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LIST OF ACRONYMS

Acronym	Definition
#	Mesh, sieve size
%	Percent
%w or %w/w	Percent by weight
°C	Degree Celsius
AACE	Association for Advancement of Cost Engineering
AAL	American Assay Labs
AG/NP	Acid Generation / Neutralization Potential
ALS	ALS Minerals
amsl	Above mean sea level
As	Arsenic
BAQ	Bureau of Air Quality
bgs	Below ground surface
BLM	Bureau of Land Management
BM	Block Model.
BMRR	Bureau of Mining Regulation and Reclamation
$CaCO_3$	Calcite
$CaMg(CO_3)_2$	Dolomite
CERCLA	Comprehensive Environmental Response, Compensation, and Liability Act
CFR	Code of Federal Regulations
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
COE	US Army Corps of Engineers
CV	Coefficient of Variation.
CWA	Clean Water Act
DG	Discrete Gaussian Model for Change of Support (Internal Dilution check).
DOI	Department of the Interior
DR	Decision Record
DWG	model output file extension
EA	Environmental Assessment
EDA	Exploratory Data Analysis.
EIS	Environmental Impact Statement
elev	Elevation
EPCM	Engineering, Procurement and Construction Management
F	Flourine grade in %
Fe	Iron
FOB	Freight on board
FONSI	Finding of No Significant Impact
G&A	General & administration
gpl	Grams per liter
GPS	Global Positioning System
GSI	GeoSystems International, Inc.
GSLM	Great Salt Lake Minerals Corporation
GUs or UEs	Geologic Units or Estimation Domains.

Acronym	Definition
H:V	Horizontal:vertical
ha	Hectares (10,000 square meters)
HDPE	High-density polyethylene
HEVs	Hybrid electric vehicles
HPZ	Hot Pond Zone
HQ core	63.5 mm diameter core
HVAC	Heating, ventilation and cooling
I/S	Illite/smectite
ICP-AES	Inductively coupled plasma atomic emission spectrometry
ICP-MS	Inductively coupled plasma mass spectrometry
IRR	Internal Rate of Return
JBR	JBR Environmental Consultants, Inc.
K	Potassium grade in %
K_2O	Potassium oxide
K_2SO_4	Potassium sulfate
KCA	Kappes Cassidy & Associates
K-UTEC	K-UTEC AG Salt Technologies
KVC	Kings Valley Clay project
LCE	Lithium carbonate equivalent
LG	Lerchs-Grossman
Li	Lithium grade, in ppm or %, as stated.
Li_2CO_3	Lithium carbonate
LOI	Loss on ignition
LOM	Life of Mine
Mg	Magnesium
MgO	Magnesium oxide
Mo	Molybdenum
MOP	Muriate of potash
MOU	Memorandum of Understanding
MSHA	Mine Safety and Health Administration
Na	Sodium grade in %
Na_2SO_4	Sodium sulfate
NAC	Nevada Administrative Code
$NaCO_3$	Sodium carbonate
NaO_2	Sodium oxide
NDEP	Nevada Division of Environmental Protection
NDOW	Nevada Department of Wildlife
NEPA	National Environmental Policy Act
NI 43-101	National Instrument 43-101
NMC	Nevada Mining Claims
NN	Nearest neighbor (statistical)
NPDES	National Pollution Discharge Elimination System
NPV	Net Present Value
OK	Ordinary kriging

Acronym	Definition
OTCQX	United States Over-the-Counter marketplace
PAEE	Preliminary Assessment and Economic Evaluation
PCD Lens	Southernmost mining area, now referred to as Stage I Lens
PFS	Preliminary Feasibility Study, Tetra Tech, January 2012
PHEVs	Plug-in hybrid electric vehicles
PoO	Plan of Operations
PQ core	85 mm diameter core
QA-QC	Quality Assurance / Quality Control Programs (used not only for Laboratories).
QP	Qualified Person
RCRA	Resource Conservation and Recovery Act
Reserva	Reserva International, LLC
RoM	Run of Mine
rpm	Revolutions per minute
Sb	Antimony
SEDAR	System for Electronic Document Analysis and Retrieval
SHPO	State Historic Preservation Office
Si	Silica
SI	International System of Units
SMU	Selective Mining Unit.
SOP	Sulfate of potash
SQM	Sociedad Quimica y Minera de Chille SA
Stage I PFS	Stage I Prefeasibility Study
Stage II Resource Estimate	Stage II Lens Resource Estimate, GeoSystems International, Inc., May 15, 2010
SWPPP	Stormwater Pollution Prevention Plans
T&E	Threatened and Endangered
TR	Technical Report
TSF	Tailings storage facility
TSX	Toronto Stock Exchange
UM	Unpatented Mining Claim
URS	URS Energy and Construction, Inc.
US	United States
USBM	US Bureau of Mines
USD	United States Dollars
USGS	US Geological Survey
WEDC	Western Energy Development Corporation
WLC	Western Lithium Corporation
WPCP	Water Pollution Control Permit
WUC	Western Uranium Corporation
X,Y,Z	Cartesian Coordinates, also “Easting”, “Northing”, and “Elevation”.
XRD	X-ray diffraction

LIST OF ABBREVIATIONS

Abbreviation	Definition
µg/m^3	micrograms per cubic meter
µm	micrometers (microns)
ac-ft	acre-feet
cfm	cubic feet per minute
cfs	cubic feet per second
cm/s	centimeters per second
cy	cubic yards
d	Day
dmt	dry metric tonne
dst	dry short ton
fpm	feet per minute
ft	Feet
ft/d	feet per day
ft/hr	feet per hour
ft^2	square foot
ft^2/tpd	square feet per ton per day
ft^3	cubic foot
ft^3/d	cubic foot per day
ft^3/hr	cubic foot per hour
ft^3/t	cubic foot per ton
g	Gram
g/cc	grams per cubic centimeter
g/t	grams per tonne
gpd	gallons per day
gpm	gallons per minute
h; hr	Hour
hp	Horsepower
in	Inch
in/yr	inches per year
kg	Kilogram
kg/m^{2}hr	kilograms per square meter per hour
km	Kilometer
kV	Kilovolt
ktonnes	Kilo tonnes
kVA	kilovolt-ampere
kW	Kilowatt

Abbreviation	Definition
kWh	kilowatt hour
kWh/t	kilowatt hour per ton
Lb	Pound
lb/t	pounds per ton
LF	linear foot
M	Meter
m^2	square meter
m^3	cubic meter
mg/L	milligrams per liter
mg/m^3	milligrams per cubic meter
Mm	Millimeter
MMBtu	million British thermal units
Mph	miles per hour
MVA	megavolt-ampere
MW	Megawatt
Opt	ounces per ton
Oz	Ounce
Pcf	pounds per cubic foot
Ph	hydrogen ion concentration
PIW	pounds per inch of width
ppm	parts per million
Psf	pounds per square foot
Psi	pounds per square inch
Rpm	revolutions per minute
SG	specific gravity
st/h	short tons per hour
t	Tonne
tpd	Tonnes per day
tpy	Tonnes per year
t/m^3	tonnes per cubic meter
Toz	Troy ounce
µm	Micrometer
stpd	short tons per day
stph	short tons per hour
stpy	short tons per year
yd^2	square yard

ABBREVIATIONS OF THE PERIODIC TABLE

actinium = Ac	aluminum = Al	amercium = Am	antimony = Sb	argon = Ar
arsenic = As	astatine = At	barium = Ba	berkelium = Bk	beryllium = Be
bismuth = Bi	bohrium = Bh	boron = B	bromine = Br	cadmium = Cd
calcium = Ca	californium = Cf	carbon = C	cerium = Ce	cesium = Cs
chlorine = Cl	chromium = Cr	cobalt = Co	copper = Cu	curium = Cm
dubnium = Db	dysprosium = Dy	einsteinum = Es	erbium = Er	europium = Eu
fermium = Fm	fluorine = F	francium = Fr	gadolinium = Gd	gallium = Ga
germanium = Ge	gold = Au	hafnium = Hf	hahnium = Hn	helium = He
holmium = Ho	hydrogen = H	indium = In	iodine = I	iridium = Ir
iron = Fe	juliotium = Jl	krypton = Kr	lanthanum = La	lawrencium = Lr
lead = Pb	lithium = Li	lutetium = Lu	magnesium = Mg	manganese = Mn
meltnerium = Mt	mendelevium = Md	mercury = Hg	molybdenum = Mo	neodymium = Nd
neon = Ne	neptunium = Np	nickel = Ni	niobium = Nb	nitrogen = N
nobelium = No	osmium = Os	oxygen = O	palladium = Pd	phosphorus = P
platinum = Pt	plutonium = Pu	polonium = Po	potassium = K	prasodymium = Pr
promethium = Pm	protactinium = Pa	radium = Ra	radon = Rn	rhodium = Rh
rubidium = Rb	ruthenium = Ru	rutherfordium = Rf	rhenium = Re	samarium = Sm
scandium = Sc	selenium = Se	silicon = Si	silver = Ag	sodium = Na
strontium = Sr	sulphur = S	technetium = Tc	tantalum = Ta	tellurium = Te
terbium = Tb	thallium = Tl	thorium = Th	thulium = Tm	tin = Sn
titanium = Ti	tungsten = W	uranium = U	vanadium = V	xenon = Xe
ytterbium = Yb	yttrium = Y	zinc = Zn	zirconium = Zr	

UNITS OF MEASURE

All dollars are presented in US dollars unless otherwise noted. Common units of measure and conversion factors used in this report include:

Linear Measure

1 inch = 2.54 centimeters
1 foot = 0.3048 meter
1 yard = 0.9144 meter
1 mile = 1.6 kilometers

Area Measure

1 acre = 0.4047 hectare
1 square mile = 640 acres = 259 hectares

Capacity Measure (liquid)

1 US gallon = 4 quarts = 3.785 liter
1 cubic meter per hour = 4.403 US gpm

Weight

1 short ton	= 2000 pounds	= 0.907 tonne
1 pound	= 16 oz	= 0.454 kg
1 oz (troy)	= 31.103486 g	

Analytical Values

	Percent	Grams per metric tonne	Troy ounces per short ton
1%	1%	10,000	291.667
1 g/tonne	0.0001%	1.0	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
10 ppb			0.00029
100 ppm			2.971

1.0 SUMMARY

This report is an update to the Western Lithium Kings Valley property. It addresses the Stage I and Stage II deposits as well as provides additional updates to the Project. The report on the Stage I deposit includes a Preliminary Feasibility Study (PFS) identifying the progression of WLC's exploration of the property along with advancements in the metallurgical process. The report on the Stage II deposit includes a mineral resource estimates for lithium (Li), along with potassium (K), fluorine (f), and sodium (Na). The report amendment also addresses recent updates to the Project.

1.1 LOCATION, PROPERTY DESCRIPTION, AND OWNERSHIP

The Project is located in Humboldt County, Nevada, USA approximately 100 km north-northwest of Winnemucca; 70 km north from Winnemucca along US Highway 95 to Orovada and then 33 km west-northwest of Orovada on paved State Highway 293.

WLC owns the Lith, Beta, BPE, Neutron, Neutron Plus 1 and Neutron R claims that encompass five areas of lithium mineralization. These five areas are covered by approximately 2,500 federal unpatented claims over an area of approximately 15,233 hectares. The Project site has been the subject of various early stage exploration programs. However, there has been no mine development or production within the Project area boundaries, and therefore no mine workings or mill tailings are present on the property. There have been three areas of focus for the Kings Valley Project as follows: (i) the PFS covering the Stage I Lens (formerly PCD Lens), which is entirely included within 219 federal unpatented claims and covers an area of approximately 1,468 hectares (ha); (ii) the resource estimate covering the Stage II Lens (formerly the South Lens); and (iii) the organoclay drilling additive operations which contemplate extraction of small amounts of clay on the Stage I Lens and manufacturing of drilling additive products at Fernley, Nevada.

1.2 GEOLOGY AND MINERALIZATION

The Kings Valley Lithium Project (the Project) is situated within the south end of the McDermitt caldera located in Humboldt County, Nevada. It lies within a well-preserved Miocene collapse structure in northwestern Nevada near the southern Oregon border. The lithium deposits occur within sedimentary and volcano-sedimentary rocks in the moat of a resurgent caldera. The extent and nature of the host rocks is well documented and understood. Several lenses of mineralization have been identified in this region, though the focus of the Project is on the southern Stage I and Stage II Lenses.

The Stage I Lens is the southernmost and smallest of the mineralized lenses in the area. The lens is composed of an approximately 3 to 5 m thick layer of alluvium underlain by lithium-enriched interbedded claystones, ash-rich clays and ash layers up to 60 to 90 m thick in the northwest and southwest ends of the Project area. These claystone-ash layers thin in the middle of the proposed pit coinciding with faulting and a predominance of brown-black basalts. Interbedded basalts occur fairly shallowly in the northwest end of the pit and are found deeper in the southeast end. The lithium-rich beds with higher lithium concentrations (>4,000 ppm) are generally found deeper in the deposit (below 30 m). The base deposit varies across the Project area averaging between 68 to 90 m and is marked by an obvious transition to an oxidized silicified claystone and ash layer.

Exploration drilling by WLC has resulted in identifying clay-rich sequences with lithium concentrations exceeding those in previous studies. In the Project area clays and clay/ashes are disseminated

throughout the deposit, but the highest lithium concentrations are broadly found between 36 and 67 m below ground surface (bgs).

The Stage II Lens is located approximately 10 km NNW of the Stage I Lens, the southernmost lithium mineralized lens in the area. The Stage II mineralized beds comprises mainly of a dark green claystone, at times intercalated with arkose beds; in the NE region of the modeled area a fanglomerate body is present. Lithium-rich beds are generally 10 to 60 m thick in most areas, and defined as interesting Li grades as greater than 1,500 ppm. Alluvium cover is variable, ranging from no cover at all to a maximum of about 6m. At the Stage II Lens, the continuity of the mineralization has been confirmed by drilling at spacings at a nominal 200 m, with a smaller area drilled at about 40 m line spacings.

For both the Stage I and 2 Lenses, mineralization consists of layered beds of lithium-bearing clay-rich volcaniclastic sedimentary rocks. The main lithium-bearing mineral is reported to be a magnesium clay mineral that includes the Hectorite and Illite group. Hectorite, a rare Li-Mg clay mineral of the smectite group. The beds exhibit very good geological lateral continuity over kilometers, as shown by drill holes spaced on the order of 500 m. The thickness of mineralization varies from less than a meter to more than 90 m with typical intercepts of about 30 m. The extent of mineralization is well known.

A report entitled *Geochemical Characterization Program* (Tetra Tech, 2011), summarizes the x-ray diffraction (XRD) and mineralogical analysis, showing that the highest lithium concentrations are associated with the mineral illite, a non-expanding dioctahedral clay, which is the dominant clay mineral at deeper depths, about 36 to 60 m. The clay minerals at shallow depths (beginning at about 30 m bgs are identified as a smectite clay. At deeper depths, the dominant clay mineral is identified as illite. A thin transition layer composed of a mixed illite/smectite exists between the two dominant clay minerals. This mineralogical work is based on using six drillholes all within the proposed pit.

A summary discussion regarding mineralisation characteristics in relation to the hectorite clay for the organoclay additive operation can be found in Section 1.12.

1.3 DEPOSIT TYPE

To date, there are no analogous deposits in operation worldwide. The hectorite deposits at Hector, California have similar mineralogy, but the geological setting is significantly different.

These deposits are believed to have formed by hydrothermal alteration of layered volcaniclastic sedimentary rocks. What is not clear is whether the alteration was essentially syngenetic with deposition of the sedimentary rocks or whether the alteration is a post depositional event. Additional work is required to resolve the origin of these deposits.

1.4 MINERAL RESOURCE

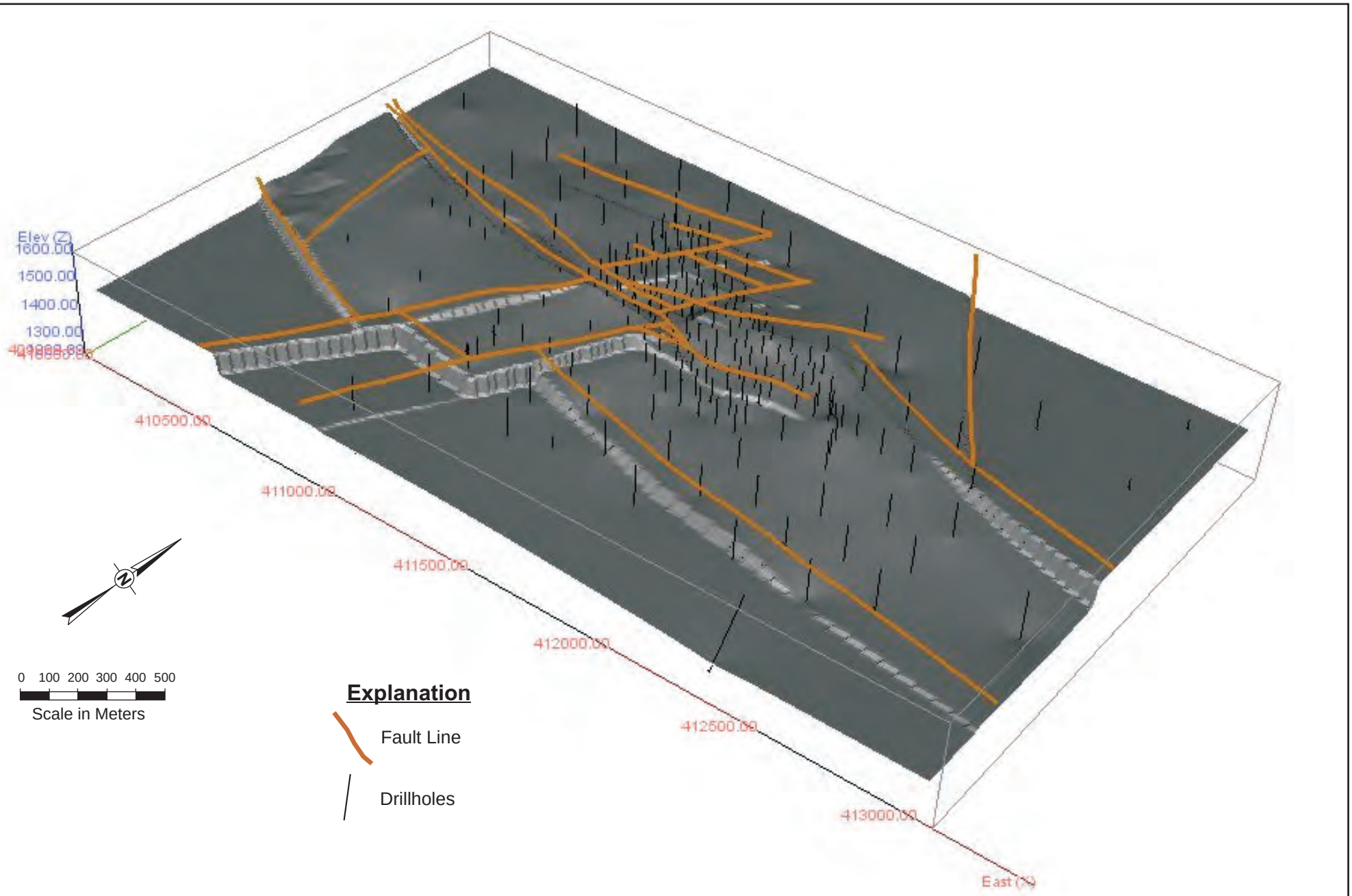
1.4.1 Stage I

WLC engaged Reserva International, LLC, to provide a block-model based mineral resource estimate for the Project. The current mineral resource estimate was developed with the WLC drillhole database available as of June 28, 2011. WLC has drilled and assayed 198 core holes, totaling 19,563 m. The resources are reported using a lithium cutoff of 3,200 ppm. Lithium carbonate is the primary product with potassium sulfate and sodium sulfate as byproducts.

Volcaniclastic moat sedimentary rocks that contain lithium-rich claystone control the Stage I Lens mineralization. Sectional interpretations (E-W and N-S) were generated from drill logs for alluvium, claystone (moat sediments), volcanics and basalt, a silicified unit, and bedrock. Two oxidation surfaces were also interpreted, one just below alluvium and another near the claystone/silicified interface. Additionally, a series of faults have been interpreted based on the drillhole data and incorporated into the geologic interpretation. The potentially economic mineralized estimation domain is the claystone. The alluvium and bedrock material have no lithium or potassium grades.

The mineral resources have been classified according to the CIM *Standards on Mineral Resources and Reserves: Definitions and Guidelines* (November 2004 and update November 2010). Resources are shown in Table 1.1. This table shows the total estimated resources for the Stage I Lens. Rounding errors occur in Table 1.1 and the contained metal does not account for mine or metallurgical recovery. To convert from Li% to Li ppm, multiply the percent value by 10,000. The conversion factor from Li% to LCE is 5.323. This factor is determined as follows:

$$\frac{\text{molecular weight } Li_2CO_3}{\text{molecular weight 2 x Li}} = \frac{(2 \times 6.941) + (12.011) + (3 \times 15.9994)}{(2 \times 6.941)} = 5.323$$



Created by:	Prepared for:	Location:		
rêserva INTERNATIONAL	*Western Lithium Corp.*	Humboldt County, NV		
	Project:	Project Number:	Date of Issue:	
	Kings Valley Lithium Project	114-311106	01/06/2012	

Figure 1.1
3D Perspective of Faulted Basement Surface

Table 1.1: Mineral Resource Statement for the Stage I Lens as of June 28, 2011

MEASURED							
Cutoff Li ppm	**KTonnes**	**Li%**	**Ktonnes LCE**	**K%**	**Ktonnes K**	**Na%**	**Ktonnes Na**
2000	50,753	0.312	843	3.27	1,660	1.13	574
2250	45,214	0.324	780	3.34	1,510	1.16	524
2500	38,858	0.338	699	3.42	1,329	1.20	466
2750	32,370	0.354	610	3.54	1,146	1.23	398
3000	24,765	0.374	493	3.71	919	1.27	315
3250	18,046	0.397	381	3.89	702	1.29	233
3500	13,100	0.420	293	4.00	524	1.29	169
3750	9,788	0.439	229	4.08	399	1.30	127
4000	7,226	0.457	176	4.14	299	1.32	95
4250	5,235	0.475	132	4.19	219	1.33	70
4500	3,475	0.494	91	4.26	148	1.35	47
4750	2,180	0.513	60	4.36	95	1.38	30
5000	1,373	0.529	39	4.44	61	1.38	19
INDICATED							
Cutoff Li ppm	**KTonnes**	**Li%**	**Ktonnes LCE**	**K%**	**Ktonnes K**	**Na%**	**Ktonnes Na**
2000	164,046	0.285	2,489	3.07	5,036	1.04	1706
2250	136,149	0.300	2,174	3.15	4,289	1.07	1457
2500	107,450	0.317	1,813	3.27	3,514	1.11	1193
2750	81,949	0.334	1,457	3.39	2,778	1.14	934
3000	58,600	0.352	1,098	3.51	2,057	1.16	680
3250	38,111	0.373	757	3.63	1,383	1.19	454
3500	24,177	0.395	508	3.73	902	1.20	290
3750	14,857	0.415	328	3.86	573	1.22	181
4000	8,796	0.435	204	3.94	347	1.26	111
4250	4,533	0.458	111	3.98	180	1.26	57
4500	2,170	0.480	55	4.06	88	1.28	28
4750	971	0.502	26	4.04	39	1.29	13
5000	478	0.517	13	4.04	19	1.33	6
INFERRED							
Cutoff Li ppm	**KTonnes**	**Li%**	**Ktonnes LCE**	**K%**	**Ktonnes K**	**Na%**	**Ktonnes Na**
2000	124,890	0.294	1,954	3.04	3,792	1.10	1374
2250	107,781	0.307	1,761	3.12	3,357	1.12	1207
2500	89,289	0.321	1,526	3.24	2,889	1.13	1009
2750	70,500	0.337	1,265	3.40	2,396	1.13	797
3000	57,348	0.348	1,062	3.43	1,969	1.15	660
3250	36,325	0.369	713	3.59	1,305	1.15	418
3500	24,234	0.386	498	3.74	907	1.15	279
3750	13,044	0.404	281	3.69	482	1.15	150
4000	7,457	0.416	165	3.64	272	1.15	86
4250	1,546	0.439	36	3.96	61	0.95	15
4500	182	0.470	5	3.22	6	1.06	2
4750	48	0.501	1	3.09	1	1.13	1
5000	19	0.524	1	3.51	1	1.04	0

Note: 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.

1.4.2 Stage II

GSI completed a review of the exploration work on the Stage II area of the Kings Valley project and has developed a lithium and potassium mineral resource estimates that conforms to Canadian National Instrument 43-101, *Standards of Disclosure for Mineral Projects* of the Canadian Securities Administrators.

The resource estimate was completed by Mr. Mario E. Rossi, CIM, AusIMM, SME, IAMG, and Principal Geostatistician of GSI. Mr. Rossi is an independent Qualified Person as defined by NI 43-101 by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience.

Table 1.2 presents the in-situ Li and K mineral resources for the Stage II Lens area, Kings Valley property, at a cut-off grade of 0.20% Li. The potassium (K) grade is considered a by-product of the Li resource. An average in-situ dry density of 1.96 t/m^3 for the mineralized volume was used as tonnage factor.

GSI is of the opinion that the exploration potential exist at the Stage II Lens to increase the current resource estimate.

Table 1.2: Kings Valley Li and K Mineral Resource Stage II Lens.

Kings Valley Li Mineral Resources*, 0.20% Li cutoff					
Category	**MTonnes**	**Li %**	**Contained KTonnes Li Carbonate Equivalent**	**K%**	**Contained Potassium Tonnes**
Indicated	95	0.27	1,365.3	3.66	3,477
Inferred	47	0.26	650.5	3.83	1,800

Rounding errors may exist.
Contained metal does not allow for mine or metallurgical recovery.
Tonnage factor used is 1.96 t/m^3.
Economic assumptions do not include any potassium credits.
Conversion factor from Li metal to Lithium Carbonate Equivalent (LCE) used is 5.323.
Economic assumptions for cutoff grade determination are:
- US$ 3.50 Li carbonate/lb; 60% metallurgical recovery; US$ 50/tonne processing; US$ 2.20/tonne mining

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues that may materially affect the resource estimate. Other relevant factors that may materially affect the resource estimate, including mining, metallurgical, and infrastructure are well understood according to the assumptions presented in the Stage II Resource Estimate.

1.5 Mineral Reserve Estimates

Two different mine plans were developed for the Stage I PFS and related reserve estimate on the Project to test the expandability of the project. The first plan, Case 1, delivers 689,850 tonnes of dry mill feed each year for 20 years. The second plan, Case 2, delivers 689,850 tonnes of dry mill feed for three years, increasing to 1,379,700 tonnes of dry mill feed for another 17 years. Measured and Indicated material from the resource model were converted to a Proven and Probable mineral reserve for both Case 1 and Case 2, by applying the pit design for each.

For Case 1, the outlines of each mining phase (1 through 5) were applied to the three dimensional grade model. Material that fell inside the phases was classified as ore if it was of a Measured or Indicated

resource class and it was 0.32% Li or higher. Material that fell inside the phases that was not classified as ore was classified as waste.

For Case 2, the outlines of each mining phase (1 through 8) were applied to the three dimensional grade model. Material that fell inside the phases was classified as ore if it was of a Measured or Indicated resource class and it was 0.32% Li or higher. Material that fell inside the phases that was not classified as ore was classified as waste.

Results are tabulated in Table 1.3 and Table 1.4 and are subdivided by logical mining phase.

Table 1.3: Mineral Reserve for Case 1

Proven Ore Tonnes (000's)	12,153		
Proven Average %Li	0.405	Proven Tonnes (000's) LCE	262
Proven Average %K	3.83	Proven Tonnes (000's) K	465
Proven Average %Na	1.46	Proven Tonnes (000's) Na	177
Probable Ore Tonnes (000's)	1,796		
Probable Average %Li	0.396	Probable Tonnes (000's) LCE	38
Probable Average %K	3.77	Probable Tonnes (000's) K	68
Probable Average %Na	1.45	Probable Tonnes (000's) Na	26
Proven+Probable Ore Tonnes (000's)	13,949		
Proven+Probable Average %Li	0.404	Proven+Probable Tonnes (000's) LCE	300
Proven+Probable Average %K	3.82	Proven+Probable Tonnes (000's) K	533
Proven+Probable Average %Na	1.46	Proven+Probable Tonnes (000's) Na	203

Summary of Dry Ore Tonnes for Proven + Probable
Cutoff = 0.320 %Li
95% Mine Recovery Factor Applied

Table 1.4: Mineral Reserve for Case 2

Proven Ore Tonnes (000's)	14,937		
Proven Average %Li	0.400	Proven Tonnes (000's) LCE	318
Proven Average %K	3.85	Proven Tonnes (000's) K	575
Proven Average %Na	1.37	Proven Tonnes (000's) Na	204
Probable Ore Tonnes (000's)	12,198		
Probable Average %Li	0.388	Probable Tonnes (000's) LCE	252
Probable Average %K	3.93	Probable Tonnes (000's) K	479
Probable Average %Na	1.36	Probable Tonnes (000's) Na	165
Proven+Probable Ore Tonnes (000's)	27,135		
Proven+Probable Average %Li	0.395	Proven+Probable Tonnes (000's) LCE	570
Proven+Probable Average %K	3.88	Proven+Probable Tonnes (000's) K	1054
Proven+Probable Average %Na	1.36	Proven+Probable Tonnes (000's) Na	369

1.6 MINERAL PROCESSING AND METALLURGICAL TESTING

Mineral processing and metallurgical testwork were completed by WLC on the Stage I deposit in connection with the Stage I PFS, Tetra Tech and K-UTEC AG Salt Technologies (K-UTEC) were provided the proprietary flowsheet and preliminary design criteria of the process as prepared by URS Energy and Construction, Inc. (URS) on behalf of WLC. Tetra Tech was charged with the review and verification of the ore preparation process and K-UTEC was focused on verifying the evaporative/crystallization recovery process.

The basic unit operations have not changed since the release of the *Kings Valley Project NI-43-101 Technical Report, Preliminary Assessment and Economic Evaluation* [PAEE]*, Humboldt County, Nevada*, URS Project No. 29916, effective date: December 31, 2009, issue date January 22, 2010 prepared by URS. It includes the following:

- Dry ore preparation process
  - ROM sizing
  - Reagent grinding
  - Reagent mixing and granulation
  - Calcination
- Wet recovery process
  - Leaching
  - Evaporation and glaserite crystallization
  - Glaserite decomposition and K_2SO_4 manufacture
  - Brine purification and Li_2CO_3 precipitation
  - Washing and finishing of Li_2CO_3
  - Dehydration and manufacture of water free Na_2SO_4

The Case 1 process has an annual target production of 13,000 tpy Li_2CO_3 (nominal). For Case 2, the first three years of annual production match Case 1. In Year 4 and beyond, the production is doubled to 26,000 tpy (nominal).

The overall recoveries are summarized below:

Lithium:	87.2%
Potassium:	77.7%
Sodium:	82.7%

1.7 MINING METHODS

1.7.1 Stage I

Tetra Tech performed an evaluation of the potential surface resource for the mineralized deposit. The surface mining work presented in this Stage I PFS is based on measured and indicated resources. Pertinent design parameters are:

- 41 degree maximum slope angle for pit walls
- 30 m wide ramps
- 10% maximum ramp grade
- 95% mining recovery with zero dilution assumed
- 3:1 H:V face angle for final waste dump faces
- 2.5:1 H:V face angle for working waste dump faces
- Design on single benching, 3 m benches

1.7.1.1 Case 1 Mine Plan

Potentially mineable pit shapes were identified using a Lerchs-Grossman (LG) analysis performed with GEMS™ Whittle pit optimization software and the Kings Valley mineral resource model. The optimization is based on preliminary estimates of operating cost, recoveries and lithium pricing. The optimization runs used only measured and indicated material for processing. All inferred material was considered as waste. Note that ore tonnes and grade will not match exactly with detailed schedule because variable cutoffs were used during first few years of operation.

An ultimate pit shell was designed based upon the selected Whittle pit shell that generated smoothed pit walls and removed adverse pit wall features such as noses. Then, phases within the ultimate pit were generated that would extract the ore in a logical sequence, and eventually enable waste to be backfilled into already mined areas in the pit. The phase designs include smoothed pit walls, haulage ramps, and pit access. A total of five phases were designed in all.

Based on the phase designs, an annual production schedule was generated that levels out total material movement per year and provides the required mill feed. The mill is designed for 2,100 tonnes per day (tpd) of dry feed, operating at 365 days per year at 90% availability. This works out to a total of 689,850 dry tonnes of ore from the mine each year. Mine plans were then produced for each year, based on the production schedule. Once mine plans were produced and annual waste production known, a series of five waste dump phases were designed. Waste Dump Phase 1 is completely external to the open pit, while the remaining dump phases are all backfill phases.

Lithium ore and waste rock will be extracted via an open-pit mine operation using end-dump haul trucks, hydraulic shovel, and one Wirtgen surface miner. Waste rock will be mined with a truck/shovel combination, while ore will be mined with the Wirtgen unit and direct loaded into haul trucks. Waste rock will be excavated via free-digging; it is assumed that no blasting will be needed.

The Case 1 LOM Production Summary is shown in Table 1.5.

Table 1.5: Case 1 LOM Production Summary

	Ore Mined (Dry Tonnes)	Waste Mined (Dry Tonnes)	Total Mined (Dry Tonnes)	Strip Ratio	Ore % Li	Ore % K	Ore % Na
Total	13,948,530	40,122,315	54,070,845	2.88	0.404	3.82	1.46

1.7.1.2 Case 2 Mine Plan

The same family of Whittle shells generated for Case 1 was used to identify a suitable ultimate shell for Case 2 which is an expanded production version of Case 1. For the first three years, the mill throughput is held constant at 689,850 tpy, the same as Case 1. However, in Year 4 and beyond, the mill throughput is doubled to 1,379,700 tpy. Waste movement holds steady at around 3.9 to 4.0 million tpy in Years 1 through 8, then increases to around 5.2 million tpy in Years 8 through 14, further increases to around 7 million tpy for Years 15 through 18 and then quickly drops off in Years 19 and 20. Note that ore tonnes and grade will not match exactly with detailed schedule because variable cutoffs were used during first few years of operation.

As in Case 1, lithium ore and waste rock will be extracted via an open-pit mine operation using end-dump haul trucks, hydraulic shovels and a Wirtgen surface miner.

The Case 2 LOM Production Summary is shown in Table 1.6.

Table 1.6: Case 2 LOM Production Summary

	Ore Mined (Dry Tonnes)	Waste Mined (Dry Tonnes)	Total Mined (Dry Tonnes)	Strip Ratio	Ore % Li	Ore % K	Ore % Na
Total	25,494,750	97,241,000	122,735,750	3.81	0.399	3.95	1.35

1.8 RECOVERY METHODS

Development of the process and design work has been completed by WLC and is the basis for this Stage I PFS. Tetra Tech and K-UTEC AG Salt Technologies (K-UTEC) were provided the proprietary flowsheet and preliminary design criteria of the process as prepared by URS Energy and Construction, Inc. (URS) on behalf of WLC. Tetra Tech was charged with the review and verification of the ore preparation process and K-UTEC was focused on verifying the evaporative/crystallization recovery process.

The basic unit operations have not changed since the release of the *Kings Valley Project NI-43-101 Technical Report, Preliminary Assessment and Economic Evaluation* [PAEE]*, Humboldt County, Nevada*, URS Project No. 29916, effective date: December 31, 2009, issue date January 22, 2010 prepared by URS. It includes the following:

- Dry ore preparation process
  - ROM sizing
  - Reagent grinding
  - Reagent mixing and granulation
  - Calcination

- Wet recovery process
  - Leaching
  - Evaporation and glaserite crystallization
  - Glaserite decomposition and K_2SO_4 manufacture
  - Brine purification and Li_2CO_3 precipitation
  - Washing and finishing of Li_2CO_3
  - Dehydration and manufacture of water free Na_2SO_4

The Case 1 process has an annual target production of 13,000 tpy Li_2CO_3 (nominal). For Case 2, the first three years of annual production match Case 1. In Year 4 and beyond, the production is doubled to 26,000 tpy (nominal).

The overall recoveries are summarized below:

Lithium: 87.2%

Potassium: 77.7%

Sodium: 82.7%

1.9 Market Studies and Contracts

1.9.1 Stage I Operation Market Parameters

The Project is capable of recovering and producing three distinct products as a result of the process design and the positive technical economic evaluation. These three products are generalized as lithium, potassium and sodium. WLC engaged an expert in the industrial minerals field to determine the market potential for each of these products. The projected market prices and values used in the economic evaluation are shown in Table 1.7.

Table 1.7: Market Prices – Cases 1 and 2

Description	Market Price	Unit	Model Value	Unit
Lithium Carbonate	$6,000	US$/t	$6,000	US$/t
Potassium Sulfate	$660	US$/t	$600	US$/t
Sodium Sulfate	$140	US$/t	$75	US$/t

1.10 CAPITAL AND OPERATING COSTS

1.10.1 Stage I

1.10.1.1 Capital Costs

Capital cost estimates include initial capital and sustaining capital for the life of the Project. Included in these estimates are equipment rebuilds and replacements, contingencies, owner's costs, EPCM costs and capitalized pre-production operating costs.

Case 1 Capital Costs

Estimated capital expenditures for the Case 1 life-of mine are US$262.7 million. This includes initial startup capital of US$237.1 million and sustaining capital of US$25.6 million. Table 1.8, Case 1 Capital Cost Estimate, details the life-of-mine, initial and sustaining capital estimate for Case 1.

Table 1.8: Case 1 Capital Cost Estimate (US$000s)

Description	Initial	Sustaining	LOM
Mine Mobile Equipment	$7,749	$10,497	$18,246
Mine Development & Tailings Handling	$17,485	$4,653	$22,139
Mine Closure	$0	$5,189	$5,189
Total Mine Capital	**$25,235**	**$20,340**	**$45,574**
Process Site Development	$1,791	$0	$1,791
Site Utilities	$10,588	$0	$10,588
Process Equipment	$115,128	$2,000	$117,128
Buildings and Structures	$6,348	$0	$6,348
Process Mobile Equipment	$2,354	$3,255	$5,609
Total Process Capital	**$136,209**	**$5,255**	**$141,464**
Indirect Costs	$35,961	$0	$35,961
Owner's Costs	$17,784	$0	$17,784
Contingency	$21,894	$0	$21,894
Total Capital	**$237,083**	**$25,595**	**$262,678**

Case 2 Capital Costs

Estimated capital expenditures for the Case 2 life-of mine are US$449.5 million. This includes initial startup capital of US$247.9 million and sustaining capital of US$201.6 million. Case 2 sustaining capital includes capitalized prestripping for the pit expansion as well as capital required for the plant expansion in Year 3 of the Project. Table 1.9, Case 2 Capital Cost Estimate, details the life-of-mine, initial and sustaining capital estimate for Case 2.

Table 1.9: Case 2 Capital Cost Estimate (US$000s)

Description	Initial	Sustaining	LOM
Mine Mobile Equipment	$7,818	$21,114	$28,932
Mine Development & Tailings Handling	$26,227	$18,424	$44,651
Mine Closure	$0	$8,519	$8,519
Total Mine Capital	**$34,046**	**$48,057**	**$82,103**
Process Site Development	$1,791	$144	$1,935
Site Utilities	$10,588	$6,461	$17,050
Process Equipment	$115,128	$105,556	$220,684
Buildings and Structures	$6,348	$1,701	$8,049
Process Mobile Equipment	$4,394	$3,510	$7,904
Total Process Capital	**$138,249**	**$117,372**	**$255,621**
Indirect Costs	$35,961	$22,458	$58,419
Owner's Costs	$17,784	$0	$17,784
Contingency	$21,894	$13,673	$35,567
Total Capital	**$247,935**	**$201,560**	**$449,495**

1.10.1.2 Operating Costs

Mine operating costs have been estimated for each year of the Project based upon the scheduled production requirements. Mine operating costs were developed from first principals and include labor, fuel and lubricants, materials, equipment and maintenance.

Process operating costs have been estimated for each year of the Project. Process operating costs are based upon material quotations and material balances which include necessary reagents, coal, grinding steel, electrical power, water, supplies, and equipment, labor and tailings facilities.

General and Administrative operating costs have been estimated for each year of the Project. General and Administrative costs include labor costs for salaried employees for site wide management, engineering, human resources and site security.

Case 1 Operating Costs

Case 1 operating costs average US$3,291/t lithium carbonate, US$99/t potassium sulfate and US$43/t sodium sulfate products over the life of the mine. Case 1 lithium carbonate cash costs net of byproducts averages US$1,397 per tonne of lithium carbonate. Table 1.10, Operating Cost Summary, details the operating costs for Cases 1 and 2.

Case 2 Operating Costs

Case 2 mine operating costs average US$3,011/t lithium carbonate, US$ 87/t potassium sulfate and US$36/t sodium sulfate products over the life of the mine. Case 2 lithium carbonate cash costs net of byproducts averages US$968 per tonne of lithium carbonate. Table 1.10, Operating Cost Summary, details the operating costs for Case 1 and Case 2.

Table 1.10: Operating Cost Summary

Description	US$ LOM Cost (000s)	US$/tonne Ore
Case 1		
Mining	$114,168	$8.18
Processing	$851,301	$61.03
G&A	$42,321	$3.03
Total	**$1,007,789**	**$72.25**
Case 2		
Mining	$197,356	$7.74
Processing	$1,418,319	$55.63
G&A	$41,798	$1.64
Total	**$1,657,473**	**$65.01**

1.11 ECONOMIC ANALYSIS

Two scenarios were evaluated. Case 1 was based on mining and processing ore at a design throughput rate of 2,100 tonnes per day (tpd). Case 2 considered starting at 2,100 tpd and then doubling ore production in Year 4.

Highlights:

- Case 1 pre-tax Net Present Value (NPV of US$262 million, with an IRR of 21.2% at a 8% discount rate.
- Case 1 post-tax NPV of US$175 million with an IRR of 17.6% at a 8% discount rate.
- Case 1 has a 21-year mine life, processing 13.9 million tonnes (dry basis) of ore at an average grade of 0.404% Li.
- Case 1 initial startup capital (including contingency of US$21.9 million), totals US$237.1 million. Sustaining capital of US$25.6 million is primarily composed of surface mine equipment, development, expansion of dry stack tailings, and surface water management and mine closure. A 10% contingency is included in the sustaining capital for all costs except surface mine equipment.
- Case 2 pre-tax NPV of US$551.8 million with an IRR of 24.4% at a 8% discount rate.
- Case 2 post-tax NPV of US$372.5 million with an IRR of 20.3% at a 8% discount rate.
- Case 2 has a 20-year mine life processing 25.5 million tonnes of ore at an average grade of 0.400% Li.
- Case 2 initial startup capital, including a contingency of (US$21.9 million), totals US$247.9 million. Sustaining capital of US$201.6 million is primarily composed of plant expansion, surface mine equipment, development, expansion of dry stack tailings and

surface water management, and mine closure. A 10% contingency is included with the sustaining capital for all costs except surface mine equipment.

- Commodity prices used in the economic evaluation are US$6,000 per tonne Li_2CO_3, US$600 per tonne K_2SO_4 and US$75 per tonne Na_2SO_4.
- The resource would be exploited using bulk surface mining methods for overburden/waste and selective mining methods for the mineralized material.
- A proprietary process would be applied.
- The mine would produce three products: lithium carbonate (Li_2CO_3), potassium sulfate (K_2SO_4) and sodium sulfate (Na_2SO_4).
- The target production rate for Case 1 is a nominal 13,000 tpy of lithium carbonate.
- The target production rate for Case 2 is a nominal 13,000 tpy lithium carbonate for Years 1 through 3 increasing to 26,000 tpy lithium carbonate in Year 4.
- Electricity would be provided from the existing grid.
- Water would be sourced from on-site wells.
- Subsequent to the Stage I PFS analysis, the Project has incurred royalty obligations that were not included in the Stage I economic estimate. The economic analysis of the Project has not been updated to account for these royalties.

Summary Project statistics are shown in Table 1.11 and Table 1.12 below:

Table 1.11: Summary Project Statistics – Case 1

Description	Statistic
Material Processed	13.9 million tonnes
Average Annual Production – Lithium Carbonate	13,000 tonnes
Lithium Carbonate Cash Operating Costs	US$3,291/t
Potassium Sulfate Cash Operating Costs	US$99/t
Sodium Sulfate Cash Operating Costs	US$43/t
Lithium Carbonate Cash Costs Net of Byproducts	US$1,397/t-LCE
Mine Life	21 yrs
Pre-production capital costs	US$237.1 million
Pre-Tax $NPV_{8\%}$	US$261.7 million
Post-Tax $NPV_{8\%}$	US$175.0 million
Pre-Tax IRR	21.2%
Post-Tax IRR	17.6%

Table 1.12: Summary Project Statistics – Case 2

Description	Statistic
Material Processed	25.5 million tonnes
Average Annual Production, Year 1 --3 – Lithium Carbonate	13,000 tonnes
Average Annual Production, After Year 4– Lithium Carbonate	26,000 tonnes
Lithium Carbonate Cash Operating Costs	US$3,011/t
Potassium Sulfate Cash Operating Costs	US$87/t
Sodium Sulfate Cash Operating Costs	US$36/t
Lithium Carbonate Cash Costs Net of By-Products	US$968/t-LCE
Mine Life	20 yrs
Pre-production capital costs	US$247.9 million
Pre-Tax NPV8%	US$551.8 million
Post-Tax NPV8%	US$372.5 million
Pre-Tax IRR	24.4%
Post-Tax IRR	20.3%

The cash flow analysis includes all mining, processing, and capital costs. Cash flow analyses were performed on both a pre-tax and post-tax basis. Applicable depletion and depreciation were calculated for determination of the Nevada State Minerals Tax and post-tax cash flow calculations. A federal income tax rate of 35% was used and working capital was set to 20% of yearly operating costs. A discount rate of 8% was used to determine the Project NPV.

The financial results of this Stage I PFS are presented in 100% owner's equity and constant 2011 US dollars for both cases. An 8% discount rate has been applied to the financial analysis. Sensitivity analyses were completed for both cases using varying lithium carbonate prices, capital cost estimates, operating cost estimates and process recoveries.

1.11.1 Case 1 Cash Flow Analysis

Case 1 entails mining 13.9 million dry tonnes of ore and 40.1 million tonnes of waste over a 21-year period. The Case 1 economic and processing parameters are as follows:

- 689,850 dry tonnes of ore mined and processed per year
- Lithium carbonate price – US$6,000 per tonne
- Potassium sulfate price – US$600 per tonne
- Sodium sulfate price – US$75 per tonne
- Lithium recovery – 87.2%
- Potassium recovery – 77.7%
- Sodium recovery – 82.7%
- Final product grade – greater than 99.0% for all products

Case 1 cash flow analysis results in a pre-tax project NPV of US$261.7 million and an IRR of 21.2%. The post-tax cash flow analysis results in a project NPV of US$175.0 million and an IRR of 17.6 %.

1.11.2 Case 2 Cash Flow Analysis

Case 2 entails mining 25.5 million dry tonnes of ore and 97.2 million tonnes of waste over a 20-year period. The Case 1 economic and processing parameters are as follows:

- 689,850 dry tonnes of ore mined and processed per year for Years 1 through 3
- 1,379,700 dry tonnes of ore mined and processed per year for Year 4 through the end of mine life
- Lithium carbonate price – US$6,000 per tonne
- Potassium sulfate price – US $600 per tonne
- Sodium sulfate price – US $75 per tonne
- Lithium recovery – 87.2 percent
- Potassium recovery – 77.7 percent
- Sodium recovery – 82.7 percent
- Final product grade – Greater than 99.0 percent for all products

Case 2 cash flow analysis results in a pre-tax project NPV of US$551.8 million and an IRR of 24.4 percent. The post-tax cash flow analysis results in a project NPV of US$372.5 million and an IRR of 20.3 percent. Table 1.13, Cash Flow Summary, summarizes the cash flow analysis for both Case 1 and Case 2.

Table 1.13: Cash Flow Summary

Description	Case 1 US$000s	Case 2 US$000s
Revenue		
Lithium Carbonate	$1,584,746	$2,864,092
Potassium Sulfate	$558,921	$1,055,473
Sodium Sulfate	$79,985	$140,159
Net Revenue	$2,223,651	$4,059,725
Operating Costs		
Mining	$114,168	$197,356
Processing	$851,301	$1,418,319
G&A	$42,321	$41,798
Total Operating Cost	$1,007,789	$1,657,473
Operating Margin	$1,215,862	$2,402,251
Initial Capital	$237,083	$247,935
LOM Sustaining Capital	$25,595	$201,560
Nevada State Minerals Tax	$48,007	$98,259
Income Tax	$233,635	$497,123
Pre-Tax Results		
Cash Flow	$905,177	$1,854,497
NPV @ 8% Discount Rate	$261,710	$551,829
IRR	21.2%	24.4%
Post-Tax Results		
Cash Flow	$671,542	$1,357,375
NPV @ 8% Discount Rate	$175,002	$372,476
IRR	17.6%	20.3%

1.11.3 Sensitivity Analysis

Sensitivity analyses were performed for the pre-tax project NPV at an 8% discount rate. Sensitivities were performed on lithium carbonate price, total operating costs, total capital expenditures and total process recovery. Values were adjusted in 5% increments ranging from negative 15% to positive 15%.

1.11.3.1 Case 1 Sensitivity Analysis

Case 1 sensitivities show that the Project is most sensitive to process recovery, followed by lithium carbonate pricing and operating costs, and least sensitive to capital expenditures. The sensitivities are graphed for Case 1 in Figure 1.2.

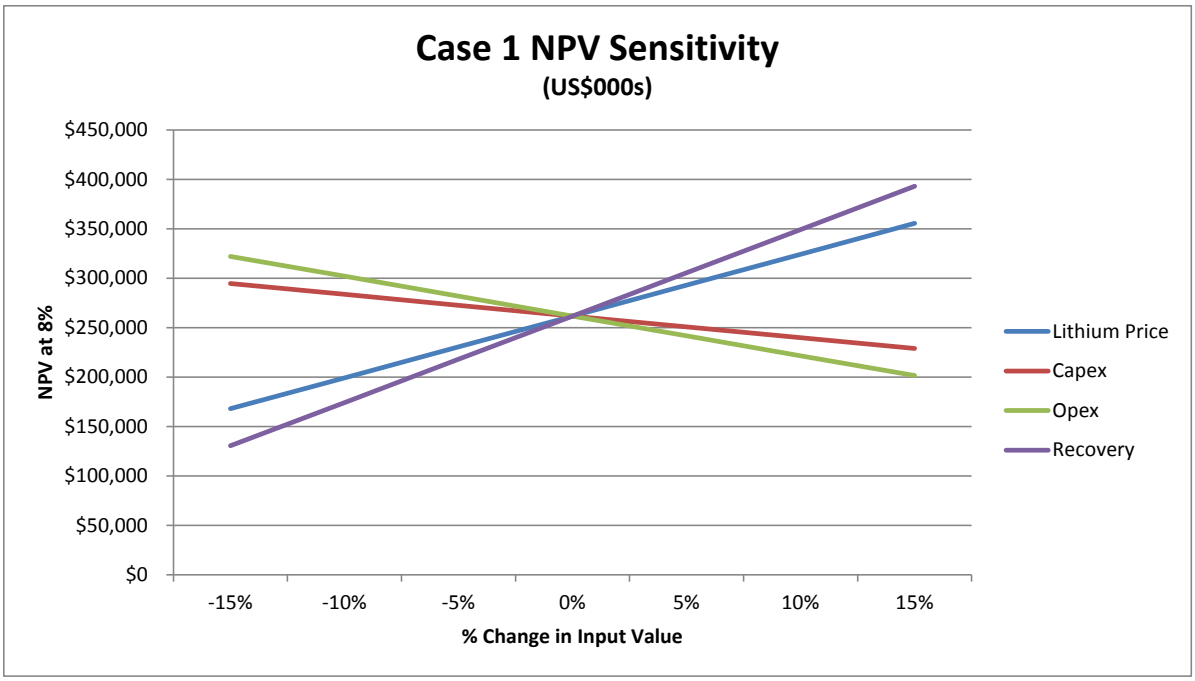


Figure 1.2: Case 1 NPV Sensitivity

1.11.3.2 Case 2 Sensitivity Analysis

Case 2 sensitivities show that the Project is most sensitive to the process recovery followed by lithium carbonate pricing and operating costs, and least sensitive to capital expenditures. The sensitivities are graphed for Case 2 in Figure 1.3.

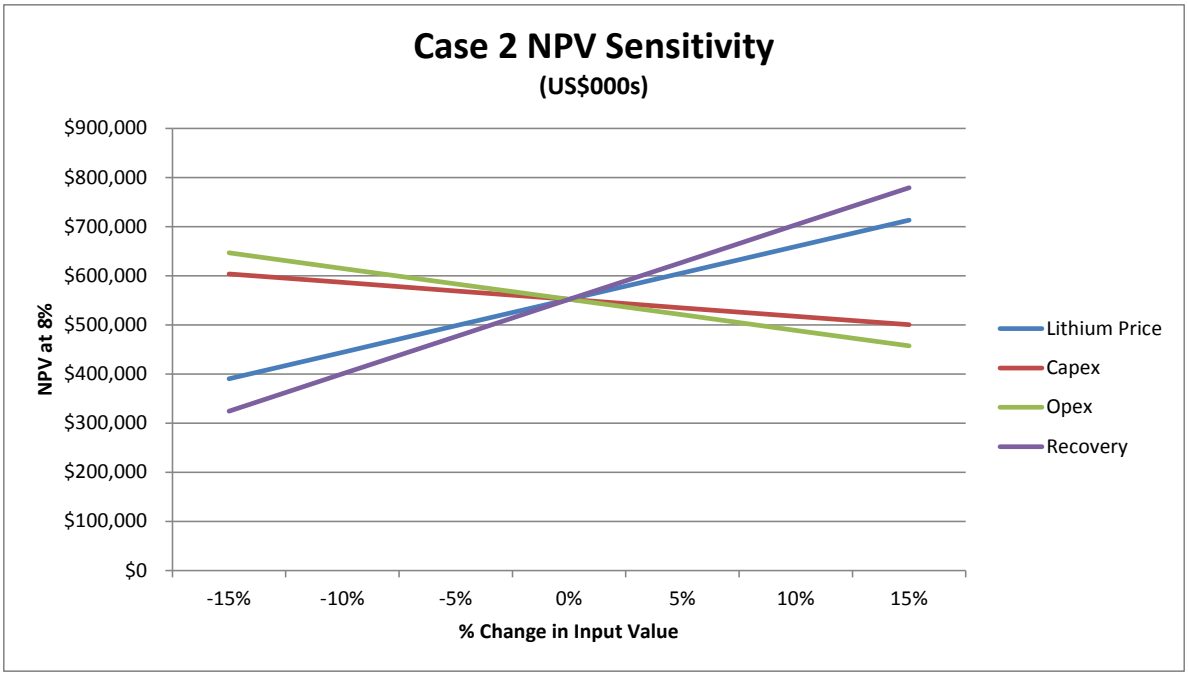


Figure 1.3: Case 2 NPV Sensitivity

1.12 Clay Operations

Western Lithium has established the Kings Valley Clay (KVC) Project internally to pursue the opportunity as a supplier of specialty drilling fluid referred to as organoclay for use in the oil and gas industry. An internal analysis assessed the areas that would have the highest potential for supplying clay for possible drilling fluids applications finding that the claystone deposit near the surface and in the oxidized zone as most suitable. Development is planned for both lithium and clay on that portion of the deposit referred to as Stage I.

The company is designing and building a 10,000 tons per year organoclay manufacturing plant in Fernley, Nevada to make the specialty clay with commissioning targeted for late summer 2014. The hectorite clay from the Kings Valley Property will be one of several raw materials (other clays) that will be utilized as feed. At present, this project does not address nor identify any mineral resource disclosure classification for the clay as described under NI 43-101 nor has Western Lithium demonstrated the economic viability of the organoclay business.

1.13 Status Of Exploration And Development

WLC has conducted only minor exploration activity on the property since the cut-off date for the Stage I resource estimate, consisting mainly of test work related to the organoclay drilling additive manufacturing process, as described in Sections 1.12 and 24.3. WLC has conducted development activity in respect of the organoclay drilling additive manufacturing facility in Fernley, and has also obtained permits to support future clay extraction operations on the property.

1.14 Conclusions and Recommendations

1.14.1 Stage I

WLC has a defined ore reserve that can be mined for lithium bearing ore and a process to recover lithium, potassium and sodium as marketable products.

The findings of this preliminary feasibility study provide a basis to advance the evaluation of the Project toward a feasibility level study.

The following recommendations for the Project should be considered:

- Continue the permitting process for the Project
- Build and operate a pilot facility to optimize the flowsheet, quantify calciner energy requirements and off-gas composition
- Confirm the required addition on anhydrite and dolomite per tonne of ore to achieve design product recovery for the next phase of the project.
- Verify the suitability of coal for direct-firing the calciner and quantify of the associated energy requirements in the next phase of testwork. Analyze the calciner off-gas from the coal-fired calcining tests to provide a basis for treatment requirements.

- Identification of corrosive gases from the calcination process in the testwork to date suggests future testwork to quantify the content and volume of gas flows for materials selection in calcining equipment and gas-handling ductwork.

- Confirm the revised flowsheet that does not incorporate the acidification step in the next phase of testwork.

Total estimated cost for these recommendations is between US$7,210,000-10,210,000.

1.14.2 Stage II

1.14.2.1 Drilling

At the present time, exploration at the Stage II (South Lens) deposit is sufficient to produce a mineral resource estimate that complies with CIM Definition Standards.

Additional drilling would be required to fill in some gaps in the drill pattern to upgrade resources from inferred to indicated, as well as indicated to measured resources. Additional infill drilling of the Stage II area is required for a detailed project definition and feasibility-level resource estimation. A nominal grid of 160 x 160 m should ensure that most resources are classified as indicated. A nominal drill hole spacing of 80 x 80 m would be required for measured resources.

1.14.2.2 QA-QC

Regarding further Stage II drilling programs, GSI agrees with AMEC's recommendation in previous Technical Reports that site specific standards be produced for the Kings Valley Li project from materials found in the project area. These standards should cover the expected grade ranges for Li, F, Na, and K.

GSI also recommends routine insertion of coarse duplicates (-10 #), in addition to the standards, pulp duplicates, and blank samples to monitor accuracy, precision and contamination. GSI also recommends that check samples be sent to a second laboratory.

1.14.2.3 Topographic Mapping

GSI recommends improving the topographic map currently available for the Stage II area. The level of detail is insufficient for (further) pre- and feasibility engineering work.

1.14.2.4 Density verification

GSI recommends that at least 50 additional dry density data be acquired from the moat sediments to better refine the in situ dry density estimate. There should also be an additional 10 density samples taken from alluvial material, as well as another 10 from the bedrock material, to improve the estimate of waste material to be moved. The density should always be determined using a wax-coat, immersion procedure; for broken up and friable samples, a soil or aggregate density method determination can be used.

1.14.2.5 Metallurgical Testing

GSI recommends that metallurgical samples from the Stage II (South Lens) be collected and tested according to the process flowsheet. This would allow refining metal recovery and process operating costs estimates. An assessment of the possible differences between the metallurgical responses of material from the Stage I (PCD Lens) and the Stage II (South Lens) should be made.

1.14.2.6 Mine Design, including Geotechnical assessments

At the present time, there are no mineral reserves at the Stage II (South Lens) deposit; mine optimization and design, including collecting and analyzing geotechnical data, would be required for future project advancement.

1.14.2.7 Proposed Budget

Total estimated cost for these recommendations is approximately US$1,030,000.

2.0 INTRODUCTION

This report is an update to the Western Lithium Kings Valley property. It addresses the Stage I and Stage II deposits as well as provides additional updates to the Project. The report on the Stage I deposit includes a Preliminary Feasibility Study (PFS) identifying the progression of WLC's exploration of the property along with advancements in the metallurgical process. The report on the Stage II deposit includes a mineral resource estimates for lithium (Li), along with potassium (K), fluorine (f), and sodium (Na). The report amendment also addresses recent updates to the Project.

2.1 TERMS OF REFERENCE

This technical report has been prepared in conjunction with the filing by WLC of a short form prospectus on Sedar.

The effective date for this Updated NI 43-101 Technical Report is April, 30, 2014.

- Preliminary Feasibility Study, Kings Valley Lithium Project, Humboldt County, Nevada
  - Effective Date of Resource Estimate: June 28, 2011
  - Effective Date of Reserve Estimate: December 14, 2011
- Stage II (South Lens) Resource Estimate, Kings Valley Project, Orvada, Humboldt County, Nevada
  - Stage II Report Effective Date: April 30, 2010

Tetra Tech and Reserva International, LLC (Reserva) were commissioned by WLC to author and prepare a NI 43-101 preliminary feasibility study (Stage I PFS) for the Project. This report provides information identifying the progression of WLC exploration of the Stage I deposit along with advancements in the metallurgical process. Costs, appropriate with the level of study, were both obtained and estimated to form the basis of the technical economic analysis presented.

GeoSystems International, Inc. was commissioned by WLC to prepare an NI 43-101 Resource Estimate for the Stage II Lens of the Project in Humboldt County, Nevada, USA.

Tetra Tech is not an associate or affiliate of WLC, or of any associated company. Tetra Tech's fee for this technical report is not dependent in whole or part on any prior or future engagement or understanding resulting from the conclusions of this report. The fee is in accordance with standard industry fees for work of this nature.

In preparing this report, Tetra Tech has relied on input from WLC and with information prepared by a number of qualified independent consulting groups as identified in Sections 2.4 and 2.7.

2.2 SCOPE OF WORK

2.2.1 Stage I

The scope of work conducted by Tetra Tech per request of WLC in connection with the Stage I PFS for the Project was to define the necessary elements from startup and construction of the project through final closure and reclamation at a ±25 percent level of accuracy. Tetra Tech, leading and working in coordination with WLC's other contractors, developed the critical design parameters for the project consisting of geology, mineral resources, mine plans, mineral reserves, process plant design, infrastructure, environmental requirements, site drainage, hydrogeology, permits, closure requirements, capital and operating cost estimates, resulting in the overall economic evaluation of the Project.

The Project includes two alternative approaches for project development: a Case 1 alternative that represents 2,100 tpd of plant throughput resulting in a normalized yearly production of 13,000 tonnes of lithium carbonate for the life of the Project and Case 2 that represents an increase in production to 4,200 tpd of lithium carbonate occurring in Year 4 of the Project through the life of the Project.

2.2.2 Stage II

GeoSystems International, Inc. (GSI) was commissioned by WLC to author and prepare an NI 43-101 compliant resource estimate and Technical Report for Stage II Lens, McDermitt caldera, Kings Valley project, located near Orovada in Humboldt County, Nevada, USA. GSI developed a mineral resource estimate for lithium (Li), along with potassium (K), fluorine (F), and sodium (Na), for an area within what is known as the South Pod, one of five identified areas within the McDermitt caldera deposit with significant lithium mineralization. This work was restricted to the Stage II Lens, only a portion of which has had sufficient drilling to produce an initial resource estimate. The estimate includes a total of 38 WLC diamond drill holes (2,150.87 m), as well as 13 older Chevron holes, of which 3 are diamond drill cores, and the remaining a mixture of reverse circulation (RC) and rotary drill holes (868.74 m).

2.3 SOURCES OF INFORMATION AND DATA

The Stage I PFS is an NI 43-101 preliminary feasibility study of the Project and is based on the June 28, 2011 resources report. Information and data used in preparation of this report are identified in Section 27 (References).

Significant input was obtained from WLC regarding exploration and other background information, and from other subject matter experts identified in Section 3.0.

2.4 QUALIFIED PERSONS

The following qualified persons were responsible for the relevant portions of this report:

Name	Title, Company	Responsible for Sections
Mario E. Rossi, FAusIMM	Principal Geostatistician GeoSystems International, Inc.	5.2, 6.2, 9.2, 10.2, 11.2, 12.2, 13.2, 14.2, 20.2, 25.2 and portions of 1, 2, 7, 18 and 26.
Edwin C. Lips, P.E.	Principal Mining Engineer, Tetra Tech	15, 16, and portions of 1, 2, 21, 22, 23, 25, 26 and 27
Timothy J. Carew, P. Geo	Principal, Reserva International, LLC	7, 8, 9, 10, 11, 12, 14 and portions of 1, 6, 15, 23, 25 and 26
D. Erik Spiller, MMSA QP	Principal Metallurgist Tetra Tech	3, 4, 5, 13, 17, 18, 19, 24 and portions of 1, 2, 6, 21, 22, 25, and 26.
Vicki J. Scharnhorst, P.E.	Principal Consultant Tetra Tech	20 and portions of 1, 6, 24, 25 and 26

2.5 UNITS OF MEASURE

The English system has been used throughout this report. All currency is in Q3 2011 US dollars ($) unless otherwise stated.

2.6 DETAILED PERSONAL INSPECTIONS

A site visit was conducted on the Project property on February 2, 2011 by D. Erik Spiller, process metallurgical QP, and on January 5, 2011 by Timothy J. Carew, resource estimate QP.

Mario E. Rossi, Principal Geostatistician, GSI, visited the Kings Valley site on February 24, 2010.

2.7 TECHNICAL CONTRIBUTORS

The disclosure provided in connection respect of the Stage I deposit included background and historical information from the PAEE (URS 2010) and updated by Tetra Tech using references identified in Section 27. In particular, information regarding metallurgy and processing, market research, mineral tenure, geological setting and the quality assurance program relied on experts (QP, or not) as described below. Tables, figures and charts received from other experts were modified to reflect Tetra Tech formatting. Additional reliance included the below listed parties.

The QPs relied on reports and statements provided by other experts who are not always Qualified Persons, as described below. Notwithstanding this reliance, none of the QPs responsible for the relevant sections of the report disclaims responsibility for their section of the report based on this information and the references cited below are for information purposes only.

2.7.1 Metallurgy and Process: Vladimir Zbranek, URS

Erik Spiller relied upon the metallurgical expertise of Mr. Zbranek, an independent consultant, formerly a general manager of Mineral Processing at URS Mining Group. He is a non-QP expert in mineral process

development who has extensive study experience in process design for projects similar to the Project. Mr. Zbranek's bachelor's and master's degrees are in Chemical Engineering. Mr. Zbranek contributed to the hydrometallurgical process development in Sections 13.1 and 17.1 of this report.

2.7.2 Marketing Research: Peter W. Harben, Inc., Las Cruces, New Mexico

Tetra Tech used confidential information produced by Peter Harben in Section 19 of the Stage I PFS. Mr. Harben has thirty years of experience in the field of industrial minerals and their international markets. Mr. Harben independently reviewed the lithium, potassium and sodium markets for WLC in support to this PFS and prepared a market report for each mineral in July and August 2011.

2.7.3 Potash Process Design: K-UTEC AG Salt Technologies, Germany

Dr. Bernd Schultheis and Dr. Reiner Zimmermann are principals of K-UTEC AG Salt Technologies, Sondershausen, Germany. K-UTEC is an internationally recognized and accredited laboratory following DIN EN ISO/IEC 17025. K-UTEC specializes in environmental technology, engineering and R&D for potash and other salt minerals.

Drs. Schultheis and Zimmermann were retained by WLC to verify the process and provide the design criteria parameters for a continuously operating facility.

Tetra Tech relied on the findings of Drs. Schultheis and Zimmermann for the hydro-process design and input. The work performed by K-UTEC is included in Sections 13.1 and 17.1 of this report.

2.7.4 Attorney: Thomas P. Erwin, Attorney at Law, of Erwin & Thompson LLP, Reno, Nevada

Mr. Erwin provided a copy of the property lease agreement prepared by him, which provides the basis for mineral tenure on the property by WLC.

He also provided a review of the validity of the Lith and Neutron claims (discussed in Section 4), which was provided in a letter to Pamela Klessig, President of WEDC and others (Kings Valley Lithium Project, Humboldt County, Nevada; Second Supplement to Mineral Status Report, July 31, 2007).

2.7.5 Barry W. Smee, Ph.D.

Dr. Barry Smee is a consulting geochemist and geologist based in British Columbia, Canada. He earned a B.Sc. in chemistry and geology from the University of Alberta, and a PhD. in geochemistry from the University of New Brunswick under the supervision of Dr. G. J. S. Govett. Dr. Smee has worked in academia, government and industry for 35 years, and has operated Smee & Associates since 1990. He has promoted the use of quality control methods in mineral exploration throughout his career, and specializes in the use of geochemical methods in covered terrain. Dr. Smee contributed to Section 11.1 of this report.

2.7.6 Stephen B. Castor, Ph.D.

Stephen B. Castor, Ph.D., is a consulting petrologist and Emeritus faculty member of the Nevada Bureau of Mines and Geology (NBMG), University of Nevada, Reno. Dr. Castor obtained a B.A. in Geology in 1965 from the University of California, Riverside; and earned graduate degrees in Geology from the University of Nevada, Reno – a Masters in 1971 and the doctorate in 1972. During his

NBMG career, Dr. Castor specialized in research on industrial minerals, but also worked on metal deposits and as a geologic mapper, producing more than 70 articles, maps, and reports. His best-known publication is probably Minerals of Nevada, a book jointly published by NBMG and the University of Nevada Press in 2003.

Following retirement from his position of 21 years as NBMG Research Geologist in 2009, Dr. Castor has been a part-time consultant for mineral exploration and mining firms. During his 42-year career, he also worked as a U.S. Peace Corps volunteer geologist for two years, in mineral exploration and mining geology for three different companies for 12 years, and for a government contracting firm for five years. The latter position included studies on uranium potential of the McDermitt caldera that began in 1980 and marked the beginning of his interest in the geology of this unique area – interest that has continued to the present, resulting in journal articles and consulting on mineral exploration projects. Dr. Castor contributed to Section 7.1 of this report.

2.7.7 Kings Valley Clay Project: Dennis Bryan, WLC

Erik Spiller relied upon Mr. Dennis Bryan, Senior Vice President for WLC for his expertise on the Kings Valley Project. Mr. Bryan prepared the "King Valley Clay Project Summary Report" dated May 5, 2014. Mr. Spiller has reviewed the report and has accepted it in its entirety. This report is presented in Section 24.

2.7.8 Western Energy Development Corporation

WEDC provided AMEC with documents relating to permits for the lithium exploration discussed in Section 4.3; AMEC reviewed the following documents:

1. *Notice of Intent to conduct lithium exploration on federal lands submitted to the Winnemucca Field Office of the U.S. Bureau of Land Management (BLM; May 29, 2007; signed by Pamela Klessig, President WEDC).*

2. *Letter response from the BLM, Winnemucca Field Office, dated 26 June 2007, signed by Dave Hayes, Assistant Field Manager, Nonrenewable Resources, acknowledging completion and acceptance of the Notice of Intent granting conditional permission to proceed.*

3. *Letter of acknowledgement of completion of the cultural survey from the BLM, Winnemucca Field Office, dated October 3, 2007, signed by Dave Hayes, Assistant Field Manager, Nonrenewable Resources, granting conditional permission to proceed.*

4. *Letter of acknowledgement of receipt of bond requirements from the State of Nevada Commission on Mineral Resources, Division of Minerals, dated 5 September 2007, addressed to WEDC, signed by Doug Dresner, Deputy Administrator, for Alan R. Coyner, Administrator, confirming that the required security bond has been paid and Receipt No. 02020 for the amount of the bond. The letter indicates that the information was forwarded to Scott R. Richey, BLM, Winnemucca Field Office.*

U.S. Department of the Interior, BLM, Surface Management Personal Bond Rider, Form OMB No. 1004-0194, which provides the details of Surface Management Bond No NVB-000493 on behalf of WEDC.

