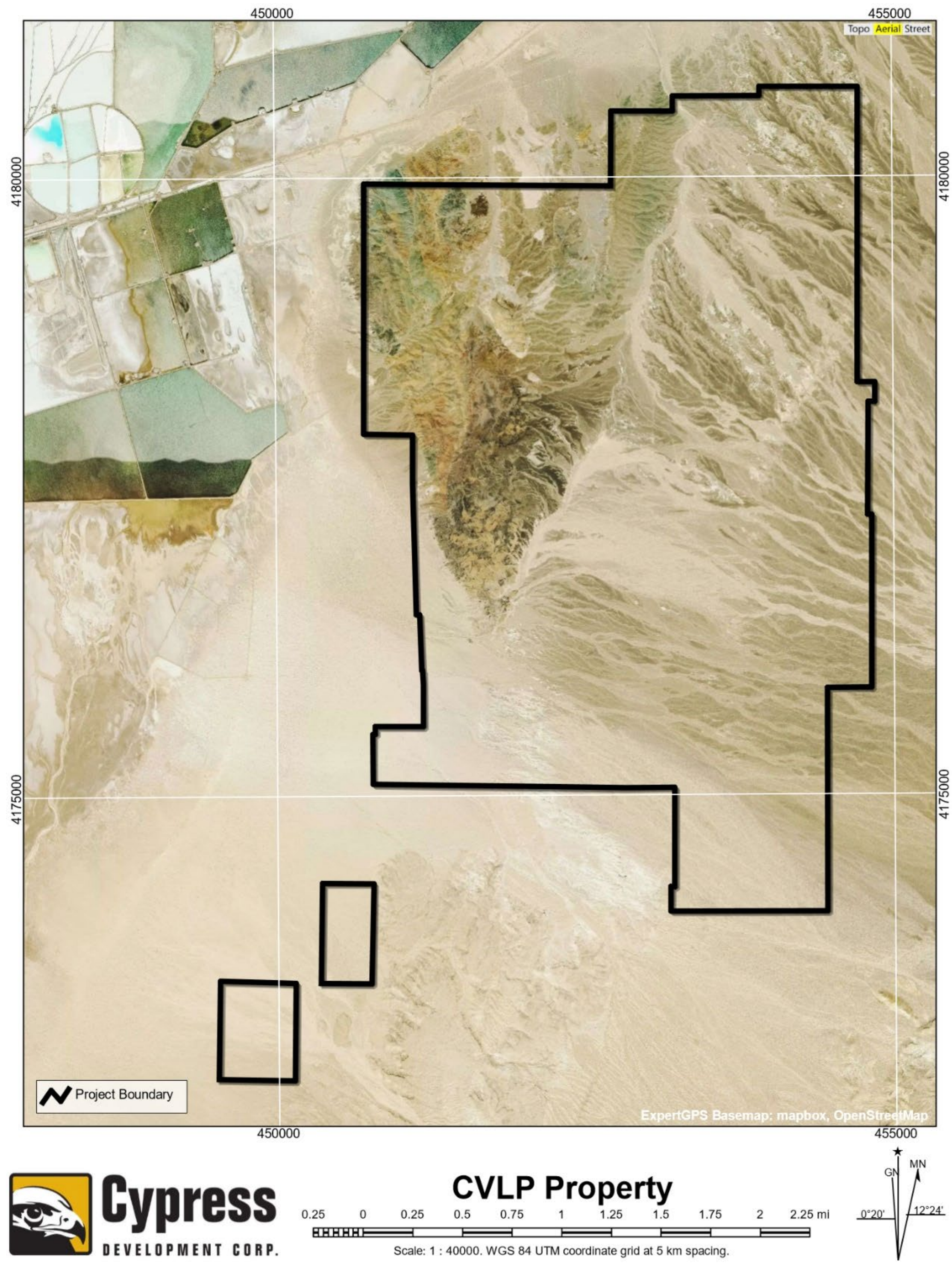
Portions of the property are controlled by placer claims or lode claims, the center portion of the property is controlled with placer claims overlaid with lode claims. The placer claims vary in size from 20 to 80 acres and were staked as even divisions of a legal section, as required under placer mine claim regulations. The lode claims are a maximum of 600 x 1,500 feet in size or 20.6 acres each.

The portion of the property which contains the Mineral Reserves is subject to a 3% net smelter return (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 $167/20 acres or claim depending on claim type. All claims are all in good standing with the BLM and Esmeralda County. The Mineral Resource and Mineral Reserve estimates defined and described in this report fall entirely on Cypress' unpatented mining claims.

Table 4-1: Active Mining Claims

NMC From	NMC To	Claims
Placer Mining Claims		
NMC1119079	NMC1119089	11
NMC1119046	NMC1119078	33
NMC1120318	NMC1120352	35
NMC1121389	NMC1121394	6
NMC1121397	NMC1121400	4
NMC1124933	NMC1124952	20
NMC1129564	NMC1129565	2
NMC1177632	NMC1177633	2
NMC1177672	NMC1177687	16
Total Placer Claims		129
Lode Mining Claims		
NMC1136414	NMC1136484	71
NMC1162324	NMC1162402	79
NMC1177644	NMC1177645	2
NMC1177656	NMC1177671	16
NMC1179592	NMC1179609	18
NMC1179614	NMC1179639	26
Total Lode Claims		212

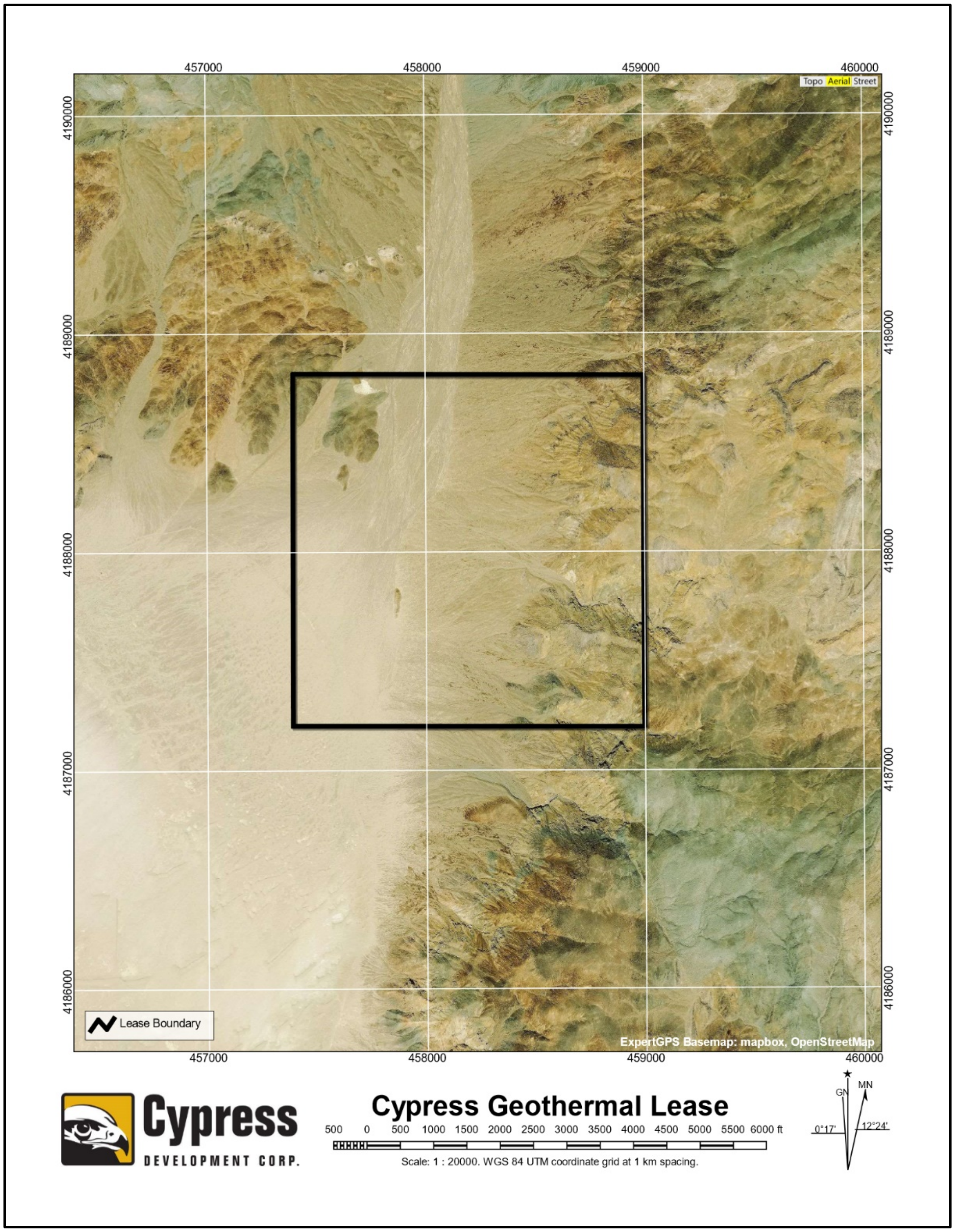
Figure 4-2: Project Property Map



4.3 Geothermal Lease

Cypress holds a geothermal lease with the BLM, NV-19-09-027, acquired in 2019 (BLM, 2019). The lease totals 640 acres in all of section 24, T1S, R40E, Mt. Diablo Meridian (Figure 4-3). The lease is located five miles north of the project near Pearl Hot Springs and Paymaster Canyon. The annual holding cost is $8,000 and the lease is subject to U. S. Federal royalties upon production.

Figure 4-3: Geothermal Lease Map



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

The project is accessed from Tonopah, Nevada, by traveling 22 miles south on US Highway 95, then 19 miles west on Silver Peak Road, a paved and well-maintained gravel road. This road is currently undergoing upgrades which will provide pavement to the project entrance when complete.

The climate of the Clayton Valley is hot in summer, with average high temperatures in mid-90 °F and cool in the winter with daily average lows between 17-32 °F (Table 5-1). Precipitation is normally in the form of thunderstorms which can be very strong and cause violent flooding even miles from the actual storm. Other precipitation events, including snowfall, are limited due to the nature of the rain shadow produced by the mountain ranges to the west. Snow cover in winter is rare, and year-round low humidity aids in evaporation. Windstorms are common all year but occur predominantly in the summer and fall.

Table 5-1: Project Weather Information

Silver Peak, Nevada Average Weather Data						
Month	Jan	Feb	Mar	Apr	May	Jun
Average high in ºF	47	54	62	69	80	90
Average low in ºF	19	24	32	38	49	57
Av. precipitation in inch	0.39	0.3	0.53	0.47	0.37	0.37
Month	Jul	Aug	Sep	Oct	Nov	Dec
Average high in ºF	98	95	86	73	57	46
Average low in ºF	62	59	50	38	26	17
Av. precipitation in inch	0.45	0.39	0.25	0.4	0.31	0.22

Source: www.usclimatedata.com/climate/silverpeak/nevada/united-states/usnv0084

Local resources available vary depending on distance from the project. Silver Peak (population 107) is the closest census designated place to the project, it consists mainly of housing, it has a post office, library and a restaurant/bar, but few other services. The next closest place is Goldfield (population 268), the Esmerelda county seat, it has housing, small stores, a restaurant, motel and government offices. Tonopah (population 2,478) is the Nye county seat and closest full-service town to the project, it has housing, grocery stores, restaurants, lodging, banks, hardware stores and government offices. Employment in Tonopah consists of service industry, military, mining and industrial jobs. Experienced processing and other technical labor should be available as the project is in a region of active lithium brine extraction, precious metals mining and solar power generation.

Infrastructure available includes paved and well-maintained gravel roads, power lines near the north side of the project, and substations in Silver Peak, Alkali Hot springs and Millers.

The project is in the Great Basin physiographic region, within the Walker Lane province of the western Great Basin. The valley has a total watershed area of about 1,430 square kilometers (km²) and the floor of the valley lies at an altitude of 4,320 ft above sea level (asl). The surrounding mountains rise several thousand feet above the valley floor, with the highest surrounding mountain, Silver Peak at 9,380 ft asl. The valley is bounded to the west by the Silver Peak Mountain Range, to the south by the Palmetto Mountains, to the east by Clayton Ridge and the Montezuma Range, and to the north by the Weepah Hills. There is no permanent surface water in the Clayton Valley watershed, all watercourses are ephemeral and only active during periods of intense precipitation. At the project itself, the terrain is dominated by mound-like outcrops of mudstone and claystone, cut by dry gravel washes across a broad alluvial fan. Access at the project is excellent due to the overall low relief of the terrain (Photo 5-1 and Photo 5-2).

Photo 5-1: Project from Flanks of Angel Island Looking East

Photo 5-2: Dry Wash Channel Cutting Claystone in Eastern Portion of Project

6.0 HISTORY

The first recorded mining activity in Clayton Valley was in 1864 with the discovery of silver at the town of Silver Peak. The playa in the center of Clayton Valley was mined for salt as early as 1906, and later explored for potash during World War II. Lithium was noted during the 1950s. In 1964, Foote Minerals acquired leases and began production of lithium carbonate at Silver Peak by 1967. Production of lithium carbonate from brine has continued to the present under several companies, currently under Albemarle Corporation (www.albamarle.com).

The occurrence of lithium in sediments of Clayton Valley was reported as early as the 1970s by the United States Geological Survey.

In 2015, Cypress acquired rights to claims on the south and east side of Angel Island. Sampling revealed high lithium concentration in surface sediments.

In 2017, Cypress drilled its first holes in the Dean claim block, followed later that year by drilling in the Glory claim block. In February 2018, Cypress reported exploration results on the Dean Property in a NI 43-101 Technical Report. Later in 2018, Cypress completed additional drilling followed by NI 43-101 technical reports Resource Estimate and a Preliminary Economic Assessment.

7.0 GEOLOGIC SETTING & MINERALIZATION

The following description of the geologic setting of the Clayton Valley Lithium Project uses information from Davis and Vine (1979), Davis et. al (1986), Munk (2011) and Bradley et. al (2013).

7.1 Regional Geology

Clayton Valley is a closed basin near the southwestern margin of the Basin and Range geo-physiographic province of western Nevada (Figure 7-1). Horst and graben normal faulting is a dominant structural element of the Basin and Range and likely 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), is within a few kilometers of the northern and eastern boundaries of Clayton Valley. 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 are the Palmetto Mountains whose arcuate form 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°E to N40°E. Near the margins of the playa surface, fault scarps with two distinct trends were 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 ya, while the last movement on the N65°E set is probably closer to 20,000 ya (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 Mya). Tectonic extension began in the late Cenozoic (16 Mya) and continues today.

East of Clayton Valley, more than 100 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 be a volcanic center to the east near Montezuma Peak, or to the south in the Montezuma Range, the Palmetto Mountains, Mount Jackson, or the Silver Peak center to the west.

Cenozoic sedimentary rocks are exposed in the Silver Peak Range, in the Weepah Hills, and in the hills due 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 Mya (Robinson, et al., 1968).

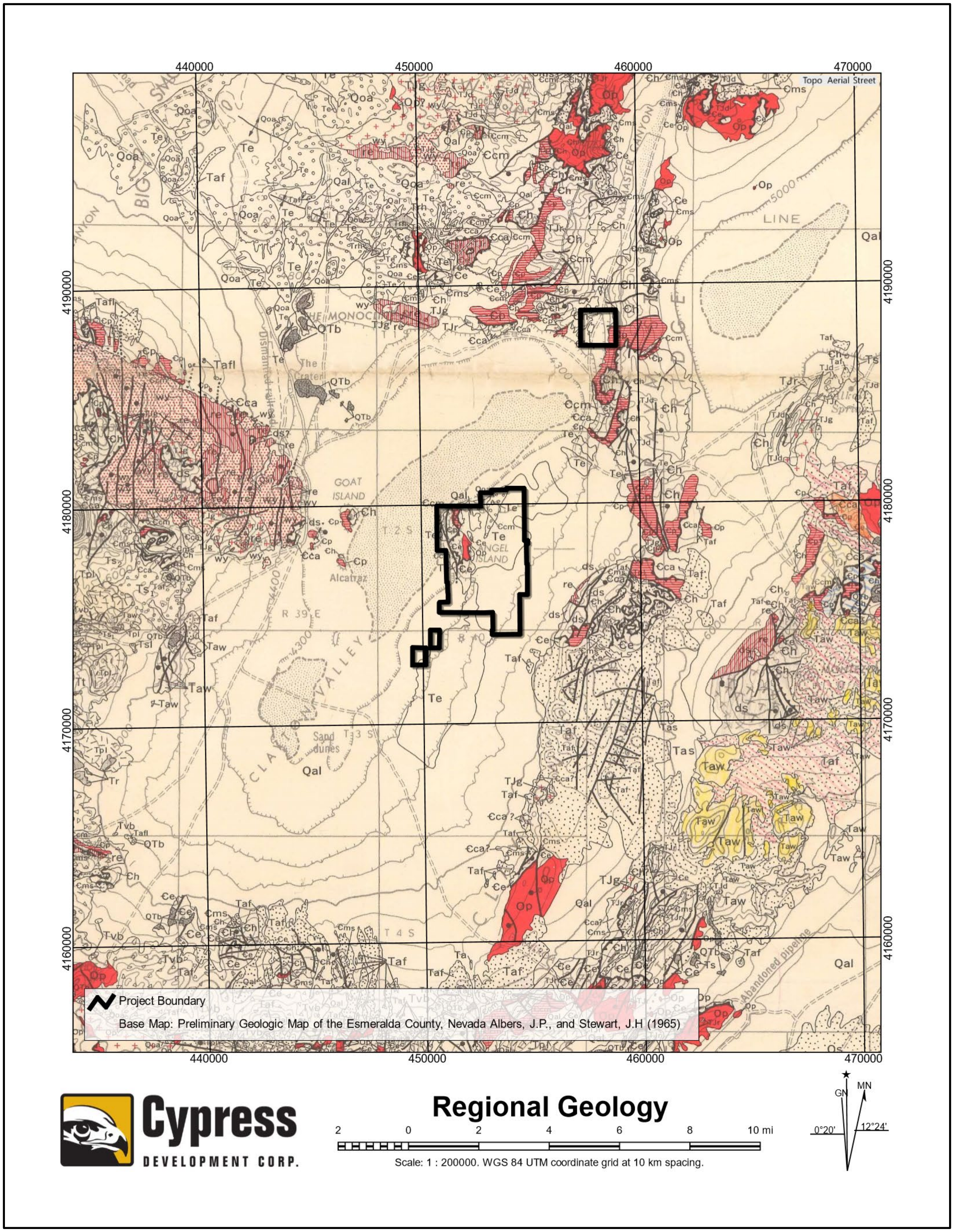
7.2 Local Geology

Clayton Valley is the lowest in elevation of a series of local playa filled basins, with a playa floor of about 100 km^2 which collects 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 as 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 was 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 sediments, by weight, are 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/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 Map



7.3 Project Geology

The western portion of the project area is dominated by the uplifted basement rocks of Angel Island which consist of metavolcanic and clastic rocks, and colluvium. The southern and eastern portions are dominated by uplifted, lacustrine sedimentary units of the Esmeralda Formation. Locally the Esmeralda Formation is comprised of fine grained sedimentary and tuffaceous units, with some occasionally pronounced local undulation and minor faulting (Photo 7-1 and Figure 7-2).

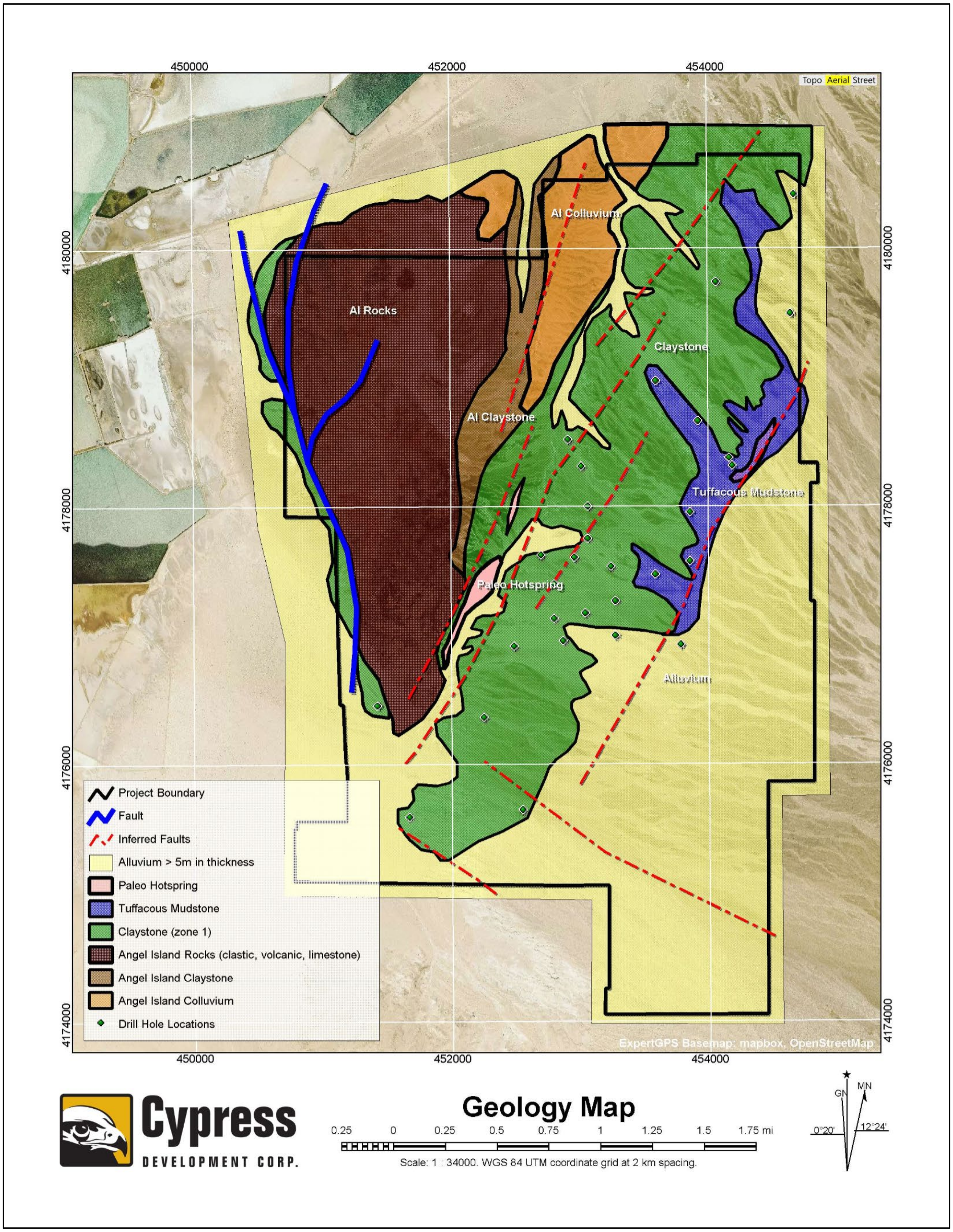
The resulting topography consists of elongate, rounded ridges of exposed Esmeralda Formation separated by washes and gullies filled with alluvial cobble, gravel and fine sediment. The ridge tops are commonly mantled weathered fragments of rock (desert pavement) sourced from the surrounding highlands. Cypress provides the following description of the stratigraphic units of the Esmeralda Formation in the project area, which form a laterally and vertically continuous stratigraphic section which underlies the south and eastern portions of the project area. Cross sections showing logged geology, geologic interpretation, and assay results from the assayed core intervals are in Section 14.0.

Photo 7-1: Exposed Esmerelda Formation in Southern Portion of Project

Alluvium—this unit consists of polylithic sand, gravel, cobble, and boulder, and covers large portions of the project. This unit varies from 0 to 10+ meters in thickness, is a thin desert pavement on the ridge or mound tops and thickens in the fluvial channels and to the east up the alluvial fan. Most of the material is from the steep canyons cutting Clayton Ridge to the east with minor amounts from the eastern flanks of Angel Island. Lithium is locally not present in this unit.

Tuffaceous mudstone—this unit consists of interbedded silty mudstone and hard tuffaceous beds, tan to reddish brown in color. At some locations, this unit grades with the alluvium creating a thin (1 to 2 meter) layer of semi-consolidated conglomerate. The unit is approximately 70% mudstone and 30% hard tuff layers. This unit is 0 to 15 meters in thickness and lithium content averages 850 ppm.

Figure 7-2: Project Geology Map



Claystone—this unit is an ash-rich claystone and the primary lithium-bearing lithology at the project, the fresh color ranges from olive green, blue-gray, tan, to reddish-brown but becomes tan-brown with a light green hue when dry. Below an interbedded top section, this unit is massive with uniform texture and color, and the grain size is consistent, and the clay is generally fat. Areas of ashy-lamina, thin tuff or zeolite layers, and ash/zeolite blebs are present, unit is generally soft and weakly ductile, breaks with conchoidal fractures and hardens when dry. The primary differences within the unit are weathering, as three distinct zones of oxidized and unaltered material. These zones do not show significant differences geochemically or metallurgically outside of higher lithium concentrations in zones one and two. This unit is 60 to 120 meters in thickness, and lithium content averages 1,060 ppm.

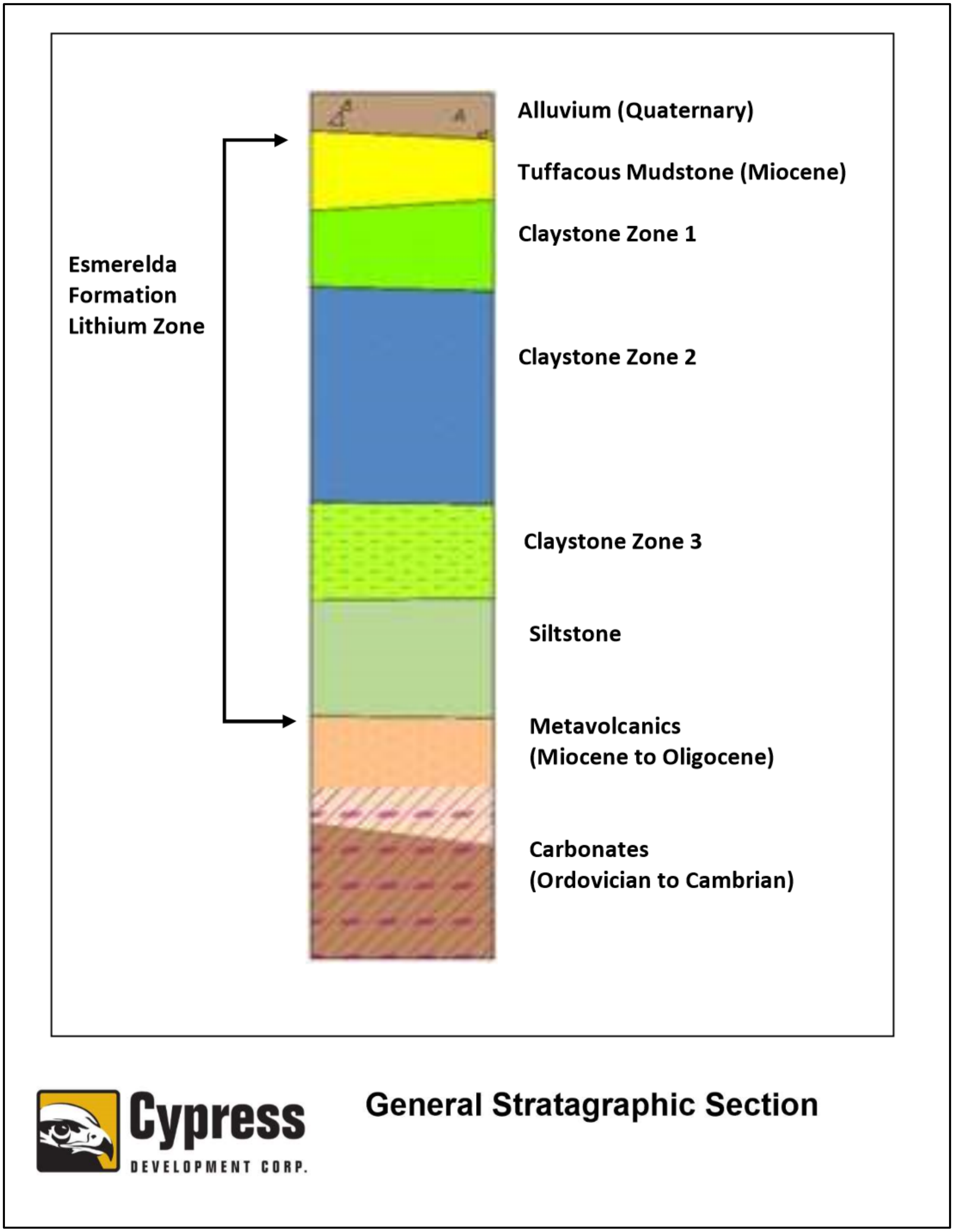
The first zone is olive to tan in color when fresh and tan when dry, oxidized and contains locally abundant iron oxide staining, hematite and partial layer replacement. The second zone begins with an interbedded area of oxidized and unaltered material, becoming completely unaltered at depth. Color is blue-gray when fresh and tan to light green when dry, unaltered and contains occasional to pervasive zones of lamina containing dark carbon and formational pyrite. The third zone typically begins below an ash-fall tuff with gradational oxidation becoming completely oxidized with depth, color is olive when fresh and dark-tan to reddish-brown when dry, zones of formational carbon and pyrite can be found high in the zone but soon become pervasive thin bands of hematite or limonite, and as depth approaches the next unit, zones of ashy/sandy or silica rich lamina and thin beds occur, and in general the grain size increases with silt and sand more prevalent.

Siltstone—this unit has a gradational upper contact and is a unit where the claystone becomes siltstone and is more firm and coarser grained than the claystone unit, color is tan to reddish-brown, the unit is oxidized, with zones of hematite, cross bedding, slump features and other signs of a higher-energy depositional environment, poorly to very well indurated with silt+sand fraction generally ~50% and higher in areas of thin beds/lamina. This unit's thickness is not known, although a 19-meter section is encountered in GCH-07, and lithium content averages 625 ppm.

7.4 Mineralization

Elevated lithium concentrations, generally > 600 ppm, are encountered in the local sedimentary units of the Esmeralda Formation from surface to at least 142 meters below surface grade (bsg). The lithium-bearing sediments primarily occur as silica-rich, moderately calcareous, interbedded tuffaceous mudstone, claystone and siltstone. The overall mineralized sedimentary suite is a laterally and vertically extensive, roughly tabular zone with at least two prominent oxidation horizons (Figure 7-3). The primary area of mineralization is in a claystone unit consisting of three zones: oxidized claystone, unaltered claystone and an oxidized claystone. The claystone unit is overlain by tuffaceous mudstone in the eastern portion of the project and underlain by a siltstone. Elevated lithium concentrations occur in all the uplifted lacustrine strata encountered; however, lithium concentrations are notably higher and more consistent in the claystone unit.

Figure 7-3: General Stratigraphic Section



8.0 DEPOSIT TYPE

Lithium occurs in potentially economic concentrations in three types of deposits: pegmatites, continental brines, and clays. Lithium is produced from pegmatites and brines, with brines the largest producer of lithium worldwide. There is no active mining of lithium clay deposits.

In clay deposits, lithium is often associated with smectite (montmorillonite) group minerals. The USGS presents a preliminary descriptive model of lithium in smectites of closed basins (Asher-Bolinder, 1991), Model 251.3(T), which suggests 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 volcanic rocks, and an arid environment.

Regional geologic traits 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 deposit type is represented by the USGS deposit model. The model consists of light-colored, ash-rich, lacustrine rocks that contain swelling clays and occur within hydrologically closed basins proximal to 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 are interbedded silica-rich, ash-rich mudstone and claystone, with interbeds of sandy and tuffaceous mudstone/siltstone and occasional poorly cemented silt and sandstone. The lithium concentrations are highest within the mudstone and claystone, but lithium is still present in a siltstone unit underlying the claystone.

The deposition of the lithium-rich sediments likely occurred late in the history of the associated paleo brine lake, based largely on the stratigraphic position of the mudstone and claystone above the thick overall sandstone- and siltstone-dominated basin fill events. 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 represent several million years of lithium input and concentration within the basin. Figure 8-1 through Figure 8-3 show a conceptual sequence of depositional, erosional, and structural events which may account for the present-day nature and occurrence of the lithium deposit.

Figure 8-1: Deposit Origin: Volcanic Events

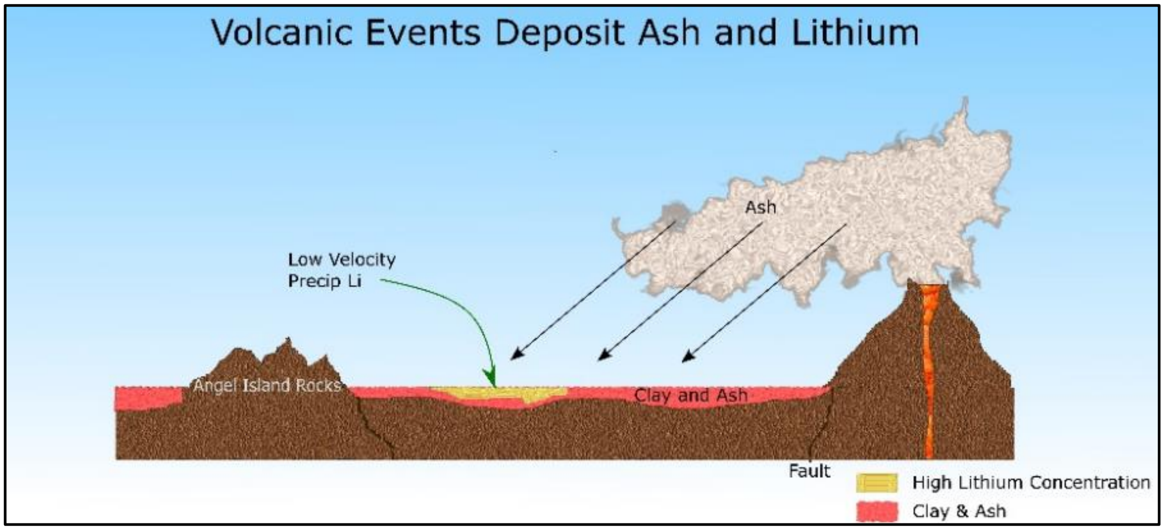


Figure 8-2: Deposit Origin: Erosion of Higher Volcanic Features

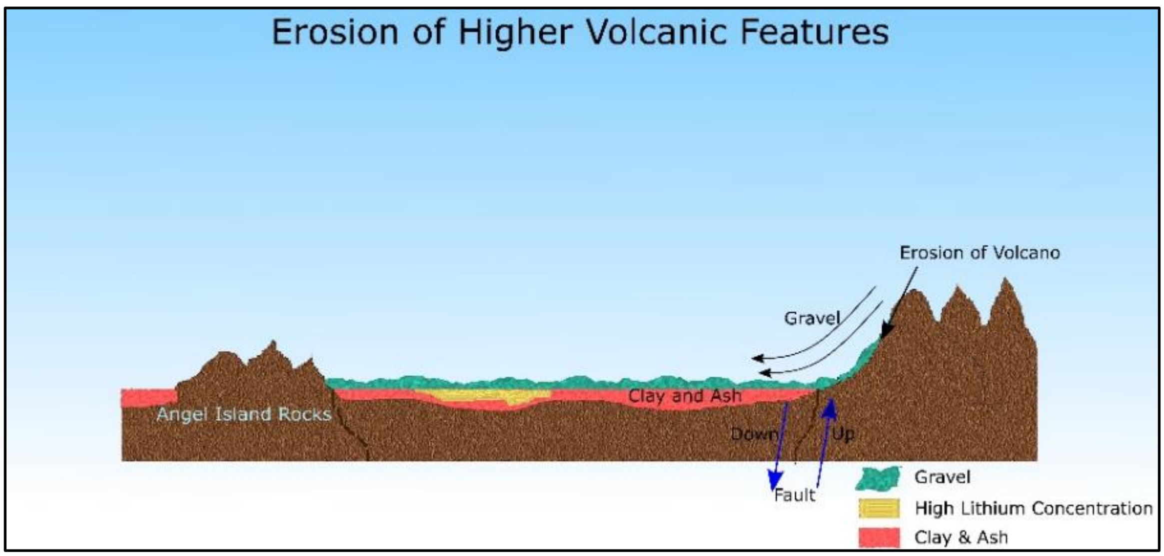
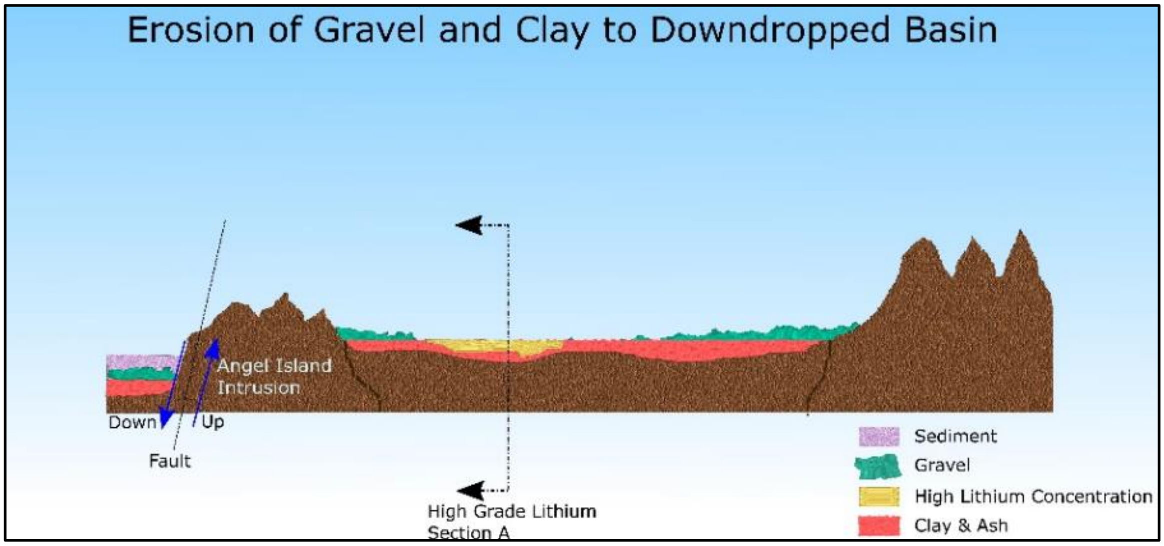


Figure 8-3: Deposit Origin: Erosion of Gravel and Clay



The lithium-bearing sediments of the deposit surround an oxidation-unaltered horizon that is recognizable in drill cores. Based on drilling to date, the highest lithium concentrations occur within a claystone unit with a central unaltered zone inter-layered between two oxidized zones. This distribution of mineralization may be the result of recent, oxidizing surface waters penetrating down dip within more permeable zones of the sedimentary package to create a series of oxidation-unaltered zones.

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.

In 2016, prior to drilling, Cypress collected 634 soil and rock chip samples. Results indicated elevated lithium concentrations over most of the project area. Cypress also conducted surface geologic mapping over most of project. The geologic information is used as a guide for exploration planning in combination with surface samples and drilling results.

The author knows of no further exploration activities carried out by Cypress, except for drilling, that warrant discussion in this report.

10.0 DRILLING

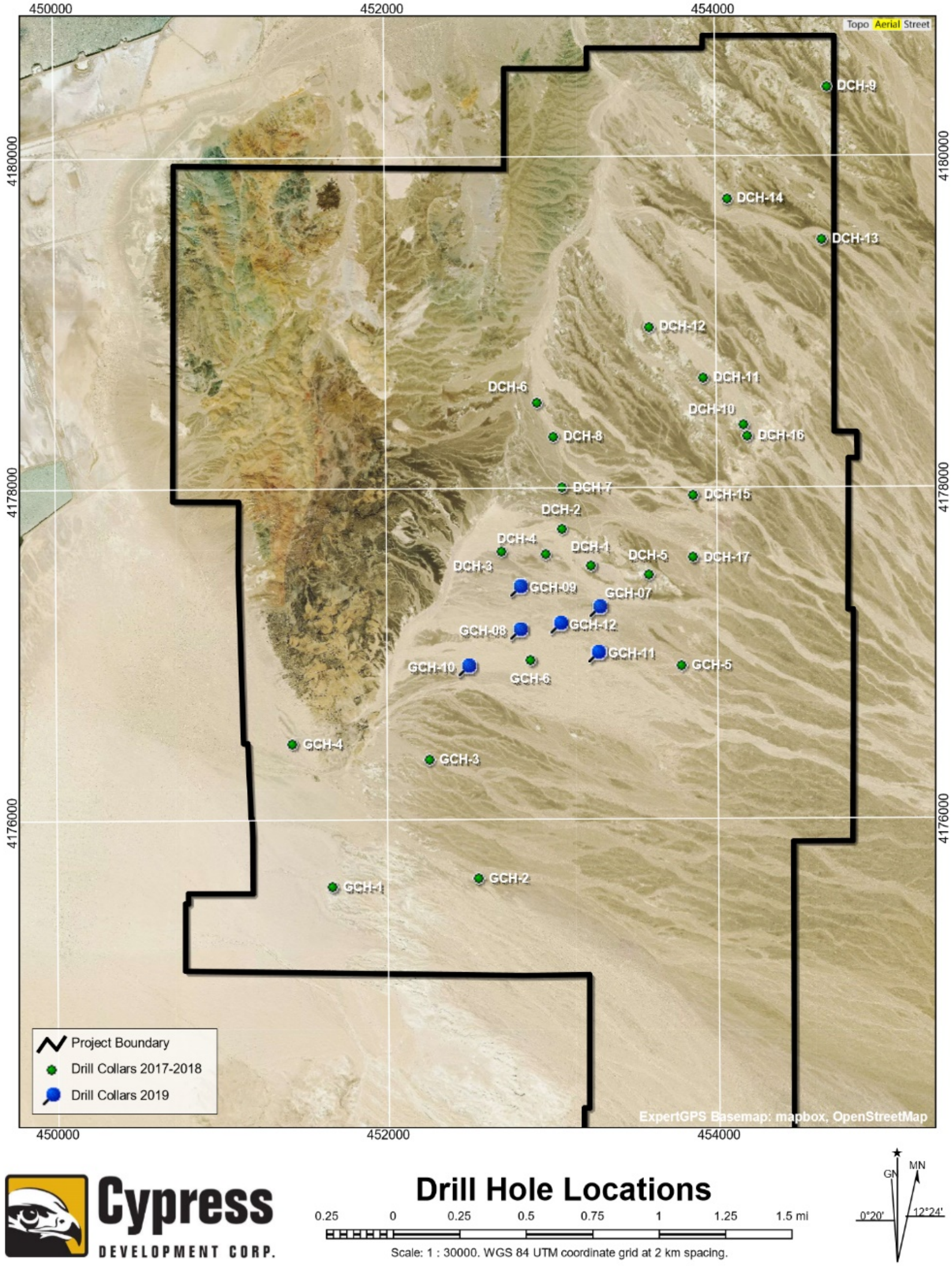
10.1 Cypress Drilling

Cypress drilled at the project in 2017, 2018, and 2019. A total of 29 vertical, NQ-size (1.87-inch diameter) core holes. Drill hole depths from 33 to 142.3 meters (108-467 feet), totaling 2,574.9 meters (8,448 feet) drilled. Downhole surveys were not collected as the holes were all drilled vertically and are relatively shallow in depth. Drilling was completed by Morning Start Drilling of Montana with either a truck- or track-mounted core drill rig. Drill hole collars are listed with coordinates in Table 10-1 and drill hole locations are shown in Figure 10-1.

Table 10-1: Drill Hole Summary

Drill Hole ID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)
DCH-01	453,237	4,177,532	1,362	36.0
DCH-02	453,060	4,177,756	1,355	112.2
DCH-03	452,694	4,177,622	1,353	76.8
DCH-04	452,958	4,177,603	1,355	72.5
DCH-05	453,584	4,177,476	1,366	79.9
DCH-06	452,911	4,178,518	1,351	39.0
DCH-07	453,065	4,178,003	1,362	78.6
DCH-08	453,010	4,178,313	1,354	75.6
DCH-09	454,675	4,180,420	1,345	106.1
DCH-10	454,163	4,178,378	1,367	64.3
DCH-11	453,916	4,178,664	1,354	103.0
DCH-12	453,591	4,178,972	1,345	66.5
DCH-13	454,641	4,179,498	1,359	112.2
DCH-14	454,066	4,179,744	1,341	81.7
DCH-15	453,857	4,177,957	1,376	127.4
DCH-16	454,184	4,178,312	1,368	122.5
DCH-17	453,853	4,177,579	1,381	124.4
GCH-01	451,662	4,175,597	1,331	32.9
GCH-02	452,544	4,175,646	1,362	39.0
GCH-03	452,249	4,176,365	1,346	60.4
GCH-04	451,425	4,176,462	1,320	51.2
GCH-05	453,779	4,176,929	1,390	129.5
GCH-06	452,870	4,176,963	1,359	100.0
2019				
GCH-07	453,275	4,177,272	1,373	142.3
GCH-08	452,795	4,177,136	1,361	111.9
GCH-09	452,798	4,177,401	1,360	118.0
GCH-10	452,485	4,176,918	1,354	93.6
GCH-11	453,273	4,177,000	1,376	124.1
GCH-12	453,039	4,177,175	1,367	113.7

Figure 10-1: Drill Hole Locations Map



10.2 2019 Drilling

Drilling in 2019 was based on the recommendations in the 2018 PEA (GRE, 2018b). The goals were to reduce drill spacing in a favorable mineralized area of the project and attempt to upgrade the resource categories from the 2018 mineral resource estimate (GRE, 2018a). The drilling was planned to generate data from deeper in the deposit, as elevated lithium concentrations persist at depth in all holes except GCH-04 where basement rocks were encountered in 2017.

Cypress utilized a truck-mounted drill rig (Photo 10-1) allowing deeper drilling depths. The six drill holes focused on a 0.5 km² area in the south-central portion of the project. GCH-07 was drilled to 142.3 meters (467 feet) and penetrated over 19 meters into siltstone, the deepest lithological unit drilled to at the project.

Photo 10-1: Drilling GCH-08

All drill cores from the program were delivered to ALS USA Inc. in Reno where they were geologically logged, photographed, and prepped for sample processing and assay. Cores from five of the six holes were processed through sample preparation in their entirety, with coarse reject material retained for use in metallurgical tests. All samples were accompanied by QA/QC samples including blanks, CRM standards and duplicates. Short, < 1-foot intervals, from GCH-09 were selected and submitted and for specific gravity testing, similar size samples were selected from GCH-10, GCH-11 and GCH-12 and submitted for geotechnical testing.

10.3 Drilling Results

Based on drilling to date the subsurface stratigraphy consists of variably interbedded lakebed deposits of silica and ash-rich mudstone and claystone, and occasional tuffaceous zones, all dipping gently to the east. These sediments are underlain by a distinct, siltstone unit in 14 of the 29 drill hole locations. Lithium values in the siltstone are lower than those within the overlying sediments, and this unit represents the extent of drilling carried out to date.

The drilling results indicate a favorable section of claystone up to 120-meters thick, where a strong, apparently planar, alternating oxidation/unaltered zone exists. These zone contacts have distinct color changes in fresh core which fade when dry. The change from oxidized to unaltered is sharp, but often interfingered indicating potential areas of varying permeability. The lithium content through these zones appears consistent, as do other geochemical factors and any specific significance of the oxidation/unaltered zones regarding lithium mineralization is not apparent. The lithium concentration does decrease with depth as the claystone grades into the siltstone unit below.

Significant drill intervals from the 2017-2018 drilling are presented in Table 10-2. Significant drill intervals from the six holes drilled in 2019 are presented in Table 10-3. The 2019 results shown are consistent with the thicknesses and grades of lithium mineralization encountered in the previous drilling.

Table 10-2: 2017-2018 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
DCH-03	8.5	36.0	27.4	999
DCH-04	1.5	51.2	49.7	1,127
DCH-05	8.5	75.6	67.1	1,129
DCH-06	14.6	31.4	16.8	1,013
DCH-07	32.2	51.2	19.0	974
DCH-09	11.3	69.5	58.2	1,093
DCH-10	8.5	64.3	55.8	1,108
DCH-11	8.2	63.4	55.2	1,209
DCH-13	23.8	106.1	82.3	1,221
DCH-15	20.1	124.4	104.2	1,106
DCH-16	14.6	122.5	107.9	1,199
DCH-17	14.6	109.1	94.5	1,050
GCH-04	3.7	29.9	26.2	1,077
GCH-05	84.7	109.7	25.0	1,018
GCH-06	3.0	100.0	96.9	1,142

Table 10-3: 2019 Significant Drill Intervals

Drill Hole ID	Depth (m)		Length (m)	Ave Li (ppm)
	From	To		
GCH-07	2.7	90.5	87.8	1,188
GCH-08	8.2	87.5	84.7	1,229
GCH-09	8.3	72.2	64.0	1,163
GCH-10	3.0	69.2	66.2	1,069
GCH-11	8.2	72.2	64.0	1,176
GCH-12	1.8	81.4	79.6	1,252

10.4 QP Opinion on Adequacy

The drilling, sampling and analytical procedures used by Cypress were reviewed. Based on the review, the QP finds no drilling, sampling, or recovery that might materially impact the accuracy or reliability of the drilling results. Photo 10-2 shows typical excellent core recovery in a 2019 hole.

Photo 10-2: Core from GCH-07

11.0 SAMPLE PRESERVATION, ANALYSES & SECURITY

11.1 Sample Preparation

Samples collected at the project comprise surface samples and NQ-size drill core. Surface samples of outcropping materials or soil were collected by Cypress geologists using standard hand tools, location and material was logged, sample was bagged and marked with number or other designation.

Drill core samples were collected at the drill rig and placed into waxed cardboard boxes by the drill crew. For holes DCH-01 through DCH-17 and GCH-01 through GCH-06, Cypress geologists photographed the core as it was received and collected core recovery information. Sample intervals were selected, primarily 10 feet in length, and split using a cleaver. One half of the core was returned to the box for geologic logging, and the other half was bagged and tagged with sample number. Geologic logging was done in the field or at facilities in Silver Peak, Nevada.

For holes GCH-07 through GCH-12, core was transported to ALS Minerals in Reno, Nevada (ALS) by Cypress personnel. A Cypress geologist utilized logging facilities where each hole was viewed in its entirety for RQD, recovery and geologic logging. The geologist selected and marked sample intervals for assay. Select holes had intervals of < 1-foot removed for geotechnical and specific gravity testing. All core was photographed by ALS staff following logging. ALS staff split any duplicate samples with saw or knife and whole-core samples were bagged and tagged as marked by the geologist for preparation and assay. GCH-12, was split in half over its entire length using saw or knife by ALS staff as marked by the geologist, the right half of the core down-hole was bagged by ALS staff for preparation and assay.

Photo 11-1 shows core from 2019 hole logged and ready for splitting. All core and surface samples were delivered to one of two ISL-certified, independent laboratories, ALS or Bureau Veritas Minerals in Reno, Nevada (BV) by Cypress personnel.

Photo 11-1: Core from GCH-12

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. Cypress has submitted 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.

Samples from holes DCH-01 through DCH-17 and GCH-01 through GCH-06 were analyzed by 33-element, 4-acid inductively coupled plasma (ICP)-atomic emission spectroscopy (AES) or ICP-mass spectrometry (MS) and soil and rock chip samples were analyzed by 33-element 4-acid ICP-AES and/or 35-element aqua regia atomic absorption spectrometry (AAS). Samples from holes GCH-07 through GCH-12, by 60-element, 4-acid ICP-MS, which added the ability to test for rare earth elements at the project.

11.3 Quality Assurance & Quality Control

For most samples collected at the project, Cypress' QA/QC procedures were limited to insertion of a certified reference material (CRM) standard at a rate of one standard sample/30 core samples. These standards were purchased in durable, pre-sealed packets. The standard sample assay results were routinely reviewed by Cypress geologists, and the results fell 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.

Samples from GCH-07 through GCH-12 included QA/QC procedures recommended in the PEA (GRE, 2018b). For every 10 samples submitted, a coarse blank or a CRM OREAS standard was inserted into the sample stream. In addition to CRMs, one sample duplicate, either ½ or ¼ core, was assayed for every 20 samples submitted. The CRMs, standards and duplicate sample assay information all fell within set tolerances and indicated no systematic errors.

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 control until released to the laboratory. Upon receipt by the laboratory, samples are tracked by a sample number assigned and recorded by the geologist. Retained core, sample reject material and pulps are stored at a secure storage facility in Silver Peak, NV (Photo 11-2) or at ALS or BV in Reno, NV.

Photo 11-2: Core Storage

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. Items to consider for the project are, 1) continue to utilize the procedures in place for data collecting, sampling, and QA/QC for analytical work, 2) increase assay confidence through systematic selection of samples for check assays at a second analytical laboratory, 3) continue to review analytical laboratories utilized for future work, and 4) catalogue locations of archived core, sample reject material and pulps.

12.0 DATA VERIFICATION

Data verification efforts included on-site inspections of the project, drilling activity, core storage facility, independent laboratory facilities, check sampling, and auditing of the project database.

12.1 Site Inspections

Since early 2018, drilling and field management has been under the supervision of QP Daniel Kalmbach, CPG. The most recent site visits made by QPs are, Terre Lane in March 2019, Todd Fayram in August 2019.and Daniel Kalmbach in February 2020

In 2018, site inspections were made by GRE consultant and QP J.J. Brown, P.G. Inspection and data verification were conducted for the 2018 Resource Estimate (GRE, 2018a) and the 2018 PEA (GRE, 2018b).

12.2 Drill Collars

Geographic coordinates for all drill hole collar locations were recorded in the field using a hand-held Trimble GPS unit. Drill holes have either temporary (short wood stake) or permanent (rebar and tag) markers erected at their collar locations (Photo 12-1). Drill hole elevations were cross referenced with professional elevation surveys conducted by Strix Imaging of Reno, Nevada in February 2018 and March 2019. All drill hole locations were visited by the author.

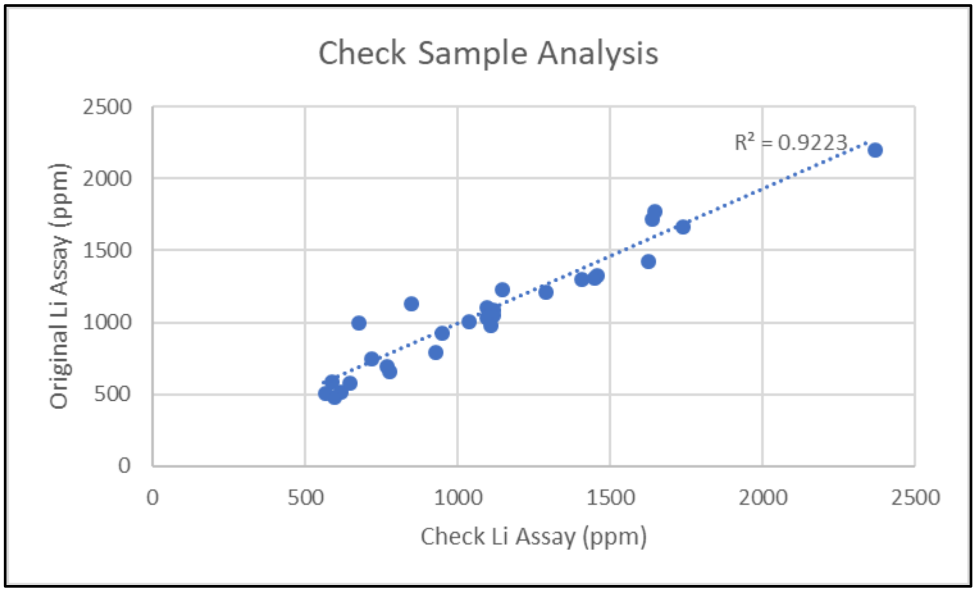
Photo 12-1: Drill Collar Marker at DCH-03

12.3 Check Sampling

In 2018, a check sampling program was conducted. 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 Minerals in Elko, Nevada for analysis using the same sample preparation and analytical procedures as were used for the original samples (ALS, 2018 - 2019). 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 0.92 (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



12.4 Database Audit

A manual audit of the digital project database was completed 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. As more data is collected as the project advances, periodic verification should be performed to maintain accuracy.

12.5 QP Opinion on Adequacy

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

13.0 MINERAL PROCESSING & METALLURGICAL TESTING

Lithium is commonly found absorbed within the lattice structure of illite and smectite clay that make up the mudstone and claystone at the project. Testing to date has determined the lithium is amenable to leaching with dilute sulfuric acid leach, followed by solution purification and concentration to produce a lithium product in the form of lithium carbonate or lithium hydroxide.

Metallurgical testing began in 2017 and is described in previous technical reports and Cypress press releases. Test work was conducted by several laboratories, including CMS (CMS, 2020), Eagle Engineering, ALS Metallurgical Laboratories, SGS Minerals Services, and Hazen Research, Inc. This work included flotation, desliming, leaching and settling tests. The results compiled in this section include new data collected since the PEA (GRE, 2018b).

The metallurgical test work discussed in the PFS was conducted by CMS at its laboratory in Butte, Montana, or under the direction of CMS at other independent third-party laboratories including Pocock Industrial, Andritz (Andritz, 2019), NORAM (NORAM Engineering and Constructors Ltd., 2020), Lilac Solutions, and Colorado School Mines. Testing was supported by analyses of samples shipped from CMS to ALS Minerals in Reno, NV and Vancouver, B.C.

13.1 Mineralogy

Samples were analyzed by x-ray diffraction and scanning electron microscope methods by SGS Mineral Services and Eagle Engineering.

The SGS investigation was conducted on several samples and identified a mixture of illite - montmorillonite clays with lesser silicates.

Eagle Engineering analyzed two samples designated as oxide and reduced material and found that both are composed of 57-61% illite, 17-26% smectite, and 17-22% other silicates. Carbonate minerals dolomite and calcite were identified composing less than 0.25% of the samples.

The designation of oxide and reduced, and color variations were used in the PEA (GRE, 2018b) to look for any differences in material behavior. Subsequent logging and testing indicate the clays are either oxidized or natural, and no significant differences exist in mineralogy with respect to variations in color. References to oxide and reduced are carried in the metallurgical testing for continuity with the previous work.

13.2 Physical Properties

ALS Metallurgical Laboratories, in 2018, determined these physical properties (Photo 13-1):

- Crusher Work Index: 2.5 kiloWatt-hour per tonne (kWh/t)
- Abrasion Index: 0.0001
- Grind Work Index: < 2 kWhr/t
- Disaggregation: material readily decomposes in water under agitation
- Density: range 1.42-1.84 grams per cubic centimeter (g/cm^3)

Photo 13-1: Split Core from DCH-10

13.3 Pulp Viscosities

Viscosity tests were performed by CMS to determine the maximum percent solids for leaching shown in Table 13-1. The tests were done using a Marsh density cup which gives an empirical value for the consistency of a fluid. The number attained depends partly on the effective viscosity at the rate of shear prevailing in the orifice, and partly on the rate of gelation.

Table 13-1: Apparent Viscosity Results

Percent Solids	Apparent Viscosity, CP	Remarks
45	Infinite	No flow, No slump in 5 minutes.
40	Infinite	Extremely viscous, flow stopping after two minutes. 50% of solution left in funnel.
35	42.2	Dripping
30	37.6	Slow Flow
25	30.8	Good flow, fluid

For the tests, clay samples were mixed with a 5% sulfuric acid solution. The mixture naturally degassed and was placed into the funnel where the time required for the funnel to empty was recorded. The results indicate leaching must be kept below 30% solids to mix, pump, and flow properly. Further rheology testing at Pocock Industrial confirmed the leached solids are pseudoplastic at 28% solids and this is an upper limit for leaching.

13.4 Leach Extraction Tests

The process design for the project is based on laboratory tests conducted by CMS, SGS Minerals Services, and Hazen Research, Inc. from 2017-2019. These results indicate lithium extractions of greater than 80% are achievable with an agitated sulfuric acid leach at elevated temperatures in two to six hours of leaching. Samples denoted as oxide and reduced behave similarly. Samples designated as oxide require slightly more time to achieve the same lithium extraction.

Additional test work conducted by CMS in 2019 determined optimum leach conditions for the project with respect to the percentage of solids in slurry, temperatures, and concentrations of sulfuric acid. These conditions were applied to larger scale tests required to generate slurries for use in determining filtration and lithium recovery methods.

Sample Selection and Variability

Prior to leaching on a larger scale, diagnostic tests were conducted on the materials available for creating large sample composites. Materials available were the crushed assay rejects from GCH-06, DCH-15, DCH-16, and DCH-17.

To confirm grades and examine variability in the samples, composite samples were prepared from the available sample intervals from each hole. The composites were characterized by relative depth, lithological horizon and oxidation/weathering state. A 200-gram sample split was prepared from each composite and leached under identical conditions of time, temperature and initial acid concentration.

Table 13-2 shows the head assays obtained from the weighted averages of the assay intervals composing each material composite. These were compared to the back-calculated head grades from the diagnostic leach tests.

Table 13-2: Head Assays of Composite Samples

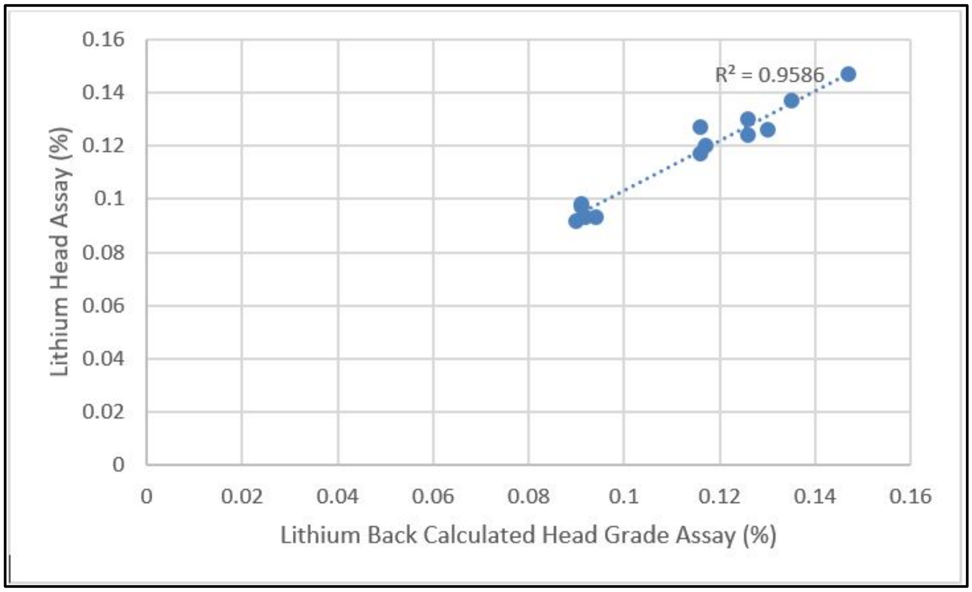
Sample	Range	Li	Li*	Li Extraction	Na	K	Ca	Mg	Al	S
GCH-06	Upper Oxide	0.130	0.126	70	0.72	6.05	4.82	2.78	6.92	0.06
DCH-15		0.091	0.097	66	1.02	4.14	4.55	2.06	6.35	0.01
DCH-16		0.094	0.093	67	1.10	4.29	5.05	2.21	6.63	0.09
DCH-17		0.090	0.092	70	1.12	3.99	5.15	2.30	6.62	0.03
GCH-06	Middle Reduced	0.147	0.147	70	0.70	5.46	4.09	2.96	6.63	0.04
DCH-15	Upper Middle Reduced	0.116	0.127	71	0.81	5.24	3.94	2.10	6.43	0.19
DCH-16		0.117	0.120	68	0.92	5.19	4.54	2.31	6.95	0.27
DCH-17		0.116	0.117	63	1.00	5.11	4.53	2.17	6.77	0.18
DCH-15	Middle Lower Reduced	0.126	0.130	74	0.82	5.00	4.80	2.59	6.72	0.19
DCH-16		0.135	0.137	71	0.83	5.47	4.64	2.50	6.79	0.20
DCH-17		0.126	0.124	72	0.99	5.40	4.59	2.69	6.85	0.17
GCH-06	Lower Reduced	0.092	0.093	63	1.13	4.74	3.54	2.57	5.95	0.01
DCH-15		0.091	0.098	70	0.97	5.32	4.38	2.46	6.51	0.01

*Back-Calculated Head Grade from Leach. All values are in %

Table 13-2 also shows a close range in lithium extractions for all samples under the diagnostic leach conditions, ranging from 63-74% extraction. The results demonstrate there is no discernable difference in lithium extraction due to oxidation state, the depth of sample, or other elements.

Figure 13-1 shows the back-calculated head grades of the composites compare closely with the average head grades from the composited assays with an R^2 of 0.96. This confirms the accuracy of the assay head grades in the composites.

Figure 13-1: Assay Correlation Plot



Large Leach Tests

To provide slurry for rheology, filtration, and lithium recovery testing, two large samples were prepared from the composites and leached at CMS (Table 13-2). Sample L-1was a 92 kg composite prepared from GCH-06 grading 1,380 ppm Li. Sample L-2 was a 41 kg composite prepared from GCH-06 and DCH-15 grading 1,330 ppm Li.

The two samples were leached in a heated 75-gallon jacketed stainless-steel leach vessel. The leach vessel used a high shear, variable speed impeller mixed in a baffled stainless-steel tank. Leaching was conducted at time, temperature and acid concentrations identified by CMS. The leaching conditions are the same as used in the process design and simulate the actual processing conditions.

Results of the large sample tests are shown in Table 13-3. The tests yielded 277 liters (L) of pregnant leach solution (PLS) from sample L-1 and 133 L of PLS from sample L-2. Extractions of lithium into the PLS were 85.5% and 86.8%, respectively. Acid consumptions as determined by titration were 125.7 kg/tonne and 127.2 kg/tonne, respectively.

Table 13-3: Large Leach Results

Sample	Feed kg	Tails kg	PLS L	Feed ppm Li	Tails ppm Li	PLS ppm Li	Extraction %	Acid kg/tonne
L-1 (GCH-06)	92	87.63	277.22	1,380	210	390	85.5	125.7
L-2 (GCH-06, DCH-15)	41.3	41.0	132.29	1,330	210	425	86.8	127.2

Counter Current Leaching

CMS examined counter-current leaching (CCL). Six separate trials were conducted, and each comprised two to four stages of leaching. As many as 16 individual 200-gram charges were run through each trial until steady-state conditions were reached. Information gained determined the effect of varying leach conditions when recycling solutions within the leach circuit.

13.5 Filtration

During the large leach tests, it became apparent the resulting leach slurries were problematic to filter by conventional means. Extensive testing was conducted at Pocock Industrial and Andritz (Andritz, 2019); the tests included:

- Sample Characterization
- Flocculent Screening and Evaluation
- Static and Dynamic Thickener Tests
- Pulp Rheology (FANN Viscosity—Pre-sheared Measurement Only)
- Vacuum and Pressure Filtrations Studies
- Centrifuge Screening Studies

Results from the above tests are:

- The leached slurry does not thicken well, the material settles very slowly and does not compress. The results rule out the use of conventional and high efficiency thickeners.
- Addition of polymer flocculant aides in the flocculation of the slurry.
- Vacuum belt filtration tests produced filtration rates that are uneconomic for the production rate required.
- Filter presses and centrifuges initially appeared viable, but further tests concluded they were uneconomic for the production rate required.
- Specific conditions and equipment were ultimately identified to achieve economic filtration rates for the project.

Solids from filtration tests simulating the final circuit were generated containing a cake moisture of 70 to 75% moisture and were readily washable. The solids generated were suitable for handling by conveyor to a dry-stack tailings facility.

13.6 Lithium Recovery

The process flowsheet in the 2018 PEA (GRE, 2018b) was based on purification-evaporation-crystallization, an approach common to the processing of lithium concentrates from hardrock mines. For the PFS, CMS worked with NORAM Engineering and Constructors Ltd. to develop an alternate approach to more efficiently concentrate the lithium, remove impurities without high reagent consumptions, and recycle sulfuric acid and water back into the leaching circuit. The flowsheet was developed in consultation with vendors. Critical key elements were tested at NORAM's subsidiary company, BC Research Inc., from December 2019 to March 2020.

The NORAM-CMS designed flowsheet uses several stages to concentrate elements into separate solution streams. The flowsheet uses commercially available equipment under process conditions determined by NORAM and CMS to remove impurities (Ca, Mg, Fe, and Al). Concentration of potential by-product rare earth elements also occur in these stages. Approximately 85% of the inflow to the lithium recovery plant is separated and recycled back to leaching. The remaining 15% is treated by evaporation. This is followed by crystallization of salts and recovery of free sulfuric acid. The salts are removed by filtration. Sulfuric acid will be recovered and returned to the leach circuit along with the water recovered from evaporation.

In testing, the resulting evaporates, following salt removal, contain a lithium concentration of 1.85%. This concentrated lithium solution is accompanied by other elements and requires further concentration and acid recovery, pre-treatment for electrochemistry, and the removal of divalents to low levels for the recovery of lithium by electrolysis. The amount of solution for commercial treatment at this point is < 2% of the inflow to the plant. NORAM and CMS are confident the resultant solution is suitable for producing battery-grade lithium in the form of lithium hydroxide monohydrate or lithium carbonate (Table 13-4).

Table 13-4: NORAM—CMS Test Results

Step	Li (ppm)	Mg (ppm)	Ca (ppm)	Fe (ppm)	Al (ppm)
Feed Solution (PLS)	380	3,340	339	2,270	1,395
Pre-Evaporation	360	54	15	40	10
Post-Evaporation and Crystallization	18,500	3,640	120	1,600	3,600

13.7 Ion Exchange Testing

Ion exchange (IX) resins are used commercially to remove metals, cations, and anions from solutions. Lilac Solutions was contracted to test their proprietary lithium IX resin. Results are positive at higher levels of pH than present in the PLS feed solution. Greater than 85% of the lithium was stripped into a solution grading 5,000 ppm Li with low levels of Na and K. Further testing of the Lilac resin remains an option.

13.8 Potential By-products

Rare earth elements (REEs) were found at elevated levels in the lithium recovery process along with Mg, Ca and other elements. Whether these elements are recoverable and represent a revenue source remains to be determined. Any contribution from by-products was not considered in this PFS.

13.9 Conclusions & Interpretation

- The processing methods are projected to effectively recover lithium from the project's mineralized materials.
- Lithium extractions of 85-87% were achieved in large sample leach tests.
- An overall recovery rate of 83% is used in the economic analysis to allow for possible losses of lithium in the recycle streams from the lithium recovery plant.
- Acid consumptions averaged 126.5 kg/tonne in the large leach test. Recovery and recycling of unused acid is expected within the processing flowsheet.
- To advance the project to the feasibility level, further test work is needed. Test work should include a pilot plant study conducted at a continuous production of at least one tonne per day (tpd) of claystone.

14.0 MINERAL RESOURCE ESTIMATE

The Mineral Resource Estimate reported for the PFS 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 Seequent Leapfrog® software and using additional information from drilling since the 2018 PEA.

14.1 Definitions

The 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.

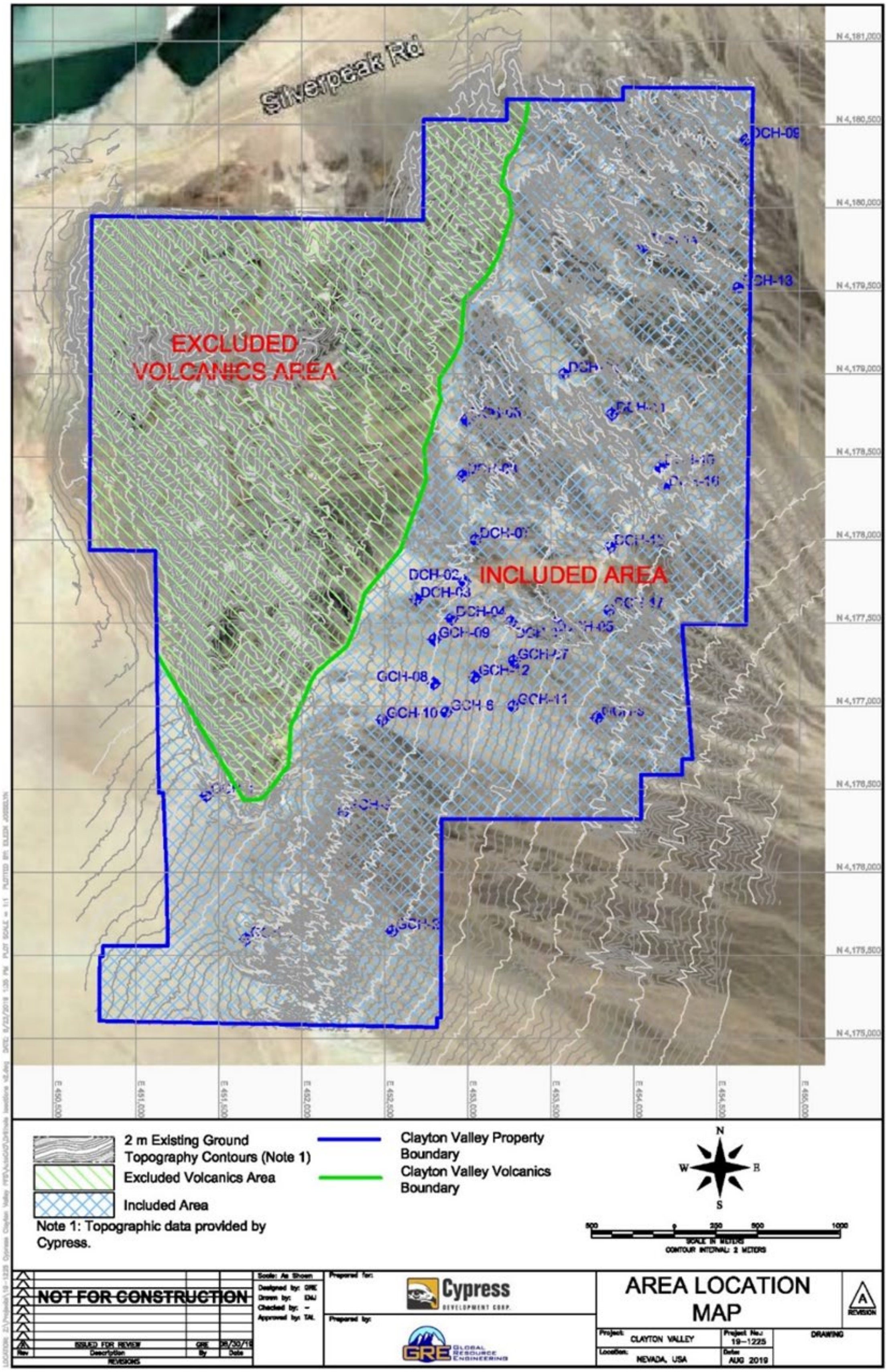
14.2 Geologic Model

The 3-D geologic model for resource estimation was constructed using Seequent Leapfrog® software.

The model is limited to Cypress property shown in Figure 14-1. This property boundary does not include Cypress claims that were under litigation in 2019 and 2020. The boundary differs from the one shown in Section 4.0 and 18.0 which post-dates the settlement of the litigation. The post-settlement boundary in these sections is inclusive of the boundary in Figure 14-1 and the mineral resource estimate.

The mineral resource estimate includes all sedimentary units located in the eastern and southern part of this property. There is no drilling or known lithium mineralization in the rock units which make up Angel Island, so this area is excluded from the mineral resource estimate.

Figure 14-1: Area Included in the Geologic Model and Mineral Resource Estimation

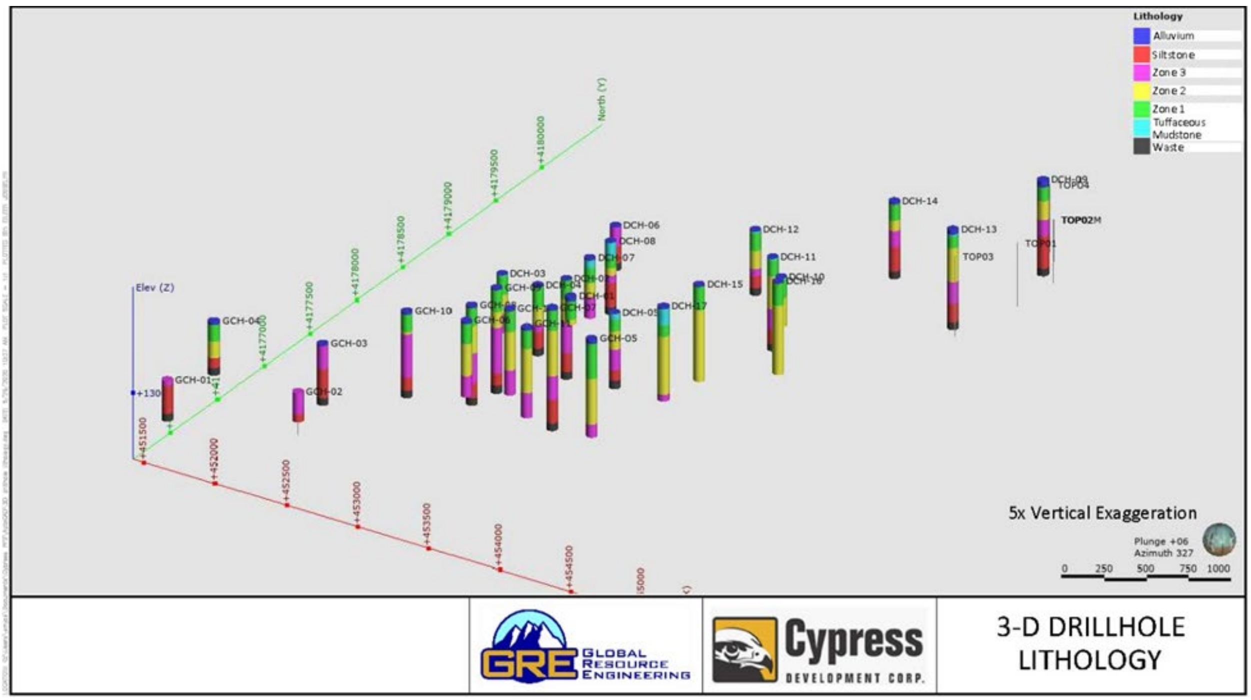


14.3 Data Used for the Lithium Estimation

14.3.1 Drill Holes

The mineral resource estimate incorporates geologic and assay results from drilling of 29 drill holes on the project. The drill hole data was compiled and verified for all drill holes, collar coordinates, drill hole direction (azimuth and dip), lithology, sampling, and assay data. This study uses 29 drill holes, totaling 2,590 meters, with an average depth of 89.3 meters per hole. All drill holes are vertical and limited to the sedimentary rock units on the property. Figure 14-2 shows a 3-D view of the drill hole lithologies. Topography was derived from aerial drone surveys completed in 2018 and 2019. Drill collars were located with GPS readings and checked against the competed topographic base.

Figure 14-2: Projected 3-D View of Drill Hole Lithologies



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 data set included 891 lithium assay values in ppm.

14.3.3 Specific Gravity

For resource modelling, a specific gravity (SG) of 1.7 g/cm^3 is used for all lithological units. Within the claystone zones that comprise most of the mineral resource, representative samples of fresh core were collected for specific gravity measurements. The samples were selected from GCH-9 (Photo 14-1), sealed in wax with specific gravities determined by ALS using the volume displacement method. The results ranged 1.47 to 1.72 g/cm^3 with a mean of 1.64 g/cm^3. The range of values were comparable to previous laboratory results. Additional lithology-specific testing is recommended for future study.

Photo 14-1: Core from GCH-09 Showing Specific Gravity Sample

14.4 Domains

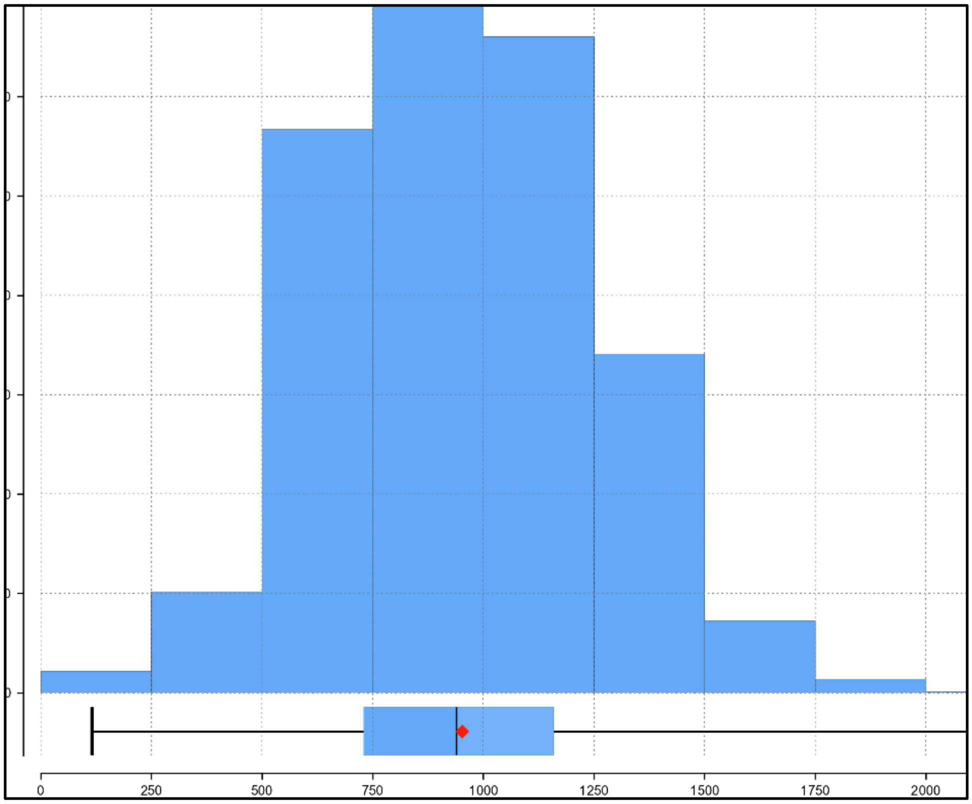
Within Leapfrog®, the alluvium lithological unit was excluded from the resource estimation. The tuffaceous mudstone and siltstone lithological units were identified as separate domains during resource estimation. The three zones of the claystone lithological unit were combined into a single domain to perform the resource estimation.