3.0 RELIANCE ON OTHER EXPERTS

3.1 MINERAL TENURE PROPERTY

In respect of the discussion regarding mineral tenure to the property set forth in Part 4, the qualified persons have relied entirety and without independent investigation, on the title opinion of Richard Harris, an attorney with Harris & Thompson, dated February 6, 2013, as updated and supplemented by the updated title opinion of Mr. Harris, dated April 28, 2014. The relevant sections of the report to which this applies are included in Section 4.0.

4.0 PROPERTY DESCRIPTION AND LOCATION

Disclosure regarding mineral tenure to the Kings Valley Property is based on the title opinion of Harris & Thompson, Nevada, counsel for WLC. This opinion is limited to Sections 4.1 through 4.4. Qualified Persons have relied on that opinion entirely, without independent investigation.

4.1 PROPERTY AREA AND LOCATION

The Kings Valley Property comprises an area of approximately 15,233 ha within Humboldt County, Nevada, that is approximately 100 km north-northwest of Winnemucca and 33 km west-northwest of Orovada, Nevada (centered on 40°42°27.24”N Latitude, 118°3°26.81”W). Situated in a remote section of northern Nevada, the King Valley Property consists primarily of sparsely populated ranch land within, and surrounded by, BLM lands on the northwest, western and southern sections of the McDermitt caldera. A small number of WLC’s claims are located, and registered, in Miller County, Oregon. Five areas of significant lithium mineralization have been identified on the Kings Valley Property – the Stage I Lens (PCD), Stage II Lens (South), Stage III Lens (South Central), Stage IV Lens (North Central), and Stage V Lens (North). To date WLC has focused its efforts on the Stage I Lens (formerly referred to as the PCD Lens), with limited work having been conducted at the Stage II Lens location.

The Stage I and Stage II Lens, being approximately 1,468 hectares, and 2,431 acres, respectively, are situated:

- with respect to the Stage I Lens, at the southern end of the McDermitt caldera in T44N, R35E within Sections 3,4, 5, 6, 7, 8, 9, 10, 15, 16, and 17 and on the USGS Thacker Pass at 7.5 min quadrangle at an approximate elevation of 1,500 m; and

- with respect to the Stage II Lens, in Township 45 North, Rangy 34 East, Sections 2 and 13; Township 46 North, Range 34 East, Sections 11 and 27.

4.2 MINERAL TENURE

The underlying title to the Kings Valley Property is held through a series of UMs. WLC holds its interests in the UMs indirectly through wholly-owned subsidiaries. UMs provide the holder with the rights to all locatable minerals on the relevant property, which includes lithium; however, this interest remains subject to the paramount title of the federal government who maintains fee simple title on the land.

The holder of an UM maintains a perpetual entitlement to the UM, provided it meets the obligations for UMs as required by the Mining Act. At this time, the principal obligation imposed on the holders of UMs is to pay an annual fee, which represents payment in lieu of assessment work required under the Mining Act. The annual fee of $140.00 per claim is payable to the BLM in addition to a fee of $10.50 per claim paid to the county recorder of the relevant county in Nevada (or, in a small number of cases, Oregon) where the UM is located. UM holders record annually an Affidavit and Notice of Intent to Hold. In 2010 the State of Nevada passed a bill requiring the payment of a fee ranging between US$85.00 and US$195.00 per UM, with such payments due in June 2011. In 2011 this bill was found to be unconstitutional and those claim fees have been refunded.

An UM does not, on its own, give the holder the right to extract and sell locatable minerals, as there are numerous other regulatory approvals and permits required as part of this process. In Nevada, such

approvals and permits include approval of a plan of operations by the BLM and environmental approvals. The Mining Act also does not explicitly authorize the owner of an UM to sell minerals that are leasable under the Mineral Lands Leasing Act of 1920, as amended (the "MLLA"), which includes potassium and sodium. The BLM is vested with a great deal of discretion in the management of the right to sell minerals governed by the MLLA, particularly where they represent a potential by-product to an economically viable mineral deposit governed by the Mining Act. WLC has initiated discussions with BLM to determine what, if any, contractual or regulatory approvals will be required to sell upgraded potassium sulfate and sodium sulfate as by-products to lithium production and to confirm WLC's priority to such approvals, but the matter has not been determined. Accordingly, at this time it is not possible to confirm the entitlement of WLC to sell potassium sulfate or sodium sulfate as a by-product to lithium production on the Kings Valley Property.

Table 4.1 sets out the UMs comprising the Kings Valley Property:

Table 4.1: Kings Valley Property, UMs

Claims	Owner
Moonlight # 1	Western Lithium Corporation, James R. Murdock
Uravada/U 23, 25-30, 46-56 (even), 61-69	Western Lithium Corporation
Moonlight No. 4	Western Lithium Corporation
Uravada/U 44	Western Lithium Corporation
Uravada/U 17-22, 24	Western Lithium Corporation and James V. LeBret
Alpha 1-56, 59-70, 72, 74-82, 84-85, 87, 90-164, 169-192, 196-210, 212, 214-230, 233-280	Western Lithium Corporation
Beta 1-51	Western Lithium Corporation
Lith 1-461, 463, 465, 467, 469, 471-473, 477, 479, 481, 484, 486, 488, 491-567, 586-677, 706-708, 713-732, 734-766, 785-1054	KV Project LLC
Neutron 31-45, 76-105, 166-190, 192, 194, 196-207, 209-225, 238-239, 347, 353-366, 379-402, 427-450, 475-498, 523-546, 555-574 and 579-627	Western Lithium Corporation
Proton 1-46	Western Lithium Corporation
Delta 1-14	Western Lithium Corporation
Rad 1-121	Western Lithium Corporation
Omega 1-124	Western Lithium Corporation
BPE 1-498	Western Lithium Corporation
Neutron Plus 1	Western Lithium Corporation
PCD Mill 1-18	Western Lithium Corporation
BPE 499-532	Western Lithium Corporation
Alpha 83R, 84-85, 86R, 88R-89R, 231R-232R	Western Lithium Corporation
Neutron 25R-30R, 70R-75R, 160R-165R, 195R, 208R, 240R-264R (even), 270R, 272R, 276R-284R (even), 285R-288R, 348R	Western Lithium Corporation
Neutron Plus 2	Western Lithium Corporation
MHC 1-99	Western Lithium Corporation

The foregoing UMs are located in Section 13, T. 44 N., R. 34 E.; Sections 1-24, T. 44 N., R. 35 E.; Sections 5-8 and 17-18, T. 44 N., R. 36 E.; Sections 1-5, 8-17, 20-25, 27-28 and 34-36, T. 45 N., R. 34 E.; Sections 24-26, 30 and 35-36, T. 45 N., R. 35 E.; Sections 3-8, 16, 21, 29-32, T. 45 N., R. 36 E.; Sections 1-3, 5, 8-19, 21-28 and 34-36 T. 46 N., R. 34 E.; Sections 33 and 34, T. 46 N., R. 36 E.; Sections 1-4, 9-11, 13-16, 20-23, 25-29 and 32-35, T. 47 N., R. 34 E.; Sections 21-22, 27-29 and 33, T. 47 N., R. 36 E.; and Section 34, T. 48 N., R. 34 E., MDM.

The UMs have been located in accordance with applicable state and federal law, and are valid and defensible, with the exception of a single claim that is not material to any of: (i) the Stage I deposit; (ii) the Stage II deposit; or (iii) the King Valley Property as a whole. Counsel has identified that WLC shares title to Moonlight #1 and Uravada 17-22 and 24.

4.3 Nature and Extent of Interest and Title

Western Lithium Corporation ("Western Lithium Nevada") is a Nevada corporation that is currently a wholly-owned subsidiary of the Canadian-based WLC. Pursuant to an agreement signed on December 20, 2007 between Western Energy Development Corporation ("WEDC"), a subsidiary of Western Uranium Corporation, and WLC (which was then also a subsidiary of Western Uranium Corporation), WEDC leased to WLC the Lith and Neutron claims for the purpose of lithium exploration and exploitation (the "Lease").

The agreement granted Western Lithium Nevada exclusive rights to explore, develop, and mine or otherwise produce any and all lithium deposits discovered on the claims, subject to royalty payments. The leased area, at that time, included the entirety of the Stage I and Stage II Lens and included 1,378 claims that covered over 11,000 ha. Lithium deposits to be exploited included, but were not limited to, deposits of amblygonite, eucryptite, hectorite, lepidolite, petalite, spodumene and bentonitic clays. Rights to all other mineral types, including base and precious metals, uranium, vanadium and uranium- or vanadium-bearing materials or ores were expressly reserved by WEDC.

The term of that lease agreement was 30 years and granted to Western Lithium Nevada the exclusive right to purchase the unpatented mining claims comprising a designated discovery, subject to the royalty and other rights to be reserved by WEDC and subject to Western Lithium Nevada obligations under the deed to be executed and delivered by WEDC on the closing of the option (PAEE URS 2010). In July, 2008, WLC ceased to be wholly-owned by Western Uranium Corporation and became an independent publicly traded company.

Effective February 4, 2011, Western Uranium Corporation, WEDC, WLC and Western Lithium Nevada entered into an agreement for the purchase by Western Lithium Nevada from WEDC of the royalties and titles for the Kings Valley Property.

In March 2011, the parties completed the transaction for the sale by WEDC to Western Lithium Nevada of the royalties and titles constituting all of the Kings Valley Property. As a result of this transaction, the existing lease and royalty arrangements between the two companies on the Kings Valley Property, including the net smelter returns and net profits royalties on any lithium project that the company developed, were terminated. WLC holds, indirectly, control and full ownership of the Kings Valley Property mining claims and leases, excluding a gold exploration target (on the Albisu property) and a 20% royalty granted by WEDC to Cameco Global Exploration II Ltd solely in respect of uranium (the "Uranium Royalty").

The UMs which are subject to the Lease authorize WLC to develop and mine minerals which are subject to location under the Mining Law of 1872, as amended. The Mining Law does not explicitly authorize the owner of an unpatented mining claim to sell minerals, which are minerals that are leasable under the Mineral Lands Leasing Act of 1920, as amended. WLC has initiated discussions with the BLM to determine the legislative and regulatory authority for its processing and sale of the potassium sulphate which will be a byproduct of WLC's production of lithium which is a mineral locatable under the Mining Law of 1872. The matter has not been formally determined.

4.4 ROYALTIES, RIGHTS AND PAYMENTS

In addition to the Uranium Royalty and those national, state and local rates identified in 4.3, the Kings Valley Property is subject to the following royalties, back-in rights, payments, and encumbrances:

- Orion Mine Finance Fund I (f.n.a RK Mine Finance (Master) Fund II L.P.) has a gross revenue royalty on the King Valley Property in the amount of 8% of gross revenue until aggregate royalty payments equal $22 million have been paid, at with time the royal will be reduced to 4.0% of the gross revenue on all minerals mined, produced or otherwise recovered. WLC can at any time elect to reduce the rate of the royalty to 1.75% on notice and payment of $20 million to Orion;

- R. McClintick and K. McClintick, Trustees of the Rick and Keel, E. McClintick Living Trust, hold a net smelter return royalty of 1.5% on production from the U 17-20 UMs. The royalty provides for an advance payment of $1,785 annually, which payments will be credited against royalty payments otherwise payable; and

- Uravada LLC holds a net smelter return royalty of 3.0% on production from the U 21-22, 24, 44, Uravada 23, 25-30, 46-56 (even), and 61-69 UMs. The royalty provides for an advance payment of $50,000 annually, which payments will be credited against royalty payments otherwise payable.

4.5 ENVIRONMENTAL LIABILITIES

Other than an accrued decommissioning obligation at the Stage I Lens of approximately $170,000, no environmental liabilities are known to exist at the Kings Valley Property.

4.6 PERMITTING

The Stage I Lens has been the subject of various early stage exploration programs. However, there has been no historical mine development or production within the project area boundaries, and therefore no mine workings or mill tailings are present at the Stage I Lens. WLC holds a current exploration permit in good standing, and has done so in each year since 2006, and also holds all necessary federal and state permits and approvals to conduct exploration activities at the Stage I Lens.

A PoO was submitted to the BLM and the NDEP in May 2008 for an extensive drilling and trenching exploration program to further delineate the resources of the Stage I Lens. That action included preparation of an environmental assessment. A revision to the PoO was filed in November 2009 and approved in January 2010.

March 7, 2014, the BLM issued a DR and FONSI for WLC's Kings Valley Mine Final EA. The DR allows the development and operation of the Company's proposed 110-acre Kings Valley hectorite clay project, see Section 24.

On May 1, 2014, and in accordance with 43 CFR 3809 Surface Management Regulations, the BLM authorized WLC to proceed with clay extraction according to its Final Plan of Operations and Reclamation Plan. On April 29, 2014, the NDEP BMRR concurrently approved WLC's Final Plan of Operations and issued a Final Reclamation Permit. On April 28, 2014, the MSHA assigned the Kings Valley Clay Mine operation an MSHA Mine Identification Number. On April 10, 2014, the Humboldt County Regional Planning Commission unanimously approved WLC's Conditional Use Permit, allowing the development of an open pit to extract clay.

WLC has documentation supporting its rights to the lithium mineralization within the Stage II Lens and that all appropriate permits for exploration have been obtained.

4.7 Other Factors or Risks

Tetra Tech is not aware of any other significant factors or risks that may affect access, title, or the right or ability to perform work on the property.

4.8 Conclusions

Based on information provided to, or researched and reviewed by Tetra Tech as a part of this preliminary feasibility study, WLC is approved by the BLM and the NDEP BMRR to conduct mineral exploration activities at the Kings Valley Lithium Property in accordance with PoO No. N85255.

As discussed above, WLC has initiated the process to obtain all necessary federal, state and local regulatory agency permits and approvals for the proposed the Project. This process, which includes the completion of environmental and natural resource baseline studies, is described in detail in Section 20 of this report.

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

5.1 STAGE I

Some information in this section was excerpted from the PAEE (URS 2010), and has been standardized for this report. All subsections have been updated.

5.1.1 Accessibility

Access to the project is via the paved US Highway 95, traveling approximately 70 km north from Winnemucca, Nevada, to Orovada and then heading west-northwest for 33 km on paved State Highway 293 toward Thacker Pass to the Project site entrance. On-site access is via several gravel and dirt roads established during the exploration phase.

5.1.2 Climate

Northern Nevada has a high desert climate with cold winters (average minimum -3°C in January) and hot summers (up to 35-40°C). Snow is expected from October to May, although it often melts quickly. Nearby mining operations operate continuously through the winter.

5.1.3 Local Resources

Because of the large-scale mining industry in the Winnemucca area, local resources include all of the amenities required for large-scale mining. The area is about 50 km north of the now depleted Sleeper gold mine and 100 km northwest of Twin Creeks, Turquoise Ridge and Getchell gold mines. Additionally, there are several other gold and copper mines in the area which could provide potential access to an experienced workforce and adequate support for mining operations. Most of the workforce may have to be sourced in Winnemucca because of the sparse population in the project area.

5.1.4 Infrastructure

The existing roads are maintained by the Nevada Department of Transportation and are in good repair. The roads are all-season roads but may be closed for short periods due to extreme weather during the winter season. The nearest railroad access is in Winnemucca.

Adequate electrical power is available and currently there is a 115 kV power line that passes through the property. Work during this preliminary feasibility study has determined this power source to be suitable for meeting the project electrical demands for Case 1. Water rights have been attained by WLC. An independent groundwater study is currently under preparation but the results of this study have not been completed for use within the Stage I PFS. It is anticipated from preliminary results from this effort that water would be sourced at the Project site; however, this remains to be confirmed.

There is sufficient space within the current Stage I Lens site to accommodate the processing plant and mine support facilities, overburden placement site, anticipated dry TSF, the limited wet TSF, water diversions, and containments (see Figure 7.1, Site Map).

5.1.5 Physiography

The Stage I Lens is oriented roughly east-west in the southern portion of the McDermitt caldera. The Project site sits at the southern end of Montana Mountains, with its western border at the divide between the Kings River Valley and Quinn River Valley. Vegetation consists of low lying sagebrush and grasslands. Elevation at the Project site is approximately 1,500 m above sea level.

5.2 STAGE II

Part of the information in this section was excerpted from the URS report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project, Humboldt County, Nevada, USA,* effective date December 31, 2009.

Access to the project is via paved road (U.S. Highway 95), 100 km north from Winnemucca to Orovada and then approximately 40 km WNW on paved State Highway 293 to the project area. On-site access is via numerous gravel and dirt roads, including the last 10 km from State Highway 293.

Northern Nevada has a high desert climate with cold winters and hot summers. Snow is expected from October to May, although it may melt quickly. Nearby mining operations operate continuously through the winter. Elevations in the area are 1,524 to 2,150 m. Vegetation consists of sagebrush and grasslands at all elevations.

Because of the large-scale gold mining industry in the Winnemucca area, local resources include all of the amenities required for large-scale mining. The area is about 50 km north of the now depleted Sleeper gold mine and 100 km northwest of Twin Creeks, Turquoise Ridge, and Getchell gold mines. Several other gold and copper mines are in the area, providing an experienced work force and adequate support for mining operations. Most of the workers may have to be sourced in Winnemucca because of the sparse population in the project area.

Adequate electrical power is available, but power lines may need to be added and/or upgraded to provide power to the project site. Currently, there is a 115 kV power line that passes through the Kings Valley Project site. Water is available in the region and can be sourced from either the Kings River Valley, west of the project, or Queens River Valley located east of the project. No surface water is present at the site. Roads are all-season and in generally good repair, but may be closed for short periods due to extreme weather in the winter. The nearest railroad access is located in Winnemucca.

Local access to the Stage II Lens is currently an unimproved gravel road that at times during the winter months can be impassable. This access will require upgrading.

6.0 HISTORY

6.1 STAGE I

6.1.1 Exploration History

In 1975 Chevron USA began an exploration program for uranium in the sediments of the McDermitt caldera which encompassed the entire caldera. Early in Chevron's program, the USGS, which had been investigating lithium sources, alerted Chevron to the presence of anomalous concentrations of lithium associated with the caldera. In 1978 and 1979 Chevron continued its focus on uranium, but added lithium to its assays, began a clay analysis program, and obtained samples for engineering work.

Results confirmed the high lithium concentrations contained in clays. From 1980 to 1987, Chevron began a drilling program which focused on lithium targets and conducted extensive metallurgical testing of the clays to determine the viability of lithium extraction. In 1985, Chevron undertook a resource estimate for a 0.25% Li cutoff. This estimate is not considered CIM compliant.

6.1.2 Ownership History

Chevron leased many of the claims that comprised the lithium project to the J. M. Huber Corporation (Huber) in 1986. In 1991, Chevron USA sold its interest in the claims to Cyprus Gold Exploration Corporation. In 1992, Huber terminated the lease. It appears that Cyprus Gold Exploration Corporation allowed the claims to lapse and provided much of the exploration data to Jim LaBret, one of the claim owners from which they had leased claims. Western Energy Development Corporation (WEDC), a Nevada corporation, leased LaBret's claims in 2005, at which time LaBret provided WEDC access to the Chevron data and to core and other samples that were available.

Pursuant to an agreement signed on December 20, 2007 between Western Energy Development Corporation ("WEDC"), a subsidiary of Western Uranium Corporation, and WLC (which was then a subsidiary of Western Uranium Corporation), WEDC leased to WLC the Lith and Neutron claims for the purpose of lithium exploration and exploitation. This agreement granted WLC exclusive rights to explore, develop, and mine or otherwise produce any and all lithium deposits discovered on the claims, subject to royalty payments. The leased area, at this time, included the entirety of the Stage I Lens and included 1,378 claims that covered over 11,000 ha.

Lithium deposits to be exploited included, but were not limited to, deposits of amblygonite, eucryptite, hectorite, lepidolite, petalite, spodumene and bentonitic clays. Rights to all other mineral types, including base and precious metals, uranium, vanadium and uranium- or vanadium-bearing materials or ores were expressly reserved by WEDC. The term of that lease agreement was 30 years and granted to WLC the exclusive right to purchase the unpatented mining claims comprising a designated discovery, subject to the royalty and other rights to be reserved by WEDC and subject to WLC's obligations under the deed to be executed and delivered by WEDC on the closing of the option (PAEE, URS 2010).

In July, 2008, Western Lithium USA Corporation ceased to be wholly owned by Western Uranium Corporation and became an independent publicly traded company.

Effective February 4, 2011, Western Uranium Corporation, WEDC, Western Lithium Corporation and WLC entered into an agreement for the purchase by Western Lithium Corporation from WEDC of the royalties and titles for the Kings Valley mineral property.

In March 2011, the parties completed the transaction for the sale by WEDC to Western Lithium Corporation of the royalties and titles constituting all of the Kings Valley mineral property. As a result of this transaction, the existing lease and royalty arrangements between the two companies on the Kings Valley property, including the Net Smelter Returns and Net Profits Royalties on any lithium project that the company developed, were terminated. Western Lithium Corporation, wholly owned by WLC, holds control and full ownership of the Kings Valley property mining claims and leases, excluding a gold exploration target (on the Albisu property) and a 20% royalty granted by WEDC to Cameco Global Exploration II Ltd solely in respect of uranium.

6.1.3 Environmental Work History

WLC initiated an environmental and natural resources baseline study program for the Project in 2007. The purpose of this on-going program is to collect data for various resources that will support the completion of the overall permitting and approval process, and the associated environmental documentation program. Section 20.1 identifies the individual studies that have been completed to date; ongoing and planned studies; and regulatory agency consultation activities.

6.1.4 Metallurgical Testwork History

From late 2007 to May 2008, WLC completed drilling 37 core holes (4,861.98 m) and eight reverse circulation (RC) holes (1,798.63 m) in the Stage I Lens. WLC performed a mineralogical study of the mineralization in 2008 (Hudson, 2008). That study identified the presence of hectorite as well as bitumen in the deposit. WLC determined the density of six samples from the Stage I Lens.

Metallurgical testwork was performed on samples from the Project deposit by investigators, including Chevron Research Company, United States Bureau of Mines (USBM), and Hazen Research (Hazen). Bench-scale laboratory studies were performed by Hazen to examine lithium extraction from a composite ore sample with about 0.35% lithium prepared from drill core samples supplied by WLC. Most of the work completed was derived from the work conducted by Chevron. Hazen found that wet attritioning is effective in liberating the clay fines from the ore. Hazen also developed methods to separate lithium-rich fines from gangue and to reduce carbonate concentration to less than 0.5% by weight. Acid plugging, curing, and water leaching were then used to extract lithium from the clay fines, with an acid dose of approximately 473.21 kg per tonne of ore required for 95% extraction. The results obtained by Hazen were consistent with those found by Chevron.

In late 2008, metallurgical testwork was performed by Kappes Cassiday & Associates (KCA) on samples from the Kings Valley lithium deposit by various researchers, based in principle on the USBM report, *Lithium and Its Recovery from Low-grade Nevada Clays* (USBM Bulletin 691, 1988). USBM investigated several extraction techniques and pursued two options, gypsum-limestone roast and chlorinating roast. Selecting samples from two different areas of the McDermitt caldera, USBM performed successful preliminary tests of gypsum-limestone roasting, which led to extensive batch-roast studies and further to larger-scale roasting in a rotary furnace. Chlorination was also tested, but proved to be economically unfeasible.

The KCA testwork was based on the prior work completed by USBM. Samples, obtained in August 2008 and October 2008 from recent drilling campaigns included both oxidized ore (comprising an estimated 15 to 20% of the deposit) and unoxidized ore (comprising 80 to 85% of the deposit). An analysis of possible ore upgrading techniques resulted in selecting wet attritioning followed by wet screening. Subsequent calcining of the ore with a mixture of limestone and gypsum was performed, followed by water leaching to recover soluble lithium and other alkali sulfates. Results confirmed that the USBM process for recovery of lithium from this ore was technically viable.

6.2 STAGE II

The information in this section was excerpted from the URS report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project, Humboldt County, Nevada, USA,* effective date December 31, 2009.

Chevron began exploration for uranium in the McDermitt caldera area in 1975. In September 1977, personnel of the U.S. Geological Survey alerted Chevron to the presence of anomalous concentrations of lithium associated with volcaniclastic moat sediments within the McDermitt caldera. Drill cuttings from rotary percussion drill holes drilled in September on radiometric anomalies were also analyzed for lithium. One hundred and forty feet of hole MJB-7-4 averaged 0.278% lithium and 85 feet of MJB-7-5 averaged 0.236% lithium. These results confirmed the presence of significant lithium hosted by a massive, green claystone within the moat sediment section.

In 1978 and 1979, Chevron's activities focused on the uranium resource hosted by volcanic rocks in the caldera and drilling to evaluate selected tailings disposal sites. The drilling focused on evaluating the thickness of the clays, obtaining samples of the clay for engineering analysis, and further investigating the lithium resource potential. Results were encouraging with respect to the concentration and consistency of the lithium contained in the clays.

In 1980 and 1981, Chevron drilled four core holes to obtain uncontaminated and undisturbed samples to more effectively determine lithium grades and coincident volcaniclastic stratigraphy. A portion of the core was sent to Chevron Research Company (CRC) for evaluating an economic process for extracting lithium from the clays.

From 1982 through 1987, Chevron drilled an additional 223 holes on lithium targets and conducted extensive metallurgical testing of the hectorite ores to determine amenability of the ores to extraction of lithium. Approximately 370 1–3 meter-deep auger holes were drilled in 1982; one sample from each hole was analyzed for lithium assays.

In 1985, Chevron produced polygonal estimates of the Li resources on their properties at Kings Valley (Table 6.1). A cutoff grade of 0.25% lithium, minimum thickness of 1.52 m, and a minimum 9.0 ft% Grade x Thickness (GT) were used for the estimate. The tonnage factor used was 1.8 g/cm^3. ***This estimate is not compliant with CIM Definition Standards for Mineral Resources and Mineral Reserves (2005) as required by NI 43-101 and is included here for historical purposes only.*** Because of the lack of documentation of criteria used to classify the historical estimate, AMEC was unable to compare the categories used by Chevron to current CIM Definition Standards and has thus not reported the tons and grade of the deposits in the Stage II Resource Estimate.

Chevron leased many of the claims that comprised the lithium project to the J. M. Huber Corporation (Huber) in 1986. In 1991, Chevron U.S.A., sold its interest in the claims to Cyprus Gold Exploration Corporation. In 1992, Huber terminated the lease and it appears that Cyprus Gold Exploration Corporation allowed the claims to lapse and provided much of the exploration data to Jim LaBret, one of the claim owners from which they had leased claims.

Table 6.1: Summary of 1985 Chevron Resources at McDermitt Caldera.

Lens	Number of Holes	Area (Acres)	Deposit Thickness (ft)	Deposit Thickness (m)	Waste Thickness (ft)	Waste Thickness (m)	Average Grade (%Li)	Deposit Tons (Mst)*	Li Tons (Mst)*
North	21	1,364	59	18.0	64	19.5	0.31	196	0.602
North Central	10	372	66	20.1	41	12.5	0.34	60	0.207
South Central	7	230	53	16.2	23	7.0	0.37	37	0.134
South	52	2,432	59	18.0	43	13.1	0.33	353	1.171
PCD	6	332	59	18.0	56	17.1	0.34	48	0.162
Total	**96**	**4,730**	—	—	—	—	**0.33**	**694**	**2.276**

* Mst = million short tons

There is no record indicating that Chevron considered potassium as a resource. From 1992 to 2005, there is no record of any activities on the project.

Western Energy Development Corp. (WEDC), a Nevada corporation, leased LaBret's claims in 2005 at which time, LaBret provided WEDC access to the Chevron data and to core and other samples that were available. In 2005, WEDC staked 1634 federal lode claims covering the area that was prospective for lithium, but subsequently dropped 320 of these claims. The claims cover much the same area as the original Chevron claims. WEDC compiled the Chevron exploration data and undertook preliminary marketing studies during 2007.

In 2007, WLC was incorporated for the purpose of lithium exploration and exploitation. WEDC provided the Chevron exploration data and access to available geological materials to WLC as part of a lease agreement for the properties described in Section 4.2 of the Stage II Resource Estimate between the two companies. Those data form the basis for that report.

From late 2007 to May 2008, WLC completed 37 core holes (4,861.98 m) and eight reverse circulation (RC) holes (1,798.63 m) in the PCD lens. WLC performed a mineralogical study of the mineralization in 2008 (Hudson, 2008). That study confirmed the presence of hectorite in the deposit. WLC determined the density of six samples from the Stage I Lens.

Additionally, beginning in the second half of 2009, WLC drilled 38 diamond drill holes in the Stage II Lens, as well as took 22 samples for density determination. The samples were sent to ALS-Chemex for analysis, using a four-acid digestion and ICP-MS for a full suite of values, including Li, K, and Na; F was assayed using two different fusion methods, depending on the sample value.

Metallurgical test work was performed on samples from the King's Valley lithium deposit by various researchers. The following list outlines the investigators and their associated reports:

- Chevron Research Company:
  - Hydrometallurgical Studies on Lithium Extraction of McDermitt Clays; Chevron Research Company, H. E. Kluksdahl, February 9th, 1982.
  - Recovery of Lithium from McDermitt Hectorite Clay; Chevron Research Company, D.A. Van Leirsburg, September 1982.
  - McDermitt Project Review; Chevron Research Company, March 24, 1983.
- United States Bureau of Mines (USBM):
  - Lithium and Its Recovery from Low-grade Nevada Clays, United States Bureau of Mines Bulletin 691; L. Crocker, R.H. Lien and Others, 1988.
- Hazen Research:
  - Extraction of Lithium from Hectorite Ore; Hazen Research Project 10677, Client Initiated Study Carried Out for WLC, Issued May, 2008.

The USBM bulletin discusses the USBM investigation for recovering a marketable lithium product from the montmorillonite-type clays of the McDermitt Caldera. The information presented in this publication is not considered to be compliant with Common Information Model (CIM) standards as required by NI 43-101, but is included here for historical purposes. Although this bulletin may not meet the standards of CIM, WLC believes the scientific basis of the report to be of sufficient quality to use as a starting point for establishing the metallurgical test program used in developing the process design.

6.2.1 Review of Chevron Research Company Test Work

In the 1980's, Chevron Resources undertook considerable metallurgical and processing studies to develop a procedure that could economically extract lithium from hectorite clay. Chevron experimented by improving the grade of the ore through upgrading techniques.

Chevron then tested lithium dissolution procedures which included dilute sulfuric acid leach, acid pug followed by water leach, and ion exchange. Chevron concluded that the most economical method would be to use the sulfuric acid pug followed by water leach method. The basis for selection included overall acid consumption and lithium recovery.

6.2.1.1 Sampling

The sample used for the test work reported here is a drill core sample denoted MJB 80-20C by Chevron. The exact location of the drill hole was not apparent from the information provided.

6.2.1.2 Upgrading

Chevron Research Company ran studies that concluded that dry grinding followed by separation of the fines coupled with upgrading of the coarse material gives the best overall upgrading of the ore. The ground material was screened into three fractions; +100 #, –100 # +400 #, and –400.

The –100 #/+400 # fraction from the grinding tests were then sent to an oleic acid float treatment to reduce the amount of acid-consuming carbonate in that material along with upgrading the lithium content. The –400 # fraction from the grinding step was mixed with the concentrate from the oleic acid float tests to obtain a concentrate material for further testing.

Chevron found that the final concentrate contained 75% of the original weight of the material and had a lithium assay of 0.37%. The head assay for lithium was 0.34%. A positive result from these studies was the fact that the carbonate content in the concentrate was significantly lower than in the feed material. This aided in reducing overall acid consumption in the subsequent stages.

6.2.1.3 Acid Pugging, Curing and Water Leaching

The acid pugging stage of the process involved intimately mixing the ore with concentrated sulfuric acid at high solids content (55–67% weight/weight percentage, or w/w). The mixture was then allowed to cure so that the acid reacted with the ore. The cured mixture formed a solid mass which was ground up and leached to solubilize the lithium. The best results showed that 85% of the lithium was extracted while consuming 915 pounds of acid per ton of ore.

6.2.2 Review of Hazen Research Test Work

Hazen Research was commissioned by WLC to examine processing options for the King's Valley lithium ore. Physical upgrading followed by sulfuric acid pugging, curing, and water leaching to extract lithium were studied. Most of the work completed was derived from the work conducted by Chevron Research Company described in Section 6.6.1. A bench-scale laboratory study was conducted to examine lithium extraction from a composite ore sample with about 0.35% lithium prepared from drill core samples supplied by WLC.

6.2.2.1 Samples

The composite sample prepared by Hazen for the work involving the upgrading and acid pugging was composed of drill core from hole WLC-10 from 40 to 100 feet with a head assay of 0.37% lithium.

6.2.2.2 Upgrading Test Work

Upgrading was designed to accomplish two goals: increase the lithium concentration while maintaining reasonable lithium recoveries and remove a substantial amount of calcite and other gangue material which would otherwise result in excessive consumption of acid in subsequent processing steps. The following physical processing steps were studied in the effort to achieve these goals:

1. Attrition or lightly grind the gangue material to disaggregate and liberate the clay. The goal was to keep the gangue relatively coarse.

2. Screen the material to remove the coarse gangue.

3. Use elutriation or cylconing to separate the lithium-rich clay fines from the coarser gangue material, which is low in lithium.

4. Use fatty acid (oleic acid) flotation for calcite removal.

After optimizing the operating parameters, Hazen found that wet attrition is effective for liberating the clay fines from the ore. In one experiment, 82% of the lithium reported to 53% of the mass in a –10 micron (~500 #) fraction after attrition. The lithium grade almost doubled to 0.6%.

After attrition, Hazen found that physical separation of the gangue on the basis of particle size, or by particle size and flotation for calcite, concentrates the lithium and rejects calcite and other gangue. Elutriation or another size classification method, such as cycloning, will separate the lithium-rich fines from the coarser gangue. Flotation with oleic acid was able to reduce the carbonate concentration to less than 0.5% (w/w) from 3.5 to 4%. Desliming after attritioning rejected over 90% of the carbonate but with higher lithium losses. When upgrading by particle size, there is a tradeoff between lithium recovery and carbonate rejection.

In the course of the project, URS has also developed a flow sheet using the upgrading process from the Hazen Research study. This work was put on hold until trade-off studies could be completed against filterability and capital for filtration equipment.

6.2.2.3 Acid Pugging, Curing and Water Leaching

The fines from the upgrading work were dried and blended to make a composite. The composite assayed 0.72% Li and 0.72% carbonate. All acid pugging and leaching experiments were performed on this composite.

The main variable examined in the acid pugging, curing, and water leaching work was acid dose, although variations in the amount of water added was also tried. In the laboratory procedure, water (if used) was added to dry clay in a porcelain mortar, followed by concentrated sulfuric acid. The mixture was worked with a pestle until homogeneity was achieved. The mixture was then covered and heated (cured) in an oven at 150°C for three hours. After curing, the solids were usually a dry hard mass. The solids were then lightly ground and water-leached for 30 minutes at 10% (w/w) solids. The lithium sulfate and other soluble sulfates dissolved in the leach.

It was found that this process was effective in extracting lithium from the clay fines, but an acid dose of near 1150 lb acid per ton of ore was required for 95% extraction. It was also revealed that extraction of lithium was nearly proportional to the acid dose. Overall results obtained by Hazen were consistent with those found by Chevron Research Company.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project in Humboldt County, Nevada, lies in the western and southern part of the McDermitt caldera complex (Figure 7.1), a Miocene volcanic feature that comprises a total surface area of about 30 km × 40 km. On the basis of U.S. Geological Survey publications (e.g., Rytuba and Glanzman, 1979), the caldera complex was considered to be formed of multiple nested calderas. More recently, the complex has been described as a single caldera formed during eruption of a zoned ignimbrite (Castor and Henry, 2000; Starkel and others, 2009).

Volcanic activity at the McDermitt caldera complex has yielded precise $^{40}Ar/^{39}Ar$ ages of 16.5 to 16.1 million years ago and was characterized by extrusion of early metaluminous and peralkaline rhyolite, followed by eruption of a voluminous ignimbrite with peralkaline rhyolite to metaluminous dacite compositions (Castor and Henry, 2000, Starkel and others, 2009). After collapse, the central part of the caldera complex was the site of resurgence, and a moat-like lake formed between this resurgent dome and the caldera walls. The lake was the site of deposition of volcaniclastic sediments that now form a nearly continuous ring within the caldera (Figure 7.1). The Kings Valley lithium deposits are lens-shaped bodies hosted by the moat sedimentary rock.

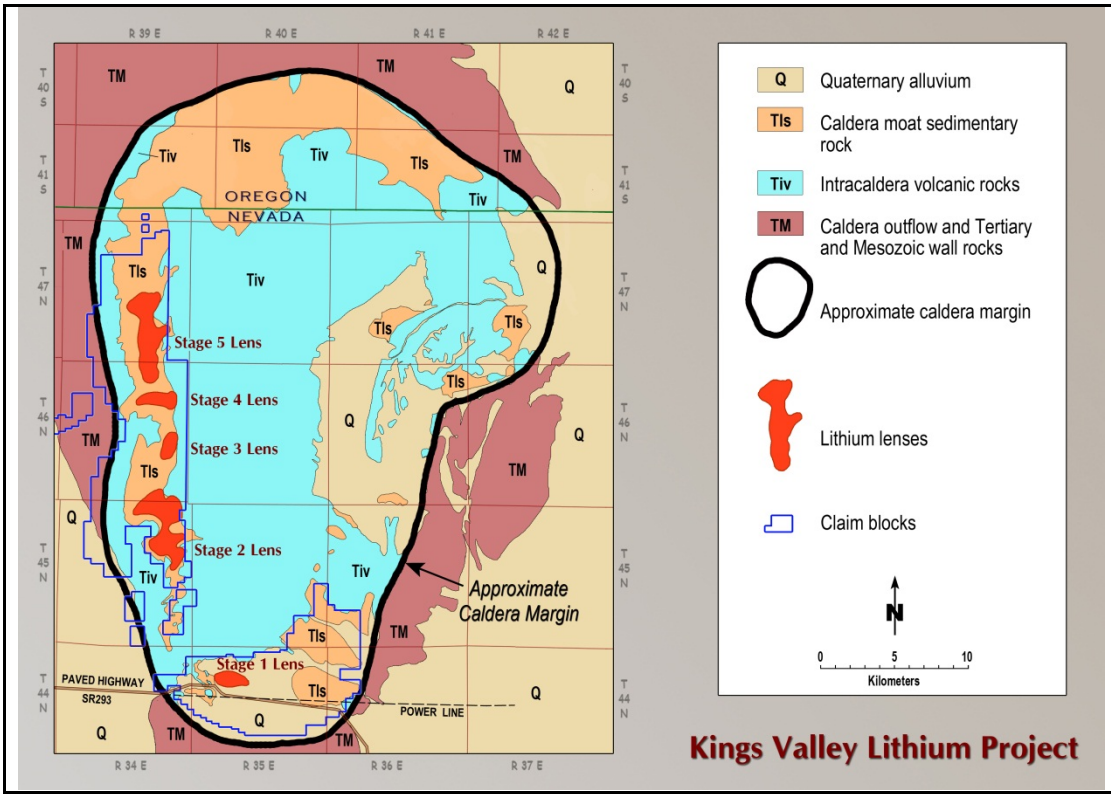


Figure 7.1: Map of the Kings Valley Lithium Project

7.2 LOCAL GEOLOGY

7.2.1 McDermitt Caldera

The Project deposits occur in a north-south zone that coincides with the western and southern part of the McDermitt caldera moat sedimentary rock. The most southerly deposit, known as the Stage I Lens, is considered to be the primary target and is best defined. The Stage II to Stage 5 lenses lie to the north.

7.2.2 Stage I Lens

The Stage I Lens is largely beneath alluvial cover at Thacker Pass. It lies at relatively low elevations in moat sedimentary rocks that are thought to have been separated from the topographically higher deposits to the north by late down-to-the south normal faulting (Castor and Henry, 2000). Exposures of the moat sedimentary rocks at Thacker Pass are limited to a few drainages and isolated road cuts; therefore, the stratigraphic sequence in the deposit is primarily derived from core drilling.

The moat sedimentary section, which has a maximum drilled thickness of about 160 m, consists of alternating layers of claystone and volcanic ash. The claystone comprises 40% to 90% of the section. In many intervals, the claystone and ash are intimately intermixed. The claystones are variably brown, tan, gray, bluish-gray and black whereas the ash is generally white or very light gray. Individual claystone-rich units may be recognized over lateral distances of more than 152 m although unit thickness can vary by as much as 20%. Ash-rich layers are more variable and appear to have some textures that suggest reworking. All units exhibit finely-graded bedding and laminar textures that imply a shallow lacustrine depositional environment.

Surficial oxidation persists to depths of 15 to 30 m in the moat sedimentary rock. Oxidized claystone is brown, tan, or light greenish-tan and contains iron oxide, whereas the ash is white with some orange-brown iron oxide. The transition from oxidized to unoxidized rock occurs over intervals as much as 4.5 m thick.

The moat sedimentary section at Thacker Pass overlies intracaldera volcanic rock that has been logged as ash-flow tuff, flows and lahars. A zone of weakly to strongly silicified sedimentary rock, the Hot Pond Zone (HPZ), occurs at the base of the moat sedimentary section in nearly every drillhole in the Stage I deposit area. The HPZ rock is locally oxidized, and the underlying volcanic rock is generally oxidized.

Core from each drillhole has been carefully examined and drill logs have been prepared that record rock type, color, accessory minerals, textures and other features of significance. On the basis of core examination, 14 lithologic codes have been recognized (Table 7.1). The core has mostly been divided into sample intervals for chemical analyses delineated on the basis of lithology. Figure 7.2 shows a generalized interpretation of the lithology for core hole WLC-43 which is located roughly in the middle of the proposed mine pit area. For this interpretation, all the claystones (lithologic units 1-6) are combined into one lithologic unit.

Table 7.1: Summary of Lithologic Units

Lithologic Code	Geology	Dominant Characteristic	Secondary Characteristic
1	Green claystone	Greenish gray claystone	With or without iron oxidation, may or may not be massive, may or may not contain thin ash layers
2	Tan claystone	Tan to light brown claystone	
3	Lt gray claystone	Light gray to medium gray claystone	
4	Gray claystone	Medium gray to dark gray claystone	
5	Carbon claystone	Dark gray claystone	Minimal to moderate carbon, may have <20% white or gray-white ash
6	Bluish claystone	Bluish gray claystone	May be massive
7	Ash/claystone	>60% gray or white ash or arkose with claystone	Ash may be layered or intermixed with claystone and may have iron oxide coatings
8	Claystone/ash	>60% green, tan, gray, dark, gray, black or blue claystone with white, tan or gray ash	Claystone may be layered or intermixed with ash
9	Laminated	Laminated claystone and ash, usually small thin layers	n/a
10	Qal (Quaternary alluvium)	Light brown to tan alluvium, ranges between 0 and 8.2 meters thick	n/a
11	HPZ (Hot Pond Zone)	Laminated, silicic and oxidized	Has to have all three dominant characteristics
12	TV (Tertiary volcanics)	Vuggy orange brown volcanics	May contain lithic fragments up to four inches in diameter, includes, voids, mudflows, breccias
13	Basalt	Brown to black basalt	May or may not be oxidized
14	Ash	>80% gray, light tan, white or yellow-brown ash	Minor arkose may be present

*Lithologic codes were provided by WLC
Codes are based on drillhole descriptions for WLC 1-37 and WLC 40-200

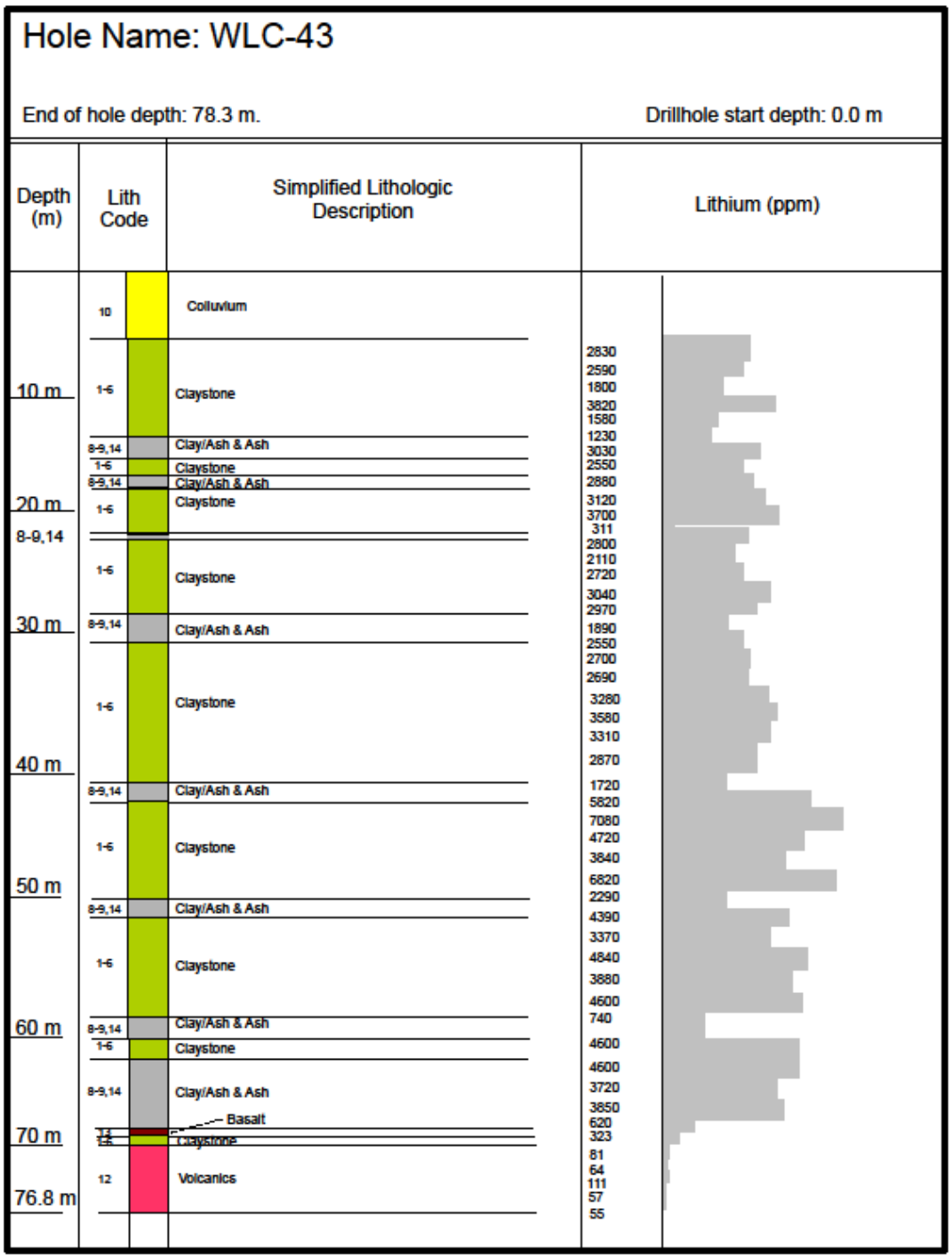


Figure 7.2: Interpreted and Simplified Sample Log for Drillhole WLC-43

Most of the moat sedimentary rocks drilled in the Thacker Pass basin contain anomalously high lithium contents (> 100 ppm). Intervals that consist mostly of ash have lithium contents of less than 800 ppm whereas intervals dominated by claystone contain more lithium (>1,000 ppm). Many intervals have very high lithium contents (>4,000 ppm). Intervals with extreme lithium contents (>8,000 ppm) occur locally.

There is no change in lithium content across the boundary between oxidized and unoxidized rock; however, the highest lithium grades generally occur in the middle and lower parts of the sedimentary rock section.

The lithium content of the Stage I Lens claystone can generally be predicted by the color and texture of the rock, as well as the amount of admixed ash. Intervals with the highest lithium grades (>4,000 ppm) generally contain gray to dark-gray or black claystone with less than 10% ash. Intervals of bluish-gray claystone with low ash content have moderate lithium content (generally 2,500 to 3,000 ppm). Intervals of light colored claystone (e.g., tan, light gray, greenish-tan) have lower lithium grades (generally 1,500 to 2,500 ppm). Intervals of mixed claystone and ash are common and have variable lithium contents (generally 1,500 to 3,000 ppm) depending on the type of claystone and proportion of ash present.

7.2.3 Stage II Lens

The Stage II Lens is located approximately 10 km NNW of the Stage I Lens, the southernmost lithium mineralized lens in the area. The Stage II Lens mineralized beds comprises mainly of a dark green claystone, at times intercalated with arkose beds; in the NE region of the modeled area a fanglomerate body is present. Figure 7.3 shows an example of a Field Log from hole SP-3. Lithium-rich beds are generally 10 to 60 m thick in most areas, defined as interesting Li grades as greater than 1,500 ppm. Alluvium cover is variable, ranging from no cover at all to a maximum of about 6m.

WLC drilling shows that the average thickness of Li mineralization is thicker than that indicated by Chevron data, because, as was the Case in the PCD lens, some of the Chevron holes stopped in mineralization. Figure 7.4 shows a thickness map of the area, for lithium grades greater than 1,000 ppm.

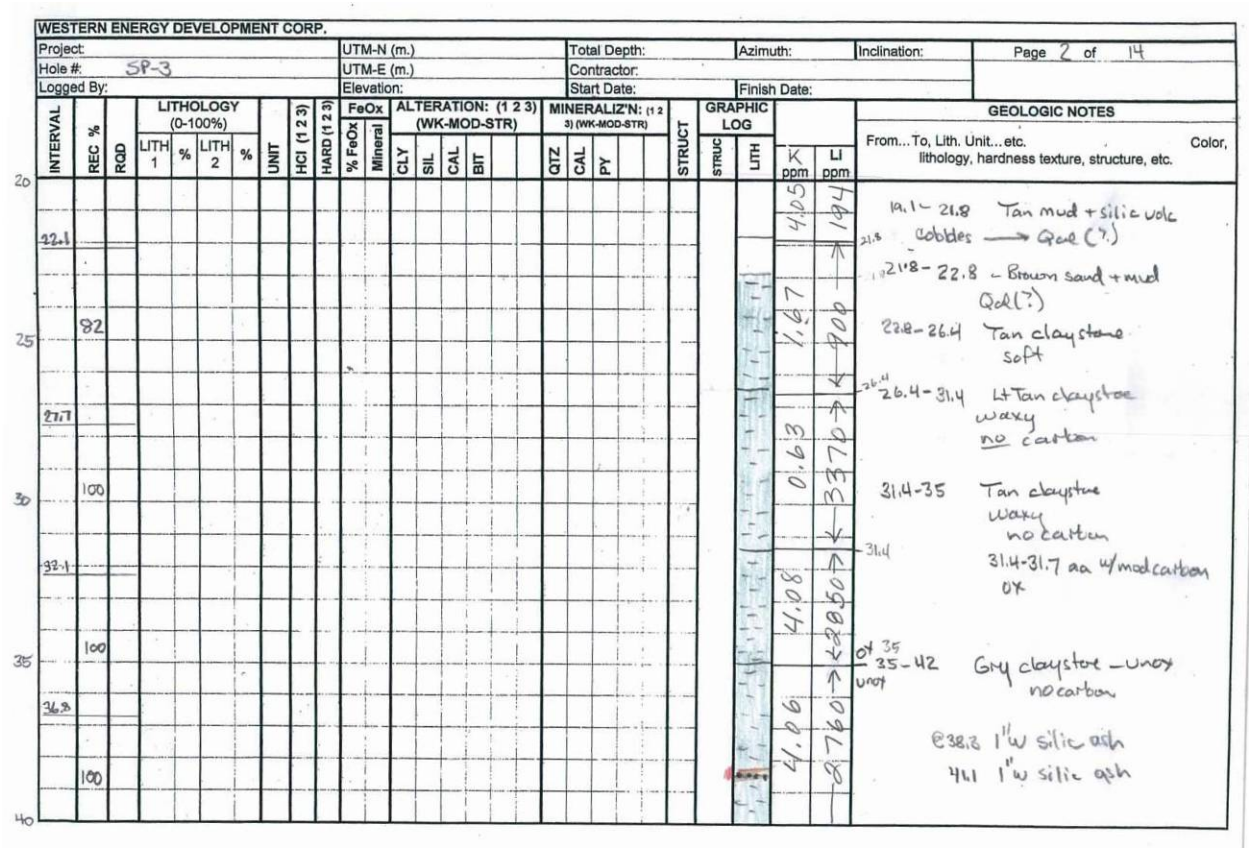


Figure 7.3: Sample Field Log for drill hole SP-3 in the Stage II Lens.

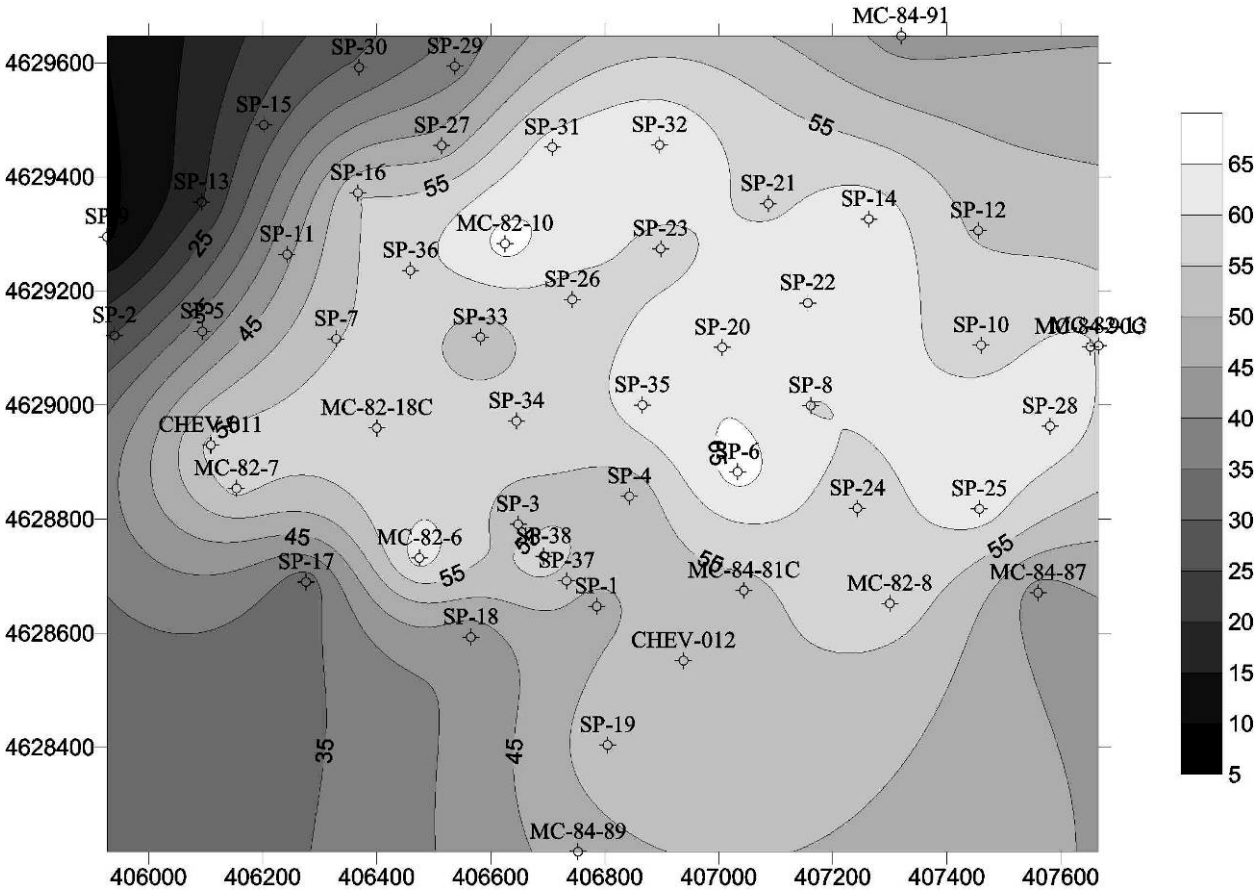


Figure 7.4: Thickness of Li mineralization (in meters) within the modelled area of the Stage II Lens (1000 Li ppm minimum grade).

7.2.4 Mineralogy

Clay in the Stage I Lens lithium deposit includes two distinctly different types on the basis of chemistry and X-ray diffraction (XRD) spectra. Clay with XRD spectra that are indicative of smectite occurs at relatively shallow depths (less than 30 m bgs) in the Stage I Lens deposit (Castor, 2010). Confirmed hectorite clay occurs elsewhere in the McDermitt caldera and has been documented by several authors (*e.g.*, Odom, 1992; Rytuba and Glanzman, 1979). Drill intervals with high lithium contents (commonly > 4,000 ppm) contain clay that yields XRD spectra that are more typical for illite than smectite (Castor, 2010). An illite-type clay occurs at moderate to deep depths in the moat sedimentary section and locally occurs in intervals that contain as much as 8,000 ppm lithium, higher than any analyzed hectorite. A relatively thin layer of interstratified smectite-illite clay is found between the smectite and illite-type clay (Castor, 2010).

Other minerals in the Stage I Lens deposit claystone include calcite, quartz, K-feldspar, plagioclase, dolomite, and fluorite. Pyrite and bitumen occur in the claystone below near-surface oxidized rock. Ash beds in the Stage I Lens deposit contain quartz and feldspar with local analcime, and minor clay and pyrite. Zeolite minerals are typically present in the north part of the caldera, but analcime is the only zeolite present in the Stage I Lens deposit (Glanzman and Rytuba, 1979).

No detailed mineralogical work has been conducted for samples from the Stage II drilling campaign.

7.2.5 Discussion

The regional geological setting of these deposits is quite well known and very well understood. The local geological setting and degree of local lithium grade variations are adequately known for the Stage I and Stage II areas to allow estimating the resources to the accuracy modeled by the resource classification applied.

8.0 DEPOSIT TYPE

8.1 DESCRIPTION OF DEPOSIT

The information in this section was transcribed in part from the URS report: NI 43-101 Technical Report *Preliminary Assessment and Economic Evaluation, Kings Valley Project, Humboldt County, Nevada, USA, effective date December 31, 2009.*

The Kings Valley lithium deposits are part of a small group of lithium-rich clay minerals known as illite and smectite and include hectorite [Na0.3(Mg,Li)3Si4O10(F,OH)2]. Hectorite was named for its first known source, the Hector Mine, 55 km east of Barstow in San Bernardino, California. This is the only hectorite deposit currently known to be in production; however, other small producers may sporadically mine hectorite. The production is used primarily for specialty clay products. Lithium is not extracted from the clays from Hector.

Harben and Bates (1984) suggest that the Hector deposits were formed when Pliocene tuff and volcanic ash were deposited in a restricted alkaline lake environment on and adjacent to travertine which was forming by hot springs emanating along the fault. The tuff was initially converted to clinoptilolite (a zeolite mineral) which, then in turn, was altered to hectorite by hot springs waters rich in lithium and fluoride.

Although no work has been reported on the genesis of the Kings Valley deposits, the geology suggests that the deposits occur over the ring fracture zone of the caldera complex. Post volcanic intrusions along those fractures likely powered hydrothermal cells that formed the deposits. Hydrothermal solutions ascended through the volcanic section along the ring fractures and other faults and passed through the volcanic section and into the lacustrine environment of the moat. These ascending fluids extracted lithium from the rhyolitic ash flow tuffs and the volcaniclastic sediments that immediately overlie the rhyolites. Glanzman and Winsor (1982) suggest that the fluids were deposited into the moat where a lithium-rich gel formed within the lake and precipitated as a stratabound, massive layer of Li-rich claystone. Thin intervals of volcaniclastic sediments were deposited below, within, and above this claystone. Another possibility is that the Li-bearing clays were the product of alteration of in situ, clay-rich horizons by the hydrothermal fluids as they traversed the geological column.

Additional work is required to determine the origin of these deposits. There are indications in some drill holes that the rocks are the product of nearby explosive volcanic eruptions, and that they have not been significantly transported by sedimentary processes. These rocks are altered and locally contain elevated Li which suggests that the rocks were deposited and then altered by hydrothermal fluids. Fine-grained volcanic and volcaniclastic material was converted to hectorite-bearing clay beds by that hydrothermal event.

8.2 BASIS OF EXPLORATION

The sediments within the Kings Valley lithium deposits host a relatively large concentration of lithium compared to most other sources. Testwork to date has shown that extraction from the deposit is economically feasible. Exploration activities have defined the geology of the deposit, defined the ore body and resulted in categories developed for measured, indicated and inferred resources for the Stage I and II Lenses of this project

9.0 EXPLORATION

9.1 STAGE I

Prior to the 2010 drilling campaign, exploration consisted of geological mapping to delineate the limits of the sedimentary rocks within the moat and drilling to determine grade and location of mineralization. Most of the Project area has been surveyed by airborne gamma ray spectrometry, though this work was not done specifically to enhance lithium exploration (PAEE URS 2010).

Survey work was completed prior to 1980 under Chevron. A collar survey was completed by WLC for the 2007-2008 drilling program using a Trimble GPS. At that time the NAD 83 global reference system was used. Comparing WLC's survey work with that done by Chevron showed near identical results for the easting and northings, elevations were off by approximately 3m and were corrected in order to conform with earlier Chevron work (PAEE URS 2010).

Downhole surveys have been performed on selected holes drilled after WLC-24. Those surveys indicate that holes are drilled vertically or very nearly vertical with the exception of hole (WLC 58) which was intentionally drilled at 70 degrees from horizontal.

Exploration work that commenced January 2010 focused on further refining the local stratigraphy and ore grades within the deposits. This work has been done in the form of systematic drilling and is covered in detail in Section 10.1. The drilling (sampling) to date covers an area of some 50 hectares, measuring approximately 1000m EW x 500m NS.

In addition to drilling, WLC developed two test pits (WLT-01 and WLT-02) in January 2010. The purpose of trenching was to obtain large bulk samples for metallurgical testwork. Alluvium was removed via bulldozer and two 4.6 m trenches were dug into the clay deposits. A total of 15 samples were collected for assay. Intervals were selected to make a composite which would approximate the ore body. These composites were shipped to the Outotec GmbH facility in Germany for testwork

The topographic surface of the Project area was mapped by aerial photography dated July 6, 2010. This information was obtained by MXS, Inc. for WLC. The flyover resolution was 0.35 m. Ground control was established by Desert-Mountain Surveying, a Nevada licensed land surveyor, using Trimble equipment. Field surveys of drillhole collars, spot-heights and ground-truthing were conducted by Mr. Dave Rowe, MXS, Inc., a Nevada licensed land surveyor, using Trimble equipment.

Reserva believes that the exploration techniques used were appropriate and that the extent and grade distribution of the deposit is reasonably well addressed. Although some sample bias was detected ti earlier RC and Sonic drilling, as described in Section 10, the vast majority (96%) of the drilling was core drilling. Based on observation of the field and logging/sampling procedures, the samples are considered to be representative and of reasonable quality. Based on geological logging data, the location and attitude of several faults that offset the base of the deposit were determined and incorporated into the overall model. The data are adequate at this stage to adequately define the local stratigraphy and grade of the deposit for mineral resource estimate purposes.

9.2 STAGE II

The information in this section is based in part on the corresponding section of the URS report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project*, *Humboldt County, Nevada, USA*, effective date December 31, 2009.

Exploration on WLC's Stage II Lens deposit has consisted of geological mapping to delineate the limits of the moat volcaniclastic sedimentary rocks and drilling to determine the grade and location of mineralization. Some, if not most, of the area has been covered by airborne gamma ray spectrometry, but those data are not pertinent to exploration for lithium. There is no record of other exploration in the area.

Section 6.2 and 10.2 discusses drilling from historical and factual perspectives. The Chevron database includes 283 core, RC, and rotary holes (25,781.8 m). Records indicate that 268 holes were drilled specifically to evaluate the lithium mineralization. The record is not clear, but it appears that of the 283 holes in the database, 223 are rotary holes (18,136.2 m), 52 are core holes (5,731.11 m) based on the "c" suffix on drill-hole names, eight are reverse circulation holes (1,798.62 m), two are sonic holes (153.7 m), and seven are PQ-size core holes for metallurgical samples (233.5 m). In addition to those holes, 370 auger holes (902.5 m) were drilled. The reason for the auger holes has not been determined.

Claim surveying for the Chevron holes was performed by Tyree Surveying Company, Albuquerque, New Mexico and Desert Mountain Surveying Company, Winnemucca, Nevada (Chevron, 1980). According to Chevron (1980) both companies used theodolites and laser source electronic distance meters to survey the claims. Records indicate that both companies surveyed drill collar locations and it is presumed that the same instrumentation was used for those locations. The reported error was within 0.1515 m horizontally and 0.303 m vertically.

Additionally, WLC has drilled to date 38 diamond drill holes in the Stage II Lens. Collar surveying for the Stage II Lens drill holes was conducted by WLC using a Trimble GPS. The NAD 83 global reference system is used. WLC compared the locations of several points surveyed by Chevron and found that the easting and northing coordinates were more or less identical. The elevation, however, was found to have a systematic difference of 3.11 m. As a result, 3.11 m was subtracted from the GPS elevations to conform to the older surveys.

WLC surveyed the initial holes in the Stage II Lens campaign (SP-1, SP-2, SP-3, SP-4, SP-5 and SP-6); later, after drilling had been completed, holes SP-12, SP-14, SP-21 and SP-23 were also surveyed for down the hole deviations. WLC found that none deviated significantly from the vertical orientation. Prior work by WLC on the Stage I (PCD Lens) had indicated as well that there is little deviation of the holes from vertical, supporting the assumption of verticality of all other holes drilled in the Stage II area. GSI accepts that the WLC and Chevron holes are vertical as a reasonable approximation, given the drilling conditions and the drill hole total depths.

GSI believes that the exploration techniques used were appropriate and that the extent and tenor of the deposits is reasonably well known. The data are adequate at this time to support inferred and indicated mineral resource categories, but additional drilling is required to adequately define the local stratigraphy and grade of the deposits in order to improve the confidence of the mineral resource estimate.

10.0 DRILLING

10.1 STAGE I

10.1.1 Type and Extent of Drilling by WLC

The Stage I Lens (formerly known as the PCD Lens) has been explored for minerals since the 1970s. Exploration began with a focus on uranium found in the area, but switched primarily to lithium in the late 1980s when Chevron still controlled the mining interests. WLC drilled 51 core holes between 2007 and 2009 to expand on the Chevron work. These holes were drilled with the primary aim of defining lithium occurrences in and the geology of the deposit. Table 10.1 lists a summary of the holes drilled. The 37 core holes (WLC-01 thru WLC-37) were drilled specifically for assay and lithologic information. The five reverse circulation (RC) holes were drilled to compare drilling techniques. The RC method proved to produce biased assay results in the Stage I Lens area so the method was abandoned. Seven PQ-sized holes were drilled with the intent to provide samples for metallurgical testwork. Two sonic holes were drilled to test the drilling procedure. It was determined that, while the sonic drill worked well, the geologic definition was not comparable to traditional coring. For more historical information on drilling refer to Section 6.1 of this technical report and to the PAEE (URS 2010).

Table 10.1: WLC Drillholes Provided in Current Database

Drilling campaign	number drilled	Type	Hole IDs in database	Approx Total Meters Drilled	Used in current resource estimate
2007-2008	5	RC	TP-1, 2,3,6, 7.	707	no
2007-2008	37	HQ	WLC-01 thru WLC-37	4,950	yes **
2009	7	PQ	WPQ-1 thru WPQ-7	233	no
2009	2	sonic	WSH-01, -02	150	yes
2010 - 2011	161	HQ	WLC-40 thru WLC-200	13,219	yes **
*WLC 38-39 were not drilled ** Excepting drillholes WLC 180, 183 and 185					

Note: Does not include drilling by Chevron

WLC initiated a carefully designed drilling campaign which commenced in January 2010 with the goal of further defining measured and indicated resource categories for lithium. An additional 161 drillholes were drilled (WLC-40 thru WLC-200). Figure 10.1 shows a plan view of drillholes in the Stage I Lens in UTM NAD 27 values. Only those drilled within the easting and northing extents on the map are shown; some drillholes to the north and east are not shown for simplicity.

Table 10.2 is a comprehensive list of WLC drillholes to date.