14.5 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.

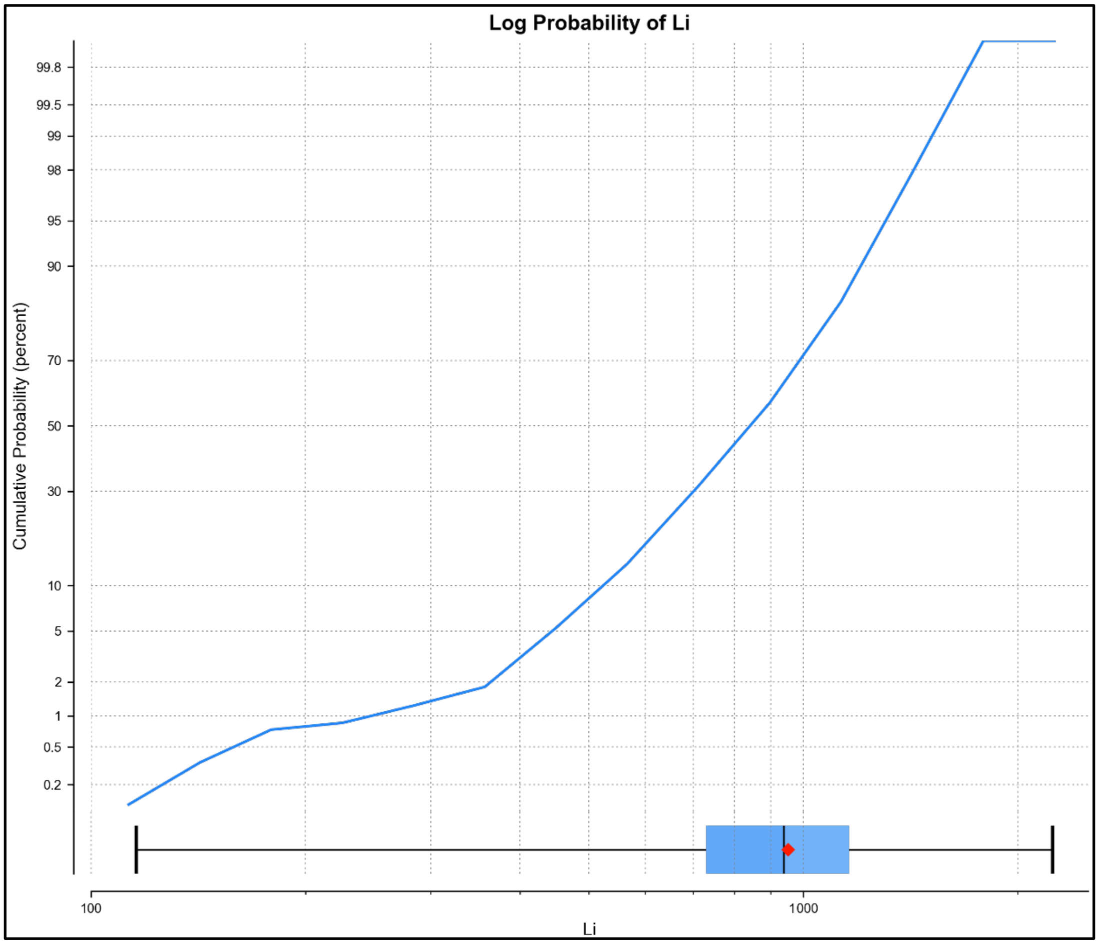
The assay data (excluding alluvium) contains a total of 891 lithium assays, ranging from 115.7 ppm to 2,240 ppm. A histogram of the project's assay data is shown in Figure 14-3.

Figure 14-3: CVLP Lithium Assay Data Histogram



A cumulative frequency plot (CFP) of the assay data is shown in Figure 14-4. The CFP 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.

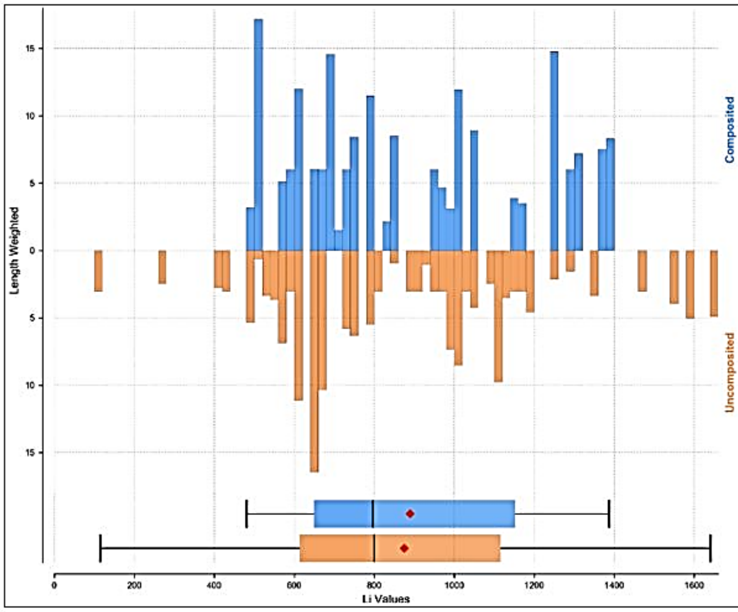
Figure 14-4: CVLP Cumulative Frequency Plot of Lithium Assay Data



14.5.1 Composite Assay Intervals

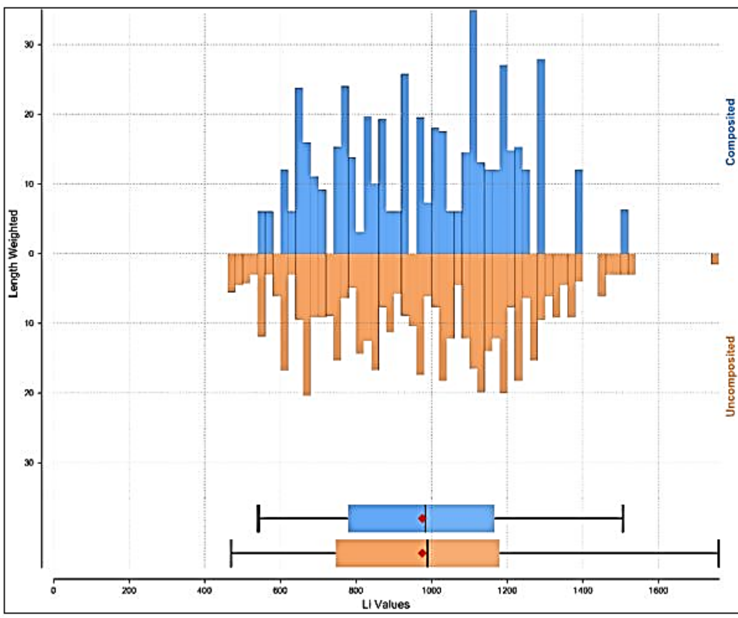
The project's assaying was done almost exclusively using 1.52- or 3.048-meter long (or 5- or 10-foot long) sample intervals. GRE composited each drill hole to 6-meter intervals within each domain. The 6-meter composite length was selected based upon the anticipated bench height in mining. The model generated using 6-meter composites was later compared to one using only three meters for the composite lengths. There was no significant difference in the grade distribution with the shorter sample length, indicating the 6-meter composites were appropriate for the resource estimate. Comparisons of the assay data and composited data, by domain, are shown in Figure 14-5 through Figure 14-7.

Figure 14-5: Tuffaceous Mudstone Comparison of Assay and Composited Data



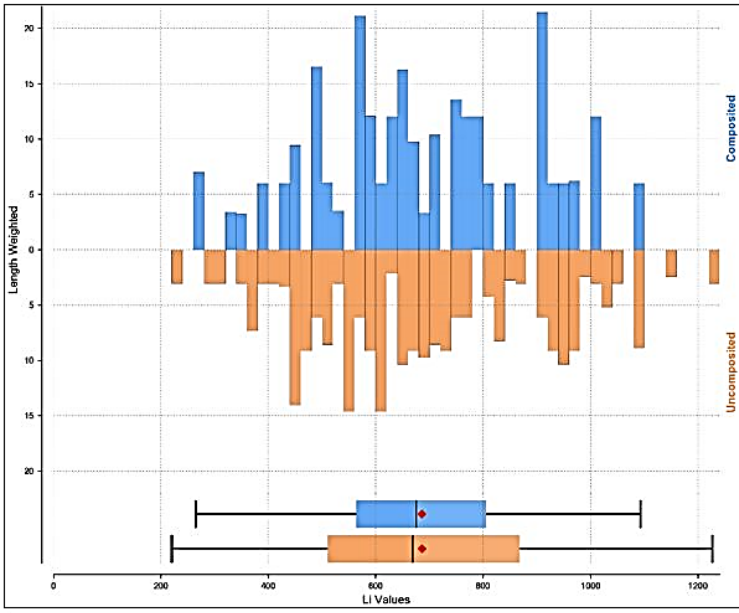
	Composited	Uncomposited
Count	34	72
Length	194.05	180.37
Mean	889.44	875.43
SD	288.67	340.47
CV	0.32	0.39
Variance	83,331.34	115,922.89
Minimum	481.60	115.70
Q1	648.83	613.10
Q2	796.99	800.00
Q3	1,153.10	1,115.90
Maximum	1,387.41	1,641.00

Figure 14-6: Claystone Comparison of Assay and Composited Data



	Composited	Uncomposited
Count	85	179
Length	508.41	499.70
Mean	975.55	975.65
SD	226.59	264.51
CV	0.23	0.27
Variance	51,340.92	69,967.06
Minimum	541.99	470.00
Q1	778.62	746.00
Q2	983.62	990.00
Q3	1,166.98	1,180.00
Maximum	1,505.72	1,760.00

Figure 14-7: Siltstone Comparison of Assay and Composited Data



	Composited	Uncomposited
Count	48	88
Length	259.45	256.64
Mean	686.08	686.32
SD	199.30	225.49
CV	0.29	0.33
Variance	39,720.22	50,844.31
Minimum	264.48	220.00
Q1	563.58	510.20
Q2	675.29	668.90
Q3	806.35	868.00
Maximum	1,093.05	1,227.00

14.6 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 lithium grade within each of the beds. All drill holes intersected the mineralized beds. The southern portion of the property appears to be in an uplifted fault block. No drill hole passed through the lowest (siltstone) unit; all drill holes ended with lithium values above 300 ppm, with exception to GCH-04 which ended in Angle Island rocks.

14.6.1 Variography

GRE generated pair-wise variograms from the composite values using Leapfrog® Edge software. The analysis was used to determine the size and orientation of the search ellipsoid for an inverse distance squared (ID^2) grade estimate. Each domain was analyzed to determine the orientation and relative length of the search ellipsoid axes, nugget, and sill. Based on the results of the variography, the search parameters used in the grade estimation were as shown on Table 14-1. Figure 14-8, Figure 14-9, and Figure 14-10 show the variograms and radial graphs for each domain, the major axis was determined to be at an azimuth of 120° for the tuffaceous mudstone and siltstone domains and at an azimuth of 34.5° for the claystone domain.

Figure 14-8: Tuffaceous Mudstone Variograms

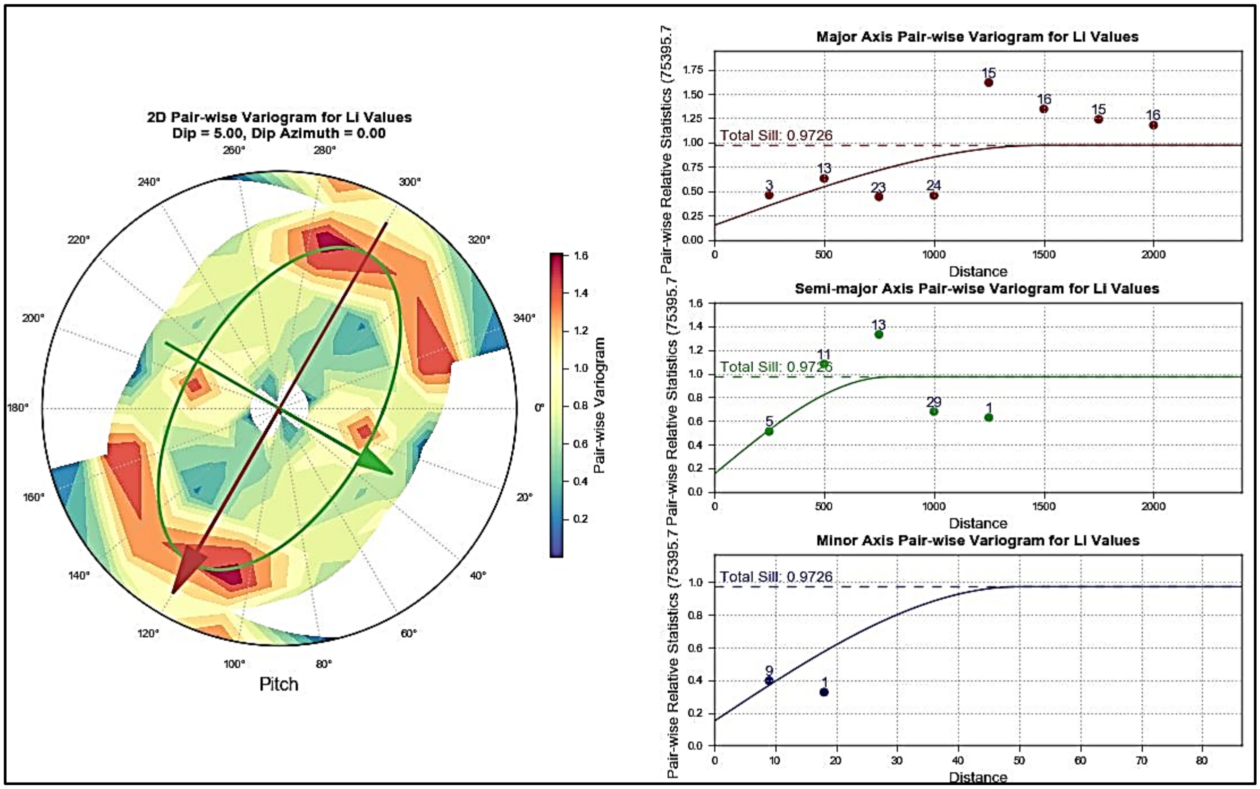


Figure 14-9: Claystone Variograms

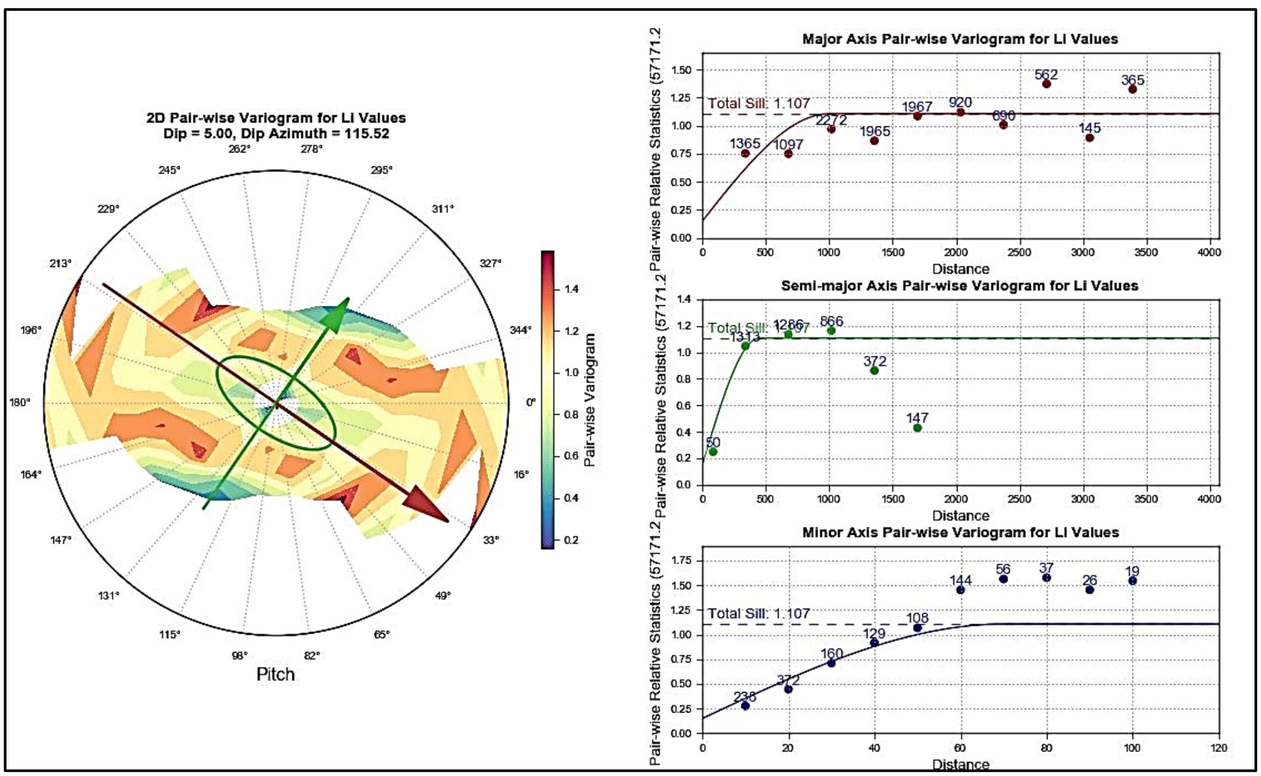


Figure 14-10: Siltstone Variograms

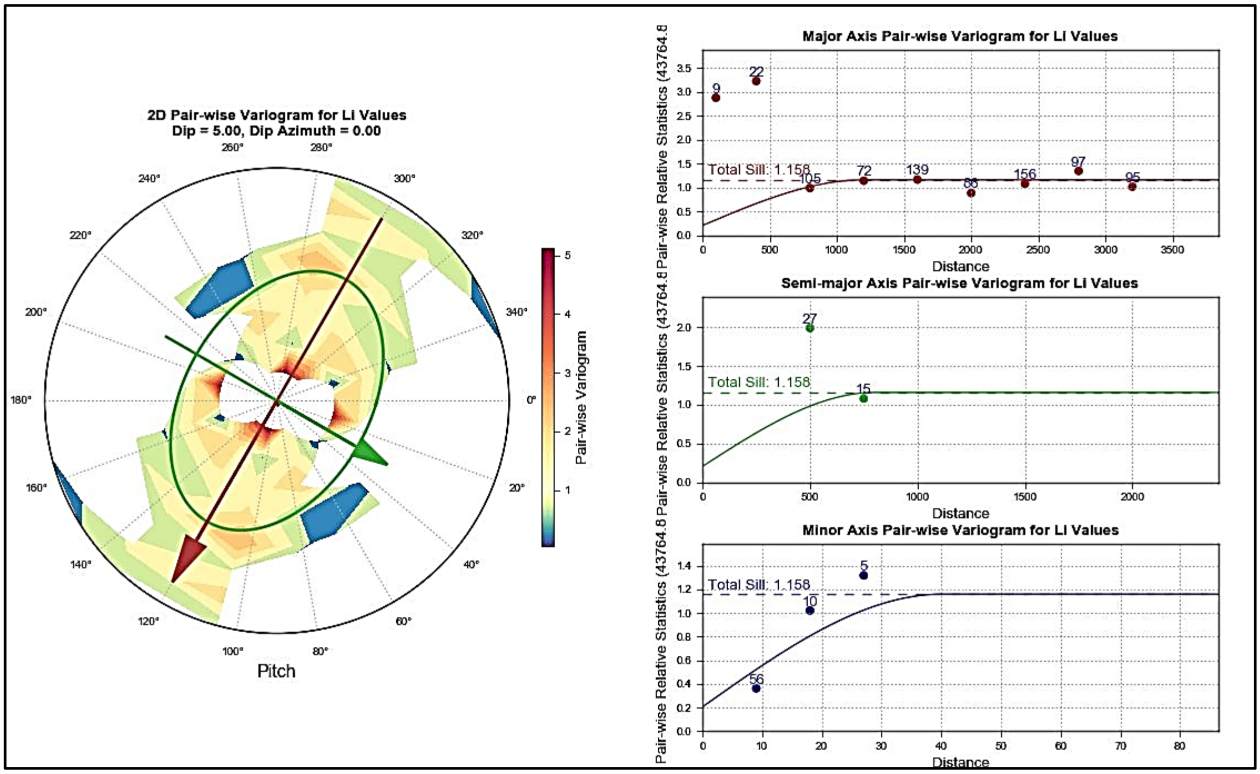


Table 14-1: Variography Results by Domain

Domain	Nugget	Sill	Orientation	Dip	Major Axis Range (m)	Semi-Major Axis Range (m)	Minor Axis Range (m)
Tuffaceous Mudstone	0.18	0.9726	120°	5°	1,500	800	50
Claystone all zones	0.2	1.107	34.5°	5°	1,000	450	70
Siltstone	0.2	1.158	120°	5°	1,200	800	40

14.6.2 Grade Modeling & Resource Categories

Grade was estimated using an ID² algorithm from a minimum of four composites and a maximum of 20 composites. The Mineral Resource was categorized as measured if there were two drill holes within the search radius, indicated if there was only one drill hole in the search radius, and all remaining mineralized areas were considered as inferred.

These parameters are consistent with typical industry practice. A plan view showing the resource category ranges is provided in Figure 14-11. For the five mineralized lithological units, average lithium grade/meter thickness is shown in Figure 14-12 through Figure 14-16.

Figure 14-11: Plan View of Resource Category Ranges

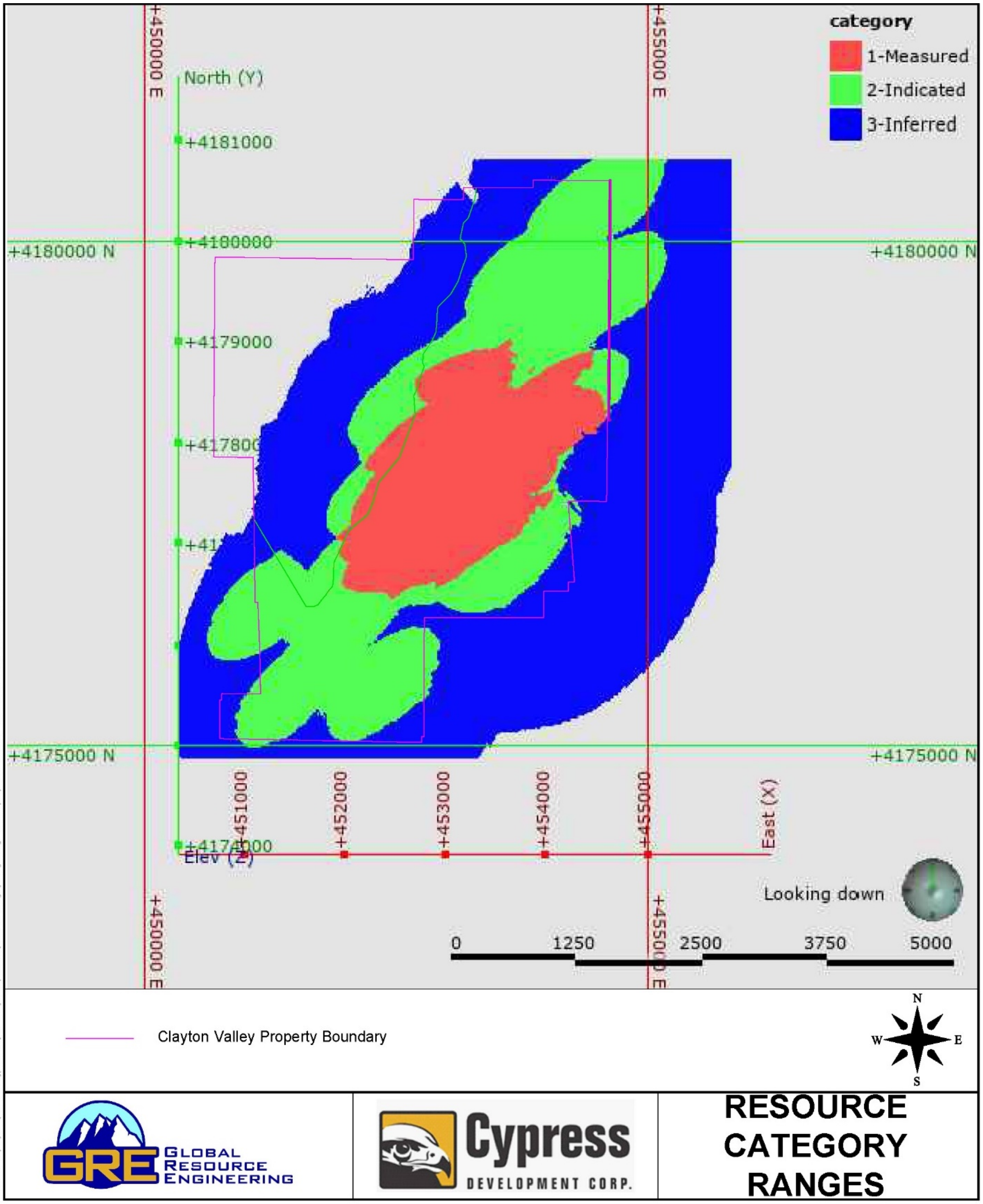


Figure 14-12: Plan View of Modeled Lithium Grades for Tuffaceous Mudstone

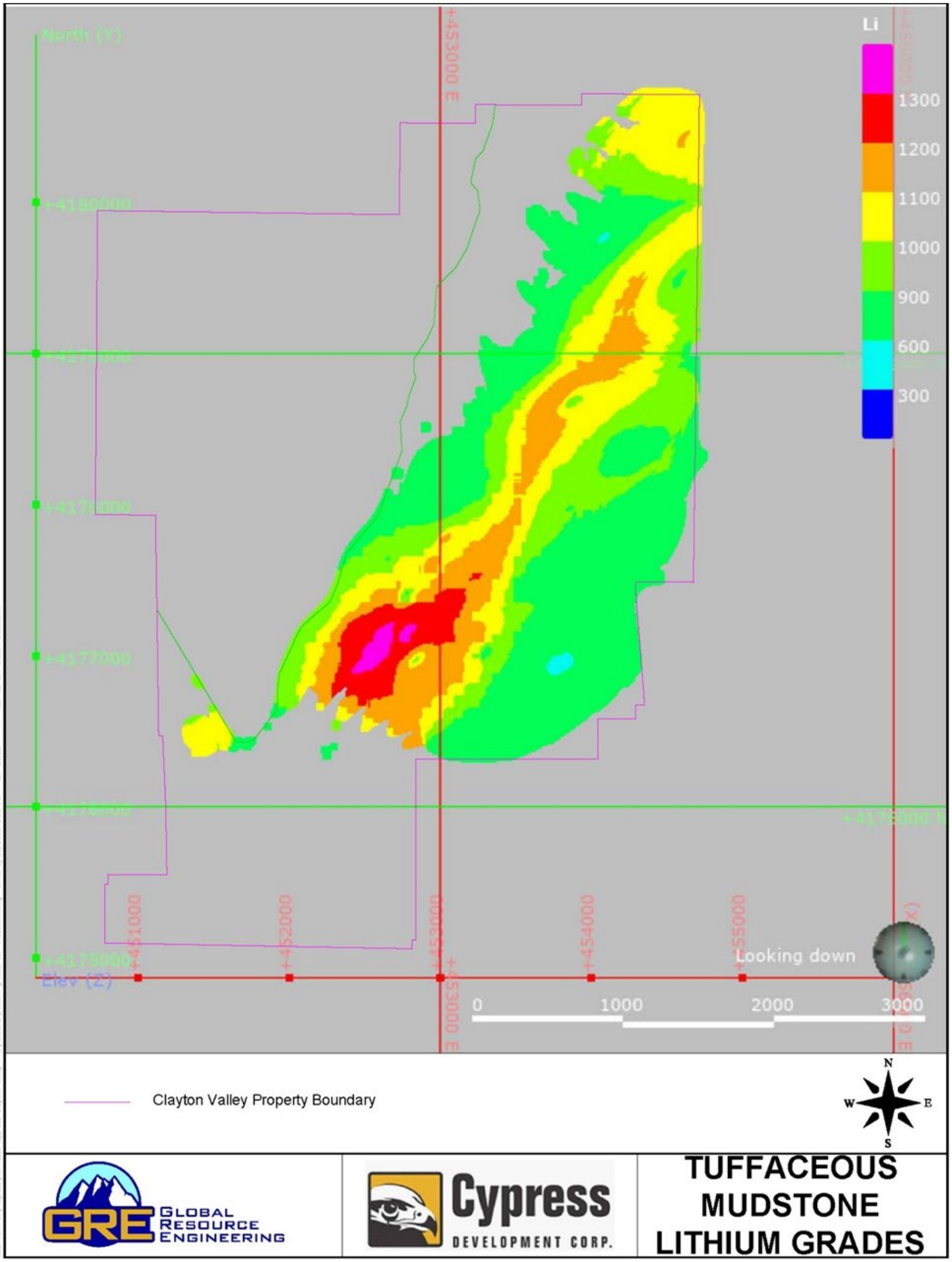


Figure 14-13: Plan View of Modeled Lithium Grades for Claystone Zone 1

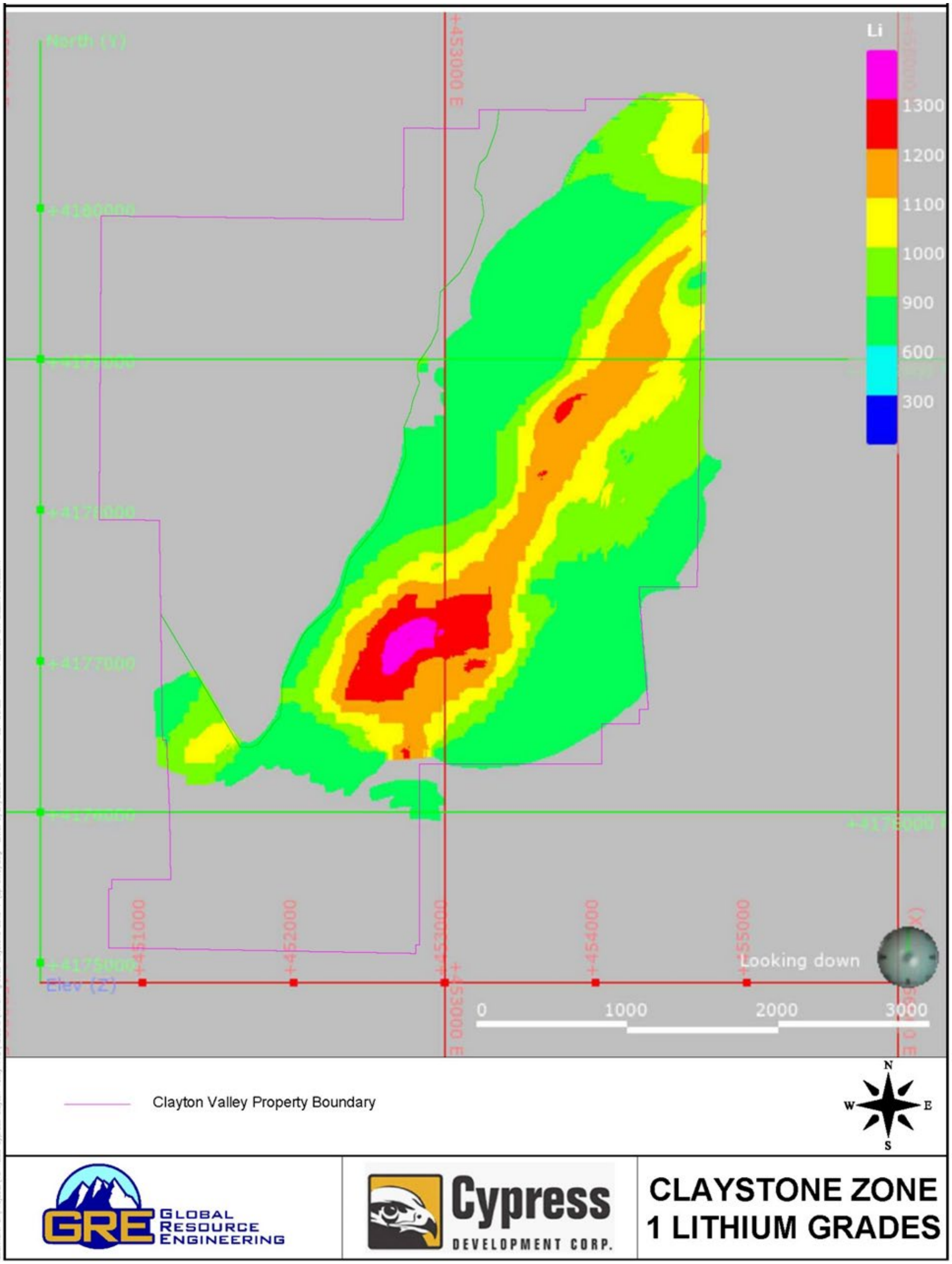


Figure 14-14: Plan View of Modeled Lithium Grades for Claystone Zone 2

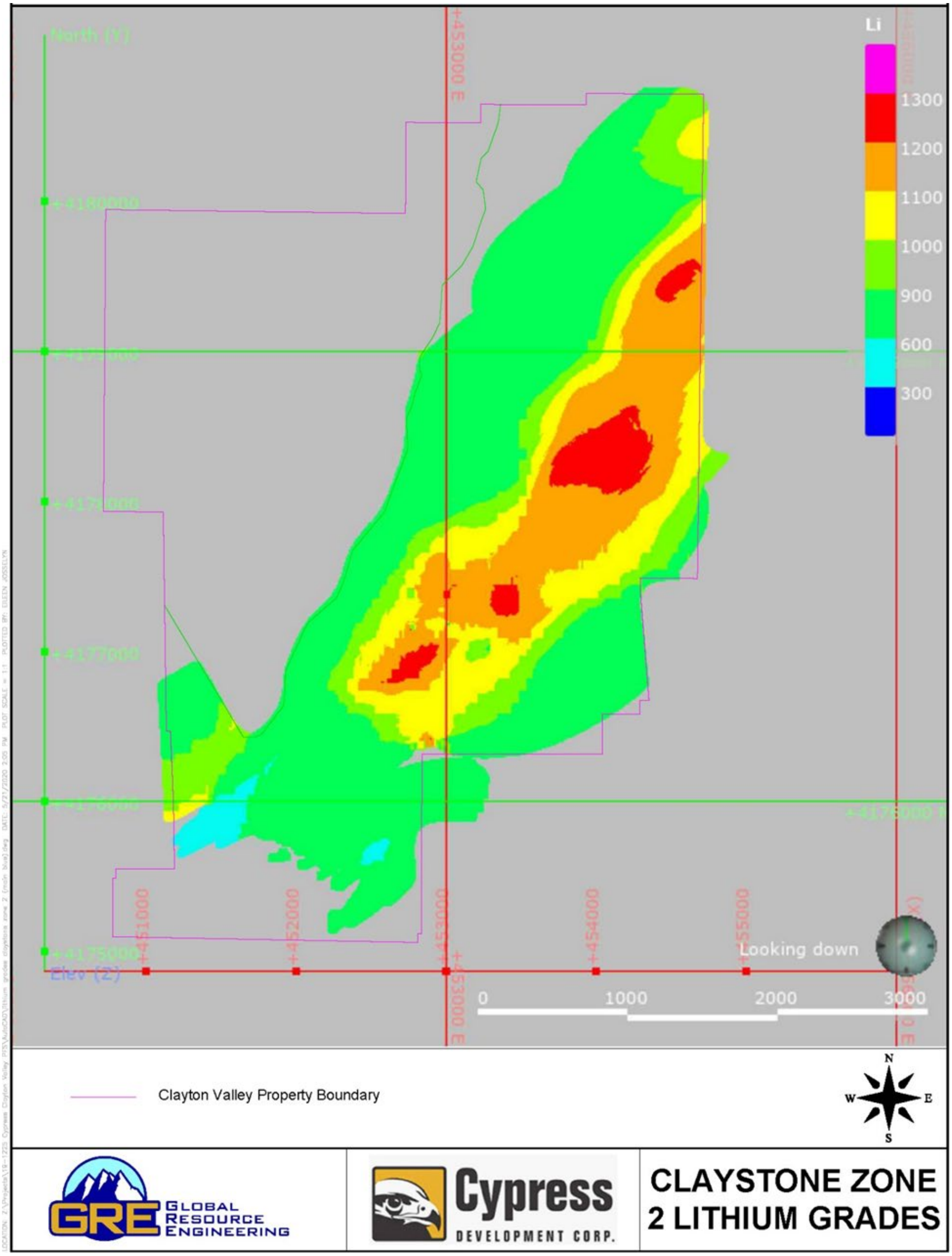


Figure 14-15: Plan View of Modeled Lithium Grades for Claystone Zone 3

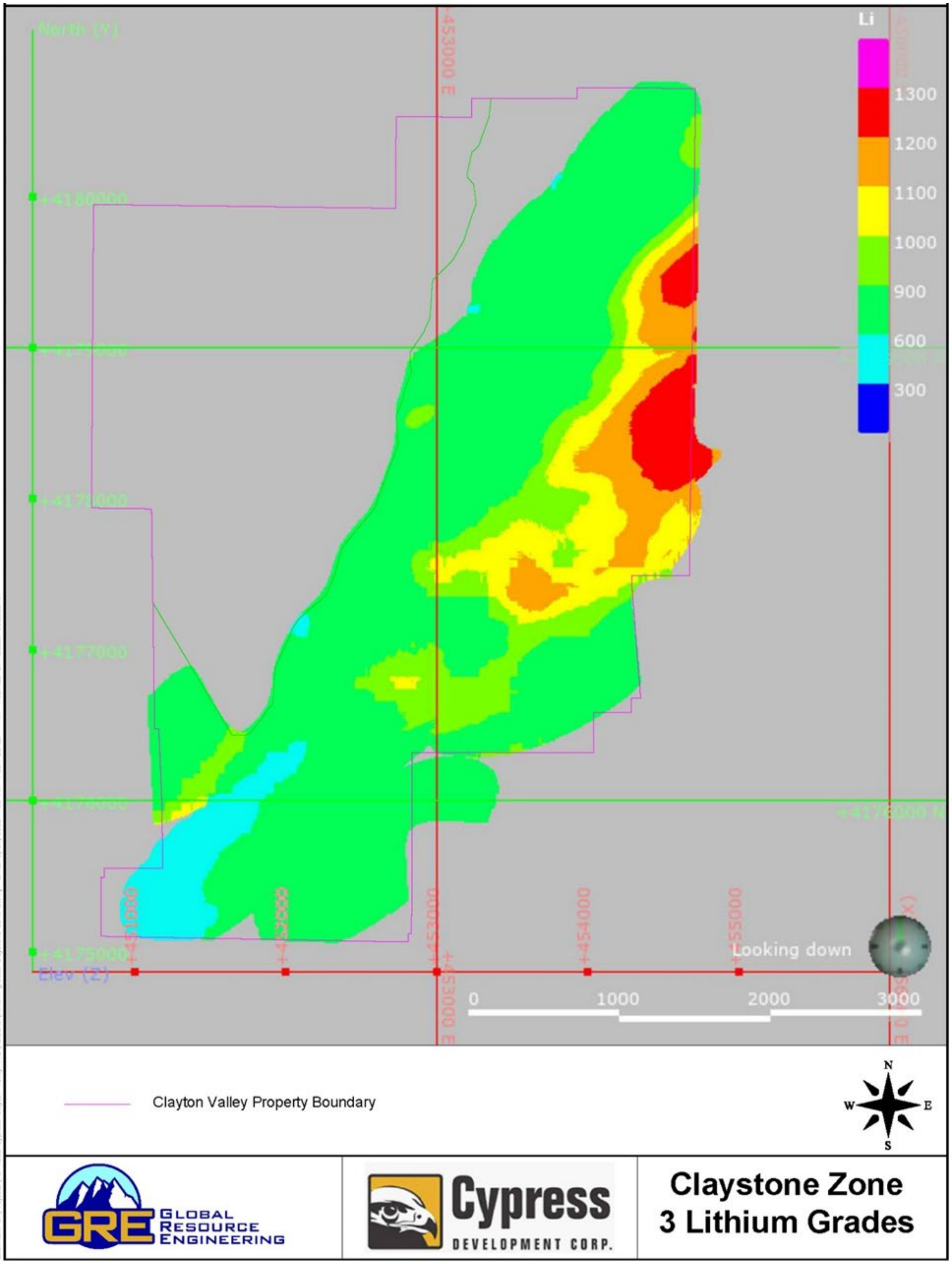
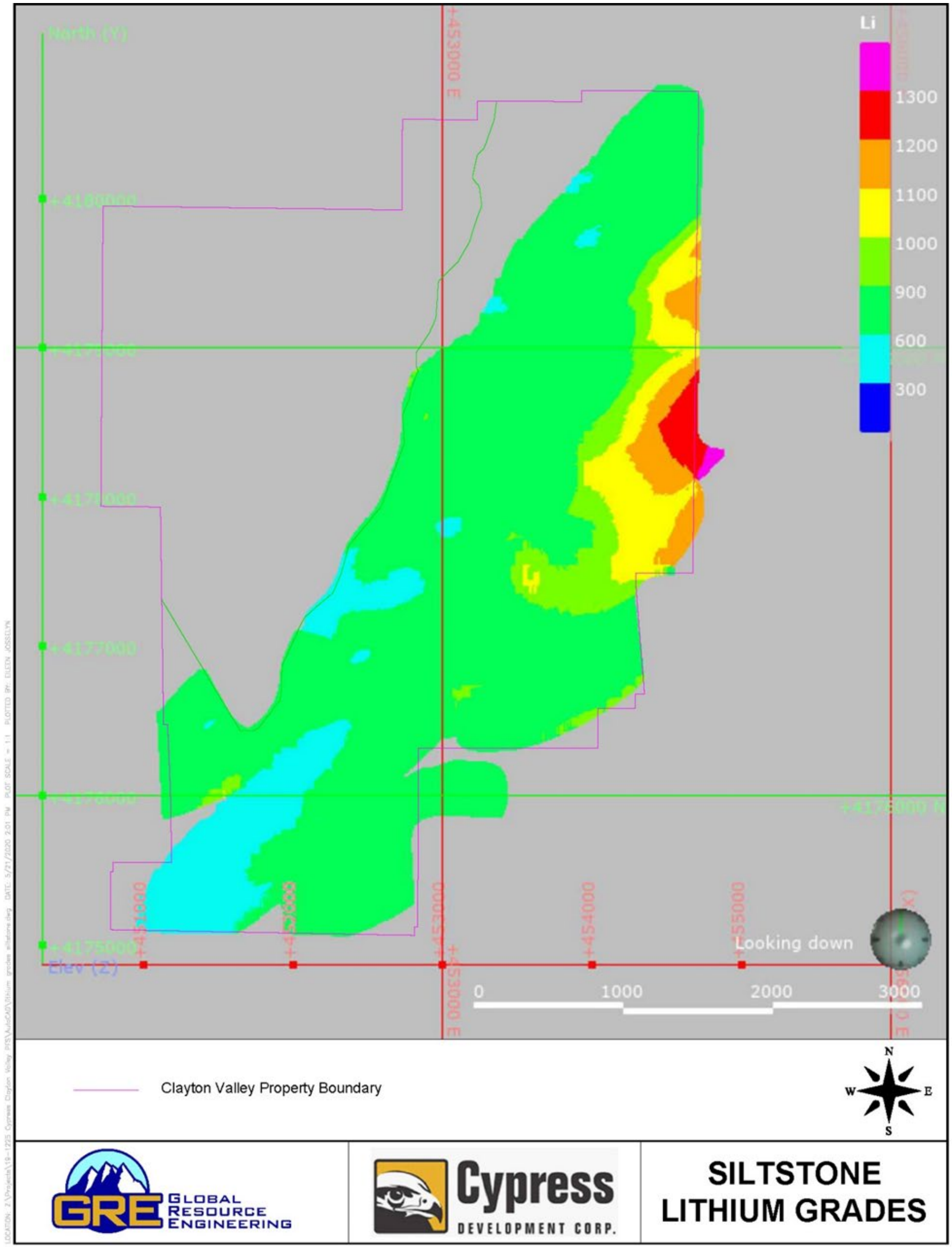


Figure 14-16: Plan View of Modeled Lithium Grades for Siltstone



14.7 Mineral Resource Estimate

The Mineral Resource Estimate is summarized in Table 14-2 and Table 14-3. The estimation uses the data from all 29 drill holes and encompasses the property as shown in Figure 14-1 and described in Section 4.0. 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.