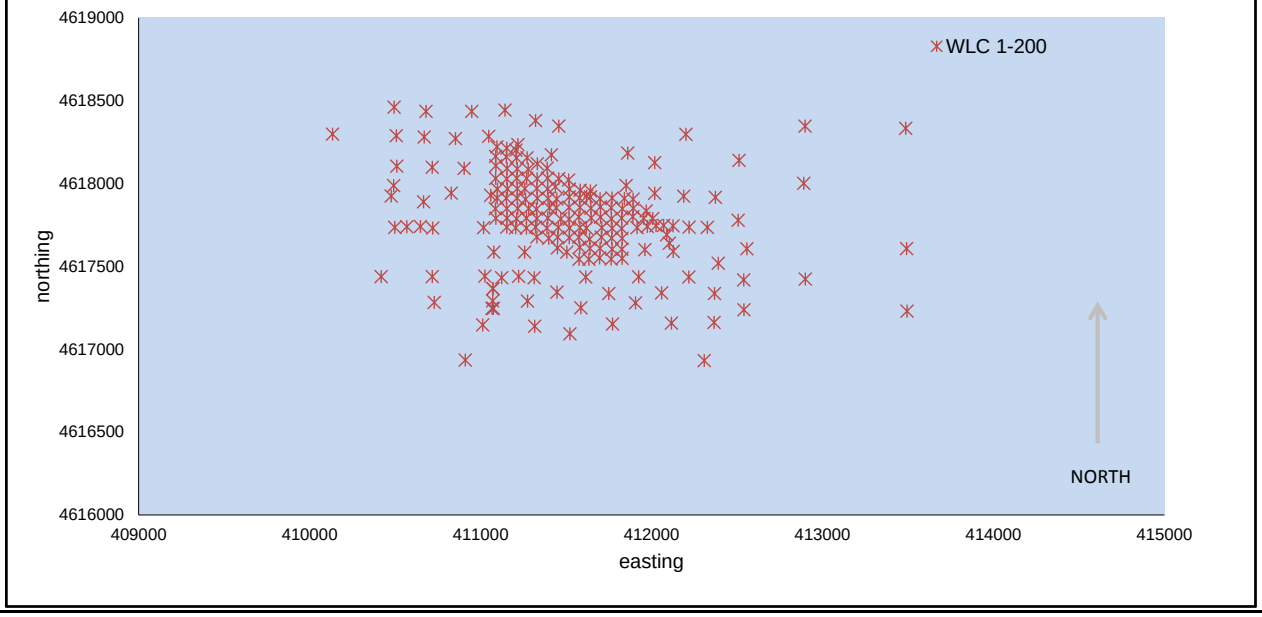


Figure 10.1: Plan View of WLC Drillholes in the Stage I Lens

Note: Map in UTM NAD 27 values. Only those drilled within the easting and northing extents on map shown, some drillholes to the north and east are not shown for simplicity

Table 10.2: Summary of Drillholes at the Western Lithium Project Area

Drilling Campaign	Hole ID	Easting	Northing	Elevation (m)	Bearing	Plunge degrees	Total Depth (m)
2007-2008	TP-1	411070.6	4617291.5	1542.7	0	90	122.0
2007-2008	TP-2	410498.6	4617735	1568.9	0	90	98.0
2007-2008	TP-3	412326.1	4617736.5	1522.9	0	90	183.0
2007-2008	TP-6	412276.9	4617732	1524.2	0	90	152.0
2007-2008	TP-7	412173.1	4617740	1527.4	0	90	152.0
2007-2008	WLC-01	411070.5	4617291.5	1542.7	0	90	125.0
2007-2008	WLC-02	410498.6	4617735	1569	0	90	96.9
2007-2008	WLC-03	412326.2	4617735	1522.9	0	90	185.3
2007-2008	WLC-04	412019	4617940.5	1541.7	0	90	139.3
2007-2008	WLC-05	412218.1	4617435.5	1510.8	0	90	182.9
2007-2008	WLC-06	411924.5	4617437.5	1517.4	0	90	256.8
2007-2008	WLC-07	411614.7	4617435.5	1521.2	0	90	188.1
2007-2008	WLC-08	410720.2	4617730.5	1561	0	90	133.2
2007-2008	WLC-09	411017.3	4617734	1543.7	0	90	96.6
2007-2008	WLC-010	411321.3	4617737	1539	0	90	104.2
2007-2008	WLC-011	411313.8	4617431	1527.7	0	90	100.3
2007-2008	WLC-012	411619.3	4617731	1537.5	0	90	171.7
2007-2008	WLC-013	411914.8	4617734	1532.9	0	90	126.9
2007-2008	WLC-014	411315.8	4617139	1529.3	0	90	181.8
2007-2008	WLC-015	411025.9	4617441	1547.7	0	90	139.4
2007-2008	WLC-016	410717	4617438	1540.5	0	90	83.5
2007-2008	WLC-017	410910.6	4616934.5	1516	0	90	274.9
2007-2008	WLC-018	411012.9	4617146	1530.8	0	90	157.6
2007-2008	WLC-019	410418.3	4617438.5	1540	0	90	105.8
2007-2008	WLC-020	412219	4617737	1525.6	0	90	156.5
2007-2008	WLC-021	412125.7	4617745	1528.6	0	90	152.4
2007-2008	WLC-022	412070.4	4617747	1530.4	0	90	138.7
2007-2008	WLC-023	411969.3	4617835	1537.3	0	90	130.1
2007-2008	WLC-024	412006.9	4617789.5	1533.7	140.2	89	182.0
2007-2008	WLC-025	411975.4	4617741	1532.5	354.8	88.8	159.1
2007-2008	WLC-026	412026.2	4617745.5	1530.6	297	89.5	146.3
2007-2008	WLC-027	412088.5	4617690.5	1526.2	358.6	89.5	115.8
2007-2008	WLC-028	412105	4617639.5	1523.2	143.3	89.8	246.0
2007-2008	WLC-029	412126.5	4617590.5	1519.9	212.2	89.6	122.8
2007-2008	WLC-030	411075.5	4617246	1540.4	217.2	89.5	99.7
2007-2008	WLC-031	411072.3	4617365	1544.4	231.1	89.7	93.6
2007-2008	WLC-032	411072.3	4617365	1544.4	0	90	30.6
2007-2008	WLC-033	411067.5	4617246.5	1540.8	0	90	30.5
2007-2008	WLC-034	411123	4617432	1542.2	126.9	89.9	99.7

Table 10.2: Summary of Drillholes at the Western Lithium Project Area

Drilling Campaign	Hole ID	Easting	Northing	Elevation (m)	Bearing	Plunge degrees	Total Depth (m)
2007-2008	WLC-035	411221.9	4617439.5	1533.5	26.5	89.7	121.2
2007-2008	WLC-036	410569.1	4617738.5	1568.5	43.8	89.5	78.3
2007-2008	WLC-037	410648.4	4617740	1564.2	0	90	69.2
2009	WPQ-1	411077.2	4617288.5	1542.1	0	90	37.3
2009	WPQ-2	411020.7	4617442	1548	0	90	37.3
2009	WPQ-3	411221.4	4617432.5	1533.9	0	90	37.3
2009	WPQ-4	412110.2	4617629.5	1522.4	0	90	19.1
2009	WPQ-5	412127.2	4617745.5	1528.5	0	90	27.8
2009	WPQ-6	412322.2	4617736.5	1523.1	0	90	37.4
2009	WPQ-7	412213.1	4617438	1511.1	0	90	37.4
2009	WSH-1	412219.7	4617258	1503.4	0	90	102.7
2009	WSH-2	412047.2	4617744	1530	0	90	47.5
2010-2011	WLC-40	411851.8	4617987	1548.5	0	90	79.8
2010-2011	WLC-41	411620.7	4617923	1549.3	0	90	56.3
2010-2011	WLC-42	411503.7	4617586.5	1528.5	0	90	91.7
2010-2011	WLC-43	411764	4617853.5	1542.6	0	90	86.0
2010-2011	WLC-44	411769.7	4617603	1528.2	0	90	116.4
2010-2011	WLC-45	411961.1	4617601	1524.6	0	90	107.2
2010-2011	WLC-46	411447.4	4617343.5	1522.5	0	90	116.4
2010-2011	WLC-47	411750.2	4617336.5	1512.2	0	90	107.3
2010-2011	WLC-48	411906.8	4617279	1507.3	0	90	122.6
2010-2011	WLC-49	412058.5	4617339.5	1510.1	0	90	115.5
2010-2011	WLC-50	412116.2	4617157.5	1499.2	194.1	89.4	154.6
2010-2011	WLC-51	412373.4	4617916	1527.3	11.4	89.3	136.3
2010-2011	WLC-52	412188.2	4617923.5	1538.1	0	90	126.2
2010-2011	WLC-53	412557.2	4617606	1511.5	0	90	121.1
2010-2011	WLC-54	412506.8	4617777.5	1518.2	0	90	87.5
2010-2011	WLC-55	412390.8	4617518.5	1511.3	0	90	124.1
2010-2011	WLC-56	412368.1	4617336	1502.4	0	90	139.4
2010-2011	WLC-57	412899.3	4617423	1504.8	261	89.2	273.5
2010-2011	WLC-58	412308.7	4616931.5	1485.4	180	70	736.4
2010-2011	WLC-59	412539.3	4617418	1506	0	90	142.4
2010-2011	WLC-60	412540.7	4617237.5	1502.7	0	90	145.5
2010-2011	WLC-61	412365.4	4617161	1495.3	0	90	157.6
2010-2011	WLC-62	411468.8	4617786	1540	0	90	63.9
2010-2011	WLC-63	411437.2	4617981	1553.1	0	90	91.6
2010-2011	WLC-64	411413.8	4618172.5	1563.8	0	90	114.9
2010-2011	WLC-65	411262.6	4617905	1549.6	0	90	108.8
2010-2011	WLC-66	411277.9	4618086	1556.9	0	90	82.9

Table 10.2: Summary of Drillholes at the Western Lithium Project Area

Drilling Campaign	Hole ID	Easting	Northing	Elevation (m)	Bearing	Plunge degrees	Total Depth (m)
2010-2011	WLC-67	410827.8	4617941.5	1559.7	0	90	108.2
2010-2011	WLC-68	411274.4	4617290	1534.2	0	90	132.4
2010-2011	WLC-69	411586.1	4617250.5	1522.8	0	90	136.2
2010-2011	WLC-70	411772	4617152	1515.7	311	89.8	153.7
2010-2011	WLC-71	411860.4	4618183.5	1564.7	0	90	130.1
2010-2011	WLC-72	412018.7	4618126	1554.7	0	90	145.4
2010-2011	WLC-73	412201.7	4618296	1548	0	90	151.4
2010-2011	WLC-74	412511.4	4618139.5	1549.4	0	90	106.1
2010-2011	WLC-75	411217.9	4618234.5	1565.2	0	90	96.0
2010-2011	WLC-76	411048.1	4618284	1574	0	90	125.6
2010-2011	WLC-77	411322.1	4618380.5	1579.3	0	90	122.6
2010-2011	WLC-78	411089.9	4618107	1565	90.71	89.3	90.1
2010-2011	WLC-79	411456.7	4618347	1576.4	0	90	131.7
2010-2011	WLC-80	411142.6	4618443.5	1579.2	0	90	113.4
2010-2011	WLC-81	411059.8	4617929	1555.5	0	90	90.0
2010-2011	WLC-82	410718	4618098.5	1579.4	0	90	98.1
2010-2011	WLC-83	410507.9	4618288.5	1594	0	90	136.3
2010-2011	WLC-84	410904.4	4618091	1564.2	0	90	89.1
2010-2011	WLC-85	410133.5	4618298.5	1567.4	0	90	61.2
2010-2011	WLC-86	410509.7	4618105	1579.9	0	90	102.7
2010-2011	WLC-87	410493.8	4618461	1604.4	151.3	89.6	134.1
2010-2011	WLC-88	410670.9	4618280.5	1589.8	0	90	125.1
2010-2011	WLC-89	410680.9	4618435.5	1598	0	90	142.4
2010-2011	WLC-90	410852.8	4618270.5	1581.3	0	90	105.8
2010-2011	WLC-91	410948	4618435	1585.9	0	90	123.7
2010-2011	WLC-92	410666.1	4617889	1564.6	0	90	97.5
2010-2011	WLC-93	411257.9	4617586.5	1535.3	0	90	57.0
2010-2011	WLC-94	411078	4617586.5	1541.3	0	90	43.3
2010-2011	WLC-95	411454.6	4617735.5	1538.8	0	90	70.0
2010-2011	WLC-96	411280.9	4617849.5	1547.9	0	90	119.5
2010-2011	WLC-97	411445.8	4617907.5	1549.3	0	90	67.7
2010-2011	WLC-98	411268.8	4618030.5	1555	0	90	89.0
2010-2011	WLC-99	411402.8	4617853	1545.5	0	90	71.8
2010-2011	WLC-100	411517.4	4617856	1546.7	0	90	67.7
2010-2011	WLC-101	411331.4	4617910.5	1550.1	0	90	117.9
2010-2011	WLC-102	411450.8	4617676.5	1536.5	0	90	69.2
2010-2011	WLC-103	411151	4617910.5	1554.4	340	89.6	90.5
2010-2011	WLC-104	411449.9	4617611.5	1533.3	0	90	75.3
2010-2011	WLC-105	411214.8	4617968.5	1554.2	0	90	98.1

Table 10.2: Summary of Drillholes at the Western Lithium Project Area

Drilling Campaign	Hole ID	Easting	Northing	Elevation (m)	Bearing	Plunge degrees	Total Depth (m)
2010-2011	WLC-106	411212.3	4618031.5	1557.1	0	90	72.8
2010-2011	WLC-107	411215	4618088.5	1559.2	0	90	89.0
2010-2011	WLC-108	411212.7	4617913	1552	0	90	103.2
2010-2011	WLC-109	411389.2	4618092	1558.3	0	90	82.9
2010-2011	WLC-110	411216.8	4617849.5	1549.7	0	90	119.5
2010-2011	WLC-111	411391.8	4618032.5	1555.4	0	90	73.8
2010-2011	WLC-112	411268.6	4617791.5	1544.5	43.2	89.4	57.0
2010-2011	WLC-113	411457.9	4618029	1556.7	0	90	77.7
2010-2011	WLC-114	411328.9	4617789.5	1544	0	90	60.0
2010-2011	WLC-115	411391.4	4617967.5	1551.7	0	90	85.3
2010-2011	WLC-116	411394.8	4617909.5	1548.9	0	90	101.2
2010-2011	WLC-117	411392.8	4617792	1542.8	0	90	61.6
2010-2011	WLC-118	411387.7	4617731.5	1538.8	0	90	40.3
2010-2011	WLC-119	411268.4	4617731.5	1541.9	0	90	27.7
2010-2011	WLC-120	411204.8	4617732.5	1545.4	0	90	25.0
2010-2011	WLC-121	411153	4617737	1547.6	0	90	21.9
2010-2011	WLC-122	411329.3	4617677	1537.6	229.5	89.8	34.1
2010-2011	WLC-123	411518.9	4617919	1550.4	0	90	58.6
2010-2011	WLC-124	411520	4617966.5	1553.9	0	90	57.0
2010-2011	WLC-125	411154.9	4617791.5	1550.6	0	90	46.4
2010-2011	WLC-126	411582.1	4617960	1553.1	0	90	58.6
2010-2011	WLC-127	411089.2	4617790	1549.1	0	90	49.4
2010-2011	WLC-128	411578.3	4617913	1550.2	217	89.6	61.6
2010-2011	WLC-129	411087.3	4617848	1551.7	0	90	90.7
2010-2011	WLC-130	411581	4617856.5	1547.1	0	90	61.6
2010-2011	WLC-131	411153.5	4617849	1552.6	0	90	99.4
2010-2011	WLC-132	411574.3	4617798.5	1543.2	0	90	69.2
2010-2011	WLC-133	411517	4617784.5	1542.3	0	90	69.2
2010-2011	WLC-134	411519.5	4617730.5	1539.1	0	90	76.9
2010-2011	WLC-135	411581.8	4617734	1539.5	270.2	89.4	81.4
2010-2011	WLC-136	411211.3	4617787	1547.5	0	90	38.4
2010-2011	WLC-137	411096.9	4617913	1555.5	230.9	89.6	90.6
2010-2011	WLC-138	411519.2	4617673	1535.6	0	90	64.7
2010-2011	WLC-139	411157.4	4617967.5	1556.4	0	90	96.7
2010-2011	WLC-140	411574.1	4617676	1536.3	0	90	92.0
2010-2011	WLC-141	411646.6	4617793	1542.2	0	90	72.2
2010-2011	WLC-142	411099	4617966.5	1558	0	90	92.0
2010-2011	WLC-143	411645.6	4617851	1546.1	0	90	67.7
2010-2011	WLC-144	411153.1	4618028.5	1559.4	0	90	96.6

Table 10.2: Summary of Drillholes at the Western Lithium Project Area

Drilling Campaign	Hole ID	Easting	Northing	Elevation (m)	Bearing	Plunge degrees	Total Depth (m)
2010-2011	WLC-145	411206.6	4618201	1564.3	0	90	88.9
2010-2011	WLC-146	411088.5	4618030	1561.1	0	90	93.6
2010-2011	WLC-147	411153.2	4618213	1565.2	0	90	87.2
2010-2011	WLC-148	411154	4618090.5	1562.2	28	89.6	89.6
2010-2011	WLC-149	411095.9	4618221	1569.2	0	90	88.9
2010-2011	WLC-150	411210.3	4618154	1561	0	90	78.4
2010-2011	WLC-151	411150.1	4618161	1565.2	0	90	92.0
2010-2011	WLC-152	411642.6	4617919.5	1550.3	0	90	55.8
2010-2011	WLC-153	411088.4	4618163	1566.3	0	90	88.1
2010-2011	WLC-154	411698.1	4617909	1548.7	0	90	51.0
2010-2011	WLC-155	411701.4	4617858.5	1545.8	0	90	67.7
2010-2011	WLC-156	411769.4	4617912.5	1548.2	0	90	61.0
2010-2011	WLC-157	411398.8	4617669.5	1534.1	0	90	40.2
2010-2011	WLC-158	411643.1	4617955.5	1552.2	0	90	55.4
2010-2011	WLC-159	411828.6	4617848.5	1541.6	0	90	76.8
2010-2011	WLC-160	411840.3	4617909.5	1544.9	0	90	70.8
2010-2011	WLC-161	411893.5	4617851	1540.6	0	90	82.9
2010-2011	WLC-162	411892.1	4617905.5	1543	0	90	71.3
2010-2011	WLC-163	411825.7	4617729	1534.9	0	90	86.0
2010-2011	WLC-164	411954.9	4617790.5	1535.8	0	90	104.2
2010-2011	WLC-165	411827.9	4617669	1531.6	0	90	70.7
2010-2011	WLC-166	411890.5	4617799	1537.5	0	90	89.0
2010-2011	WLC-167	411827.2	4617603	1527.9	0	90	101.0
2010-2011	WLC-168	411825.4	4617791	1538.3	0	90	87.5
2010-2011	WLC-169	411828.2	4617548.5	1525.1	0	90	104.2
2010-2011	WLC-170	411764.8	4617787	1538.9	0	90	79.9
2010-2011	WLC-171	411764.2	4617544	1525.1	0	90	95.1
2010-2011	WLC-172	411766	4617670	1532	0	90	92.1
2010-2011	WLC-173	411701.2	4617796.5	1540.4	0	90	75.3
2010-2011	WLC-174	411769.8	4617729.5	1535.6	0	90	83.0
2010-2011	WLC-175	411576.5	4617543.5	1527.2	0	90	86.0
2010-2011	WLC-176	411708.5	4617733.5	1536.6	0	90	82.9
2010-2011	WLC-177	411580.3	4617615	1531.3	0	90	88.8
2010-2011	WLC-178	411706.5	4617676	1533.2	0	90	87.5
2010-2011	WLC-179	411699.1	4617608.5	1529	0	90	92.0
2010-2011	WLC-180	413491.2	4617607.5	1497.3	0	90	140.9
2010-2011	WLC-181	412890.7	4618001	1534.3	0	90	30.7
2010-2011	WLC-182	412897.8	4618347	1549	0	90	30.5
2010-2011	WLC-183	413494	4617230.5	1482.2	0	90	30.5

Table 10.2: Summary of Drillholes at the Western Lithium Project Area

Drilling Campaign	Hole ID	Easting	Northing	Elevation (m)	Bearing	Plunge degrees	Total Depth (m)
2010-2011	WLC-184	411522.6	4617093	1521.5	0	90	37.2
2010-2011	WLC-185	413486.4	4618333	1525.5	0	90	95.0
2010-2011	WLC-186	411332.3	4618120	1558.3	0	90	86.0
2010-2011	WLC-187	411633.5	4617543	1526.7	0	90	101.3
2010-2011	WLC-188	411514.2	4618021	1555.8	0	90	61.6
2010-2011	WLC-189	411640.2	4617609	1530.4	0	90	96.9
2010-2011	WLC-190	411272	4618155	1558.8	0	90	74.5
2010-2011	WLC-191	411637.4	4617664.5	1533.5	0	90	98.1
2010-2011	WLC-192	411330.4	4618029	1552.9	0	90	92.1
2010-2011	WLC-193	411696.6	4617551.5	1525.6	0	90	104.3
2010-2011	WLC-194	411329.7	4617970.5	1549.4	0	90	110.3
2010-2011	WLC-195	411442.3	4617853.5	1543.8	0	90	70.7
2010-2011	WLC-196	411259.6	4617969.5	1550.9	0	90	116.4
2010-2011	WLC-197	410476.7	4617924	1570.2	0	90	92.4
2010-2011	WLC-198	411327	4617845.5	1544.3	0	90	79.8
2010-2011	WLC-199	410491.4	4617987.5	1570.9	0	90	115.7
2010-2011	WLC-200	410729	4617282.5	1525.3	0	90	73.8

The latest drilling campaign began in January 2010 and concluded in June 2011. All core was logged by a geologist on-site who recorded hole ID, easting, northing, elevation, total depth and a lithologic description.

Assays for drillholes bored prior to January 2010 (WLC-01-WLC-37) had analytical work done by American Assay Labs (AAL) in Nevada. The data from AAL's analytical work was not used in the model. These holes had assays for lithium repeated during the latest campaign. These re-assays, as well as all subsequent assays, were performed at ALS Minerals in Reno, Nevada. The assays included results for lithium, potassium and sodium and a suite of common elements of interest, while the re-assays were analyzed for lithium only. Samples selected for assay were chosen by the geologist based on lithology and color.

The drillhole spacing was prescribed by the geostatistical methods which included variography to determine optimal spacing for inclusion in Inferred, Indicated and Measured categories. The geologic model included intervals from holes WLC-01 thru WLC-200 (excluding WLC-180, WLC-183 and WLC -185) and WSH-01 and WSH-02 for a total of 197 holes and a total length of approximately 18,500 m. Chevron drillholes were excluded from consideration in the model. Please refer to Section 14.

All drillholes included in the resource estimate were drilled essentially vertically (88.8 to 90 degrees) with the exception of one hole (WLC-58). All mineralization thicknesses recorded in boreholes are treated as true thicknesses.

All core drillholes were drilled using standard core drilling techniques by Marcus & Marcus Exploration Inc. Holes WSH-1 and WSH-2 were drilled by Boart Longyear Company with a sonic drill. Core was

stored in a secure workshop while being processed and then locked in CONEX containers or a warehouse after sampling was completed.

10.1.2 Accuracy and Reliability of Results of Drilling

The author relied on Reserva to ascertain the accuracy and reliability of the most current drilling program. Based on field and core shack observations, Reserva found no factors in the drilling techniques, core recovery and sample collection procedures that would materially affect the accuracy and reliability of the results. The procedures and methodologies are consistent with industry standards, and the data is considered to be acceptable for estimation purposes.

10.2 STAGE II

The information in this section is based in part on the corresponding section of the URS report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project*, *Humboldt County, Nevada, USA*, effective date December 31, 2009.

10.2.1 Drilling

Cuttings from rotary percussion drill holes drilled by Chevron in 1977 were analyzed for lithium. Those drill holes confirmed the presence of interesting grades of lithium hosted by a massive, green claystone within the moat sediment section.

Between 1979 and 1987 Chevron drilled a total of 34 rotary percussion holes, plus 227 core and RC holes for initial exploration, to develop lithium targets, and to conduct extensive metallurgical testing. Those holes were drilled to test the thickness of the clays, to obtain samples of the clay for engineering analysis, and to further investigate the lithium resource potential. Results were encouraging with respect to the level and consistency of the lithium contained by the clays.

WLC has recovered some of the historical core and has been in the process of re-logging that core. Of the total 283 holes that appear in the database, 227 have lithium assay data. There are indications that an additional six holes were drilled, but there are no locations for those holes. The record indicates that 213 rotary percussion and 15 core holes were drilled to test the lithium mineralization between 1980 and 1984. These drill procedures were standard for the industry at that time.

A total of 38 additional HQ (63.5 mm) diameter core holes were drilled by WLC in the Stage II Lens area during 2009 (Figure 10.2). Two of the Chevron core holes, MC-84-81C and MC-84-90C, were re-analyzed by WLC during the 2009 campaign, and thus have grades for all four elements of interest, Li, K, F, and Na.

Core or diamond drilling provides better geological information and quality of sample compared to percussion and RC methods. GSI agrees with previous recommendations that only core drilling should be used for these deposits. Although reverse circulation (RC) drilling can reduce drilling costs, at PCD RC drilling was shown to produce biased assay results. WLC did not drill any RC holes at Stage II Lens.

10.2.2 Logging

Chevron core was collected from the drills twice a day and descriptively logged by geologists at Chevron's field camp. Chip samples from rotary drills were logged at the drill. Two composite samples

were collected every five feet and bagged. The geologist logging the hole made a chip board at the drill site. The chip boards consisted of drill cuttings glued to a 1-inch x 4-inch board whose vertical scale was 1in = 10ft. Lithological logging of both core and chip samples stressed lithologic units, their contacts, mineralization, alteration and brecciation.

WLC core is collected at the end of each shift and transported back to the Orovada field office. Core is cleaned and logged for lithology, oxidation, alteration, and core recovery. All core is photographed with high resolution digital cameras, and the samples are stored in a lockable steel storage container.

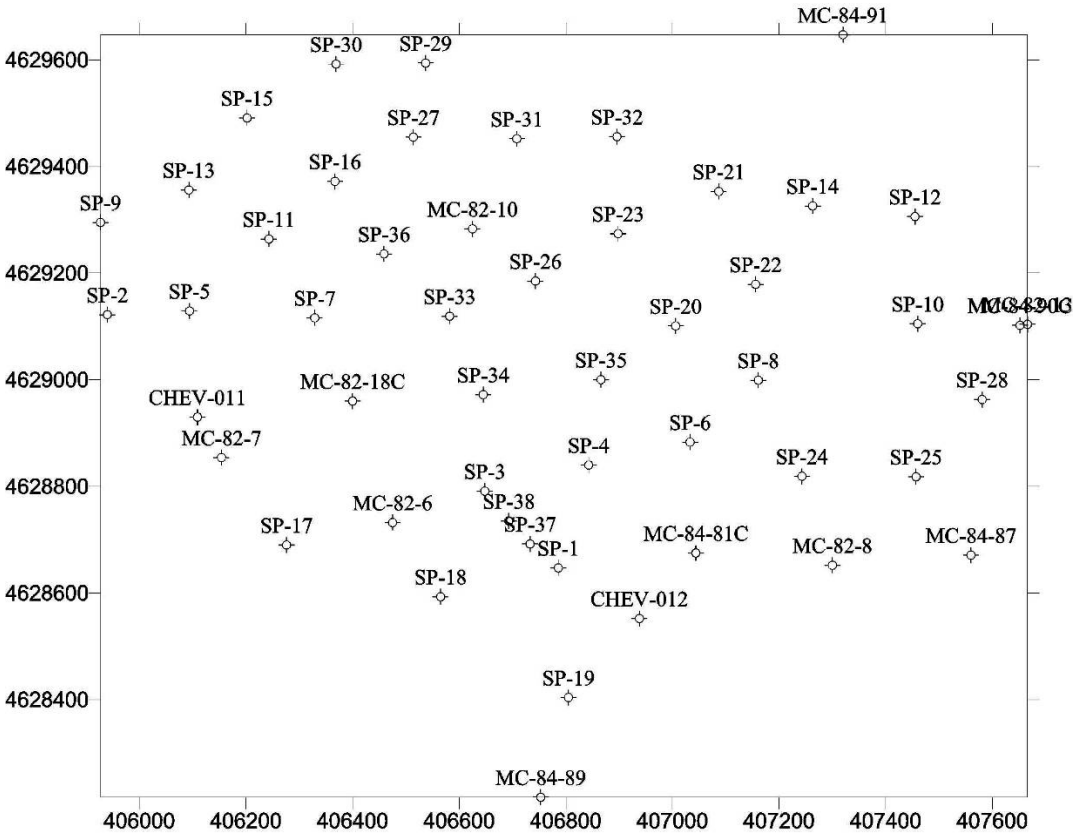


Figure 10.2: Stage II Lens Drill Hole Location Map.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 STAGE I

ALS Minerals (ALS) of Reno, Nevada, was used as the primary assay laboratory for the WLC Stage I Lens drill program. ALS is an ISO 9001:2000 certified laboratory and an ISO/IEC 17025-certified Quality Systems Laboratory, and participates in the Society of Mineral Analysts round-robin testing.

The quality assurance-quality control (QA-QC) protocols used during the Stage I Lens sampling and analytical program were developed by Dr. Barry Smee of Smee & Associates Consulting Ltd, an international specialist in QA-QC procedures who is familiar with the requirements of NI 43-101. Mr. Tim Carew of Reserva International (Reserva), a consulting firm that specializes in ore reserve estimation and reporting, evaluated the QA-QC procedures used during the program.

11.1.1 Sample Preparation

After the drilled core was brought to the core shed from the field, the boxes of core were logged, photographed, cut and sampled by WLC employees and consultants. The length of the assay samples is determined by the geologist by lithology and averaged 1.46 m. The core was cut in half with diamond blade saws and the right half bagged for sampling. For duplicate samples, one half of the core is cut in half again and the two halves are bagged and sampled separately to test sampling and assay precision. Each sample was assigned a unique identification number to ensure security and anonymity. The samples were picked up by ALS in trucks that arrive from Winnemucca or are delivered to ALS in Winnemucca or Reno by WLC employees.

WLC submitted a total of 11,157 samples to ALS, of which 9,884 were mineralized samples. Randomly inserted in the sample stream were 1,273 QA-QC samples, which represent 11 % of the total assays. The QA-QC samples include blanks to test for contamination, high and low grade lithium standards to test for accuracy and duplicates to test for precision.

At ALS, the samples were dried at a maximum temperature of 60°C. The entire sample was then crushed with a jaw crusher to 90% passing a ten mesh screen. Nominal 250 gram splits were taken for each sample using a riffle splitter. This split is pulverized using a ring mill to 90% passing a 150 mesh screen. These preparation procedures were deemed appropriate by Reserva for the mineralized material from the Stage I Lens.

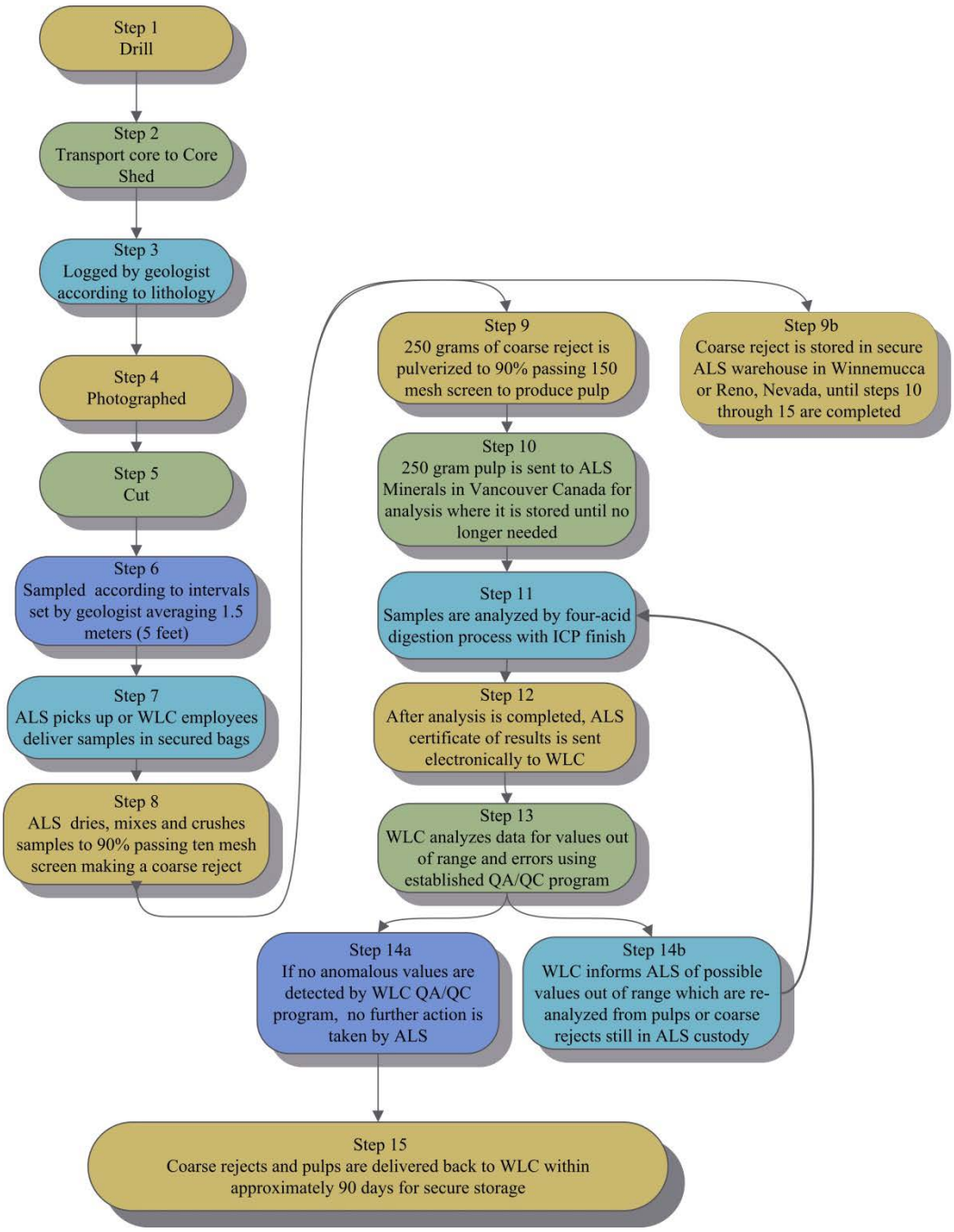


Figure 11.1: Sample Preparation Flow Diagram

11.1.2 Analysis

Inductively coupled plasma mass spectroscopy (ICP-MS) combined with atomic emission plasma spectroscopy (ICP-AES) are used to provide sensitive analytical capability. This combination is used to provide high precision analyses for precious metal exploration and reconnaissance programs. Reserva deems these methods to be appropriate and applicable to the Stage I Lens drill samples.

Following standard four-acid digestion, ALS's ME-MS61 analytical package initially analyzes the samples using ICP-AES to ensure that elevated metal concentrations were not present which would interfere with ICP-MS analyses. Along with a full geochemical suite, this method was used to analyze the three elements of commercial interest to the Project: lithium, potassium and sodium.

11.1.3 Density

WLC submitted 25 core samples from the Stage I Lens program to ALS for density determination. Density was measured by two methods: 24 samples were tested using the paraffin-coated method (ASTM Designation C914-95), while one sample was tested using the method described in ASTM C127-04, a test for aggregate materials. Both are standard methods. The material was weighed, the weights recorded and then the samples were dried at 75º C for 24 hours. The dry material was weighed again and the weight recorded. A rock density test was then performed on the dry material using a wax immersion procedure. Table 11.1 shows the dry density values that were used in the resource model.

Table 11.1: Average Density Values Used in the Resource Model

Lithology	Average of Density Determination (tonnes per m^3)
Alluvium	1.52
Claystone	1.79
Basalt	2.51
Volcanics	1.96

Note that dry density is the standard reference for density measurements for industrial minerals.

11.1.4 Quality Control

The QA-QC protocol included analyses of blank, duplicate and standard samples.

The QA-QC program was set up so that at least 10% of all samples are checks or tests against the laboratory assays. ALS analyzed samples in batches of 40, with six of those samples being ALS standards and controls. In addition, WLC randomly inserted two standards (one low grade and one high grade lithium), one duplicate and one blank in every 34 samples. Approximately 12% of all WLC samples were QA-QC checks.

Adhering to the QA-QC program developed by Dr. Smee (2010), WLC identified a small number of assay samples from ALS that were below the criteria established within acceptable limits. These QA-QC elements included core sample duplicates; high and low grade standards for lithium and blanks of dolomite. The pulps or coarse rejects of these 76 samples were re-tested and analyzed by ALS, including an average of 10 samples above and below the questionable result. ALS compiled reports and found that there were a few samples that were mislabeled by the lab. Certificates were re-issued by the assay

company to correct for the sample switches. Generally, the remaining QA-QC samples delivered assay results that were similar to the original results, which indicate possible errors in sampling, handling and processing. In total, 6% (76 of 1,273) of the QA-QC samples did not conform to the established criteria.

During the QA-QC process undertaken by WLC, an error was observed in the high grade lithium standards where 17% of the standards had a higher ppm rate than was expected by three standard deviations (SD). ALS began using a new higher grade lithium standard to compensate for the discrepancy. WLC re-assayed the highest 16 lithium values for drillholes WLC-01 through WLC-37 and WLC-40 through WLC-200. Similar samples were sent to both ALS Reno (for a blind repeat test) and Activation Laboratories (Actlabs), Ancaster, Ontario, Canada, for lithium analysis. Of the 16 re-assays, 14 had lower and two had higher lithium values, with the largest difference being 11% less. The overall difference of the original and the repeat assay values for ALS was lower by 5%. The difference between the ALS re-assays and Actlabs was 3% lower. Dr. Smee concluded that the overall deposit estimates may be lower by at most 2-3%, which is considered within industry standards.

As an added QA-QC measure, Actlabs was chosen to run assays on duplicate pulps produced by ALS for comparison to ALS assay results. A total of 112 random pulp samples were sent to Actlabs in April 2011. The data analyzed by Dr. Smee was found to comply with industry standards, being within 3% of the assay values returned by ALS.

11.1.4.1 Blank Samples

Blank samples composed of dolomite are used to check for possible contamination during sample preparation and analysis. The dolomite was chosen because it is known to have low lithium content and is therefore a good indicator of contamination from one sample to another during crushing and assaying. The raw dolomite is not an established laboratory standard and was obtained from a mine in Winnemucca, Nevada. The warning limit of 100 ppm was determined by Dr. Smee as five times the average concentration of the blank of 20 ppm lithium. There were 314 blanks analyzed for the Stage I program or approximately 3% of the total number of assays sent for analysis. Sixteen blank samples reported anomalously high lithium. All non-conforming samples were submitted for re-run (See Section 11.1.5 for details).

The practical detection limit for both ICP-MS and ICP-AES was reported at 0.2 ppm by the lab. Since the range of lithium values that WLC considers to be the low range of economic interest (1,000 to 2,000 ppm) is 10 to 20 times the practical detection limit, Reserva does not consider this to be a significant issue. Figure 11.2 shows blank assay results against lithium values represented on the y-axis. Non-conforming samples (i.e., those above the red 100 ppm limit line) were re-analyzed by ALS. These results were found to be accurate.

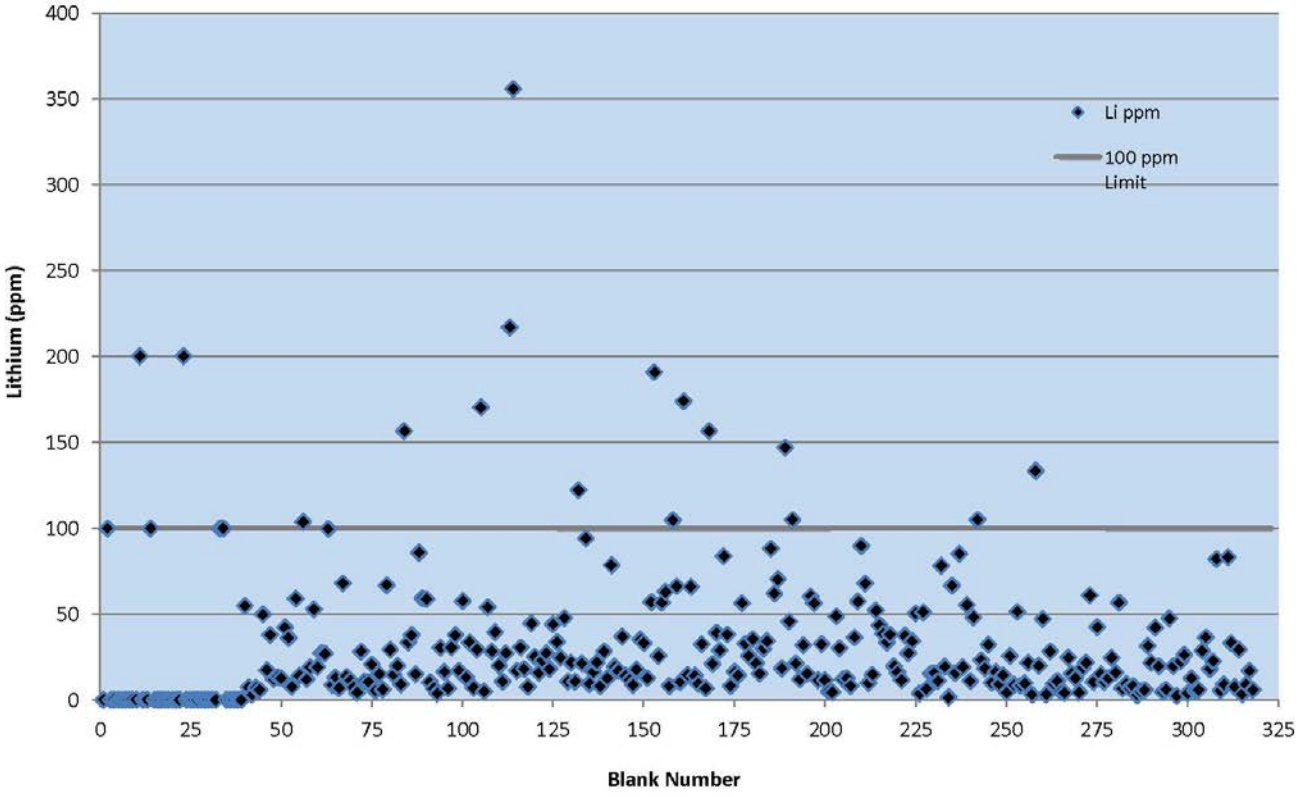


Figure 11.2: Lithium Concentration of Blanks

(Gray line indicates the warning limit for lithium)

Five percent of the blanks (16 of 314 samples) were above the WLC QA-QC warning limit. The samples were re-run by ALS and, again, determined to be accurate values. High values may indicate contamination in the crushing process or an anomalously high concentration of lithium in the given sample. Reserva considers the amount of contamination suggested by the blanks analyses to be acceptable.

11.1.4.2 Duplicate Samples

Pulp samples for duplicate analyses were prepared in an identical manner as the non-duplicate samples. A total of 316 sets of duplicates were submitted "blind" to ALS. The lithium concentrations for these duplicates are shown in Figure 11.3.

Duplicate samples provide information about the precision of the sampling and analytical procedure. Precision can be estimated by calculating the average relative error of the duplicate pairs. WLC considers the duplicate assay precision to be inaccurate if the average relative error between two duplicate samples is 500 ppm or more.

Twenty-two outliers (7% of the duplicate sets) did not meet the relative error criteria and were re-examined by ALS. The re-analyzed values were within 5% of the original value. Poor duplicate replication can indicate lack of precision in all phases of the sampling procedures. Potential causes include errors introduced during sampling (i.e., mislabeled sample, splitting) or during analytical testing. Figure 11.3 shows the laboratory performance results for duplicate samples.

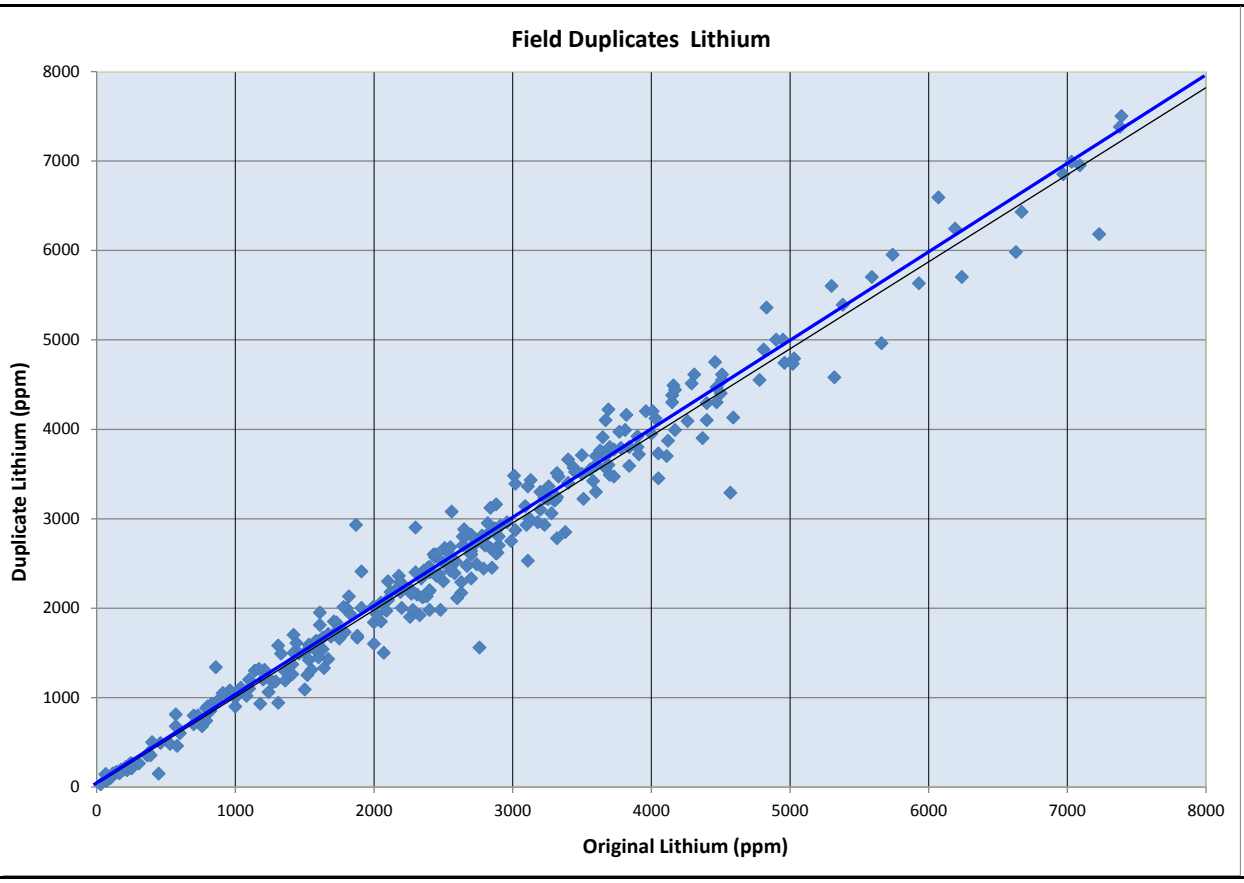


Figure 11.3: Lithium Content of Field Duplicates

Note: The heavy blue line represents 1:1 equivalency. The thin line is a linear regression curve that shows near 1:1 equivalency for all duplicate pairs.

Overall, duplicates show good correlation to original samples.

11.1.4.3 Standards

Two standards consisting of lithium-bearing clays from the Project area were used for the Stage I Lens drilling program: a low-grade standard (LG) with a certified value of 3,526 ppm lithium, and a high-grade standard (HG) with a certified value of 4,470 ppm lithium. These values were based on round-robin results from four analytical laboratories: Kappes, Cassiday & Associates, Reno, NV; ALS Minerals, Reno NV; Acme Labs, Vancouver, British Columbia, Canada; and ActLabs, Ancaster, Ontario, Canada.

To certify the low and high grade standards, a round-robin was completed in June 2010 in which ten standards of each grade were sent to six labs for testing. The resulting assays were analyzed and calculated to determine an average lithium, potassium, sodium and fluorine value for each standard. The results from two of the labs, seen as two clusters at bottom left in Figure 11.4 were discarded because they were not deemed to be compatible with the analytical methods employed. The average of the four labs that gave similar analytical results was used to calculate the certified values.

Three hundred and nine LG and 320 HG standards, representing 6% of the entire sample program, were inserted randomly into the sample stream to evaluate analytical accuracy. These standards were used

in the Stage I Lens exploration work and show lithium data to be reasonably accurate. Figure 11.4 and Figure 11.5 show laboratory performance with respect to the two standards.

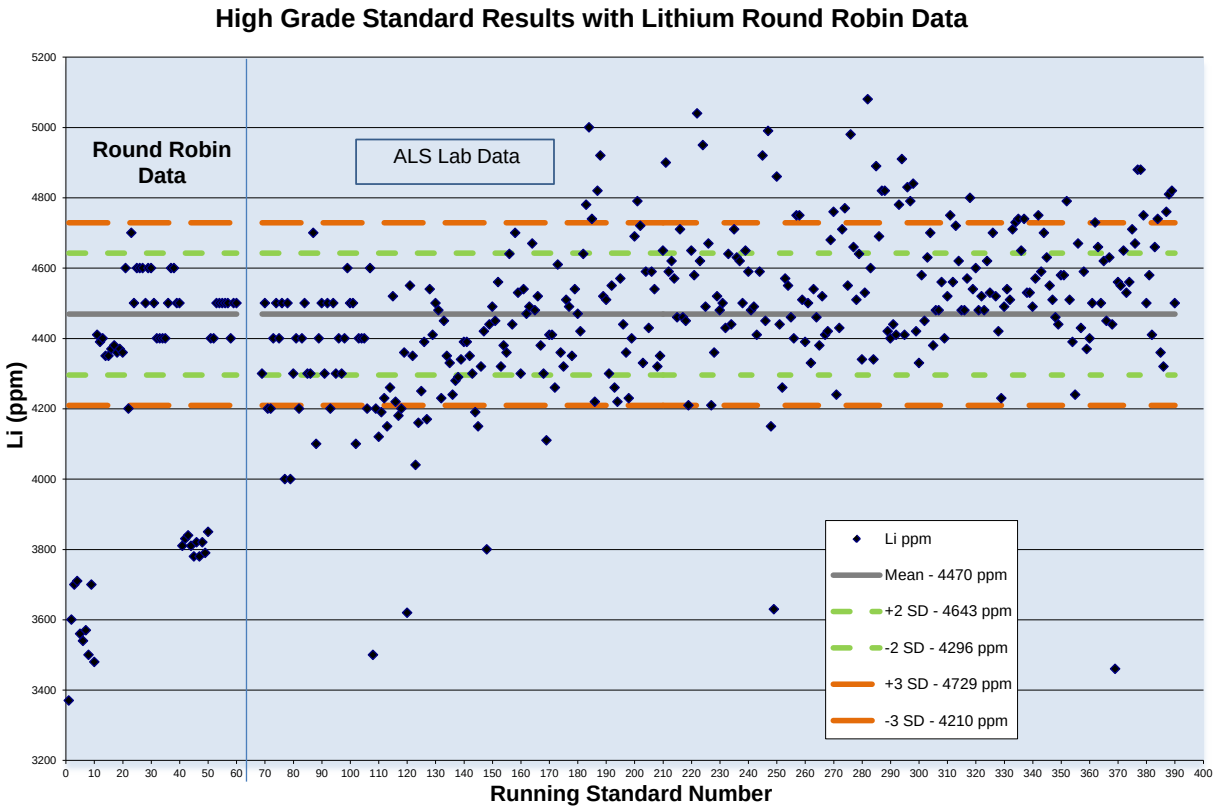


Figure 11.4: High Grade Standard Results with Lithium Round Robin Data

Note: As determined by four labs (round robin) on the left and by ALS on blind standards inserted in drill sample batches (lab data) on the right.

By identifying results that deviate more than three standard deviations (SD), ALS has developed in-house controls to better evaluate the lithium samples. Figure 11.4 shows that the data was trending below the -3 SD line at the beginning of the program. By identifying these samples that deviate with the WLC QA-QC program and having ALS re-analyze them, the assays became more accurate. In the middle of the program, the assays began to drift above the +3 SD line. The upward drift has been corrected by introduction of a new higher grade lithium standard that ALS has recently inserted into their QA-QC program. The low grade lithium standard did not drift above and below the 3 SD lines because the lithium ppm of the standards that the laboratory used was close to WLC lithium standards from the beginning.

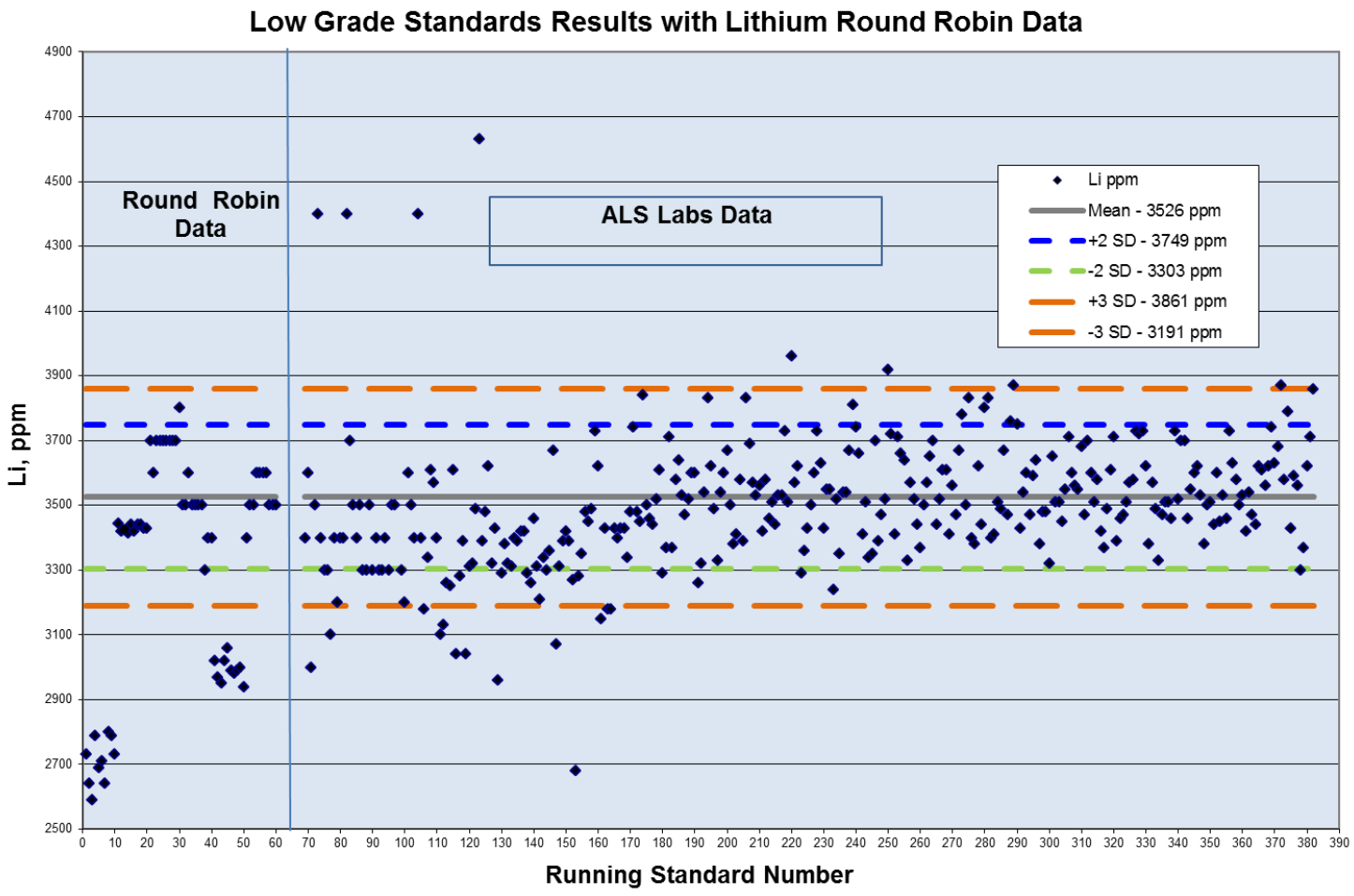


Figure 11.5: Low Grade Standard Results with Lithium Round Robin Data

Note: As determined by analyses by four labs (round robin) on the left and by ALS on blind standards inserted in drill sample batches (lab data) on the right.

11.1.5 Discussion of QA-QC

When samples that deviated from the accepted criteria were identified, these samples as well as a batch of 10 to 20 samples that are run before and after that sample, were re-analyzed to check for accuracy. Samples were re-analyzed using the pulps or coarse rejects that had been securely stored at the ALS warehouses in Winnemucca or Reno. Re-analysis values generally reported similar lithium ppm results compared to the original assay values with a margin of difference between zero and ±10%.

The WLC QA-QC program used standard samples obtained from the Project area, duplicate pulp samples and laboratory blanks to determine the precision and accuracy of the laboratory results. Dr. Smee monitored the data throughout the Stage I Lens drilling program. A final review of the data was performed by Dr. Smee on August 20, 2011 and he concluded: "*The Kings Valley analytical data appears to conform to the requirements of NI 43-101 Best Practices. The Li data is accurate and free from contamination. The sampling precision is within Industry Best Practices expectations.*"

Reserva reviewed the data and concurs that both the precision and accuracy of lithium analysis for the Stage I Lens are acceptable.

The total number of blank, duplicate and standard samples analyzed by the laboratory was 11% of the total samples assayed. There are additional checks that are being recommended for future drilling programs. These include:

- Check coarse duplicates (after first crush, usually -10 mesh material)
- Send pulp and coarse duplicates to a second laboratory
- Add a standard in the 1,500 to 2,500 ppm Li range (to ensure adequate accuracy around the presumed economic cutoff grades)
- Add of a potassium standard (obtained from the Stage I area)

Finally, Reserva recommended that the protocols and procedures for QA-QC be compiled and made part of an overall QA-QC document for the Project, to include field sampling practices, sample preparation and assaying protocols, laboratory QA-QC and database validation.

11.1.6 Security

WLC stores core from the current drill program in a secure core logging/sampling facility that is locked when authorized personnel are not present and until they are shipped to the assay laboratory.

Current WLC security is considered adequate and meets industry standards. The possibility of significant tampering with samples in the custody of WLC is unlikely because of the amount of lithium required to significantly change the tenor of core or cuttings samples. Reserva recommends that all future sampling programs employ an expanded sample security protocol that includes formal chain of custody documentation. The security procedures should be part of a larger QA-QC program to ensure consistent practices along the entire sequence of processes, from the field to the building of the electronic database.

11.2 STAGE II

Most of the information presented in this section is based on the corresponding section of the URS report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project*, *Humboldt County, Nevada, USA*, effective date December 31, 2009. The text has been standardized and updated for this report.

11.2.1 Sample Preparation – Chevron

Few records of sample preparation procedures exist for the project. Hand-written notes indicate that core was split and one-half was archived. The other half was crushed in a jaw crusher and then split "until a single representative sample bag" was obtained. The mass of the sample is not specified. The remainder of the split was retained in labeled bags.

Chip samples were split and one-half retained. The second split was prepared as above. The mass or granulometry of the final analytical split is not specified nor has AMEC located records of those data.

Details of crushing, splitting, and pulverization are not provided. During the time covered by this Cone Geochemical Inc. (the primary analytical laboratory) routinely dried the samples at 120°C, crushed to 10#, split 150 g minimum with a riffle splitter, and pulverized to 150 # with a steel ring and puck mill unless otherwise directed by their customer. There is no record of variance from this procedure for these samples.

The record suggests that sample splitting and bagging was performed by Chevron employees and that the entire sample was sent to the analytical laboratory for final preparation and analysis. There is no indication in the record that company employees were involved with final sample preparation.

11.2.2 Sample Preparation – WLC

WLC has used ALS Chemex (ALS) of Reno, Nevada as their primary assay laboratory for their Stage II Lens 2009 drill campaign. ALS is an ISO 9001:2000 certified laboratory, also an ISO/IEC 17025-certified Quality Systems, and participates in the Society of Mineral Analysts round robin testing.

Sample preparation at ALS begins with drying the sample at a typical 110-120 °C. Then, the entire sample is crushed with a roll crusher to 90% passing -6 #. A nominal 250 g split is taken using a riffle splitter, which is pulverized using a ring mill to 90% passing -150 #. These preparation procedures are deemed appropriate for the mineralized material from Stage II Lens.

11.2.3 Analysis – Chevron

Samples were analyzed for Li at Cone Geochemical, Inc. (Cone) and Skyline Labs, Inc. (Skyline) by dissolving 0.1 g of sample in boiling HF-HNO_3-$HClO_4$ boiled to dryness. The residue was then dissolved in 6N HCl, diluted to 100 ml with H_2O, and read on an atomic absorption spectrometer (AA). This four acid digestion is an industry standard procedure. Skyline experimented with a number of other procedures, but the record indicates that they used the four acid digestion followed by AA determination as the primary analytical procedure. In most cases, Li was reported as ppm, but in some cases, Li_2O was reported as percent. In the database, Li_2O was converted to Li (ppm).

A small number of samples were analyzed for As, Sb, Au, Ag, Zn, Mo, and MgO. Arsenic, Antimony and Au were determined by digestion in aqua regia followed by AA. Mo, Zn, Ag, and MgO were determined by four acid digestion followed by AA. These are standard analytical procedures.

There were no available certifications for Cone and Skyline at the time the work was performed; however, both labs were well respected and widely used by the mining and environmental industries.

11.2.4 Analysis – WLC

ALS used Inductively Coupled Plasma Mass Spectroscopy (ICP-MS) combined with conventional atomic emission plasma spectroscopy (ICP-AES) to provide extended sensitivity analytical capability. While originally designed for very low, geochemical assays for metal exploration and reconnaissance programs, GSI believes that these methods can be applied to the Stage II Lens samples.

After a 4-acid digestion, ALS's ME-MS61 analytical package initially analyzes the samples using ICP-AES to pre-screen them, ensuring that no elevated metal concentrations are present. Elevated metal concentrations (defined as >1% of an individual base metal, or >3% cumulative) cannot be introduced to a plasma mass spectrometer without causing cross contamination. Samples showing elevated metal

concentrations are not analyzed by ICP-MS but will only have ICP-AES data reported. Along with a full geochemical suite, this method was used to analyze three of the four elements of interest to the Stage II Resource Estimate, lithium, potassium, and sodium. Several other elements were also analyzed and reported, but they are beyond the scope of this Technical Report.

Fluorine was analyzed with a specific fusion method, which uses sodium carbonate and potassium nitrate to solubilize it in an alkaline environment to prevent volatilization. The method was calibrated differently depending on the concentration: ALS's F-ELE81a method is used when the concentration of F is less than 20,000 ppm; F-ELE82a is used for F concentrations greater than 20,000 ppm. Both methods are standard for the industry.

11.2.5 Density

WLC submitted 26 core samples from the Stage II Lens to ALS Chemex in Reno Nevada for density determinations. Density was determined by two methods; 21 samples were tested using the paraffin coated method (ASTM Designation C914-95), while one other sample was tested using the method described in ASTM C127-04, test for Aggregate materials. Both are standard methods. The remaining 4 samples were lost due to disintegration when applying the wax coat or when hydrating.

The material received was weighed, the weight reported and then dried at 75°C for 24 hours. The dry material was then weighed again and the weight reported. A rock density test was then completed on the dry material using a wax immersion procedure. Table 11.2 show the in-situ dry density values used in the resource model.

Table 11.2: Density Data Used for Resource Estimation.

Unit	Average Density Value (t/m^3)	Density Value used in the Resource Model
Alluvial	1.52	1.52
Claystone Average	1.95	1.96
Ash Average	1.99	
Fang. Average	1.91	
Silicified	1.94	
Bedrock Average	2.23	2.23

Density determinations were performed using standard procedures and are adequate for resource estimation. GSI recommends, however, that additional density data be collected during the next round of drilling.

11.2.6 Quality Control – Chevron

AMEC found four duplicate samples analyzed by Cone in the data provided. Those results suggest that the precision was adequate, but there are insufficient data to reach any meaningful conclusions.

Otherwise, no independent quality control measures were in place. During this time period, Cone normally analyzed a single duplicate sample and a single standard sample in each analytical batch, but

those data were not routinely provided to their customers and are thus not part of the record of this project.

These are the only results that can be identified as quality control results. This is typical of the time period although it was normal to send a number of samples to a second laboratory specifically for check assays. Relative to current industry standards, quality control for the Chevron analytical work on this project was substandard. The results were verified by drilling three twin holes in the PCD lens and comparing Chevron grades to WLC grades (see Section 11.2.8). Also, GSI compared globally the Chevron Li grades to the WLC grades globally for the claystone unit in the Stage II Lens, and found no significant differences.

11.2.7 Quality Control - WLC

Quality assurance-quality control (QA-QC) by WLC consists of standard samples, blank analyses, and duplicate analyses. Duplicate analyses were performed on pulp samples at ALS. Those samples were prepared and analyzed in the same batch as the original sample.

11.2.7.1 Blank Samples

Blanks are used to check if there is any possible contamination during sample preparation and analyses. There were 41 blanks analyzed for the 2009 Stage II Lens campaign, about 3% of the total number of assays sent for analysis.

There are some slightly anomalous values for Li when a practical detection limit 10 times the nominal detection limit is defined, in this Case 20 ppm. This value is also very close to the average Li value of all 41 blank samples. Given the range of values for Li that are considered mineralized is 50 to 100 times the returned Li values, GSI does not consider this to be a significant issue.

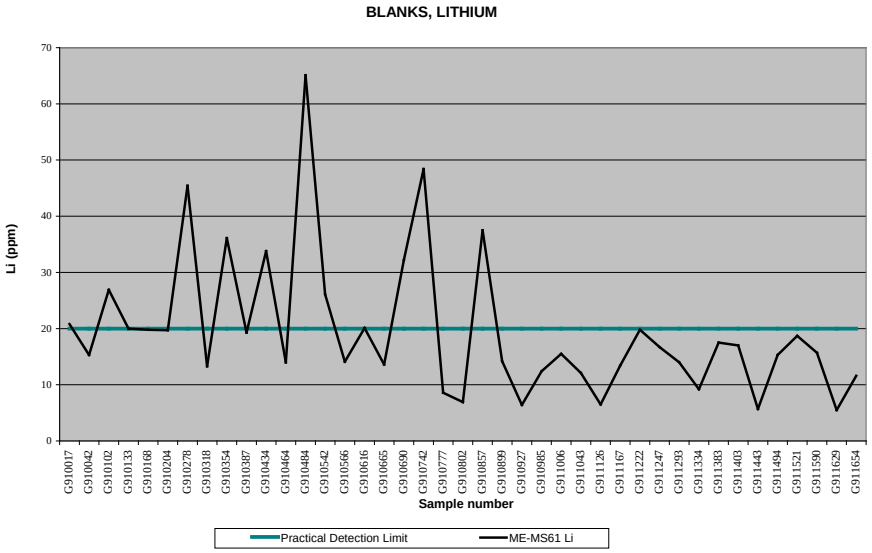


Figure 11.6: Blanks, Lithium.

The corresponding graphs for K, F, and Na are shown in Technical Reference F. In all cases, the same conclusions can be made: blank samples appear to indicate that there is little, if any, contamination in the analytical process. GSI considers results from the blanks to all be acceptable.

11.2.7.2 Duplicate Samples

Duplicate samples were run from the same pulps as the original assay. They provide information about the precision of the analytical procedure. Precision can be estimated by calculating the relative error of the duplicate pairs:

$$Relative\ Error = \frac{|S1 - S2|}{(S1 + S2)/2}$$

The cumulative frequency of the relative error (RE) is generally calculated for samples containing more than 20 times the laboratory-reported detection limit. Samples near the detection limit tend to result in high relative errors due to their low values. The 80th and 90th percentile relative error are reasonable measures often used to estimate precision. Table 11.3 shows both RE percentiles for all samples and for the four elements of interest, Li, K, F, and Na.

Table 11.3: Relative Errors from duplicate samples, Stage II Lens WLC drilling.

Element	80th Percentile RE (%)	90th Percentile RE (%)
Li	±11.1%	±13.1
K	±10.6%	±14.2
F	±15.3%	±19.5%
Na	±18.2%	±22.2%

Figure 11.7 shows the cumulative data frequency vs. RE for Li. GSI considers that the results from the duplicate samples are acceptable for Li and K, the two main economic drivers of the project; the precision of F and Na is lower than desired.

Sodium is not a major project driver, nor is the relative lack of precision of its samples (compared to the other elements) expected to impact project economics. In the Case of F, it does impact metallurgical processing. GSI believes that the relative low precision of the F samples is more related to the use of different methods in the analysis. Also, as shown in Section 14.2, Li and F have a very strong positive correlation, which suggests that the F sampled values could be checked or calibrated stochiometrically from Li values. The strong correlation suggests that the only source of F appears to be the same hectorite clay.

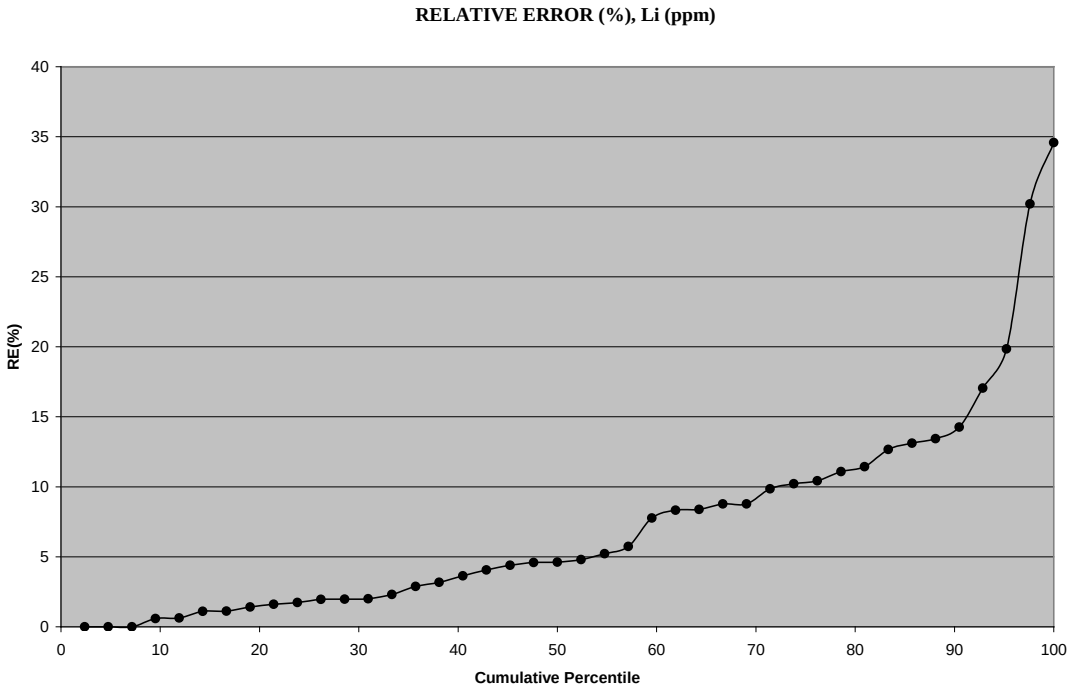


Figure 11.7: Cumulative Relative Error (%), Lithium.

Scatterplots and Cumulative RE plots for the other elements are shown in Technical Reference F.

11.2.7.3 Standards

WLC used two standards from Li-bearing clays found in the project area, one whose accepted value is 3,379 ppm Li, and the other 4,217 ppm Li. Those standards were used in the 2009 exploration work of the Stage II Lens and show that the Li data are reasonably accurate. Figure 11.8 and Figure 11.9 show the laboratory performance with respect to the two standards. Note that there is only sample (G911056) that is significantly lower than the standard, which was sent to reanalysis.