14.7.1 Cutoff Grades

Cutoff grades for lithium of 300 ppm, 600 ppm, and 900 ppm are used in the tables. The tonnage listed for each cutoff is the total amount of material within the resource classification above that Cutoff, where the 300-ppm cutoff includes all material above the 600 and 900-ppm cutoffs, and the 600-ppm cutoff includes the material above 900 ppm.

The three cutoffs are selected by GRE based on economics, distribution of grades in the block model, and optimization of the mine schedule.

- The economic break-even Cutoff grade for the parameters within this PFS is between 300 and 600 ppm and is closer to 300 ppm as follows:
  - Break-even = operating cost / (recovery x price)
    - Price/tonne for lithium = \$9,500/t x 5.323 = \$50,568/t
    - Break-even = \$16.78 / (83% × \$50,568/kg x 10^6) = 400 ppm
- Most of the material in the block model is above 600 ppm Li. Table 14-1 shows the resources are similar for the 300 ppm and 600 ppm cutoff grades. This is particularly true for the material in Zones 1-3.
- For purposes of grade optimization, GRE selected 900 ppm as the cutoff grade for mill feed. All material between 600 ppm and 900 ppm is designated “low grade” for stockpile. Material below 600 ppm is designated waste.

Based on these considerations and the mine production schedule, the 900-ppm cutoff grade for mill feed is used for reporting the Mineral Resource.

14.7.2 Global Mineral Resources

As listed in Table 14-2, at a cutoff grade of 900 ppm, the Measured Resources total 500.8 million tonnes at an average grade of 1,086 ppm Li. The Indicated Resource totals 287.2 million tonnes at 1,033 ppm Li for a total of Measured plus Indicated Resources of 788 million tonnes at 1,067 ppm Li. The Inferred Resource is 79.9 million tonnes averaging 1,033 ppm Li, which is about 10% of the Measured and Indicated totals.

Comparison with the figures in the 600 ppm and 300 ppm columns shows the large size of the deposit, which has 1.6 to 1.7 billion tonnes in Measured and Indicated categories and another 405 to 415 million tonnes in Inferred.

14.7.3 Mineral Resource

The reported Mineral Resource is pit constrained. GRE created an "ultimate" pit that extends to most property boundaries and is bounded by Angel Island rocks in the west, as shown in Figure 14-17. The ultimate pit shell uses the slope angles described in Section 16.0 with no set-back from property lines.

Using a 900 ppm Li cutoff, the pit-constrained Mineral Resources total 432.4 million tonnes averaging 1,088 ppm Li in the Measured Resource and 160.9 million tonnes at 1,032 ppm Li in the Indicated Resource, for a total of 593.3 million tonnes at 1,073 ppm Li in Measured and Indicated Resources. The Inferred Resource is 2.3 million tonnes averaging 1,005 ppm Li.

The application of the boundary constraints as seen reduce the Measured and Indicated total material by about 25%, from 788 million tonnes to 593.3 million tonnes. Application of the ultimate pit shell removes most of the Inferred Resource blocks, dropping from 79.9 million tonnes to 2.3 million tonnes.

Lithium contained in the pit-constrained Measured and Indicated Resources totals 636.4 million kg Li, or 3.387 million tonnes of LCE.

The constrained pit shell and Mineral Resource is used to derive the Mineral Reserves in Section 14.8.2 and the mine production schedule in Section 16.0.

Table 14-2: Global Mineral Resource Estimate

Domain	**Tonnes Above Cutoff (millions)**	**Li Grade (ppm)**	**Li Contained (million kg)**	**Tonnes Above Cutoff (millions)**	**Li Grade (ppm)**	**Li Contained (million kg)**	**Tonnes Above Cutoff (millions)**	**Li Grade (ppm)**	**Li Contained (million kg)**
	300 ppm Cutoff			**600 ppm Cutoff**			**900 ppm Cutoff**		
Measured									
Tuffaceous mudstone	49.4	883	43.7	48.2	891	42.9	19.9	1,060	21.1
Claystone all zones	633.2	1,015	642.8	633.0	1,015	642.7	459.3	797	366.2
Siltstone	164.9	702	115.7	121.3	774	93.9	21.6	1,021	22.1
Total	**847.5**	**947**	**802.2**	**802.5**	**971**	**779.5**	**500.8**	**1,086**	**544.0**
Indicated									
Tuffaceous mudstone	33.0	867	28.6	31.6	880	27.8	15.1	1,040	15.7
Claystone all zones	595.7	870	518.2	589.8	873	514.8	210.5	1,039	218.8
Siltstone	228.9	773	177.0	181.8	834	151.6	61.5	1,009	62.1
Total	**857.7**	**844**	**723.8**	**803.2**	**864**	**694.2**	**287.2**	**1,033**	**296.6**
Measured + Indicated									
Tuffaceous mudstone	82.4	876	72.2	79.7	887	70.7	35.0	1,052	36.9
Claystone all zones	1,228.9	945	1,161.0	1,222.8	947	1,157.5	669.8	1,074	719.6
Siltstone	393.8	743	292.7	303.1	810	245.5	83.1	1,012	84.1
Total	**1,705.2**	**895**	**1,525.8**	**1,605.7**	**918**	**1,473.7**	**788.0**	**1,067**	**840.6**
Inferred									
Tuffaceous mudstone	12.8	683	8.7	12.7	684	8.7	0.2	930	0.2
Claystone all zones	287.3	757	217.4	284.0	759	215.6	31.2	1,004	31.3
Siltstone	115.1	869	100.0	108.7	886	96.4	48.5	1,051	51.0
Total	**415.2**	**786**	**326.2**	**405.4**	**791**	**320.6**	**79.9**	**1,033**	**82.5**

Figure 14-17: Constrained Pit Outline

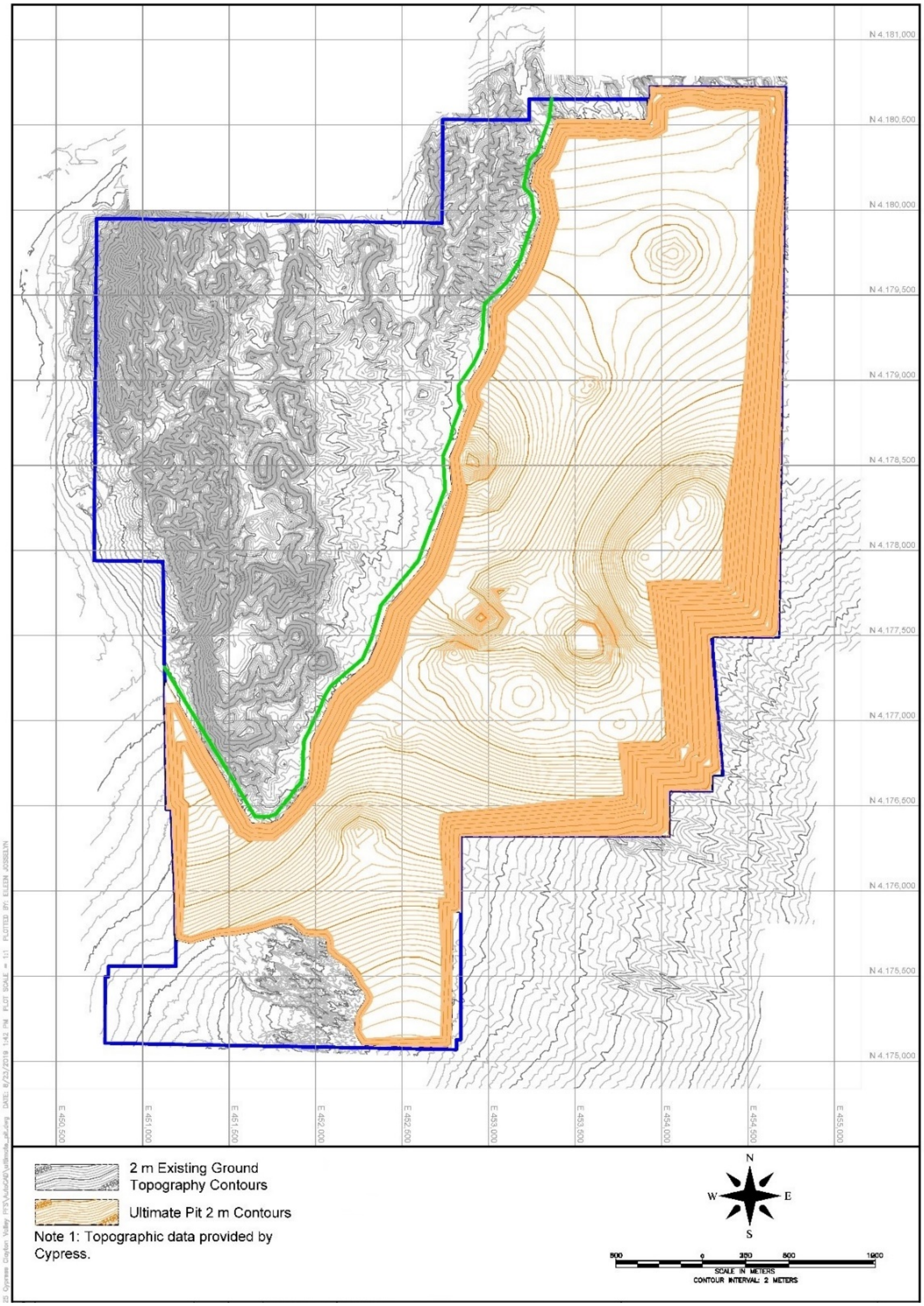


Table 14-3: Pit Constrained Mineral Resource Estimate

Domain	Tonnes Above Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)	Tonnes Above Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)	Tonnes Above Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)
	300 ppm Cutoff			**600 ppm Cutoff**			**900 ppm Cutoff**		
Measured									
Tuffaceous mudstone	47.9	885	42.4	46.6	894	41.7	19.6	1,062	20.8
Claystone all zones	555.4	1,020	566.5	555.3	1,020	566.4	412.0	1,089	448.7
Siltstone	8.6	690	**6.0**	6.3	763	4.8	0.9	974	0.9
Total	**612.0**	**1,005**	**614.83**	**608.2**	**1,008**	**612.9**	**432.4**	**1,088**	**470.4**
Indicated									
Tuffaceous mudstone	29.0	880	25.5	27.6	896	24.7	14.5	1,043	15.1
Claystone all zones	408.3	868	354.4	406.9	869	353.6	146.2	1,031	150.7
Siltstone	3.0	746	2.2	3.0	749	2.2	0.20	963	0.2
Total	**440.3**	**868**	**382.2**	**437.5**	**870**	**380.6**	**160.9**	**1,032**	**166.0**
Measured + Indicated									
Tuffaceous mudstone	76.9	883	67.9	74.2	895	66.4	34.1	1,054	35.9
Claystone all zones	963.7	956	920.9	962.3	956	920.1	558.2	1,074	599.4
Siltstone	11.6	704	8.2	9.3	759	7.0	1.1	972	1.1
Total	**1,052.2**	**947**	**997.0**	**1,045.7**	**950**	**993.5**	**593.3**	**1,073**	**636.4**
Inferred									
Tuffaceous mudstone	1.1	715	0.8	1.1	715	0.8	0.1	933	0.1
Claystone all zones	56.7	709	40.2	55.4	712	39.4	2.2	1,009	2.2
Siltstone	1.6	648	1.1	1.5	655	1.0	0.0	0	0.0
Total	**59.4**	**707**	**42.0**	**57.9**	**711**	**41.2**	**2.3**	**1,005**	**2.3**

Table 14-4: Mineral Resource Estimate Summary

Domain	Tonnes Above Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)
	900 ppm Cutoff		
Measured			
Tuffaceous mudstone	19.6	1,062	20.8
Claystone all zones	412.0	1,089	448.7
Siltstone	0.9	974	0.9
Total	**432.4**	**1,088**	**470.4**
Indicated			
Tuffaceous mudstone	14.5	1,043	15.1
Claystone all zones	146.2	1,031	150.7
Siltstone	0.20	963	0.2
Total	**160.9**	**1,032**	**166.0**
Measured + Indicated			
Tuffaceous mudstone	34.1	1,054	35.9
Claystone all zones	558.2	1,074	599.4
Siltstone	1.1	972	1.1
Total	**593.3**	**1,073**	**636.4**
Inferred			
Tuffaceous mudstone	0.1	933	0.1
Claystone all zones	2.2	1,009	2.2
Siltstone	0.0	0	0.0
Total	**2.3**	**1,005**	**2.3**

1. The effective date of the Mineral Resource Estimate is May 19, 2020. The QP for the estimate is Ms. Terre Lane of Global Resource Engineering Ltd. and is independent of Cypress.
2. The Mineral Resource estimate was prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the with generally accepted Canadian Institute of Mining's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019).
3. 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 the 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.

14.8 Estimate Validation

Of the 593 million tonnes in Measured and Indicated Resources, 73% is in the Measured Category.

The CIM requirement for classification of a Measured Resource is drill holes must be "spaced closely enough to confirm both geological and grade continuity". It is the opinion of the QP that the Measured Resources meet this requirement

Validation of the resource model is supported by the following checks and comparisons.

14.8.1 Model to Drill Hole Validation

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 southeast to northwest. Figure 14-19 and Figure 14-20 present cross sections showing modeled lithology; Figure 14-21 and Figure 14-22 present cross sections with modeled Li grades. Figure 14-18 shows the cross-section locations.

Figure 14-18: Cross Section Locations

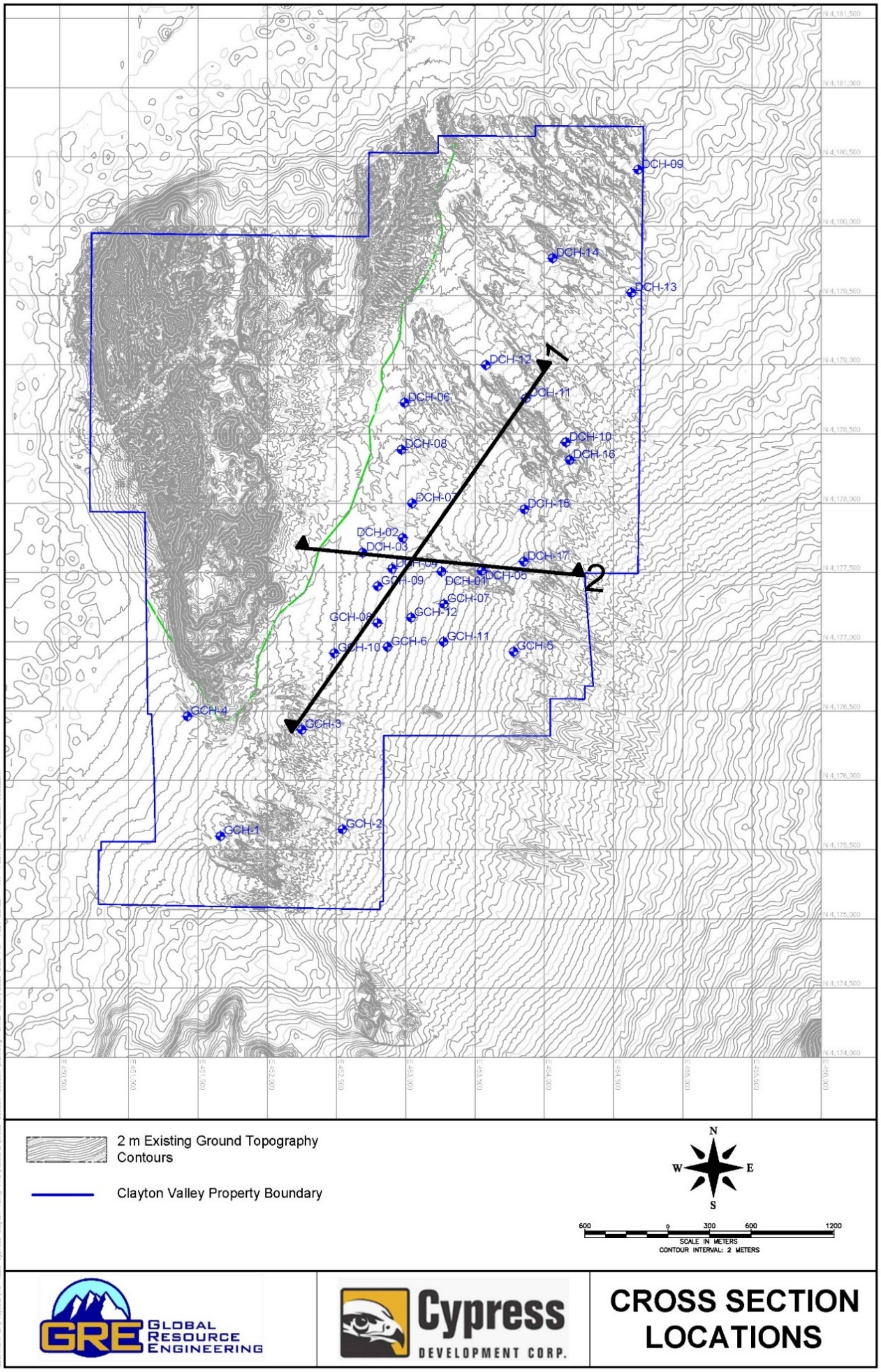


Figure 14-19: Cross Section 1 with Lithology

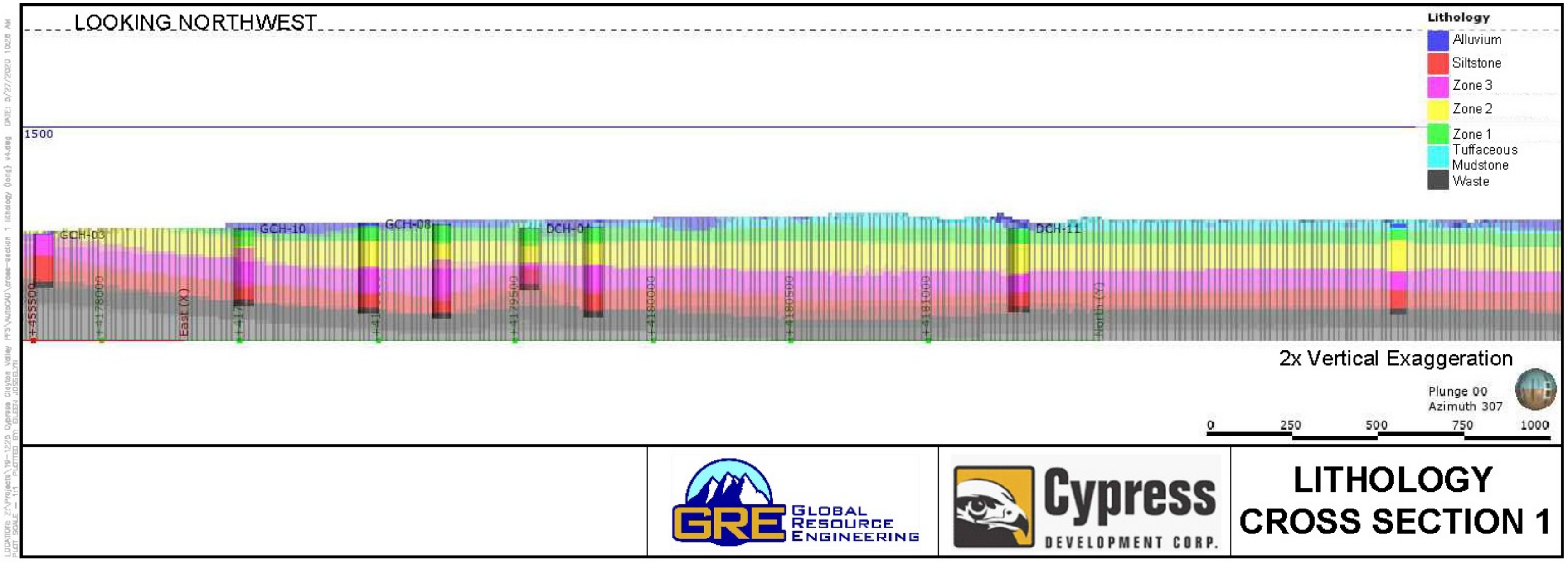


Figure 14-20: Cross Section 2 with Lithology

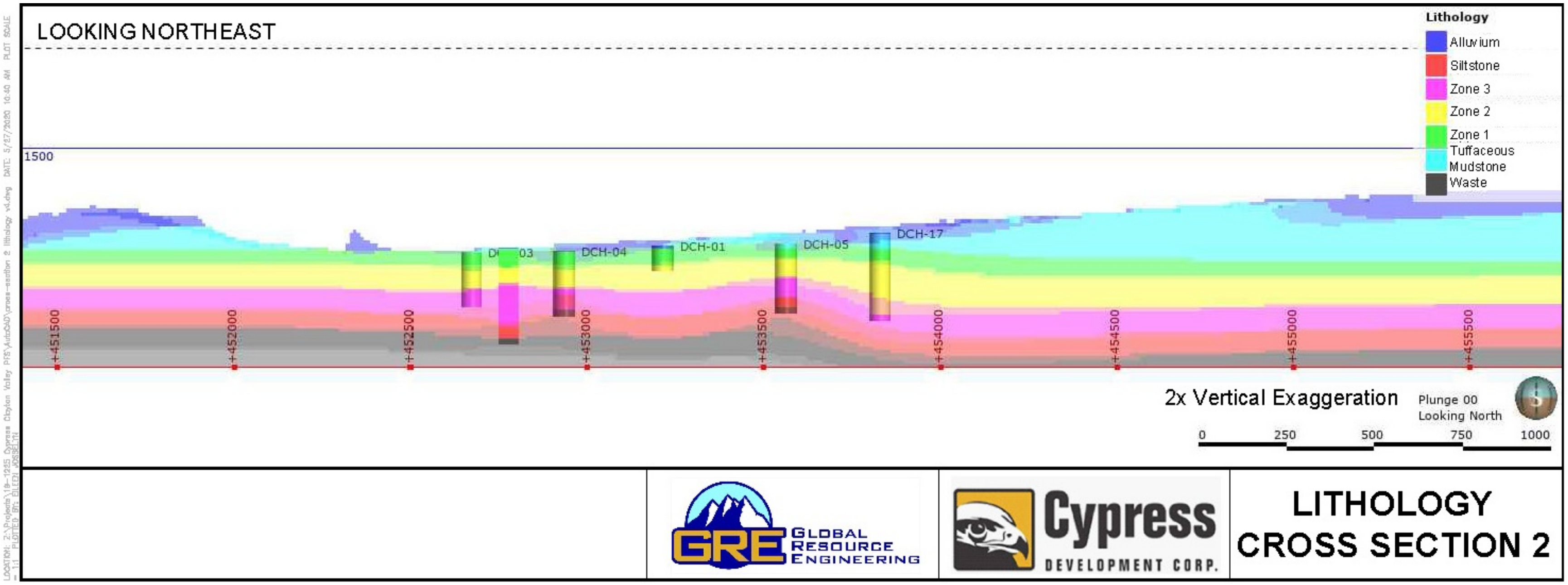


Figure 14-21: Cross Section 1 with Lithium Grade

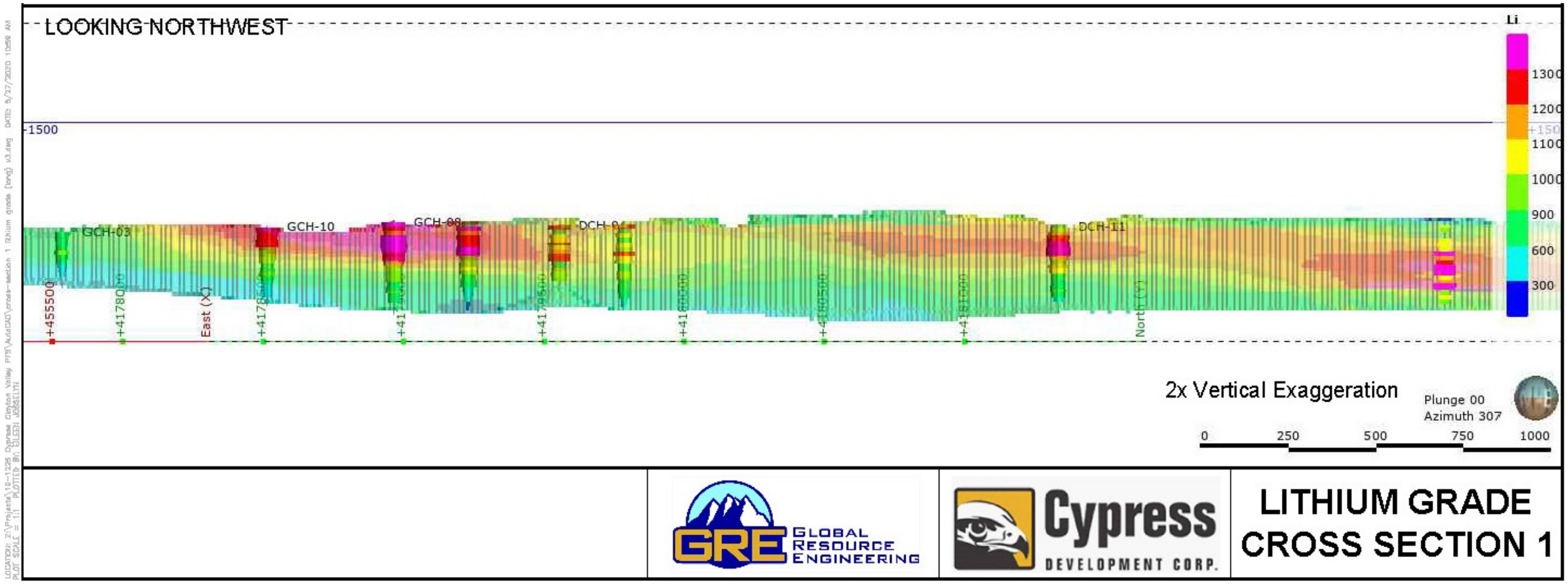
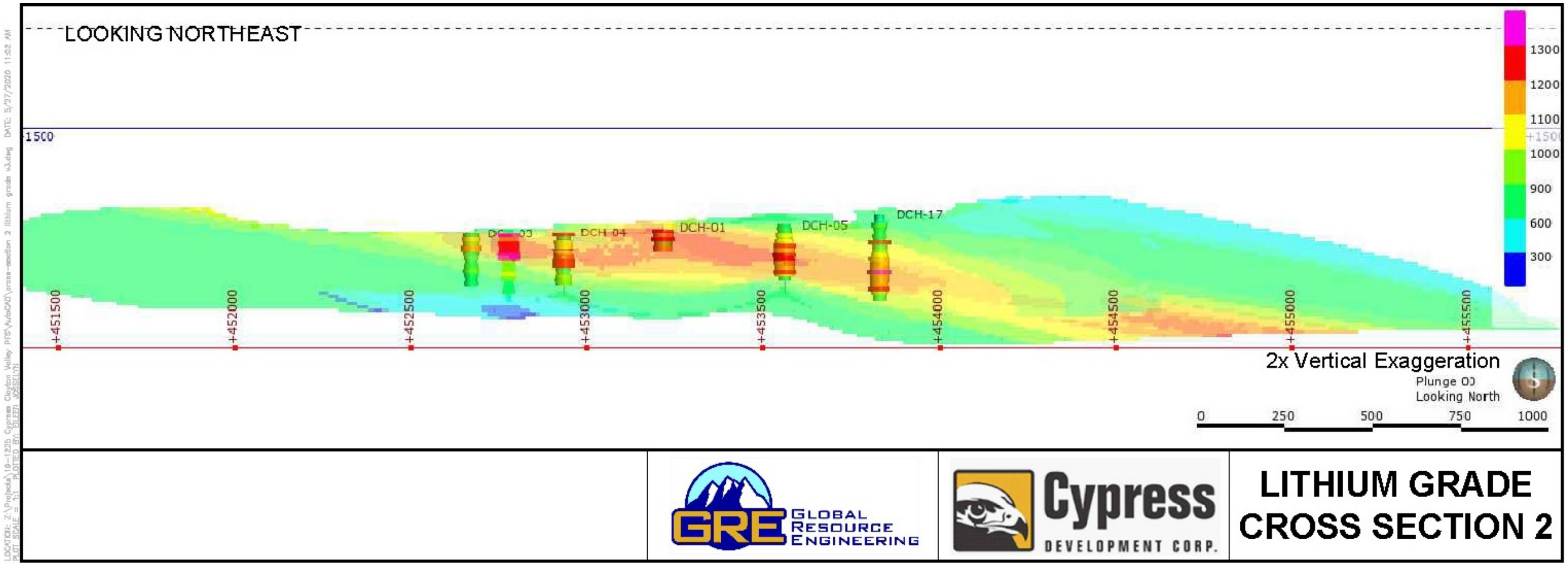


Figure 14-22: Cross Section 2 with Lithium Grade



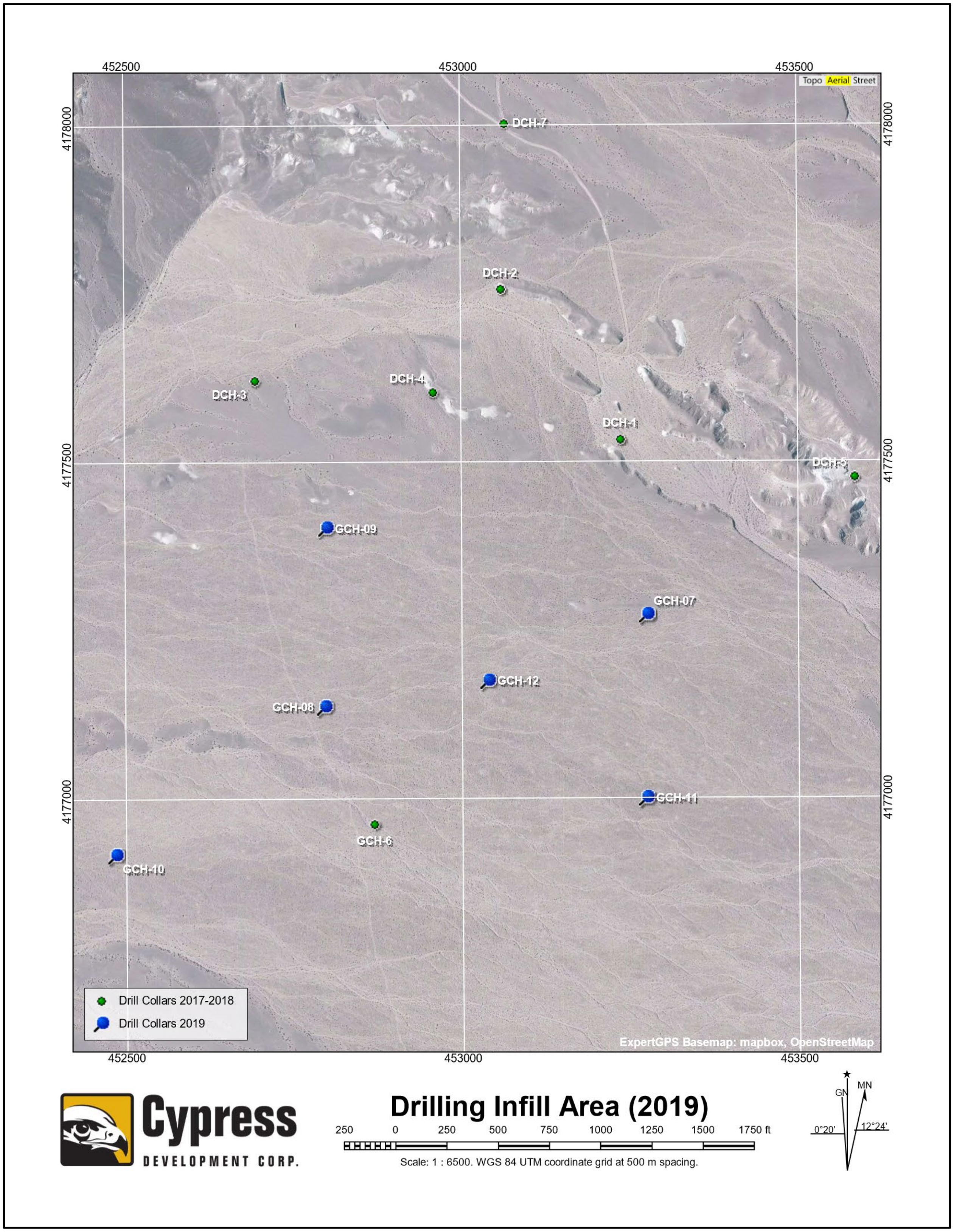
14.8.2 Drill Hole to Drill Hole Comparison

In 2019, Cypress drilled infill holes in the area around and north of GCH-06 (Table 14-5 and Figure 14-23). The distribution and similarity in values support the range and search parameters used in developing the resource model. Spacing in the in-fill program averaged 200 meters in claystone, variogram show range of 1,000 meters in major (NE) axis and 450 meters in minor (SE, downdip) axis.

Table 14-5: Infill Drill Hole Comparison

Drill Hole ID	Depth (m)		Length (m)	Ave Li (ppm)
	From	To		
GCH-06	3.0	100.0	96.9	1,142
GCH-07	2.7	90.5	87.8	1,188
GCH-08	8.2	87.5	84.7	1,229
GCH-09	8.3	72.2	64.0	1,163
GCH-10	3.0	69.2	66.2	1,069
GCH-11	8.2	72.2	64.0	1,176
GCH-12	1.8	81.4	79.6	1,252

Figure 14-23: CVLP 2019 Infill Drill Hole Locations



14.8.3 Model to Model Comparison

GRE superimposed pit shells from Phase 1 to Phase 8 onto both the PFS Mineral Resource Estimate model and the 2018 PEA (GRE, 2018b) Mineral Resource Estimate model. Table 14-6 shows the comparison phase by phase and total. The comparison shows the changes from infill drilling and refinement of the modeling parameters.

- Conversion of all Inferred blocks to Measured and Indicated.
- Increase in the material above a 900-ppm cutoff grade to 222.8 million tonnes.
- Increase in grade to 1,141 ppm Li.

Table 14-6: Comparison of PFS and PEA Models

Pit Phase	A. PEA Model with PFS Pit Phases Imposed, 900 ppm Li Cutoff				B. PFS Model with PFS Pit Phases, 900 ppm Li Cutoff			
	Measured+ Indicated Tonnes (millions)	Li Grade (ppm)	Inferred Tonnes (millions)	Li Grade (ppm)	Measured+ Indicated Tonnes (millions)	Li Grade (ppm)	Inferred Tonnes (millions)	Li Grade (ppm)
1	19.3	1,161	14.7	1,102	33.9	1,199	-	-
2	12.8	1,197	7.2	1,138	18.4	1,165	-	-
3	23.0	1,063	0.8	1,068	27.0	1,122	-	-
4	9.2	1,048	3.8	1,032	13.9	1,169	-	-
5	25.2	1,103	9.4	1,069	37.8	1,109	-	-
6	26.5	1,134	1.9	1,096	36.9	1,131	-	-
7	9.3	1,125	5.3	1,049	16.0	1,140	-	-
8	4.6	1,116	23.2	1,044	38.9	1,125	-	-
Total	**129.9**	**1,118**	**66.3**	**1,072**	**222.8**	**1,141**	**-**	**-**

14.8.4 Nearby Mineral Resources

Two companies, Noram Ventures and Enertopia, have reported drill results and mineral resources with similar grades. (See Section 23.0). The resources identified are adjacent to the northeast and on-trend with Cypress' resources. While there are differences in modeling methods, the general comparison indicates continuity within the mineralized system.

15.0 MINERAL RESERVE ESTIMATE

Reserves are classified in order of increasing confidence into Probable and Proven categories to be compliant with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore Canadian National Instrument 43-101. CIM mineral reserve definitions are as follows.

15.1 Mineral Reserves

A Mineral Reserve is the economically mineable part of a measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at prefeasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Prefeasibility Study or Feasibility Study.

15.1.1 Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

15.1.2 Proven Mineral Reserve

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

15.1.3 Modifying Factors

Modifying factors may include mining, processing, metallurgical, economic, marketing, legal, environmental, infrastructure, social and governmental factors.

In the opinion of the authors, Modifying Factors apply to the project.

As a source of lithium, sedimentary-hosted clay, claystone or ash-derived deposits are a new class of deposit. As of this report, there are no operations or projects in the world at a large enough scale to say that the extraction of lithium for this class is commercially proven.

Based on these considerations, all Measured and Indicated Mineral Resources within the mine schedule of Section 16.0 are classified as Probable Reserves.

15.1.4 Inferred Mineral Resource

The Mineral Reserves estimates exclude the Inferred Mineral Resource. All Inferred Resources, along with material within the mine plan below the cutoff of 600 ppm Li, are classified in the mine schedule as waste.

15.2 Mineral Resource Statement

The pit-constrained Mineral Resources were used to derive the Mineral Reserves. This was accomplished by building a mine production schedule from an optimized sequence of pit shells which capture the measured and indicated blocks. The pit shells are nested within the ultimate pit-constrained shell. As such, the Mineral Resources in Section 14.0 include the Mineral Reserves.

15.2.1 Cutoff Grades

As described in Section 14.0, a cutoff grade of 900 ppm Li is used to determine mill feed. This value was chosen by GRE based on successive iterations to produce a uniform production schedule with minimal waste. Material between 600 and 900 ppm Li is designated as low grade material to stockpile and is not included in the Mineral Reserve.

15.2.2 Mine Life and Phases

Within the ultimate pit shell, 16 pit phases were constructed which contain a total of 281 million tonnes of Measured and Indicated Resources. The phases begin in the southwest and expand northeast, where mining is deeper and encounters increasing amounts of low- grade ore and waste. For the mine production schedule and economic analysis, only the first eight phases are used, which result in a mine life of approximately 40 years.

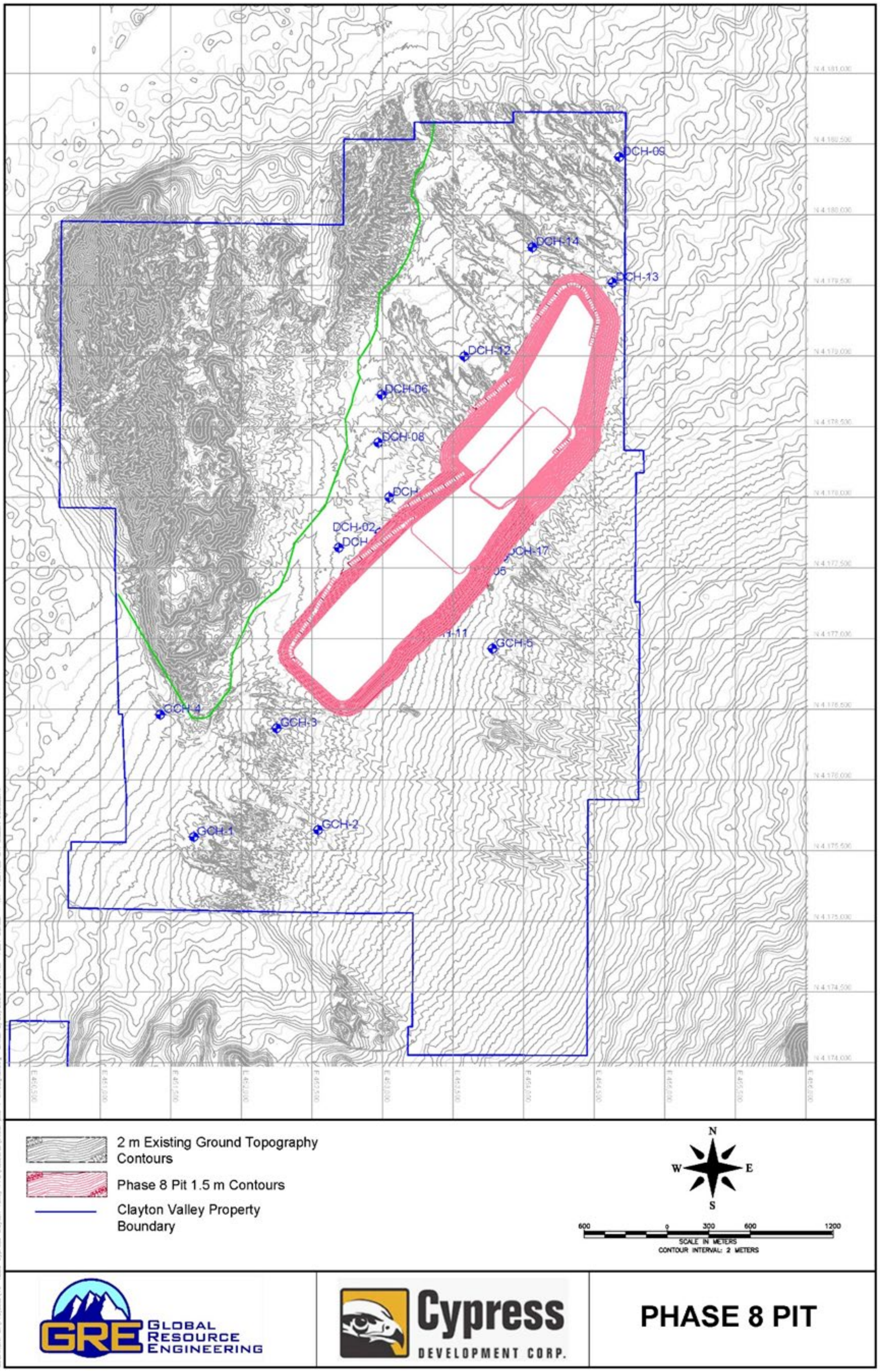
The cumulative result for all eight phases forms the Mineral Reserves in Table 15-1 and shown in Figure 15-1. All Measured and Indicated Mineral Resources in this pit were converted to Probable Mineral Reserves as defined by NI 43-101. Inferred Mineral Resources are not part of the Mineral Reserve statement.

Table 15-1: Mineral Reserve Estimate

Domain	Tonnes Above Cutoff (millions)	Li Grade (ppm)	Li Contained (million kg)
Probable Reserve			
Total	**222.8**	**1,141**	**254.3**

1. The effective date of the Mineral Reserve Estimate is May 19, 2020. The QP for the estimate is Ms. Terre Lane of Global Resource Engineering Ltd. and is independent of Cypress.
2. The Mineral Reserve estimate was prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the with generally accepted Canadian Institute of Mining's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019).
3. Mineral Reserves are reported within the pit design at a mining cutoff of 900 ppm.
4. The cutoff of 900 ppm is an optimized cutoff selected for the mine production schedule.
5. The Mineral Reserves are derived from and not separate from the Mineral Resources.
6. Mineral Reserves are estimated based on delivery to the mill stockpile.
7. No Inferred Resources are included in the Mineral Reserves or given value in the economic analysis

Figure 15-1: Plan View CVLP Pit Phase 8



The Probable Mineral Reserve contains 254.3 million kg of Li, or 1.353 million tonnes LCE.

The eight pit phases which make up the mine production schedule in Section 16.0 include 26.6 million tonnes of low grade, averaging 732 ppm Li, and 21.6 million tonnes classified as waste.

To the best of the QP's knowledge, there are no known legal, political, environmental, permitting, title, taxation, socio-economic, marketing, mining, metallurgical, or other factors that would further materially affect the Mineral Reserves reported herein.

16.0 MINING METHODS

All materials within the project's resource area are relatively flat lying soft sediments. The deposit is covered by a thin veneer of alluvial gravels and extends to a depth of 100 to 140 meters.

Mining will be carried out using conventional surface methods. Excavation will use a single Caterpillar 6020B or equivalent shovel (hydraulic excavator configuration) with a 12 m³ bucket capacity. The material is very soft, so drilling and blasting will not be required.

Truck haulage was studied as an alternative. Conveyor transport to the mill is preferred due to reduced traffic and water for dust control. Conveyors should also result in lower operating cost due to the consumption of electric power instead of diesel.

Material at the mining face will be fed directly to a mobile feeder-breaker and then moved out of the pit using a series of jump conveyors. The material will then be transferred to over-land conveyors and transported to a radial stacker and run-of-mine (ROM) stockpile located at the processing plant.

16.1 Pit Geotechnical Analysis

Sampling and physical testing of in situ soils from drill holes in the pit limit were performed to supplement the pit stability analysis needed to determine the appropriate slope angles for pit design.

16.1.1 Pit Geotechnical Sampling & Testing

A total of 13 samples were collected at various depths from drill holes GCH-10, GCH-11, and GCH-12 for laboratory testing. The tests were completed (April 2019) by Advanced Terra Testing following the technical standards of the American Society for Testing and Materials (ASTM).

The laboratory tests included:

- Atterberg Limits (ASTM D4318)
- Shrinkage Limits (ASTM D4943)
- Specific Gravity (ASTM D854 – Method 8)
- Grain Size Analysis with Hydrometer (ASTM D6913, D7928)
- One-Dimensional Consolidation (ASTM D2435)
- Direct Shear (ASTM D3080)
- Consolidated Undrained Staged Triaxial Compression (ASTM D4767)

Table 16-1 and Table 16-2 show the samples collected and tests performed, respectively.

Table 16-1: Collected Pit Geotechnical Samples

Sample ID	Source Drill Hole	Depth (m)	
		From	To
512012	GCH-12	4.0	4.2
512013		20.1	20.3
512014		32.1	32.3
512015		51.6	51.8
512016		68.0	68.2
512018		105.1	105.3
512020	GCH-10	20.0	20.2
512022	GCH-11	11.0	11.2
512023		23.9	24.3
512024		44.6	44.8
512025		61.6	61.8
512026		87.6	87.8
512027		120.8	121.0

Table 16-2: Pit Geotechnical Samples Testing Completed

Testing	Sample(s)
ASTM D4318	Composite (512014, 512015, 512016); 512020; 512026
ASTM D4943	Composite (512014, 512015, 512016); 512027 (x2); 512020 (x2)
ASTM D854 – Method 8	Composite (512014, 512015, 512016)
ASTM D6913, D7928	Composite (512014, 512015, 512016); 512020; 512026
ASTM D2435	512012; 512016; 512018; 512023; 512025; 512026
ASTM D3080	Composite (512014, 512015, 512016); 512022
ASTM D4767	512012; 512014; 51218

16.1.2 Materials Classifications

Testing revealed the tuffaceous mudstone and the three claystone zones have the USCS Classification of Fat Clay, and the siltstone has the USCS Classification Silty Sand (Table 16-3). The resulting particle size distributions are displayed in Figure 16-1 through Figure 16-3.

Table 16-3: Material Characteristics of Lithologies

Unit	Source	USCS	PL	LL	PI
Tuffaceous Mudstone	GCH-10	Fat Clay	20	73	53
Claystone zones 1-3	GCH-12	Fat Clay	23	73	50
Siltstone	GCH-11	Silty Sand	0	0	Not Plastic

Figure 16-1: Particle Size Distribution—Tuffaceous Mudstone

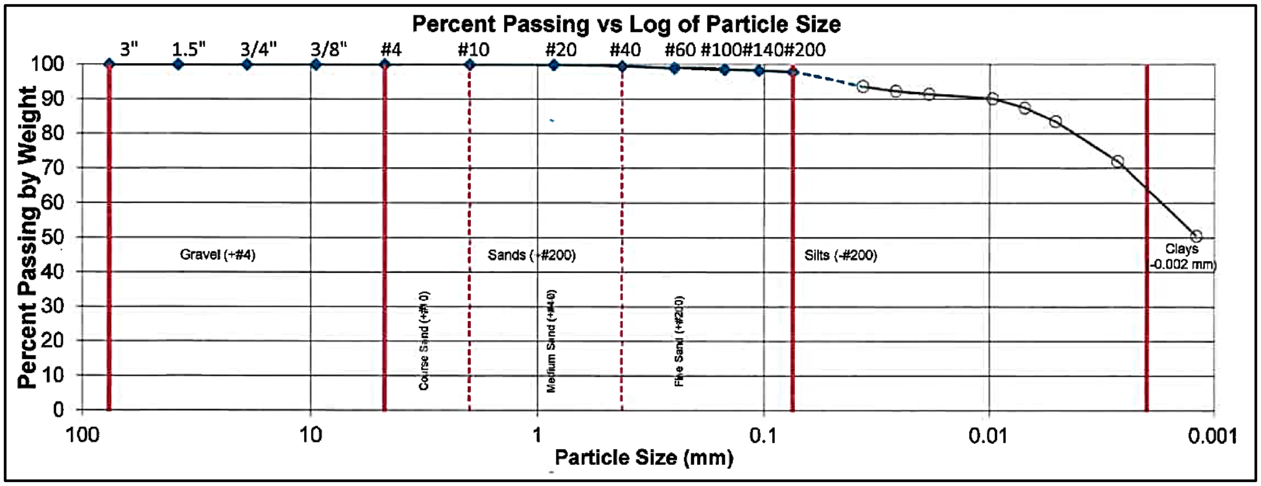


Figure 16-2: Particle Size Distribution—Claystone Zones 1-3

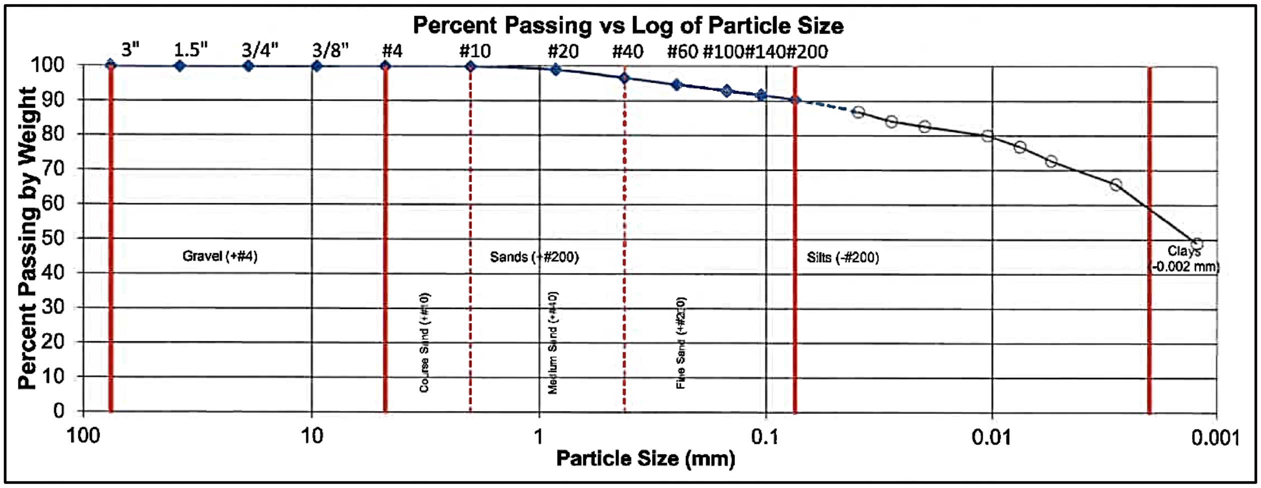
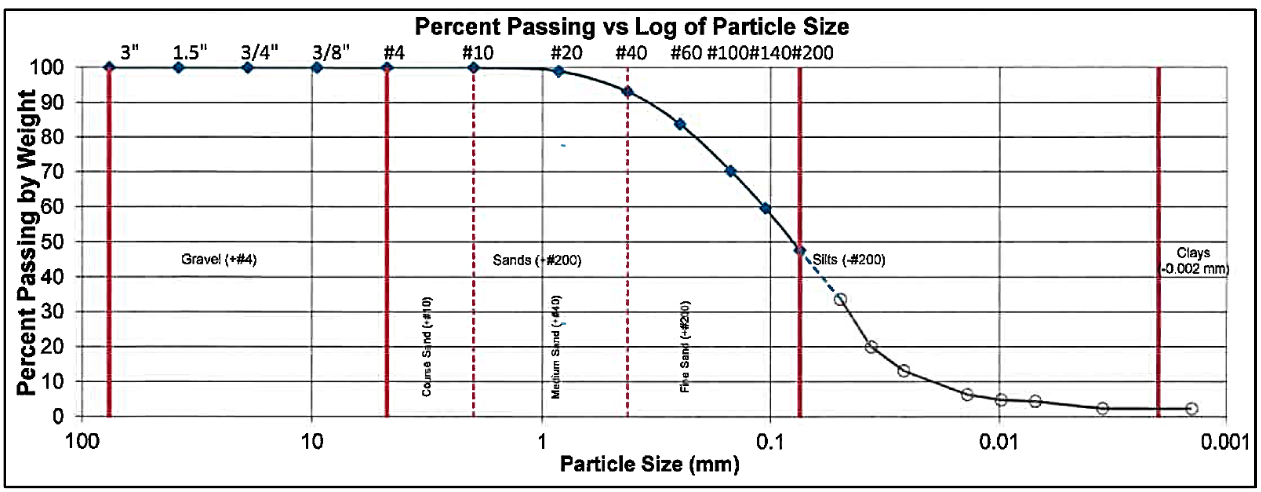
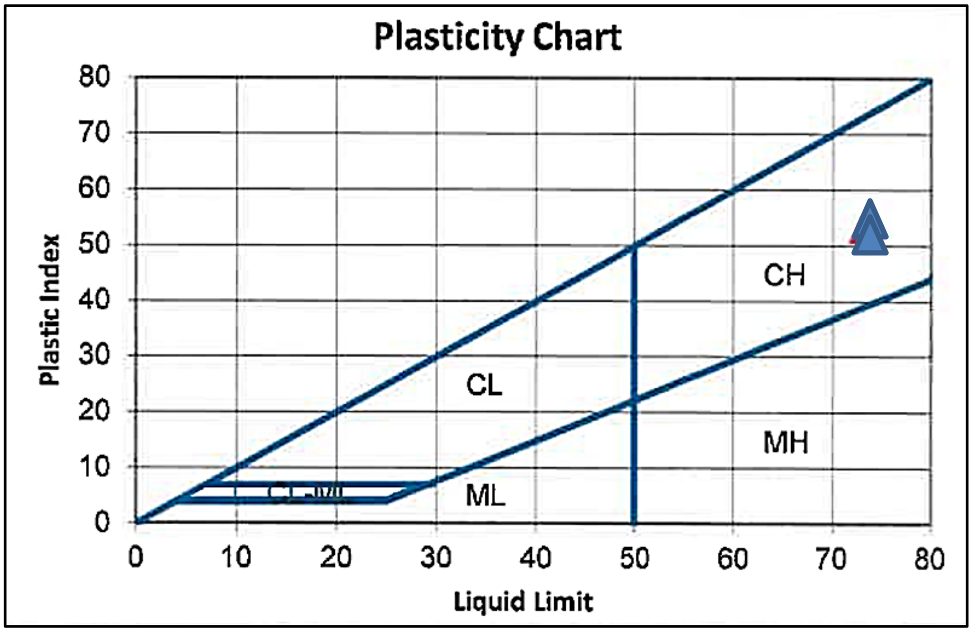


Figure 16-3: Particle Size Distribution—Siltstone



Of important note to both mining and processing, the tuffaceous mudstone and the claystone units have similar high capacities to retain water, as reflected in their Plastic Limit (PL), Liquid Limit (LL), and Plasticity Index (PI). Plastic limit is the percent moisture content by weight where a soil begins to behave as a plastic, liquid limit is the upper moisture content where it becomes fluid, and the plasticity index is the difference in percent between the two. The mudstone and claystone units plot similarly on a Plasticity Chart, with the Atterberg Classification CH, or high clay (Figure 16-4).

Figure 16-4: Plasticity Chart



Plot showing Atterberg Classification for tuffaceous mudstone and the claystone zones.

16.1.3 Pit Slope Stability Analysis

A schematic two-dimensional cross-section of the pit was analyzed using the SLOPE/W software by GeoStudio, version 2019. The cross-section was laterally divided into the three claystone zones with pit slopes that incorporate 6-m width benches and 7.5-m bench heights (the tuffaceous mudstone was not considered in the stability cross-section). The overall pit slope for each lithology was varied by varying the bench face angle (BFA) until a static Factor of Safety (FOS) of 1.3 was attained.

The shear strengths of the claystone were modeled using the Mohr-Coulomb constitutive model which defines the shear strength of the soil in terms of the normal stress, cohesion, and internal friction (phi) of the material. The cohesion and internal friction were determined from the direct shear and triaxial tests. The claystone unit weights were averaged from the laboratory tests for each zone. Groundwater pore-pressure was not applied in the stability analysis because no groundwater was encountered during drilling and is assumed to be below the pit limit. Figure 16-5 shows the general analyzed pit cross-section, and Table 16-4 shows the analyzed material strength properties.

Figure 16-5: General Pit Stability Cross Section

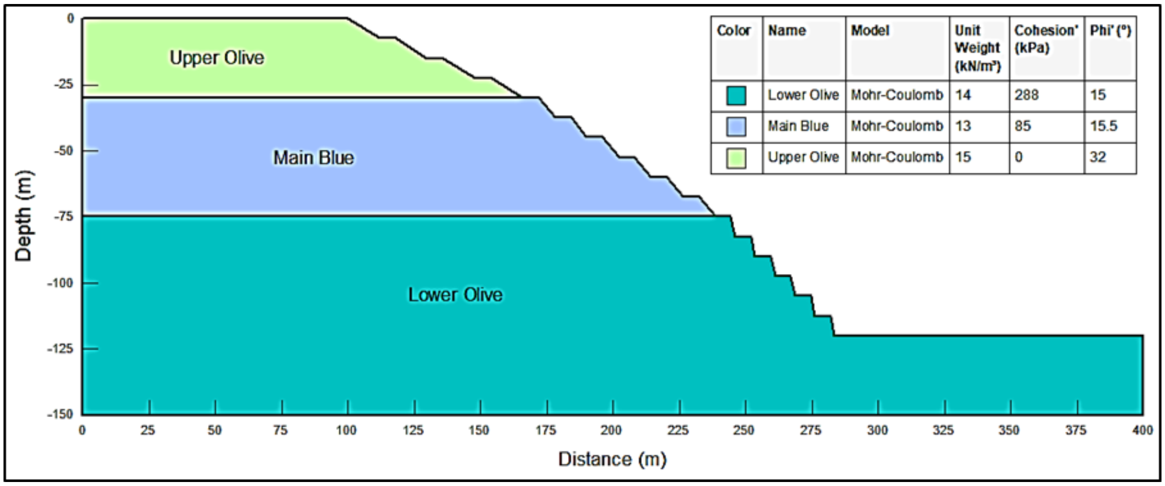


Table 16-4: Pit Stability Material Strength Properties

Material	Unit Weight (kN/m3)	Cohesion (kPa)	Internal Friction Angle(degree)
Claystone zone 1 (UO)	14	288	15
Claystone zone 2 (MB)	13	85	15.5
Claystone zone 3 (LO)	15	0	32

The pit stability analysis resulted in the following pit slope for each claystone zone:

- Claystone zone 1: overall pit slope of 23 degrees and BFA of 32 degrees
- Claystone zone 2: overall pit slope of 32 degrees and BFA of 51 degrees
- Claystone zone 3: overall pit slope of 43 degrees and BFA of 85 degrees

16.2 Mine Plan

16.2.1 Pit Design

An ultimate pit shell was used to limit the mine plan and was generated using the variable pit slope angles above (Section 16.1.2).

The bench height and width were set at 7.5 meters and 6 meters, respectively, based on operating equipment reach and minimum road width.

Within the ultimate pit shell, 16 pit phases were generated. The first eight of these were used to design a production schedule with uniform mill feed and minimal waste. At the design nominal production rate of 15,000 tpd, the mine life represented by these eight phases is about 40 years, and yields the Mineral Reserves described in Section 14.8.2.

The eight phases used the variable slope angles by material type and were designed with maximum road grades in pit of 8%. The eight phases are illustrated in Figure 16-6 through Figure 16-7.

Figure 16-6: CVLP Phase 1 Pit Phases 1 through 4

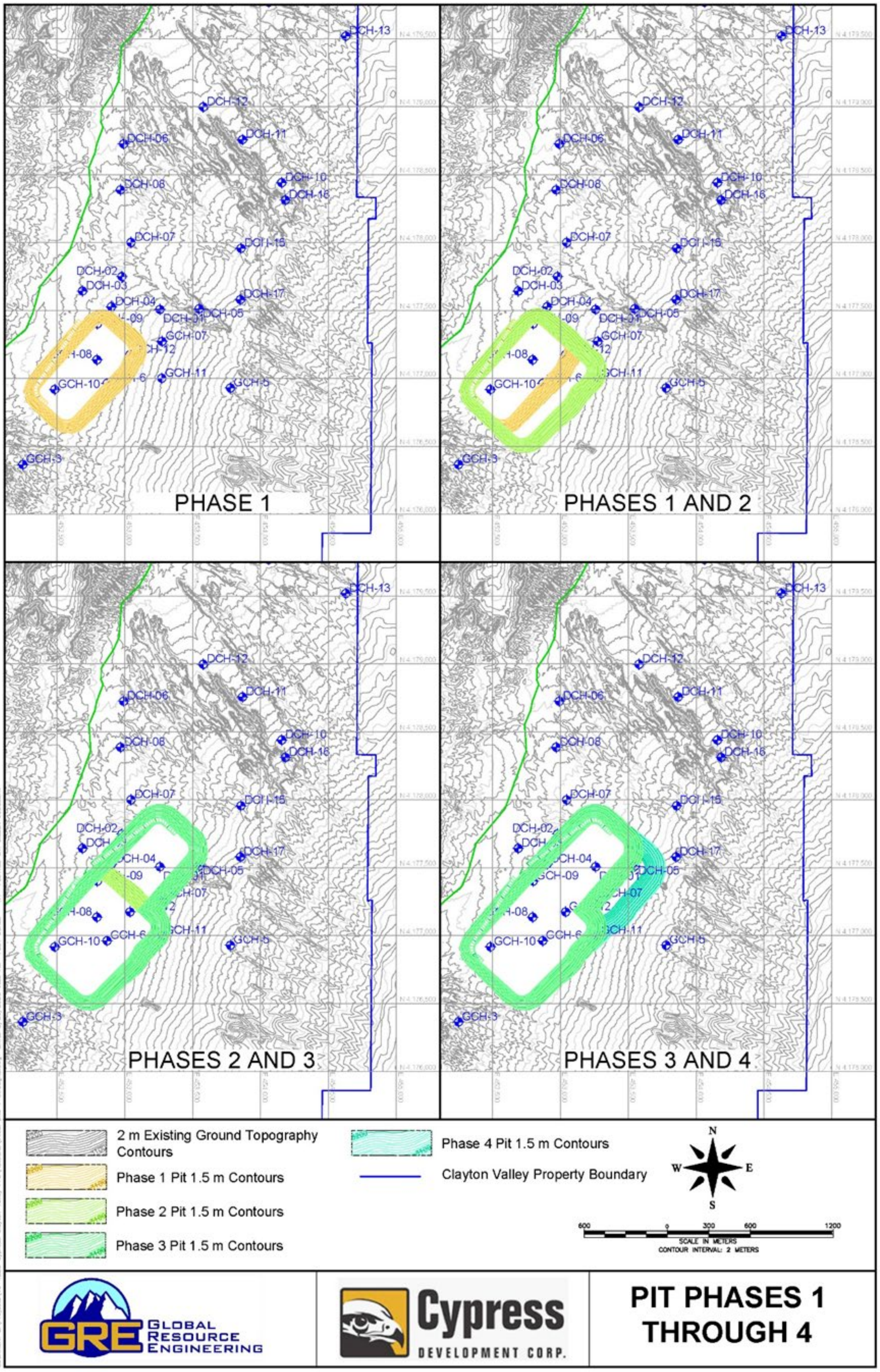
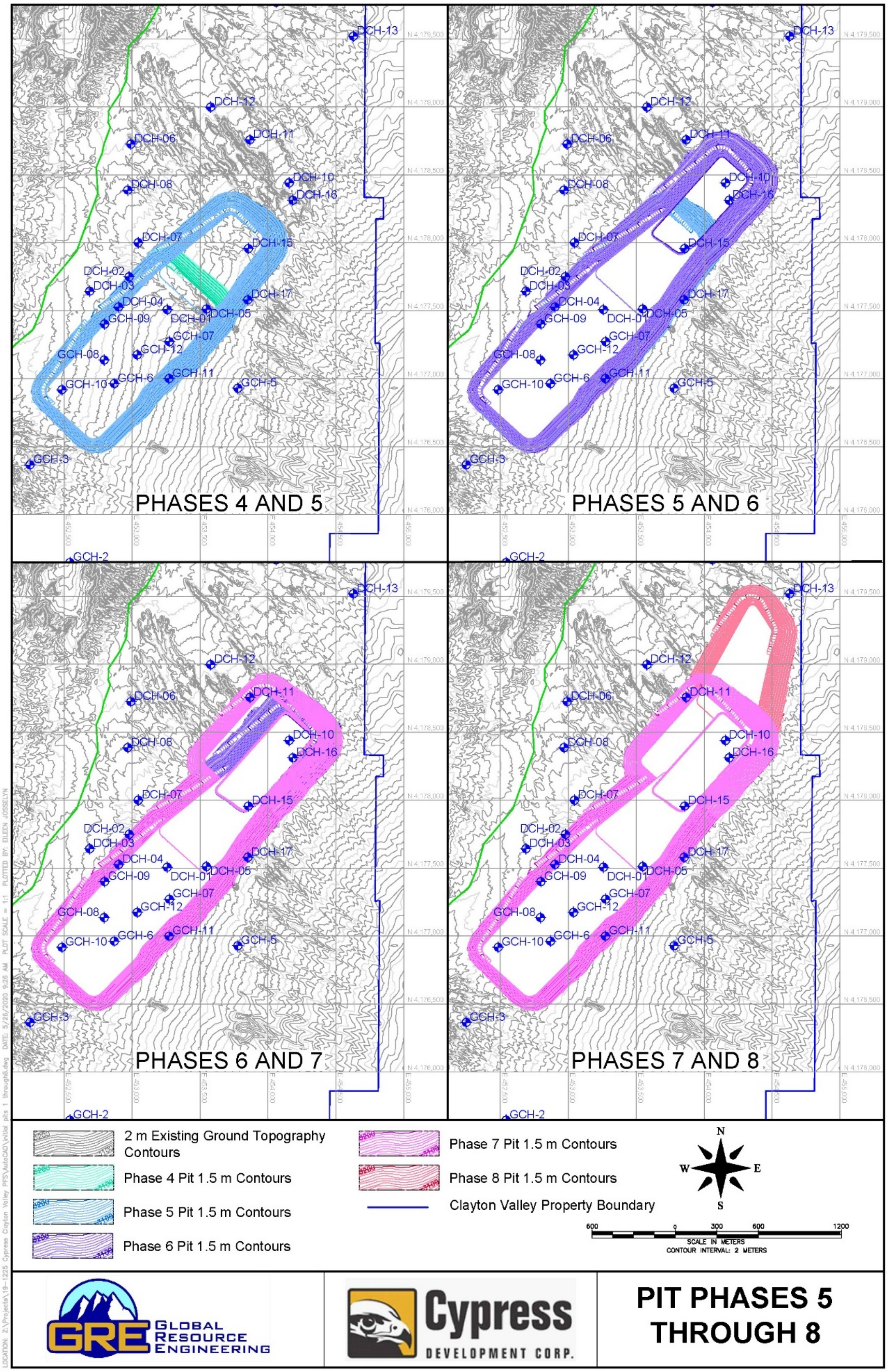


Figure 16-7: CVLP Phase 2 Pit Phases 5 through 8



16.2.2 Pit Production

Within each phase, overburden and waste material will be removed using CAT 657G or equivalent scrapers with a waste removal rate of 166 tonnes/hour. Ore mining will be done with a hydraulic shovel with a bucket capacity of 12 cubic meters and a production rate of 1,265 tonnes/hour. The shovel will dig and feed material to a mobile feeder breaker (see Photo 16-1 and Photo 16-2). Material from the feeder breaker will be transferred to a series of portable jump conveyors, to move the material out of the pit. Finally, the material will be transferred to over-land conveyors and directed to the processing plant or a stockpile as appropriate.

Photo 16-1: Example of a Feeder Breaker

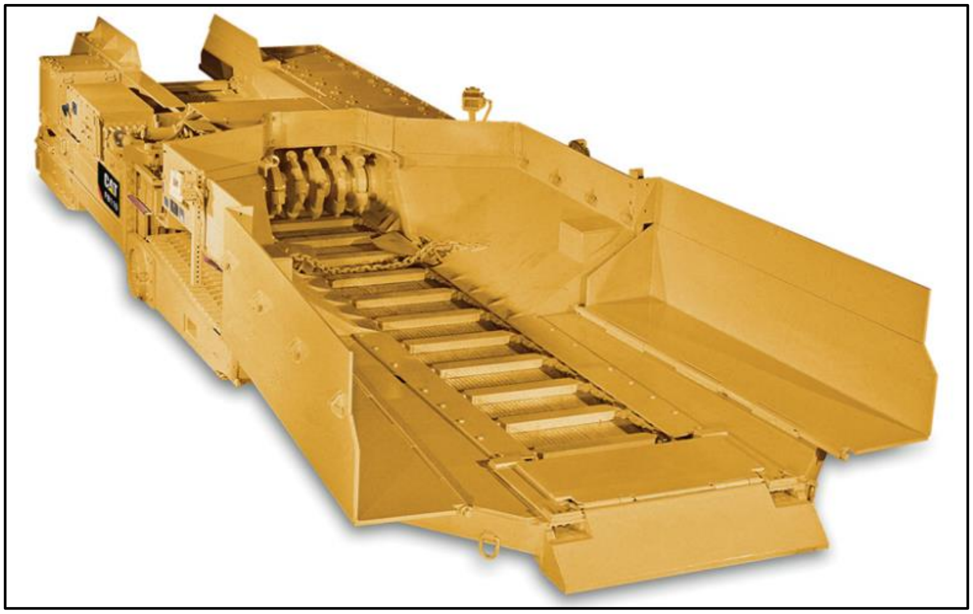


Photo 16-2: Example of a Loader Loading a Track Mounted Feeder Breaker

Each pit phase bench will be subdivided into 10-meter wide sections longitudinally. The feeder-breaker will initially be set up at one end of the first 10-meter-wide section on the side of the pit phase closest to the plant. Portable jump conveyors, each approximately 30.5 meters long, will be positioned from each end of the pit phase along the outside edge of the 10-meter wide section, converging at the mid-line of the pit phase to transport excavated ore from the working area to the mid-line of the pit phase (see Figure 16-8 through Figure 16-11).

Figure 16-8: Mining Method Schematic Plan

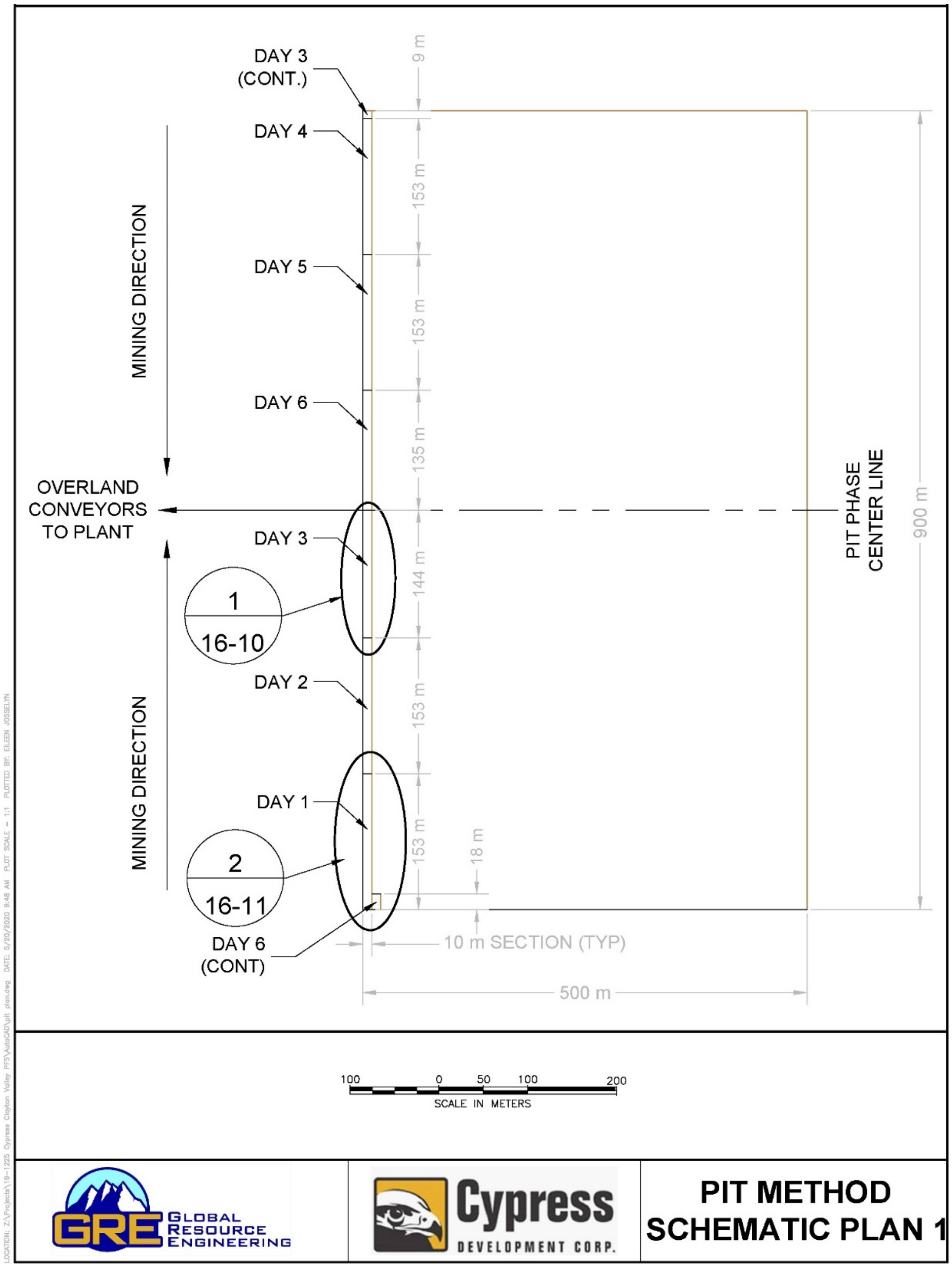


Figure 16-9: Mining Method Schematic Plan Detail Day 2

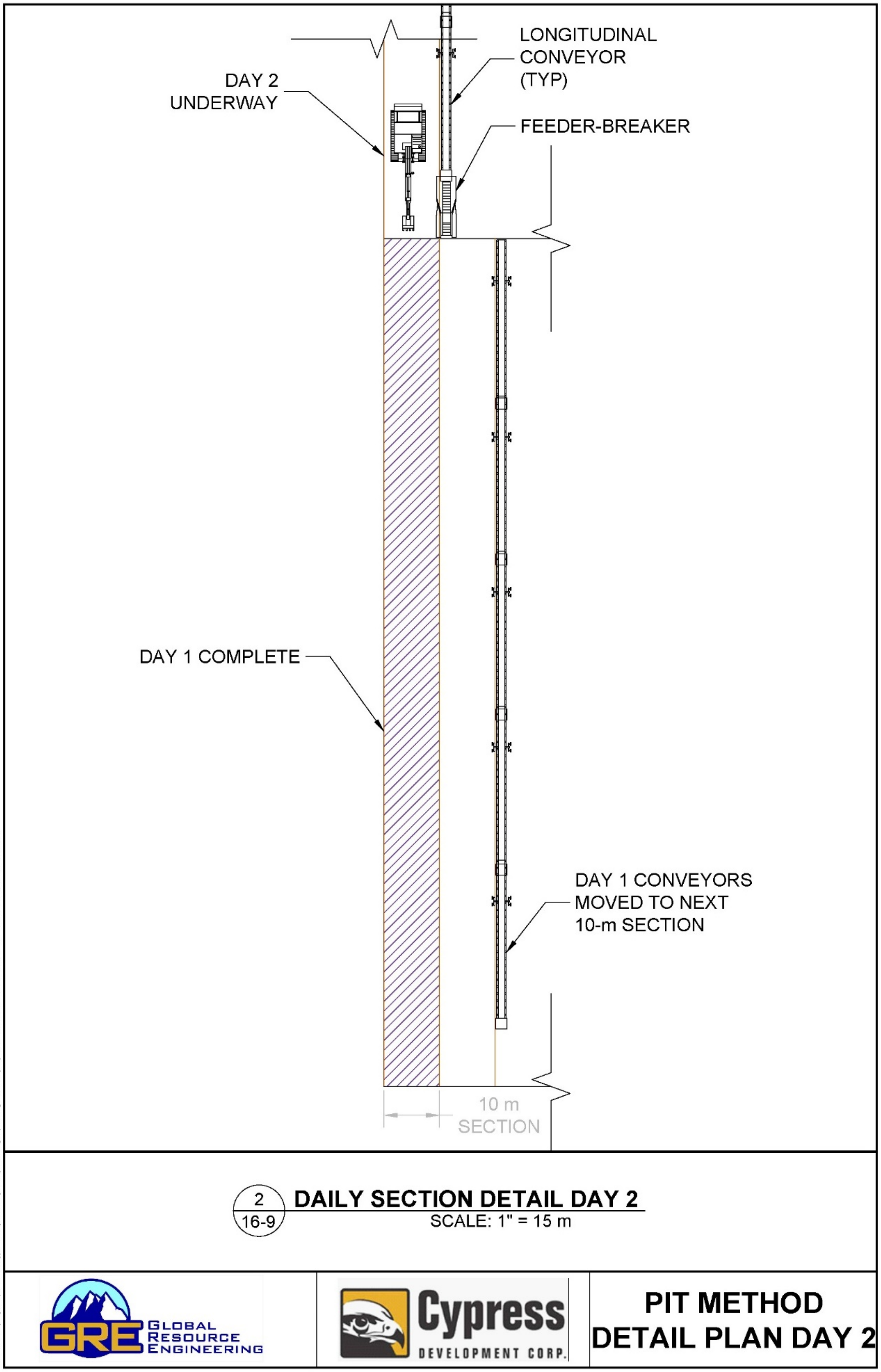
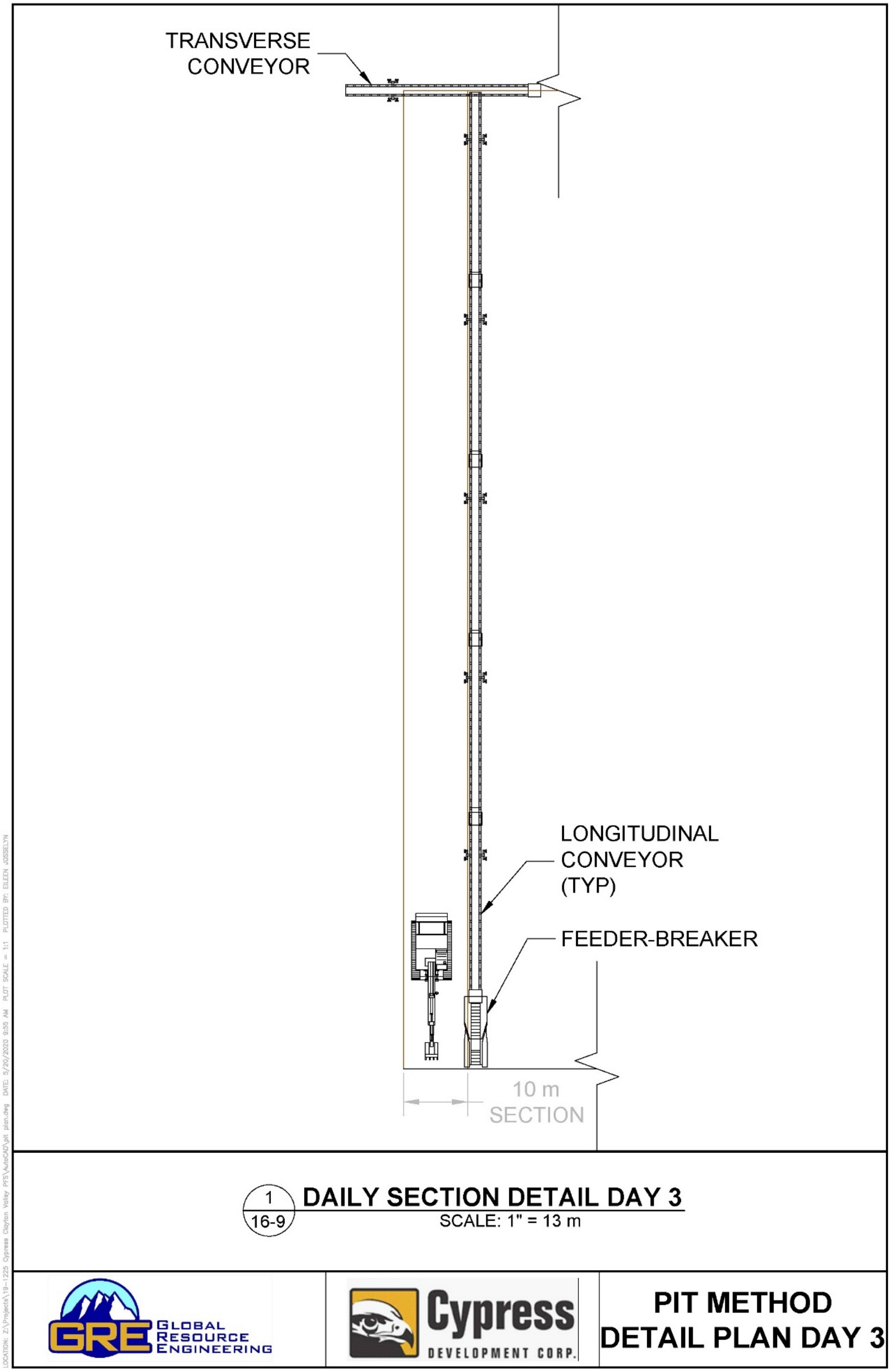