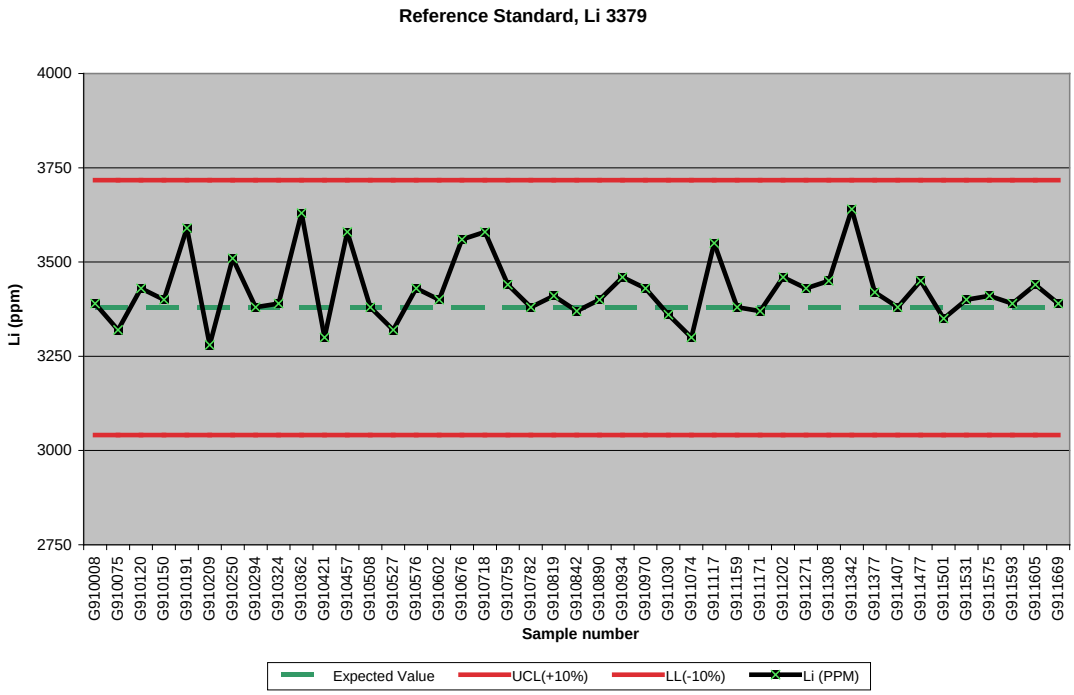


Figure 11.8: WLC Standard Li-3379, Stage II Lens assays.

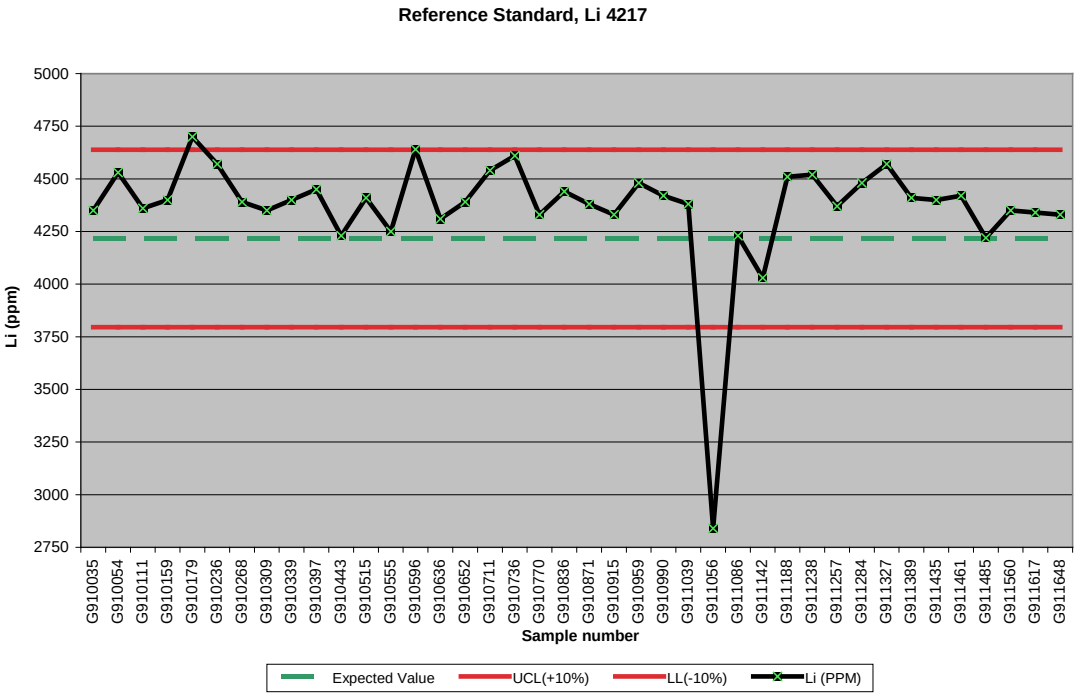


Figure 11.9: WLC Standard Li-4217, Stage II Lens assays.

11.2.8 Discussion of QA-QC

QA-QC for the Chevron data is substandard relative to current best practices. However; twin holes drilled by WLC at PCD largely confirmed the location and tenor of the mineralization; also, estimating the resources using Ordinary Kriging and the Chevron holes compared to the WLC holes only result in very similar estimates; in GSI's opinion, this validates at least globally that the Chevron data are adequate to use for future resource work.

WLC QA-QC utilizes standard samples, duplicate samples, and blanks to check the precision and accuracy of the laboratory results. GSI believes that both the precision and accuracy of Li and K analyses for the Stage II Lens is adequate. Likewise, the F and Na analysis are deemed adequate for resource estimation, although further improvements to increase data precision are suggested.

The total number of check samples analyzed by the laboratory was about 8% of the total samples assayed. There are additional checks that are being recommended for future drilling at the Stage II Lens, including checking coarse duplicates (after first crush, usually -10 # material); and sending pulp and coarse duplicates to a second laboratory. Adding also a standard in the 1,500 to 2,500 ppm Li range would allow ensuring adequate accuracy around the presumed economic cutoff grades, as well as adding standards for K and F, at least, all from moat sediment material from the same McDermitt caldera.

Finally, GSI recommends that the protocols and procedures for QA-QC be written down and made part of an overall QA document for the project, to include field sampling practices, sample preparation and assaying protocols, laboratory QA/QC, and database validation.

11.2.9 Security

11.2.9.1 Chevron

GSI is not aware of any security measures for samples from this project. Based on the time-frame in which the data were collected, it is reasonable to suspect, as AMEC did, that no formal security measures were in place. Because Li occurs in relatively high concentrations and the generally unavailability of Li compounds that could be used to tamper with the samples, AMEC believes that tampering with the samples is unlikely, and GSI agrees that this conclusion also applies to Stage II Lens material.

11.2.9.2 WLC

WLC stores core from the current drill program in a lockable core logging/sampling facility that is locked when no one is present. Samples are stored in a locked facility until they are shipped to the assay laboratory.

11.2.9.3 Discussion

Security during the Chevron drill programs was typical for the time period. Current WLC security is adequate. The possibility of significant tampering with samples in the custody of Chevron or WLC is unlikely because of the amount of Li required to significantly change the tenor of core or cuttings samples. GSI recommends that all future sampling programs have a rigorous sample security, including formal chain of custody documentation. This document should be part of a larger QA-QC document, to ensure consistent practices along the entire sequence of processes, from the field to the building of the electronic database.

12.0 DATA VERIFICATION

12.1 STAGE I

12.1.1 Data Verification Procedures

For resource estimation purposes Reserva compiled an assay and lithological database from assay compilations and summary geological logs supplied by WLC, in spreadsheet format. WLC maintains a tracking chart (Excel spreadsheets) that is used to match analytical data from ALS (provided electronically in the form of both Excel spreadsheets, and secured PDF assay certificates) to the intervals logged by the geologists, and referenced to duplicate sample tags (Sample ID) stapled into the core boxes. WLC also maintains a master chart to track and manage QA-QC samples, the data provided to Reserva was excerpted from this database. Reserva obtained the certified assay certificates for a sample of 10% of the assay intervals, chosen at random, for comparison with the assay data imported into the Reserva database. No discrepancies were noted in this comparison exercise. Reserva detected a relatively small number of inconsistencies in intervals in the import and data validation process, which were well below 1% of the total intervals and were corrected with WLC.

12.1.2 Drill Core and Geologic Logs

Reserva has reviewed the WLC drill core and logging procedures and finds them to be adequate for the study. Some inconsistent and duplicate geologic codes were discussed with WLC and rectified through the use of standard codes, both in ongoing logging and retroactive modification.

12.1.3 Topography

The topographic surface utilized in the estimate was provided by WLC in 3D DTM and contour format (DWG), and was based on aerial photography dated July 6, 2010. The flyover resolution was 0.35 m. Ground control was established by Desert-Mountain Surveying, a Nevada licensed land surveyor, using Trimble equipment. Field surveys of drillhole collars, spot-heights and, ground truthing was conducted by Mr. Dave Rowe, MXS, Inc., a Nevada licensed land surveyor, using Trimble equipment. A comparison of surveyed collars to the topography DTM highlighted inconsistencies for a small number of collars, which were investigated and rectified in conjunction with Mr. Rowe.

12.1.4 Verification of Analytical Data

Reserva has not undertaken any verification of analytical data in the form of independent samples.

12.1.5 Data Adequacy

Based on the various reviews, validation exercises and remedies outlined above, Reserva concludes that the data provided is adequate for use in the resource estimate.

12.2 STAGE II

The information in this section is taken from the corresponding section of the URS report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project*, *Humboldt County, Nevada, USA*, effective date December 31, 2009. The text has been re-formatted for this report.

Western Energy Development Corp. acquired the physical records of the project from Jim LaBret (now deceased), the owner of a number of claims covering uranium prospects in the area. Those records were subsequently transferred to WLC.

AMEC during the preparation of its initial Technical Report on the property (AMEC, 2008) completed extensive work regarding data verification. This included:

1. Comparing about 50% of lithological and assay data to the original documentation, which included laboratory certificates and summary graphic logs.

2. AMEC also collected a large sample from the Huber pit and chip from core boxes that were subsequently analyzed, confirming the presence and general grade of the samples.

Based on these tests, AMEC concluded that the grades indicated by Chevron are reasonable and that there is no reason not to rely on those data for this evaluation.

GSI did not consider necessary to repeat the process for the Stage II Lens, as it is part of the same deposit and it was surveyed using the same equipment and resulted in the same general conclusions with respect to the Chevron holes as for Stage I Lens.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 STAGE I

13.1.1 Introduction

WLC has continued process development for lithium extraction from the Kings Valley clay deposits that was originally based on the USBM work in the McDermitt caldera reported in 1988. Mineralogy of the Kings Valley resource is discussed in Section 7.1. The USBM studied several options and selected two with potential for further development, including the thermal process favored by Western Lithium. The USBM report was the basis for the URS *Kings Valley Project NI 43-101* Technical *Report, Preliminary Assessment and Economic Evaluation* (PAEE) issued January 22, 2010. Kappes Cassidy & Associates (KCA) was commissioned by WLC to perform metallurgical testwork in support of the PAEE that confirmed that the USBM process for recovery of lithium from the mineralized material is technically viable.

The metallurgical testwork commissioned by WLC for the PFS included programs specific to calcination and the evaporative crystallization process.

The ore preparation process involves calcining the ore mixed with anhydrite and dolomite to convert the silicates to sulfates for leaching. Recoverable metals include lithium, potassium and sodium. The calcine is leached in water recovering the sulfates to solution.

The wet recovery process includes evaporation and crystallization stages to recover potassium and sodium as sulfates, and lithium as a carbonate, a material suitable for battery manufacture. The products would be purified to meet specifications for marketing.

13.1.2 Metallurgical Testwork

13.1.2.1 Hazen

Hazen Research, Inc. (Hazen) in Golden, Colorado was contracted by WLC to continue process development, define process parameters for calcining and Li_2CO_3 production, and adapt the process to semi-continuous operation.

The work was performed in two phases. Phase I explored bench-scale kiln operating parameters, investigated purification concepts including ion exchange, and provided recommendations for scale-up to a semi-continuous process. The objective of Phase II was to incorporate the results from the Phase I work and produce 1 kg of lithium carbonate.

<u>Phase I</u>

Testwork was performed on samples of oxidized and unoxidized mineralized material from drillholes WLC-13 and WLC-20.

Twenty-eight rotary kiln experiments and accompanying leaches were performed to examine optimal temperature, residence time, gas environment, powdered versus pelletized feed and reagent blend ratios for lithium recovery.

Two fluid-bed calcining experiments were performed for comparison with rotary kiln results.

An investigation to identify a suitable ion exchange (IX) resin for purification of the Li_2CO_3 product was completed.

Two mineralogical evaluations were undertaken.

Results were reported in the report entitled *Phase I Lithium Recovery Pilot Plant Development*, Hazen Project 11103—07, issued June 13, 2011.

Phase II

The samples for this work were obtained from test pit WLT-02 located in the south end of the McDemitt caldera near the reserve defined in this study, (K-UTEC Sample, December 7, 2011, Technical Reference E). The location relative to the ore deposit is shown in Figure 1, Technical Reference E.

Semi-continuous pilot calcination was conducted at 1000°C for 45 minutes on a mixture of one part ore to 0.4 parts each of anhydrite and dolomite. Leaching was conducted at 90-95°C for a retention time of one hour. Ion exchange was utilized to remove calcium from the brine. Potassium and sodium were evaporated to glaserite prior to Li_2CO_3 precipitation with Na_2CO_3. Purification of the Li_2CO_3 was accomplished by repulping and CO_2 dissolution, a step considered unnecessary in the final flowsheet.

The report, *Phase II Process Parameter Summary for Lithium Recovery Pilot Plant Development*. Hazen Project 11103-07, was issued August 16, 2011.

13.1.2.2 K-UTEC

Laboratory-scale testwork to verify and advance the WLC wet recovery process was performed at K-UTEC AG Salt Technologies, Sondershausen, Germany (K-UTEC). Three phases of work were commissioned by WLC: Phase 1 performed process verification on synthetic brine, Phase 2 verified the Phase 1 work on calcined ore (by Outotec) from the deposit, and Phase 3 provided design parameters, a mass balance and capital and operating cost estimates for the wet plant.

The Phase 1 Report, *Process Design to Manufacture Lithium Carbonate from a Hectorite Containing Ore*, is dated August 30, 2010.

The Phase 2 Report, *Process Design to Manufacture Lithium Carbonate from a Hectorite Containing Ore*, is dated September 16, 2010.

The Phase 3 Report, *Evaporation and Crystallization Process and Preliminary Design to Extract Li_2CO_3, K_2SO_4 and Na_2SO_4 for the Kings Valley Leach Solution*, is dated September 12, 2011. Based on the K-UTEC test results, overall recovery of the metals is 87.2% Li, 77.7% K and 82.7% Na.

13.1.2.3 CICITEM

WLC engaged Centro de Investigación Científico Tecnológico para la Minería (CICITEM) at the University of Antofagasta in Chile to develop baseline information of phase diagrams for the crystallization process that included the separation of sodium and potassium sulfates from lithium sulfate.

The results of the CICITEM work are published in Potassium Sulfate Recovery Process from Li_2CO_3 "Barren Solution," Without Addition of Potassium Chloride – Phase 1 of Test Work Program, dated July 27, 2010.

Additional flowsheet development is based on this testwork performed on simulated lithium barren solutions. A review of this testwork by URS suggests the flowsheet can be simplified to remove the acidification step following Li_2CO_3 precipitation. This modification has been incorporated in the flowsheet and process description presented in Section 17.1.

13.1.2.4 Outotec

Outotec GmbH, Oberursel/Frankfurt am Main, Germany (Outotec) was commissioned by WLC to perform advanced granulating and calcining tests utilizing a laboratory fluidized bed calciner. Additional granulation testwork was performed confirming the ability of the granules to maintain integrity through the calcination process. The tests confirmed a fluidized bed unit is suitable for the WLC Kings Valley process; however the results indicated the recovery of lithium in the subsequent leach circuit was seven points lower than that achieved utilizing a rotary calciner. WLC elected to advance the project incorporating a rotary unit. A portion of the calcine generated in the testwork was forwarded to K-UTEC for the Phase II testing.

Two Outotec reports were issued: *Granulation and 200 mm Fluid Bed Testwork with Lithium Containing Clay*, December 2010, Project No. 6040520, and *Beneficiation, Granulation and Roasting of 1700 kg Lithium containing clay Mixture in Outotec's 200 mm Fluid Bed Pilot Plant*, February 2011, Project No. 6042408.

13.1.3 Other Factors

Confirmation of the required addition of anhydrite and dolomite per tonne of ore to achieve design product recovery is recommended for the next phase of the project.

Verification of the suitability of coal for direct-firing the calciner and quantification of the associated energy requirements is recommended for the next phase of testwork.

Analysis of the calciner off-gas from the coal-fired calcining tests is recommended to provide a basis for treatment requirements.

Identification of corrosive gases from the calcination process in the testwork to date suggests future work to quantify the content and volume of gas flows for materials selection in calcining equipment and gas-handling ductwork.

Confirmation of the revised flowsheet that does not incorporate the acidification step is recommended.

13.2 STAGE II

WLC engaged KCA in 2009 to undertake experimental laboratory work with the goal of studying a thermal process for extracting lithium from lithium-bearing clays obtained from the Kings Valley deposit in Nevada. The basis of the test work is derived from prior work completed by the USBM in 1988 and from the experience of KCA personnel. The results of the work conducted by KCA are presented in *Kings Valley Lithium Project—Metallurgical Test Work Summary,* Report #39801, November 2009.

The findings of the KCA test work program were the basis for the process design criteria and are described in detail in Section 16 of URS' Technical Report: *NI 43-101 Technical Report Preliminary Assessment and Economic Evaluation, Kings Valley Project*, *Humboldt County, Nevada, USA*, effective date December 31, 2009.

Overall, the process originally proposed by the USBM to extract lithium from the lithium bearing clays described above was proven to be viable. GSI believes that the metallurgical samples taken are representative of the material being tested. While KCA performed the metallurgical tests on material from the Stage I Lens, GSI believes that the previous USBM and the mineralogical similarities between the Stage I and Stage II lenses are a reasonable basis for assuming that Li-rich clays from the Stage II Lens could be beneficiated in a similar manner. Therefore, it is reasonable to assume that this Stage II Lens estimate indeed qualifies as a resource. However, further metallurgical testing, specific to Stage II Lens clays, would be necessary to estimate mineral reserves and perform economic analyses.

The process development work conducted under Stage II has been superseded by technologies advanced in the Stage I development program.

14.0 MINERAL RESOURCE ESTIMATES

14.1 STAGE I

WLC engaged Reserva International, LLC to complete a resource estimate for the Stage I Lens within the Project, located in Humboldt County, Nevada. The resource estimate was made from a three-dimensional block model using commercial mine planning software (Gemcom GEMS™). The main estimated elements are lithium (Li) and potassium (K), but estimates were also made for sodium, magnesium, aluminum, calcium and iron, in addition to minor elements molybdenum, rubidium, strontium, chromium, titanium, arsenic and barium. The estimates are current as of June 28, 2011, the effective date of this section of the report.

Table 14.1 shows the total estimated resources for the Stage I Lens for a range of Li cutoff values. Reserva is of the opinion that, at a 3,200 ppm (0.32%) lithium cutoff, the Stage I Lens has reasonable prospects for economic extraction by open-pit mining. Section 15.1 of this report provides the parameters for the derivation of the cutoff. Lithium carbonate would be the primary product, associated with potassium and sodium as byproduct. The resources are reported using a lithium cutoff.

- Measured resources are in blocks (30 m × 30 m x 3 m) estimated using at least three drill holes and four composites within a 200 m × 150 m search radius in the horizontal plane and 20 m in the vertical direction.

- Indicated resources are in blocks estimated using at least two drillholes and five composites within a 100 m × 75 m search radius in the horizontal plane and 10 m in the vertical direction.

- Inferred resources are blocks estimated with at least three composites within a search radius of 300 m × 225 m in the horizontal plane and 30 m in the vertical plane.

The density values used in the model are shown in Table 14.2. See Section 11.1.3 for details on how density was determined.

Note that rounding errors may occur in Table 14.1 and that the contained metal does not account for mine or metallurgical recovery. To convert from Li % to Li ppm, multiply the % value by 10,000. The conversion factor from Li % to LCE (lithium carbonate equivalent) is 5.323. This factor is determined from the proportion of lithium in the molecular weight of lithium carbonate (Li_2CO_3), as follows:

$$\frac{\text{molecular weight } Li_2CO_3}{\text{molecular weight 2 x Li}} = \frac{(2 \times 6.941) + (12.011) + (3 \times 15.9994)}{(2 \times 6.941)} = 5.323$$

Table 14.1: Resources Statement Tabulation by Lithium Cutoff

MEASURED							
Cutoff Li ppm	**KTonnes**	**Li%**	**Ktonnes LCE**	**K%**	**Ktonnes K**	**Na%**	**Ktonnes Na**
2000	50,753	0.312	843	3.27	1,660	1.13	574
2250	45,214	0.324	780	3.34	1,510	1.16	524
2500	38,858	0.338	699	3.42	1,329	1.20	466
2750	32,370	0.354	610	3.54	1,146	1.23	398
3000	24,765	0.374	493	3.71	919	1.27	315
3250	18,046	0.397	381	3.89	702	1.29	233
3500	13,100	0.420	293	4.00	524	1.29	169
3750	9,788	0.439	229	4.08	399	1.30	127
4000	7,226	0.457	176	4.14	299	1.32	95
4250	5,235	0.475	132	4.19	219	1.33	70
4500	3,475	0.494	91	4.26	148	1.35	47
4750	2,180	0.513	60	4.36	95	1.38	30
5000	1,373	0.529	39	4.44	61	1.38	19
INDICATED							
Cutoff Li ppm	**KTonnes**	**Li%**	**Ktonnes LCE**	**K%**	**Ktonnes K**	**Na%**	**Ktonnes Na**
2000	164,046	0.285	2,489	3.07	5,036	1.04	1706
2250	136,149	0.300	2,174	3.15	4,289	1.07	1457
2500	107,450	0.317	1,813	3.27	3,514	1.11	1193
2750	81,949	0.334	1,457	3.39	2,778	1.14	934
3000	58,600	0.352	1,098	3.51	2,057	1.16	680
3250	38,111	0.373	757	3.63	1,383	1.19	454
3500	24,177	0.395	508	3.73	902	1.20	290
3750	14,857	0.415	328	3.86	573	1.22	181
4000	8,796	0.435	204	3.94	347	1.26	111
4250	4,533	0.458	111	3.98	180	1.26	57
4500	2,170	0.480	55	4.06	88	1.28	28
4750	971	0.502	26	4.04	39	1.29	13
5000	478	0.517	13	4.04	19	1.33	6
INFERRED							
Cutoff Li ppm	**KTonnes**	**Li%**	**Ktonnes LCE**	**K%**	**Ktonnes K**	**Na%**	**Ktonnes Na**
2000	124,890	0.294	1,954	3.04	3,792	1.10	1374
2250	107,781	0.307	1,761	3.12	3,357	1.12	1207
2500	89,289	0.321	1,526	3.24	2,889	1.13	1009
2750	70,500	0.337	1,265	3.40	2,396	1.13	797
3000	57,348	0.348	1,062	3.43	1,969	1.15	660
3250	36,325	0.369	713	3.59	1,305	1.15	418
3500	24,234	0.386	498	3.74	907	1.15	279
3750	13,044	0.404	281	3.69	482	1.15	150
4000	7,457	0.416	165	3.64	272	1.15	86
4250	1,546	0.439	36	3.96	61	0.95	15
4500	182	0.470	5	3.22	6	1.06	2
4750	48	0.501	1	3.09	1	1.13	1
5000	19	0.524	1	3.51	1	1.04	0

Note: 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.

Table 14.2: Density Values used in the Stage I Lens Resource Model

Lithology	Average Density (t/m^3)
Alluvium	1.52
Claystone (average)	1.79
Basalt	2.51
Volcanics	1.96

All of the holes have been drilled vertically except for WLC-58 (-70 degree dip, 180 degree azimuth) and are assumed to not have deviated. WLC has checked this assumption with downhole surveys of 23 holes in the Stage I Lens. Results indicate very little deviation of vertical holes (minimum dip: -90, maximum dip: -88.2, average dip: 89.5) and support the assumption of verticality for previously drilled holes.

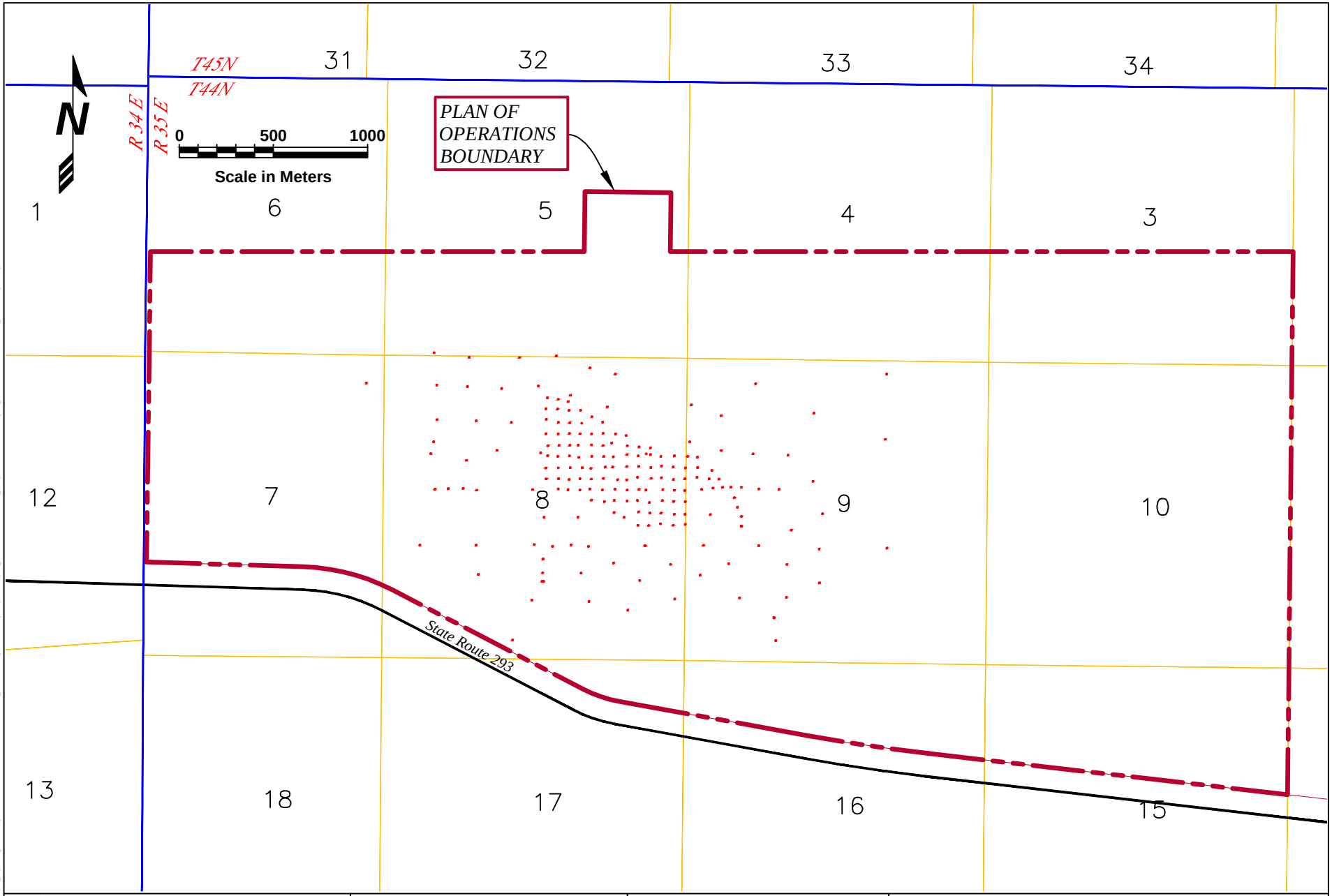
14.1.1 Geologic Model

Project limits and coordinates are in UTM NAD 27 easting northing and elevation values and are in meters. The block model geometry is summarized in Table 14.3. The block size selected for the Stage I Lens modeling is 30m × 30m × 3m, which is considered reasonable given the drillhole spacing and the fact that the block model represents open pit resources.

The drillhole database used for geologic modeling and resource estimation was closed on June 28, 2011. The last drillhole for which assay data was available was WLC-200. A total of 197 holes drilled by WLC (see Section 10.1, Drilling), and cores were assayed at ALS Labs and were used for estimation purposes. A plan view of the drillhole collars is shown in Figure 14.1 below. The average horizontal spacing of the drillholes varies from a nominal 180 m × 180 m, to 61 m × 61 m in the more closely drilled central portion of the deposit.

Table 14.3: Gemcom Model Extent, UTM Coordinates (m)

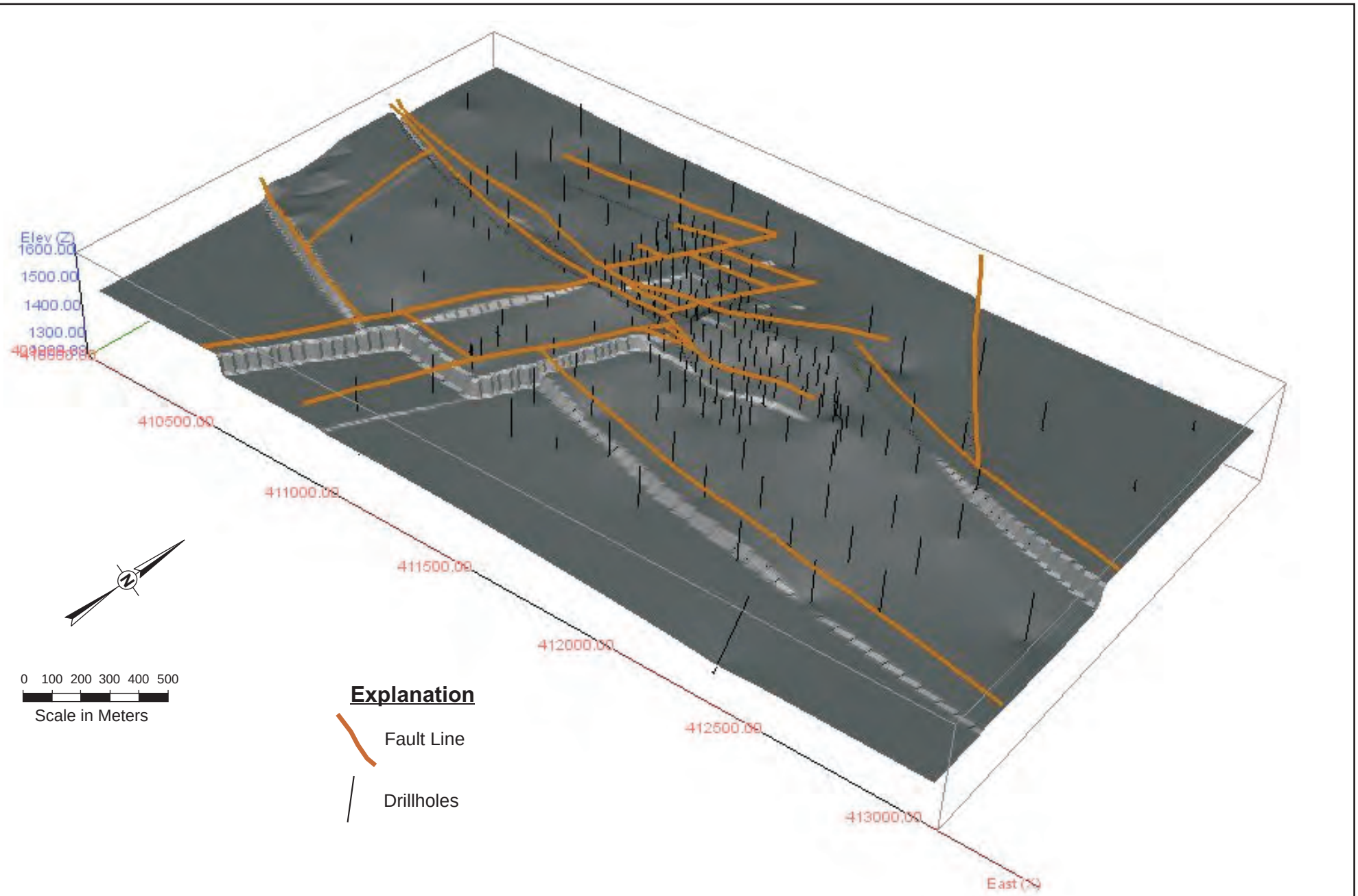
UTM Coordinates (m)	Minimum	Maximum	Block Size (m)	No. of Blocks
Easting	410,000	413,000	30.0	100
Northing	4,616,800	4,618,600	30.0	60
Elevation	1,306	1,615	3.0	103



Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: and Provide by: ***Western Lithium Corp.***	Location: Humboldt County, NV	**Figure 14-1** **Drillhole Collar Locations**
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	

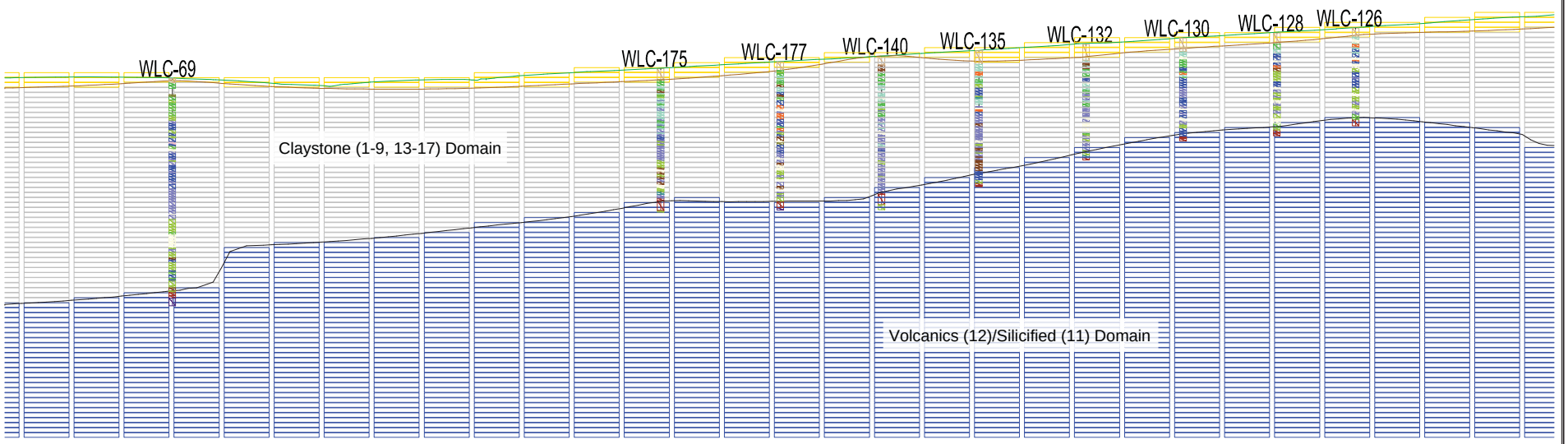
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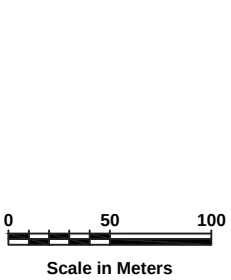
The Stage I Lens mineralization is controlled by volcaniclastic moat sedimentary rocks containing lithium-rich claystone. Sectional interpretations (for E-W and N-S transects) were generated by WLC from drill logs for alluvium, claystone (moat sediments), basalt, a silicified unit, and bedrock. Two oxidation surfaces were also interpreted, one just below alluvium and another near the claystone/silicified interface. Additionally, a series of faults have been interpreted by WLC based on drillhole data and were incorporated in the geologic interpretation. The approach taken was to develop a set of 3-D triangulated surfaces representing the topographic surface and the interfaces between the overburden, claystone and volcanics/silicified rocks. A fault surface was created in order to enhance the quality of interpretation. The volume (solid) between the modeled overburden and basement was treated as a single domain for estimation and resource reporting purposes. A 3-D perspective of the faulted basement surface and surface expression of the faults is shown in Figure 14.2. The surfaces and solids were used to code the block model by lithology domains within GEMS™. A lithologic domain is a volume considered as a single entity for modeling purposes. All claystone rock types, intercalated ash layers and occurrences of basalt were treated as a single domain. How the presence of basalt was handled in the domain is outlined in Section 14.1.5. Figure 14.3 shows an N-S cross section (looking west) with the interpreted domain surfaces shown; also illustrated in this figure is the topography and the drillholes color coded by the rock type codes. Reserva checked the consistency of the interpreted domain codes against WLC drillhole logs in section and plan and found it acceptable.



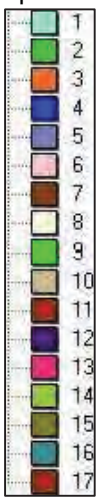
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	Project:	Project Number:	Date of Issue:
	Kings Valley Lithium Project	114-311106	01/06/2012

Figure 14.2
3D Perspective of Faulted Basement Surface





Explanation



Created by: rêserva INTERNATIONAL	Prepared for: ***Western Lithium Corp.***	Location: Humboldt County, NV	**Figure 14.3** **N-S Cross Section at 411580E Showing Interpreted Lighology Domains (Looking West)**
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	

Note that the integer codes representing the rock type (RT) are shown in Table 14.3

Table 14.4: Rock Type Integer Codes and Descriptions

Description	Abbreviation	Integer Code
Green claystone	Green clyst	1
Tan claystone	Tan clyst	2
Light gray claystone	Lt gry clyst	3
Gray claystone	Gry clyst	4
Carbonaceous claystone	Carb clyst	5
Bluish claystone	Bluish clyst	6
Ash/claystone	Ash/clyst	7
Claystone/ash	Clyst/ash	8
Laminated claystone	Laminated	9
Quaternary alluvium	Qal	10
Silicified pond zone	HPZ	11
Volcanics	TV	12
Basalt	Basalt	13
Ash	Ash	14
Fault	Fault	15
Unspecified	Blank	16
Volcanic mud	Volc mud	17

14.1.1.1 Assays

As of June 28, 2011 a total of 197 core holes had been drilled by WLC in the Stage I Lens. Assay data for these holes was used for estimating grade. Detailed and/or summary lithological logs were available for all holes and were used in the modeling of the lithological surfaces and domains. See Section 11.1 for a description of the core sampling process. The data was imported into the GEMS™ project database. The average length of the assay intervals was approximately 1.5 m.

14.1.2 Exploratory Data Analysis

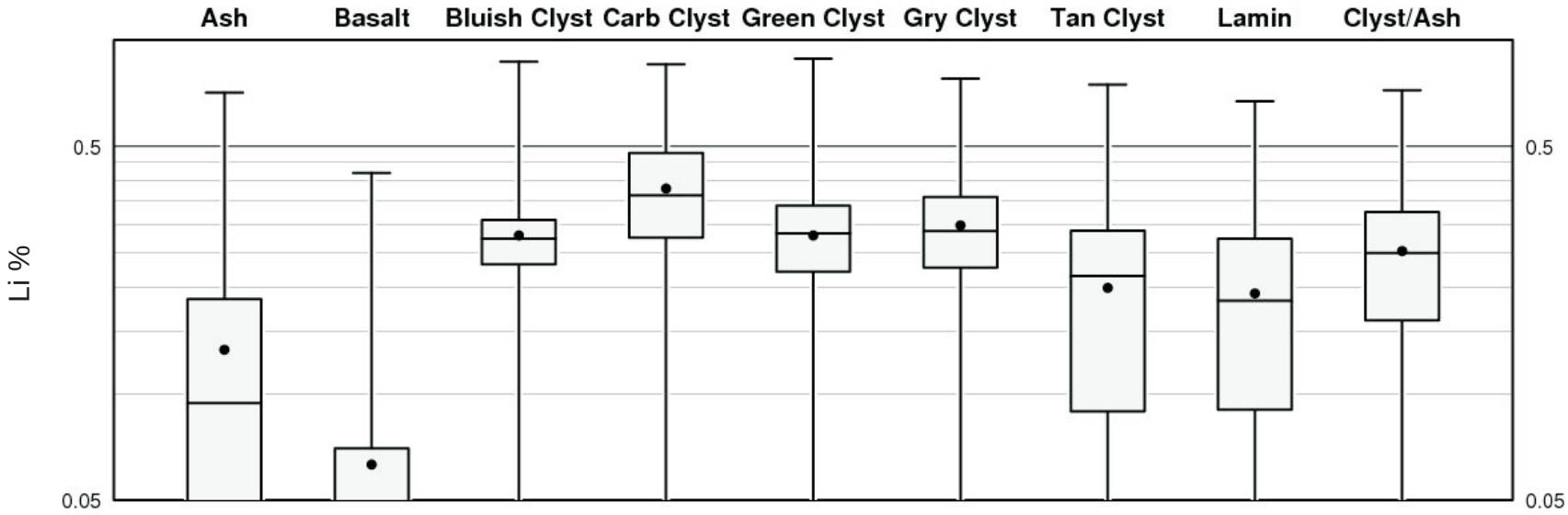
The relationships among the assay values for the major elements (lithium, potassium) and among the other elements of interest were analyzed by constructing a correlation matrix of the data, as illustrated in Table 14.5. Correlations greater than the absolute value of 0.5 are highlighted.

Table 14.5: Assay Values – Correlation Matrix (All Rock types)

	Li	K	Na	Mg	Al	Ca	Fe	Mo	Rb	Sr	Cr	Ti	As	Ba
Li	1.000	0.289	-0.056	0.536	-0.570	0.252	-0.411	0.460	0.763	0.229	-0.125	-0.071	0.235	-0.212
K	0.289	1.000	-0.083	-0.231	0.176	-0.111	-0.104	0.433	0.620	-0.297	-0.111	0.144	0.523	-0.162
Na	-0.056	-0.083	1.000	-0.194	0.396	-0.096	0.236	0.004	0.106	-0.106	-0.009	0.032	0.129	0.020
Mg	0.536	-0.231	-0.194	1.000	-0.519	0.446	-0.369	-0.078	0.249	0.722	-0.188	-0.135	-0.277	0.015
Al	-0.570	0.176	0.396	-0.519	1.000	-0.391	0.632	-0.230	-0.123	-0.352	0.129	0.319	0.078	0.157
Ca	0.252	-0.111	-0.096	0.446	-0.391	1.000	-0.209	0.079	0.024	0.624	-0.068	-0.048	-0.110	0.006
Fe	-0.411	-0.104	0.236	-0.369	0.632	-0.209	1.000	-0.177	-0.196	-0.232	0.405	0.373	0.066	0.137
Mo	0.460	0.433	0.004	-0.078	-0.230	0.079	-0.177	1.000	0.460	-0.131	0.034	-0.044	0.591	-0.229
Rb	0.763	0.620	0.106	0.249	-0.123	0.024	-0.196	0.460	1.000	0.026	-0.099	0.108	0.393	-0.160
Sr	0.229	-0.297	-0.106	0.722	-0.352	0.624	-0.232	-0.131	0.026	1.000	-0.125	-0.109	-0.283	0.152
Cr	-0.125	-0.111	-0.009	-0.188	0.129	-0.068	0.405	0.034	-0.099	-0.125	1.000	0.139	0.009	0.003
Ti	-0.071	0.144	0.032	-0.135	0.319	-0.048	0.373	-0.044	0.108	-0.109	0.139	1.000	0.081	0.047
As	0.235	0.523	0.129	-0.277	0.078	-0.110	0.066	0.591	0.393	-0.283	0.009	0.081	1.000	-0.192
Ba	-0.212	-0.162	0.020	0.015	0.157	0.006	0.137	-0.229	-0.160	0.152	0.003	0.047	-0.192	1.000

Lithium and potassium, with a correlation of 0.289, can be considered to be not strongly correlated. The strongest correlation for both Li and K elements is with the minor element rubidium. Lithium exhibits a moderate positive correlation (0.536) with magnesium and exhibits a negative correlation (-0.570) with aluminum. These correlations are related to the mineral composition. The moderate to strong correlations (both positive and negative) among the minor elements are not significant to the resource estimate.

A boxplot (Figure 14.4) generated with assay lithium values illustrates the variation in concentrations with respect to rock type.



	Ash	Basalt	Bluish Clyst	Carb Clyst	Green Clyst	Gry Clyst	Tan Clyst	Lamin	Clyst/Ash	
Number of data	1695	87	217	1922	705	1683	866	719	800	Number of data
Mean	0.133	0.063	0.28	0.38	0.28	0.299	0.199	0.192	0.253	Mean
Maximum	0.709	0.421	0.869	0.854	0.885	0.778	0.747	0.671	0.72	Maximum
Upper quartile	0.185	0.07	0.31	0.479	0.34	0.36	0.289	0.274	0.326	Upper quartile
Median	0.094	0.04	0.274	0.364	0.284	0.288	0.215	0.183	0.25	Median
Lower quartile	0.032	0.021	0.232	0.276	0.221	0.227	0.089	0.09	0.161	Lower quartile
Minimum	0.001	0.005	0.012	0.007	0.009	0.004	0.007	0.0	0.007	Minimum
Variance	0.0171	0.0047	0.0102	0.0236	0.0135	0.0163	0.0142	0.0146	0.0142	Variance
CV	0.979	1.101	0.36	0.404	0.415	0.428	0.597	0.629	0.47	CV
Skewness	1.609	2.868	1.699	0.218	0.496	0.609	0.15	0.551	0.476	Skewness

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Project: Kings Valley Lithium Project

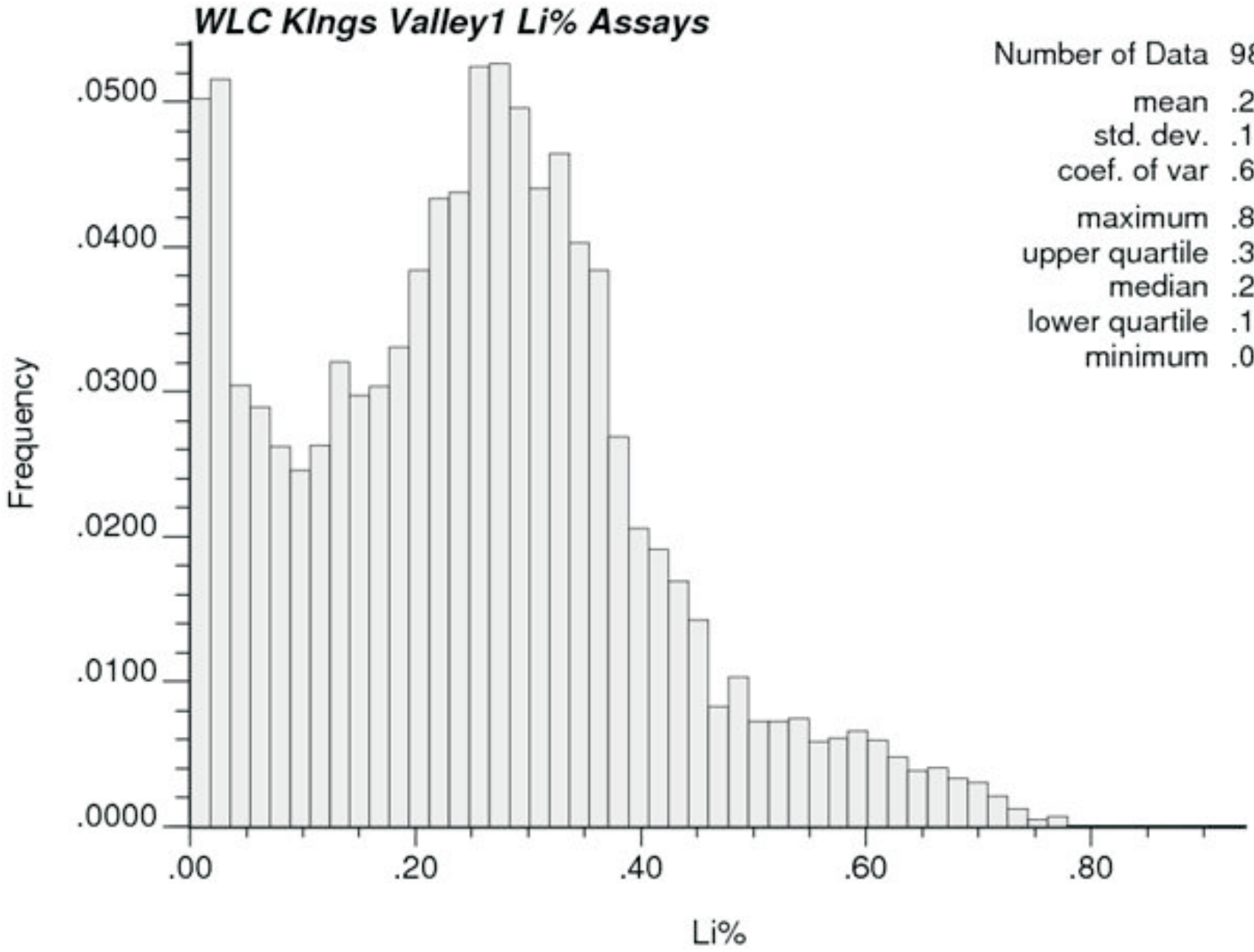
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Figure 14.4
Boxplot of Lithium by Rock Type

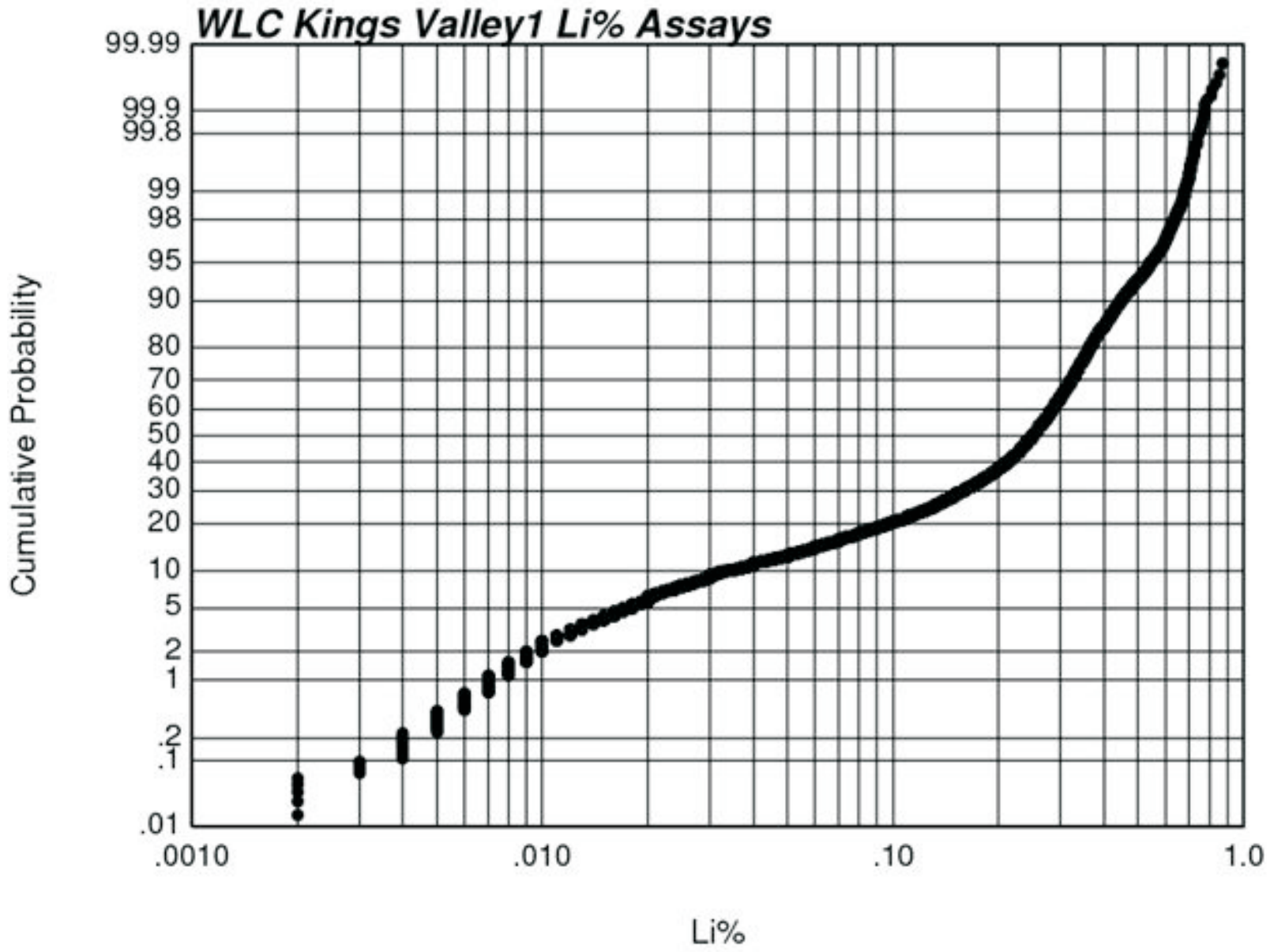
Figure 14.5 illustrates that there is a distinct range of lithium values in the claystone rock types, with the highest values occurring in the carbonaceous claystones. These relationships are consistent across the deposit. Lithium values in the intercalated ash layers and basalt are distinctly lower. It should be noted that the high maximum value exhibited is likely due to differences in the interpretations made by various geologists during logging.

Although there are notable variations in the lithium content by rock type, it has not been possible to establish the lateral continuity of individual rock types at this time due to the nature of the deposit and current drillhole spacing. Given the additional indication that bulk mining methods would be employed, it was decided to treat the various claystone units as a single geological domain for modeling purposes. The following histograms and probability plots (Figure 14.5, Figure 14.6, Figure 14.7 and Figure 14.8 for lithium and potassium are, therefore, based on all rock types.



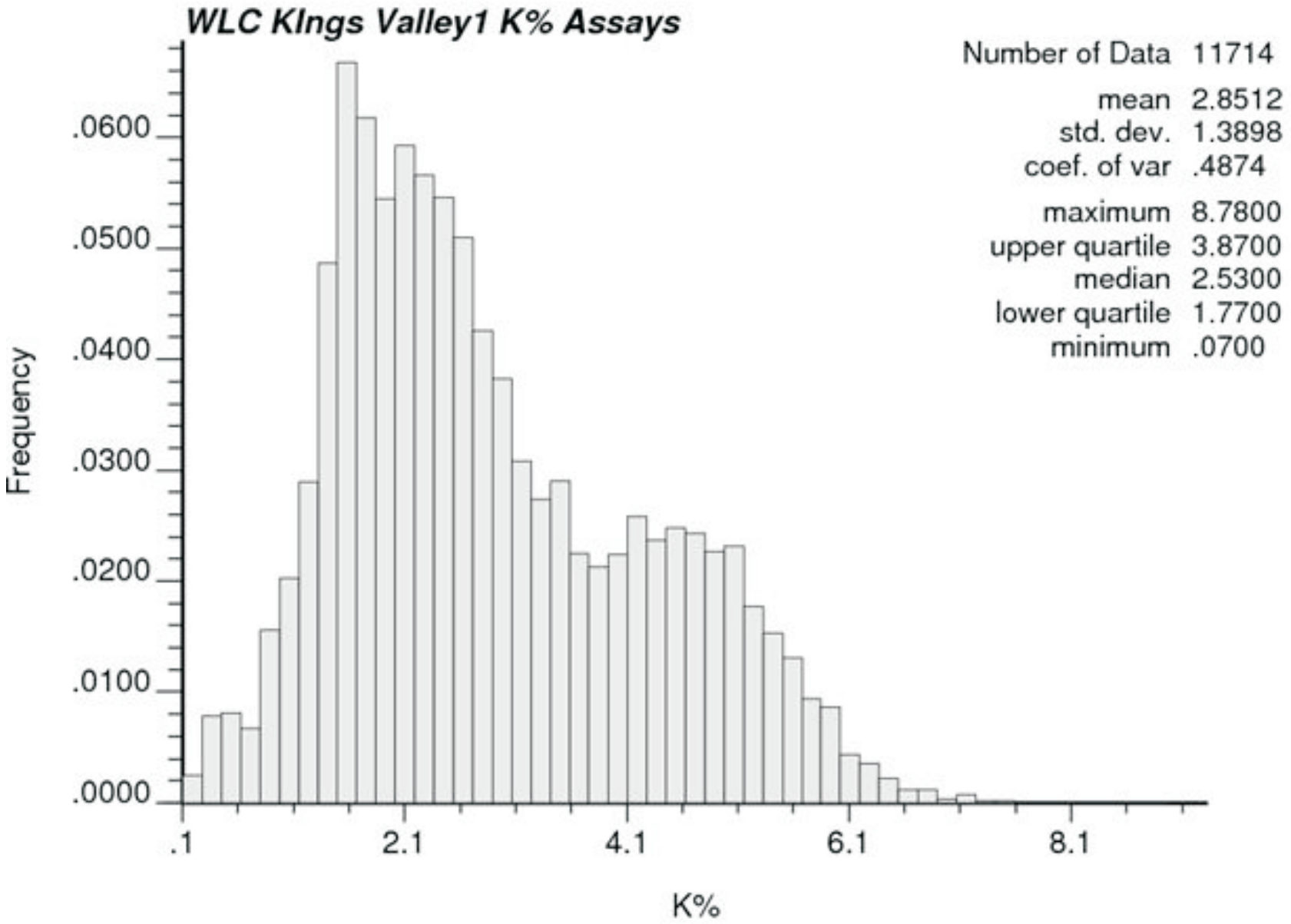
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Figure 14.5
Histogram and Basic Statistics
Lithium Assays



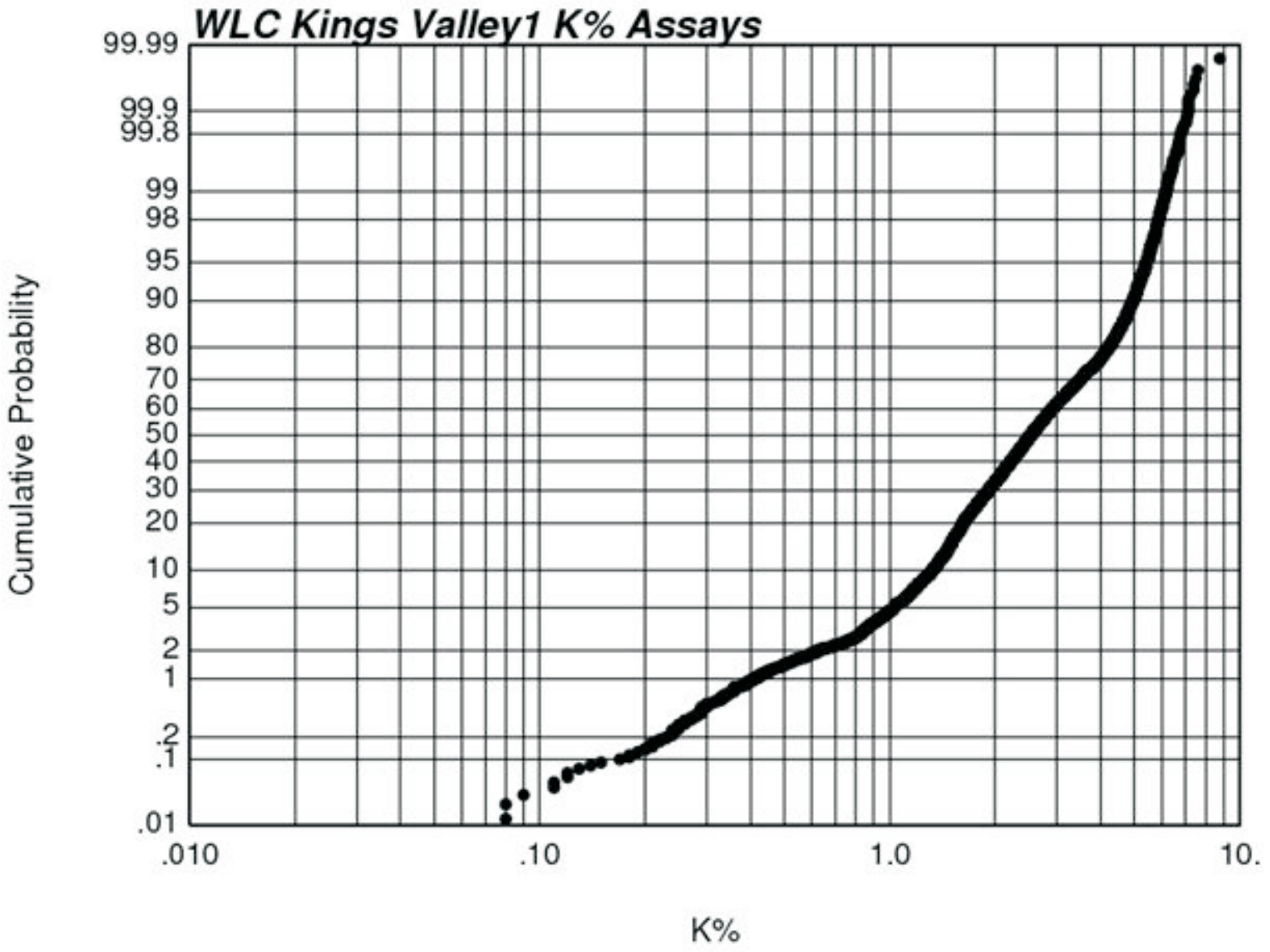
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Figure 14.6
Log Probability Plot, Lithium Assays



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Figure 14.7
Histogram and Basic Statistics
Potassium Assays



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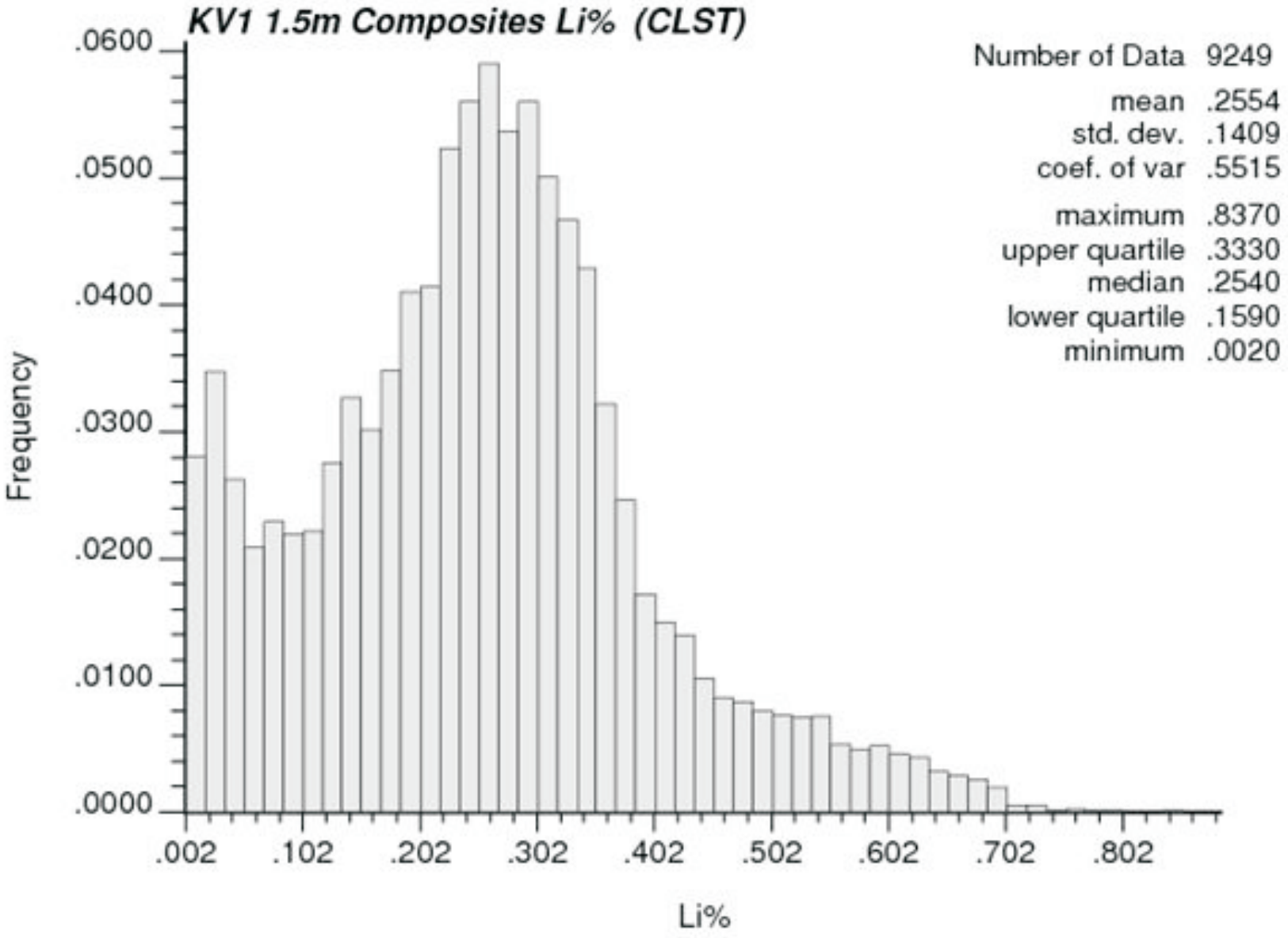
Figure 14.8
Log Probability Plot, Potassium Assays

14.1.3 Composites

Assay values were not capped prior to compositing because probability plots showed continuity and the domain composite coefficient of variation (CV; standard deviation/mean), was relatively low. The overall CV for lithium assays was 0.627, while potassium had a CV of 0.487. Figure 14.9 and Figure 14.10 show the histogram and probability plots of the lithium and potassium assay values. High grade restrictions on composites were, however, used in the estimation, as detailed in Section 14.1.6.

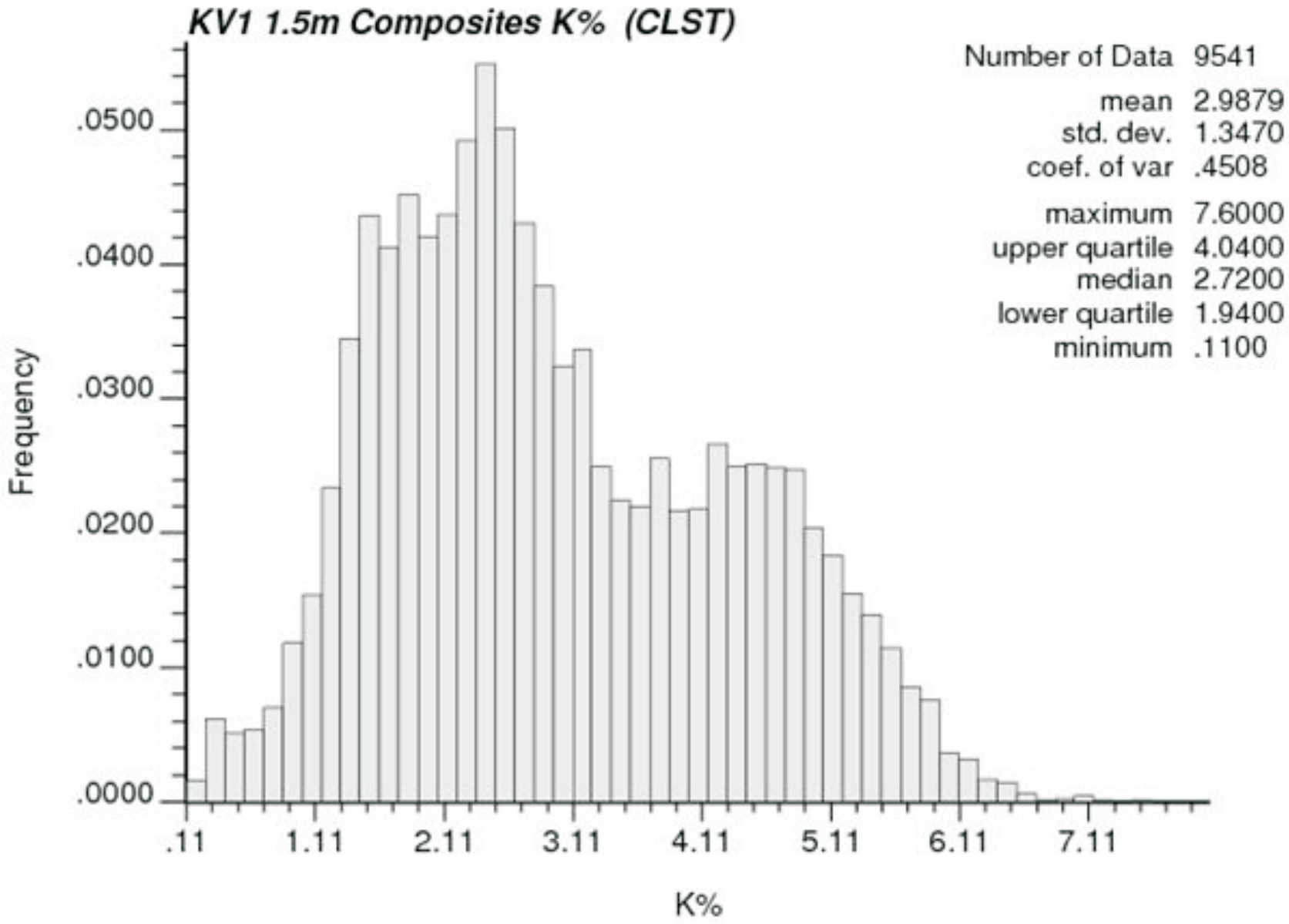
Down-hole compositing was done for 1.5 meter lengths. This length was chosen because it is one-half the vertical block size of 3 meters. This limited the effects of smoothing in grade interpolation. The composites were coded by back-tagging against the rock type block model reflecting the lithology domains. Since there a limited number of assays from the alluvial zone, composites with lithium grades start at the bottom of the alluvial surface, and continue down to the last assayed interval. There were a total of 9,249 Li% composites and 9,541 K% composites within the main mineralized unit. The difference in count reflects a greater frequency of zero Li% intervals than for K%. A histogram and basic statistics of the claystone domain are shown in Figure 14.9 for lithium composite grades.

Figure 14.10 shows the corresponding histogram for potassium composite grades within the same domain.



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Figure 14.9
Histogram and Basic Statistics
Claystone Composites , Lithium



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Figure 14.10
Histogram and Basic Statistics
Claystone Composites , Potassium

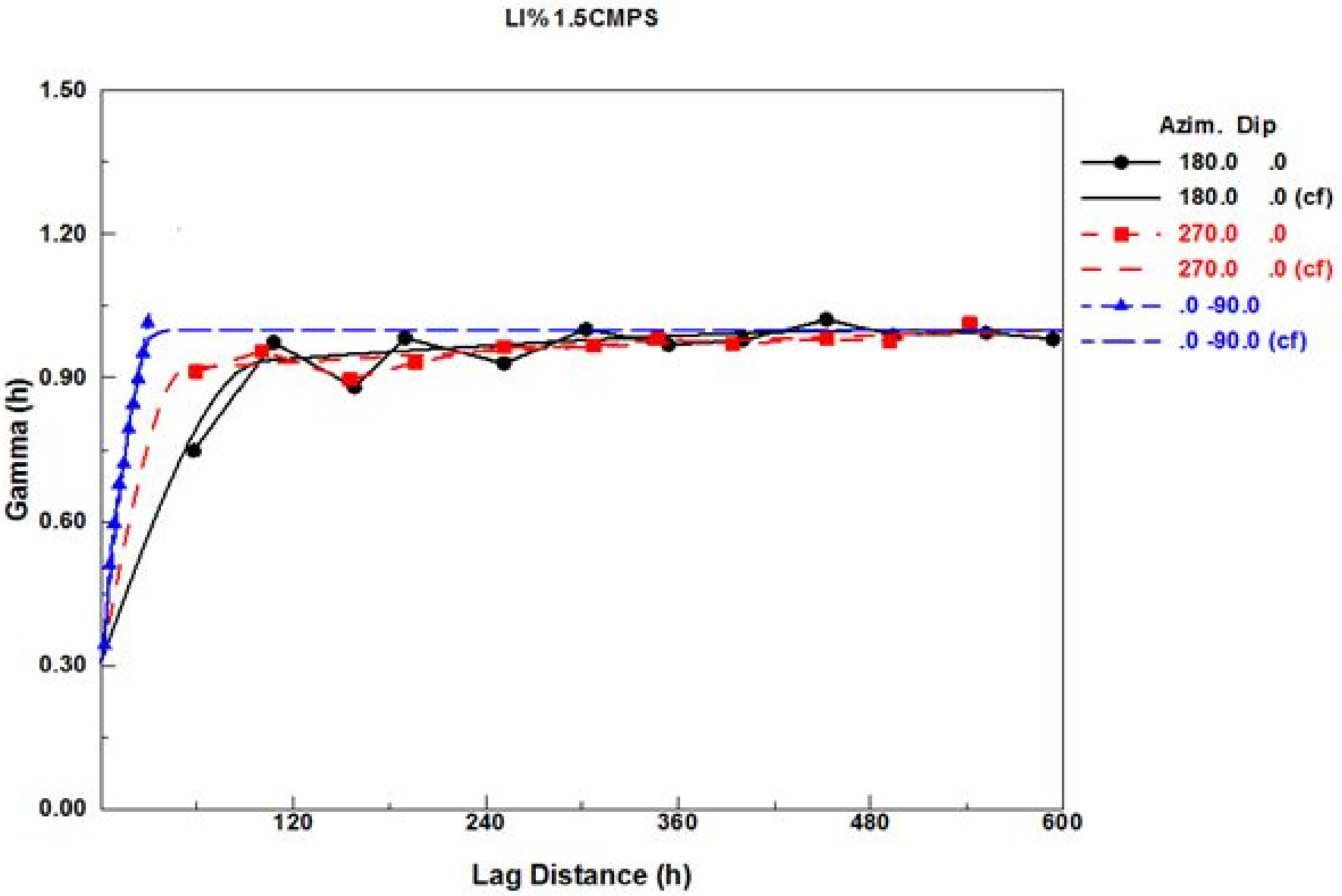
14.1.4 Variography

With most mineral deposits, the measure of continuity for any given metal depends on the separation distance between points of measurement and the direction from position to position. Variability increases and correlation decreases with the sample-to-sample separation distance. When the rate of change in variability is dependent on the direction, the measure of spatial variability is described as anisotropic and it is characterized in terms of an ellipsoid with axes of anisotropy. There are several mathematical functions that measure variability in space, and Reserva elected to use the correlogram, which is a variant of the semi-variogram. The correlogram was probably the first measure of continuity (the converse of variability) developed. It measures the correlation coefficient between two sets of data, with values at the heads and values at the tails of vectors with similar direction and magnitude. Multiple studies have found that the correlogram provides a stable estimate of spatial continuity. For ease of modeling, the correlogram value is subtracted from one and is presented in a similar graphical form to the variogram. The variography was completed using Sage 2001® software and 1.5 m composites. Reserva calculated experimental correlograms with 30 degree increments in the azimuth and 45 degree increments in the dip direction and the vertical direction. These correlograms were computed for lithium and potassium.

The correlograms were modeled with a nugget component and two nested spherical components, used in the resource estimation step. The lithium and potassium parameters used are tabulated in Table 14.6, and the experimental correlograms and fitted models for the directions most closely corresponding to the calculated average correlogram are shown in Figure 14.11.

Table 14.6: Variogram Parameters – Lithium and Potassium

Element	Sill	Range (m)			Rotation (Azimuth/Dipº)		
		X	Y	Z	X	Y	Z
Li	0.290						
	0.617	135.7	62.5	29.7	19/10	291/-16	258/71
	0.093	1420.3	49.7	527.9	294/1	223/-86	204/4
K	0.08						
	0.682	68.6	48.5	16.7	190/-56	47/-28	128/17
	0.238	1077.1	569.2	968.3	74/1	345/-3	232/87



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Figure 14.11
Experimental Correlograms/Fitted Models for Average Correlogram

14.1.5 Estimation

The resources were estimated using either ordinary kriging (OK) or inverse distance (power of two) interpolation, with estimation restricted to the claystone domain. Alluvium and bedrock material have no lithium or potassium grades. A block model was defined to cover the volume of interest, as detailed in Table 14.7 following:

Table 14.7: Block Model Geometry

Axis	Minimum	Maximum	Block Size(m)	# Blocks
X(East)	410,000	413,000	30	100
Y(North)	4,616,800	4,618,600	30	60
Z(Elevation)	1,306	1,615	3	103

Lithium and potassium block grades were interpolated in three passes using the parameters detailed below. The three pass approach was used to ensure that an adequate number of well distributed samples were available to inform most blocks and to reduce the smoothing effect inherent in interpolation.

Pass 1:
Minimum # of samples 5
Maximum # of samples 8
Maximum # samples/hole 2 (Minimum # of holes = 3)

Pass 2:
Minimum # of samples 4
Maximum # of samples 10
Maximum # samples/hole 3 (Minimum # of holes = 2)

Pass 3:
Minimum # of samples 3
Maximum # of samples 12
Maximum # samples/hole N/A

The search ellipsoids used are as detailed in Table 14.8 following. The search orientation is based on the orientation of the second spherical component of the lithium correlogram.

Table 14.8: Search Ellipsoid Geometry (Lithium and Potassium)

Pass	Range (m)			Azimuth/Dip (Deg)		
	X	Y	Z	X	Y	Z
1	100	75	10	114/0	24/0	0/-90
2	200	150	20	114/0	24/0	0/-90
3	300	225	30	114/0	24/0	0/-90

The minor elements (aluminum, arsenic, barium, calcium, chromium, iron, molybdenum, magnesium, sodium and rubidium) were interpolated in a single pass, using inverse distance interpolation (power of three), and the Pass 3 search ellipsoid as shown in Table 14.8.

Although some sectional interpretations of the basalt occurrences were developed by WLC, it was difficult to correlate them and develop a physical 3D model of the rock type. A probabilistic approach using indicators was therefore developed, creating a basalt percent block model (attribute) that could be used to represent the occurrence of basalt and to dilute the lithium grades in the affected blocks, based on the proportion of basalt. This proportion was also used to calculate a weighted average density for the blocks affected. The primary documentation of the presence of basalt is in the lithology table, and this data was used to assign an integer code (basalt = 1, non-basalt = 0) to composites in the claystone domain. These basalt indicators were interpolated into the block model using inverse distance cubed interpolation. The search parameters were based on basalt indicator variography. The interpolated block estimate in this Case is a value between 0 and 1 that represents the proportion of the block that is basalt.

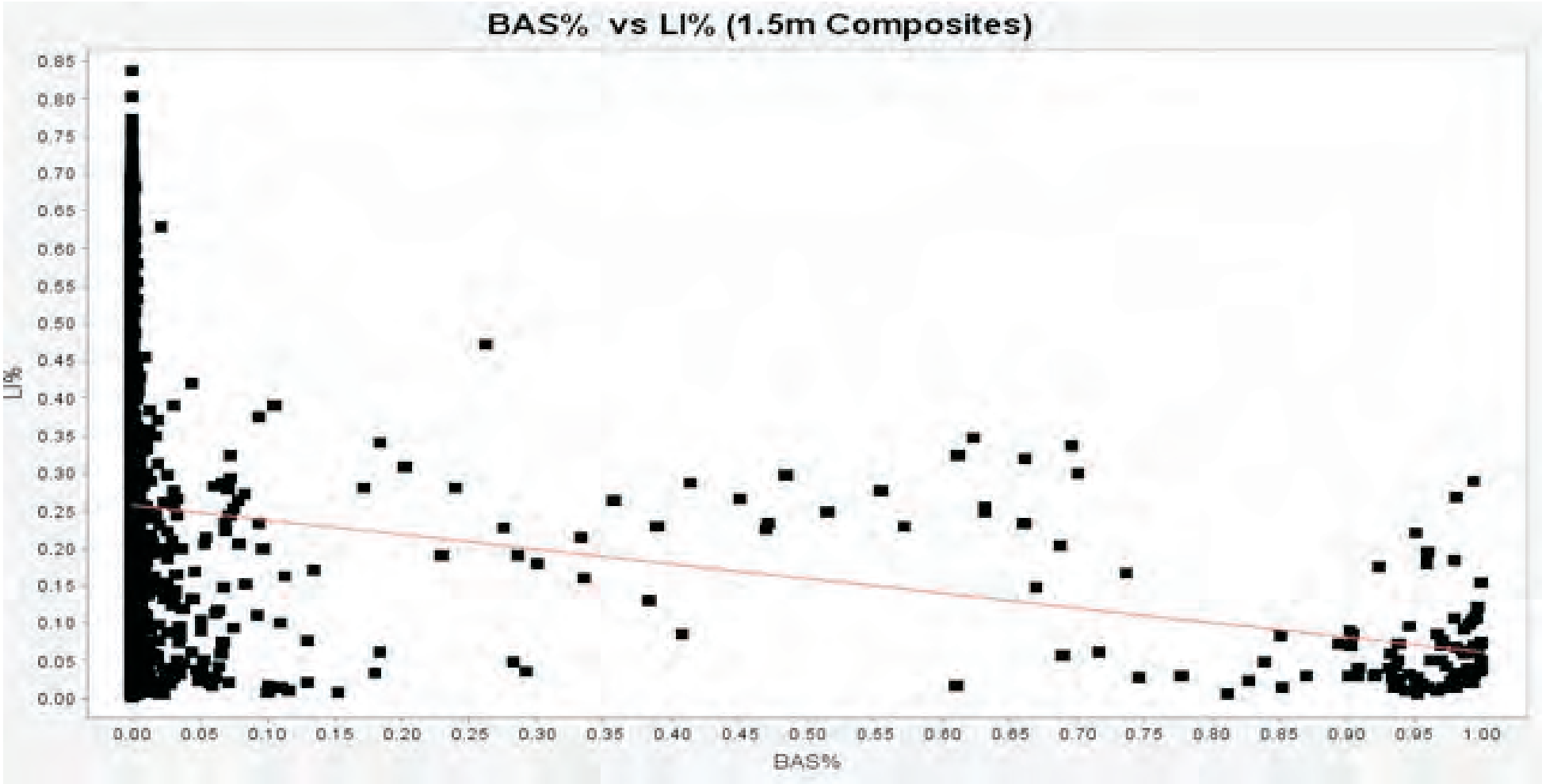
As indicated above, the proportion of basalt estimated in a block was used to adjust the block element grades and density as in the following examples:

Adjusted Li% = Li% × (100% – Basalt%)

Adjusted Density = 1.79 × (100% – Basalt%) + 2.51 × Basalt%

(Where 1.79 t/m^3 and 2.51 t/m^3 are the density of claystone and basalt respectively. The basalt density of 2.51 t/m^3 is based on a single determination.)

With respect to the interpolation of the main element lithium, there was some concern that grades in the vicinity of basalt occurrences (i.e., blocks with high basalt content) were lower than composites from areas with little or no basalt. This could possibly bias the estimation of blocks with high basalt content. The relationship between grade and basalt content was investigated. The basalt percentage was interpolated using indicator geostatistics. The claystone composites were back-tagged with the estimated percentage of basalt. This composite basalt percentage was graphed against the lithium grades as illustrated in Figure 14.12. The x-axis shows the basalt percentage where 1.0 is equal to 100%.



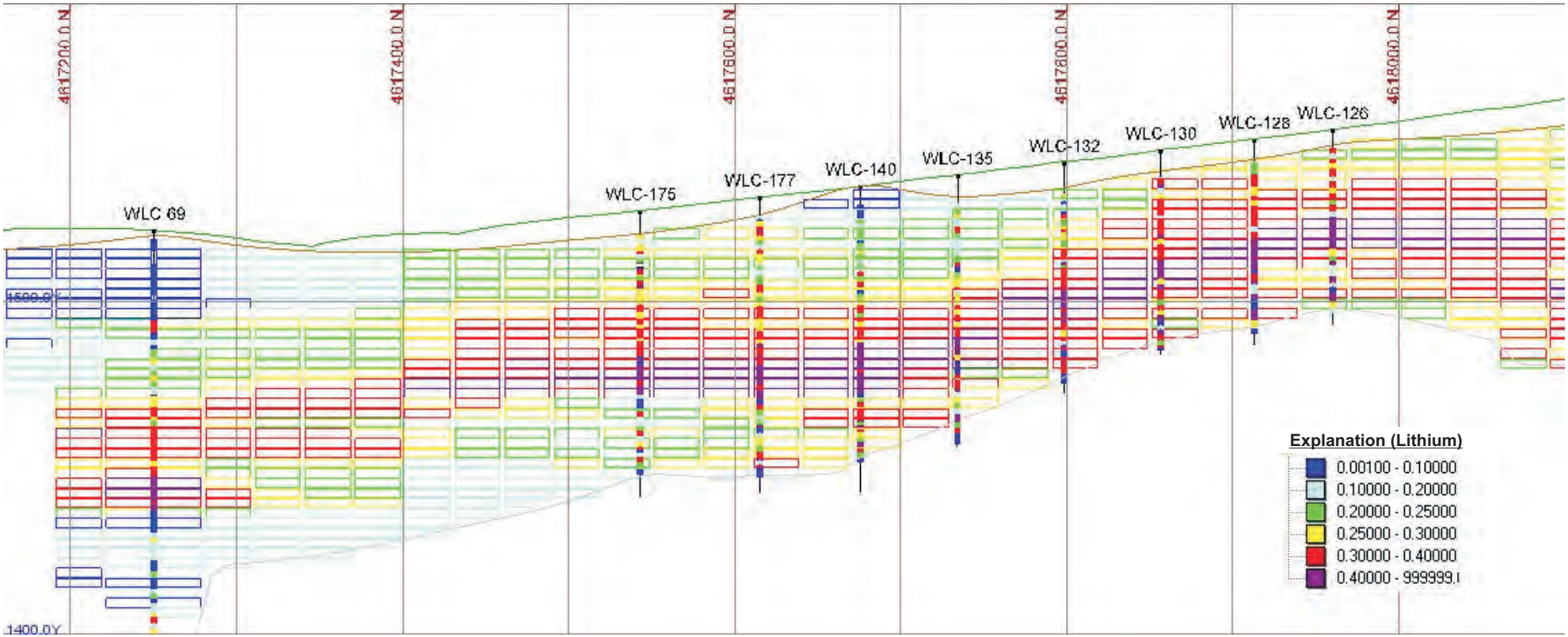
Created by:	Prepared for:	Location:		
rêserva INTERNATIONAL	*Western Lithium Corp.*	Humboldt County, NV		**Figure 14.12**
	Project:	Project Number:	Date of Issue:	**Claystone Composites, Lithium Grade vs Basalt**
	Kings Valley Lithium Project	114-311106	01/10/2011	

A basalt percentage of 10% was selected as a threshold for differentiating two sub-domains within the claystone domain, composed of claystone blocks with a basalt percentage of <10% and ≥10% basalt respectively. In order to prevent the possible bias referred to above, only composites with basalt percentages of <10% were used when interpolating grades into blocks with <10% basalt and, similarly, with composites with a basalt percentage of >10% and blocks with >10% basalt. Overall, blocks with a significant percentage of basalt (≥ 75%) comprise 0.9% of the model blocks with estimated values. Although this is a small percentage, the basalt is still accounted for in the interpolation of grade values in blocks containing basalt as described above. High grade composites were spatially restricted at the time of lithium and potassium grade estimation, which means that although the composites were used, their spatial influence was restricted with a smaller search. This allows the high grade value to be recognized, but prevents excessive smearing of these values. The definition of what was considered "high grade" was obtained from the composite cumulative frequency plots (Figure 14.7 and Figure 14.9) and generally corresponds to the 99% or higher percentile of the distribution. Table 14.9 shows the details of this restriction. The horizontal ranges used were selected to limit the influence to the blocks immediately adjacent to the sample. The anisotropy and orientation of the restricted search ellipsoids are the same as for the full search, as tabulated in Table 14.8. (Search ellipsoid geometry)

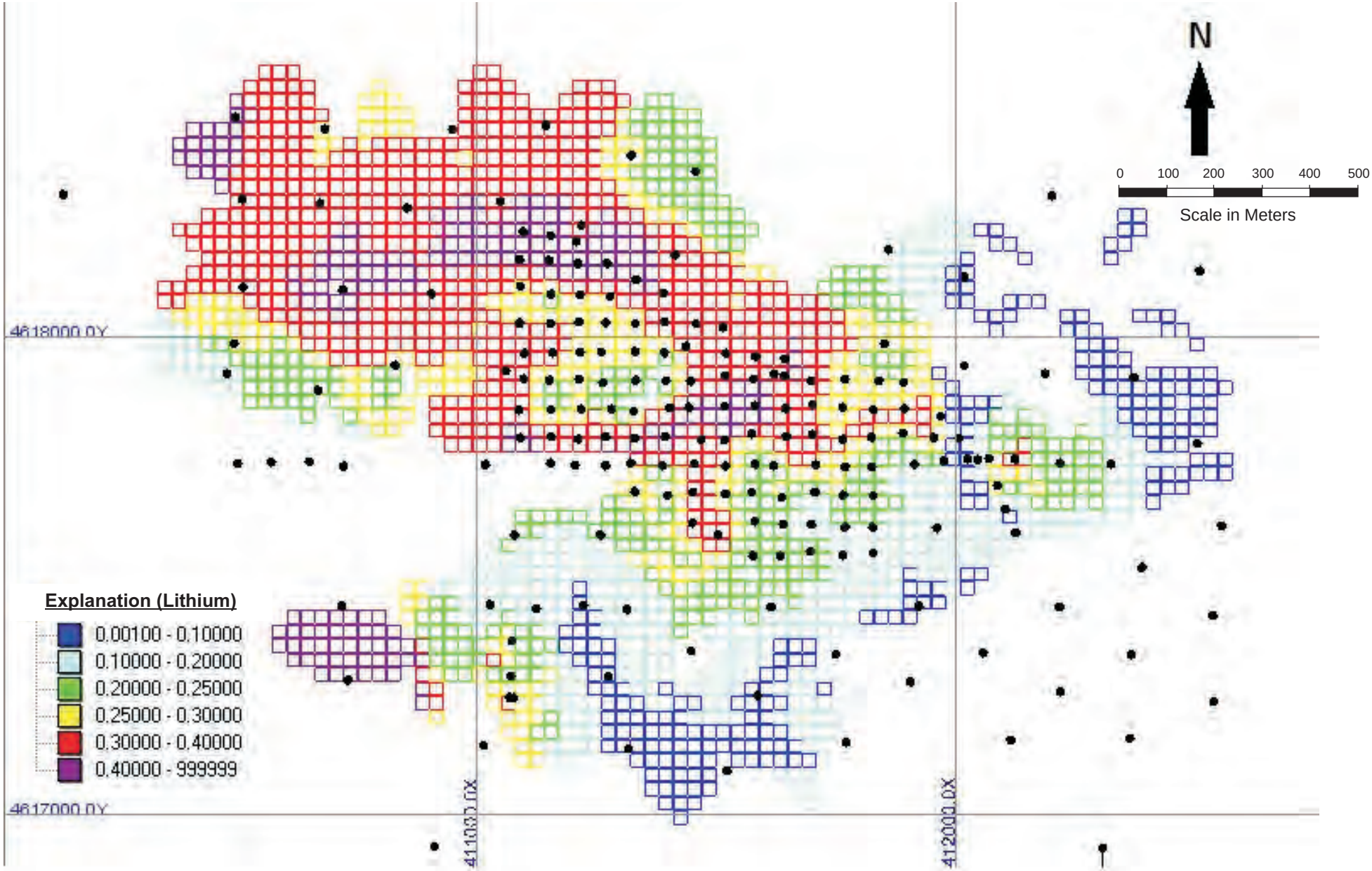
Table 14.9: High Grade Restriction Parameters

Variable	High Grade Threshold	Restricted Search (m)
Li	0.65%	42.5 × 42.5 × 2
K	6.5%	42.5 × 42.5 × 2

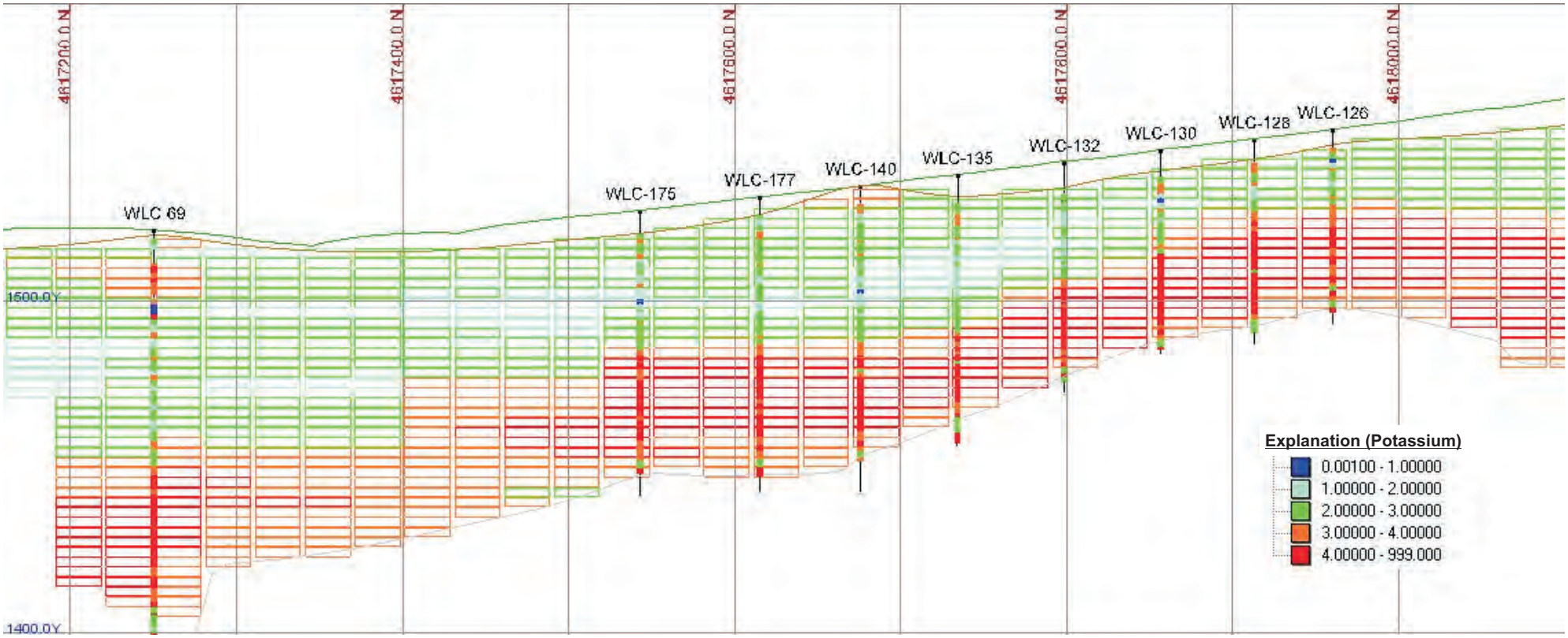
Figure 14.13 and Figure 14.14 show an N-S cross section at 411580E (looking west, Measured & Indicated blocks only) and a plan view, respectively, through the lithium OK block model. Figure 14.15 and Figure 14.16 show the corresponding cross section (looking west) and plan view of the potassium OK block model.



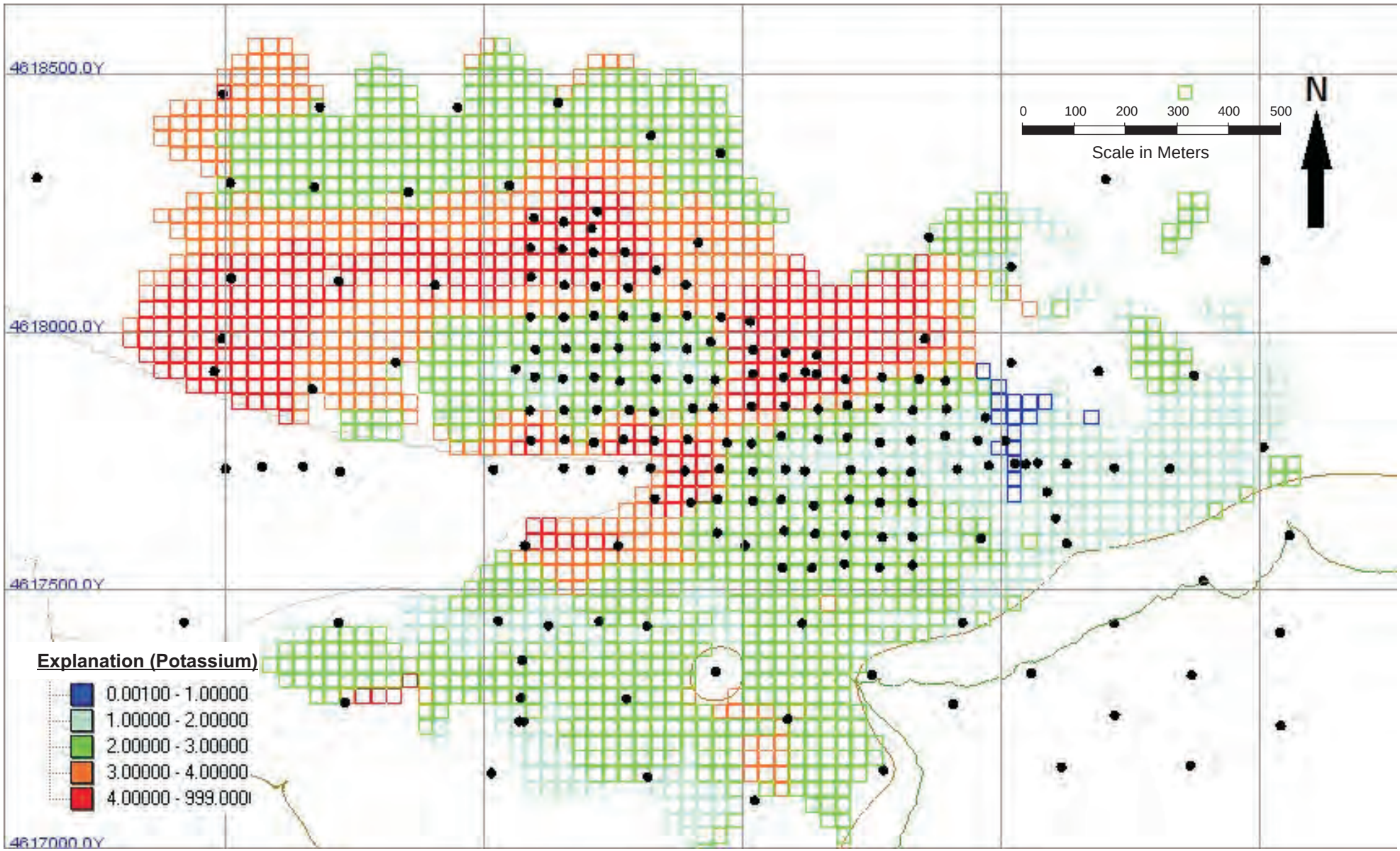
Created by: rêserva INTERNATIONAL	Prepared for: **Western Lithium Corp.**	Location: Humboldt County, NV	**Figure 14.13** **N-S Cross Section at 411,580E** **OK Model with Drillholes, Lithium %**
	Project: Kings Valley Lithium Project	Project Number: 114-311106 / Date of Issue: 01/10/2011	



Created by: rêserva INTERNATIONAL	Prepared for: *Western Lithium Corp.*	Location: Humboldt County, NV	**Figure 14.14** **Plan View - Level 1510** **OK Model with Drillholes, Lithium %**
	Project: Kings Valley Lithium Project	Project Number: 114-311106 / Date of Issue: 01/10/2011	



Created by: rêserva INTERNATIONAL	Prepared for: *Western Lithium Corp.*	Location: Humboldt County, NV	**Figure 14.15** **N-S Cross Section at 411,580E** **OK Model with Drillholes, Potassium %**
	Project: Kings Valley Lithium Project	Project Number: 114-311106 / Date of Issue: 01/10/2011	

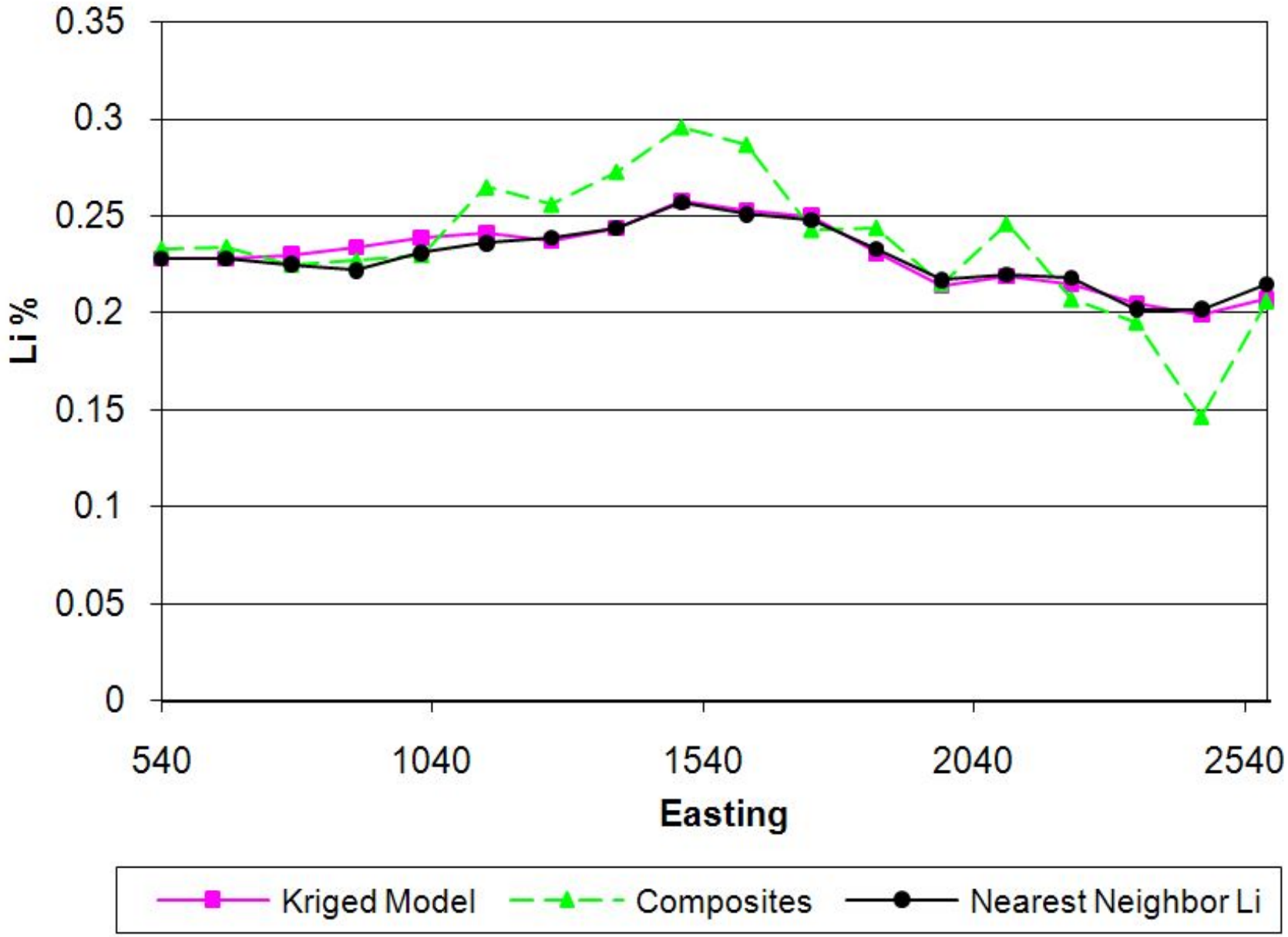


Created by: rêserva INTERNATIONAL	Prepared for: *Western Lithium Corp.*	Location: Humboldt County, NV	**Figure 14.16** **Plan View - Level 1510** **OK Model with Drillholes, Potassium %**
	Project: Kings Valley Lithium Project	Project Number: 114-311106 / Date of Issue: 01/10/2011	

14.1.6 Model Validation

Reserva completed a detailed visual inspection of the model to check for proper coding of drillhole intervals and block model cells, in both sections and plans. The coding was found to be correct. Grade interpolation was checked relative to drillhole composites and found to be reasonable.

Reserva checked the block model estimates for global bias by checking the mean nearest neighbor (NN) estimate for lithium and potassium against model OK grade estimates. Mean grades were found to match very well. Reserva also checked for local trends in the grade estimate by comparing the mean grade estimate from the NN model and composites against the OK model in 100 m wide swaths through the model on Easting and Northing, and 50m high swaths for elevation. The average grade of all the interpolated blocks (OK and NN) and composites that fall in the swath is calculated and plotted to allow a local comparison of the sample and interpolated values. The block model behaves as expected and reproduces the general grade trends well, with some evidence of the smoothing common in interpolation procedures Figure 14.17 shows an example of a swath plot for lithium on N-S sections. The pink line is the block model lithium average, the black line is the NN average and the green line is the composite average.



Created by:	Prepared for:	Location:		
rêserva INTERNATIONAL	*Western Lithium Corp.*	Humboldt County, NV		
	Project:	Project Number:	Date of Issue:	
	Kings Valley Lithium Project	114-311106	01/10/2011	

Figure 14.17
Lithium Swath Plot
Blocks vs NN Model and Composites

In summary, the lithium and potassium estimates are considered reasonable and show no marked anomalies, such as large deviations between the averaged values.

14.1.7 Mineral Resource Classification

The mineral resources of the Stage I Lens were classified consistent with the CIM Definition Standards incorporated by reference into the Canadian NI 43-101 as follows:

- *Inferred Mineral Resource*

  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 drillholes.

- *Indicated Mineral Resource*

  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 drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

- *Measured Mineral Resource*

  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 drillholes that are spaced closely enough to confirm both geological and grade continuity.

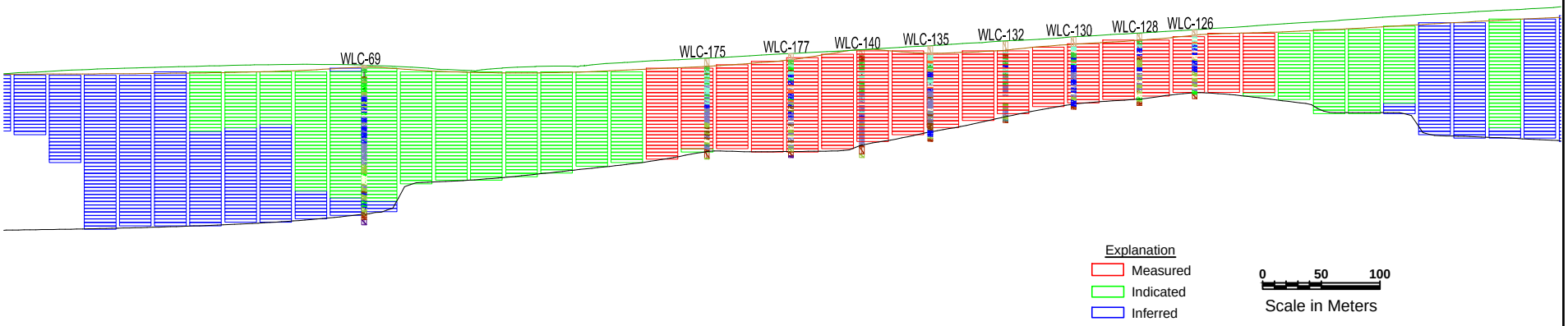
The mineralization of the Project satisfies sufficient criteria to be classified into measured, indicated and inferred mineral resources. Resources are tabulated in Table 14.1 and the criteria are listed below.

- Measured resources are in blocks estimated using at least three drillholes and five composites within a 100 × 75 m search radius in the horizontal plane and 10 m in the vertical direction.
- Indicated resources are in blocks estimated using at least two drillholes and four composites within a 200 × 150 m search radius in the horizontal plane and 20 m in the vertical direction.
- Inferred resources are blocks estimated with at least three composites within a search radius of 300 × 225 m in the horizontal plane and 30 m in the vertical plane.

Figure 14.18 shows an N-S cross section (looking west) at 411580E, illustrating the classification.

14.1.8 Risk

Reserva is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that will materially affect the mineral resource estimate.



Created by: rêserva INTERNATIONAL	Prepared for: *Western Lithium Corp.*	Location: Humboldt County, NV	**Figure 14.18** **N-S Cross-Section at 411580E Illustrating the Classification (Looking West)**
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	

Aga, Lee – T:\Mining\Projects\Western Lithium – Kings Valley PFS\Report\Figures–Drawings\Tim Carew\Fig14-18.dwg – 1/18/2012 1:16 PM

14.2 STAGE II

WLC engaged GeoSystems International, Inc. to complete a resource estimate for the Stage II Lens area within the Kings Valley Project, located in Humboldt County, Nevada. The resource estimate was made from a three-dimensional (3D) block model using commercial mine planning software, (MineSight®). The estimated elements are lithium, potassium, fluorine, and sodium, and are current as of May 15, 2010, the effective date of the estimates.

The resource estimate was completed by Mr. Mario E. Rossi, CIM, AusIMM, SME, IAMG, and Principal Geostatistician of GSI. Mr. Rossi is an independent Qualified Person as defined by NI 43-101 by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience. Also, Mr. Rossi is independent of WLC applying all of the tests in Section 1.5 of NI 43-101.

Table 14.10 shows the total estimated resources for the Stage II Lens area using a nominal 2,000 ppm Li cutoff. GSI is of the opinion that at a 0.20% lithium cutoff, the Stage II Lens has reasonable prospects for economic extraction by open-pit mining. As with other areas within the Kings Valley Project, lithium carbonate would be the primary product with potassium as a by-product. Therefore, the resources are reported in all cases using a Li cutoff.

The estimated indicated resources (Table 14.10) are approximately 95 million metric tons at 0.27% Li and 3.66% K, while the inferred resources (Table 14.11) are about 47 million metric tons at 0.26% Li and 3.83% K. In all cases, rounding errors may exist.

Indicated resources are in blocks estimated using at least 2 drill holes, 6 composites, or 3 octants within a 160 m search radius in the horizontal plane and 16 m in the vertical direction. A later smoothing algorithm was used to avoid isolated blocks of indicated within inferred, or vice-versa; this algorithm defined indicated and inferred areas with a minimum 80 x 80 x 10 m volume.

The density values are derived from the Stage II Lens determinations. The average values obtained are shown in Table 14.12; the in situ dry density used for all mineralized material (moat sediments) is 1.96 t/m^3.

The Stage II Lens estimate considers only a portion of what is known as the South Pod, centered in the area where the new WLC drilling exists. While GSI chose not to interpolate grade into areas without recent drilling, neighbouring Chevron holes indicate that there is potential to increase laterally the estimated resource tonnage with further drilling.

Table 14.10: Indicated Resources, Stage II Lens

Li Cutoff (%)	Million Metric Tonnes	Li (%)	Contained KTonnes Li Carbonate Equivalent	K (%)	Contained KTonnes K	F (%)	Na (%)
0.05	135	0.24	1,725	3.88	5,238	0.53	1.36
0.10	133	0.24	1,699	3.89	5,174	0.53	1.37
0.15	127	0.25	1,690	3.88	4,928	0.54	1.40
0.20	**95**	**0.27**	**1,365**	**3.66**	**3,477**	**0.57**	**1.55**
0.25	50	0.31	825	3.15	1,575	0.68	1.85
0.30	27	0.34	489	2.88	778	0.73	2.06

Rounding errors may exist.
Contained metal does not allow for mine or metallurgical recovery.
Tonnage factor used is 1.96 t/m^3.
Conversion factor from Li metal to Lithium Carbonate Equivalent (LCE) used is 5.323.
Economic assumptions do not include any potassium credits.
Economic assumptions for cutoff grade determination are:
- US$ 3.50 Li carbonate/lb; 60% metallurgical recovery; US$ 50/tonne processing; US$ 2.20/tonne mining

Table 14.11: Inferred Resources, Stage II Lens

Li Cutoff (%)	Million Metric Tonnes	Li (%)	Contained KTonnes Li Carbonate Equivalent	K (%)	Contained KTonnes K	F (%)	Na (%)
0.05	104	0.19	1,052	3.89	4,046	0.50	1.07
0.10	87	0.21	973	3.94	3,428	0.52	1.18
0.15	74	0.23	906	3.97	2,938	0.54	1.26
0.20	**47**	**0.26**	**650**	**3.83**	**1,800**	**0.58**	**1.43**
0.25	20	0.30	319	3.31	662	0.70	1.76
0.30	9	0.34	163	3.02	272	0.81	1.99

Rounding errors may exist.
Contained metal does not allow for mine or metallurgical recovery.
Tonnage factor used is 1.96 t/m^3.
Conversion factor from Li metal to Lithium Carbonate Equivalent (LCE) used is 5.323.
Economic assumptions do not include any potassium credits.
Economic assumptions for cutoff grade determination are:
- US$ 3.50 Li carbonate/lb; 60% metallurgical recovery; US$ 50/tonne processing; US$ 2.20/tonne mining

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues that may materially affect the resource estimates. Other relevant factors that may materially affect the resources, including mining, metallurgical, and infrastructure are well understood according to the assumptions presented in the Stage II Resource Estimate.

Table 14.12: Density Values used in the Stage II Lens Resource Model

Lithology	average of density determinations	as used in resource model
Claystone Average	1.95 t/m^3	1.96 t/m^3
Ash Average	1.99 t/m^3	
Fang. Average	1.91 t/m^3	
Silicified	1.94 t/m^3	
Alluvial	1.52 t/m^3	1.52 t/m^3
Bedrock Average	2.23 t/m^3	2.23 t/m^3

The topography used is the Nad27 DEM sheet provided by WLC, which is deemed inadequate for feasibility-level studies. GSI recommends that WLC increase the certainty of the resource estimate through the following steps:

1. Completing metallurgical test work specific to the Stage II Lens;
2. Define the importance of oxidized vs. un-oxidized material;
3. Obtain a fly-over digital topography of the Stage II Lens;
4. Prioritize infill drilling by the mining extraction sequence. Drill on a 160 m grid for indicated mineral resources and a 80 m grid for measured mineral resources. These suggested drill hole spacings could be checked and adjusted as further drilling is completed.

14.2.1 Geologic Model

Project limits and coordinates are in UTM (m) Easting and Northing values, and the block model geometry is summarized in Table 14.13. Elevation is also in meters. The Stage II Lens block size selected is 20 x 20 x 5 m, which is considered reasonable given the current drill hole spacing and that the block model represents open pit resources.

Table 14.13: MineSight Model Extent, UTM Coordinates (meters)

UTM Coordinates (m)	Minimum	Maximum	Block Size (m)	No. of Blocks
Easting	404,000	410,000	20.0	300
Northing	4,626,500	4,632,500	20.0	300
Elevation	1,500	2,200	5.0	140

The drill hole database used for geologic modeling and resource estimation was closed on December 31, 2009 with the last hole drilled in the Stage II Lens, SP-38. The Stage II Lens mineralization is controlled by volcaniclastic moat sedimentary rocks containing lithium-rich claystone. Sectional interpretations (approximately E-W and N-S) were generated by WLC from drill logs for alluvium, moat sediments (claystone), fanglomerates, silicified unit, and bedrock. Also two oxidation surfaces were interpreted, one just below alluvium, and another near the claystone/silicified interface. Additionally, a series of faults have

been interpreted by WLC based on drill hole data, and incorporated in the geologic interpretation. These sectional interpretations were digitized and transformed into strings, which were then used to create surfaces and three-dimensional solids in MineSight®.

The surfaces and solids were used to code the block model by lithology domains within MineSight®. The percentage of each block within each unit was coded in order to capture the mixtures of units at the interfaces. Table 14.14 summarizes the lithologic and oxidation units in the Stage II Lens geologic model.

The estimation domains were derived from these lithologic units, which are also shown in Table 14.14. The Li grades within the oxidized and un-oxidized claystones, as well as the smaller fanglomerate/volcanics did not warrant separate estimation domains, according to the statistical analysis discussed in Section 14.2.4 below.

Table 14.14: Lithologic and Oxidation Units modeled in the Stage II Lens

Unit	Block Model Code	Estimation Domain
Alluvial	10	10
Claystone Upper Oxidized	20	25
Claystone Un-oxidized	30	25
Fanglomerates/volcanics	40	25
Claystone Lower Oxidized	50	25
Silicified	60	60
Bedrock	70	70

Figure 14.19 shows a cross section with the interpreted geologic and oxidation units; also shown is the topography and the drill holes annotated with histograms representing Li grades in ppm, looking north. GSI checked the consistency of the interpreted domain codes against WLC drill-hole logs in section and plan and found a reasonable match with the block domain coding.

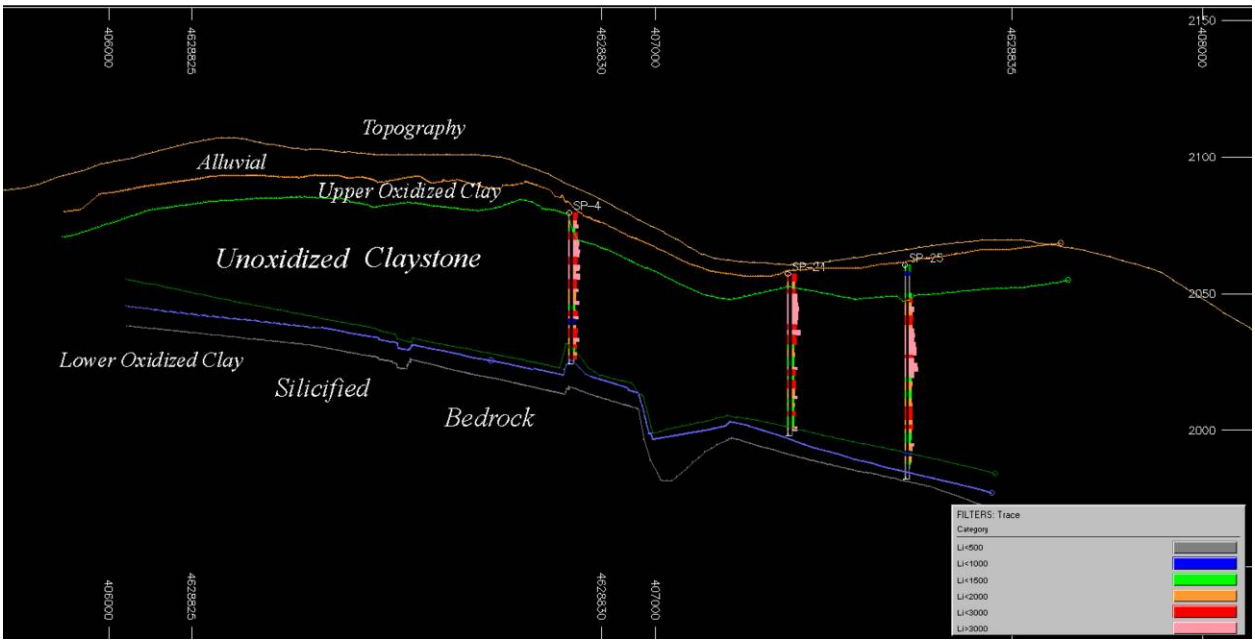


Figure 14.19: Cross Section showing interpreted lithology units.

14.2.2 Assays

A total of 51 drill holes were used for estimating grade, 38 SP core holes drilled by WLC, and 13 older Chevron holes. Only 2 of the 13 older Chevron holes (MC-84-81C and MC-84-90C) have grades for the four estimated elements, since they were sent for re-analysis by WLC in 2009. The remaining 11 Chevron holes only have lithium grades.

The total drilling was 3,019.61 m, of which 2,150.87 m were drilled by WLC (SP holes), and the remaining 868.74 m were drilled by Chevron. A total of 2,813.31 m correspond to the moat sediments domains (lithologies 2 through 5), while 109.01 m correspond to the silicified domain (lithology unit 60), and the remaining drilled meters to the alluvial or bedrock domains, which were not estimated.

Assay values were not capped prior to compositing because probability plots showed continuity and the domain composite coefficient of variation (CV; standard deviation/mean), was low. The overall CV for Li assays was 0.50, while potassium had a CV of 0.36. Figure 14.20 shows the histogram of all Li assay values. However, high grade restrictions on composites were used in the estimation, see below.

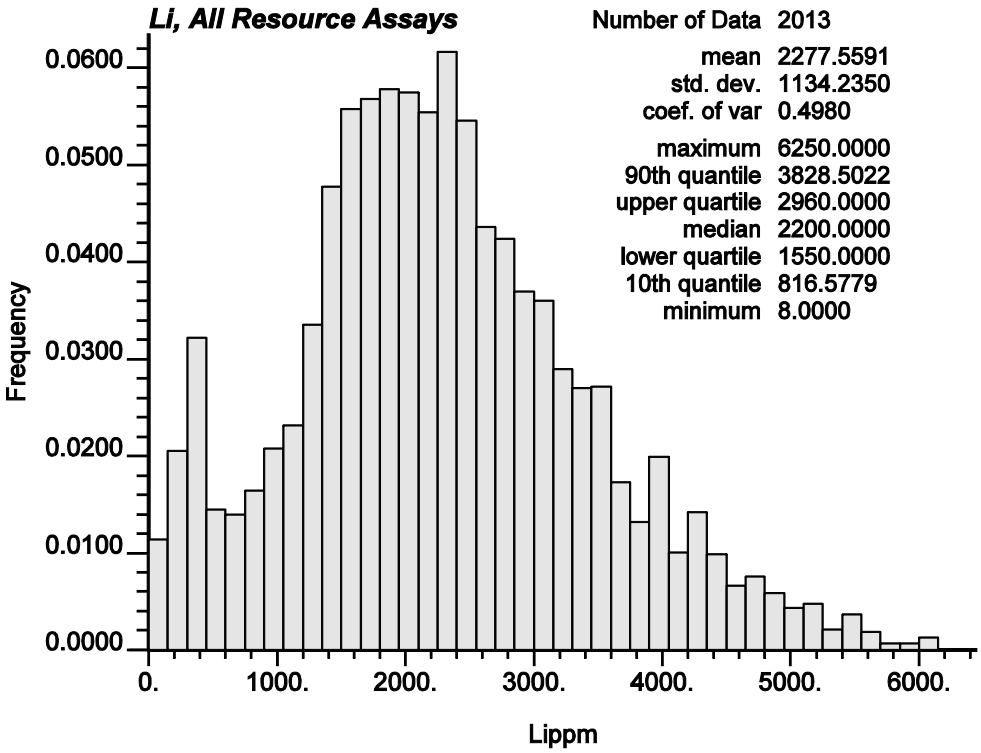


Figure 14.20: Histogram and basic statistics, all lithium assays.

Fluorine values have the particularity of being derived from two different assaying methods, based on the F concentration of the sample. For samples with greater than 2% F, a variant of the Fusion method discussed in Section 11.2 is used. This creates an artificial discontinuity, because it appears that values that are close to 2% F will tend to underestimate the F value, while values derived from the second assaying method will result in values that are higher than 2%. The end result is a relative deficiency of values around 2% F, which clearly appears artificial, see Figure 14.21. This issue needs to be addressed in future resource estimates and economic evaluations.

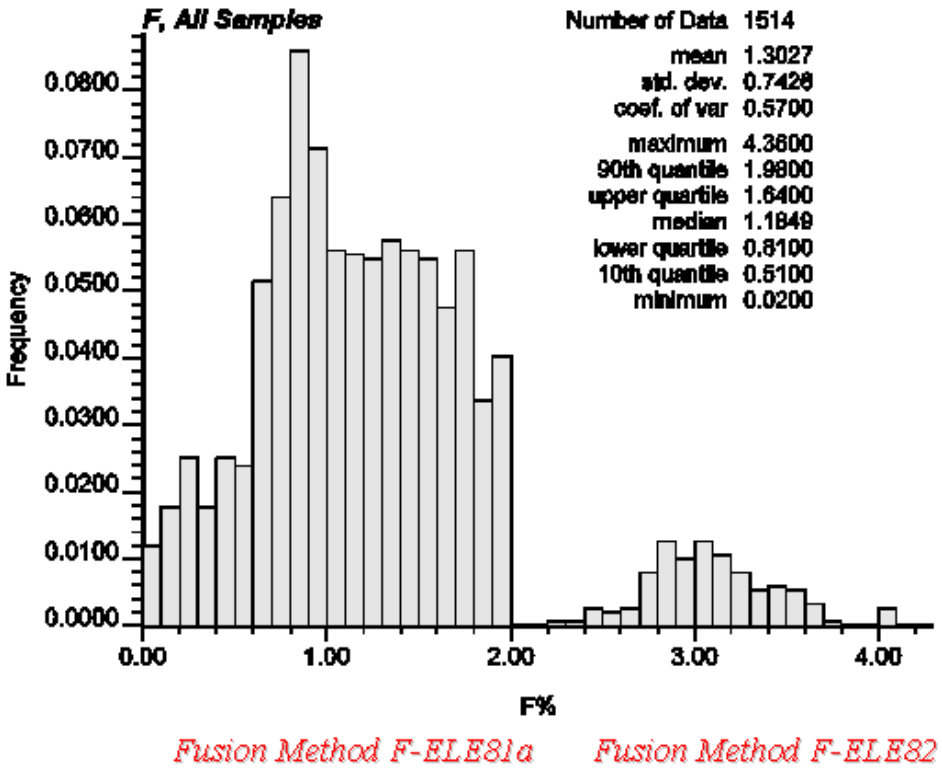


Figure 14.21: Histogram and basic statistics, all fluorine assays.

14.2.3 Composites

Assay values were composited into 1.5 m long down the hole composites and coded by lithology domains. Composites with less than 0.5 m in length were not used in grade interpolation. Since there are no assays from the alluvial zone, composites with Li grades start at the bottom of the alluvial surface, down to the last assayed interval. There were a total of the 1,655 composites within the main mineralized unit, the claystone (moat sediment) domain, the histogram and basic statistics of which are shown in Figure 14.22. Figure 14.23 shows the corresponding histogram for potassium grades within the same domain.

Additional histograms and basic statistics, including those corresponding to F and Na, are shown in Technical Reference G.

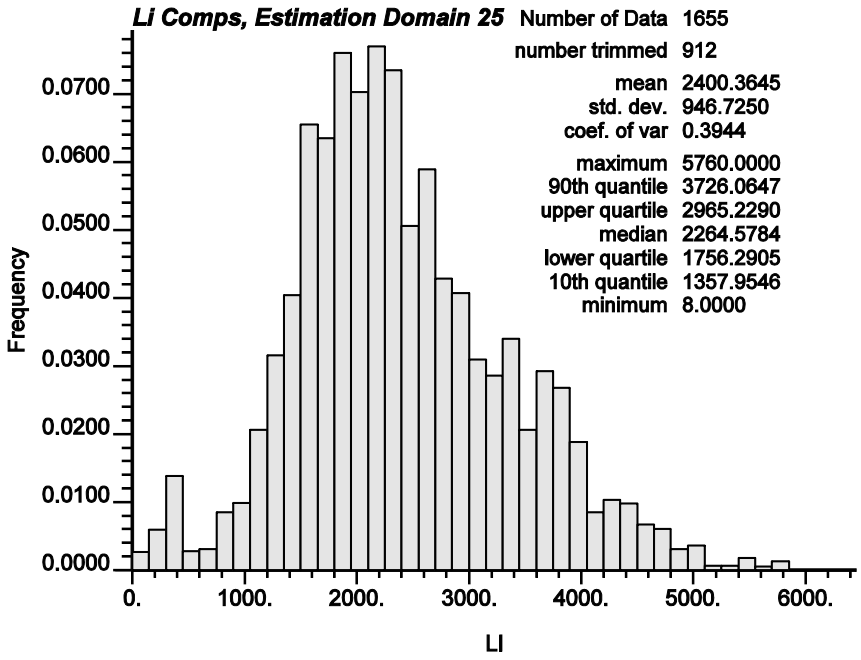


Figure 14.22: Histogram and basic statistics, lithium composites, domain 25 (Claystone).

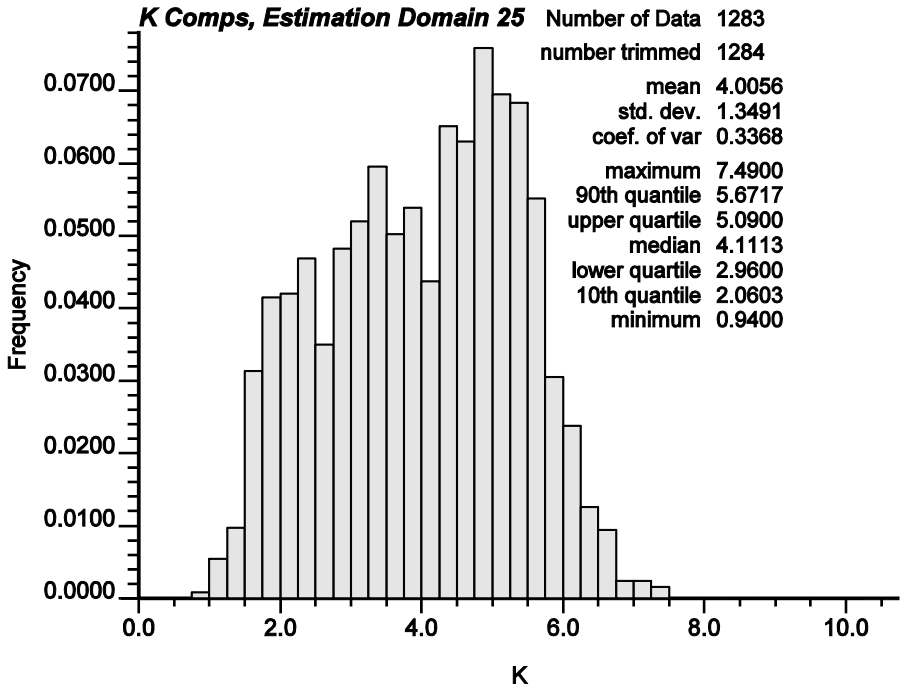


Figure 14.23: Histogram and basic statistics, potassium composites, domain 25 (Claystone).

14.2.4 Exploratory Data Analysis and Estimation Domains

In addition to the histogram and basic statistics described above, other histograms, cumulative frequency plots, and conditional mean plots were completed for all 4 elements. All CVs for each of the domains were low, so that in all cases ordinary kriging can be used in grade interpolation.

Histograms and probability plots for each of the lithology units suggest that there is little grade variation from one unit to the next. In the Case of the claystone unit (moat sediments), oxidation does not appear to impact the grade distribution; and the fanglomerate unit also has significant Li mineralization, with slightly lower grades compared to the claystone.

The only unit that was separated from the claystone was the thin silicified unit laying over the bedrock that clearly shows a drop in grade. With more drilling, it may be possible for WLC geologists to identify a higher-grade population domain within the moat sediments, probably in the form of thin layers of clays enriched in hectorite, relative to their surroundings. It was not necessary to do so for grade interpolation at this time.

Scatterplots and conditional mean plots assess the degree of correlation between variables. Conditional mean plots were done based on lithium grade classes only. These plots are similar to scatterplots, except that they show the average of each secondary variable corresponding to each Li grade class. They provide an indication of the correlation between two variables by synthesizing the scatterplot cloud. Figure 14.24 shows the potassium mean grades conditional to lithium grade ranges. The correlation between the two is excellent, and suggests that the only source of F in this deposit is the same Li-rich hectorite clay; if this assumption could be verified, prediction of F grades at the time of mining would be easier.

Figure 14.25 shows the scatterplot for the same variables composites in domain 25 (claystone). The linear correlation coefficient between the two is 0.94; note that for the higher Li grades, there appears to be an additional kick on F. Compare to Figure 14.26, which shows the scatterplot for Li-K; note that the correlation is negative, which is also observed in the conditional mean plots and the resource grade tonnage curves: higher Li grades generally mean lower K grades.

Technical Reference G presents the additional cumulative frequency plots, scatterplots, and conditional mean plots obtained for Li, K, F, and Na.

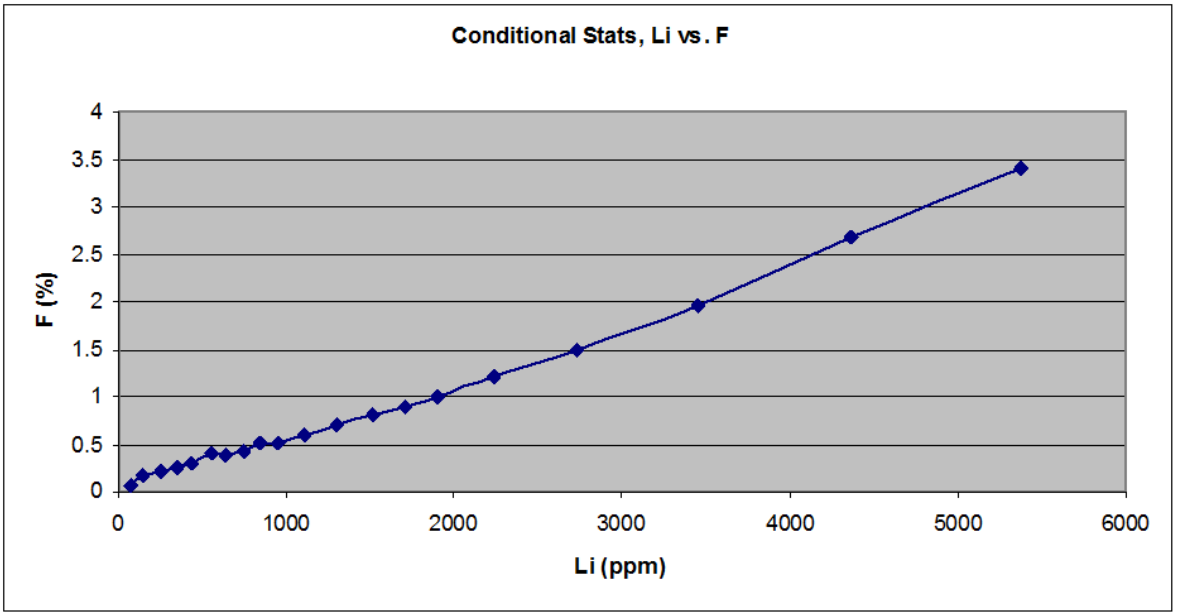


Figure 14.24: Potassium expected means for lithium grade ranges, all composites.

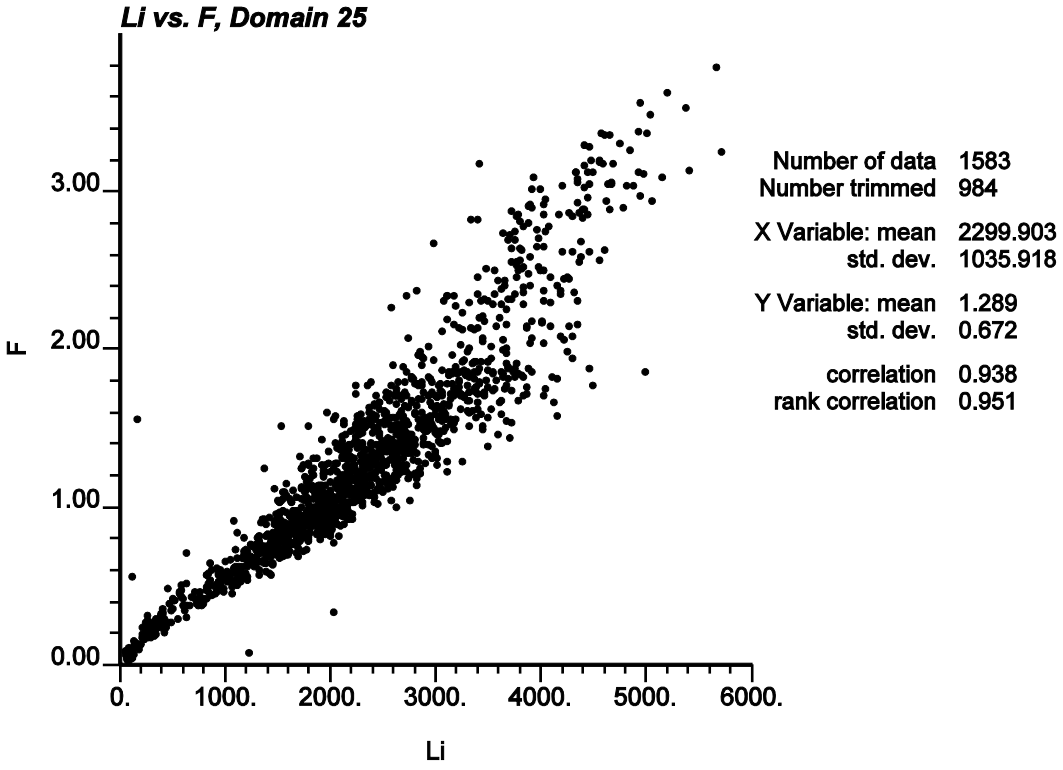


Figure 14.25: Scatterplot and basic statistics, Li vs. F composites, domain 25 (Claystone).

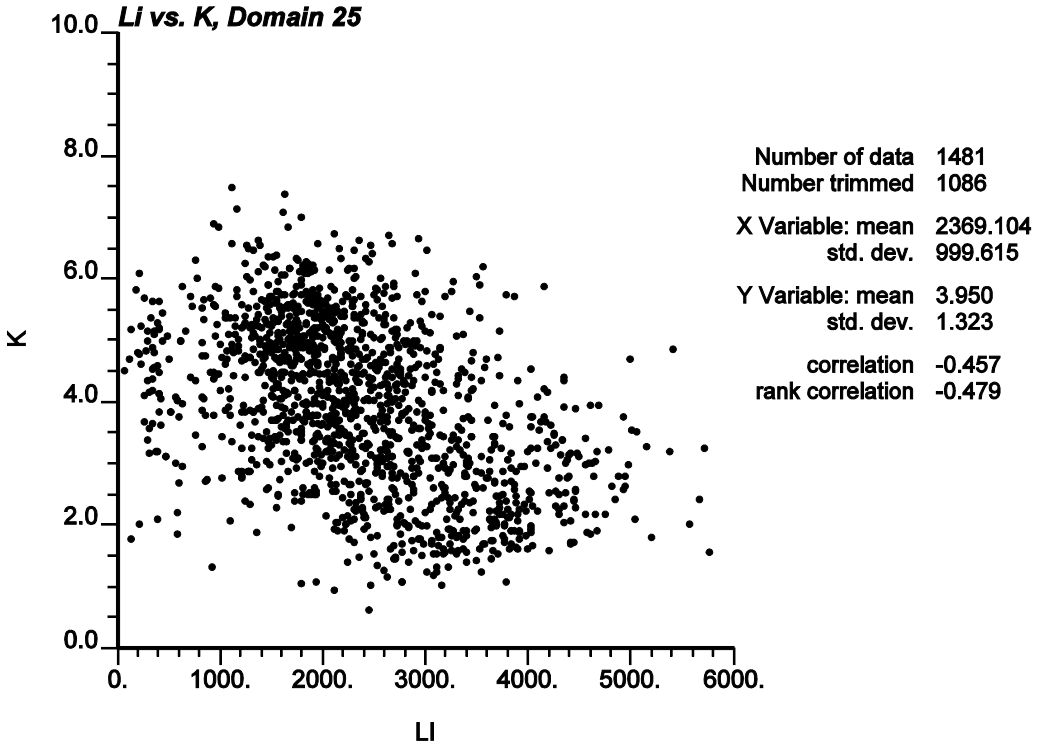


Figure 14.26: Scatterplot and basic statistics, Li vs. K composites, domain 25 (Claystone).

14.2.5 Variography

Variography graphically shows the spatial variability of an attribute. For the Kings Valley project, variography was completed using Sage 2001® software. GSI calculated experimental correlograms with 30° increments in the azimuth, 45° increment in the dip direction, and the vertical direction. These correlograms were computed for all 4 elements. The correlograms were modeled and used in the resource estimation step.

14.2.6 Estimation

The reported estimate is an Ordinary Kriging (OK) based on estimation domains. The main mineralized estimation domain is the claystone, with a small fanglomerate unit and a silicified unit just above bedrock carrying medium-range Li grades. Alluvium and bedrock material have no Li or K grades.

Two passes (outside in) were used for Ordinary Kriging (OK) grade estimation. The first pass was used to estimate blocks with more local information, searching out to what approximately meets the indicated classification criteria. The second pass was to fill the claystone envelope. Composite and blocks were matched on domain code, although Domain 60 composites (silicified) were used to estimate claystone material, but no claystone composites were used to estimate the silicified domain.

High grade composites were spatially restricted at the time of estimation, which means that although the composites were used, its spatial influence was restricted to a smaller radius. The definition of what was considered "high grade" was obtained from the cumulative frequency plots, and generally correspond to the 99% or higher percentile of the distribution. Table 14.15 shows the details of this restriction.

Table 14.15: High grade restrictions.

Variable	High grade threshold	Restricted search radii
Li	5,000ppm	14 x 14 x 14 m
K	6.5%	14 x 14 x 14 m
F	3.5%	14 x 14 x 14 m
Na	1.8%	14 x 14 x 14 m

Figure 14.27 and Figure 14.28 show an N-S cross sections and plan view respectively through the Li OK block models looking north. Figure 14.29 and Figure 14.30 show the corresponding cross sections and plan view of the K OK block models looking north. The models are shown using contoured values, and the figures can be zoomed into for more clarity. Red contours are Li grades greater than 3,000 ppm.

Note how the better potassium grades tend to be lower in elevation, compared to good Li grades. It appears that the lower claystone are richer in potassium than the upper horizons, while the opposite is true for Li, with better grades closer to the surface. This behavior explains the negative correlation found in drill holes between Li and K and discussed above.

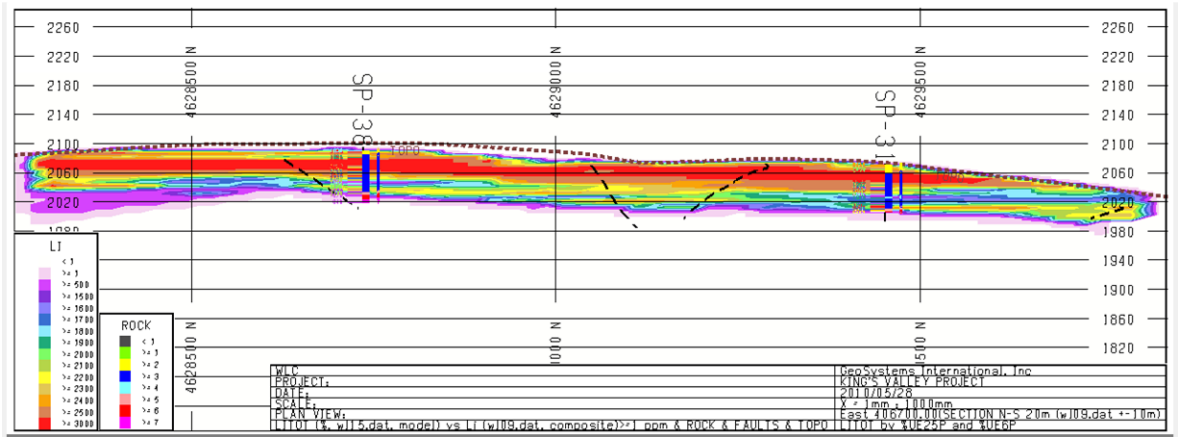


Figure 14.27: N-S Cross Section at 406700 E, Li (ppm) OK Model with drill holes.