Figure 16-10: Mining Method Schematic Plan Detail Day 3

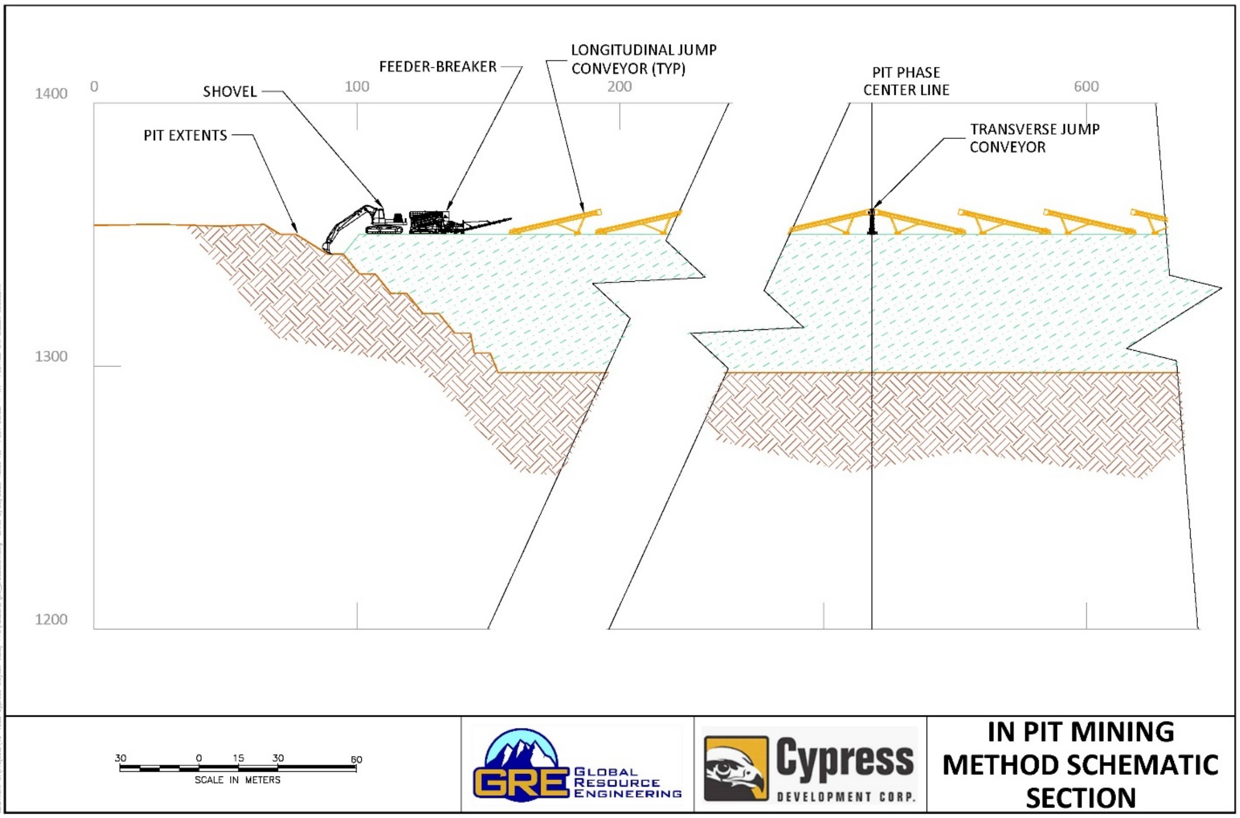


Additional jump conveyors will be positioned transverse to the longitudinal conveyors along the mid-line of the pit phase to transport excavated ore out of the pit. Excavation will proceed from the distant end of the pit phase bench toward the pit phase mid-line. To achieve the production, 153 linear meters of 10-meter wide 7.5-meter-deep section will be excavated daily.

As the excavation proceeds, in-pit longitudinal conveyors along the previous days' excavation will be moved to the next 10-meter wide section.

This mining method has a low operating cost and requires the least amount of support equipment. There is very little traffic on the haul roads, which reduces road maintenance requirements, water usage, and related costs.

Figure 16-11: Mining Method Schematic Profile



16.3 Mine Production Schedule

The distribution of material is shown by pit phase in Table 16-5 and by bench and bench in Table 16-6. Mining will progress from southwest to northeast to defer low grade and waste handling until later in the mine life.

Table 16-5: Production by Pit Phase

Phase	Ore Tonnes (millions)	Low Grade Ore Tonnes (millions)	Waste Tonnes (millions)	Li Grade (ppm)	Low Grade Li Grade (ppm)	Stripping Ratio
1	33.93	0.41	0.79	1,199	872	0.02
2	20.04	0.00	1.18	1,148	880	0.06
3	28.62	0.59	2.92	1,128	856	0.10
4	14.31	1.62	1.83	1,165	822	0.11
5	36.47	4.55	7.07	1,109	805	0.17
6	33.50	3.80	9.27	1,144	714	0.25
7	16.19	0.81	2.42	1,136	817	0.14
8	38.40	0.64	10.59	1,126	876	0.27
Total	**221.46**	**12.42**	**36.07**	**1,143**	**788**	**0.15**

Table 16-6: Pit Resource by Phase and Bench

Bench	Ore Tonnes (millions)	Low Grade Ore Tonnes (millions)	Waste Tonnes (millions)	Li Contained (millions kg)	Li Grade (ppm)	Low Grade Li Contained (millions kg)	Low Grade Li Grade (ppm)	Stripping Ratio	Pre-Stripping
Phase 1									
1387.5	-	-	-	-		-			
1380	-	-	-	-		-			
1372.5	2	5	92	2	915	4	736	13.72	Pre-Strip
1365	22,943	5	118,530	24,814	1,082	3	702	5.17	
1357.5	1,423,600	25,414	542,031	1,620,826	1,139	21,831	859	0.37	
1350	4,338,886	138,988	129,925	5,023,238	1,158	119,636	861	0.03	
1342.5	5,330,815	22	41	6,703,413	1,257	17	763	0.00	
1335	4,793,117	18	14	6,033,718	1,259	13	707	0.00	
1327.5	4,362,243	30	25	5,471,430	1,254	20	669	0.00	
1320	3,985,771	2	11	4,924,352	1,235	1	665	0.00	
1312.5	3,620,458	2	4	4,269,194	1,179	2	861	0.00	
1305	3,286,054	6,418	1	3,682,666	1,121	5,772	899	0.00	
1297.5	2,768,807	240,919	(0)	2,918,757	1,054	211,917	880	0.00	
1290	-	-	-	-		-			
1282.5	-	-	-	-		-			
1275	-	-	-	-		-			
Phase 2									
1387.5	-	-	-	-		-			
1380	-	-	55	-		-			Pre-Strip
1372.5	1	7	111,615	1	916	5	751	14236.14	Pre-Strip
1365	374,222	4	735,930	361,788	967	3	701	1.97	
1357.5	2,019,841	8	303,634	2,139,527	1,059	5	715	0.15	
1350	2,454,609	13	10,219	2,811,444	1,145	11	779	0.00	
1342.5	2,389,375	22	282	2,854,859	1,195	17	760	0.00	
1335	2,302,357	19	1,013	2,744,455	1,192	13	710	0.00	

Bench	Ore Tonnes (millions)	Low Grade Ore Tonnes (millions)	Waste Tonnes (millions)	Li Contained (millions kg)	Li Grade (ppm)	Low Grade Li Contained (millions kg)	Low Grade Li Grade (ppm)	Stripping Ratio	Pre-Stripping
1327.5	2,224,436	29	3,624	2,628,078	1,181	19	672	0.00	
1320	2,158,292	2	3,543	2,534,323	1,174	1	658	0.00	
1312.5	2,092,122	2	3,431	2,427,872	1,160	2	858	0.00	
1305	2,036,988	143	3,597	2,318,442	1,138	129	899	0.00	
1297.5	1,991,654	3,747	1,902	2,181,354	1,095	3,311	884	0.00	
1290	-	-	-	-		-			
1282.5	-	-	-	-		-			
1275	-	-	-	-		-			
Phase 3									
1387.5	-	-	-	-		-			
1380	-	-	-	-		-			
1372.5	-	-	179,347	-		-			Pre-Strip
1365	297,769	211,082	856,993	293,233	985	180,014	853	1.68	
1357.5	2,116,507	116,517	1,529,275	2,302,659	1,088	101,186	868	0.68	
1350	3,764,311	-	357,568	4,308,511	1,145	-		0.09	
1342.5	3,703,634	176,635	(0)	4,187,745	1,131	149,682	847	0.00	
1335	3,575,389	83,674	-	4,040,273	1,130	72,115	862	0.00	
1327.5	3,433,072	-	-	3,900,192	1,136	-		0.00	
1320	3,225,080	-	-	3,687,405	1,143	-		0.00	
1312.5	3,015,938	-	-	3,446,478	1,143	-		0.00	
1305	2,827,151	-	-	3,198,048	1,131	-		0.00	
1297.5	2,662,428	-	-	2,919,654	1,097	-		0.00	
1290	-	-	-	-		-			
1282.5	-	-	-	-		-			
1275	-	-	-	-		-			
Phase 4									
1387.5	-	-	-	-		-			
1380	-	-	145,540	-		-			Pre-Strip
1372.5	7,225	198,373	1,055,896	6,568	909	163,662	825	5.14	Pre-Strip
1365	233,831	1,040,314	427,078	216,761	927	859,407	826	0.34	
1357.5	1,347,294	346,269	16,750	1,437,750	1,067	277,924	803	0.01	
1350	1,639,085	30,175	28,194	1,773,030	1,082	26,460	877	0.02	
1342.5	1,645,373	-	42,007	1,863,999	1,133	-		0.03	
1335	1,608,650	-	42,062	1,910,083	1,187	-		0.03	
1327.5	1,577,643	-	26,637	1,938,309	1,229	-		0.02	
1320	1,565,299	-	26,001	1,942,006	1,241	-		0.02	
1312.5	1,565,983	-	13,517	1,927,579	1,231	-		0.01	
1305	1,565,202	-	1,310	1,876,092	1,199	-		0.00	
1297.5	1,558,107	-	198	1,785,876	1,146	-		0.00	
1290	-	-	-	-		-			
1282.5	-	-	-	-		-			
1275	-	-	-	-		-			
Phase 5									
1387.5	-	-	13,214	-		-			Pre-Strip
1380	-	-	760,653	-		-			Pre-Strip
1372.5	22,100	240,079	2,426,106	20,051	907	191,936	799	9.25	Pre-Strip
1365	88,301	1,559,315	2,839,598	80,318	910	1,276,927	819	1.72	Pre-Strip
1357.5	2,095,578	1,917,883	1,032,371	2,034,939	971	1,538,235	802	0.26	
1350	3,961,701	813,324	-	4,011,788	1,013	638,088	785	0.00	
1342.5	4,518,723	17,850	0	4,839,031	1,071	15,689	879	0.00	
1335	4,283,637	-	-	4,733,186	1,105	-		0.00	

Bench	Ore Tonnes (millions)	Low Grade Ore Tonnes (millions)	Waste Tonnes (millions)	Li Contained (millions kg)	Li Grade (ppm)	Low Grade Li Contained (millions kg)	Low Grade Li Grade (ppm)	Stripping Ratio	Pre-Stripping
1327.5	4,024,265	-	-	4,602,527	1,144	-		0.00	
1320	3,820,510	-	-	4,457,781	1,167	-		0.00	
1312.5	3,641,440	-	-	4,269,029	1,172	-		0.00	
1305	3,494,803	-	-	4,069,954	1,165	-		0.00	
1297.5	3,334,811	-	-	3,806,786	1,142	-		0.00	
1290	3,179,359	-	-	3,527,967	1,110	-		0.00	
1282.5	-	-	-	-		-			
1275	-	-	-	-		-			
Phase 6									
1387.5	-	-	8,485	-		-			Pre-Strip
1380	-	-	435,206	-		-			Pre-Strip
1372.5	-	-	1,891,628	-		-			Pre-Strip
1365	-	49,144	3,461,713	-		32,230	656	70.44	Pre-Strip
1357.5	265,679	1,490,179	2,878,606	257,368	969	1,023,069	687	1.64	
1350	2,092,319	1,917,600	598,612	2,055,801	983	1,391,338	726	0.15	
1342.5	3,931,533	342,289	-	4,013,394	1,021	264,996	774	0.00	
1335	3,942,022	850	0	4,121,463	1,046	765	900	0.00	
1327.5	3,661,065	-	-	3,987,759	1,089	-		0.00	
1320	3,420,701	-	-	3,872,178	1,132	-		0.00	
1312.5	3,186,711	-	-	3,774,261	1,184	-		0.00	
1305	2,963,340	-	-	3,629,680	1,225	-		0.00	
1297.5	2,771,133	-	-	3,491,774	1,260	-		0.00	
1290	2,579,615	-	-	3,270,861	1,268	-		0.00	
1282.5	2,423,065	-	-	3,046,314	1,257	-		0.00	
1275	2,262,448	-	-	2,788,719	1,233	-		0.00	
Phase 7									
1387.5	-	-	2,016	-		-			Pre-Strip
1380	-	-	19,894	-		-			Pre-Strip
1372.5	-	-	34,676	-		-			Pre-Strip
1365	-	-	248,245	-		-			Pre-Strip
1357.5	19,950	144,114	1,295,969	23,059	1,156	118,212	820	7.90	Pre-Strip
1350	780,940	619,434	656,575	860,456	1,102	502,950	812	0.47	
1342.5	2,059,290	5,407	42,773	2,358,441	1,145	4,123	762	0.02	
1335	2,022,407	-	26,672	2,356,625	1,165	-		0.01	
1327.5	1,994,916	-	15,166	2,351,490	1,179	-		0.01	
1320	1,940,510	-	15,305	2,294,163	1,182	-		0.01	
1312.5	1,907,823	-	16,165	2,213,539	1,160	-		0.01	
1305	1,877,296	-	15,571	2,107,667	1,123	-		0.01	
1297.5	1,818,466	6,709	15,888	1,961,004	1,078	5,981	891	0.01	
1290	1,769,461	36,343	12,139	1,864,141	1,054	32,239	887	0.01	
1282.5	-	-	-	-		-			
1275	-	-	-	-		-			
Phase 8									
1387.5	-	-	-	-		-			
1380	-	-	-	-		-			
1372.5	-	-	216,352	-		-			Pre-Strip
1365	-	-	1,539,579	-		-			Pre-Strip
1357.5	-	-	4,015,346	-		-			Pre-Strip
1350	519,802	197,992	4,814,316	483,006	929	173,078	874	6.71	
1342.5	4,885,281	430,747	4	5,082,946	1,040	377,813	877	0.00	
1335	4,953,946	8,080	0	5,448,335	1,100	7,226	894	0.00	

Bench	Ore Tonnes (millions)	Low Grade Ore Tonnes (millions)	Waste Tonnes (millions)	Li Contained (millions kg)	Li Grade (ppm)	Low Grade Li Contained (millions kg)	Low Grade Li Grade (ppm)	Stripping Ratio	Pre-Stripping
1327.5	4,640,106	-	-	5,233,472	1,128	-		0.00	
1320	4,404,573	-	-	5,049,130	1,146	-		0.00	
1312.5	4,196,967	-	-	4,877,166	1,162	-		0.00	
1305	3,990,447	-	-	4,662,417	1,168	-		0.00	
1297.5	3,784,819	-	-	4,405,003	1,164	-		0.00	
1290	3,594,183	-	-	4,138,491	1,151	-		0.00	
1282.5	3,425,258	-	-	3,862,105	1,128	-		0.00	
1275	-	-	-	-		-			

All pre-stripping and waste handling is carried out by CAT 657G or equivalent scrapers. Pre-stripping of waste is conducted if there is no ore present on a bench or if the amount of waste exceeds 20 times the amount of ore. Benches to be pre-stripped are shown in Table 16-6.

For all other benches, all waste and low grade on a bench is scheduled to be mined over the same duration as the ore on that bench. This method resulted in years with higher waste quantities to be mined.

GRE moved the high pre-stripping waste production years to previous periods where there is very low waste handling to smooth out the production and generate an efficient production schedule.

A portion of the waste material may be suitable for construction gravel. Whether material classified as low grade will be stockpiled to be processed later or instead is milled when it is mined will be a function of efficiency and future lithium prices.

16.4 Mine Operation & Layout

A preliminary mining schedule was generated from the constrained pit mineral reserve estimate. GRE used the following assumptions to generate the schedule.

- Process production rate: 15,000 tpd
- Mine operating days/week: 7
- Mine operating weeks/year: 52
- Mine operating shifts/day: 2
- Mine operating hours/shift: 10

A summary of the production schedule is shown in Table 16-7 and Figure 16-12.

Ore will be transported from the open pit to the process plant via over-land conveyors. Low-grade material between 600 and 900 ppm Li will be transported to the low-grade stockpile via over-land conveyors. Waste rock will be transported to the waste dump using scrapers.

Main access and haul roads are designed with a width of 30.5 meters to accommodate the proposed equipment fleet, including ditches and berms. The access roads are wide enough to accommodate two-way traffic. The maximum road gradient is 8%. Figure 16-13 shows a typical haul road profile.

Pages 114 through 116 are intended to print n
Landscape on Tabloid or 11 x 17-inch paper.

Table 16-7: Mine Schedule

Pit Phase	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35	Year 36	Year 37	Year 38	Year 39	Year 40	Total
Ore Tonnes (millions)																																										
1	0.00	3.53	5.37	5.48	5.48	5.48	5.48	3.14																																		33.93
2								2.34	5.48	5.48	5.48	1.28																														20.04
3												4.20	5.48	5.48	5.48	5.48	2.53																									28.62
4																	2.96	5.48	5.48	0.41																						14.31
5																			0.07	5.11	5.48	5.48	5.47	5.47	5.48	3.91																36.47
6																										1.83	5.48	5.48	5.48	5.48	5.47	4.30										33.50
7																																1.20	5.48	5.48	4.04							16.19
8																																			1.43	5.48	5.48	5.47	5.48	5.48	5.48	34.28
Waste Tonnes (millions)																																										
1	0.00	0.72	0.07	0.00	0.00	0.00	0.00	0.00																																		0.79
2								1.14	0.02	0.01	0.01	0.00																														1.18
3												2.73	0.19	0.00																												2.92
4																	1.67	0.12	0.03	0.00																						1.83
5																			4.82	2.25	0.00	0.00																				7.07
6																										9.12	0.15	0.00														9.27
7																																2.27	0.07	0.04	0.03							2.42
8																																			10.59	0.00	0.00					10.59
Li Kg (millions)																																										
1	0.00	4.05	6.53	6.89	6.87	6.71	6.29	3.33																																		40.67
2								2.44	6.41	6.48	6.27	1.40																														23.00
3												4.63	6.22	6.20	6.25	6.22	2.77																									32.28
4																	3.14	6.47	6.60	0.47																						16.68
5																			0.07	5.08	5.81	6.09	6.32	6.41	6.32	4.36																40.45
6																										1.79	5.59	5.84	6.20	6.62	6.92	5.35										38.31
7																																1.34	6.37	6.35	4.34							18.39
8																																			1.43	5.79	6.08	6.23	6.34	6.39	6.34	38.59
Li Grade (ppm)																																										
1	915	1,149	1,216	1,258	1,256	1,226	1,148	1,062																																		1,199
2								1,044	1,171	1,183	1,145	1,095																														1,148
3												1,105	1,136	1,132	1,141	1,136	1,097																									1,128
4																	1,062	1,181	1,206	1,146																						1,165
5																			909	995	1,060	1,113	1,155	1,170	1,154	1,116																1,109
6																										981	1,022	1,066	1,132	1,210	1,264	1,244										1,144
7																																1,117	1,164	1,159	1,072							1,136
8																																			1,000	1,057	1,110	1,138	1,158	1,167	1,157	1,126
Low Grade Ore Tonnes (millions)																																										
1	0.00	0.09	0.07	0.00	0.00	0.00	0.01	0.24																																		0.41
2								0.00	0.00	0.00	0.00	0.00																														0.00
3												0.33	0.17	0.09																												0.59
4																	1.61	0.00																								1.62
5																			1.13	3.20	0.22	0.00																				4.55
6																										2.97	0.83	0.00														3.80
7																																0.76	0.00		0.04							0.81
8																																			0.28	0.35	0.01					0.64
Low Grade Li Kg (millions)																																										
1	0.00	0.08	0.06	0.00	0.00	0.00	0.01	0.21																																		0.36
2								0.00	0.00	0.00	0.00	0.00																														0.00
3												0.28	0.14	0.08																												0.50
4																	1.32	0.00																								1.33
5																			0.92	2.57	0.17	0.00																				3.66

Pit Phase	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35	Year 36	Year 37	Year 38	Year 39	Year 40	Total
6																										2.09	0.62	0.00														2.71
7																																0.62	0.00		0.04							0.66
8																																			0.24	0.31	0.01					0.56
Low Grade Li Grade (ppm)																																										
1	736	860	861	731	676	691	899	880																																		872
2								725	760	684	885	884																														880
3												858	847	860																												856
4																	822	877																								822
5																			815	802	792	879																				805
6																										705	746	900														714
7																																813	762		888							817
8																																			875	877	894					876

Figure 16-12: Mine Schedule

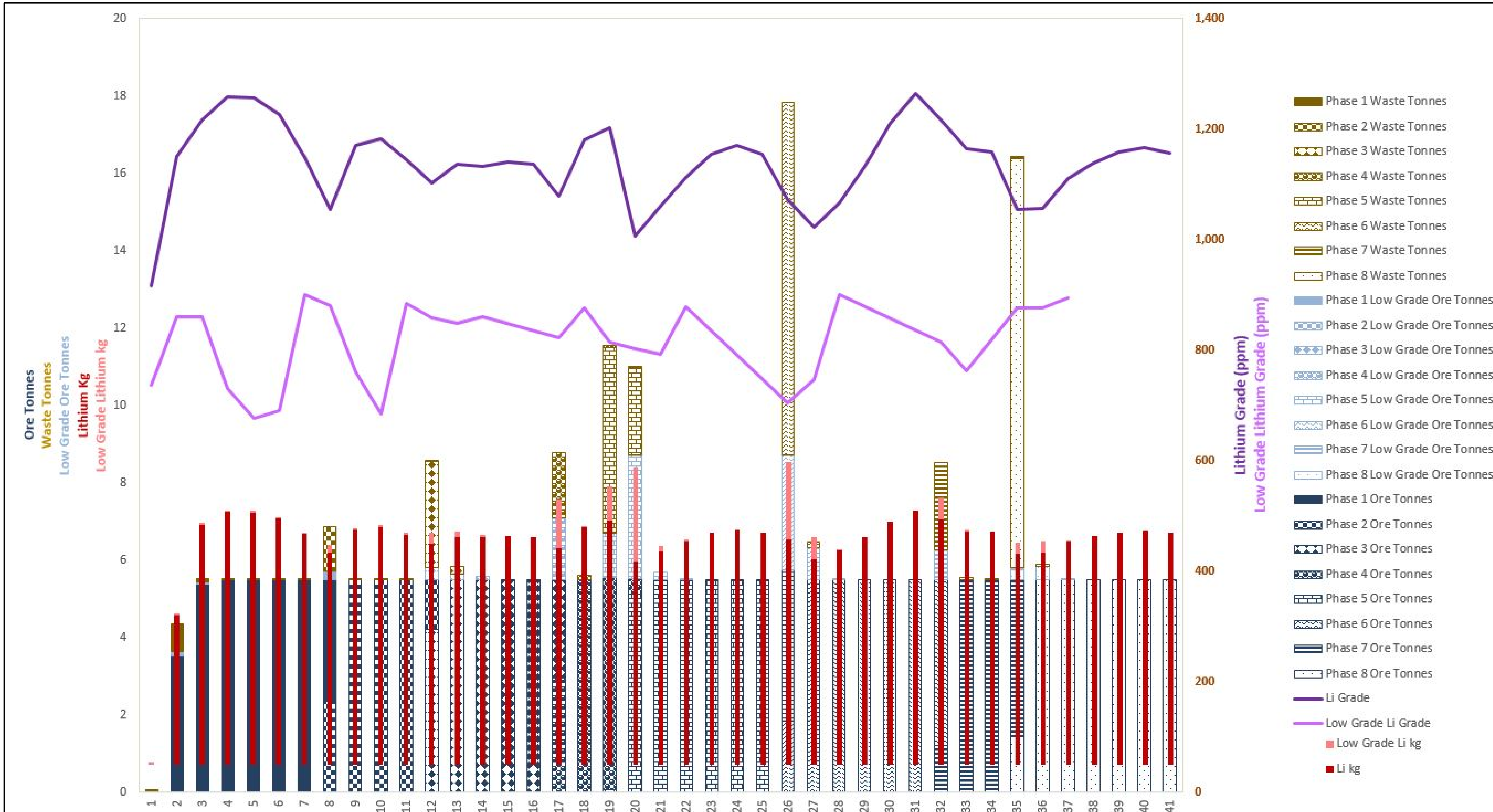
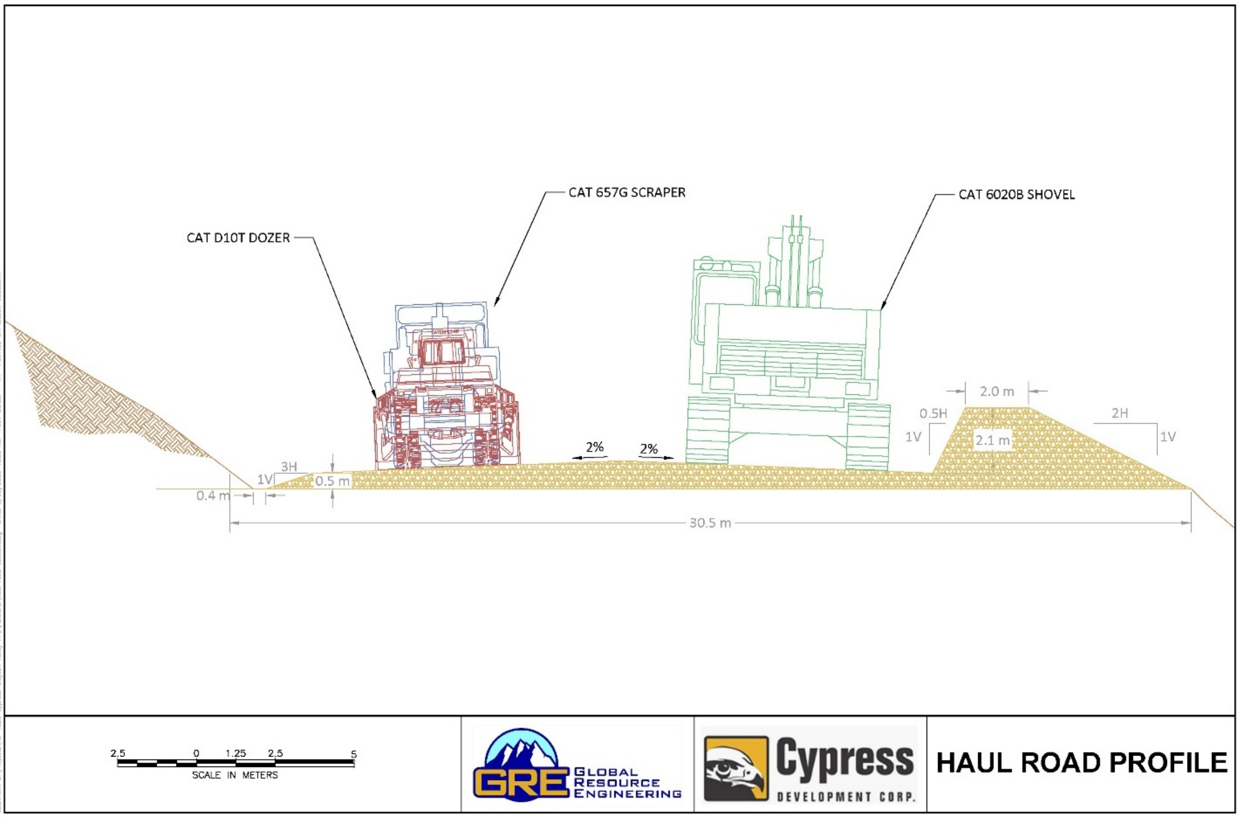


Figure 16-13: Typical Haul Road Profile



16.5 Hydrology

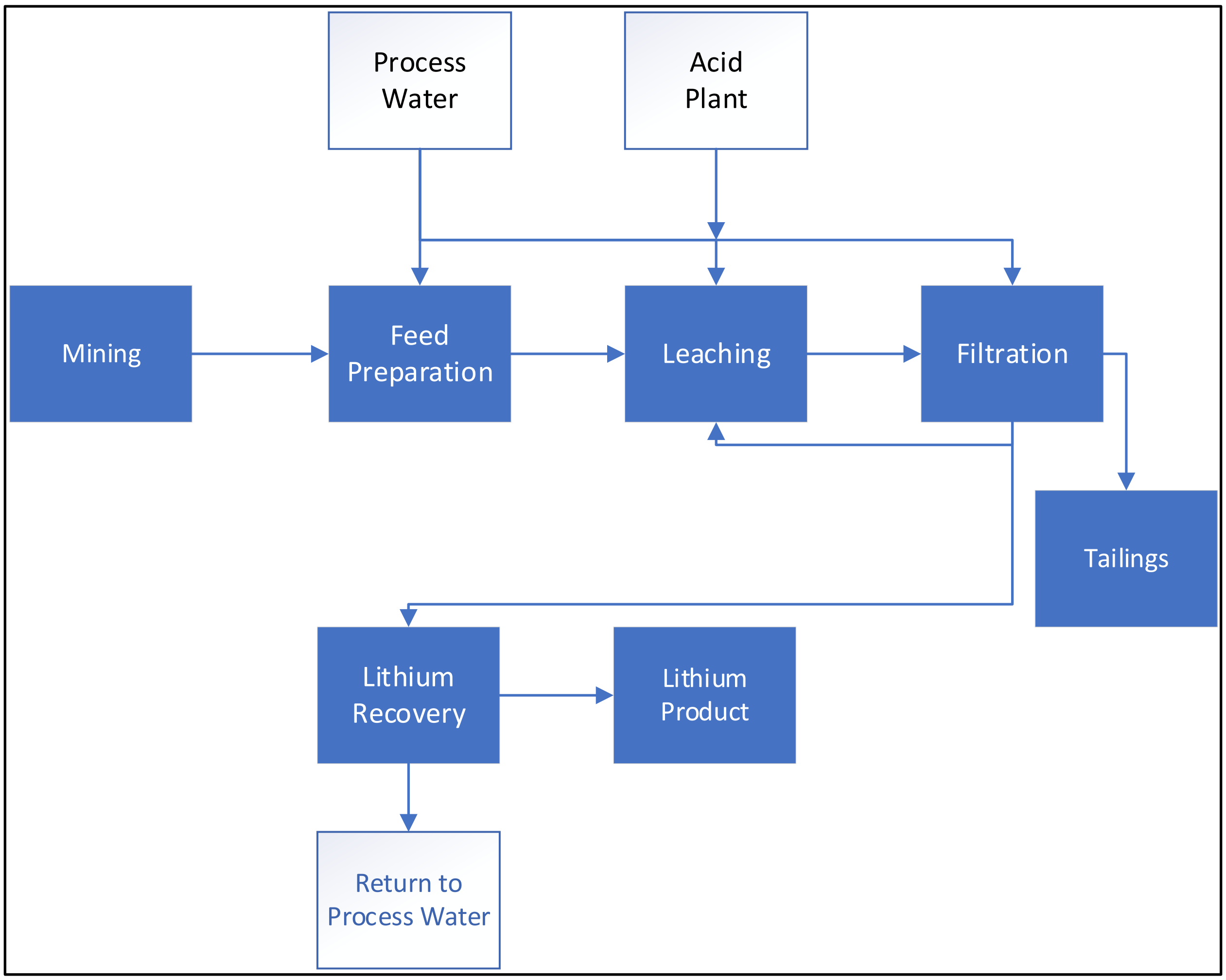
The project site has no permanent or ephemeral surface drainage. The annual average precipitation is 4.38 inches, with a 100-year, 24-hour peak event of 2.4 inches. The project area is relatively flat, with higher ground immediately to the west (Angel Island) and 2 km to the east and southeast (Split Mountain). The topography within the pit area is flat to moderate, ranging in elevation from 1,330 to 1,420 meters.

Runoff from the east has the potential to impact the mine area. A drainage ditch will be required to divert storm run-off around the pit and was designed upgradient of the project area to capture the 100-year, 24-hour peak runoff. The drainage ditch will be a minimum of 1.8 meters deep, 1.5 meters wide at the base, and will have 2:1 side wall slopes.

17.0 RECOVERY METHODS

The generalized recovery process for the project is shown in Figure 17-1. The processing follows a similar flowsheet developed in the 2018 PEA (GRE, 2018b) but with changes in materials handling with respect to filtration and lithium recovery. The processing methods continue to use industry-standard, commercially available equipment and are the basis for the capital and operating costs in Section 21.0.

Figure 17-1: Generalized Process Diagram



17.1 Design Basis

The target rate for production is set at a nominal 15,000 tpd (dry weight) of feed to the plant. This rate was established in the 2018 PEA (GRE, 2018b) based on a generalized goal of producing more than 20,000 tpy of lithium product and upon limitations identified from infrastructure items that include water, power, and consumption and supply of sulfuric acid. These considerations are unchanged. With the 15,000 tpd rate and estimated lithium grades and recoveries for the project, the design basis for the PFS results in an estimated production rate of 80 to 90 tpd of lithium product in the form of lithium hydroxide. The plant will also have the capability to produce a lithium carbonate product. For reporting purposes, all production is quoted in terms of lithium carbonate equivalent (LCE).