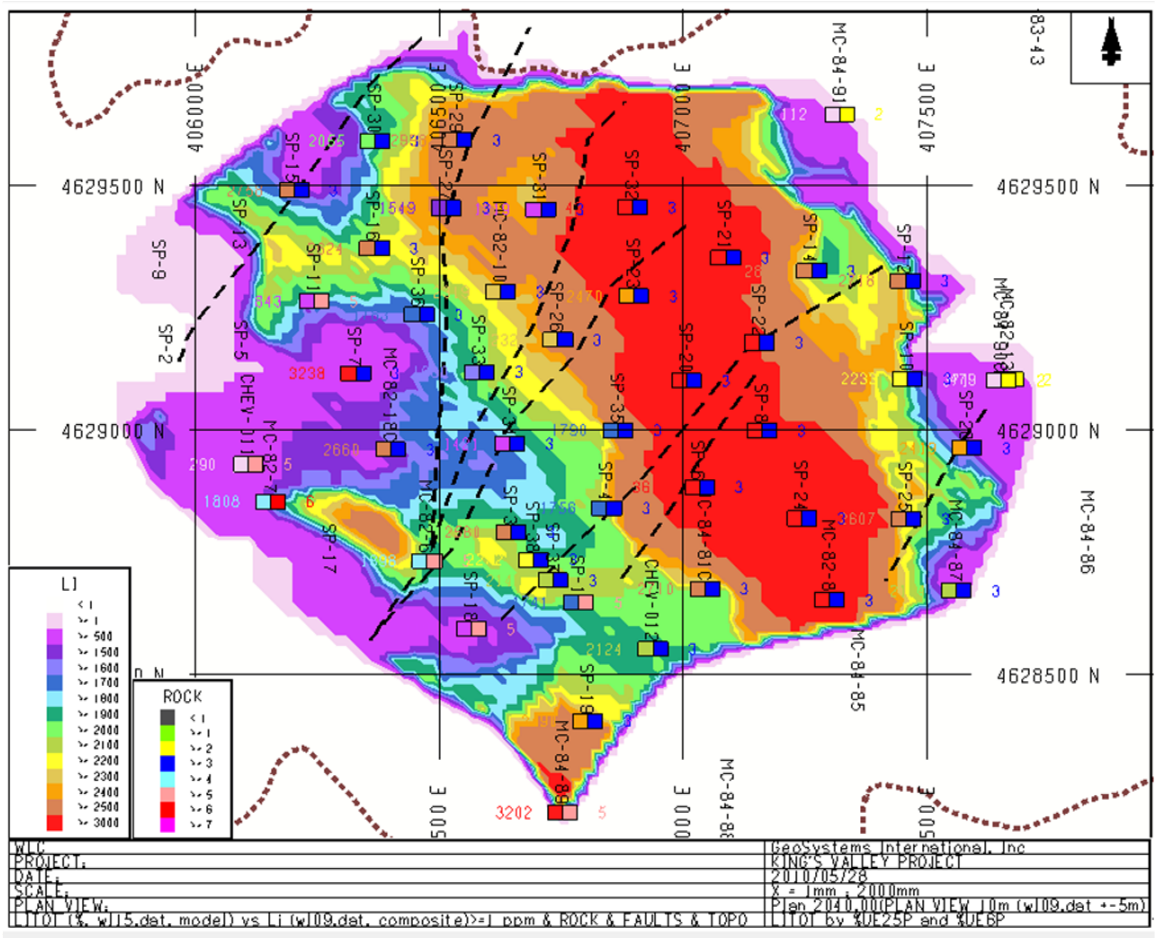


Figure 14.28: Plan view, Level 2040 m, Li (ppm) OK Model.

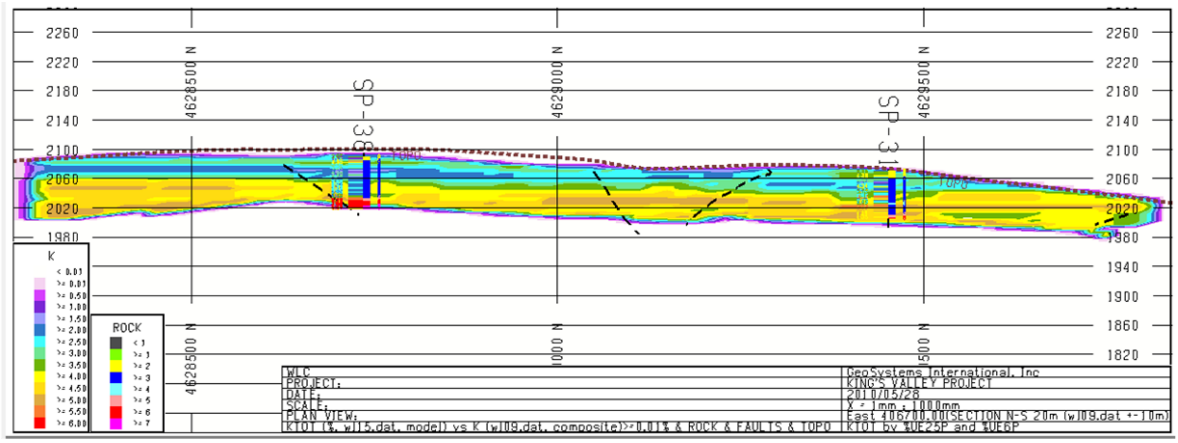


Figure 14.29: N-S Cross Section at 406700 E, K (%) OK Model with drill holes.

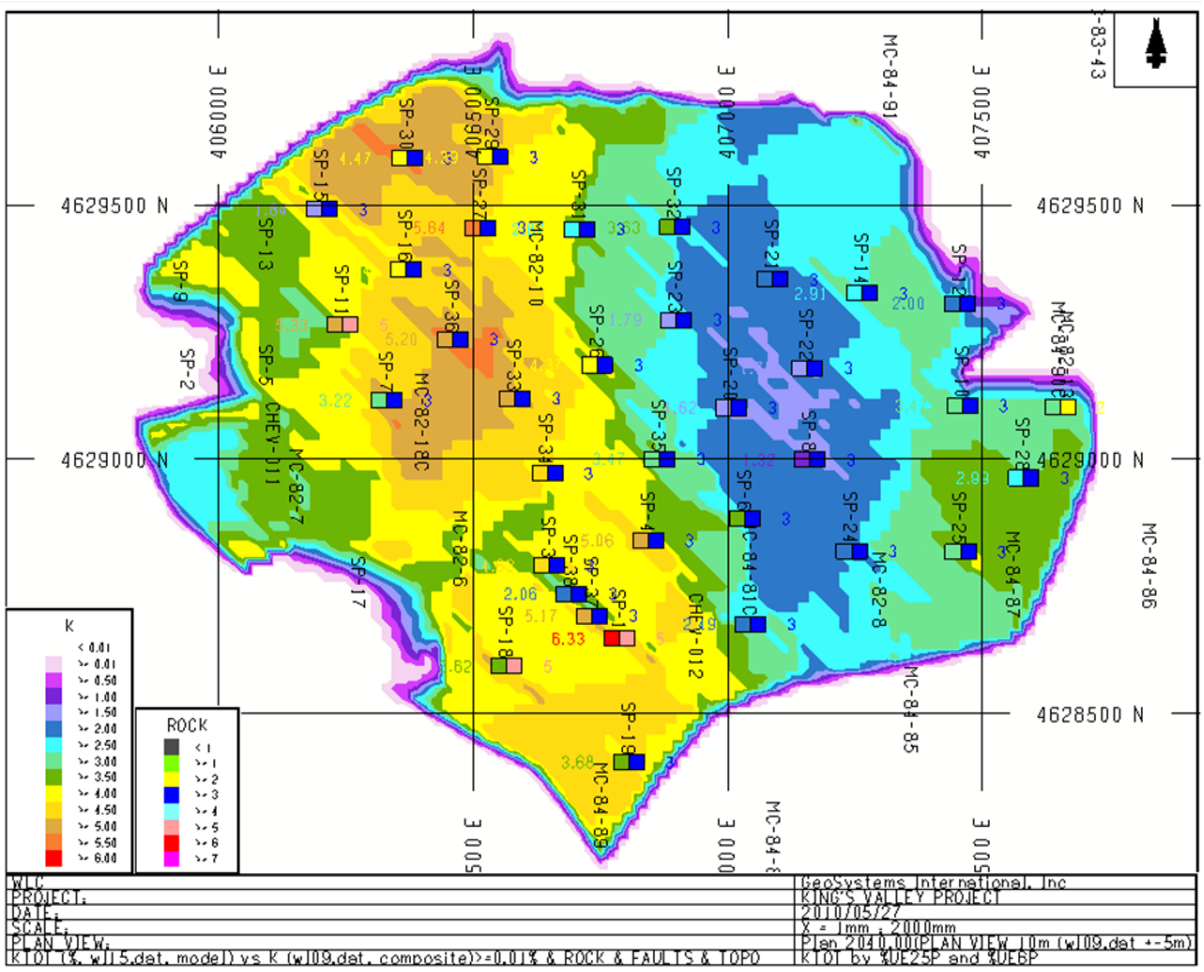


Figure 14.30: Plan view, Level 2040 m, K (%) OK Model.

14.2.7 Model Validation

GSI completed a detailed visual inspection of the model to check for proper coding of drill-hole intervals and block model cells, in both sections and plans. The coding was found to be correct. Grade interpolation was checked relative to drill-hole composites and found to be reasonable.

GSI checked the block model estimates for global bias by checking the mean nearest neighbor (NN) estimate for Li and K against model OK grade estimates. Mean grades were found to match very well. GSI also checked for local trends in the grade estimate by comparing the mean grade estimate from the NN model against the OK model in swaths through the model on easting, northing, and elevation. The block model behaves as expected and reproduces well the general grade trends. Figure 14.31 shows an example of a swath plot for Li on N-S sections. The red line is the block model Li average, the black line the NN average, and the dotted line the number of blocks (second Y axis) used in the average. Additional graphs are shown in Technical Reference H.

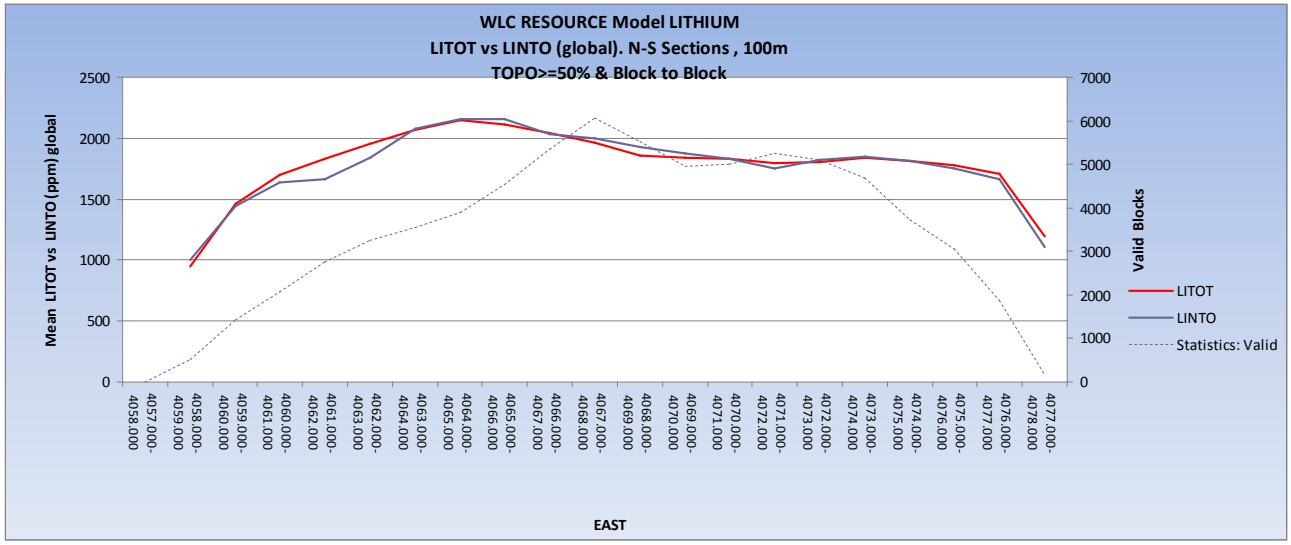


Figure 14.31: Li swath plot, blocks vs. NN model, N-S Sections.

In addition to the nearest neighbour (NN) model, an Indicator-modified OK (IMOK) model was run on Li to check if there were any opportunities to separate out dilution and increase grade. The IMOK model confirmed the OK estimate, being very similar in tonnage and grade above cutoff. The similarity of both models confirms the very good grade continuity of Li grades and suggests that, with current drilling density, there are few if any opportunities to separate out lower grade material.

In summary, the Li, K, and Na estimates are thought to be robust and show no anomalies. The potassium grade drops slightly for higher Li averages, which is somewhat unexpected, considering that it is behaving differently than what was observed for the PCD lens. The F model is also internally consistent with the data and parameters used in the estimation, but is believed to be under-representing the population around 2% F due to the assaying issue discussed elsewhere.

14.2.8 Recommended Drill Spacing

GSI verified the prior recommendation made by AMEC for the Stage I Lens using the same methodology for the Stage II Lens. AMEC uses the statistical criterion that yearly ore production grade and tonnage

should be known at least to ±15% with 90% confidence in order to fall in the indicated mineral resource category. The criterion for measured mineral resources is ±15% with 90% confidence for quarterly production.

Unfortunately, this criterion is sensitive to small changes in production rates, as well as dependent on whether the material sent to plant annually comes from a single or multiple mining fronts. GSI ran the confidence limits test for the Stage II Lens using the same estimated annual production rate of 5,000 tons per day, i.e., the assumption is that the plant will be fed entirely with South material. Applying both the results of this test, as well as practical experience and criteria derived from the correlogram models and the current drill hole spacing, the recommended nominal drill spacing for measured mineral resources and indicated mineral resources are:

1. Measured mineral resources: 80 × 80 m; and

2. Indicated mineral resources: 160 × 160 m

14.2.9 Mineral Resource Classification

Mineral resources of the Stage II Lens were classified consistent with the CIM Definition Standards incorporated by reference into NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified as an indicated mineral resource and inferred mineral resource. Resources are tabulated in Table 14.11 and the criteria listed below.

- Inferred mineral resource: For a block to be classified as an inferred mineral resource, interpolated blocks had to be within the resource envelope drawn around the WLC holes, which include some Chevron core holes. In addition, the blocks have to be within 320 m in the horizontal plane, and 32 m in the vertical plane of the nearest drill hole, with a minimum of 3 composites that result in a grade greater than or equal to 0.20 Li%, and reasonable prospects for economic extraction.

- Indicated mineral resource: For a block to be classified as an indicated mineral resource, interpolated blocks had to have at least two drill holes within 160 m distance from its centroid, with a minimum of 6 composites that result a grade greater than or equal to 0.20 Li%, and reasonable prospects for economic extraction.

- There are no measured mineral resources at this time in the Stage II Lens area.

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues that may materially affect the resource estimates. Other relevant factors that may materially affect the resources, including mining, metallurgical, and infrastructure are well understood according to the assumptions presented in the Stage II Resource Estimate.

15.0 MINERAL RESERVE ESTIMATES

15.1 STAGE I

15.1.1 Relevant Technical and Economic Considerations

In establishing a mineral reserve, Tetra Tech referred to the mineral resource model prepared by Reserva (June 2011), the PAEE (URS 2010), results from the K-UTEC process test report (August 2011) and the results of a geotechnical study published by AMEC (March 2011). Tetra Tech confirmed that the geologic resource model, the mining and the processing approaches were adequate, relevant and justified for economic extraction. Additionally, Tetra Tech has established that the Project has a reasonable expectation to establish the necessary extraction facilities and obtain the necessary government approvals. Tetra Tech converted the mineral resource into a mineral reserve using what it considered the economically minable part of the Measured and indicated Mineral Resource

15.1.2 Pit Limit Analysis

For the pit limit analysis and determination of the optimum pit process, Tetra Tech used the block model generated and reported by Reserva in the mineral resource section of Stage I PFS, effective as of June 28, 2011. The Lerchs-Grossmann (LG) Gemcom Whittle™ Strategic Mine Planning Software (Whittle) was used to develop the pit shells. Mineral resources classified as Measured and Indicated were considered as potential ore throughout the LG evaluations. The extent of the block model used in the pit analysis for the mineral reserves falls within the reported mineral resource estimates. Block grade values of %Li, %K, and %Na were used, along with a bulk density model, and model for material classification (measured, indicated and inferred).

15.1.3 Pit Limit Model Evaluation Parameters

The Project deposit is situated at a relatively shallow depth and amenable to open pit mining methods that are similar to those used at operating clay mines in the upper central United States. Mineralized layers of lithium bearing claystone occur in horizontal and sub-horizontal layers and are interbedded with ash layers. The mineralized zone is overlain by a layer of waste alluvium which varies in thickness from 0.3 m to 11.3 m with an average thickness of 4.9 m, and by a low-grade (<0.32% lithium) claystone on the order of 10 m thick in the southeast portion of the deposit, increasing to an approximate maximum of 40 m in the northwest part of the deposit. The mining methodology considered consists of overburden removal using truck/shovel bulk mining and the removal of ore using selective mining techniques to minimize dilution.

Tetra Tech reviewed the 2010 PAEE by URS and found the mining approach be reasonable. To advance the project to PFS level cost parameters were adjusted to what was deemed appropriate for the current timeline. The mining cost and process costs were increased slightly from the PAEE (URS 2010) to account for a lesser economy of scale and increases in energy costs since the publication of the report. Mining costs are based on internal Tetra Tech databases of other projects similar in size. Additionally, Tetra Tech used a lower mining recovery factor (95%) to account for loss of ore due to the inability to achieve perfect separation of ore and waste because of the layered nature of the deposit. Process recovery was adjusted upward to match the results from WLC's extensive testing program. Additionally, the results of the geotechnical study performed by AMEC entitled "Prefeasibility Level

Geotechnical Study Report, Kings Valley Lithium Project Humboldt County, Nevada" (March 2011), identified an inter-ramp pit slope of 41 degrees as appropriate for pit walls up to 90 m high. Although potassium and sodium grades contribute roughly 25% of the revenue stream, lithium is the main profit source and pit optimization was based solely on modeled lithium grade. After the pit designs were complete, the grades for potassium and sodium from the 3-D blocks that are above the ore cutoff are reported along with the lithium grade.

The pertinent parameters used in the 3-D Lerchs-Grossman open pit optimization program are shown in Table 15.1 below.

Table 15.1: Gemcom Whittle Pit Optimization Parameters

Assumption	Value	Units
Pit Slopes	41	Degrees
Mining Cost (Overburden)	1.50	US$/Tonne
Mining Cost (Other)	2.75	US$ /Tonne
Mill Cost	66.00	US$ /Tonne Ore
Selling Price	13.89	US$ /lb Lithium
Mining Recovery	95.0	%
Process Recovery	87.2	%

15.1.4 Cutoff Grade Modeling Methods

Tetra Tech calculated the breakeven mining cutoff, based on the parameters in Table 15.1, to be 0.23% Li. Two different pit shells were selected from the family of economic shells that were generated by the Whittle program. At the request of WLC, a cutoff of 0.32% Li, which is slightly higher than the breakeven cutoff, was applied to the material within the shells in order to generate an average ore grade of 0.40% Li. Selection criteria were based on finding a shell that contained sufficient ore at the 0.32% cutoff to supply the mill for 20 years of production. For Case 1, this requirement was 13.8 million dry tonnes of ore. For Case 2, the requirement was 25.6 million dry tonnes of ore.

To check the validity of applying the higher cutoff within an optimized shell, a second Whittle optimization was performed in which a fixed cutoff of 0.32% Li was specified for ore. Two shells were chosen from this optimization that contained roughly the same amount of ore as was in the original Whittle shells at the 0.32% Li cutoff. The two newer shells appeared to be almost identical in size and shape to the Shells 8 and 10 that were chosen as a basis for ultimate pit design. Furthermore, the average ore grade and strip ratios are nearly identical. A comparison of the four shells is shown in Table 15.2 below.

Table 15.2: Whittle Shell Summary

	Whittle Shell Number 8 (Case 1)	Whittle Shell Number 16 Based on 0.32 Fixed Cutoff	Whittle Shell Number 10 (Case 2)	Whittle Shell Number 21 Based on 0.32 Fixed Cutoff
Measured + Indicated Ore Tonnes (000s)	13,333	14,265	27,231	27,052
Waste Tonnes (000s)	36,594	38,743	89,430	87,579
Average %Li	0.408	0.409	0.397	0.399
Average %K	3.84	3.77	3.89	3.80
Average %Na	1.46	1.42	1.36	1.32
Strip Ratio	2.74	2.72	3.28	3.24

Applying a cutoff higher than the breakeven cutoff to material within an economic shell lowers the net value of that shell if time value of money is not taken into account. However, application of a higher cutoff sometimes increases the net present value of a project. This is especially true in a Case such as the Project where the processing cost is on the order of ten times the mining cost per tonne of ore.

The Case 1 Whittle Shell 8, Case 1 ultimate design based on Shell 8, typical E-W and N-S sections through the Case 1 final designed pit and Case 1 Phase sequence are shown in Figure 16.1, Figure 16.2, Figure 16.3, Figure 16.4 and Figure 16.5.

The Case 2 Whittle Shell 10, Case 2 ultimate design based on Shell 10, typical E-W and N-S sections through the Case 2 final designed pit and Case 2 Phase sequence are shown in Figure 16.6, Figure 16.7, Figure 16.8, Figure 16.9 and Figure 16.10.

15.1.5 Mineral Reserve Estimate

Two different mine plans were developed for the Project to test the expandability of the project. The first plan, Case 1, delivers 690,850 tonnes of dry mill feed each year for 21 years. The second plan, Case 2, delivers 690,850 tonnes of dry mill feed for three years, increasing to 1,379,700 tonnes of dry mill feed for another 17 years. Measured and Indicated material from the resource model was converted to a Proven and Probable mineral reserve for both Case 1 and Case 2, by applying the pit design for each.

For Case 1, the outlines of each mining phase (1 through 5) were applied to the three dimensional grade model. Material that fell inside the phases was classified as ore if it was of a Measured or Indicated resource class and it was 0.32% Li or higher. Material that fell inside the phases that was not classified as ore was classified as waste.

For Case 2, the outlines of each mining phase (1 through 8) were applied to the three dimensional grade model. Material that fell inside the phases was classified as ore if it was of a Measured or Indicated resource class and it was 0.32% Li or higher. Material that fell inside the phases that was not classified as ore was classified as waste.

Results are tabulated in Table 15.3 and Table 15.4 and are subdivided by logical mining phase.

Table 15.3: Mineral Reserve for Case 1

Proven Ore Tonnes (000's)	12,153		
Proven Average %Li	0.405	Proven Tonnes (000's) LCE	262
Proven Average %K	3.83	Proven Tonnes (000's) K	465
Proven Average %Na	1.46	Proven Tonnes (000's) Na	177
Probable Ore Tonnes (000's)	1,796		
Probable Average %Li	0.396	Probable Tonnes (000's) LCE	38
Probable Average %K	3.77	Probable Tonnes (000's) K	68
Probable Average %Na	1.45	Probable Tonnes (000's) Na	26
Proven+Probable Ore Tonnes (000's)	13,949		
Proven+Probable Average %Li	0.404	Proven+Probable Tonnes (000's) LCE	300
Proven+Probable Average %K	3.82	Proven+Probable Tonnes (000's) K	533
Proven+Probable Average %Na	1.46	Proven+Probable Tonnes (000's) Na	203

Summary of Dry Ore Tonnes for Proven + Probable
Cutoff = 0.320 %Li
95% Mine Recovery Factor Applied

Table 15.4: Mineral Reserve for Case 2

Proven Ore Tonnes (000's)	14,937		
Proven Average %Li	0.400	Proven Tonnes (000's) LCE	318
Proven Average %K	3.85	Proven Tonnes (000's) K	575
Proven Average %Na	1.37	Proven Tonnes (000's) Na	204
Probable Ore Tonnes (000's)	12,198		
Probable Average %Li	0.388	Probable Tonnes (000's) LCE	252
Probable Average %K	3.93	Probable Tonnes (000's) K	479
Probable Average %Na	1.36	Probable Tonnes (000's) Na	165
Proven+Probable Ore Tonnes (000's)	27,135		
Proven+Probable Average %Li	0.395	Proven+Probable Tonnes (000's) LCE	570
Proven+Probable Average %K	3.88	Proven+Probable Tonnes (000's) K	1054
Proven+Probable Average %Na	1.36	Proven+Probable Tonnes (000's) Na	369

Note: Ore tonnes and grade will not match exactly with detailed schedule because variable cutoffs were used during first few years of operation and the average cutoff used for the Case 2 mine plan is 0.327% Li (see Table 16.7).

15.1.6 Potential Effects of Relevant Factors

Any increase in product price could potentially increase the amount of mineable reserve, since the breakeven cutoff could be lowered.

An increase in overall process recovery would have a similar effect, since that would also lower the breakeven cutoff and allow lower grade material to be mined at a profit.

A decrease in operating costs such as fuel, electricity, and raw materials (reagents) could decrease the breakeven cutoff and potentially increase the mineable reserve.

If additional geotechnical studies show that the overall pit slopes can be steepened from the current 41 degree overall slope, this can potentially decrease the strip ratio and potentially increase the mineable reserve.

15.2 STAGE II

The Stage II Resource Estimate report is preliminary in nature and addresses only the resource estimate. As such, this section is not applicable to the Stage II area.

16.0 MINING METHODS

16.1 STAGE I

16.1.1 Criteria and Design Parameters

The Project deposit is situated at a relatively shallow depth and amenable to open pit mining methods. Mineralized layers of lithium ore occur in horizontal and sub-horizontal claystone layers, interbedded with low grade ash layers. A Wirtgen surface miner was selected as suitable equipment to selectively mine the lithium claystone beds. Overburden would be mined using conventional truck and shovel methods. Per WLC's directive, this study assumes that the overburden would be mined via free-digging, no blasting would be required. Tetra Tech has not independently confirmed the viability of free-digging.

Based on Tetra Tech's recommendation, an overall mining recovery factor of 95% was applied. This factor was considered to be reasonable for the mining bench height chosen, and the type of mining equipment used. No dilution factor was applied to ore grade as it was assumed that sufficient dilution was built into the 3-D grade model by using bench composited assay data.

Two operating scenarios were chosen for study. The first, Case 1, calls for the processing of 2,100 dry tonnes of ore per day, 365 days per year, at 90% availability. The target mine life of operation is 21 years, and the nominal average mill feed grade is 0.40% lithium. The second, Case 2, calls for processing of 2,100 tonnes of ore per day, 365 days per year, at 90% availability for the first 3 years. Then, beginning in Year 4, the production rate is doubled to 4,200 tonnes of ore per day, 365 days per year, at 90% availability for an additional 17 years. The nominal average mill feed grade is 0.40% lithium for Case 2.

16.1.1.1 Geotechnical Data

WLC engaged AMEC of Sparks Nevada to perform an independent geotechnical investigation to support the preliminary feasibility effort. Preliminary results of the AMEC study were published in a report entitled *Prefeasibility Level Geotechnical Study Report, Kings Valley Lithium Project Humboldt County, Nevada* (March 2011). Final results were published in an addendum entitled *Addendum to Prefeasibility Level Geotechnical Investigation, Supplemental Pit Slope Recommendations and Geotechnical Recommendations Pertaining to Groundwater Data; Kings Valley Lithium Project, Humboldt County, Nevada* (May 10, 2011). The objectives of the AMEC study were to:

- Characterize the subsurface soil and bedrock at the site
- Provide preliminary earthwork and foundation recommendations
- Provide preliminary mine pit slope design recommendations
- Provide recommendations for design level geotechnical studies

Pit slope inputs used for the open pit design are based on a final report issued by AMEC. The unit weight, friction angle and cohesion were measured for the three main rock types encountered in the deposit, which are alluvium, basalt and claystone/ash layers. Direct shear testing was also performed.

Limit equilibrium stability analyses were performed using Spencer's method of analysis with the computer program SLIDE.

Slope recommendations presented in the AMEC report are based on limited data, but are considered adequate for a preliminary feasibility level of design. Limited oriented structural data was obtained during the study, and these analyses are largely predicated on use of resource geologic cross sections for interpretation of the geology and structure around the pit. In addition, limited groundwater data is available, and the stability of the pit slopes has been analyzed with the assumption that groundwater will have no adverse effect to the stability of the slopes.

On April 25, 2011, personnel from Tetra Tech performed a site visit for the purpose of verifying sampling methods and other items related to the geotechnical investigation done by AMEC.

Tetra Tech determined that the sampling and calculations methods used were appropriate for the level of study. The primary result was the recommendation of an overall pit slope angle up to 41 degrees for an overall slope height of 90 m. The analysis included a probability-based component in the calculation. This result is commensurate with the weak overburden that was tested to identify shear strength, cohesion, friction angles, unconfined compressive strength and other parameters. The soils investigation resulted in the recommendation that most foundations could be supported by conventional spread foundations and floor slabs with certain recommended engineering design. The report concluded that the TSF and waste dump can be established on properly prepared existing soils sites. Seismicity calculations were properly reduced to a maximum credible earthquake and operating basis earthquake for design purposes. The report concluded with appropriate geotechnical recommendations that would be needed for a feasibility study.

Results of the AMEC report were incorporated into the Stage I PFS and presented in the Section 26.6 of this report.

16.1.1.2 Pit Wall Design

The open pit designs have all been constructed using an overall interramp slope angle of 41 degrees. This slope is the maximum angle that can be used for a medium consequence of failure (20% probability of failure) and a maximum highwall height of 90 m.

Designs are all based on single benching, using a 3 m bench height. This is a conservative approach, and it may be possible to combine multiple benches, especially in the upper zones of the pit where waste is being bulk mined. In these areas, a possible pit wall configuration would be to use triple benching with a 65 degree face angle and 6.4 m wide catch bench.

16.1.1.3 Groundwater

The groundwater regime at Kings Valley is currently being characterized at present. WLC has engaged Schlumberger Water Services to conduct an extensive groundwater investigation and prepare a report. The results of this investigation were not complete and available for inclusion in the Stage I PFS.

16.1.1.4 Haul Roads

Haul roads are 30 m wide with a maximum grade of 10%, measured along the inside of a curve. The designed road widths are more than adequate for the largest haul trucks that would be used

(Caterpillar 773). The 773C truck is 5.4 m wide and has a tire height of 2.2 m. For haul roads inside the pit, a single safety berm on the pit side of the roadway would be required to be at least mid-axle height of the largest self-propelled mobile equipment traveling on the road. Tetra Tech assumes that safety berms can be created at a 1.5 horizontal to 1 vertical slope using run-of-mine material. A flat top on the berms of 0.33 m is assumed, and a berm height of 1.2 m allows for one-half the tire height plus a 10% margin. The resulting width of the required safety berm is 3.93 m.

Haul roads inside of pits where only one safety berm is required are designed to be 30 m wide for two-way traffic. Subtracting berm widths, this provides 4.8 times the width of a 773C truck.

In most cases, ramps were constructed in areas that would later be mined out, to minimize the tonnes mined external to the ultimate shell. Exceptions include the Case 1 haul road for Phase 4, and the Case 2 haul road for Phase 3 which were both constructed in waste external to the ultimate pit boundary which marked the extent of ore. Also, the haul roads in the west highwall for the final phase of both Case 1 and Case 2 were constructed external to the shell in waste. It is possible that refinements to the schedule could be made, and final access could be via roads constructed with fill material on the internal backfilled waste pile. This option was not explored for this technical report.

16.1.1.5 Pit Limit Determination

The ultimate pit limit was determined using GEMCOM Whittle™ Strategic Mine Planning software (Whittle). Only grade model blocks that are classified as measured or indicated were considered as ore. Lithium was the only revenue-generating product considered in the Whittle™ analysis. Pertinent economic factors for the Whittle run are shown below in Table 16.1. Mining cost of overburden is for waste material that lies above the lithium orebody. Mining cost (Other) is the cost for mining ore and also any waste that is intermixed within the main orebody.

Table 16.1: Whittle™ Pit Optimization Parameters

Assumption	Value	Units
Pit Slopes	41	Degrees
Mining Cost (Overburden)	1.50	US$/Tonne
Mining Cost (Other)	2.75	US$/Tonne
Mill Cost	66	US$/Tonne Ore
Selling Price	13.89	US$/lb Lithium
Mining Recovery	95.0	%
Process Recovery	87.2	%

A series of economic shells were generated by the Whittle program. These shells were used to define the ultimate limits of the pit, as well as to determine the location of the first mining phase. Shell 8 of the optimization contained sufficient ore at an average grade of 0.42% Li to feed the mill for over 20 years. A portion of Shell 8 included an isolated pit in the southwest corner of the property. WLC requested that this target not be included as part of the mine design because it is situated in a separate drainage area. Results of the Whittle run are shown in Table 16.2.

Table 16.2: Whittle Output

Shell #	Li_2CO_3 Price US$/lb	Total (tonnes)	Ore (tonnes)	Strip Ratio	%Li
1	6.87	39,103	9,183	3.26	0.59
2	7.74	89,891	18,365	3.89	0.57
3	7.99	606,006	123,966	3.89	0.51
4	8.26	1,011,459	202,019	4.01	0.5
5	8.55	2,351,264	500,457	3.7	0.48
6	8.86	7,200,077	1,469,232	3.9	0.46
7	9.20	41,085,820	7,001,809	4.87	0.45
8	9.56	52,598,072	10,247,893	4.13	0.44
9	9.95	64,393,853	14,315,829	3.5	0.42
10	10.37	121,072,410	26,290,070	3.61	0.4
11	10.83	144,223,227	36,303,567	2.97	0.39
12	11.34	221,661,015	58,081,040	2.82	0.37
13	11.89	256,095,331	72,137,027	2.55	0.36
14	12.50	286,180,146	84,207,218	2.4	0.35
15	13.89	337,957,115	104,230,095	2.24	0.34
16	15.28	359,858,019	116,502,773	2.09	0.33
17	16.67	386,713,857	126,098,694	2.07	0.32
18	18.06	398,711,831	132,926,032	2	0.32
19	19.45	407,516,220	137,723,993	1.96	0.31
20	20.84	413,214,216	141,856,207	1.91	0.31
21	22.22	417,267,258	144,904,864	1.88	0.31
22	23.61	421,015,265	147,205,130	1.86	0.3
23	25.00	424,155,045	148,839,651	1.85	0.3
24	26.39	426,180,301	150,474,171	1.83	0.3
25	27.78	428,021,959	151,640,374	1.82	0.3

Note: Shell 8 was used as basis for Case 1; Shell 10 was used as basis for Case 2.
Results include certain smaller pits to the southwest that were not included in the final design.

16.1.1.6 Cutoff Grade

The breakeven cutoff grade for the economic parameters assumed in the pit shell analysis was calculated using the following formula:

Value of Ore = Cost of extracting and processing ore

($13.89/lb Li)*(22.04 lb Li/% Li)*(95% mining recovery)*(87.2% process recovery) = (66+2.75)

Using the above equation, the percent Li for the breakeven cutoff is 0.27% Li.

At the request of WLC, the nominal feed grade to the mill was required to be 0.40% Li. In order to achieve this average feed grade from the Kings Valley Lithium deposit, it was necessary to raise the cutoff grade for Case 1 to 0.32% Li. For Case 2, the variable cutoff grade was used (See Table 16.7). The average cutoff grade for Case 2 is 0.327% Li. Therefore, based on the design economics, there is

some material between 0.27% Li and the applied mining cutoff that could be processed at a profit. For this study, this material was considered to be waste and it was not separately stockpiled.

The objective of running the Whittle program was to identify economic shells which contained sufficient ore at an average grade of approximately 0.40% Li to mill for roughly 20 years. After deducting material in the Whittle shells that fell in unmineable areas except the isolated southwest portion and applying the higher design cutoff, it was found that Shells 8 and 10 were a good fit for Case 1 and Case 2 respectively.

16.1.1.7 Ultimate Design Pits

Ultimate pits, phases, and annuals were constructed using PolyMap™ software. For each case, an ultimate pit was first designed, without ramps that defined the final extent of the pit. Whittle shell outlines were used as a guide. Pit walls were smoothed to remove noses and smooth out bends, and pit bottoms were adjusted to allow for sufficient equipment operating room. After completing the ultimate design without ramps, mineable reserves were calculated within the designed shape to verify that sufficient ore was contained in the pit for approximately 20 years of production.

16.1.1.8 Mining Phases and Scheduling

Within the ultimate shell, a series of smaller mining phases were designed which included ramp access. Phases were generally developed in a southeast to northwest pattern. A minimum pushback width of 50 m was used between phases.

When the mining phase designs were complete, a detailed material movement schedule was generated using the MicroMODEL® Mansched program. The scheduling process "mines" the phases in order, and the general procedure is to first open up sufficient ore in the first mining phase by prestripping sufficient waste prior to production (pre-production). Then, as ore is being mined from the current phase, the next phase or phases are being concurrently prestripped to expose ore faces that would be mined when the current phase is mined out. The schedule needed to meet the required mill feed each year and yearly waste mining tonnages were adjusted by forward smoothing in order to level out the total material movement in each year.

16.1.1.9 Annual Dig Plans

After the detailed annual schedule was in place, annual dig plans were designed to match the annual schedule as close as possible with regard to ore tonnes, head grade, and waste tonnes. Generating the dig plans ensures that the proposed schedule is physically achievable with regards to access, number of benches mined per year, etc.

16.1.1.10 Waste Dump Design

The required waste dump volume was calculated based on the total tonnes of waste to be disposed, the average in situ density of the waste, and the assumption of an overall swell factor of 30% (swell plus subsequent compaction). Final dump faces were designed using a 3H:1V slope (18.4 degrees) while working faces were designed using a 2H:1V slope (26.6 degrees). Working faces would later be flattened to a 3H:1V slope when mining has finished. Haul roads in waste dumps were designed using the same criteria as in-pit haul roads which is 30 m width and 10% maximum grade. Waste dump parameters are summarized in Table 16.3.

Table 16.3: Waste Dump Parameters Summary

Waste Dump Design Criteria	
Final Slope Angle (degrees)	18.4
Working Slope Angle (degrees)	26.6
Haul Road Width (m)	30
Haul Road Maximum Grade (%)	10
Overall Swell Factor (%)	30

Once the detailed schedule of ore and waste movement was formulated, a plan for disposing of the waste rock was needed. From the annual plans, the year in which backfilling of waste into the pit first became feasible was identified. The amount of external volume needed is the sum of all waste produced, prior to the year in which backfilling can commence (with applied swell factor). An external waste dump was designed on the east edge of the final pit which contained the required waste volume.

Subsequent waste dump phases were designed in which all waste was used to backfill the pit. These dump phases were placed in locations which became available for backfilling as the overall mining sequence progressed. Each dump phase contained from two to seven years' worth of waste production. Dumps were placed as close to the currently mined areas as possible, to minimize waste haul distances.

16.1.1.11 Haul Profiles

For each production year, ore and waste are generally mined from one or two mining phases. Waste is hauled to, at most, two dump phases in a single year. For each year, average haul profiles were generated for the ore haul out of each mining phase to the mill, and for the waste haul from each mining phase to each waste dump. A weighted (by tonnes ore/waste) haul time was then calculated to get the average haul time for waste and ore in each year of operation.

Ore tonnes, grade, and waste tonnes, along with average haul times were consolidated and transferred to an Excel spreadsheet. A detailed cost model was developed, based on these production figures.

16.1.1.12 Case 1 Mine Design

A mine plan was developed to deliver lithium ore to the mill, operating at a rate of 2,100 dry tpd, based on 365 days of operation per year at 90% availability. This is equivalent to 689,850 dry tpy of ore. Ore is mined with a Wirtgen surface miner and loaded directly into haul trucks. The ore is then hauled and dumped on a series of drying pads adjacent to the mill. Waste is mined with a hydraulic excavator, loaded into haul trucks, then hauled and dumped. Waste is initially dumped outside the pit to the east and northeast in early years, and then backfilled into the pit in later years as soon as it is feasible.

As noted earlier, the Whittle Shell 8 was used to design an ultimate pit without ramps. A comparison of material contained in Shell 8, (excluding that in the isolated southwest area) and the final design without ramps is presented in Table 16.4. The designed pit contains more ore at a slightly lower grade, and a slightly higher strip ratio. A plan view map of Whittle Shell 8 is presented in Figure 16.1 and a plan view map of the Case 1 Final Design is presented in Figure 16.2. Cross sectional views through the Case 1 pit are provided in Figure 16.3 and Figure 16.4.

Table 16.4: Comparison of Whittle Shell 8 to Case 1 Final Design

	Whittle Shell 8	Case 1 Final Design
Measured + Indicated Ore Tonnes (000's)	13,333	13,949
Waste Tonnes (000's)	36,594	40,122
Average % Li	0.408	0.404
Average % K	3.84	3.82
Average % Na	1.46	1.46
Strip Ratio	2.74	2.88

Cutoff = 0.320 % Li
95% Mine Recovery Factor Applied

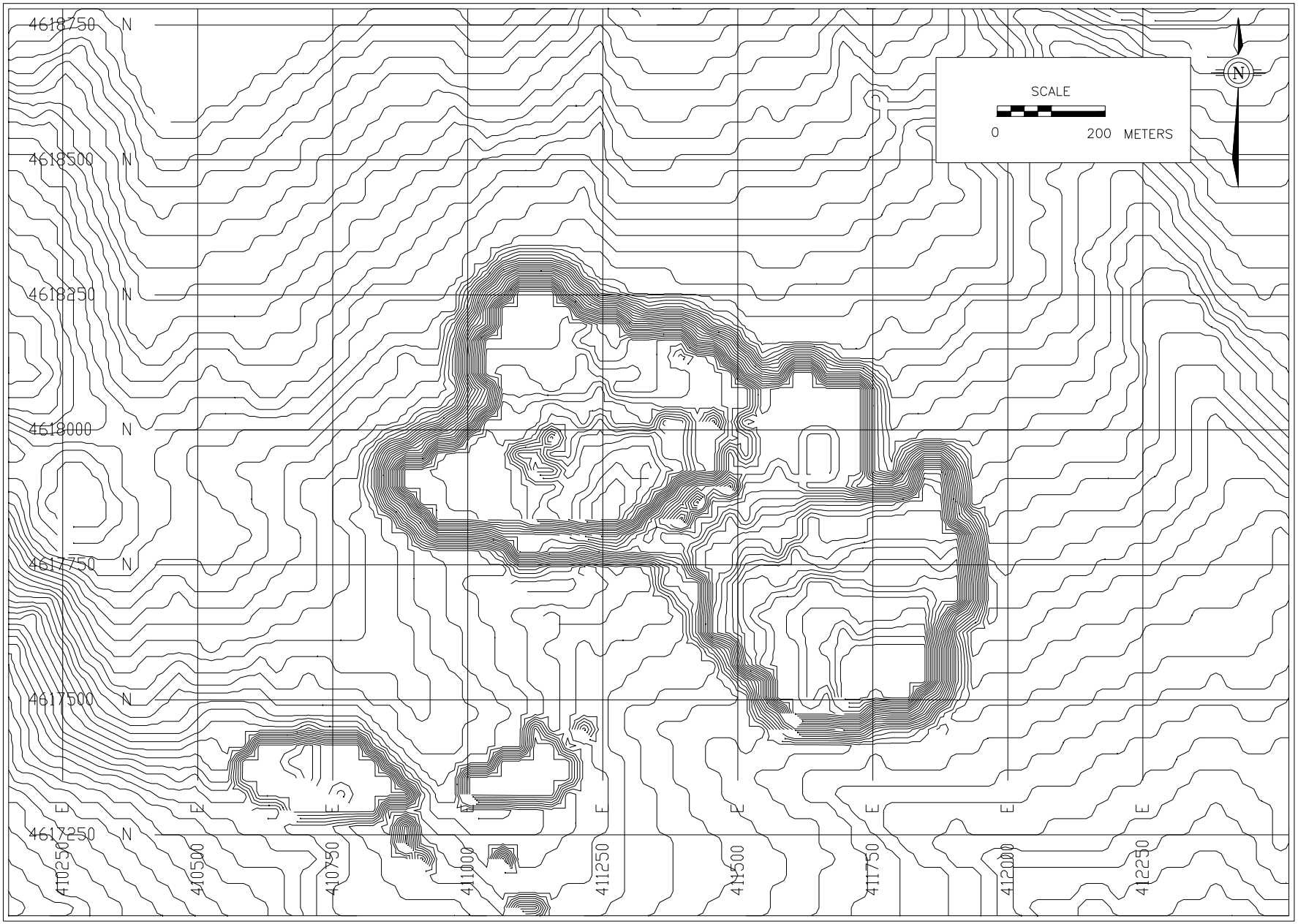
Within the limits of the ultimate pit, a series of five mining phases were designed including ramp access. A plan view drawing showing the order in which the phases are mined is presented in Figure 16.5.

. The first mining phase was situated roughly in the location of Whittle Shell 6, which identifies a higher profitability area containing approximately three years of production. Other phases were developed in a southeast to northwest pattern. In general, a minimum pushback width of 50 m was used between phases. The Case 1 phase reserves are shown in the following Table 16.5.

Table 16.5: Case 1 Mine Phases, Summary of Dry Tonnes

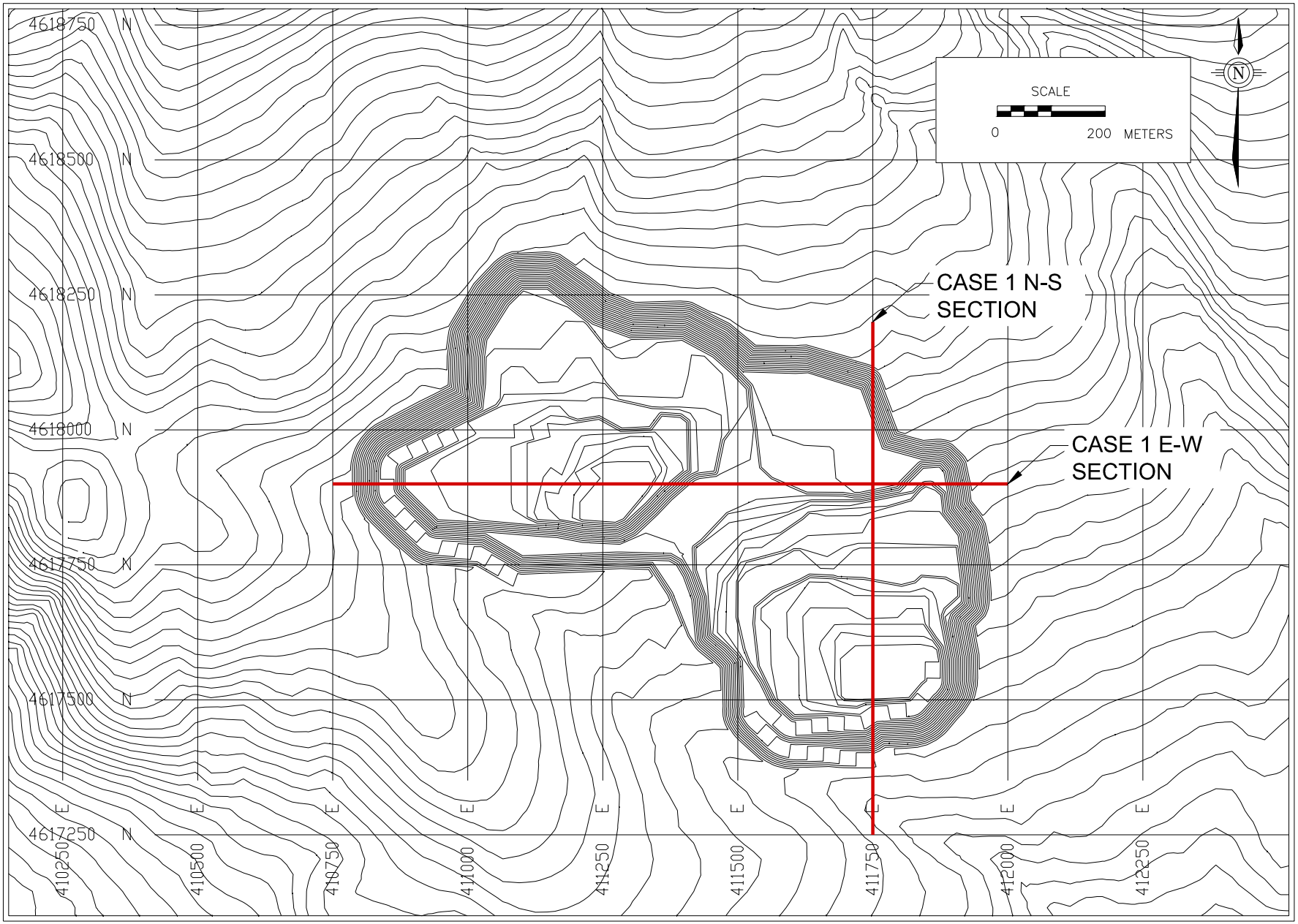
	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Total
Proven Ore Tonnes (000's)	2,286	2,052	2,043	2,521	3,251	12,153
Proven Average % Li	0.402	0.403	0.410	0.394	0.414	0.405
Proven Average % K	3.70	3.83	3.82	3.69	4.03	3.83
Proven Average % Na	1.55	1.35	1.49	1.43	1.47	1.46
Probable Ore Tonnes (000's)	60	455	321	138	822	1,796
Probable Average % Li	0.391	0.402	0.378	0.389	0.401	0.396
Probable Average % K	3.54	3.93	3.12	3.64	3.97	3.77
Probable Average % Na	1.65	1.23	1.66	1.27	1.50	1.45
Proven+Probable Ore Tonnes (000's)	2,346	2,507	2,364	2,659	4,073	13,949
Proven+Probable Average % Li	0.402	0.403	0.406	0.394	0.411	0.404
Proven+Probable Average % K	3.70	3.85	3.72	3.69	4.02	3.82
Proven+Probable Average % Na	1.55	1.33	1.51	1.42	1.48	1.46
Waste Tonnes (000's)	5,670	7,769	5,437	10,091	11,155	40,122
Strip Ratio	2.37	3.10	2.30	3.80	2.74	2.85

Summary of Dry Ore Tonnes for Proven + Probable
Cutoff = 0.320 %Li
95% Mine Recovery Factor Applied



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Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: ***Western Lithium Corp.***	Location: Humboldt County, NV	FIGURE 16.1 WHITTLE SHELL 8
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	



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Prepared for: ***Western Lithium Corp.***

Location: Humboldt County, NV

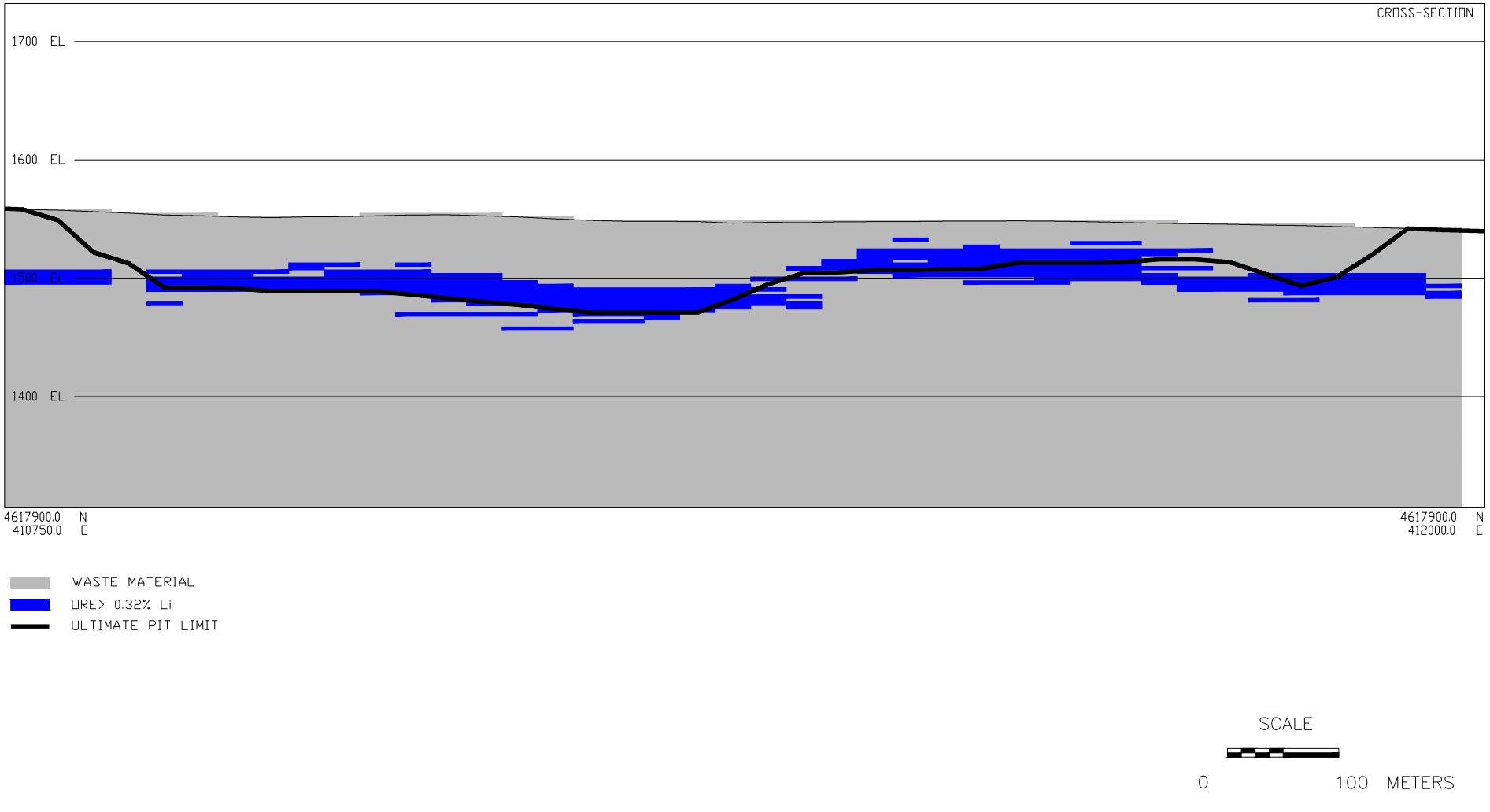
Project: Kings Valley Lithium Project

Project no.: 114-311106

Date of Issue: 12/2011

FIGURE 16.2

CASE 1 FINAL ULTIMATE DESIGN



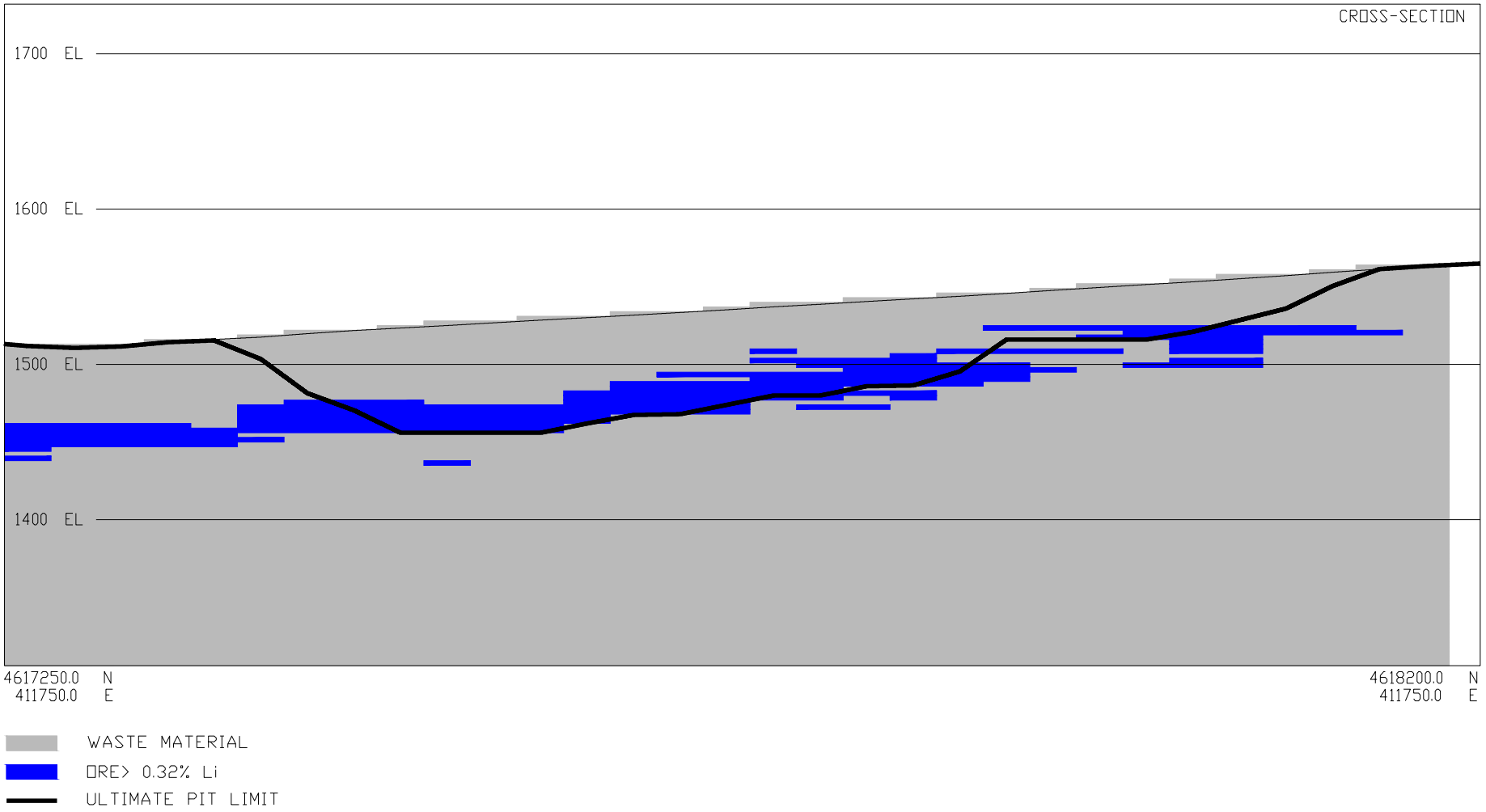
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Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: ***Western Lithium Corp.***	Location: Humboldt County, NV	FIGURE 16.3 CASE 1 E-W SECTION
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	

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Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: ***Western Lithium Corp.***	Location: Humboldt County, NV	FIGURE 16.4 CASE 1 N-S SECTION
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	

Case 1 Waste Dump Design

A series of four waste dump phases were designed for the Case 1 scenario. Dumps were designed assuming an overall swell factor of 30%. Phase 1 is external to the pit, and is needed for handling waste from pre-production through Year 7. The remaining three phases are backfill phases and are needed to handle waste from production for Years 8 and beyond. The capacity of each Case 1 dump phase is presented in Table 16.6.

Table 16.6: Case 1 Waste Dump Volume Summary

Dump Phase	Volume (000s m^3)	Minimum Bench Elev (m)	Maximum Bench Elev (m)	In Use During Years
1	13,882	1,504	1,582	-1 to 7
2	1,553	1,513	1,582	8 to 9
3	5,632	1,480	1,597	9 to 12
4	8,786	1,456	1,519	13 to 20
Total	29,853	1,456	1,597	

Case 1 Waste Mining Schedule

Prestripping of 2.5 million dry tonnes of waste would occur in the year prior to the start of production. Waste movement in production Years 1 through 7 holds generally steady at around 2.2 million dry tpy. Waste production expands to 2.5 million dry tpy in Years 8 through 15, and then quickly declines in production Year 16 and beyond.

Case 1 In-Pit Inferred Resources

For Case 1, inferred resources were considered as waste and not used in the economic analysis. Note that CIM standards define inferred resources as:

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 for locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.

The Case 1 pit designs contain a total of 44,000 tonnes of inferred material above a 0.320% Li cutoff. The average grade of the inferred material is 0.330% Li, 3.42% K, and 1.34% Na.

16.1.1.13 Case 2 Mine Design

A second mine plan, Case 2, was developed to deliver lithium ore to the mill, operating at a rate of 2,100 dry tpd, based on 365 days of operation per year at 90% availability for the first three years of operation. Then, in Year 4, the production rate is doubled. This is equivalent to 689,850 dry tpy of ore

for Years 1 through 3, and 1,379,700 dry tpy of ore for Years 4 and beyond. Ore is mined with a Wirtgen surface miner and loaded directly into haul trucks. The ore is then hauled and dumped on a series of drying pads adjacent to the mill. Waste is mined by hydraulic excavator, loaded into haul trucks, then hauled and dumped. Waste is initially dumped outside the pit to the east and northeast in early years, and then backfilled into the pit in later years as soon as it is feasible.

One change in design criteria for Case 2 vs. Case 1 was that WLC requested a variable cutoff strategy be implemented that would maintain a mill feed grade close to or exceeding 0.40% Li in the first half of the project. In order to meet this requirement, the ore cutoff was set according to the schedule presented in Table 16.7.

Table 16.7: Case 2 Mining Cutoff Summary

Year(s)	%Li Cutoff
1	0.33
2	0.345
3-4	0.325
5	.335
6.20	.325
Average	0.327

As noted earlier, the Whittle Shell 10 was used as the template for designing an ultimate pit without ramps. A comparison of material contained in Shell 10, excluding the shapes that are in the separate drainage, vs. the final design without ramps is presented in Table 16.8. The designed pit contains less ore at a slightly lower grade, and a higher strip ratio. A plan view map of Whittle Shell 10 is presented in Figure 16.6 and a plan view map of the Case 1 Final Design is presented in Figure 16.7. Cross sectional views through the Case 1 pit are provided in Figure 16.8 and Figure 16.9.

Table 16.8: Comparison of Whittle Shell 10 to Case 2 Final Design

	Whittle Shell 10	Case 2 Final Design
Measured + Indicated Ore Tonnes (000's)	27,231	27,153
Waste Tonnes (000's)	89,430	95,598
Average % Li	0.397	0.395
Average % K	3.89	3.88
Average % Na	1.36	1.36
Strip Ratio	3.28	3.52

Cutoff = 0.327% Li
95% Mine Recovery Factor Applied

Within the limits of the ultimate pit, a series of eight mining phases were designed including ramp access. A plan view drawing showing the order in which the phases are mined is presented in Figure 16.10. The phases were developed in a southeast to northwest pattern. In general, a minimum pushback width of 50 m was used between phases. Case 2 phase reserves are shown in the following Table 16.9.

Table 16.9: Case 2 Mine Phases, Summary of Dry Tonnes

	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6	Phase 7	Phase 8	Total
Proven Ore Tonnes (000s)	1,006	1,208	5,427	3,834	3,076	386	-	-	14,937
Proven Average %Li	0.400	0.398	0.400	0.393	0.408	0.430	-	-	0.400
Proven Average %K	2.87	3.86	3.86	3.87	4.07	4.14	-	-	3.85
Proven Average %Na	0.77	1.44	1.43	1.34	1.45	1.40	-	-	1.37
Probable Ore Tonnes (000s)	330	409	1,001	1,166	202	3,769	2,213	3,108	12,198
Probable Average %Li	0.372	0.377	0.381	0.388	0.377	0.381	0.392	0.398	0.388
Probable Average %K	2.50	3.46	3.21	3.86	3.75	4.04	4.32	4.00	3.93
Probable Average %Na	1.23	1.39	1.21	1.20	1.48	1.45	1.38	1.33	1.36
Proven+Probable Ore Tonnes (000s)	1,336	1,617	6,428	5,000	3,278	4,155	2,213	3,108	27,135
Proven+Probable Average %Li	0.393	0.393	0.397	0.392	0.406	0.386	0.392	0.398	0.395
Proven+Probable Average %K	2.78	3.76	3.76	3.87	4.05	4.05	4.32	4.00	3.88
Proven+Probable Average %Na	0.88	1.43	1.40	1.31	1.45	1.45	1.38	1.33	1.36
Waste Tonnes (000s)	9,552	5,143	10,978	14,267	8,158	19,708	9,126	18,667	95,599
Strip Ratio	7.05	3.18	1.71	2.85	2.49	4.74	4.12	6.01	3.52

Note: Ore tonnes and grade will not match exactly with detailed schedule, because variable cutoffs were used during first few years of operation.
Cutoff = 0.327 % Li
95% Mine Recovery Factor Applied

Case 2 Mining Schedule

A mining schedule was developed based on mining the sequence of eight phases. Initial top-down schedules were used to determine peak waste mining years, and waste was forward scheduled in order to level out yearly fluctuations. The goal of the schedule was to develop a sequence of material movement, which could be used as a guide in developing detailed annual dig plans. By keeping the total material movement constant for long periods of time, fluctuations in fleet requirements are reduced assuming relatively constant haul distances.

Detailed dig plans were developed to cover one year of pre-production, and each of the 20 years of production. Each dig plan was designed to come within a few thousand tonnes of the targeted production for ore and waste for that period. A summary of mined ore and waste material was generated for each period, based on the dig plans.

A series of six waste dump phases were designed for the Case 2 scenario. The capacity of each dump phase is presented in Table 16.10.

Table 16.10: Case 2 Waste Dump Volume Summary

Dump Phase	Volume (000s m^3)	Minimum Bench Elev (m)	Maximum Bench Elev (m)	In Use During Years
1	18460	1498	1618	-1 to 5
2	3224	1468	1552	5 to 6
3	6488	1456	1588	6 to 8
4	19469	1456	1546	8 to 13
5	18812	1480	1579	13 to 17
6	5821	1468	1513	17 to 20
Total	72274	1456	1618	

Case 2 Waste Mining Schedule

For this model, prestripping of six million dry tonnes of waste is assumed to occur in the 18 months prior to the start of production. However, an option remains for WLC to defer the prestripping to a later year depending on market conditions. Waste movement in Years 1 through 8 would hold generally steady at around 3.9 to 4.0 million dry tpy. Waste production would expand to about 5.2 million dry tpy in Years 8 through 14, ramp up further to around seven million dry tpy for Years 15 through 18, and then quickly decline in Years 19 and 20

In the Case 2 scenario that Tetra Tech evaluated the additional stripping required was extended from pre-production through the first 3 years of production in order to levelize the equipment and manpower requirements associated with the additional waste removal. WLC could potentially pushout this additional stripping by starting the project using the Case 1 schedule and making the decision to increase production later. This option accounts for the additional waste stripping to later in the project life. Tetra Tech did not evaluate this scenario.

Case 2 In-Pit Inferred Resources

For Case 2, inferred resources were considered as waste and not used in the economic analysis.

The Case 2 pit designs contain a total of 257,000 tonnes of inferred material above a 0.320% Li cutoff. The average grade of the inferred material is 0.374% Li, 3.81% K and 1.55% Na.

16.1.2 Equipment Selection

Both Case 1 and Case 2 have been planned as an open-pit mine using end-dump haul trucks, hydraulic excavators for waste removal, and a Wirtgen surface miner for selective ore mining. The Wirtgen surface miner was selected as the primary ore stripping machine to aid in selective mining of the ore. The Wirtgen can achieve a cutting depth of 0 to 28 cm, allowing for greater selectivity and grade control. The Wirtgen machine also makes a primary crushing circuit unnecessary, as cut material is sized to less than 50 mm.

The Wirtgen machine would also allow for selective removal of ash bands present within the mineralized claystone. This level of detail was not included in the Stage I PFS as the ash bands had yet to be identified within the geological model.

16.1.2.1 Operating Schedule

The mining operations have been scheduled on 350 operating days per year with two 12-hour shifts per day. Equipment operations and manpower would use a rotating four crew shift schedule. Equipment mechanical availability has been set to 90% with an effective utilization of 84%. Mechanical availability takes into account planned and unplanned mechanical down time due to maintenance and equipment failure. Utilization takes into account down time associated with shift changes, employee breaks and weather delays. Using these factors results in 8,400 hours of available working time per year and 6,350 operating hours per year.

16.1.2.2 Case 1 Equipment Selection

Primary waste removal is achieved using a single Hitachi EX1200-6 excavator with a 6.3 m^3 bucket matched with Caterpillar 770 haul trucks with a rated payload of 36 tonnes. Primary ore removal is achieved with a Wirtgen 2200SM surface miner. This machine would direct load Caterpillar 770 haul trucks using an available elevated conveyor discharge system mounted to the front of the machine.

Excavator and Wirtgen productivities were calculated based upon the scheduled time, availability, utilizations, truck, and material properties. Productivities are reflective of minimal down time due to the loading operations waiting on trucks. The Wirtgen surface miner productivity exceeds the production rate required to feed the process plant. Tetra Tech determined that ore and waste operations would result in significant downtime thus increasing operational costs utilizing a dedicated ore and waste operation. Therefore, waste operations would occur for approximately five days per week with the remaining two days used for ore operations. This allows for the most effective use of operating labor and decreased demand for trucks. The trucks would be shared between the ore and waste operations. The excavator and Wirtgen would be operated using the same operator thus saving labor costs. The Wirtgen hourly productivity was quoted to Tetra Tech from Wirtgen America as 768 short tons per scheduled hour. This estimate equates to 627 tonnes per hour with a 10% decline in productivity due to direct loading of haul trucks. Table 16.11, Case 1 Excavator Productivity Estimate, shows the excavator productivity calculation.

Truck productivity is based on truck-cycle times from the loading point to either the ROM ore stockpile or the appropriate waste dump location. Truck cycle times were developed for each year based upon a weighted average truck cycle time from the various mining phases utilized in each year to either the ROM ore stockpile for ore truck cycle times or to the waste dump phase for waste truck cycle times. The truck cycle times were then used to determine the number of truck hours required for ore and waste removal and subsequently the number of trucks required to meet the mine plan schedule.

The following major mining equipment is utilized in the mine plan:

- 1 Hitachi EX1200-6 Excavator
- 1 Wirtgen 2200SM Surface Miner
- 3 to 5 Caterpillar 770 Haul Trucks
- 1 Caterpillar D9 Dozer
- 1 Caterpillar 980 Front End Loader
- 1 Caterpillar 16M Motor Grader
- 1 5,000 Gallon Water Truck

Table 16.11: Case 1 Excavator Productivity Estimate

Description	Units	Hitachi EX1200-6
In-Situ Density	dry tonnes/ m^3	1.76
Moisture Content	%	18.0
Swell Factor	%	25
Bucket Capacity	m^3	6.3
Bucket Fill Factor	%	100
Nominal Bucket Capacity	m^3	6.3
Tonnes per Bucket	loose wet tonnes	10.8
Nominal Truck Size	wet tonnes	36
Truck Fill Factor	%	100
Expected Truck Capacity	wet tonnes	36.0
Number of Passes/Truck	passes	4
Loader Cycle Time	seconds/pass	30
Spot Time/Load	seconds	45
Total Loading Time	minutes	2.8
Scheduled Days/Year	days	350
Scheduled Hours/Day	hours/day	24
Scheduled Hours/Year	hours/year	8,400
Shift Length, Hours	hours	12
No. of Annual Shifts	shifts/year	700
Total Time/Shift	minutes	720
Max Tonnage/Shovel Scheduled Shift	wet tonnes/shift	9,425
Max Tonnage/Shovel Scheduled Year	wet tonnes/year	6,597,818
Tonnage/Scheduled Hour	wet tonnes/scheduled hour	785

Case 1 Support and Utility Equipment

Support equipment was selected to perform operations supporting ore and waste removal. This includes a Caterpillar D9 dozer working on the waste dump, spreading waste material as it is dumped by the haul trucks. The dozer would also perform miscellaneous road work, pit cleanup and other support functions as required. A Caterpillar 980 class rubber tired front end loader would be used to support the hydraulic excavator and to perform secondary loading of the haul trucks when required. The hydraulic excavator would not be mobile enough or have sufficient reach to clean up the active pit area. A Caterpillar 16M motor grader and 5,000-gallon water truck would perform haul road maintenance and provide dust suppression.