Table 17-1: Process Design Basis

Item	Units	Value
Mine production	kt/yr	5,475
Average lithium grade	% Li	0.114
Overall lithium recovery	%	83
Nominal processing rate	tpd	15,000
Operating schedule	days/year	350
Plant availability	%	92
Feed preparation rate	tph	738
Leach rate (solids), 4 trains	tph	171 x4
Retention time, 2 tanks	min	120 x2
Slurry flow each. train	gpm	2,243
Acid addition, total leach section	tph	86
Filtration rate, 8 filter units	tph (dwt)	92
Tailings to conveyor	tph (wet)	1,200
PLS to lithium recovery	gpm	7,700
Solution to evaporators	gpm	1,100
Make-up water to plant	gpm	2,000
Li Product (LCE)	tpd	72

17.2 Process Flowsheet

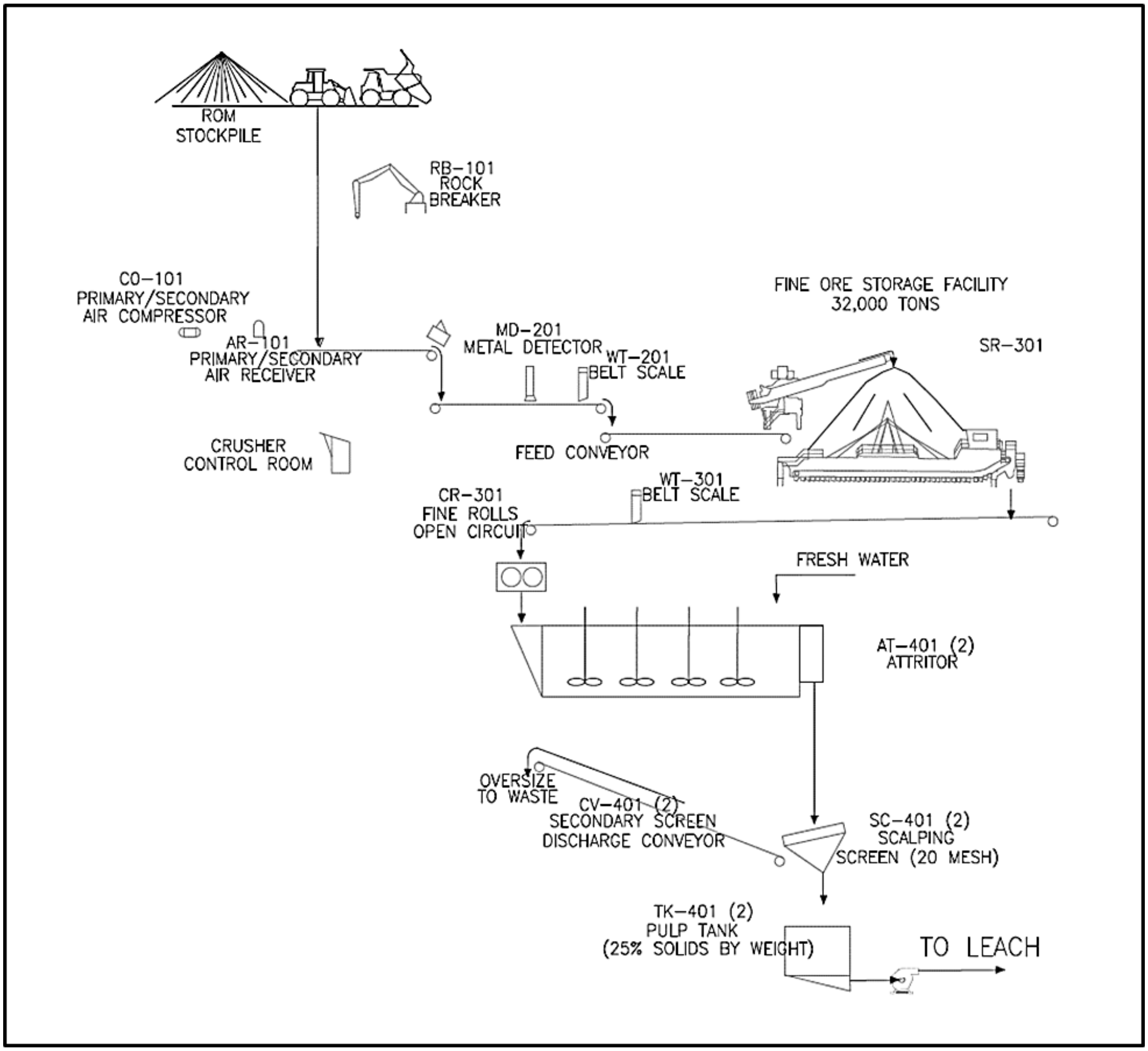
17.2.1 Mine to ROM Stockpile

Mine production will use a backhoe type excavator to dig below grade and dump material into a mobile feeder/breaker. The mobile feeder/breaker will break large lumps of material and effectively lower the run of mine (ROM) material to a nominal 125 mm to allow for conveying. The feeder breaker will feed the claystone onto a belt conveyor extending from the mine pit face to a ROM stockpile at the processing plant via a series of jump- and mainline-mobile conveyors. Conveyor haulage from the mine was selected to eliminate truck haulage and allow better efficiency in mine production and is also intended to conserve water by minimizing heavy equipment and dust control for haul road maintenance. Mine feed will be conveyed to a ROM stockpile and will be stored in a 30,000-tonne stockpile by a linear stacker.

17.2.2 Feed Preparation

Material in the ROM stockpile will be fed to the plant via a linear reclaimer discharging into a two-way splitter and pair of roll crushers with 125-mm openings and discharging into two 350-tonne fine ore bins. Material from the bins will be fed onto variable speed drive feed conveyors and into a set of four stainless steel rotary attritors which will disaggregate the clay by rotary action and reclaim water. Each attritor will operate at 175 tph and feed slurry into slurry feed tanks where additional water will be added to adjust the slurry to 20-25% solids. Figure 17-2 identifies the comminution flowsheet for the current process.

Figure 17-2: Feed Preparation Simplified Flowsheet



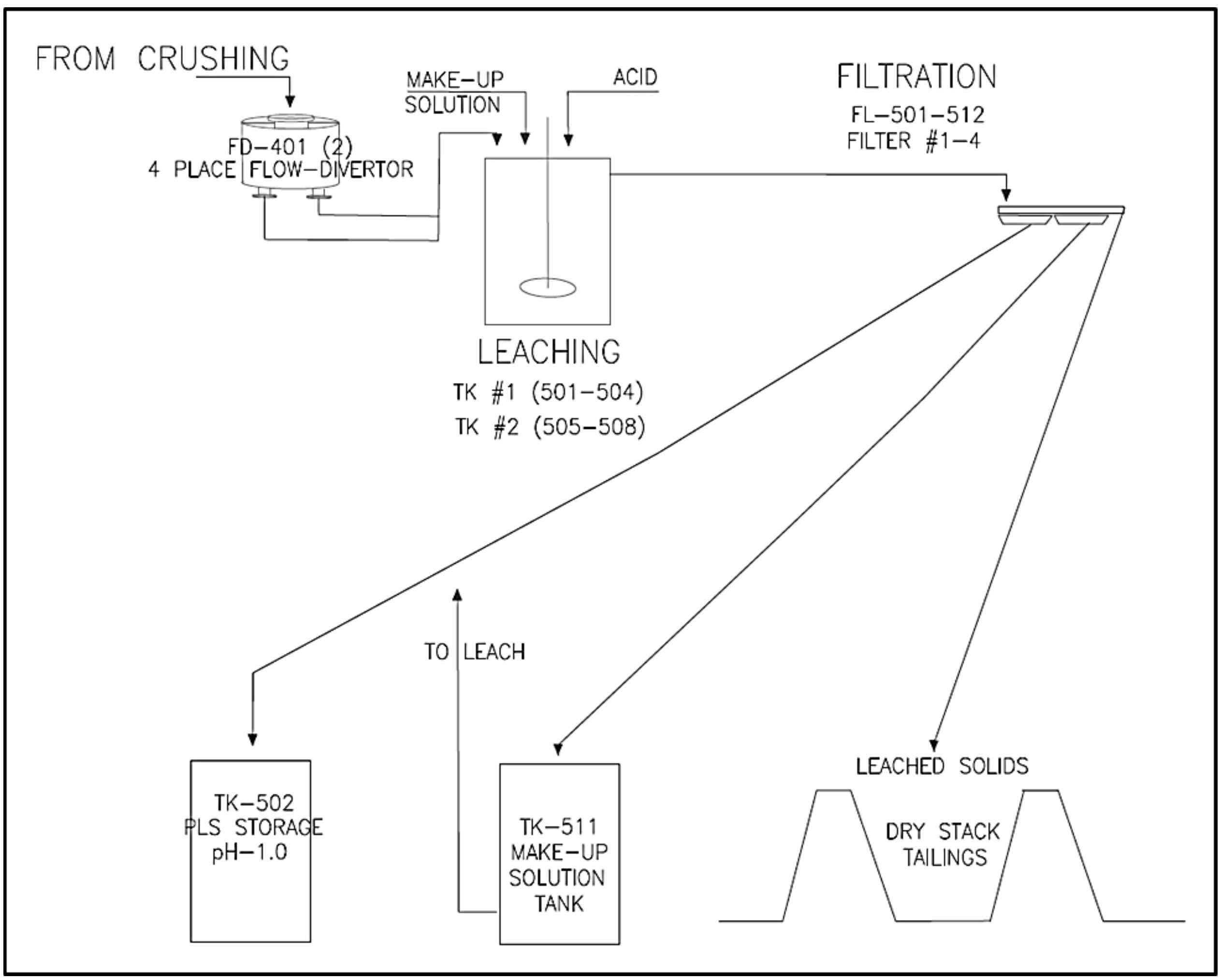
17.2.3 Leaching & Filtration

Non-acidified slurry discharging from the attritor will pass through a scalping screen and into a pump box. The scalping screen will remove oversize material. The slurry will then be pumped to a four-way splitter with each slurry split feeding into one of four leach trains. Each leach train will be equipped with two 10-meter diameter by 10-meter high stainless-steel or fiberglass tanks. The tanks will be insulated and covered to prevent heat loss and evaporation and equipped with mechanical agitators.

Sulfuric acid will be added to the first tank of each train to bring the sulfuric acid concentration to 5-10% sulfuric acid by weight. The first tank will also be equipped with steam coils to bring slurry temperatures to 60-70 °C using steam from the sulfuric acid plant or a backup boiler. Slurry from the first tank in each train will overflow into the second tank co-currently with a 2-hr retention time in each tank.

Discharge from each train of leach tanks will feed into a slurry conditioning tank and then the flow divided into one of eight filter units where the slurry will be distributed, drained of PLS and then washed with water and drained a second time. The PLS will be pumped to storage tanks at the lithium recovery plant. The wash solutions will be recycled to the reclaim water tanks for further use. The drained filter cake at near-neutral pH will be discharged onto a conveyor which will transport the filtered tailings to the tailings facility.

Figure 17-3: Leaching and Filtration Simplified Flowsheet



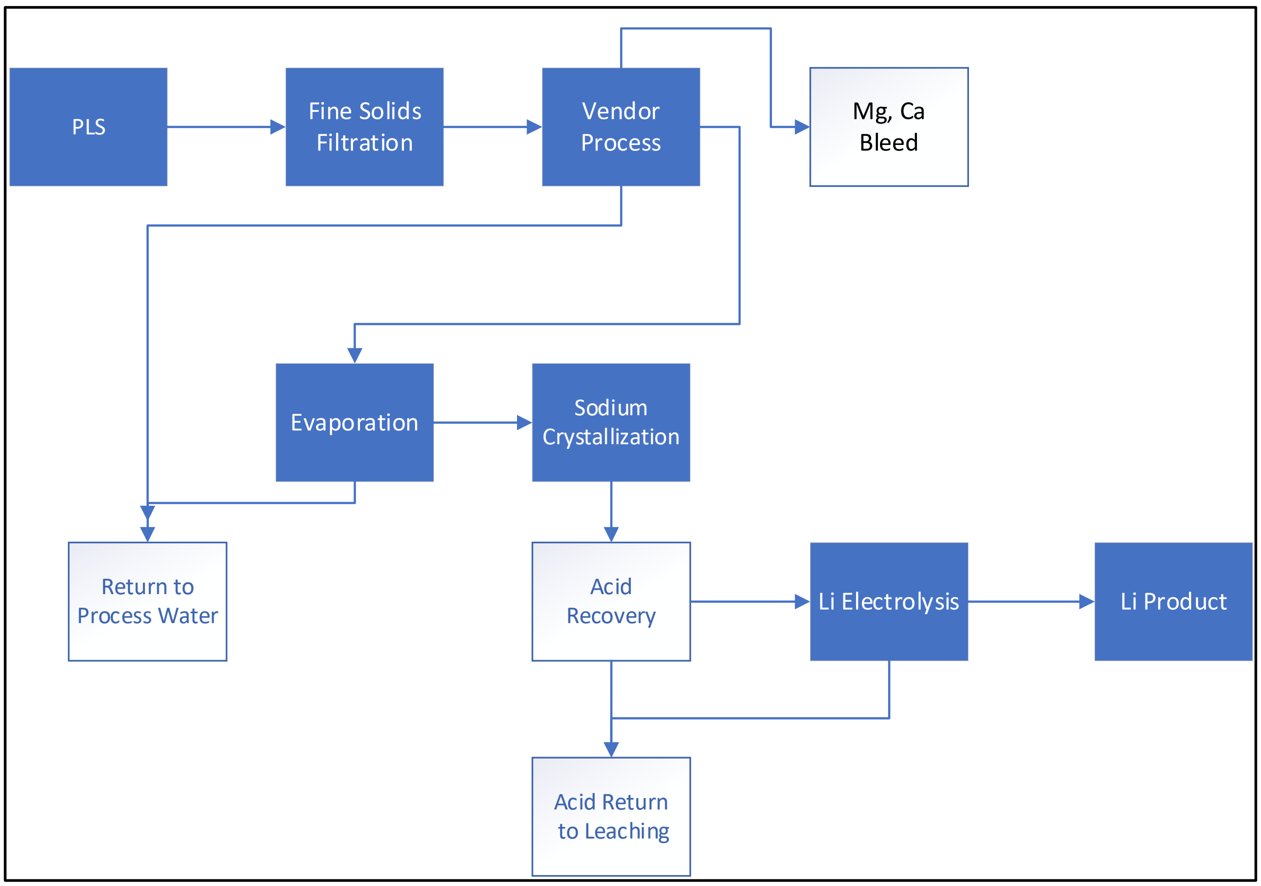
17.2.4 Lithium Recovery Plant & Production

The PLS storage tanks will feed a NORAM-CMS-designed arrangement of commercially available equipment (Figure 17-4). The units will operate under process conditions established by NORAM and CMS to remove magnesium, calcium, and other elements to a separate bleed stream prior to evaporation. Sulfuric acid and water will be recovered and returned to the leach circuit.

Concentrated lithium sulfate solution will be converted to lithium hydroxide solution via electrolytic cells followed by crystallization of lithium hydroxide monohydrate crystals. The crystals will be washed, dried and bagged for shipping.

The plant will potentially recover other products which could include rare earth elements, potassium, and other salts; and have the ability to produce lithium carbonate.

Figure 17-4: Lithium Recovery Process Diagram



18.0 PROJECT INFRASTRUCTURE

18.1 General Arrangement

The project is located within township 2 south, range 40 east and township 3 south, range 40 east, Mt. Diablo Meridian, as shown in Figure 18-1. The project is accessible by way of Silver Peak Road, a paved two-lane road north of the property that connects Silver Peak with US Highway 95 to the east. The east slope of Angel Island was identified for the plant location (Photo 18-1 and Photo 18-2). The location was selected based upon proximity to the mine area, topography, access to Silver Peak Road, power, and probably geotechnically stable subsurface for plant construction.

18.1.1 Access Roads

Primary access to the operation will be via a road developed south from Silver Peak Road to the proposed plant site as shown in Figure 18-1. This road will be adequate for semi-truck traffic. Additional access roads will be constructed to allow heavy equipment traffic between the mine and internally within the plant site. Mine roads will be minimal due to the use of conveyors in lieu of truck haulage.

18.1.2 Buildings & Yards

Structures and facilities to be installed on-site include administration, laboratory, warehouse, reagent storage, sulfuric acid plant, crushing, leaching and lithium recovery areas, mine shop, and fuel and reagent storage areas as shown in Figure 18-2. The access road to the site will enter a parking area accessible to the administration building. The processing areas and other site access points will be fenced and gated.

Administration will be housed in a building sized to accommodate supervision, accounting, safety and technical personnel. The site will be connected to communications using local phone and internet services.

The laboratory will house sample preparation and analytical equipment to handle the daily requirements of the mine and processing plant.

The mill workshop and warehouse building will be located adjacent to the processing plant and will include dry storage areas for parts, reagents and supplies. Contained tankage will be provided for acid, recycled water and liquid chemicals.

The crushing, leaching and filtration areas will be open-air contained enclosures. The process building will house the lithium recovery and product manufacturing equipment and work areas.

The building will include offices, overhead cranes, HVAC, and fire protection systems. The building will include drying and bagging equipment and area to allow for indoor storage and loading of final product.

Photo 18-1: View of Plant Site Area from Pit Looking Northwest

Photo 18-2: View from Plant Site Area Looking Toward Pit Looking Southeast

Figure 18-1: General Arrangement of Facilities

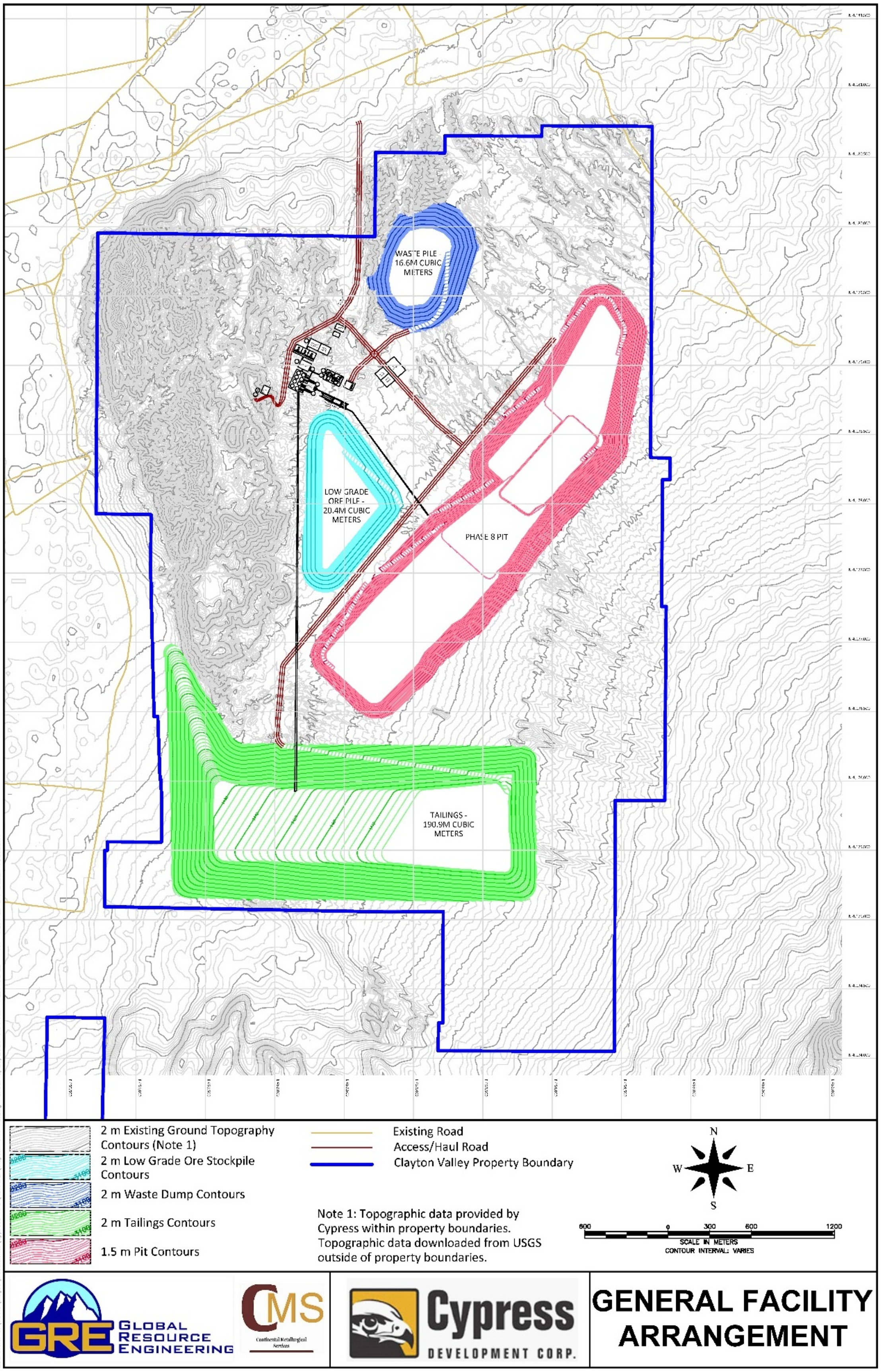
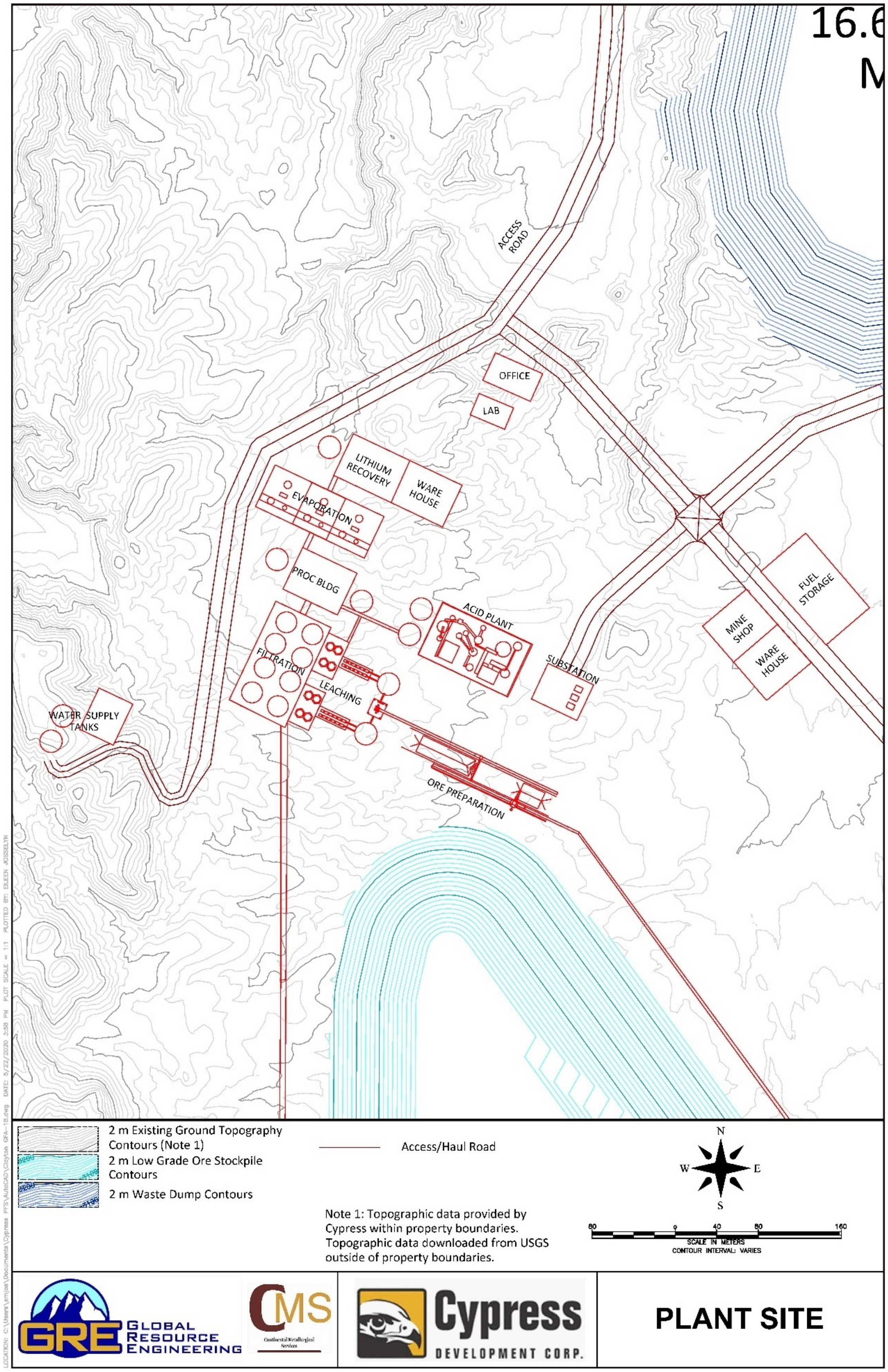


Figure 18-2: Plant Site



The mine shop will allow for two service bays and include offices, an overhead crane, compressed air, tool rooms, lubrication availability, and storage for conveyor and other repair parts.

Fuel and lube storage will be in a contained open-air area that will service the mine and plant mobile equipment. Diesel fuel will be delivered in tanker trucks and stored in tanks with 10,000 gallons total capacity.

18.2 Sulfuric Acid Plant

The sulfuric acid plant is a Dupont MECS plant with full energy recovery (Dupont, 2020). The plant can produce 2,500 tons/day (100 weight% H_2SO_4 basis) of sulfuric acid by burning elemental sulfur. The process generates large amounts of heat which is captured as steam to heat leach tanks and other processes in the plant and generate 27.5 megawatts (MW) of power.

The plant will be equipped to meet National Ambient Air Quality Standards emission limits in accordance with the State of Nevada Implementation Plan.

Elemental sulfur in dry form will be delivered to the site by truck at the rate of 800 tpd.

Sulfuric acid will be stored on site in tanks adjacent to the leach plant. The tank storage area will include load-out to provide the option for shipping and sales of excess acid.

The plant is sized to meet 100% of the power needs of the mine and process facilities with surplus power. A main substation will be located adjacent to the sulfuric acid plant for power distribution to the site. The substation will be connected to the regional power grid and have the capability to send surplus generated power for sales off site.

Cooling for the acid plant is provided a closed indirect water circulation loop and directly at the turbine condenser.

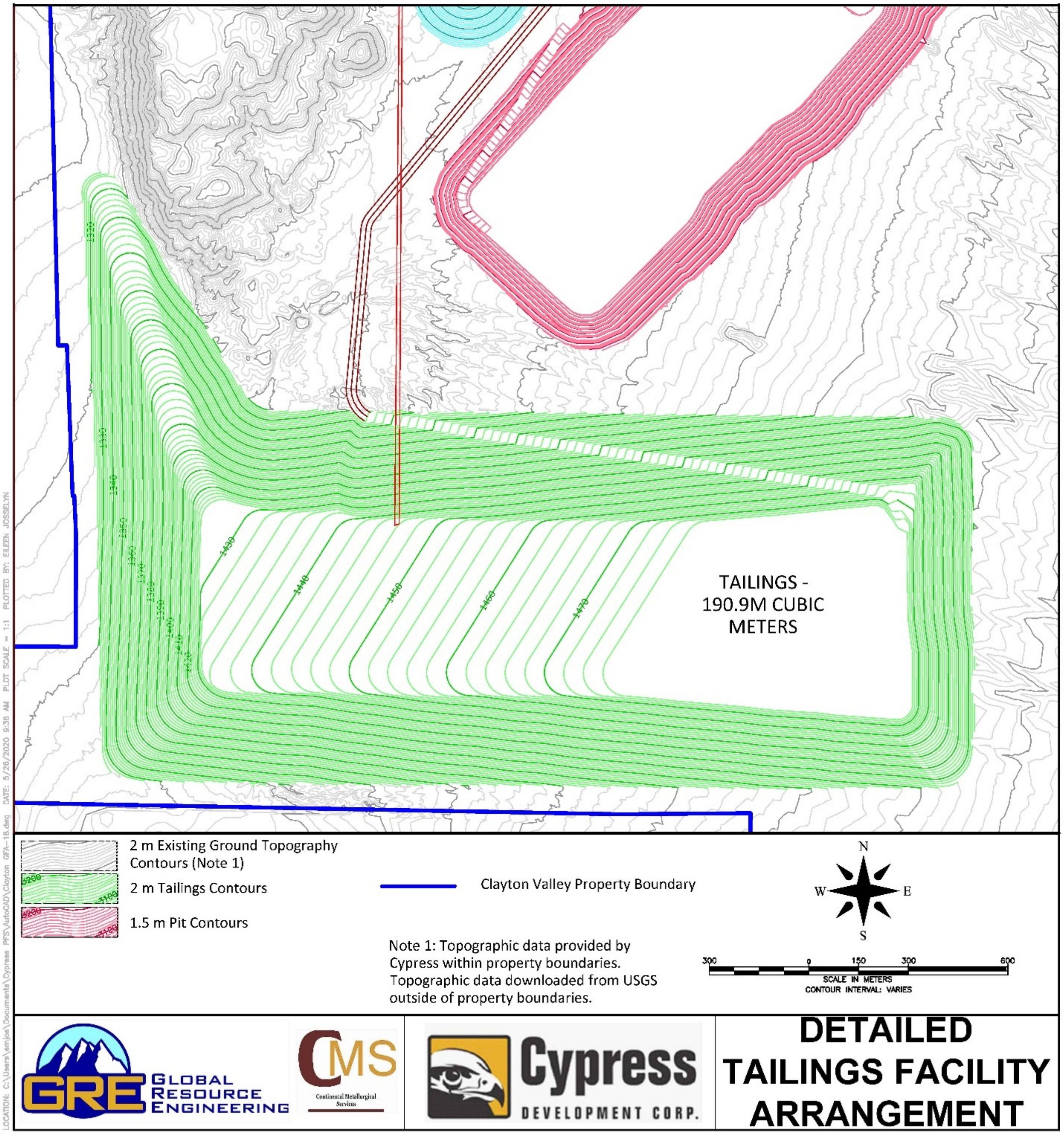
18.3 Tailings Facility

Tailings will be conveyed from the filtration plant to a facility south as shown in Figure 18-3. The tailings will be placed via a stacking conveyor. Dozers will be used for final spreading and contouring. Tailings will be allowed to dry and compacted as necessary to a target 90 to 95% of the standard Proctor density, which will minimize any possibility of solution migration. The stacking operation will support a 30-meter high stack.

Pocock Industrial performed physical testing on the tailings. The tailings are expected to be stable when placed and compacted. The following the physical characteristics were determined:

- Tailings Median Size – 5.5 microns (Hazen - Horiba Particle Size Analyzer)
- Tailings Hydraulic Conductivity – 5.3 x 10^{-8} centimeters per second at a compaction to 94.6 pounds per cubic foot @ 75% moisture (IGES – ASTM D5084 – Method C)
- Solids SG – 2.60 to 2.73

Figure 18-3: Dry Stack Tailings Area at Life of Mine



18.3.1 Construction

An initial starter berm will be constructed with waste material. Concurrent tailings placement and berm construction will occur throughout the life of the repository. Waste material will be advanced ahead of the tailings level in successive lifts using the upstream construction method.

The berms will accommodate haul traffic and outer slopes generally of 3H:1V with benches to achieve an overall sloped facility of 3.5H:1V.

Once the perimeter berms are placed across the drainages and washes, stormwater run-on will be limited upstream of the dry stack areas. During operation, the tailings surface will be sloped away from the edge of the facility to limit potential water impoundment overtopping the dam and eroding the facility sides. Perimeter ditches will be constructed around the outer edges of the berms to move water in and around the facility.

18.4 Power Supply

Power on site will be provided primarily by the sulfuric acid plant. Secondary power will be provided by connection to the regional grid. Clayton Valley is served by two 69 kV transmission lines, one of which is located just north of the project by Silver Peak Road.

Discussions with NV Energy, the local utility company, indicate the existing lines can provide the required power to the project. Provisions are included in the capital costs for upgrading 50 km of the line to assure start-up and operation of the project when the acid plant is not operating.

Power on-site will be distributed from a main substation located adjacent to the sulfuric acid plant. Line feed to areas of the plant and mine will be via overhead and buried lines as required and stepped down to appropriate voltages. The estimated power requirements by area are presented in Table 21-6.

Cypress holds a geothermal lease five miles north of the project. The lease is a potential source of additional power that the Company plans to evaluate in conjunction with the project.

18.5 Water Supply

A water balance model was prepared based upon water requirements for the mine and processing plant, with consideration of losses to evaporation and tailings. Total water use in processing is estimated at 8,000 gpm. Approximately 75% of the water will be recycled from the processing plant and be returned to the leaching circuit. Makeup water required for the project is estimated at 2,000 gpm.

The Clayton Valley basin has groundwater to support the project, but the water rights are fully allocated and held by several parties. Cypress has evaluated options for securing makeup water. The options are dependent on future conditions and agreements with other entities. For the PFS, the cost of acquiring a source for makeup water was not included. The costs of supply wells, pipeline, and power to provide makeup water to the project site are included.

The project will have a dedicated water system to provide fire protection to all areas of the processing plant.

18.6 Waste Management

Other than treated effluent from the site septic systems, the project will have no water discharge to the environment. 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 drain field are sized for the building occupancy.

Solid waste will be placed in dumpsters or other appropriate containers for transport off-site.

Hazardous waste will be placed in appropriate containers to be transported offsite by a licensed contractor.

18.7 Storm Water Handling

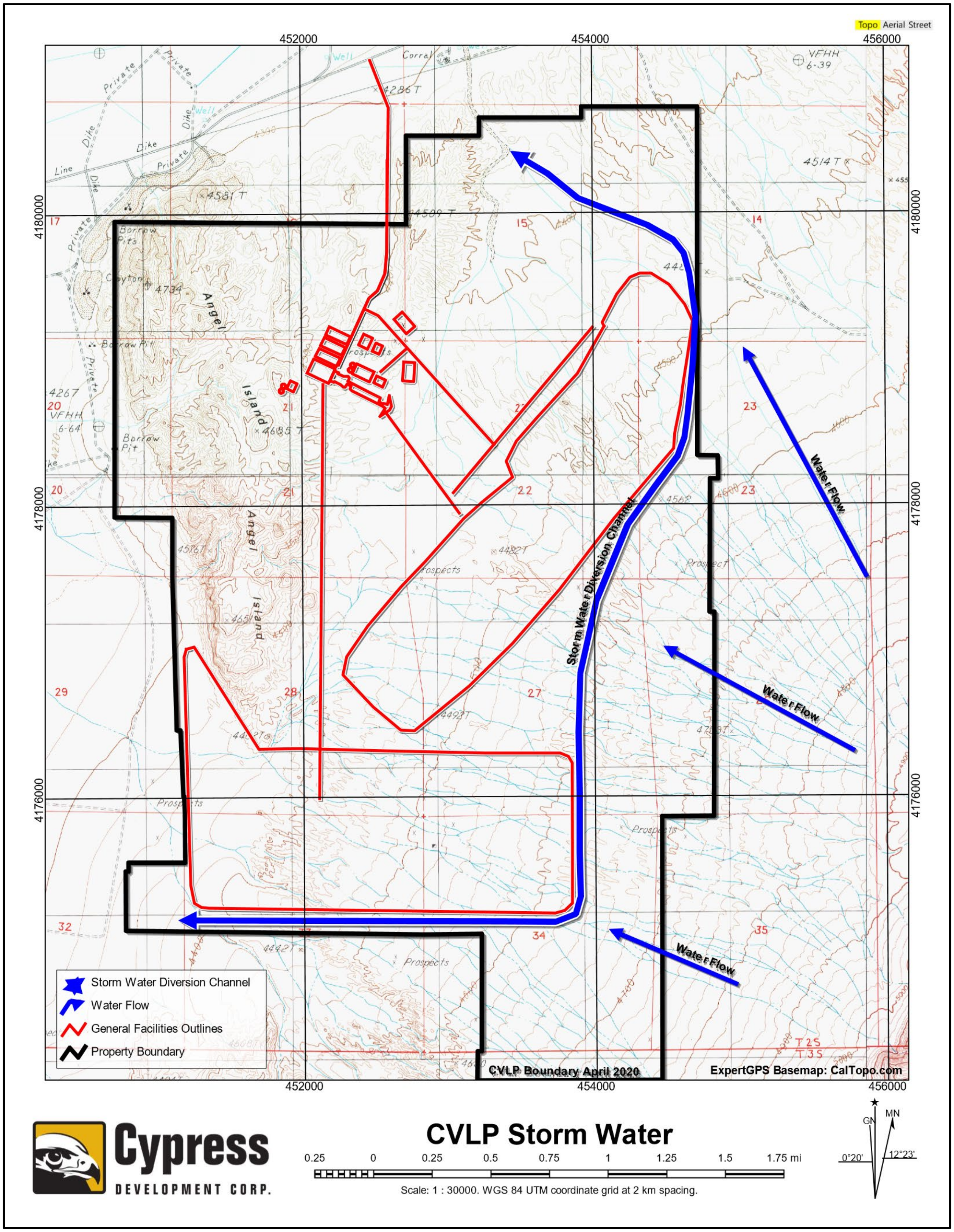
The mine site is located at the base of an alluvial fan. As shown in Figure 18-4, the alluvial fan is fed by a canyon two miles east of the project and covers an area of several square miles. Minor fans emit from the canyons north and south and contribute to the surface run-off. The surface run-off flows mostly north to the playa across Silver Peak Road, or south around the southern tip of Angel Island.

Storm water flowing over the alluvial fan will be diverted around the eastern perimeter of the mine area, leaving the surface flows unchanged from their present course.

The plant site will be located on the east slope of Angel Island, unaffected by surface run-off from the east. The access road to the plant will follow a minor depression avoiding the major outflow point which is presently the north access route onto the property.

Storm water in and around the plant area will be diverted to settling ponds. Storm water within containment areas will be treated accordingly prior to discharge.

Figure 18-4: General Storm Water Flow



19.0 MARKET STUDIES & CONTRACTS

Cypress has no agreements or contracts in place for the sale of lithium products or for the purchase or sale of any other commodities, resources or supplies except for the underlying royalty agreement described in Section 4.0.

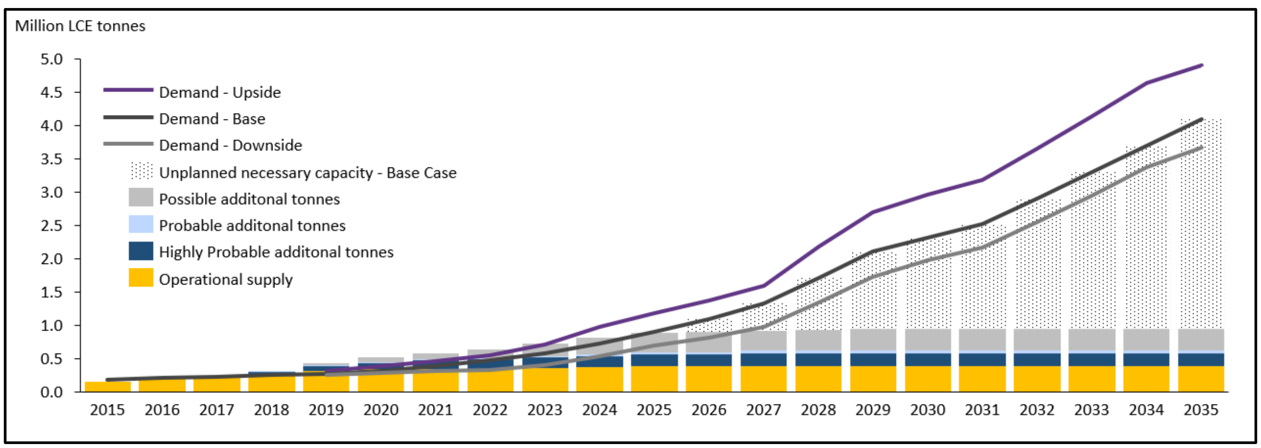
The following describes the price assumptions for lithium and the major consumable items affecting the project.

19.1 Lithium Supply & Demand

The outlook for lithium was examined in the 2018 PEA (GRE, 2018b) and is the subject of numerous published reports and analyst reviews.

Lithium is an indispensable element in lithium-ion batteries for which substitution appears unlikely. Current global annual consumption for all uses of lithium totals 248,000 tonnes of LCE (Benchmark Mineral Intelligence., 2018), and shown in Figure 19-1. Approximately half of this demand is attributed to batteries for electric vehicles, grid storage, and portable electronic equipment, which is divided roughly equally between carbonate and hydroxide forms of lithium. As concluded by Benchmark and others, the demand for lithium is set to grow rapidly with the adoption and increased demand for electric vehicles. Forecasts favor the growth in demand for lithium in hydroxide form outpacing that for lithium carbonate.

Figure 19-1: Lithium Demand—Supply Balance



19.2 Lithium Price Assumption

In the 2018 PEA, Benchmark (Benchmark Mineral Intelligence., 2018) identified a shortfall in lithium supply beginning in 2025. They determined the additional supply to meet demand will require a price of $13,000/tonne of LCE to support the development of new higher capex projects.

Since that time, lithium prices have decreased. As of the effective date of this report, LME lists a price of $8,000/tonne for lithium carbonate and $9,750/tonne for lithium hydroxide monohydrate (battery grade, minimum 99.5% Li_2CO_3 for lithium carbonate and 56.5% $LiOH_2O$ for lithium hydroxide monohydrate), as the spot prices CIF for China, Japan and Asia.

The project is expected to produce lithium hydroxide and suitable for purchase by Tier 1 battery producers as described by Benchmark (Benchmark Mineral Intelligence., 2019). A price of $9,500/tonne is used in the economic analysis. This price is applied to the production as converted to tonnes of LCE (kg Li x 5.323 / 1000). The price assumption reflects variations expected over time, including lower prices during the early years of start-up and higher prices later when the operation may receive a premium for its product.

19.3 Elemental Sulfur

Elemental sulfur is required by the operation to generate sulfuric acid and is a major part of the project operating cost.

Ausenco (Ausenco Engineering Canada Inc., 2020) conducted a market survey and logistics study and determined the required sulfur demands of the project can be filled domestically. Ausenco was quoted prices F.O.B. California in the range of $100 to $150/tonne for elemental sulfur as of Q1 2020. Sulfur supply and demand are linked to the oil and agricultural industries. World sulfur prices within last 20 years have fluctuated in a broad range, from virtually free to over $600/tonne. For this report, a base price of $100/tonne is used excluding transportation.

19.4 Electric Power

Published commercial power rates for Nevada are used for the PFS. Based upon the project's connected demand and use, the weighted cost for grid power is calculated at $0.066/kilowatt-hour (kWh). The project's acid plant includes 27.5 MW of generating capacity, enough to offset 100% of the power requirements of the operation (21.6 MW) when the acid plant is running. The local utility NV Energy indicated the sale of surplus power is possible but at uncertain terms that are negotiable. No credits are therefore assumed in the PFS for surplus power sales.