A fleet of utility vehicles would be on site for maintenance and utility work around the mine site. This includes a small backhoe loader for general site maintenance; fuel and lube trucks for in-pit refueling and lubrication of equipment; a service truck for in-pit equipment service; light plants for lighting of the in-pit and waste dump operations; and pickup trucks for supervisory personnel, surveying and crew transportation.

16.1.2.3 Case 2 Equipment Selection

For the first seven years of the mine plan, primary waste removal is achieved using a single Hitachi EX1200-6 excavator with a 6.3 m^3 bucket matched with Caterpillar 770 haul trucks with a rated payload of 36 tonnes. After year seven the Hitachi EX1200-6 would be replaced with a Hitachi EX1900-6 with an 11.0 m^3 bucket. A fleet of larger Caterpillar 773 haul trucks with a rated payload of 54 tonnes would be purchased which match the larger excavator. The existing Caterpillar 770 haul trucks would be moved to ore operations resulting in an excess of trucks available for ore operations. Primary ore removal is achieved with a Wirtgen 2200SM surface miner. This machine would direct load Caterpillar 770 haul trucks using an available elevated conveyor discharge system mounted to the front of the machine.

Excavator and Wirtgen productivities were calculated based upon the scheduled time, availability, utilizations, truck, and material properties. Productivities are reflective of minimal down time due to the loading operations waiting on trucks. The Wirtgen surface miner productivity exceeds the production rate required to feed the process plant. As a result, ore operations would be performed during day shift only utilizing a rotating two crew shift schedule. The Wirtgen hourly productivity was quoted to Tetra Tech from Wirtgen America as 768 short tons per scheduled hour. This estimate equates to 627 tonnes per hour with a 10% decline in productivity due to direct loading of haul trucks. Table 16.12, Case 2 Excavator Productivity Estimate, shows the excavator productivity calculation.

Truck productivity is based on truck-cycle times from the loading point to either the ROM ore stockpile or the appropriate waste dump location. Truck cycle times were developed for each year based upon a weighted average truck cycle time from the various mining phases utilized in each year to either the ROM ore stockpile for ore truck cycle times or to the waste dump phase for waste truck cycle times. The truck cycle times were then used to determine the number of truck hours required for ore and waste removal and subsequently the number of trucks required to meet the mine plan schedule.

The following major mining equipment is utilized in the mine plan:

- 1 Hitachi EX1200-6 Excavator through Year 7
- 1 Hitachi EX1900-6 Excavator for Year 8 through LOM
- 1 Wirtgen 2200SM Surface Miner
- 9 Caterpillar 770 Haul Trucks
  - 3 for Ore operations through Year 7
  - 6 for waste operations through Year 7
  - 9 for ore operations from Year 8 through LOM
- 4 to 7 Caterpillar 773 Haul Trucks
- 1 Caterpillar D9 Dozer
- 1 Caterpillar 980 Front End Loader
- 1 Caterpillar 16M Motor Grader
- 1 5,000 Gallon Water Truck

Table 16.12: Case 2 Excavator Productivity Estimate

Description	Units	Hitachi EX1200-6	Hitachi EX1900-6
In-Situ Density	dry tonnes/ m^3	1.76	1.76
Moisture Content	%	18.0	18.0
Swell Factor	%	25	25
Bucket Capacity	m^3	6.3	11.0
Bucket Fill Factor	%	100	100
Nominal Bucket Capacity	m^3	6.3	11.0
Tonnes per Bucket	loose wet tonnes	10.8	18.9
Nominal Truck Size	wet tonnes	36	54
Truck Fill Factor - Nominal Truck Size	%wet tonnes	10036	10054
Expected Truck Capacity Truck Fill Factor	wet tonnes%	36.0100	54.0100
Number of Passes/Truck Expected Truck Capacity	passes wet tonnes	436.0	354.0
Number of Passes/Truck	passes	4	3
Loader Cycle Time	seconds/pass	30	30
Spot Time/Load Loader Cycle Time	seconds seconds/pass	4530	4530
Total Loading Time Spot Time/Load	minutes seconds	2.845	2.345
Total Loading Time	minutes	2.8	2.3
Scheduled Days/Year	days	350	350
Scheduled Hours/Day Scheduled Days/Year	hours/day days	24350	24350
Scheduled Hours/Year Scheduled Hours/Day	hours/year hours/day	8,40024	8,40024
Shift Length, Hours Scheduled Hours/Year	hours hours/year	128,400	128,400
No. of Annual Shifts Shift Length, Hours	shifts/year hours	70012	70012
Total Time/Shift No. of Annual Shifts	minutes shifts/year	720700	720700
Total Time/Shift	minutes	720	720
Max Tonnage/Shovel Scheduled Shift	wet tonnes/shift	9,425	17,280
Max Tonnage/Shovel Scheduled Year ax Tonnage/Shovel Scheduled Shift	wet tonnes/year wet tonnes/shift	6,597,8189,425	12,096,00017,280
Tonnage/Scheduled Hour ax Tonnage/Shovel Scheduled Year	wet tonnes/scheduled hour wet tonnes/year	7856,597,818	1,44012,096,000
Tonnage/Scheduled Hour	wet tonnes/scheduled hour	785	1,440

Case 2 Support and Utility Equipment

Support equipment was selected to perform operations supporting ore and waste removal. This includes a Caterpillar D10 dozer working on the waste dump spreading waste material as it is dumped by the haul trucks. The dozer would also perform miscellaneous road work, pit cleanup and other support function as required. A Caterpillar 988 class rubber tired front end loader would be used to support the hydraulic excavator and perform secondary loading of the haul trucks when required.

The hydraulic excavator would not be mobile enough or have sufficient reach to clean up the active pit area. A Caterpillar 16M motor grader and 5,000-gallon water truck would perform haul road maintenance and provide dust suppression.

A fleet of utility vehicles would be on site for maintenance and utility work around the mine site. This includes a small backhoe loader for general site maintenance; fuel and lube trucks for in-pit refueling and lubrication of equipment; a service truck for in-pit equipment service; light plants for lighting of the in-pit and waste dump operations; and pickup trucks for supervisory personnel, surveying and crew transportation.

16.1.3 Production Schedule

A detailed yearly mine schedule for Case 1 appears below in Table 16.13.

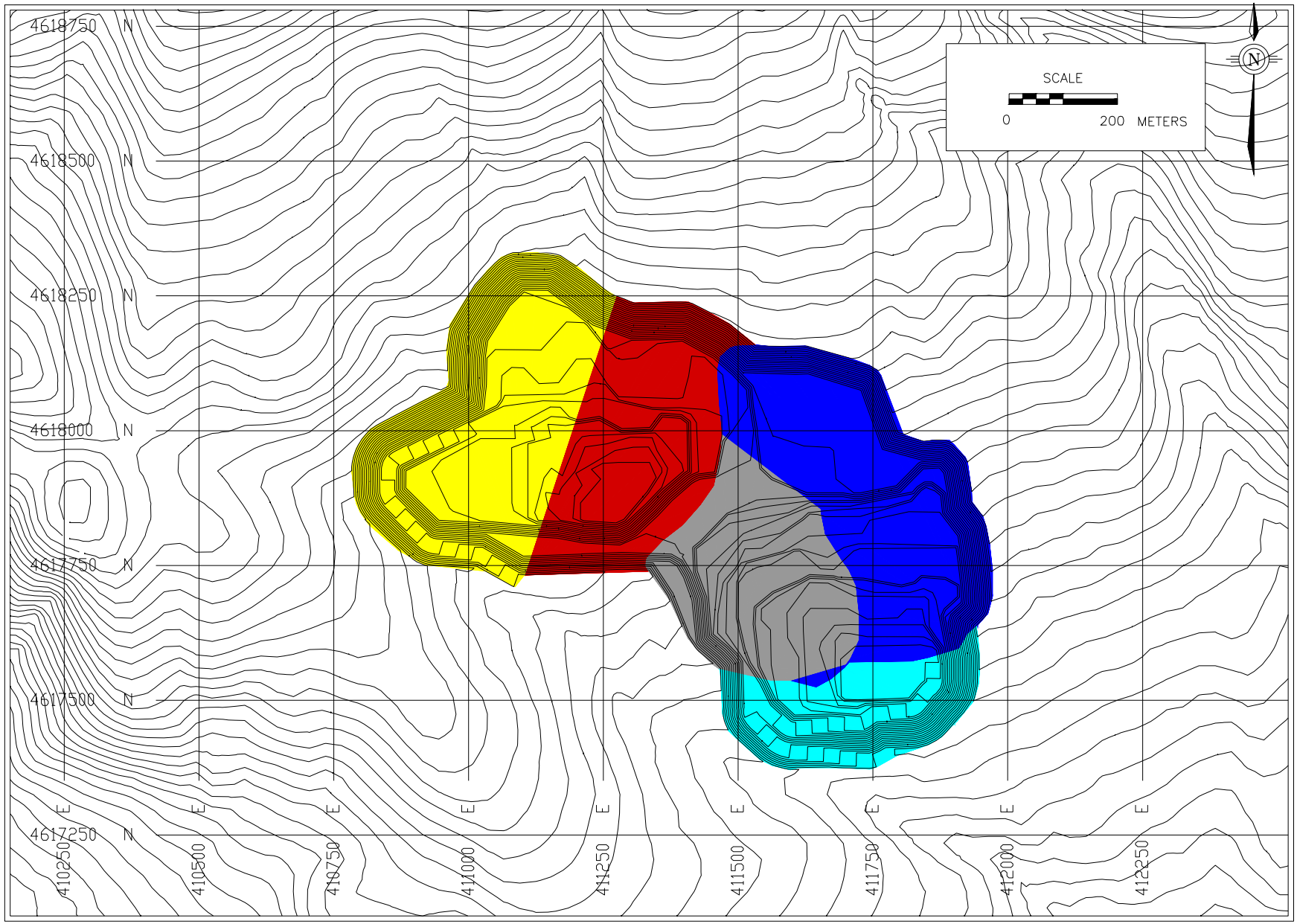
Table 16.13: Case 1 Detailed Mine Schedule

Year	Ore Mined Dry Tonnes	Waste Mined Dry Tonnes	Total Mined Dry Tonnes	Strip Ratio	Li%	K%	Na%
-1	-	2,500,013	2,500,013	-	-	-	-
1	689,850	2,217,582	2,907,432	3.21	0.395	3.52	1.64
2	689,850	2,214,701	2,904,551	3.21	0.390	3.68	1.48
3	689,850	2,214,368	2,904,218	3.21	0.397	3.56	1.57
4	689,850	2,215,362	2,905,212	3.21	0.414	3.83	1.38
5	689,850	2,215,930	2,905,780	3.21	0.390	3.94	1.32
6	689,850	2,218,977	2,908,827	3.22	0.413	3.99	1.37
7	689,850	2,291,599	2,981,449	3.32	0.415	3.92	1.33
8	689,850	2,506,152	3,196,002	3.63	0.374	3.05	1.55
9	689,850	2,510,535	3,200,385	3.64	0.409	3.72	1.50
10	689,850	2,505,897	3,195,747	3.63	0.405	3.88	1.54
11	689,850	2,507,454	3,197,304	3.63	0.404	3.74	1.37
12	689,850	2,509,308	3,199,158	3.64	0.379	3.52	1.39
13	689,850	2,508,871	3,198,721	3.64	0.402	3.93	1.38
14	689,850	2,505,910	3,195,760	3.63	0.408	3.90	1.45
15	689,850	2,506,152	3,196,002	3.63	0.363	3.15	1.57
16	689,850	1,297,660	1,987,510	1.88	0.405	3.82	1.52
17	689,850	438,304	1,128,154	0.64	0.391	3.86	1.61
18	689,850	113,575	803,425	0.16	0.423	4.24	1.50
19	689,850	65,245	755,095	0.09	0.463	4.52	1.39
20	689,850	50,746	740,596	0.07	0.432	4.48	1.33
21	151,530	7,974	159,504	0.05	0.430	4.35	1.37
TOTAL	13,948,530	40,122,315	54,070,845	2.88	0.404	3.82	1.46

A detailed yearly mine schedule for Case 2 appears in Table 16.14.

Table 16.14: Case 2 Detailed Mine Schedule

Year	Ore Mined Dry Tonnes	Waste Mined Dry Tonnes	Total Mined Dry Tonnes	Strip Ratio	Li %	K %	Na %
-2	-	2,000,667	2,000,667	99.99	-	-	-
-1	-	4,001,333	4,001,333	99.99	-	-	-
1	689,850	3,899,000	4,588,850	5.65	0.401	3.17	0.94
2	689,850	3,894,000	4,583,850	5.64	0.403	2.74	0.93
3	689,850	3,974,000	4,663,850	5.76	0.399	3.76	1.47
4	1,379,700	3,967,000	5,346,700	2.88	0.401	3.89	1.41
5	1,379,700	3,971,000	5,350,700	2.88	0.399	3.80	1.49
6	1,379,700	3,968,000	5,347,700	2.88	0.401	3.85	1.36
7	1,379,700	3,967,000	5,346,700	2.88	0.401	3.67	1.31
8	1,379,700	3,972,000	5,351,700	2.88	0.401	3.82	1.33
9	1,379,700	5,178,000	6,557,700	3.75	0.390	3.67	1.26
10	1,379,700	5,167,000	6,546,700	3.75	0.391	3.85	1.33
11	1,379,700	5,166,000	6,545,700	3.74	0.403	4.27	1.31
12	1,379,700	5,170,000	6,549,700	3.75	0.410	4.02	1.48
13	1,379,700	5,171,000	6,550,700	3.75	0.422	4.30	1.45
14	1,379,700	5,183,000	6,562,700	3.76	0.379	3.78	1.44
15	1,379,700	6,966,000	8,345,700	5.05	0.395	4.04	1.46
16	1,379,700	6,961,000	8,340,700	5.05	0.389	4.23	1.39
17	1,379,700	6,967,000	8,346,700	5.05	0.406	4.27	1.38
18	1,379,700	6,756,000	8,135,700	4.90	0.389	4.43	1.34
19	1,379,700	658,000	2,037,700	0.48	0.395	3.76	1.36
20	1,350,000	284,000	1,634,000	0.21	0.415	4.51	1.26
TOTAL	25,494,750	97,241,000	122,735,750	3.81	0.399	3.95	1.35



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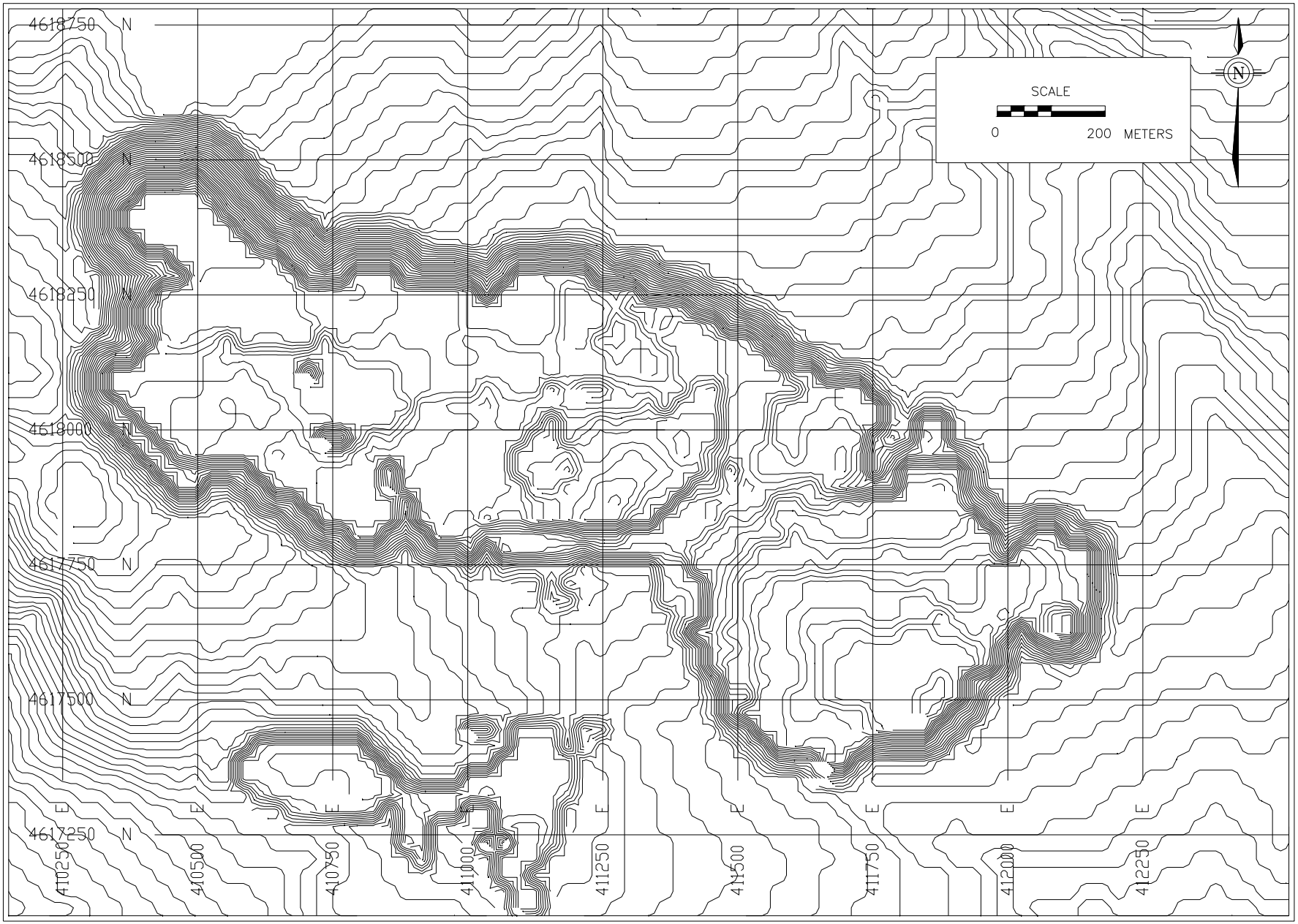
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FIGURE 16.5

CASE 1 MINING PHASES



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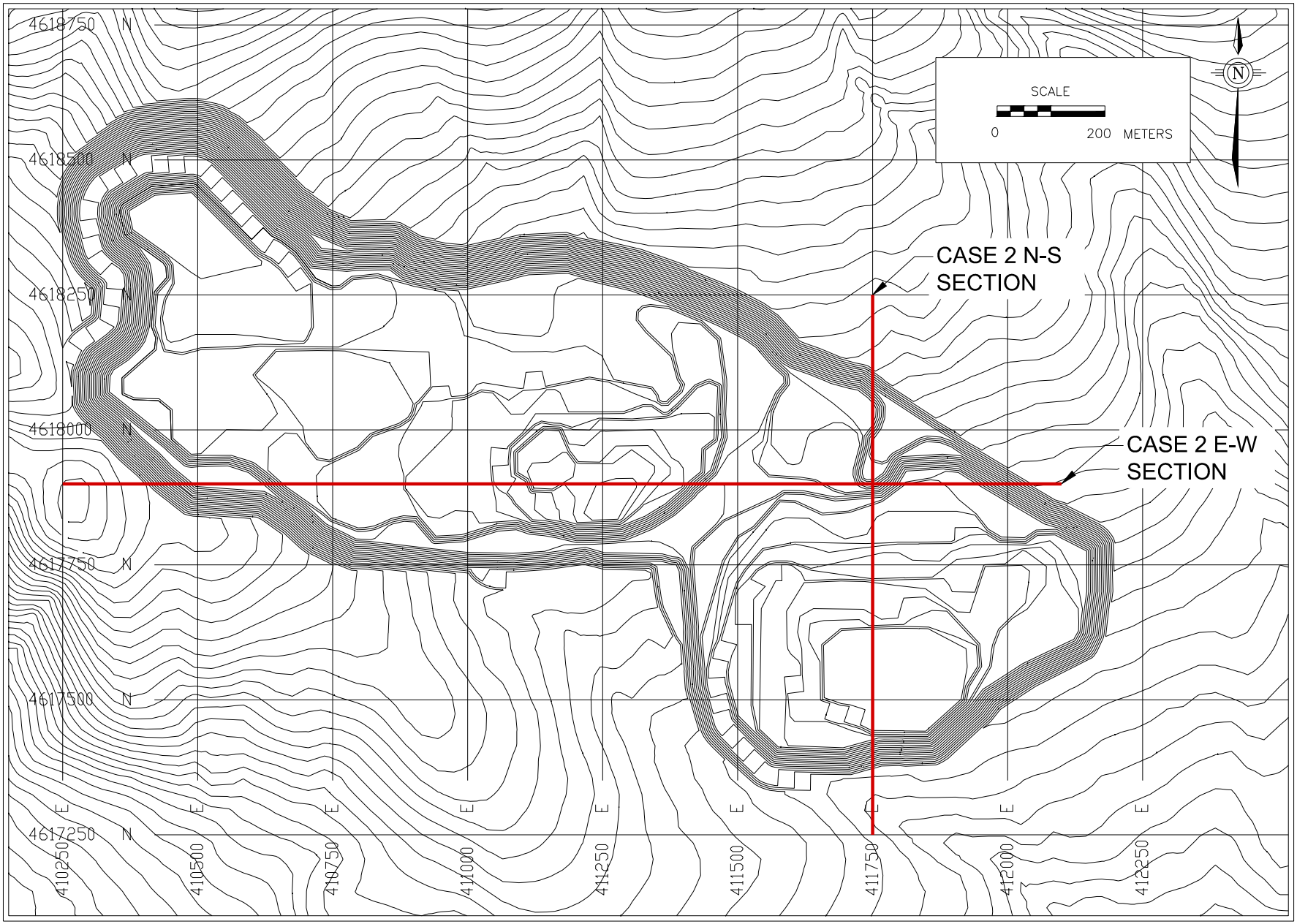
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FIGURE 16.6
WHITTLE SHELL 10



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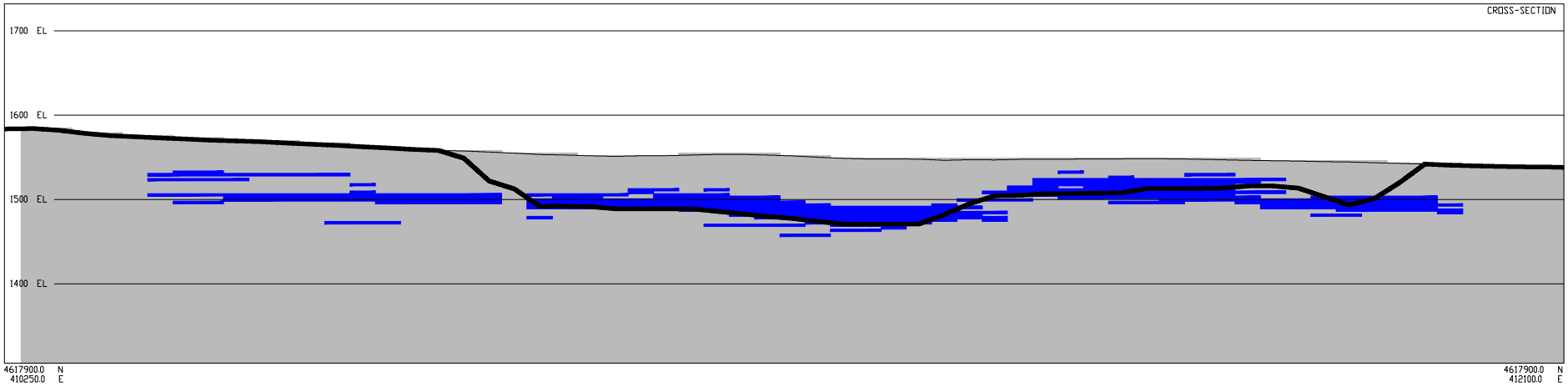
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FIGURE 16.7

CASE2 FINAL ULTIMATE DESIGN

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WASTE MATERIAL

ORE> 0.32% Li

ULTIMATE PIT LIMIT

SCALE

0 100 METERS

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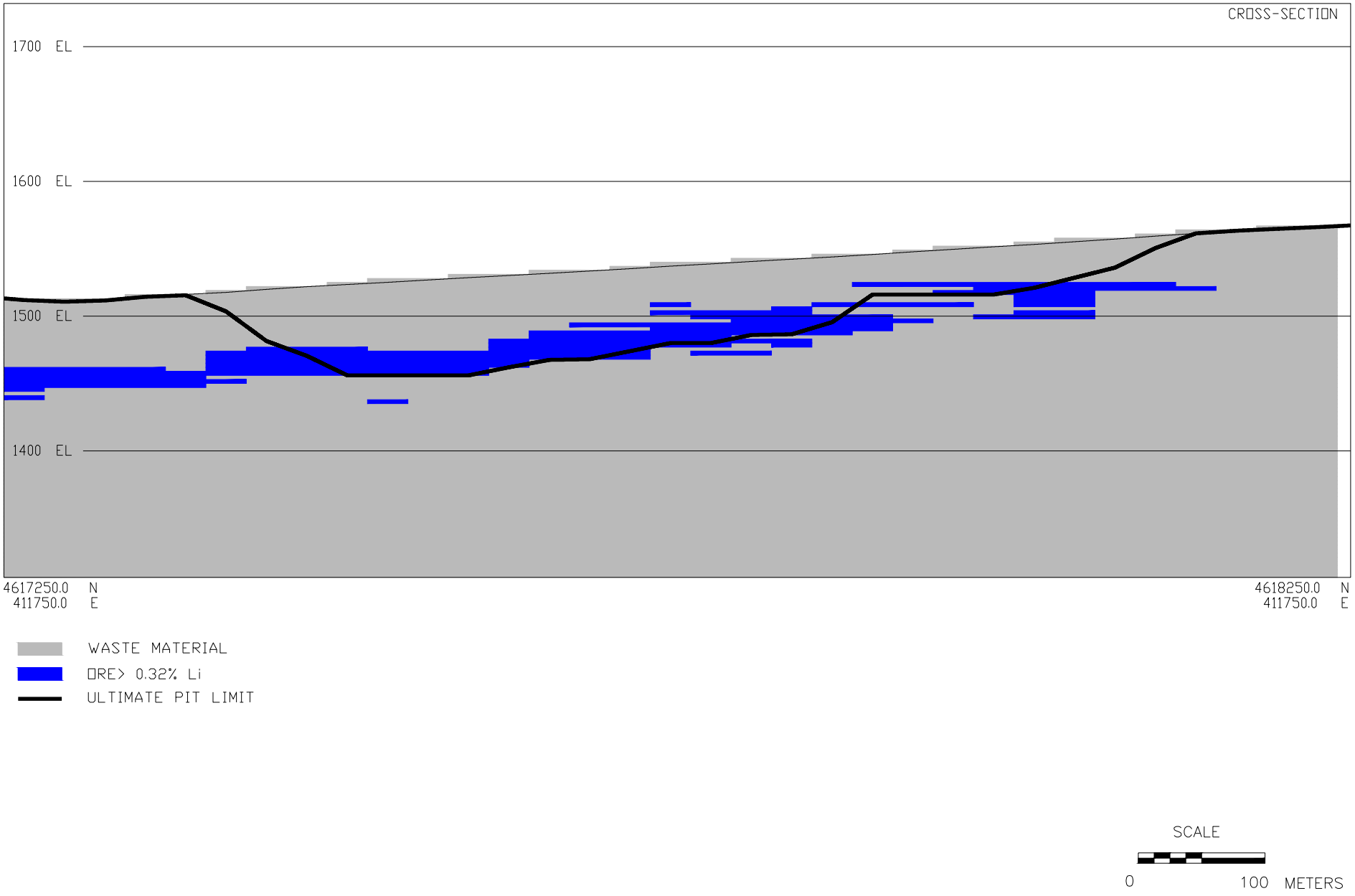
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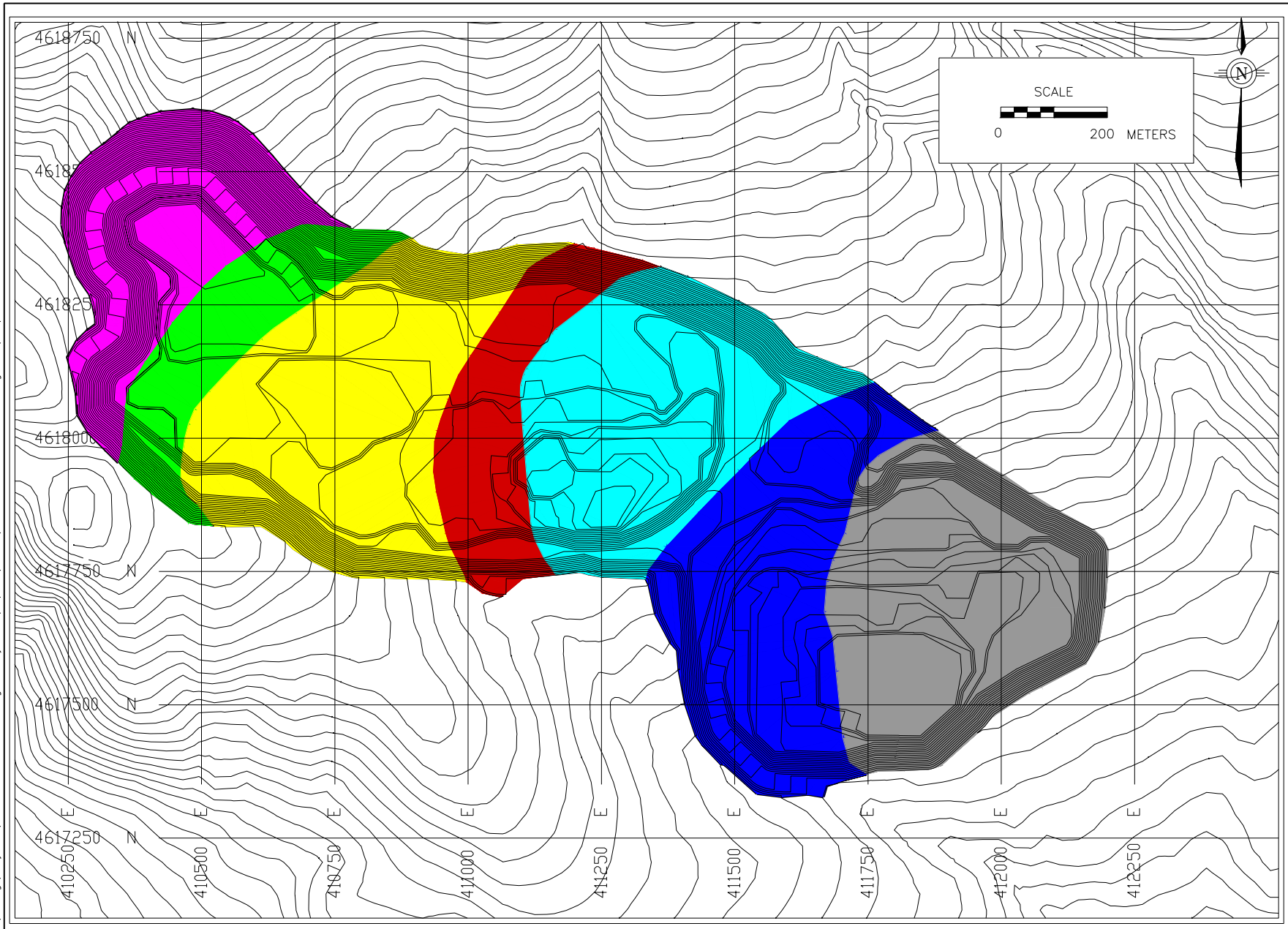
FIGURE 16.8

CASE 2 E-W SECTION



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Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: ***Western Lithium Corp.***	Location: Humboldt County, NV	FIGURE 16.9 CASE 2 N-S SECTION
	Project: Kings Valley Lithium Project	Project no.: 114-311106 / Date of Issue: 12/2011	



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Project:
Kings Valley Lithium Project

Project no.:
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FIGURE 16.10
CASE 2 MINING PHASES

16.2 STAGE II

The Stage II Resource Estimate report is preliminary in nature and addresses only the resource estimate. As such, this section is not applicable to the Stage II area.

17.0 RECOVERY METHODS

17.1 STAGE I

17.1.1 Introduction

The Kings Valley process plant has a design ore throughput of 2,100 tpd for Case 1. The plant would operate 365 days per year and process a total of 689,850 tpy of ore with nominal production of 13,200 tpy of Li_2CO_3, 46,600 tpy of K_2SO_4, and 53,400 tpy of Na_2SO_4. The plant would consist of two sections: a size reduction and thermal ore preparation circuit designed by Tetra Tech and a wet recovery circuit designed by K-UTEC.

17.1.2 Process Design Criteria

Major process design criteria are shown in Table 17.1. Detailed criteria for the dry ore preparation circuit are included in Technical Reference E.

Table 17.1: Major Process Design Criteria

DESCRIPTION	UNIT	VALUE
Climate		
Daily Max Temp	ºC	40
Daily Min Temp	ºC	-16
Mean Daily Max Temp	ºC	29
Mean Daily Min Temp	ºC	-9
Wind (gust)	km/h	93
Elevation	m	1,525
	ft	5,000
Type Of Deposit		
	Open Pit	Lithium
ROM Ore Characteristics		
Specific Gravity (dry)		1.79
Swell Factor	%	25
Moisture Content	%	18
Operating Schedule		
Shifts/Day		2
Hours/Shift	h	12
Hours/Day	h	24
Days/Year	d	365
Plant Availability/Utilization		
Overall Plant Feed (dry)		
Design	t/d	2,100
Crushing	%	80
Reagents	%	90
Calcining	%	90
Leaching	%	90
K-UTEC	%	90
Production		
Head Grade		
Li	%	0.4
K	%	3.82
Na	%	1.46
Mg	%	5.59
Al	%	3.46
Fe	%	1.76
Recovery in Leach		
Li	%	88.9
K	%	78.8
Na	%	83.4
Recovery Overall		
Li	%	87.2
K	%	77.7
Na	%	82.7
Production (avg for LOM)		
Li_2CO_3	t/y	13,174
K_2SO_4	t/y	46,587
Na_2SO_4	t/y	53,414

17.1.3 Process Flowsheet Development

The simplified flowsheet schematics for the dry ore preparation circuit is illustrated in Figure 17.1, Dry Ore Preparation Process Flow Diagram and for the wet recovery circuit is illustrated in Figure 17.2, Wet Recovery Process Flow Diagram, were used as the design basis for this study. The flowsheet is based on the process developed by WLC as described in the PAEE (URS 2010), updated by recent metallurgical testing and development work at Hazen, CICITEM, K-UTEC and Outotec.

Conventional equipment used in the mineral processing and chemical industries is depicted in the flowsheets. The ore would be reduced in size with roll crushers designed to process wet clay in advance of roller mills that would provide the final stage of size reduction. The calciner would be a rotary machine while the dryer and cooler associated with the calciner would be fluidized bed units, all common in the cement industry. The vessels, tanks, evaporators, crystallizers, filters and other equipment in the wet recovery circuit are standard in the salt industry.

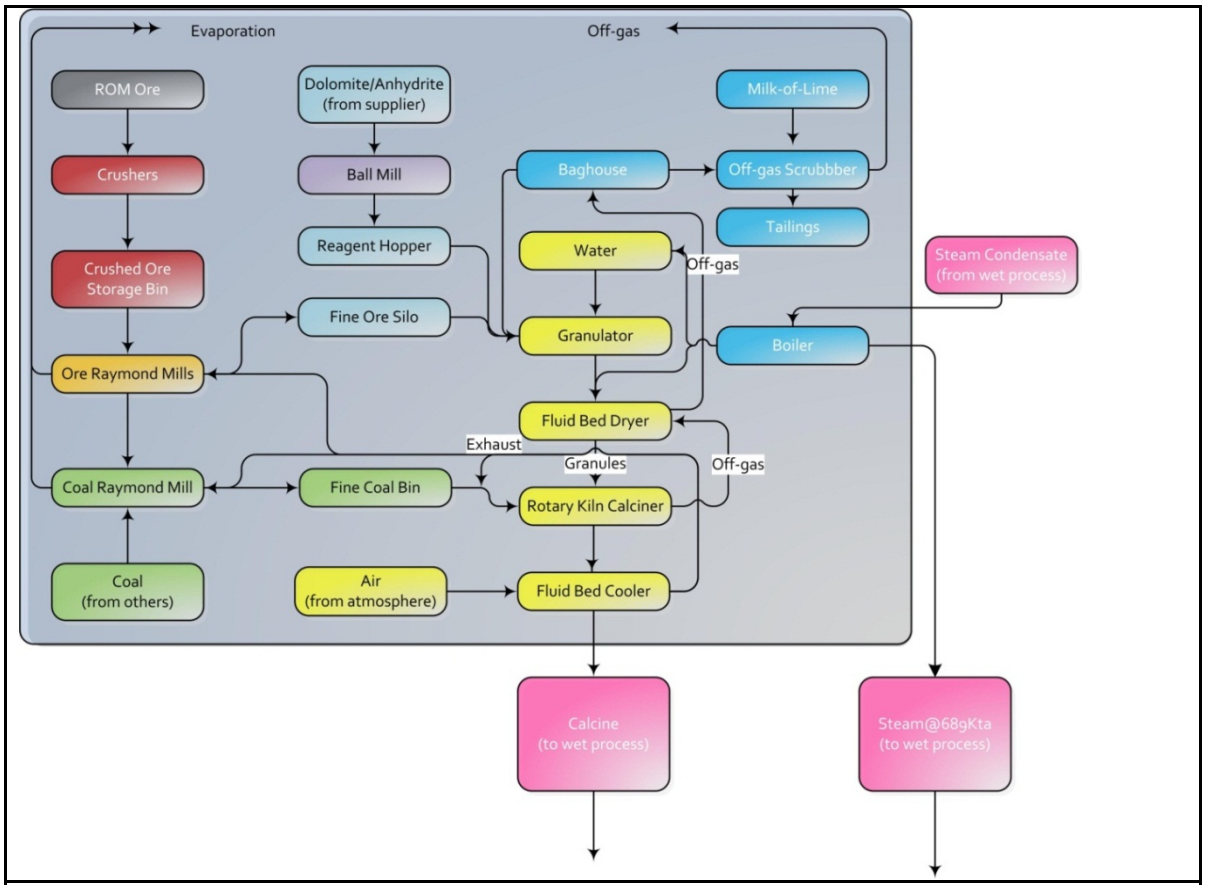


Figure 17.1: Dry Ore Preparation Process Flow Diagram

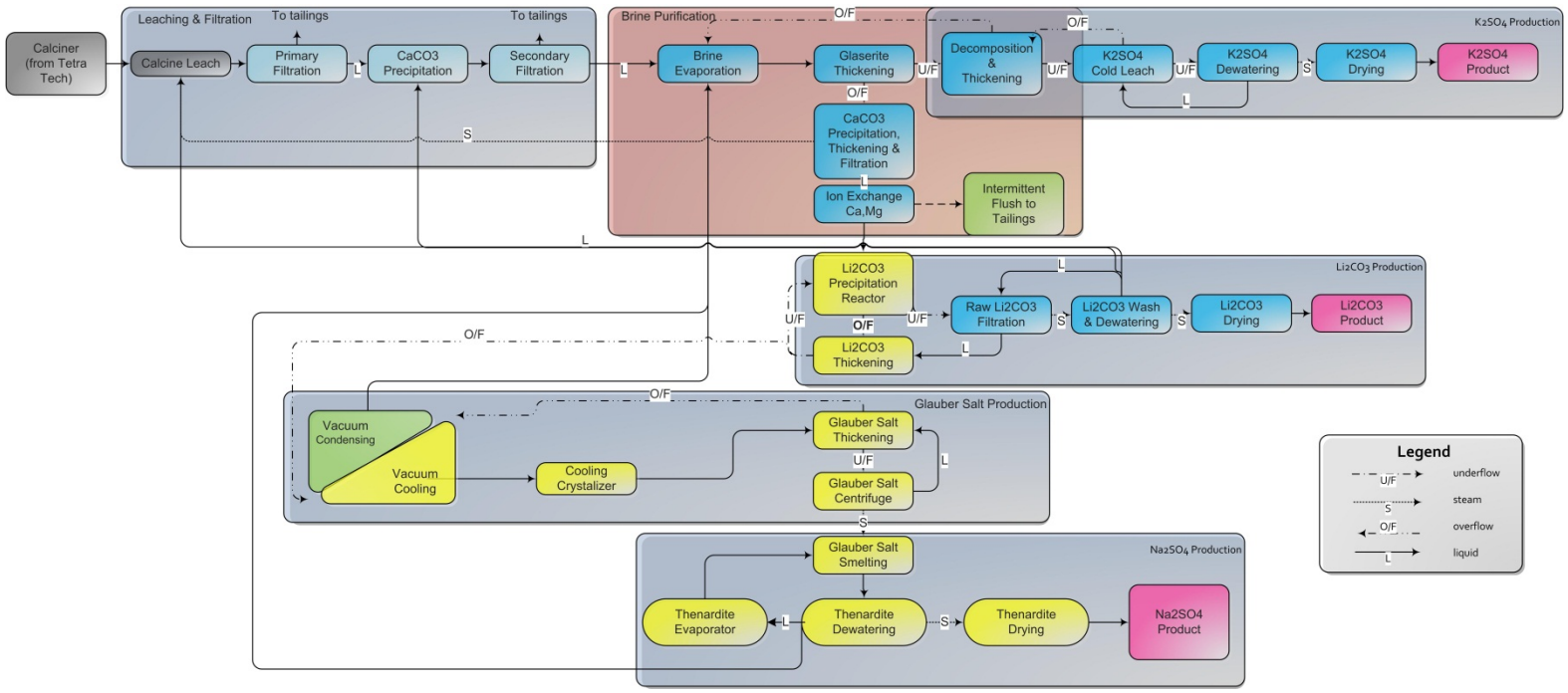


Figure 17.2: Wet Recovery Process Flow Diagram

17.1.4 Process Plant Siting

Considerations dictating the location of the process plant and ancillary facilities include:

- Favorable topography to minimize earthworks
- Proximity to the mine to minimize ore transport cost
- Proximity to existing infrastructure

The identified plant site is on the south side of the pit. The coarse ore stockpiles and grizzly would be located at the edge of the pit to minimize ore transport. The mine truck shop would be adjacent to the plant site.

The fresh water source for process and site requirements is anticipated to be drawn from a Project area well field. WLC has acquired water rights for the project area and initiated a groundwater study.

17.1.5 Dry Ore Preparation Process Description

The first section of the process plant would include size reduction, blending of the ore with reagents, and calcining to convert the silicate minerals to sulfates.

17.1.5.1 Size Reduction

Ore would be mined from the deposit with a Wirtgen surface miner. A Wirtgen has the ability to electively mine ore from the ash layers to minimize dilution and can be configured to produce a minus 50 mm product, eliminating the requirement for a primary crusher. Ore would be trucked to one of eight ore stockpiles, each with a capacity of approximately one day of plant throughput that would be tilled with a disk to facilitate evaporation of a portion of the contained moisture. Ore would be reclaimed from the stockpiles to a grizzly with a front-end loader.

Grizzly undersize would drop onto an apron feeder to a conveyor delivering the ore to two roll crushers that would reduce the ore size to minus 6 mm.

Roll crusher discharge would be conveyed to a crushed ore storage bin for distribution to two Raymond ore mills that would reduce the ore size to minus 149 µm and reduce the moisture content from 16% to 12%. The mill systems would include classifiers and recovery cyclones. Exhaust from the calcine cooler would provide heated air for the mills with propane-fired burners available for supplemental heat. Ground ore would be stored in a dedicated silo.

17.1.5.2 Reagent Preparation

Reagents, minus 10-mm anhydrite and minus 13-mm dolomite, would be trucked to the property and delivered to dedicated silos. The reagents would be fed to a dry grinding circuit for size reduction to minus 149 µm. The dry grinding circuit would include a classifier and recovery cyclone. Product would be conveyed to a reagent hopper.

Ground ore and reagents, anhydrite and dolomite, would be proportionally metered to a granulator for mixing and production of 2-mm-diameter granules at 12% moisture. The granulator would discharge to a rotary table feeder that would provide continuous feed to the dryer.

The granules would be dried with heat recovered from the calciner off-gas in a fluidized bed dryer. The dryer system would include a forced draft blower and an exhaust fan. Moisture in the granules would be reduced from 12% to 0%.

17.1.5.3 Calcining

Dried granules would be conveyed to a coal-fired rotary calciner and heated to 1,050°C for 15 minutes. Calcine would discharge the calciner and be elevated to the fluidized bed cooler. Off-gas from the calciner would exhaust the dryer to a baghouse for recovery of fines containing lithium and a scrubber before venting to atmosphere.

Ambient air would be forced through the cooler to reduce the calcine temperature to 200°C. Cooled calcine would be conveyed to a leach circuit. A bunker would be provided for storage ahead of the leach circuit to allow for orderly reduction in calciner operation in Case of leach circuit downtime. Recovered heat in the cooler exhaust would be utilized for calciner combustion air, Raymond mill air and coal pulverized air.

Coal would be trucked to a 5,000-t stockpile from a transload site on the railroad and reclaimed via a feeder to a coal pulverizing system.

A lime slaker would be provided to produce milk-of-lime for the off-gas scrubber.

A coal-fired boiler would be is provided to supply steam requirements as calculated by K-UTEC.

17.1.6 Wet Recovery Process Description

17.1.6.1 Leaching

Calcine would be leached with water at 70°C in a series of three agitated tanks. Solids from each tank would be filtered and rinsed before advancing to the next tank. Brine would be piped countercurrent to the solids flow to maximized concentration of lithium and potassium. Overflow brine from the first leach tank would advance to a calcium removal stage.

Leach residue from the third leach tank would be filtered and conveyed to a drystack tailings storage facility.

17.1.6.2 First Calcium Removal Stage

Leach brine would be clarified, advanced to a $CaCO_3$ precipitation vessel to effect removal of calcium and filtered before advancing to evaporation.

17.1.6.3 K_2SO_4 Production

Potassium sulfate recovery is based on glaserite crystallization followed by glaserite decomposition and the addition of water.

Leach brine from the first calcium removal stage would be combined with Glauber salt mother liquor, glaserite decomposition liquor and Thenardite mother liquor (Thenardite is anhydrous Na_2SO_4,) for evaporation in a triple effect system. The combined brine would be heated with steam in dedicated heat exchangers. Glaserite suspension from the third evaporator would be thickened, cycloned and centrifuged before advancing via conveyor to an agitated decomposition reactor.

Glaserite decomposition would be affected by the controlled addition of water to the reactor. Decomposition reactor underflow would be cycloned and centrifuged, producing a raw K_2SO_4 product that would advance to a cold-leaching reactor for purification. Overflow from the leaching reactor would be clarified and returned to the evaporators as glaserite decomposition liquor.

Raw K_2SO_4 product would be leached in a reactor with cold water, cycloned, centrifuged and conveyed to a dryer. Dryer discharge, the final product, would be conveyed to the packaging area. Dryer exhaust would be cycloned and filtered before discharging to atmosphere. Cold leach brine would be recycled to the decomposition reactor.

17.1.6.4 Second Calcium Removal Stage

Pre-heated glaserite mother liquor from the evaporation stage would be mixed with pre-heated sodium carbonate to precipitate calcium carbonate in an agitated precipitation reactor at 60°C. Precipitate would be filtered and the sludge returned to the leach circuit. Overflow would be thickened to separate contained $CaCO_3$ solids and the purified overflow advanced to ion exchange for removal of the remaining calcium as well as magnesium. Purified brine would advance to the Li_2CO_3 precipitation reactor.

17.1.6.5 Li_2CO_3 Production

Lithium carbonate recovery is based on precipitation from the glaserite mother liquor with Na_2CO_3.

Sodium carbonate would be added to the agitated reactor to affect precipitation of Li_2CO_3 at 90°C. The underflow would be pumped to a belt filter for washing before advancing to an agitated washing vessel. Overflow would be thickened to clarify the Li_2CO_3 mother liquor and pumped to Glauber salt crystallization.

Underflow from the washing vessel would be filtered, dried and conveyed to the packaging area. Exhaust from the dryer would be cycloned and filtered for recovery of lithium carbonate before venting to atmosphere. Overflow from the washing vessel would be returned as washing brine to the first leach tank and the Li_2CO_3 precipitate belt filter.

17.1.6.6 Na_2SO_4 Production

Sodium sulfate recovery is based on crystallization of barren Li_2CO_3 liquor to produce Glauber salt, and evaporation of water from the suspension.

Li_2CO_3 mother liquor would be pumped to a cooling crystallizer for precipitation of the Glauber salt. The Glauber salt suspension would be pumped to a thickener to separate the solids from the mother liquor. Thickener underflow would be centrifuged and advanced to a conversion vessel.

Conversion of the Glauber salt suspension to Thenardite would be affected by heating the suspension to 60°C. The resulting water-free product would be cycloned, centrifuged, dried and conveyed to the packaging area. Dryer exhaust would be cycloned and scrubbed before venting to atmosphere.

Thenardite mother liquor would be pumped to a triple-effect evaporation system. The liquor would be heated with steam in dedicated heat exchangers. The resulting Thenardite suspension would be returned to the conversion vessel to increase the yield of Thenardite.

17.1.6.7 Recoveries

Overall recoveries from the WLC process are listed below:

Lithium	87.2%
Potassium	77.7%
Sodium	82.7%

17.1.7 Product Packaging

All three products would be packaged in 1000-kg super sacks in vendor-supplied packing lines. A fourth packing line would be provided to pack Li_2CO_3 in 12.5-kg bags.

17.1.8 Equipment Sizing

Details of the equipment that was used to develop the capital cost estimate for Case 1 are shown in the equipment list found in Technical Reference E.

17.1.9 Energy Recovery

The flowsheet is designed to recover energy from the calcination and steam generation processes to reduce overall energy consumption.

17.1.10 Expanded Case

The flowsheet for Case 2, with production doubled in Year 4, would be the same as for Case 1. For the PFS, it is assumed the Case 1 process equipment would be duplicated where required to increase capacity for Case 2.

17.1.11 Process Energy and Water Requirements

The project will require electricity, coal fuel and water to support the process operations.

Electrical power demand for the process will be supplied from the existing 115 kV power line. Based on the estimated electrical demand from the operations, there is sufficient electrical power supply to meet the demand for the Case 1 scenario. This power line is owned and maintained by Harney Electric, the local co-op utility provider. WLC would engage in a contractual agreement with Harney Electric for the supply electricity. Power supply has not been evaluated for Case 2.

Coal would be used to fuel both the calciner and the necessary boiler for steam requirements. While WLC has not selected a coal supplier at this time, the Project has begun looking into potential suppliers from Wyoming or Utah. Coal would be railed near the project site to an unloading facility near Winnemucca. From there, the coal would be hauled and stockpiled at the Project site via truck.

Water rights have been attained by WLC. An independent groundwater study is currently under preparation, but results of the study have not been completed. It is anticipated from preliminary results from this effort that water would be sourced at the Project site; however, this remains to be confirmed.

17.2 STAGE II

The Stage II Resource Estimate report is preliminary in nature and addresses only the resource estimate. As such, this section is not applicable to the Stage II area.

18.0 INFRASTRUCTURE

The following applies to the Project. Note, in preparation of the Stage II Resource Technical Report, infrastructure is not considered and have no economics applied.

18.1 EXISTING PROJECT AND SITE ACCESS

The Project site in northern Nevada is near several established large-scale mining operations. The site has the advantage of good accessibility on paved all-season roads from Interstate 80 at Winnemucca 70 km north via US Highway 95 to Orovada, and 33 km west-northwest on State Highway 293 to the project site.

Roads accessing the Stage I area are all-season and in generally good repair, but may be closed for short periods due to extreme weather in the winter. Local access to the Stage II Lens is currently an unimproved gravel road that at times during the winter months can be impassable. The access to this area will require upgrading.

Rail service for potentially transloading coal is available at Winnemucca, Golconda (24 km east of Winnemucca), and the Valmy power station.

18.2 PROXIMITY OF PROPERTY TO POPULATION CENTERS

Orovada is a center for the local agricultural community with an estimated area population of less than 1,000. Winnemucca is a major mining supply hub with all the services of a town with a population of 7,400, including health and medical.

Elko, 264 km east of Winnemucca, and Reno, 264 km southwest of Winnemucca (both on Interstate 80), offer commercial air service.

Because of the large-scale gold mining industry in the Winnemucca area, local resources include all of the amenities required for large-scale mining. The area is about 50 km north of the now depleted Sleeper gold mine and 100 km northwest of Twin Creeks, Turquoise Ridge, and Getchell gold mines. Several other gold and copper mines are in the area, providing an experienced work force and adequate support for mining operations. Most of the workers may have to be sourced in Winnemucca because of the sparse population in the Project area.

The proximity to Winnemucca eliminates the need for a man camp at site. Employees could be bussed from Winnemucca on a daily basis.

18.3 AVAILABLE UTILITIES AND RESOURCES

Adequate electrical power is available, but power lines may need to be added and/or upgraded to provide power to the site as the Project develops. A 115-kV power line runs adjacent to State Highway 293 and has adequate capacity for the Case 1 process plant equipment and infrastructure. The electrical capacity needs for Case 2 has not been evaluated.

No surface water is present at the site. However, water is available in the region and can be sourced from either the Kings River Valley, west of the project, or Queens River Valley located east of the project. The nearest railroad access is located in Winnemucca.

18.4 STAGE I PFS INFRASTRUCTURE DESIGN

The road access for Stage I are laid out to provide access to the site, the open pit mining operation, the waste rock storage facility, processing plant, and dry stack TSF. All roads are with a width of 30 m, not including any required safety berms.

Process fluids will be contained in the Barren Purge Containment Pond, located south of the processing facilities. The lined pond will be constructed with 2.5H:1V side slopes and a total depth of 5.0 m, which includes a minimum freeboard requirement of 1.0m. The pond will be constructed with a bottom elevation of 1498 m above mean sea level (amsl) and a total capacity of approximately 51,000 m^3 of solution at a maximum water elevation of 1501.5 amsl. A small pipeline corridor will be constructed to convey piped flows from the processing facilities to the Barren Purge Containment Pond. Both the pond and the pipeline corridor will be lined with a 60-mil LLDPE geomembrane liner underlain by a 0.3 m thick (minimum) subgrade layer consisting of reworked and compacted fine grained subgrade.

A basic water balance was developed to determine the sizing of the pond. The results of the water balance indicated that the pond could fully contain all of the inflows from the processing facilities as well as precipitation. The only outflow for the system is evaporation. Normal conditions for precipitation and evaporation were assumed and average values were extracted from the Kings River Valley and Rye Patch Dam weather stations; however, the actual evaporation rate may be lower due to the composition of the process solution. This assumption will be verified in more advanced studies.

Surface water will be diverted away from the open pit mining operations, waste rock storage facility, processing plant, and dry stack TSF via surface water diversion channels. The diversion channels were sized to contain the flows from a 100-year, 24-hour storm event, and the cross-section was designed so that the diversion channels can be constructed using equipment available at the site. The diversions will have a bottom width of 5.0 m, side slopes of 2H:1V, and a total depth of 2.0 m, including 1.0 m of freeboard. The diversions will be constructed in stages in order to accommodate the changing pit and waste rock storage facility footprints.

19.0 MARKET STUDIES AND CONTRACTS

19.1 STAGE I

19.1.1 Introduction

WLC commissioned a market study for the three saleable products recovered from the process described in this study: lithium carbonate, potassium sulfate, and sodium sulfate. The reports were updated in mid-year 2011 and contain detailed reviews of the supply and demand outlook for each commodity. In its initiation of the market evaluation, WLC engaged a consultant in each associated area, to determine the market potential for the lithium, potassium, and sodium products. The results of these efforts are presented below.

WLC has not entered into any contractual off-take agreements for the sale of the three products.

19.1.2 Lithium Carbonate (Li_2CO_3)

19.1.2.1 Demand

The largest consumers of lithium are the manufacturers of glass, ceramics and batteries. Future demand for lithium will be driven by the volume of production in these industries.

19.1.2.2 Specifications

The minimum Li_2CO_3 content for product used by the battery industry is 99.5%.

19.1.2.3 Prices

The market study concluded that US$6000 per tonne of Li_2CO_3 is a reasonable estimate in the current economic environment and has been used FOB the Project site as the basis for the PFS.

19.1.2.4 Market Entry

An established market exists and entry is competitive and contingent on price and the ability to insure delivery over the life of a contract

19.1.3 Potassium Sulfate (K_2SO_4)

19.1.3.1 Demand

Ninety-five percent of current global consumption of potassium is in fertilizers. Potassium sulfate is the second most common potassium fertilizer. Future demand for production from Nevada will be dependent on agricultural consumption on the west coast of the US, and exports.

19.1.3.2 Specifications

The minimum potash (K_2O) content for potassium sulfate fertilizer is 50%.

19.1.3.3 Prices

The market study concluded that US$600 per tonne is a reasonable estimate of the average selling price in 2011 and this price FOB the project site has been used as the basis for the PFS.

19.1.3.4 Market Entry

An established market exists and entry is competitive and contingent on price and the ability to ensure delivery over the life of a contract Potassium Sulfate (K_2SO_4).

19.1.4 Sodium Sulfate (Na_2SO_4)

19.1.4.1 Demand

The largest consumers of sodium sulfate are the manufacturers of soap and detergents and glass, and pulp and paper industry. Future demand for lithium will be driven by the volume of production in these industries.

19.1.4.2 Specifications

The minimum sodium sulfate content for commercial grade is 99%.

19.1.4.3 Prices

The market study relied on a USGS estimate of US$140 per tonne. A price of US$75 per tonne FOB the project site has been used as the basis for the PFS due to the distance to potential consumers.

19.1.4.4 Market Entry

An established market exists and entry is competitive and contingent on price and the ability to ensure delivery over the life of a contract Sodium Sulfate (Na_2SO_4).

19.2 STAGE II

Neither WLC nor the QPs have engaged in the review of market studies on contracts with respect to Stage II.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 STAGE I

This section summarizes the overall permitting and approval program that will be and have been completed by WLC under both the Case 1 and Case 2 Project scenarios in order to construct, operate and close the proposed Kings Valley Lithium mine and ore processing facilities (Project). This program includes obtaining the necessary permits and approvals from various federal, state and local government agencies as appropriate, and the completion of a baseline study program to collect data for environmental, natural and socio-economic resources that will be used to support the permitting and approval process. As part of the overall permitting and approval process, an environmental documentation program completed in accordance with the National Environmental Policy Act of 1969 (NEPA) will be completed to assess the potential impacts to the human and natural environment that could result from the implementation of project activities.

20.1.1 Environmental Baseline Studies

A baseline study program has been initiated to collect the environmental, natural resource and socio-economic data required to support the completion of the multi-federal and state agency permitting and approval program, and the anticipated environmental documentation process that will be required under NEPA. This baseline program includes, but is not limited to, studies for the following standard resource issues and topics:

- General vegetation
- General wildlife (including extensive surveys for Greater Sage-grouse)
- Special status vegetation and wildlife species including those species managed under the requirements of the Federal Endangered Species Act of 1973, as amended
- Invasive, non-native plant species including noxious weeds
- Soils
- Paleontology
- Water quality and quantity including surface and groundwater hydrology (including surface drainage studies, seep and spring studies, and pit lake studies)
- Geochemistry
- Jurisdictional wetlands and waters of the United States as required by Section 404 of the Federal Clean Water Act of 1977, as amended
- Air quality as required by the Federal Clean Air Act of 1963, as amended
- Cultural resources as managed under the National Historic Preservation Act of 1966, as amended, and the Archaeological Resources Protection Act of 1979
- Native American traditional values as regulated by various federal laws and regulations including the American Indian Religious Freedom Act of 1978, as amended, the Native

American Graves Protection and Repatriation Act of 1990, and Executive Order 13175 – Consultation and Coordination with Tribal Governments

- Environmental Justice in accordance with Executive Order 12898 – Federal Actions to Address Environmental Justice in Minority Populations and Low-Income Providers
- Hazardous materials and solid waste
- Range management
- Social and economic impacts
- Aesthetics including noise and visual assessments

20.1.1.1 WLC Baseline Study Program

The following baseline studies and agency consultation activities have been completed, are ongoing or are proposed for the Project

Completed Studies and Agency Consultation Activities

- Air Sciences, Inc., 2012. *Quality Assurance Project Plan*, for the Western Lithium Corporation Rev1, Kings Valley Lithium Project. Air Sciences Project No. 270-5. June 2012.
- AMEC, 2013. *Supplemental Geotechnical Recommendations Kings Valley Clay Mine Project Humboldt County, Nevada*. Prepared by AMEC Earth and Environmental, Inc. Amec Project No. 10-417-00961. June 2013.
- AMEC, 2011. *Prefeasibility Level Geotechnical Study Report, Kings Valley Lithium Project, Humboldt County, Nevada*. Prepared by AMEC Earth and Environmental, Inc. Amec Project No. 10-417-00961. March 2011.
- ASM Affiliates, Inc., 2008. *A Class III Cultural Resources Inventory of 1230 Acres for the Western Energy Development Corporation Thacker Pass Lithium Exploration Project Humboldt County, Nevada*. November, 2008.
- Chambers, 2011. *A Class III Cultural Resource Inventory of the Kings Valley Lithium Project Expansion, Humboldt County, Nevada*. January 2011.
- Chambers Group, Inc., 2009. *Letter Report of Findings for Baseline Vegetation Surveys for the Kings River Valley Exploration Project, Humboldt County, Nevada*. Letter Prepared for Mr. Doug Bowden, Western Energy Development Corporation. June 8, 2009.
- Enviroscientists, Inc., 2008. *Memorandum Results for the Kings Valley Lithium Exploration Project Pygmy Rabbit Survey, Humboldt County, Nevada.* Prepared by Ms. Sara Thorne, Enviroscientists, Inc., Prepared for Mr. Dennis Bryan, Western Lithium Corporation and Mr. Ken Detweiler, Bureau of Land Management. December 3, 2008.
- Enviroscientists, Inc., 2010. *Memorandum Results for the Kings Valley Lithium Amended Plan of Operations Biological Survey, Humboldt County, Nevada*. Prepared by Ms. Sara Thorne, Enviroscientists, Inc., Prepared for Ms. Celeste Mimnaugh and Ms. Janet Hook, Bureau of Land Management. November 16, 2010.
- Great Basin Ecology, Inc.; 2012. *Sage-Grouse Habitat Assessment – Kings Valley Lithium Project Area.* Report Prepared for Western Lithium Corporation. September 2012.

- Great Basin Ecology, Inc.; 2013. *2013 Kings Valley Clay Mine Project 2013 Winter Sage-Grouse Survey*. Report Prepared for Western Lithium Corporation. March 2013; Revised November 2013.
- Great Basin Ecology, Inc.; 2013. Memorandum. Habitat Assessment. To: Catherine Clark, From Gary Back. August 8, 2013.
- JBR Environmental Consultants, Inc., 2011a. *Waters of the U.S. Jurisdictional Determination, Western Lithium Corporation, Kings Valley Lithium Project, Humboldt County, Nevada (Draft)*. December 14, 2011.
- JBR Environmental Consultants, Inc., 2011b. *Baseline Vegetation Transect Survey Report, Western Lithium Corporation, Kings Valley Lithium Project, Humboldt County, Nevada (Draft)*. December 16, 2011.
- JBR Environmental Consultants, Inc., 2012. *Baseline Biological Survey Report, Western Lithium Corporation, Kings Valley Lithium Project, Humboldt County, Nevada.* July 9, 2012.
- JBR Environmental Consultants, Inc. 2012. Winter Greater Sage-Grouse Survey Report Western Lithium Corporation Kings Valley Lithium Project Humboldt County, Nevada. April 4, 2012.
- J.C. Brennan & Associates, Inc., 2013. Noise Contour Modeling – Kings Valley Clay Mine Project. November 26, 2013.
- J.C. Brennan & Associates, Inc., 2012. *Environmental Noise Assessment, Kings Valley Lithium Project, Humboldt County, Nevada.* February 8, 2012.
- Kautz Environmental Consultants, Inc. (Kautz), 2007. *A Cultural Resources Inventory of Drill Pads and Associated Access Routes for Proposed Lithium Exploration, Thacker Pass, Humboldt County, Nevada*. Prepared by Dwight D. Simons, Ph.C. and Robert R. Kautz, Ph.D.
- Lumos and Associates, Inc., 2011. *Preliminary Groundwater Investigation for the Kings Valley Lithium Project (Stage I Property).* Lumos Project Number 7618.300. May 11, 2011.
- Lumos and Associates, 2011. *Kings Valley Lithium Project Spring Survey.* Lumos Project Number 7618.400. July 25, 2011.
- McCabe, et. al. 2012. *Volume 1: Report, A Class III Inventory of a 2,256 Acre Parcel for Western Lithium's Kings Valley Lithium Project, Humboldt County, Nevada*. BLM Report Number CR2-3157(P). Prepared by Far Western Anthropological Research Group, Davis, California. April 2012.
- Nevada Department of Wildlife, 2010-2013. *PDF Maps Showing Bighorn Sheep Movement Associated with the Kings Valley Uranium Exploration Project and the Kings Valley Lithium Exploration Project Boundaries*. Maps provided weekly to Western Lithium Corporation.
- Schlumberger Water Services, 2011. *Technical Memorandum. Preliminary Results for the Western Lithium Corporation King's Valley Lithium Project Surface Water Evaluation.* Prepared by Alonso Vidal, Schlumberger Water Services; Prepared for Catherine Clark, Western Lithium Corporation. October 20, 2011.
- Schlumberger Water Services, 2012. *Groundwater Occurrence at the Western Lithium Kings Valley Clay Mine*. Technical Memorandum. October 2012.

- Schlumberger Water Services, 2013. Technical Memorandum. King's Valley Springs Impact and Monitoring Analysis. Prepared by Tyler Cluff, PG, Schlumberger Water Services; Prepared for Catherine Clark, Western Lithium Corporation. July 19, 2013.
- SRK Consulting, (U.S.), Inc. 2014. Whole Rock Analysis on clay surface samples and clay ore/waste rock. (no formal letter or report prepared. Information summarized in Kings Valley Clay Mine Final Environmental Assessment 2014).
- SRK Consulting, (U.S.), Inc., 2013. Kings Valley Lithium Project Supplemental Baseline Biological Survey Report. SRK Project Number 357800.070. June 2013.
- SRK Consulting, (U.S.), Inc., 2013b. *Western Lithium Corporation, Spring and Seep Survey June 2013, Kings Valley Clay Mine.* SRK Project Number 357800.020. July 2013.
- SRK Consulting (U.S.), Inc. 2013c. Revised Waste Rock Characterization Report for the Kings Valley Clay Mine – Final. SRK Project Number 357800.060. Revised August 2013.
- SRK Consulting, (U.S.), Inc., 2012a. *Western Lithium Corporation, Spring and Seep Survey September 2011, Kings Valley Lithium Project.* SRK Project Number 357800.020. January 2012.
- SRK Consulting, (U.S.), Inc., 2012b. *Western Lithium Corporation, Spring and Seep Survey December 2011, Kings Valley Lithium Project.* SRK Project Number 357800.020. February 2012.
- SRK Consulting (U.S.), Inc., 2012c. *Western Lithium Corporation, Spring and Seep Survey June 2012, Kings Valley Lithium Project*. SRK Project Number 357800.020. July 2012.
- SRK Consulting (U.S.), Inc., 2012d. *Memorandum. Kings Valley Lithium Project – Waste Rock Characterization Program Summary.* Prepared by: Amy Prestia, Ruth Warrender, Rob Bowell; SRK Consulting (U.S.), Inc.; Prepared for Janet Hook, Bureau of Land Management. January 24, 2012.
- SRK Consulting (U.S.), Inc., 2012e. *Waste Rock Characterization Report for the Kings Valley Clay Mine*. September 2012.
- Tetra Tech, 2011. *Western Lithium Corporation Geochemical Characterization Program, Kings Valley Project,* Humboldt County, USA. August 29, 2011.
- U.S. Department of Interior, Bureau of Land Management; Nevada Department of Wildlife, Cooperating Agency, 2009a. *Environmental Assessment DOI-BLM-NV-W010-2010-0001-EA Kings Valley Lithium Exploration Project*. December 2009.
- U.S. Department of Interior, Bureau of Land Management, 2009b. *Decision and Finding of No Significant Impact Kings Valley Lithium Exploration Project Environmental Assessment DOI-BLM-NV-WO10-0001-EA*. December, 31 2009.
- U.S. Department of Interior, Bureau of Land Management, 2014. Environmental Assessment DOI-BLM-NV-W010-2013-0046-EA. Western Lithium Corporation Kings Valley Clay Mine, March 2014.
- U.S. Department of Interior, Bureau of Land Management, 2014. *Decision and Finding of No Significant Impact Western Lithium Corporation Kings Valley Clay Mine Environmental Assessment DOI-BLM-NV-W010-2013-0046-EA,* March 5, 2014.

- Western Lithium Corporation, 2011. *Western Lithium Corporation Kings Valley Lithium Exploration Project Humboldt County, Nevada, Amendment #1 Plan of Operations N85255 Nevada Reclamation Permit No. 0301.* May 2011.
- Western Lithium Corporation (WLC), 2012a. *Western Lithium Corporation, Spring and Seep Survey March 2012, Kings Valley Lithium Project.* April 2012.
- Western Lithium Corporation, 2012b. *Sage Grouse Lek Survey Report,* Western Lithium Corporation, Kings Valley Lithium Project, Humboldt County, Nevada. May 23, 2012.
- Western Lithium Corporation, 2012c. *Kings Valley Clay Mine Plan of Operations and Reclamation Permit Application*, October 2012.
- Western Lithium Corporation, 2012d. *Kings Valley Clay Mine Noxious and Invasive Species Management Plan.* October 2012.
- Western Lithium Corporation, 2014. Memorandum. *Waters of the State Assessment, Humboldt County, Nevada.* Memorandum to the Project File. Prepared by Catherine Clark. March 14, 2014.
- Western Lithium Corporation, 2013b. *Kings Valley Project Spring and Seep Survey December 2012* Report. March 2013.
- Western Lithium Corporation, 2013c. *Kings Valley Project, Spring and Seep Survey March 2013.* April 2012.

The results of all baseline surveys performed to date indicate the following:

- The Project area does not contain any jurisdictional waters of the US or wetlands, and would not require a Section 404 permit under the CWA. On October 12, 2012, the U.S. Army Corps of Engineers concurred with the survey findings and verified that there are no jurisdictional waters of the U.S. (including wetlands) within the Kings Valley survey area. Project activities will not impact any threatened or endangered vegetation or wildlife species. Based on these survey results, a Section 7 Consultation Process with the US Fish and Wildlife Service is not required for the Project (WLC, 2011d).