20.0 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

In 2019, a Phase I Environmental Site Assessment of the project was conducted by Stantec Consulting Services Inc. The study found no existing environmental liabilities. Stantec in 2019 also completed a Threatened and Endangered Species Preliminary Study. The study investigated various sources of public records to lay the groundwork for further field work.

Also, in 2019, U.S. Department of Interior Bureau of Land Management published an Environmental Assessment (EA) (DOI-BLM-B000-2019-0009-EA) titled September 2019 Competitive Geothermal Lease Sale EA (BLM, 2019). The EA covers lands in and around the Geothermal Lease Sale Parcel NV-19-027 now held by Cypress.

Cypress is presently bonded under existing Notice Level permits with Bureau of Land Management. These permits are in good standing or were closed upon satisfactory completion of reclamation work. There are no mine workings or tailings of significance within Cypress' claims.

20.1 Permits Required

Environmental permitting requirements for the Project are expected to be like other mines in Nevada. The permitting process consists of submitting a Plan of Operations to the Bureau of Land Management, who will act as lead agency, conducting environmental baseline studies, and preparing an Environmental Impact Statement along with other permit applications prior to site development and operations. The applications will include 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. Plans and permits are expected to include the following in order of importance:

- Plan of Operations under 43 CFR 3809, State of Nevada and U.S. National Environmental Policy Act compliance, Bureau of Land Management
- EA or Environmental Impact Study
- Reclamation Permit, Nevada Department of Environmental Protection (NDEP)
- Water Pollution Control Permit, NDEP Bureau of Water Pollution Control
- Stormwater NPDES General Permit, NDEP Bureau of Water Pollution Control
- Waters of the U.S., Corps of Engineers
- Class II Air Quality Operating Permit, NDEP Bureau of Air Pollution Control
- Permit to Appropriate Public Waters, Nevada Department of Water Resources State Engineer
- Industrial Artificial Pond Permit, Nevada Department of Wildlife
- Hazardous Materials Permit, NDEP Bureau of Waste Management
- Solid Waste Permit, NDEP Bureau of Waste Management
- Onsite Sewage Disposal System General Permit, NDEP Bureau of Water Pollution Control
- Potable Water Permit, NDEP Bureau of Safe Drinking Water

20.2 Timeline

In order to secure the above permits, data from the following studies will be collected.

- 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
- Initiate Plan of Operation

- In 2017, the U.S. President issued Executive Order 13817, “A Federal Strategy to Ensure Secure and Reliable Supplies of Critical Minerals” which is intended to streamline the permitting processes for mineral exploration and development for critical minerals including lithium (Exec. Order No. 13817, 2017). Subsequently, the Department of Interior issued Secretarial Order No. 3355. Under the order, BLM is limited to 12 months to complete an Environmental Impact Statement from the time of issuing a Notice of Intent. With the above studies, the time frame for permitting the project is estimated at 18 to 24 months.

21.0 CAPITAL & OPERATING COSTS

21.1 Capital Costs

The capital and operating costs are estimated according to accepted methods for prefeasibility studies. The estimates constitute a Class 4 estimate, as defined by the AACE International, and have an accuracy of +30%/-15%. General arrangement drawings, process flow diagrams and material mass balances were used to develop the major equipment list for each of the operating area in the project. Responsibilities for the estimates are as follows:

- Mining GRE
- Processing CMS
- Site G&A CMS
- Owner's Costs CMS/CYP

Vendor quotes, internal data and public information were used in the estimates. Factors were applied to processing plant and to building-related items to allow for construction and installation of fixed equipment. Indirect costs allow for EPCM, freight, sales tax and Owners Costs and are added prior to the application of contingency.

All costs are presented in Q1 2020 US$. No forward escalation is applied. A summary of the capital costs is shown in Table 21-1.

Table 21-1: Capital Cost Summary

Area	$ x 1,000
Facilities	5,891
Mine	34,757
Plant	306,855
Infrastructure	25,907
Owners Costs	24,992
Contingency & Working Capital	94,883
Total Capital Cost	**493,284**

The initial capital costs total $493 million, which includes $95 million in contingency plus working capital. The items and breakdown of estimates for each area are as follows.

21.1.1 Direct Costs

Site Development and Facilities

Factored budgetary estimates are used for earthworks for buildings: main office building, laboratory, mill maintenance shop and warehouse, safety/first aid building, and mine maintenance shop.

The building estimates are inclusive of office furnishings, HVAC, septic, electrical and communications fire protection and security systems, and shop and laboratory equipment.

Included are administration and processing plant mobile equipment: pickups, ambulance, flatbed truck, mobile crane, front end loader, and forklifts (Table 21-2).

Table 21-2: Site Facilities Summary

Area	$ x 1000
Offices & shops	4,458
Mobile Equip	800
Total Direct	5,258
Indirect	632
Total	**5,891**

Mining

Mine development costs include access and haul roads, a heavy equipment workshop and mine warehouse, and fuel station. Estimates are made from factored published and internal data. The mine shop and warehouse are inclusive of offices, dry, tools, overhead crane, HVAC, septic, electrical and communications, and fire protection (Table 21-3).

Table 21-3: Mine Capital Summary

Area	$ x 1000
Development	4,388
Production Equipment	20,872
Support Equipment	3,895
Other Mining	592
Total Direct	29,747
Indirect	5,009
Total	**34,757**

Estimates for mine production and support equipment are made from vendor quotations for major items (Caterpillar, Superior and MMD) and internal data for minor equipment.

Mine production equipment consists of a 6020B shovel, two D10T dozers, a 657G scraper. Transportation from mine to mill stockpile consists of a 500-horsepower (hp) feeder-breaker, 66 30-hp 100-foot mobile jump conveyors, and two 400-hp overland conveyors. A radial stacker feeding the mill stockpile is included in the processing capital.

Mining support equipment consists of 150-hp grader, 5000-gallon water truck, service/tire truck, light stands, pumps, pickups, and compactor. The dozers and support equipment will also provide road and yard maintenance and service the tailings facility as needed.

Other mining supplies and equipment includes surveying equipment, computers, software, plotter, and radios, which are estimated using factored internal data. Included is allowance for initial consumables, diesel fuel and tires, which are estimated on one month use in operating costs.

Processing Plant

Estimates for processing capital are made by vendor quotes, and published or internal data, which are factored to the size or rate of operation where appropriate (Table 21-4).

Table 21-4: Processing Capital Summary

Area	$ x 1000
Feed Preparation	10,731
Leaching	14,358
Filtration	32,211
Tailings Handling	3,589
Li Recovery	44,930
Acid Plant	102,585
Construction Directs	56,858
Total Direct	265,262
Indirect	41,593
Total Plant	**306,855**

Feed preparation area includes rail stacker, stacked ore area, rail reclaimer, chutes, conveyors, metal detector, magnet and weightometer, fine ore roll crushers, fine ore bins and support structures.

Leaching area consists of attritors, pulp tank, pumps, flow divertor, and support structure, followed by covered Leach tanks equipped with agitators and heating coils.

Filtration area includes flocculation tank and equipment, flow divertor, filtration units, discharge chutes, PLS tanks, reclaim water tanks, and pumps. The filtration units are quoted by a single vendor and make up 90% of the direct capital cost in this area. The vendor's quote includes piping, motors, electrical controls and instrumentation, engineering and installation supervision.

Tailings handling includes conveyors from the filtration area to the tailings facility and a radial stacker.

Lithium recovery area includes PLS handling in filtration, concentration and acid recovery units, and lithium production to lithium hydroxide and carbonate, drying and bagging equipment. The estimates for PLS handling and lithium production areas were developed by CMS using multiple vendor quotes and include pumps, piping, electrical distribution and instrumentation. The estimated equipment costs are roughly divided between the two areas as shown below. The lithium recovery building includes offices, dry, overhead crane, HVAC, septic, electrical and communications, and fire protection.

- PLS handling 42%
- Lithium production 51%
- Ancillary equipment & building 9%

Acid plant and distribution includes a 2,500 tpd acid plant with sulfur melting, burner, acid storage, steam and electricity production, blower and environmental controls. The plant is quoted by a single vendor inclusive of all piping, electrical and instrumentation, foundation and support structures and represents 39% of the total direct costs of the processing plant.

Processing Plant Construction

With exception of the acid plant, construction allowances are applied to the plant capital equipment items above to arrive at the total processing plant cost. The Construction Direct Costs allow for installation, concrete, steel, piping, electrical and instrumentation controls, and are estimated by percentages of the equipment costs based upon internal and published data for similar installations. The acid plant is quoted as a turn-key installation by a vendor.

The net construction allowances on the basic plant equipment total 54% as follows:

- Installation, concrete, and steel: 35%
- Piping, electrical and instrumentation: 19%

Table 21-5: Plant Construction Costs

Area	$ x 1000
Plant equipment (exclusive of acid plant)	105,819
Installation, concrete & steel	36,413
Piping, electrical & instrumentation	20,243
Total Plant	**162,474**

The following general costs were used to review the construction costs:

- Earthwork
    - Grading and Leveling - $3,100/acre
    - Structural Excavation - $4.50/yd³
- Concrete: $400/yd³ – medium to large structural footings with vibration.
- Structural Steel:
    - General $827/t
    - Stainless $3,650/t
- Detail and Fabrication $5,280/t
- Construction Labor Cost
    - Central Nevada rates: average for all trades of $80/hr

Infrastructure

Infrastructure items consist of electrical supply, water supply and tailings facility. Estimates are made from quantities and costs from internal and published data (Table 21-6).

Electrical costs include a main substation, switch gear and power distribution to buildings and working areas. Included is an allowance for upgrading 50 km of 69 KV line. This allowance amounts to 72% of the total electrical costs.

Table 21-6: Infrastructure Capital Summary

Area	$ x 1000
Power	14,595
Water Supply	5,705
Tailings	2,597
Total Direct	22,897
Indirect	3,010
Total	**25,907**

Water supply costs allow for drilling four wells and an allowance for constructing a 14-inch pipeline over seven miles to the project, main and secondary water tanks, and installing electrical and piping distribution to the plant and buildings. The cost of obtaining water through rights acquisition, ongoing purchase, or other arrangements is excluded from the estimate.

The tailings facility costs allow for earthwork for initial embankment, diversion ditches, liner if needed, and monitoring wells.

21.1.2 Indirect Costs

Cost Parameters

Allowances are made on percentage basis for EPCM, freight, and sales tax.

EPCM costs are assumed at 8% of the Direct Costs and are intended to cover contractor mobilization, construction-related site studies and engineering, procurement, and construction travel housing and management exclusive of owner's costs.

Freight costs are applied at 3% of the direct costs of equipment. The allowance assumes most equipment is sourced in North America or FOB North America.

A local sales tax is applied at 6.85% of the direct costs on equipment. Exemptions to the sales tax may apply due to the operation being a new mine or a producer of critical metals

Owner's Costs

Allowances are made under Owners Costs for pre-production items including owner's team in project management, further testing and feasibility study, permitting and bonding, construction insurance, commissioning, recruitment and training, first-fills and spare capital items, and buy-down of royalty (Table 21-7).

Table 21-7: Owners Costs Summary

Area	$ x 1000
Project Management & Insurance	6,000
Feasibility Study	5,250
Start-up	6,700
Permitting & Bond	4,750
Royalty Buy-Down	2,000
Freight & Tax	291
Total	**24,992**

Costs for acquiring makeup water are not included.

Contingency

An allowance of 20% is made on the above direct and indirect costs to account for project changes incurred during the normal course of construction.

21.1.3 Other Capital

Working Capital

An allowance of two months of operating costs is added to cover delays and costs beyond those included in Owners Costs. Because of the long length of the mine schedule, working capital recovery is not included.

Sustaining Capital

Sustaining capital is included in the cash flow model and varies from $3.5 to $8.4 million/year.

Mine sustaining capital includes additional equipment as called for by the production schedule and mine equipment replacement estimated from 10% of the mine mobile equipment cost/year.

Sustaining capital includes the costs of maintaining a reclamation bond. Assumptions are 15% of the bond amount will occur in the first year of production and 3% will occur annually thereafter. Total cost of the bond based on comparable projects in Nevada is assumed at $15 million.

Replacements within the processing plant and administration are expensed as maintenance.

Allowance for expansion of the tailings facility is made on a per tonne basis in the processing plant operating costs.

21.2 Operating Costs

The project operating costs were developed from estimates of labor, operating and maintenance supplies, power, and fuel. The operation was sized to the nominal production rate of 15,000 tpd.

Responsibility for each area of the estimates is as follows:

- Mining GRE
- Processing CMS
- G&A CMS

The estimated operating costs total an average of $91.2 million/year, or $16.78/t. Distribution of the estimated costs is shown in Table 21-8.

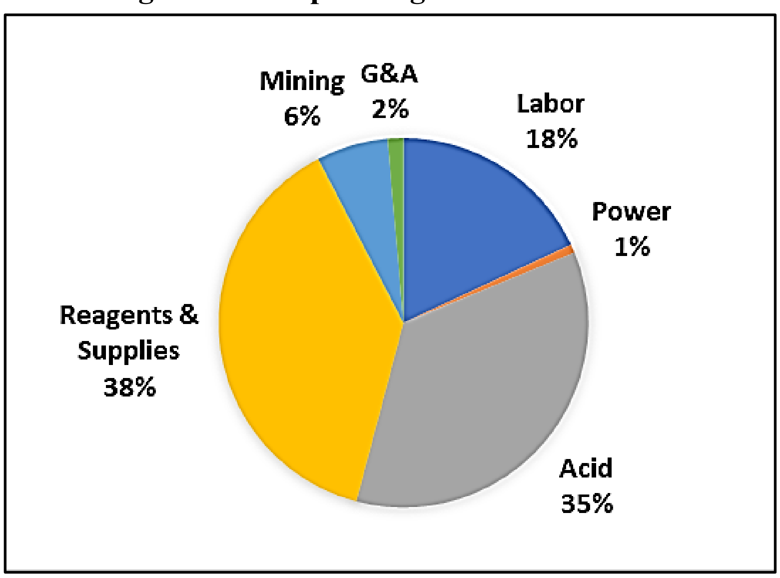
Table 21-8: Operating Cost Summary

Area	Avg Annual $ x 1000	Mill Feed $/t
Mining	9,932	1.83
Processing	77,735	14.30
G&A	3,550	0.65
Total	**91,218**	**16.78**

21.2.1 Key Components

The distribution of key operating components is shown in Figure 21-1.

Figure 21-1: Operating Cost Distribution



Labor

The projected total labor force required for the operation is estimated at 183 on-site personnel (Table 21-9).

Table 21-9: Labor Requirements

Area	Salary	Operations	Total
Mine	9	37	46
Plant	22	93	115
G&A	16	6	22
Total	**47**	**136**	**183**

Supervision and technical staff were allocated based on similar size and type of operations.

Operating and maintenance labor was allocated by operating area, pieces of equipment and number of crew shifts required

Labor rates by job function were based on typical current Nevada rates.

A burden factor of 40% was applied to all positions to allow for benefits, holidays, vacations, sick leave and payroll taxes.

Sulfuric Acid

Acid plant operations are a major component in the operating costs and account for approximately one third of the total operating cost.

Operating hours for the plant was assumed to be 24 hours/day, 7 days/week, for 52 weeks/year with an availability of 95%.

The total sulfur price delivered to mine site is estimated at a delivered cost of $145/t based on Q1 2020 cost of sulfur in dry form ($100/t) and freight ($45/t).

At full operating rate, the acid plant has capacity of 867,000 tonnes/year. Acid requirements of the operation at 15,000 tpd are estimated at 684,000 tonnes/year, or 79% utilization of acid plant capacity. The acid plant is equipped to operated continuously at rates under full utilization.

Power

The acid plant is equipped for power generation. It is assumed most of the power requirements of the operation will be met by the acid plant.

Generation capacity of the acid plant is 27.5 MW. The running power required by the operation is estimated at 21.6 MW and shown in Table 21-10.

There will be times during operation when the acid plant is not running and purchased power is required. It is estimated 93% of the total power requirement will be supplied by the sulfuric acid plant and 7% will be purchased. Cost of purchased power is estimated at $0.066/kWh based on connected and running loads.

No allowances are made in the operating cost estimates for potential power sales or offsets in utility costs that might occur from the operation placing surplus power onto the regional grid.

Table 21-10: Connected and Consumed Power Loads

Location	Connected HP	Demand KW	Running KW
Mine	2,085	1,437	770
Tailings	725	500	293
Leach & Filter	9,180	6,327	4,019
Li Recovery	17,675	12,025	11,860
Acid Plant	6,713	4,258	4,254
Water Supply	1,510	1,014	338
Buildings & Labor	275	191	106
Total	**38,163**	**25,752**	**21,641**

Operating and Maintenance Supplies and Fuel

Operating and maintenance supplies are estimated for each area based on estimate consumption and current bulk prices.

Diesel and gasoline will be delivered to on-site fuel storage for use primarily by mine equipment. Diesel is assumed at cost of $3.00/gallon.

21.2.2 Area Distribution

Mining

Mine operating costs include stripping, excavation, road maintenance, waste handing, conveying and stacking, and tailings placement (Table 21-11).

Table 21-11: Distribution Summary of Operating Costs

Area	$/yr x 1000	Mill Feed $/t
Mining		
Production Equipment	5,263	0.97
Support Equipment	425	0.08
Mine Labor	4,244	0.78
Mine Operating Costs	**9,932**	**1.83**
Processing		
Reagents & Consumables	67,122	12.35
Power	678	0.12
Plant Labor	9,935	1.83
Process Operating Costs	**77,735**	**14.30**
G&A		
Services and Supplies	1,266	0.23
G&A Labor	2,284	0.42
Total G&A Operating Costs	3,550	0.65
Total Operating Costs	**91,218**	**16.78**

Mining production equipment hours were estimated from the equipment productivity estimates, the scheduled tonnages of leach material and waste and the number of equipment required.

Mining support equipment hours were calculated from the number of pieces of equipment times the operating hours/day, assuming utilization of 90% and availability of 85%, times the operating days/year.

All power costs related to operation of feeder breaker and conveyors are accounted for in the plant processing power costs.

Processing Plant

The plant operating costs account for feed preparation, leaching, filtration, tailings handling, lithium recovery and acid plant operations, and are grouped by reagents and maintenance supplies, power and labor (Table 21-11).

Operating hours for plant functions were assumed to be 24 hours/day, 7 days/week, for 52 weeks/year with an availability of 92%.

Laboratory operating hours were set at 2 shifts/day, 8 hours/shift, and 260 operating days/year.

Feed preparation costs include allowances for crusher liners and screens.

Leaching costs consist mostly of sulfuric acid generated by the acid plant. This cost accounts for 48% of the reagents and supplies.

Other reagents and supplies include flocculent, filters, anti-scalent, and sodium carbonate, all estimated based on unit rates of consumption.

Electric power is the major consumable in lithium recovery, and accounts for 54% of the total power consumption of the operation. The savings in electric costs from the use of power from the acid plant are estimated at $9.4 million/year, or $1.72/tonne.

Lump sum estimates are made for maintenance supplies in each area, equipment and vehicle operation and laboratory supplies.

General & Administrative

General & Administrative (G&A) operating costs consist of site management and support and include lump sum allocations based on similar operations (Table 21-11).

Included are allocations for site insurance, offices supplies, legal costs, property maintenance, training and recruitment, subscriptions, travel, miscellaneous equipment rentals, vehicle operating and maintenance, site safety, environmental, and sanitary services. Corporate overhead costs are not included in the estimate.

State and local taxes are not included in the G&A costs but are included in the cash flow analysis.

22.0 ECONOMIC ANALYSIS

A discounted cash flow model was prepared using the information and estimates from the previous sections of this report. The model includes federal, state, and local taxes. Responsibilities for the model assumptions and economic analysis are as follows:

• Mine Production Schedule	GRE
• Mining Capital & Operating Costs	GRE
• Processing Recovery & Product Sales	CMS
• Processing Capital & Operating Costs	CMS
• G&A Costs	CMS
• Owner's Costs	CYP
• Tax Model Rates, Royalties	CYP

22.1 Model Assumptions

Capital costs of $493 million are distributed over a two-year period for pre-production construction with 39% of the capital assumed spent in Year -2 and 61% in Year -1.

Ramp-up to full production is assumed in the first two years of operation with 64% of the annual production rate assumed in Year 1 and 98% in Year 2. The time for permitting, feasibility and other studies prior to a construction decision is not included in the model. The costs for these studies, however, were included in Owner's Costs.

The nominal production rate at full operations is set at 15,000 tpd, or 5.475 million tonnes/year. At this rate, the project mine life is substantially long. For the cash flow model, the mine life is truncated at the end of pit Phase-8 or 40 years.

In the mine production schedule, lithium grades vary from 1,101-1,258 ppm Li. Recovery is estimated at 83% of the lithium tonnes processed and results in production ranging from 18,000 tpy of LCE in year-1 to between 25,000 and 30,000 tpy LCE though out years 2 through 30, averaging 27,400 tpy of LCE.

For the analysis, all material in the production schedule grading less than 900 ppm is regarded as waste to be placed in either low grade stockpile or waste dump. The mine schedule results in 223 million tonnes averaging 1,141 ppm Li. All lower grade material is assumed either stockpiled or placed in a waste dump.

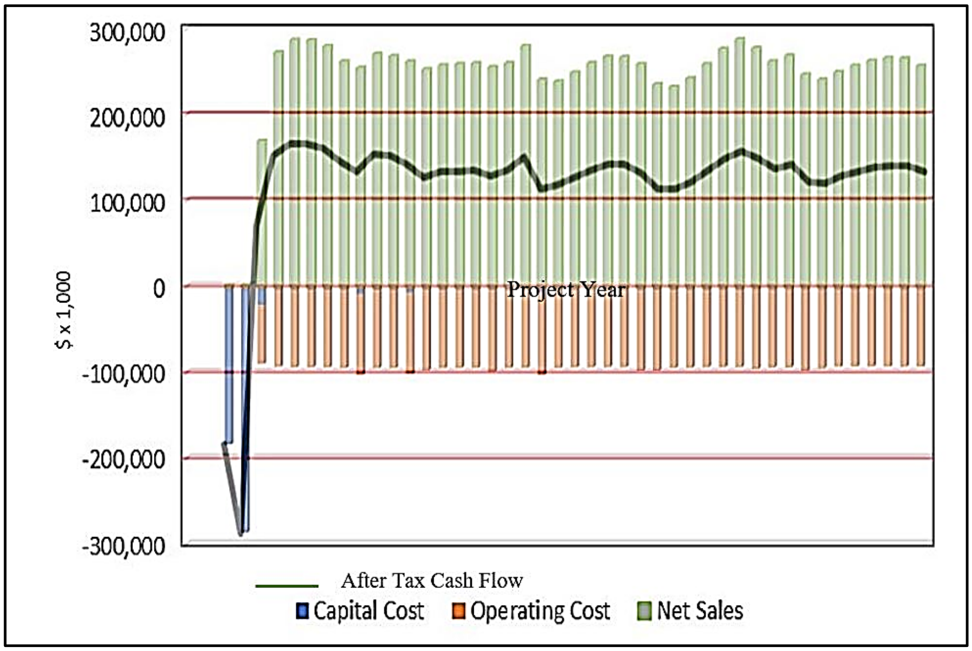
The base price for lithium product is $9,500/tonne of LCE based on the information in Section 19.0. All production is given in terms of lithium carbonate equivalent. Additional value is possible by producing lithium hydroxide but no premium on price is included. Any premium that does occur is assumed to offset lower prices in the first two years of operation when production of technical grade product may occur. The base price is assumed to be F.O.B. the project site.

The project has potential to generate surplus sulfuric acid and power. No credit is taken for power sales or offsets on purchased electricity.

No allowance was included to obtain a source of makeup water. Such costs are dependent on future conditions and agreements with other entities.

The royalty rate in the model is 1% NSR. Buy-down of the royalty to this rate is assumed in the Owner's Costs.

Figure 22-1: Cash Flow Model



The only revenue stream considered is from the sale of lithium products (Figure 22-1). No revenues are included for any other by-products. Such revenues remain to be determined.

The model is on a 100% equity basis with no debt leveraging.

An 8% discount rate is used to report Net Present Values.

Assumptions made for the tax calculations are:

- Federal Income Tax is applied at 21% after deductions for depletion, depreciation and state and local taxes.
    - Depreciation is calculated using basic straight-line method with five years on mobile equipment and 10 years on all other plant and facilities.
    - The depletion allowance is calculated from the lesser of 15% of net profits after operating costs or 50% of the net profits after depreciation.
- State and local taxes are applied at full rates. Certain deductions or exemptions may apply and remain to be determined.
    - Nevada Net Proceeds Tax is applied at 5% of net profits after depreciation and depletion.
    - An effective property tax rate of 1.05% is applied on the book value of capital.
    - A sales tax of 6.85% was applied to equipment capital costs based on the rate for Esmeralda County.

22.2 Results

Results for the project base case are:

- Average annual production of 27.4 million kg of LCE
- Cash operating cost of $3,329/tonne LCE
- A $1.052 billion after-tax NPV at an 8% discount rate
- A 25.8% after-tax IRR
- Payback period of 4.4 years
- Break-even price (0% IRR) of $4,025/t LCE

22.3 Sensitivity Analyses

Sensitivity of the project was evaluated to changes in lithium price, capital costs, and operating costs, these results are shown in Table 22-1, Figure 22-2 and Figure 22-3.

Table 22-1: Sensitivity Assessment

Variation	60%	100% Base Case	150%
Lithium Price $/t LCE	$5,700	$9,500	$14,250
NPV-8%	$130 million	$1.052 billion	$2.173 billion
IRR	10.5%	25.8%	41.1%
Capital Cost	$296 million	$493 million	$740 million
NPV-8%	$1.352 billion	$1.052 billion	$673 million
IRR	30.1%	25.8%	20.0%
Operating Cost	$1,997/t LCE	$3,329/t LCE	$4,993/t LCE
NPV-8%	$1.229 billion	$1.052 billion	$828 million
IRR	39.6%	25.8%	17.9%

Note: IRR (internal rate of return) and NPV (net present value) are both shown after-tax

Figure 22-2: Sensitivity in After-Tax NPV

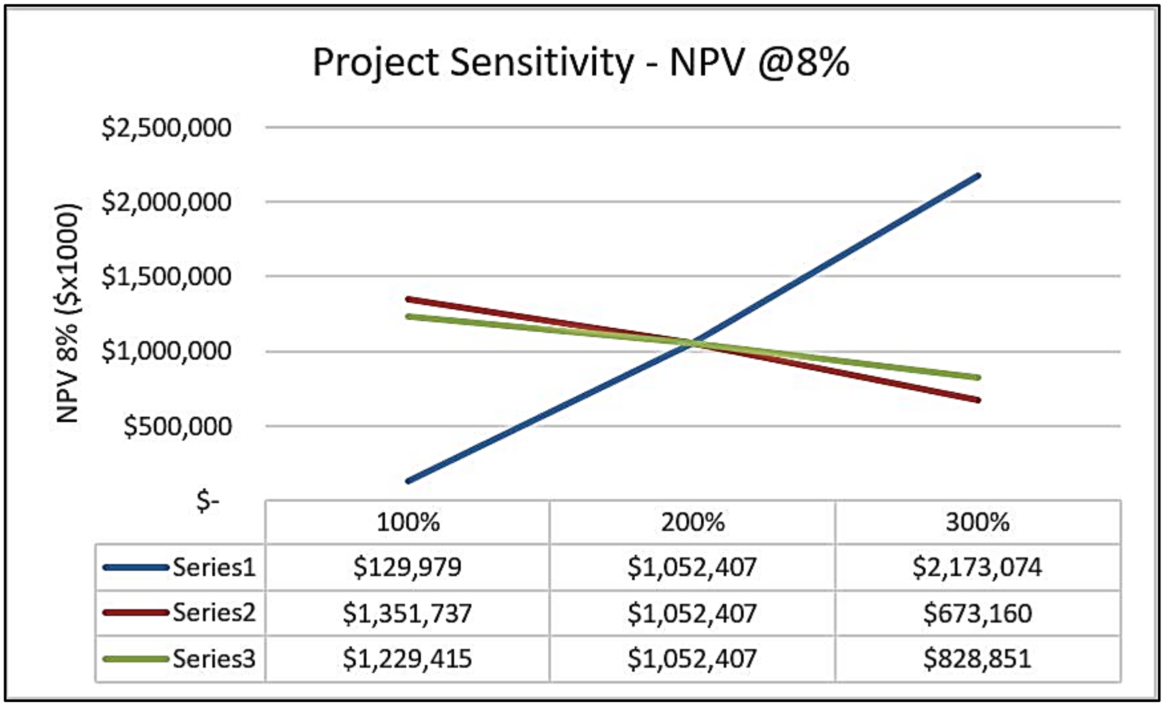
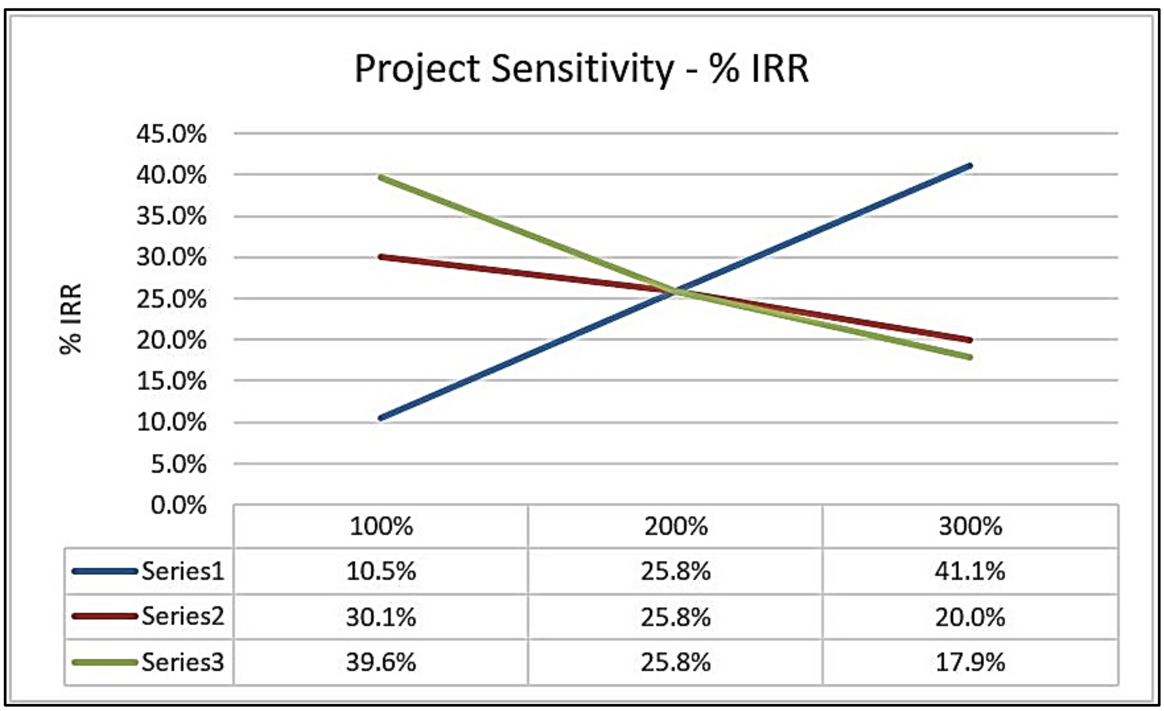


Figure 22-3: Sensitivity in After-Tax IRR



Lithium Price

The cash flow model is most sensitive to changes in lithium price. At 60% of the base case, or $5,700/t LCE, the after-tax NPV@ 8% is $130 million, and the after-tax IRR is 10.5%. At 150%

of the base case, or $14,250/t LCE, the after-tax NPV@ 8% is $2.17 billion, and the after-tax IRR is 41.1%.

Capital and Operating Costs

As expected, capital and operating costs affect the cash flow model either positively or negatively with changes from the base case.

The cash flow model is less sensitive to changes in capital cost. At 60% of the base case capital cost, or $296 million, the after-tax NPV is $1.352 billion, and the after-tax IRR is 30.1%. At 150% of the base case capital cost or $740 million, the after-tax NPV is $673 million, and an after-tax IRR is 20.0%. As the vendor quote for the acid plant is a considerable portion of the capital cost, a large reduction to the low end of the range is not expected.

The cash flow model is more sensitive to changes in operating cost. At 60% of base case operating costs, or $1,997/t LCE, the after-tax NPV is $1.229 billion and the after-tax IRR is 39.6%. An 150%, or $4,993/t LCE, the after-tax NPV is $828 million, and the after-tax IRR is 17.9%. The low end of the range, at an operating cost of $1,997/t LCE, is doubtful without significant by-product credit sales.

23.0 ADJACENT PROPERTIES

Seven companies hold lithium properties adjacent to the project.

23.1 Lithium in Sediments

Three public companies and two private entities have properties immediately adjacent to the project with mineral resources or exploration results for lithium-bearing clays.

- Noram Ventures Inc. holds property northwest of the project and reported in February 2019 an inferred resource of 145 million tonnes at 1,145 ppm Li (Peek Consulting Inc., 2019). In 2019-2020, Noram Ventures announced results from additional drilling including deepening of several drill holes. In February 2020, the Noram Ventures announced an indicated resource of 124 million tonnes at 1,136 ppm Li and an Inferred Resource of 77 million tonnes at 1,045 ppm Li.
- Enertopia Corporation holds property northwest of the project where five holes were drilled in 2019. In April 2020, the company announced an indicated resource (report pending) of 81.7 million tonnes at 1,121 ppm Li and an Inferred Resource of 18.1 million tonnes Li at 1,131 ppm Li.
- Spearmint Resources holds property southeast of the project and drilled three holes in 2018.
- Two private companies have properties east of the project and conducted exploration drilling in 2018-2020.

23.2 Lithium in Brine

Brine production in the Clayton Valley has been ongoing for over 50 years. Two public companies have properties immediately adjacent to the project with active production or mineral resources for lithium-bearing brines.

- Albemarle Corporation owns a commercial brine operation west and north of the project. It consists of the wells and evaporation ponds, and a lithium production plant in the town of Silver Peak.
- Pure Energy Minerals holds property west and north of the project and in 2017 published a NI 43-101 technical report and inferred resource for lithium brine. In 2019, Pure Energy Minerals announced an earn-in agreement on their property with Schlumberger Limited.

24.0 OTHER RELEVANT DATA & INFORMATION

Section 27.0 provides a list of documents that were consulted in support of the PFS. No further data or information is necessary in the opinion of the authors to make this report understandable and not misleading.

25.0 INTERPRETATION & CONCLUSIONS

The information within this report supports the presence of economic lithium mineralization and further work on the project.

The mineralization occurs within a large lithium-bearing clay deposit. The estimated Mineral Reserves for the project are large and capable of supporting a mine life of more than 40 years.

The project as outlined is based on a production rate of 15,000 tpd. This is identical to the PEA and is selected based on constraints in transportation and market considerations. The capital costs for the project are estimated at $493 million, of which the acid plant is a major component. A lower production rate could be considered as a means of capital cost reduction by deferring purchase of the acid plant and buying sulfuric acid instead; this alternative was not studied.

At the design rate of 15,000 tpd, the project has an estimated production rate of 27,400 tpy LCE. The operation is expected to produce lithium in the form of battery-grade lithium hydroxide. The estimated average operating cost of $16.78/t of material equates to $3,329/tonne LCE.

The analysis in this report uses a base price of $9,500/tonne LCE on lithium sales. The results are an after-tax NPV@8% of $1.052 billion and an after-tax IRR of 25.8%. The results are positive.

The project is exposed to risks typical of a mining project at a prefeasibility level of study.

- Recovery of lithium from the project was not proven at a commercial scale. Further testing in a pilot plant is needed to confirm all parts of the process flowsheet.
- Production is potentially limited by the availability and cost of sulfur and its transportation.
- The project is most sensitive to lithium market prices which are currently dependent on the demand for lithium batteries in electric vehicles and energy storage.
- A source of makeup water has not been secured. Options to obtain water through rights acquisition, purchase or other agreements should be pursued.
- Environmental permitting is subject to presence of flora, fauna or other conditions which are yet to be determined.

Further work is needed to evaluate the project as described within the recommendations of Section 26.0.

26.0 RECOMMENDATIONS

The recommendations to advance the project are:

- Processing—Additional test work is needed to confirm the process flowsheet and determine recoveries and reagent consumptions at the pilot stage. Critical information includes,
    - confirm steps and equipment in leaching and filtration
    - conduct further work to enhance solid-liquid separation and reduce acid consumption
    - determine lithium and acid losses in the processing plant, if any
    - optimize solution handling in the plant and determine if bleed streams or additional treatment are needed to recycle solutions
    - determine whether K, Mg, REEs, and other elements have commercial value
- Mining—Drilling or limited test mining is required to obtain material for metallurgical testing.
- Permitting—A field program is required to determine if any species of concern are present and to gather data to prepare a Plan of Operations.
- Infrastructure—Feasibility-level designs for the mine, plant and tailings storage areas can begin. Further determination of project power and water supply are needed.

26.1 Program Costs

Although the project uses off-the-shelf equipment and design, a pilot plant will be required to ensure all the processes work together as a single unit and to identify any scale-up or operational issues.

The pilot plant is projected to operate at approximately one tonne/day, and parts of the plant will be able to operate 24 hours/day for an entire month. The plant will be designed to ensure proper interaction of components. The estimated cost of the pilot plant study is $6.75 million and covers the capital, sample procurement, construction, and operation for six months, and includes a contingency allowance of 25%.

Table 26-1: Estimated Pilot Plant Costs

Area	$ x 1000
Pre-program studies	150
Sample procurement	500
Equipment	
Leaching	650
Lithium Recovery	2,600
Operating expenses	1,500
Contingency	1,350
Total Program	**6,750**

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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 Prefeasibility Study Technical Report of the Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of May 19, 2020 and an Issue date of May 19, 2020* (the "Technical Report").

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 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 prefeasibility, 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. My most recent visit was August 1, 2019.
8. I take responsibility for the information in Sections 13, 17-19, and 21-22 and co-responsibility for Sections 1-3 and 24-27, and the overall composition of the Technical Report.
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 PFS has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
11. As of the effective date of the PFS, to the best of my knowledge, information and belief, the PFS contains all scientific and technical information that is required to be disclosed to make the PFS not misleading.

Todd S. Fayram
"Todd Fayram"
Principal and Owner, Continental Metallurgical Services, LLC
Date of Signing: May 19, 2020

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 Prefeasibility Study Technical Report (PFS) of the Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of May 19, 2020 and an Issue date of May 19, 2020 (the "PFS"), 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 visited the property on March 19, 2019.
9. I take responsibility for the information in Sections 14-16 and co-responsibility for Sections 1-3, 21-22 and 24-27, and the overall composition of the Technical Report.
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 and Preliminary Economic Model of the Clayton Valley Project issued June 5, 2018 and September 4, 2018, respectively.
12. I have read National Instrument 43-101 and Form 43-101F1. The PFS has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
13. As of the effective date of the PFS, to the best of my knowledge, information and belief, the PFS contains all scientific and technical information that is required to be disclosed to make the PFS not misleading.

Terre A. Lane
"Terre A. Lane"
Principal Mining Mining Engineer
Date of Signing: May 19, 2020

CERTIFICATE OF QUALIFIED PERSON

I, Daniel W. Kalmbach, CPG, do hereby certify that:

1. I am an independent geologist at: 908 E 10Th Street, Davenport, Iowa 52803
2. I hold a Bachelor of Science in Geology (1999) from the College of Mining and Earth Resources at the University of Idaho.
3. I am a member in good standing and Certified Professional Geologist (CPG-11732) with the American Institute of Professional Geologists.
4. I have worked as a geologist since 1999; I have held staff and management positions with private and public companies in the field of geology, mining, exploration, development and environmental science, and as an independent geologist. I have supported or authored multiple technical reports on minerals properties, and am actively involved in resource estimation, geologic work in exploration, development and mining, and project management.
5. I have read the definition of "Qualified Person" (QP) 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.
6. I am a co-author of the technical report titled "*NI 43-101 Prefeasibility Study Technical Report of the Clayton Valley Lithium Project, Esmeralda County, Nevada, USA" with an effective date of May 19, 2020 and an Issue date of May 19, 2020* (the "Technical Report"). I take-responsibility for the information in Sections 4-12, 20 and 23 and co-responsibility for Sections 1-3 and 24-27, and the overall composition of the Technical Report.
7. I most recently visited the property that is the subject of this Technical Report on February 10-11, 2020. I have visited the property many times since August 2017 and have supervised the geologic work and property management as an independent consultant and QP to Cypress since April 2018.
8. As of the effective date of the Technical Report, I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the Issuer as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. 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.
11. I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Daniel W. Kalmbach, CPG
"Daniel Kalmbach"
Independent Geologist
Date of Signing: May 19, 2020