- Based on the geochemical characterization results, the waste rock material associated with the project presents a low risk for acid rock drainage and metal leaching (ARDML). Humidity cell tests were run for 47 weeks (and 74 weeks for cell 9). Results from the humidity cell tests indicate acid generation is not predicted for any of the waste rock material types that will be mined (SRK, 2013c; SRK, 2012d; SRK, 2012e).

Agency Consultation Activities

In June 2011, WLC initiated consultation and coordination activities with appropriate federal and state regulatory agency specialists to ensure that the environmental and natural baseline study data is collected using approved procedures and protocols to meet appropriate data adequacy standards that will support the multi-federal and state agency permitting program and the anticipated NEPA environmental documentation process (WLC, 2011i). Since this time, WLC has continued to keep agencies involved as to the progress of the project. This consultation and coordination program will continue through the completion of the definitive feasibility study process and up to the initiation of the

NEPA environmental documentation program and the completion of the permitting and approval process.

In accordance with Nevada Bureau of Land Management (BLM) guidance outlined in Instructional Memorandum IM NV-2011-004, In January 2012, WLC held a pre-mine plan of operations review meeting with the management and Interdepartmental (ID) Team of the BLM's Winnemucca District Office. A follow-up meeting between the BLM and WLC was performed in April 2014.

Environmental Baseline Study Data Gap Analysis

Tetra Tech's review of the completed and ongoing baseline study program indicates that WLC has initiated the essential environmental and natural resource baseline studies to support the permitting and approval program and the NEPA environmental documentation process for the Project. The studies completed to date and the ongoing studies appear to be adequate and in conformance with what are typically performed for mining projects located on BLM-managed public lands in Nevada.

As part of the ongoing baseline study program and the definitive feasibility study, WLC will develop a scope of work for an air quality monitoring and modeling program. The scope of work will be based on the components selected for the final process engineering design. WLC will coordinate this activity with the key regulatory agencies including the Nevada Division of Environmental Protection (NDEP), Bureau of Air Quality (BAQ), and the BLM to ensure that the program will support the state air quality permitting process and the NEPA environmental documentation program. Note that the NDEP BAQ has primacy for air quality activities in Humboldt County under the Federal Clean Air Act of 1970, as amended.

20.1.1.2 Environmental Baseline Study Cost Estimate

Between 2007 and 2014, WLC spent an estimated US$1,300,000.00 for baseline studies and associated analysis. The 2014 cost estimate will be revised as Project development activities progress, which will include the completion of the definitive feasibility study.

The estimated cost to complete the air quality monitoring and modeling program will be developed after the final process engineering design has been selected and the proposed air quality program scope of work has been reviewed with the NDEP BAQ and the BLM.

20.2 STAGE II

There are currently no known environmental, socio-economic, or cultural issues that will materially affect the implementation of Stage II. The Stage II report, issued May 15, 2010 titled, "NI 43-101 Technical Report State II (South Lens) Resource Estimate Kings Valley Project" states that no environmental liabilities are known to exist at the Project site.

20.2.1 Existing Environmental Documents

1. Notice of Intent to conduct lithium exploration on federal lands submitted to the Winnemucca Field Office of the U.S. Bureau of Land Management (BLM; May 29, 2007; signed by Pamela Klessig, President WEDC).

2. Letter response from the BLM, Winnemucca Field Office, dated 26 June 2007, signed by Dave Hayes, Assistant Field Manager, Nonrenewable Resources, acknowledging completion and acceptance of the Notice of Intent granting conditional permission to proceed.

3. Letter of acknowledgement of completion of the cultural survey from the BLM, Winnemucca Field Office, dated October 3, 2007, signed by Dave Hayes, Assistant Field Manager, Nonrenewable Resources, granting conditional permission to proceed.

4. Letter of acknowledgement of receipt of bond requirements from the State of Nevada Commission on Mineral Resources, Division of Minerals, dated 5 September 2007, addressed to WEDC, signed by Doug Dresner, Deputy Administrator, for Alan R. Coyner, Administrator, confirming that the required security bond has been paid and Receipt No. 02020 for the amount of the bond. The letter indicates that the information was forwarded to Scott R. Richey, BLM, Winnemucca Field Office.

5. U.S. Department of the Interior, BLM, Surface Management Personal Bond Rider, Form OMB No. 1004-0194, which provides the details of Surface Management Bond No NVB-000493 on behalf of WEDC.

20.2.2 Baseline Studies

There are no new baseline studies have been performed beyond the work outlined in Section 20.2.1.

20.2.3 Permitting

WLC provided documents relating to permitting for the lithium exploration. The Notice of Intent to conduct lithium exploration on federal lands was submitted to the Winnemucca Field Office of the U.S. Bureau of Land Management (BLM; May 29, 2007). The letter response from the BLM signed by Dave Hayes, Assistant Field Manager Nonrenewable Resources, Winnemucca Field Office (dated June 26, 2007), indicated that the Notice of Intent was complete and that other permits were not required because the area of disturbance is less than five acres. That letter required that WLC have a cultural resources study performed and acknowledgement that the required financial guarantee was accepted for the project prior to beginning work on the project.

On October 3, 2007, WEDC received a letter from Dave Hayes of BLM that acknowledged receipt and acceptance of the cultural survey and permitted operations as soon as the financial security was received. Financial security was acknowledged in a letter from the State of Nevada Commission on Mineral Resources, Division of Minerals dated 5 September 2007. The letter indicated that copies were sent to the appropriate BLM Field Offices. AMEC was also provided with the Surface Management Personal Bond Rider form OMB No.1004-0194 detailing Surface Management Bond No. NVB-000493, and a receipt for the total amount of the bond.

A Plan of Operation (POO) was submitted to the BLM and the Nevada Department of Environmental Protection (NDEP) in May 2008 for an extensive drilling and trenching exploration program to further delineate the resources of the PCD lens. That action included an Environmental Assessment (EA). A revision to the POO was filed on November 10, 2009 and approved by the BLM in January 2010. Subsequent approvals from the BLM were issued under separate “work plans”. WLC submitted Work

Plan #6 to the BLM in September 2011. The BLM granted approval/authorization in October 2011, allowing for additional exploration drilling.

20.3 WASTE AND TAILING DISPOSAL, SITE MONITORING, AND WATER MANAGEMENT

The management of waste rock, tailings disposal facilities, water management and site monitoring during operations are key issues for any mine and ore processing operation located in the State of Nevada. The BLM's Surface Management Regulations contained in Title 43 of the Code of Federal Regulations (43 CFR 3809) require that mining operations authorized by the federal mining laws or anyone intending to develop mineral resources on public lands must prevent unnecessary or undue degradation of the land and reclaimed areas. The State Mining Regulations contained in Nevada Administrative Code (NAC) 445A.350 through NAC 445A.447 mandate that mine and ore processing/fluid management system operations "will not degrade waters of the State". These federal and state regulatory requirements are met by constructing and managing mine and ore processing/fluid management operations in accordance with approved engineering design plans and regulatory agency approved operating plans.

20.3.1 Waste Rock Management

For both the Case 1 and Case 2 Project options, waste rock from the open pit will be managed using a combination of two methods: construction of a surface waste rock storage facility (WRSF) and in-pit backfilling.

During operations, the WRSF will be constructed by end dumping in lifts to a stable, overall slope configuration of 3.0 Horizontal to 1.0 Vertical (3.0H:1.0V). This construction method will facilitate final reclamation, revegetation and closure of the waste rock storage facility.

Waste rock will also be backfilled into a portion of the open pit during mine operations. Mining operations will be phased to allow the in-pit backfill to be completed. The in-pit backfill will be designed and constructed to maintain long term stability. The in-pit backfilling will reduce the amount of total mine site surface disturbance.

Based on the geochemical characterization results, the waste rock material associated with the project presents a low risk for acid rock drainage and metal leaching (ARDML). Humidity cell tests were run for 47 weeks (and 74 weeks for cell 9). Results from the humidity cell tests indicate acid generation is not predicted for any of the waste rock material types that will be mined (SRK, 2013c; SRK, 2012d; SRK, 2012e).

The waste rock management program will be refined as the Project progresses.

20.3.2 Tailings Management

For both the Case 1 and Case 2 Project options, tailings will be dewatered in the process facility, then placed (dry stacked) on a linear low density polyethylene (LLDPE) lined surface storage facility. The TSF will be constructed in lifts to a stable, overall slope of 3.0H:1.0V. This construction method will facilitate final reclamation, revegetation, and closure of the TSF. Runoff from the TSF will be collected in an HDPE lined pond and pumped back to the process facility for reuse.

The tailings management program will be refined based on the results of the ongoing process engineering design study and the definitive feasibility study.

20.3.3 Site Monitoring

Both the BLM and the NDEP BMRR will stipulate requirements for the monitoring of the mine and ore processing operations, including the fluid management system, to ensure that they do not contribute to any unnecessary or undue degradation of public lands or degrade waters of the State. The BLM monitoring stipulations will be issued as part of its Record of Decision for approving the Project under its Surface Management Regulations contained in 43 CFR 3809. The NDEP BMRR monitoring requirements will be stipulated in the Water Pollution Control Permit issued for the Project in accordance with the State Mining Regulations contained in NAC 445A.350 through NAC 445A.447.

20.3.4 Water Management

WLC has conducting detailed surface and groundwater hydrologic studies and a detailed geochemical characterization study for the proposed Project. Based on "Western Lithium Corporation the Project Groundwater Quantity and Quality Impacts Analysis Revision 3, August 2013; prepared by Schlumberger Water Services USA, Inc. (August 2013), key results obtained for dewatering and water supply production are:

Sustainable pumping between 280 and 350 gpm in the claystone unit is possible, although production yields may decline as dewatering reduces the saturated interval.

Additional water supply pumping of 80 gpm could be obtained from the area of well WSH-3 and other locations east of the open pit in basalt and claystone units.

Drawdown impacts for post-closure partial pit backfill include the following key results (Schlumberger 2013):

The 10 foot drawdown contour at the end of mining is less than six miles wide, and extends approximately one mile beyond the project area boundary. The maximum drawdown extent is slightly bigger than at the end of mining.

Maximum drawdown occurs under the partial backfill scenario at 12 years.

95% recovery is achieved after 55 years.

At 100 years (equilibrium), the largest drawdown is ~210 feet along the north edge of the west pit's pit lake.

Water levels recover 20 feet above pre-mining levels in the partial backfill scenario.

The maximum 10 foot drawdown contour encompasses springs SP-001, SP-002, SP-010, and SP-011 with SP-012 near the edge of the 10 foot drawdown contour. SP-001 is affected directly by open pit mining and dewatering. The impact to other springs is uncertain due to the potential perched nature of the springs.

After equilibrium, the particle backfill scenario's 10 foot drawdown contour still encompasses spring SP-001.

The 10 foot drawdown contours do not reach other water users in Kings River or Quinn River Valleys, nor do they extend into the alluvial basins of either valley.

According to the Schlumber report (August 2013), groundwater flow impacts include the following key results:

- A pit lake forms in the west pit at an elevation of 4,920 feet. The realistic range of pit lake elevations is 4,900 to 4,933 feet.
- The pit lake formed is predicted to result in 5.5 gpm of flow through at equilibrium. Flow out of the pit lake follows the general groundwater gradient towards the southeast. The realistic range of flow through could be 4.7 to 7.7 gpm.
- The largest inflow contribution to the pit lake is groundwater, estimated at 20 gpm at equilibrium. The larges outflow is ET, estimated at 28 gpm.

20.4 Regulatory Agency Permitting Process

This section summarizes the permitting and approval process for the key federal and state agencies. A list of federal, state and local permits and approvals is presented Table 20.1.

20.4.1 Federal, State and Local Agency Permitting

A multi-agency regulatory process will be completed to obtain the required federal, state and local agency permits and approvals necessary to construct, operate and ultimately close the Kings Valley mine and ore processing operations. The Project site is located in northern Humboldt County, Nevada, approximately 100 km northwest of the City of Winnemucca, on federal public lands administered by the Winnemucca District Office of the BLM.

20.4.1.1 BLM Permitting and Approval

As lead agency, the Winnemucca District Office of the BLM would issue federal approval for the proposed Project in accordance with its Surface Management Regulations contained in 43 CFR 3809. The BLM will require the submittal of a PoO, prepared in accordance with 43 CFR 3809, inclusive, that details the proposed mine and ore processing/fluid management system operations, along with reclamation and closure activities. The BLM will also require the placement of a financial guarantee (reclamation bond) to ensure that mine and process site reclamation is completed in accordance with the approved plan. The BLM will also require the establishment of a long-term financial trust program to ensure that adequate funds are available to manage post-closure site activities into a pre-determined future period. Note that the PoO and financial guarantee, including the long-term trust, are coordinated with the Reclamation Branch of the NDEP, BMRR as discussed below.

The BLM will also coordinate the cultural resources review program with the Nevada State Historic Preservation Office (SHPO) that is required under Section 106 of the Federal NHPA. BLM will also coordinate the government-to-government consultation process with appropriate Native American tribal governments in accordance with various federal laws and regulations including the American Indian Religious Freedom Act of 1978, as amended; the Native American Graves Protection and

Repatriation Act of 1990; and Executive Order 13175 – Consultation and Coordination with Tribal Governments.

The NDEP BMRR will be a key State of Nevada agency for the overall mine and ore processing permitting and approval process. The Regulation Branch of the BMRR will issue the State of Nevada WPCP for the mine and ore processing/fluid management system operations in accordance with NAC 445A.350 through NAC 445A.447. The WPCP will stipulate requirements for the management and monitoring of the mine and ore processing operations, including the fluid management system, to ensure that they do not degrade waters of the state. The permit will also stipulate proposed procedures for temporary, seasonal and permanent closure of mine and ore processing operations. This WPCP is referred to as a "zero discharge permit" as it does not allow for the off-site discharge of mine water or the release/spills of process fluids and other pollutants from the fluid management containment system. This WPCP is issued under State of Nevada statute and regulations. It is not a "National Pollution Discharge Elimination System (NPDES) permit" issued under State of Nevada primacy in accordance with the requirements of Section 402 of the Federal Clean Water Act (CWA).

The Reclamation Branch of the BMRR will issue the State of Nevada reclamation permit for the project in accordance with NAC 519A, inclusive. The reclamation permit, with the associated PoO and Reclamation Plan, will stipulate procedures for the reclamation and closure of the mine and ore processing facilities. As mentioned above, the BLM required Mine Plan of Operations and Reclamation Plan (PoO) will be a joint plan submitted concurrently to both the BLM and the Reclamation Branch of the BMRR for review and approval under a Memorandum of Understanding (MOU) between these two agencies. The Reclamation Branch of the BMRR will review and approve the joint plan and the financial guarantee for reclamation activities and the long-term trust program to ensure that adequate funds are available to manage post-closure site activities into a pre-determined future period, and will issue the State of Nevada Reclamation Permit.

Other federal, state and Humboldt County agencies will issue additional permits, approvals or concurrences for various mine operations and activities in accordance with applicable federal, state and county ordinances, guidelines, regulations and laws, as appropriate.

Table 20.1 presents a list of the key permits or approvals, by regulatory agency, that will be required for the Project, based on the current mine and ore processing plans. Additional permits and approvals will be identified as necessary during the completion of the definitive feasibility study process.

Table 20.1: Summary List of Key Permits and Approvals by Agency

Agency	Permit or Approval
Federal Agencies	
Bureau of Land Management, Winnemucca District Office	Approval for mine and ore processing operations under 43 CFR 3809. Includes preparation PoO; posting of the financial guarantee for reclamation activities; and establishment of the long-term trust for funding of post-closure management activities. Completion of the NEPA environmental review and documentation process, either an environmental assessment (EA) or an environmental impact statement (EIS). Consultation process with the Nevada SHPO for assessing potential impacts to cultural resources under Section 106 of the Federal NHPA. Coordinate the Native American government-to-government consultation process under various federal laws and regulations.
Humboldt County Departments	
Regional Planning Department	Conditional use permit for mine and ore processing operations in accordance with the Humboldt County Zoning Code Chapter 17.44.040 Conditional Uses, M-3 Open Land Use District,
Building Department	Various permits for structures and facilities including building, electrical, plumbing and mechanical. These permits require standard design criteria for buildings including seismicity, wind, frost depth and snow loads. The permits are issued in accordance with County Ordinance 04-18-11.
State of Nevada Agencies	
NDEP BMRR, Regulation Branch	WPCP (zero discharge permit) for mine and ore processing operations/fluid management system issued in accordance with NAC 445A.350 through NAC 445A.447. The WPCP will stipulate requirements for the management and monitoring of the mine and ore processing operation, including the fluid management system, to ensure that they do not degrade waters of the state. The permit will also stipulate proposed procedures for temporary, seasonal and permanent closure of mine and ore processing operations.
NDEP BMRR, Reclamation Branch	Reclamation permit for the mine site issued in accordance with NAC 519A.010 through NAC 519A.415. The reclamation permit will stipulate procedures for the reclamation and closure of the quarry and ore processing facilities. The PoO is a joint plan approved by the BLM and BMRR. This includes review and approval of the financial guarantee for reclamation activities and the long-term trust fund for post-closure management activities.
NDEP Bureau of Air Quality (BAQ)	Air quality permit to construct the mine site. Air quality operating permit. These two permits are issued in accordance with NAC 445B.001 through NAC 445B.3689. They meet the requirements of the Federal Clean Air Act.
NDEP Bureau of Water Pollution Control (BWPC)	State groundwater permit for on-site sewage disposal. Issued in accordance with applicable sections of NAC 445A.070 through NAC 445A.925.
Division of Water Resources (NDWR)	Change the point of diversion and manner of use for water wells owned by WLC at its Quinn River Ranch to the mine site for use as production wells for process water.
Nevada Department of Wildlife (NDOW)	Industrial artificial pond permit for impoundments containing process solutions that can impact wildlife species. Issued under NAC 502.460 through NAC 502.495. Consultation and coordination on wildlife issues, including special status species, that are managed in accordance with the State wildlife laws and regulations contained in NRS 501.001 through NRS 501.395; and NAC 501.010 through NAC 501.340.

Agency	Permit or Approval
State Historic Preservation Office (SHPO)	Consultation process with the BLM for assessing potential impacts to cultural resources under Section 106 of the Federal NHPA. Coordinate the Native American government-to-government consultation process with the BLM under various federal laws and regulations.
Nevada State Fire Marshal Division	Permit to store hazardous materials. Issued in accordance with NRS 477.045 and NAC 477.325.
Nevada State Fire Marshal Division	Review of structural building plans in accordance with NRS 447; NAC 447.740 through NAC 447.750.
Nevada Division of Minerals	Mine registry forms. Submitted within 30 days after mine operations begin. In accordance with NRS 513.063 and NRS 513.073
Nevada Mine Safety and Training Division	Notification of the opening and closing of a mine. Notification submitted to the Division before a mine operations commence or close in accordance with NRS 512.160.

Experience indicates that the federal, state and local agency permitting and approval program could be completed within a 18-month time period using a focused agency consultation program, and if all required data needed to support an individual permit process has been collected. Figure 20.1 presents a preliminary time line for the completion of the Project permitting and approval activities.

20.4.1.2 Permitting and Approval Cost Estimate

Excluding the NEPA documentation process, the estimated cost to complete the multi-agency permitting and approval process based on the current mine and ore processing plans discussed in this preliminary feasibility study, is US$337,000. Table 20.2 summarizes the cost estimate by the individual permit or approval process. This cost estimate will be revised as needed based on the results of ongoing baseline studies, the process facility design, and the completion of the definitive feasibility study.

Table 20.2: Estimated Costs of Permits and Approvals by Agency

Agency Permit or Approval	Estimated Cost US$
BLM	
Approval under 43 CFR 3809. Includes preparation of the Mine Plan of Operations and Reclamation Plan	50,000
Section 106 consultation process for cultural resources with the Nevada SHPO (and implementation of required mitigation). Native American government-to-government consultation process	75,000
Humboldt County Planning Department	
Conditional Use Permit	5,000
Humboldt County Building Department	
Building Permits	5,000
NDEP BMRR Regulation Branch	
Water Pollution Control Permit	80,000
NDEP BMRR Reclamation Branch	
Reclamation Permit	25,000
NDEP Bureau of Air Quality	
Air Quality Permit to Construct	25,000
Air Quality Permit to Operate (Note: the type of operating permit is dependent on the results of the air quality monitoring and modeling program that will be initiated by WLC in 2012)	75,000
NDEP Bureau of Water Pollution Control	
Stormwater Permit to Construct	0
Stormwater Permit to Operate	0
NDEP Bureau of Water Pollution Control	
Groundwater Permit for On-Site Sewage Disposal	15,000
Nevada Division of Water Resources	
Change the point of diversion and manner of use for water wells owned by WLC at its Quinn River Ranch to the mine site for use as production wells for process water.	10,000
Nevada Department of Wildlife	
Industrial Artificial Pond Permit	25,000
Nevada SHPO	
Coordinate the NHPA Section 106 consultation process with the BLM. Coordinate the Native American consultation process with the BLM [(1)]	0
Nevada State Fire Marshal Division	
Permit to Store Hazardous Materials	5,000
Review of Structural Building Plans	5,000
Nevada Division of Minerals	
Mine Registry Forms	1,000
Nevada Mine Safety and Training Division	
Notification of the Opening and Closing of a Mine	1,000
Total Estimated Cost	$402,000

[1]This cost is included as part of the BLM Section 106 cultural resources and Native American consultation process

20.4.1.3 NEPA Environmental Documentation Process

Because the proposed project is on public lands and requires approval from the BLM, the proposed Project constitutes a federal action. The federal action will be assessed for potential environmental impacts as required by NEPA. NEPA is not a permit or approval action. NEPA is a "law of disclosure" which analyzes and discloses to the public the potential impacts to the environment that could result from the proposed action and or alternatives; assesses the level of significance for each identified impact; and proposes mitigation measures if needed to reduce the potential impact from the selected proposed action to a less than significant level. The results of the NEPA analysis are used by the BLM as part of their 43 CFR 3809 decision making process.

The NEPA analysis program for the proposed Project would take the form of an Environmental Assessment (EA) or an Environmental Impact Statement (EIS). The EA process is less stringent than the more comprehensive EIS process in terms of public scoping and depth of analysis. In Nevada, the BLM normally requires an EIS program to analyze and assess a new mine project in accordance with NEPA. However, based on the size and scope of the proposed Project, the BLM could determine that an EA is the proper NEPA analysis. As lead agency, the BLM will determine the appropriate NEPA program to assess the proposed project upon formal review of the Draft Mine PoO and the potential impacts that could result from the implementation of the proposed action or alternatives.

Note that formal decisions approving mining projects located on BLM managed public lands in Nevada can be appealed by various organizations, including special interest groups or individuals. These appeals are based on determined inadequacies in the NEPA assessments. Under the less stringent EA process, the burden of proof to show that the environmental analysis was scoped and properly completed is placed on the BLM. If an appeal to an EA is ultimately upheld, then the BLM and the proponent, at minimum, could be required to revise portions of the EA. Depending on the appeal issue, the proponent could be allowed to move forward with construction and operation activities while the appeal process is completed. However, if an EA appeal is upheld, the BLM could be required to re-analyze the proposed action under an EIS, and a legal stay of operations could be required. This new EIS program could require several years and significant monies to complete, and delay the implementation of the proposed action.

Under an EIS program, the burden of proof is placed on the appellant to show that BLM did not adequately scope and complete the EIS process; which is normally a very challenging technical and legal activity to complete. An appeal to an EIS, should it be actively pursued, could simply require a re-assessment of a specific resource issue. Under an EIS appeal, the proponent could be allowed to move forward with construction and operation activities while the appeal, and possibly a resource re-assessment, is worked through; although a legal stay of operations could be required. Although an EIS is generally more time consuming, with a greater cost, ultimate time and cost savings could be realized if the Finding of No Significant Impact (FONSI) and Decision Record (DR) for an EA is appealed.

<u>NEPA Cost Estimate</u>

The cost to complete an EA program for a new mine project in Nevada could range between US$100,000 and US$200,000, or more. The cost to complete an EIS program for a new mine project in Nevada could range between US$800,000 to US$1,000,000 or more. Scoping and consultation meetings with the BLM's Winnemucca District Office would assist in determining the detailed cost estimate to complete the selected NEPA process.

NEPA Time Frames

This section describes estimated time frames to complete an EA or an EIS process under NEPA. Note that coordination with the BLM is required to determine an approved time frame estimate for the completion of the selected NEPA process.

Under 43 CFR 3809, the BLM has 30 days to review and accept a Draft Mine PoO as technically complete after it is submitted by WLC. Note that BLM generally assumes these are 30 working days, not calendar days. Once the BLM determines that the Draft PoO is technically complete, then the process to determine the appropriate NEPA action will commence. This would include selecting the ID Team which is composed of the specific BLM specialists who will work on the NEPA program and representatives of WLC. Experience indicates this activity could be completed within 30 working days.

Once the ID Team is selected and the appropriate NEPA action is determined, the process of selecting the NEPA contractor can start. Under an EA process, the BLM will usually allow the project proponent to select its own third-party NEPA contractor without going through a formal bid process. The BLM could review the qualifications of the contractor, but that generally is very quick, especially if they are familiar with the firm. Experience indicates that this selection process can be completed within 15 to 30 working days.

Under the EA process, pre-project public scoping is not required. Once the ID Team is in place, key EA issues will be discussed during a formal ID Team meeting; baseline study needs will be outlined and a data gap analysis will be completed of existing studies that have been completed or are ongoing; perhaps a draft work plan will be prepared for guidance purposes; a draft schedule will be prepared; and the process will be initiated. An EA could potentially be completed in 18 months, although the time frame could be longer. This 18 month period assumes that the BLM would accept the results of the completed and ongoing baseline studies. Additional time could be required if BLM requires the completion of additional baseline studies or resource data collection.

Under an EIS process, selection of the third-party NEPA contractor can take three months or more. This would include preparation of the draft and final EIS bid document/data adequacy standards; a standard 30 day bid preparation process; 15 to 30 days to review bids and select the contractor; and preparation of the formal contract and other paperwork for the EIS process.

For an EIS, the proponent should plan on a minimum of 18 months to complete the process once it commences. This would be the minimum time to complete the pre-EIS public scoping, address comments, prepare the work plan, complete the analysis, complete administrative draft documents and agency reviews, prepare the draft final document for public notice and review, address comments and prepare the Final EIS. Note that an EIS process could easily require 24 months or more to complete.

Permitting Strategy

Suggestions for a permitting strategy to develop positive community support and streamline final project approval include the following:

- Maintain regular consultation activities with the appropriate federal, state and local regulatory agencies. These agencies include the Winnemucca BLM, the various NDEP Bureaus, the appropriate Humboldt County departments and other federal and state

agencies as deemed appropriate. These meetings will keep the regulatory agencies up to date on project activities and allow them to provide insight on issues as appropriate.

- Continue to hold regular public information (town hall meetings) with local communities, including the City of Winnemucca, and the community of Orovada. These meetings are beneficial in developing and maintaining community support by highlighting positive social and economic aspects of the project. They also provide a forum for individuals to address concerns up front, which will allow WLC to address these issues at the earliest possible opportunity and avoid potential "eleventh hour delays".

- Continue to hold regular meetings with appointed and elected local, state and federal officials. These types of meetings provide WLC with the opportunity to keep key officials updated on Project issues, and set the stage for political assistance, if needed, at the local, state and national levels. Humboldt County and the appropriate State of Nevada officials can be contacted on a regular basis. Nevada's Congressional delegation, including Senators Harry Reid and Dean Heller, and Congressman Mark Amodei, all maintain offices in Reno, Nevada that are staffed on a daily basis.

20.5 Social or Community Impacts

The construction and operation of the Kings Valley Lithium mine and ore processing operations should not impact local or regional social or community infrastructure. It is expected that employees will come from the surrounding area which already has established social and community infrastructure including housing, retail and commercial facilities such as stores and restaurants; and public service infrastructure including schools, medical and public safety departments and fire and police/sheriff departments.

Based on the projected mine life, the number of potential hourly and salaried positions, and the projected salary ranges, Project operations would have a long-term positive impact in regards to direct, indirect and induced local and regional economics. Project operations are considered basic industries, as they would contribute dollars to the local and regional area in terms of operation purchases and employee hiring (direct impacts). Additional positive economic links would result from the purchases of goods and services from the local service sectors including businesses such as restaurants, gas stations, hotels, and other retail businesses; as earnings increase in these businesses; they hire additional people and buy more inputs from other businesses (indirect impacts). Both the direct and indirect impacts would change the flow of dollars to the local households, which would alter their consumption accordingly. The effect of this change in local household consumption upon businesses in a regional economic area is referred to as an induced impact. The positive change in the local and regional economic area works its way throughout the entire local economy (Harris and Dobra, 2009).

An additional and positive economic benefit would be the creation of several hundred short-term positions for construction activities. To the greatest extent possible, the majority of these construction positions would be hired from the regional labor pool, along with a to-be-determined percentage of outside workers who are brought in. The construction period could last for approximately 12 months (estimated Project construction time frame), and would also contribute to positive direct, indirect and induced economic benefits to the local and regional communities (Harris and Dobra, 2009).

20.6 MINE RECLAMATION AND CLOSURE

Reclamation and closure of the mine and ore processing operations will be completed in accordance with the joint BLM and BMRR approved Mine Plan of Operations and Reclamation Plan, and other approved closure plans prepared as part of the State of Nevada WPCP that will be issued by the NDEP/BMRR. These plans will be updated on a regular basis, in consultation with the BLM and the NDEP/BMRR, to ensure that they remain up to date in terms of the following:

- The latest federal and state regulatory requirements for reclamation and closure as contained in 43 CFR 3809; NAC 519A; and NAC 445A.350 through NAC 445A.447
- The latest and appropriate reclamation and closure technologies and procedures
- Ensuring that the posted reclamation bond and post-closure long-term trust fund remain sufficient to reclaim and close the mine site, and fund post closure monitoring activities, respectively

Project facilities will be reclaimed using standard reclamation techniques and procedures as summarized in the following list:

- During construction activities, suitable and available growth medium material will be stripped from sites scheduled for surface disturbance and stockpiled for future reclamation activities.
- WLC will conduct concurrent reclamation of sites no longer required for mine and ore processing operation activities.
- Buildings and other structural facilities including power lines and substations will be dismantled and removed off site to appropriate storage or disposal facilities.
- Process plant components will be removed off site and transported to approved storage or disposal facilities.
- Concrete foundations will be broken up and buried on site, or removed off site to an approved disposal area.
- The TSF runoff collection pond and the barren purge containment pond will be reclaimed according to the following procedures: (1) remaining solutions will be allowed to evaporate; (2) pond liners will be ripped to allow percolation of meteoric waters and folded in; (3) the ponds will be backfilled with stockpiled soil material to a sufficient elevation above the original ground surface; (4) the final backfilled surface will be graded to provide positive surface drainage and revegetated with an approved reclamation seed mix.
- The overall 3.0H:1.0V TSF slopes will be capped with stockpiled growth medium and revegetated with an approved reclamation seed mix.
- The overall 3.0H:1.0V waste rock storage facility slopes will be regraded as needed, capped with stockpiled growth medium and revegetated with an approved reclamation seed mix.

- The backfilled open pit will be left in its final configuration. The final pit slopes will be designed for long-term stability. The backfilled portion of the pit will be allowed to revegetate naturally without the use of a reclamation seed mix. To inhibit public access, pit access roads will be reclaimed, and the pit will be enclosed with a rock berm or fence. Warning signs will be placed at suitable locations along the rock berm or fence to notify the public of a hazardous condition.

- Roads not needed for long term monitoring access will be regraded and revegetated using an approved reclamation seed mix.

- The surface water diversion ditches will remain in place to divert surface water flows around the reclaimed mine site area.

The majority of the Draft Mine Plan of Operations and Reclamation Plan has been prepared and will be further refined based on the results of the ongoing process engineering design study. This draft plan will include the initial reclamation bond cost estimate for the project.

The Nevada BLM and the State of Nevada have initiated a long-term trust fund program as part of the federal and state permitting program to provide for the funding of site maintenance and monitoring activities following the completion of final reclamation and closure activities. The financial method for securing and placement of the trust fund, the trust fund cost and the determined long-term duration varies by project. Consultation with the BLM and the NDEP BMRR would determine the specifics of the long-term trust fund program.

20.7 Conclusions and Recommendations

A multi-agency permitting and approval process will be completed by WLC to construct, operate and close the Project in accordance with applicable federal, state and local regulations. This program will include the acquisition of numerous permits and approvals from various regulatory agencies. To streamline the time required to complete the overall permitting and approval process it is recommended that representatives of WLC hold regular meetings with personnel from the various regulatory agencies including the Winnemucca BLM, the NDEP BMRR, Humboldt County, and other federal, state and local agencies as appropriate to provide updates on Project activities. These meetings, which can be held as individual meetings or a joint meeting, will provide an open forum to discuss overall progress and any issues or concerns that may arise during development activities. It is also recommended that WLC hold regular public information meetings in the regional area including the City of Winnemucca and the community of Orovada to provide regular Project updates and to continue developing positive public support. Finally, WLC should maintain regular contact with local, state and federal elected officials. The value in keeping elected officials updated on Project activities cannot be stressed enough in terms of potential assistance and support that might be needed to assist the agencies and WLC in completing the overall permitting and approval process for the Project.

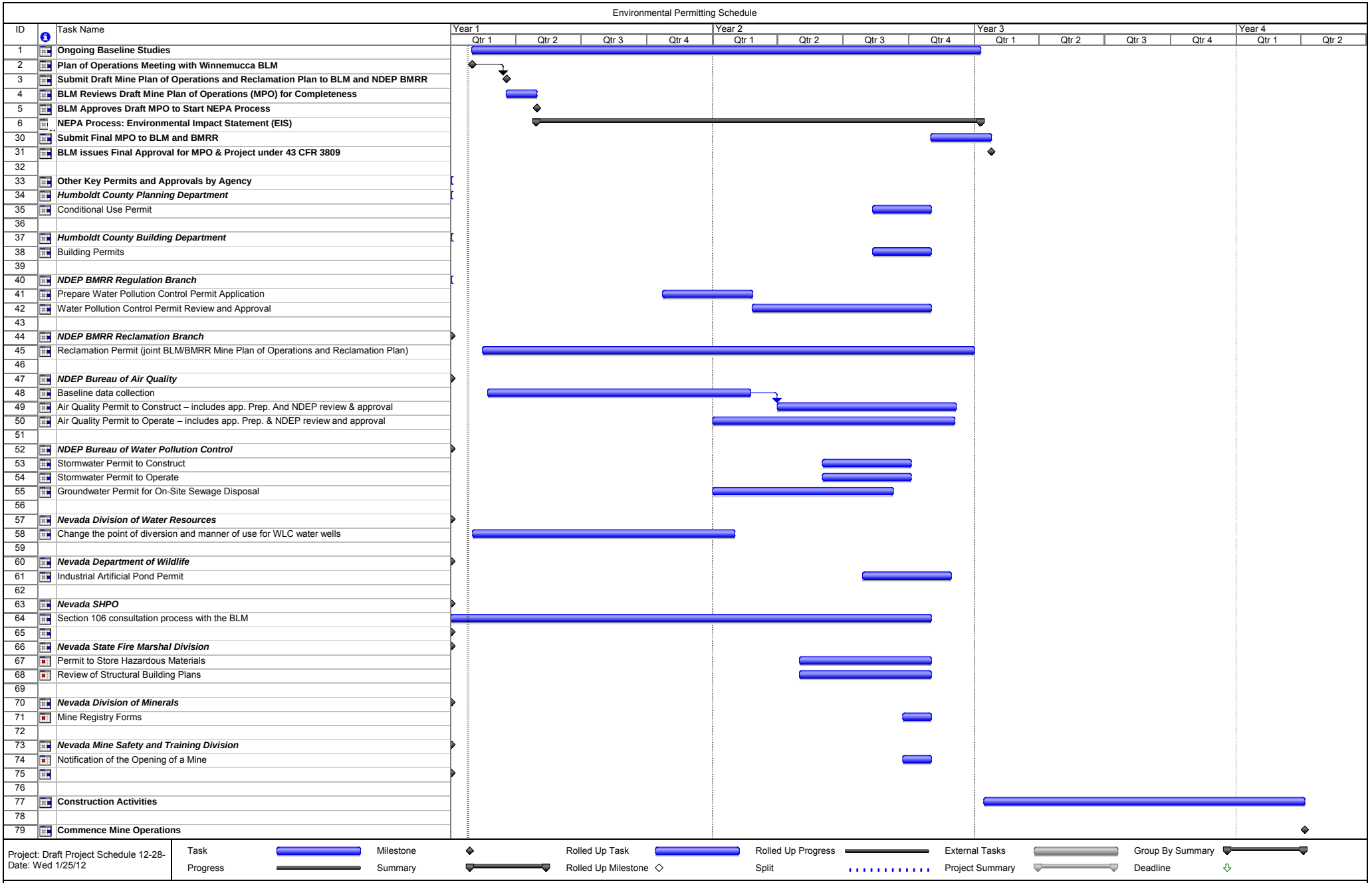


Figure 20.1. Environmental Permitting Schedule

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 STAGE I

21.1.1 Cost Estimating Criteria

All capital and operating costs are expressed in constant third quarter 2011 US dollars (US$).

Capital costs were estimated with an accuracy of ±25% in line with requirements of a preliminary feasibility study under CIM guidelines. Capital costs were divided into three primary categories:

- Mine Equipment Capital
- Process Equipment Capital
- Tailings Facility and Water Management Capital

Mine equipment capital was determined based upon the material movements required to meet the mine plans. The basis of capital requirements for mine equipment was gathered from vendor budgetary quotes or from *Mine and Mill Equipment Costs: An Estimator's Guide* published by InfoMine USA (InfoMine). All mine equipment capital cost estimates obtained are in 2011 US dollars.

The process equipment capital estimate was compiled and calculated by Tetra Tech Wardrop Engineering (Tetra Tech Wardrop). Pricing input was provided by Tetra Tech Wardrop, K-UTEC and WLC. The process capital estimate was prepared in accordance with the Association for Advancement of Cost Engineering (AACE). All budgetary quotes for process capital items were obtained in the third quarter of 2011. The process capital estimate is based upon the following:

- Budget quotations for the dry process equipment and systems including crushers, Raymond mills, ball mills, granulation, table feeders, dryer, coal pulverization, calciner, and cooler. The Tetra Tech Wardrop in-house database was used for the balance of the equipment.
- Budgetary quotations for the wet process equipment and systems were obtained by K-UTEC.
- Preliminary material take-offs for civil work were costed by Tetra Tech Wardrop.
- Electrical, instrumentation, piping and HVAC were expressed as a percentage for process equipment cost.
- Platework and structural steel was estimated using similar recent projects as benchmarks by Tetra Tech Wardrop.
- Major electrical equipment costs were estimated by Tetra Tech Wardrop.

All equipment and material costs were based on FCA (free carrier) manufacturer plant and were exclusive of spare parts, taxes, duties, freight and packaging. These costs, if appropriate, were covered in the indirect section of the estimate.

Tailings facility and water management capital cost includes materials and construction for the barren purge containment pond, dry stack pad, surface water diversions and site access roads. These costs were estimated from the designed volumes and materials required using the RS Means Heavy Construction Cost Data handbook and vendor estimates obtained by Tetra Tech for projects in Nevada.

21.1.1.1 Exclusions

The following items were excluded from the capital estimate:

- Force Majeure
- Taxes
- Overtime
- Costs Outside Battery Limits
- Interest
- Sunk Costs

Technical Reference C, Capital Basis of Estimate, provides a detailed report regarding the basis of estimate used and associated methodology.

21.1.2 Capital Cost Estimate

Tetra Tech estimated the required capital expenditures for both Case 1 and Case 2 mining scenarios on a 100% owner equity basis. Capital costs, where applicable, include expenditures for the following items:

- Mine mobile equipment
- Mine development and tailings handling
- Process site development
- Site utilities
- Process equipment
- Buildings and structures
- Process mobile equipment
- Mine closure costs
- Indirect costs
- Owner's costs
- Contingency

Mobile equipment capital includes purchases of new equipment, replacement of existing mobile equipment after the end of useful life and the major rebuilds of large equipment such as excavators, haul trucks, dozers and the surface miner.

Indirect costs include contractor indirect costs, initial fills (i.e., diesel, reagents, oil, etc.), spares, commissioning and start-up, freight and EPCM.

Owner's costs were provided to Tetra Tech by WLC and include planned expenses for permitting, insurance and WLC staffing.

Contingency costs are comprised of a 15 percent provision for all capital expenditures except for mobile equipment and the K-UTEC equipment. The K-UTEC equipment contingency was applied at 10 percent. Contingency for equipment was included in the budgetary quotes used in the estimate.

Capital costs include initial capital required for project development and sustaining capital required for continued production.

21.1.2.1 Case 1 Capital Estimate

The Case 1 capital estimate results in a total Life of Mine capital cost of US$262.7 million. Of this, US$237.1 million represents initial capital with the remaining US$25.6 million in sustaining capital. Sustaining capital consists primarily of surface mine equipment, expansions of the dry stack tailings, surface water management and anticipated mine closure activities. Sustaining capital contingency was included in each line item rather than a separate estimate as was used for the initial capital. Table 21.1, Case 1 Capital Estimate, summarizes the initial, sustaining and Life of Mine capital required.

Table 21.1: Case 1 Capital Estimate (US$000s)

Description	Initial	Sustaining	LOM
Mine Mobile Equipment	$7,749	$10,497	$18,246
Mine Development & Tailings Handling	$17,485	$4,653	$22,139
Mine Closure	$0	$5,189	$5,189
Total Mine Capital	**$25,235**	**$20,340**	**$45,574**
Process Site Development	$1,791	$0	$1,791
Site Utilities	$10,588	$0	$10,588
Process Equipment	$115,128	$2,000	$117,128
Buildings and Structures	$6,348	$0	$6,348
Process Mobile Equipment	$2,354	$3,255	$5,609
Total Process Capital	**$136,209**	**$5,255**	**$141,464**
Indirect Costs	$35,961	$0	$35,961
Owner's Costs	$17,784	$0	$17,784
Contingency	$21,894	$0	$21,894
Total Capital	**$237,083**	**$25,595**	**$262,678**

21.1.2.2 Case 2 Capital Estimate

Capital requirements for Case 2 include a factored estimate based upon the Case 1 capital requirements for the process site development, site utilities, process equipment, buildings and structures and indirect costs. Tetra Tech factored the capital estimate from Case 1 by one of three ways. The capital cost not altered (i.e., fencing, reagent unloading, water systems, etc.), doubled (i.e., process equipment, substations, fuel storage, etc.) or factored (i.e., communications, lime storage, truckshop, etc.).

Based on Tetra Tech's experience it used a seven-tenths factor as seen in the following equation (Lowrie, 2002) for factored capital estimates:

*Capital Case 2 = Capital Case 1 * (Rate Case 2/Rate Case 1)$^{0.7}$*

Where:

Capital Case 2 = Factored Capital Cost at Rate Capital Case 2

Capital Case 1 = Original Capital Cost at Case 1.

Rate Case 1 = 2,100 tpd

Rate Case 2 = 4,200tpd

The Case 2 capital estimate results in a total life of mine capital cost of US$449.5 million. Of this, US$247.9 million represents initial capital with the remaining US$201.6 million in sustaining capital. Sustaining capital consists primarily of surface mine equipment, expansions of the dry stack tailings, surface water management, process plant expansion and anticipated mine closure activities. Table 21.2, Case 2 Capital Estimate, summarizes the initial, sustaining and Life of Mine capital required.

Table 21.2: Case 2 Capital Estimate (US$000s)

Description	Initial	Sustaining	LOM
Mine Mobile Equipment	$7,818	$21,114	$28,932
Mine Development & Tailings Handling	$26,227	$18,424	$44,651
Mine Closure	$0	$8,519	$8,519
Total Mine Capital	**$34,046**	**$48,057**	**$82,103**
Process Site Development	$1,791	$144	$1,935
Site Utilities	$10,588	$6,461	$17,050
Process Equipment	$115,128	$105,556	$220,684
Buildings and Structures	$6,348	$1,701	$8,049
Process Mobile Equipment	$4,394	$3,510	$7,904
Total Process Capital	**$138,249**	**$117,372**	**$255,621**
Indirect Costs	$35,961	$22,458	$58,419
Owner's Costs	$17,784	$0	$17,784
Contingency	$21,894	$13,673	$35,567
Total Capital	**$247,935**	**$201,560**	**$449,495**

21.1.3 Operating Cost Estimate

Tetra Tech estimated the mine and process operating cost requirements associated with both Case 1 and Case 2 operations.

Total operating costs include expenditures for the mining, processing and general and administrative (G&A) functions. The Case 1 operating costs average US$72.25 per tonne of ore mined or US$3,816 per tonne of LCE. Pre-production operating costs have been capitalized and therefore deducted from the total operating costs in the year in which the costs were capitalized. Table 21.3, Case 1 LOM Operating Cost Summary, details the yearly total operating costs for Case 1.

The Case 2 operating costs average US$65.01 per tonne of ore mined or US$3,472 per tonne of LCE. Pre-production operating costs and operating costs associated with waste stripping in support of the production increase have been capitalized and therefore deducted from the total operating costs in the year in which the costs were capitalized. Table 21.4, Case 2 Estimated Total Operating Costs, details the yearly total operating costs for Case 2.

Table 21.3: Case 1 LOM Operating Cost Summary

Description	Units	LOM Totals
Waste	dry tonnes (000s)	40,122
Ore	dry tonnes (000s)	13,949
Total Material	dry tonnes (000s)	54,071
Strip Ratio	waste:ore	2.88
Lithium Carbonate Production	tonnes (000s)	264
Mine Operating Costs	US$ (000s)	$118,123
Process Operating Costs	US$ (000s)	$853,452
General and Administrative Costs	US$ (000s)	$44,411
Capitalized Operating Costs	US$ (000s)	($8,196)
Total Operating Costs	US$ (000s)	**$1,007,789**
Total Operating Costs per Ore Tonne	US$/tonne	**$72.25**
Total Operating Costs per LCE Tonne	US$/tonne	**$3,816**

Note: mine operating costs in pre-production time frame are capitalized.

Table 21.4: Case 2 LOM Operating Cost Summary

Description	Units	LOM Totals
Waste	dry tonnes (000s)	97,241
Ore	dry tonnes (000s)	25,495
Total Material	dry tonnes (000s)	122,736
Strip Ratio	waste:ore	3.81
Lithium Carbonate Production	tonnes (000s)	477
Mine Operating Costs	US$ (000s)	$215,648
Process Operating Costs	US$ (000s)	$1,420,471
General and Administrative Costs	US$ (000s)	$44,908
Capitalized Operating Costs	US$ (000s)	($23,553)
Total Operating Costs	US$ (000s)	**$1,657,473**
Total Operating Costs per Ore Tonne	US$/tonne	**$65.01**
Total Operating Costs per LCE Tonne	US$/tonne	**$3,472**

Note: mine operating costs in pre-production time frame and pit expansion for production increase are capitalized.

Mine operating costs were derived from first principles and represent expenses incurred due to scheduled production volumes, derived equipment productivities and haul distances. Operating costs include the following items:

- Fuel (off-road diesel and on-road gasoline)
- Electric power
- Lubricants (*i.e.*, engine oil, transmission fluid and hydraulic fluid)
- Rubber tires
- General repair parts and wear parts (*i.e.*, rollers, bushings, filters, and cutting teeth)
- Labor

A diesel fuel price of US$3.00 per gallon was used to calculate mine operating costs.

An electricity cost of US$0.064 per kilowatt hour (kWh) was used. The electricity price was derived from quoted rate schedules applicable to the Project and includes a base electricity cost of US$0.063 per kWh and a demand charge of US$5.70 per kWh of demand. WLC provided Tetra Tech with the electricity rate schedule as per Harney Electric Cooperative Inc.

Lubricants, rubber tires, and general repair parts and wear parts were quoted from the InfoMine handbook.

Labor rates were provided by WLC and were based on current local market non-union labor rates and adjusted for customary benefit packages. For Case 1, the manpower required for operation of the mine is 37 hourly and 9 salaried employees during a typical year. For Case 2, the manpower required for operation of the mine is 64 hourly and 14 salaried employees during a typical year.

Process operating costs include the following major categories:

- Dry process:
  - Anhydrite
  - Dolomite
  - Lime
  - Propane
  - Coal
  - Grinding steel
- Wet process:
  - Soda ash
  - Sulfuric acid
  - Resin
  - Coal

- General process:
  - Process mobile equipment
  - Water
  - Electrical power
  - Manpower
  - Maintenance and miscellaneous operating expenses

Process operating costs include: cost of reagents, utilities, supplies, maintenance parts, general wear items, operating labor and supervision. Reagent costs were obtained by WLC from local and national suppliers. Cost of maintenance and operating supplies were factored from total process operating costs. Electrical power charges and water costs were provided by WLC. Labor rates were provided by WLC and were based on current local market non-union labor rates and adjusted for customary benefit packages. Major equipment rebuilds have been capitalized rather than expensed.

Tetra Tech and WLC developed a manpower schedule for both the mine and process operations. Hourly and salary wages were applied to the appropriate personnel based upon the position's job description and number of applicable employees. To the base rate, a payroll burden of 36% was applied to all wages. The payroll burden, as provided by WLC, includes the following items:

- Federal and tax payroll taxes, *i.e.*, Medicare, social security and unemployment
- Life, medical, dental, vision and accidental death and dismemberment insurance
- 401k plan contributions
- Paid time-off
- Disability income
- Paid holidays
- Shift differentials
- Bonuses

For Case 1, the manpower required for operation of the process plant is 70 hourly and 6 salaried employees during a typical year. For Case 2, the manpower required for operation of the process plant is 86 hourly and 6 salaried employees during a typical year.

21.1.3.1 Case 1 Operating Costs

Table 21.5, Case 1 Mine Operating Costs, summarize the annual mine operating costs for Case 1. The average life of mine operating cost is estimated to be US$2.11 per tonne moved or US$432 per tonne of LCE. Note that mine operating costs during the pre-production period have been capitalized in the cash flow estimated by Tetra Tech.

Process operating costs for Case 1 average US$61.03 per tonne of ore processed or US$3,223 per tonne of LCE. Process operating costs incurred during the pre-production period include general staff training and site orientation. Note that process operating costs during pre-production have been capitalized and

deducted from the total process operating cost in the year in which the cost is capitalized. Table 21.6, Case 1 Process Operating Costs, summarizes the yearly process operating costs for Case 1.

Process related unit consumptions were developed based upon heat/mass/energy balances and applied on an applicable per tonne product (LCE) or per tonne ore basis. The primary unit costs are presented in Table 21.7, Case 1 Process Unit Costs.

Table 21.5: Case 1 LOM Mine Operating Costs

Description	Units	LOM Totals
Waste	dry tonnes (000s)	40,122
Ore	dry tonnes (000s)	13,949
Total Material	dry tonnes (000s)	54,071
Strip Ratio	waste:ore (000s)	2.88
Mine Equipment Costs	US$ (000s)	$45,267
Mine Hourly Labor Costs	US$ (000s)	$52,298
Mine Salary Labor Costs	US$ (000s)	$20,559
Capitalized Mine Operating Costs	US$ (000s)	($3,955)
Total Mine Operating Costs	US$ (000s)	**$114,168**
Total Mine Operating Costs per Waste Tonnes	US$/tonne	**$2.85**
Total Mine Operating Costs per Ore Tonnes	US$/tonne	**$8.18**
Total Mine Operating Costs per Total Tonnes	US$/tonne	**$2.11**

Note: mine operating costs in pre-production time frame are capitalized.

Table 21.6: Case 1 LOM Process Operating Costs

Description	Units	LOM Totals
Ore Mined	dry tonnes (000s)	13,949
Lithium Carbonate Product	tonnes (000s)	264
Lithium Carbonate Process Cost	US$ (000s)	$714,237
Potassium Sulfate Process Cost	US$ (000s)	$92,678
Sodium Sulfate Process Cost	US$ (000s)	$46,537
Capitalized Process Operating Cost	US$ (000s)	($2,152)
Total Processing Cost	**US$ (000s)**	**$851,301**
Processing Cost per Ore Tonne	**US$/tonne**	**$61.03**
Processing Cost per Lithium Carbonate Tonne	**US$/tonne**	**$3,223**

Note: Process Operating Costs in Pre-Production time frame are capitalized.

Table 21.7: Case 1 Process Unit Costs

Description	Cost per Unit Tonne (US$/tonne)	Cost per Year (US$000/year)	Cost per Ore Tonne (US$/tonne)	Cost per LCE Tonne (US$/tonne)
Dry Process Reagents				
Anhydrite	$31.10	$7,509	$10.89	$575.25
Dolomite	$20.39	$4,219	$6.12	$323.20
Lime	$184.63	$1,779	$2.58	$136.29
Propane	$1,188.28	$30	$0.04	$2.30
Coal - Calciner	$80.32	$6,040	$8.76	$462.70
Grinding Steel	$1,018.50	$280	$0.41	$21.48
Wet Process Reagents				
Soda Ash	$200.70	$4,157	$6.03	$318.44
Sulfuric Acid	$462.97	$7	$0.01	$0.57
Resin	$9,936.00	$261	$0.38	$20.00
Coal - Boiler	$80.32	$2,638	$3.82	$202.06
General Process				
Water	N/A	$91	$0.13	$6.97
Electrical Power	N/A	$6,585	$9.55	$504.44
Manpower	N/A	$5,515	$7.99	$422.52
Maintenance Supplies	N/A	$1,564	$2.27	$119.85
Misc. Operating Expenses	N/A	$391	$0.57	$29.96
Process Mobile Equipment	N/A	$1,023	$1.48	$78.36
Total Processing Cost		**$42,089**	**$61.01**	$3,224.38

An estimate of maintenance supplies was made by adding four percent to operating costs. Miscellaneous operating supplies (fuel, lubricants, lab supplies, etc.) were estimated by applying a one percent factor to operating costs.

21.1.3.2 Case 2 Operating Costs

Table 21.8, Case 2 Mine Operating Costs, summarize the annual mine operating costs for Case 2. The average life of mine operating cost is estimated to be US$1.61 per tonne moved or US$413 per tonne of LCE. Note that mine operating costs during the pre-production period have been capitalized in the cash flow estimated by Tetra Tech. Tetra Tech has also capitalized a portion of the mine operating costs in Years 1 through 3 based upon the additional waste stripping required for the expected production increase in Year 4.

Case 2 process operating costs include necessary reagents, power, water, equipment, labor and miscellaneous operating costs associated with the processing of ore and related facilities. Process operating costs for Case 2 average US$55.63 per tonne of ore processed or US$2,971 per tonne of LCE. Process operating costs incurred during the pre-production period include general staff training and site orientation. Note that process operating costs during pre-production have been capitalized and deducted from the total process operating cost in the year in which the cost is capitalized. Table 21.9, Case 2 Process Operating Costs, summarizes the yearly process operating costs for Case 2.

Process related unit consumptions were developed based upon heat/mass/energy balances and applied on an applicable per tonne product or per tonne ore basis. The primary unit costs are presented in Table 21.10, Case 2 Process Unit Costs.

Table 21.8: Case 2 LOM Mine Operating Costs (000s)

Description	Units	LOM Totals
Waste	dry tonnes (000s)	97,241
Ore	dry tonnes (000s)	25,495
Total Material	dry tonnes (000s)	122,736
Strip Ratio	waste:ore	3.81
Mine Equipment Costs	US$ (000s)	$97,551
Mine Hourly Labor Costs	US$ (000s)	$86,673
Mine Salary Labor Costs	US$ (000s)	$31,424
Capitalized Mine Operating Costs	US$ (000s)	($18,292)
Total Mine Operating Costs	US$ (000s)	**$197,356**
Total Mine Operatng Costs per Waste Tonnes	US$/tonne	**$2.03**
Total Mine Operatng Costs per Ore Tonnes	US$/tonne	**$7.74**
Total Mine Operatng Costs per Total Tonnes	US$/tonne	**$1.61**

Note: mine operating costs in pre-production time frame and pit expansion for production increase are capitalized.

Table 21.9: Case 2 LOM Process Operating Costs (000s)

Description	Units	LOM Totals
Ore Mined	dry tonnes (000s)	25,495
Lithium Carbonate Product	tonnes (000s)	477
Lithium Carbonate Process Cost	US$ (000s)	$1,199,565
Potassium Sulfate Process Cost	US$ (000s)	$152,620
Sodium Sulfate Process Cost	US$ (000s)	$68,285
Capitalized Process Operating Cost	US$ (000s)	($2,152)
Total Processing Cost	US$ (000s)	**$1,418,319**
Processing Cost per Ore Tonne	US$/tonne	**$55.63**
Processing Cost per Lithium Carbonate Tonne	US$/tonne	**$2,971**

Table 21.10: Case 2 Process Unit Costs

		2,100 tpd Production Rate			4,200 tpd Production Rate		
Description	Cost per Unit Tonne (US$/tonne)	Cost per Year (US$000/year)	Cost per Ore Tonne (US$/tonne)	Cost per LCE Tonne (US$/tonne)	Cost per Year (US$000/year)	Cost per Ore Tonne (US$/tonne)	Cost per LCE Tonne (US$/tonne)
Dry Process Reagents							
Anhydrite	$31.10	$7,509	$10.89	$578.98	$15,018	$10.89	$582.31
Dolomite	$20.39	$4,219	$6.12	$325.29	$8,438	$6.12	$327.16
Lime	$184.63	$1,779	$2.58	$137.18	$3,558	$2.58	$137.96
Propane	$1,188.28	$30	$0.04	$2.31	$60	$0.04	$2.33
Coal - Calciner	$80.32	$6,040	$8.76	$465.70	$12,080	$8.76	$468.38
Grinding Steel	$1,018.50	$280	$0.41	$21.62	$561	$0.41	$21.74
Wet Process Reagents							
Soda Ash	$200.70	$4,157	$6.03	$320.50	$8,313	$6.03	$322.34
Sulfuric Acid	$462.97	$7	$0.01	$0.57	$15	$0.01	$0.57
Resin	$9,936.00	$261	$0.38	$20.13	$522	$0.38	$20.25
Coal - Boiler	$80.32	$2,638	$3.82	$203.37	$5,275	$3.82	$204.54
General Process							
Water		$91	$0.13	$7.01	$182	$0.13	$7.05
Electrical Power	find $/ tonne vlaue	$6,585	$9.55	$507.71	$10,697	$7.75	$414.76
Manpower	N/A	$5,515	$7.99	$425.26	$6,575	$4.77	$254.93
Maintenance Supplies	N/A	$1,564	$2.27	$120.63	$2,852	$2.07	$110.57
Misc. Operating Expenses	N/A	$391	$0.57	$30.16	$713	$0.52	$27.64
Process Mobile Equipment	N/A	$1,112	$1.61	$85.76	$1,139	$0.83	$44.15
Total Processing Cost		**$42,179**	**$61.14**	**$3,252.18**	**$75,997**	**$55.08**	**$2,946.68**

An estimate of maintenance supplies was made by adding four percent to operating costs. Miscellaneous operating supplies (fuel, lubricants, lab supplies, etc.) were estimated by applying a one percent factor to operating costs.

21.1.4 General and Administrative

General and Administrative expenses were estimated for the project by Tetra Tech and include salaried personnel required for supervision and administrative duties applicable to both the process plant and mine site. These costs include general managers, human resources, accounting, environmental staff, safety and security, dispatch personnel and administrative assistants. General and Administrative costs average US$3.03 per tonne of ore for Case 1 and US$1.64 per tonne of ore for Case 2. Note that General and Administrative costs incurred during the pre-production time period have been capitalized. The capitalized costs have been deducted from the operating costs in the year in which the costs were capitalized.

21.1.5 Risks

Capital estimates are directly tied to the price of steel. Fluctuations in the supply and demand of steel may affect the prices quoted.

Changes caused by the ongoing hydrology studies may have an impact on capital and operating costs related to pit slope stability, waste dump placement and construction and other factors.

There is currently no allowance for drilling and blasting of the waste material. Due to the presence of basalt, although minor, drilling and blasting may be required for small volumes of material. This may

increase the operating costs due to the requirement of a contract drill and blast operation; however, this expense will most likely be minor given the small amount of basalt present within the area.

21.2 STAGE II

The Stage II Resource Estimate report is preliminary in nature and addresses only the resource estimate. As such, this section is not applicable to the Stage II area.

22.0 ECONOMIC ANALYSIS

22.1 STAGE I

Tetra Tech performed a pre-tax and after-tax cash flow analysis using the revenues, operating costs and capital costs laid out for the project for both Case 1 and Case 2 mining scenarios. All costs are stated in constant Q3 2011 US dollars.

22.1.1 Principal Economic Parameters and Assumptions

The cash flow analyses were performed using the following key parameters and assumptions:

- Product revenues
  - Lithium carbonate -US$6,000 per tonne FOB project site
  - Potassium sulfate –US$600 per tonne FOB project site
  - Sodium sulfate - US$75 per tonne FOB project site
- Process recoveries
  - Lithium – 87.2%
  - Potassium – 77.7%
  - Sodium – 82.7%
- 100% Owner's Equity
- 8% discount rate
- 35% federal income tax rate
- 20% of operating costs for working capital
- Tax losses are carried forward
- Three-year pre-production period
- No royalties; Subsequent to the Stage I PFS analysis, the Project has incurred royalty obligations that were not included in the Stage I economic estimate. The economic analysis of the Project has not been updated to account for these royalties.
- Mine operation runs 350 days per year process operation runs 365 days per year at 90% availability
- Case 1 nominal production rate of 13,000 tpy of LCE
- Case 2 nominal production rate of 13,000 tpy of LCE in Years 1, 2 and 3 increased to 26,000 tpy of LCE in Year 4 through end of mine life
- Sunk capital and equipment salvage values have not been included

22.1.2 Taxes

Included in the cash flow analysis are applicable federal and state taxes.

Nevada State Minerals Tax is an ad valorem property tax assessed on minerals mined or produced in Nevada when they are sold or removed from the state. The tax is applied at a rate of 5% for the project and applies to gross proceeds minus allowable deductions. Allowable deductions include operating costs, depreciation, and royalties.

A federal income tax rate of 35% was applied to net revenue less operating costs, depreciation, depletion, Nevada State Minerals Tax and loss carry forward. An approximate rate of 35% was used to allow for differences in tax payments due to Alternative Minimum Tax and adjustments to the corporate tax rate based upon revenue.

22.1.3 Cash Flow Analysis

22.1.3.1 Case 1 Cash Flow Analysis

A cash flow analysis was performed for Case 1 with results showing a pre-tax project NPV of US$261.7 million and an Internal Rate of Return (IRR) of 21.2% over a 21-year mine life. The post-tax cash flow analysis results in a project NPV of $175.0 million and an IRR of 17.6%. Payback on a pre-tax and after-tax basis occurs in Years 5 and 7, respectively. Table 22.1, Case 1 Estimated Cash Flow, details the yearly production, costs and cash flow for Case 1.

Table 22.1: Case 1 Estimated Cash Flow

Description	Units	LOM Totals	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Production Statistics														
Waste	dry tonnes (000s)	40,122	0	0	2,500	2,218	2,215	2,214	2,215	2,216	2,219	2,292	2,506	2,511
Ore	dry tonnes (000s)	13,949	0	0	0	690	690	690	690	690	690	690	690	690
Total Material	dry tonnes (000s)	54,071	0	0	2,500	2,907	2,905	2,904	2,905	2,906	2,909	2,981	3,196	3,200
Strip Ratio	waste:ore	2.88	0.00	0.00	0.00	3.21	3.21	3.21	3.21	3.21	3.22	3.32	3.63	3.64
Lithium Carbonate Production	dry tonnes (000s)	264.12	0.00	0.00	0.00	12.78	12.61	12.84	13.39	12.61	13.36	13.42	12.10	13.23
Potassium Sulfate Production	dry tonnes (000s)	931.53	0.00	0.00	0.00	42.47	44.40	42.95	46.21	47.54	48.14	47.30	36.80	44.88
Sodium Sulfate Production	dry tonnes (000s)	1,066.47	0.00	0.00	0.00	55.39	52.21	54.27	52.02	49.36	51.77	51.19	52.39	53.82
Commodity Prices														
Lithium Carbonate Price	US$	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000
Potassium Sulfate Price	US$	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600
Sodium Sulfate Price	US$	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75
Revenue														
Lithium Carbonate	(US$000s)	$1,584,746	$0	$0	$0	$76,652	$75,682	$77,040	$80,339	$75,682	$80,145	$80,533	$72,577	$79,369
Potassium Sulfate	(US$000s)	$558,921	$0	$0	$0	$25,483	$26,641	$25,772	$27,727	$28,523	$28,885	$28,378	$22,080	$26,931
Sodium Sulfate	(US$000s)	$79,985	$0	$0	$0	$4,154	$3,915	$4,070	$3,901	$3,702	$3,883	$3,839	$3,929	$4,037
Net Revenue Before Taxes	**(US$000s)**	**$2,223,651**	**$0**	**$0**	**$0**	**$106,289**	**$106,238**	**$106,883**	**$111,967**	**$107,907**	**$112,913**	**$112,751**	**$98,586**	**$110,336**
Operating Costs														
Mining	(US$000s)	$114,168	$0	$0	$0	$5,874	$5,913	$6,092	$6,002	$5,987	$5,977	$5,910	$6,792	$7,330
Process	(US$000s)	$851,301	$0	$0	$0	$41,996	$41,942	$42,018	$42,202	$41,942	$42,191	$42,213	$41,769	$42,148
G&A	(US$000s)	$42,321	$0	$0	$0	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090
Total Operating Costs	**(US$000s)**	**$1,007,789**	**$0**	**$0**	**$0**	**$49,960**	**$49,945**	**$50,200**	**$50,294**	**$50,019**	**$50,258**	**$50,212**	**$50,651**	**$51,568**
Operating Margin	**(US$000s)**	**$1,215,862**	**$0**	**$0**	**$0**	**$56,329**	**$56,293**	**$56,683**	**$61,674**	**$57,887**	**$62,655**	**$62,539**	**$47,935**	**$58,768**
Nevada State Minerals Tax	**(US$000s)**	**$48,007**	**$0**	**$0**	**$0**	**$1,631**	**$681**	**$1,127**	**$1,718**	**$1,790**	**$2,235**	**$2,324**	**$1,590**	**$2,114**
Capital Costs	**(US$000s)**	**$262,678**	**$8,075**	**$71,434**	**$157,573**	**$0**	**$9**	**$36**	**$2,282**	**$613**	**$796**	**$1,130**	**$3,394**	**$759**
Pre-Tax Cash Flow	**(US$000s)**	**$905,177**	**($8,075)**	**($71,434)**	**($157,573)**	**$54,698**	**$55,603**	**$55,519**	**$57,673**	**$55,485**	**$59,624**	**$59,084**	**$42,952**	**$55,896**
Cumulative Pre-Tax Cash Flow	**(US$000s)**	**$905,177**	**($8,075)**	**($79,510)**	**($237,083)**	**($182,385)**	**($126,783)**	**($71,264)**	**($13,590)**	**$41,895**	**$101,519**	**$160,603**	**$203,555**	**$259,450**
Pre-Tax Net Present Value @ 8%	**(US$000s)**	**$261,710**												
Pre-Tax Internal Rate of Return		**21.2%**												
Depreciation	(US$000s)	($255,726)	$0	$0	$0	($23,708)	($42,675)	($34,141)	($27,317)	($22,095)	($17,951)	($16,053)	($16,140)	($16,494)
Depletion	(US$000s)	($244,602)	$0	$0	$0	($11,692)	($11,686)	($11,757)	($12,316)	($11,870)	($12,420)	($12,403)	($10,844)	($12,137)
Federal Income Tax @ 35%	(US$000s)	($233,635)	$0	$0	$0	($6,754)	($438)	($3,380)	($7,113)	($7,747)	($10,517)	($11,116)	($6,776)	($9,808)
Changes in Working Capital	(US$000s)	$0	$0	$0	$0	($9,992)	$3	($51)	($19)	$55	($48)	$9	($88)	($183)
Post Tax Cash Flow	**(US$000s)**	**$671,542**	**($8,075)**	**($71,434)**	**($157,573)**	**$37,952**	**$55,168**	**$52,088**	**$50,542**	**$47,793**	**$49,060**	**$47,978**	**$36,088**	**$45,904**
Cumulative Post-Tax Cash Flow	**(US$000s)**	**$671,542**	**($8,075)**	**($79,510)**	**($237,083)**	**($199,132)**	**($143,964)**	**($91,876)**	**($41,334)**	**$6,460**	**$55,519**	**$103,497**	**$139,585**	**$185,489**
Post-Tax Net Present Value @ 8%	**(US$000s)**	**$175,002**												
Post-Tax Internal Rate of Return		**17.6%**												

Description	Units	LOM Totals	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
Production Statistics															
Waste	dry tonnes (000s)	40,122	2,506	2,507	2,509	2,509	2,506	2,506	1,298	438	114	65	51	8	0
Ore	dry tonnes (000s)	13,949	690	690	690	690	690	690	690	690	690	690	690	152	0
Total Material	dry tonnes (000s)	54,071	3,196	3,197	3,199	3,199	3,196	3,196	1,988	1,128	803	755	741	160	0
Strip Ratio	waste:ore	2.88	3.63	3.63	3.64	3.64	3.63	3.63	1.88	0.64	0.16	0.09	0.07	0.05	0.00
Lithium Carbonate Production	dry tonnes (000s)	264.12	13.10	13.07	12.26	13.00	13.20	11.74	13.10	12.65	13.68	14.97	13.97	3.05	0.00
Potassium Sulfate Production	dry tonnes (000s)	931.53	46.81	45.13	42.47	47.42	47.06	38.01	46.09	46.57	51.16	54.54	54.05	11.53	0.00
Sodium Sulfate Production	dry tonnes (000s)	1,066.47	54.27	51.18	49.87	51.22	52.86	52.02	53.91	54.59	54.75	55.44	52.32	11.62	0.00
Commodity Prices															
Lithium Carbonate Price	US$	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$0
Potassium Sulfate Price	US$	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$0
Sodium Sulfate Price	US$	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$0
Revenue															
Lithium Carbonate	(US$000s)	$1,584,746	$78,593	$78,398	$73,547	$78,010	$79,175	$70,442	$78,593	$75,876	$82,086	$89,848	$83,832	$18,329	$0
Potassium Sulfate	(US$000s)	$558,921	$28,089	$27,075	$25,483	$28,451	$28,234	$22,804	$27,654	$27,944	$30,695	$32,722	$32,432	$6,917	$0
Sodium Sulfate	(US$000s)	$79,985	$4,070	$3,838	$3,741	$3,842	$3,965	$3,901	$4,043	$4,094	$4,106	$4,158	$3,924	$871	$0
Net Revenue Before Taxes	**(US$000s)**	**$2,223,651**	**$110,751**	**$109,312**	**$102,770**	**$110,303**	**$111,373**	**$97,148**	**$110,290**	**$107,914**	**$116,887**	**$126,728**	**$120,188**	**$26,118**	**$0**
Operating Costs															
Mining	(US$000s)	$114,168	$7,410	$7,308	$6,065	$6,036	$6,076	$6,085	$4,838	$4,106	$3,230	$3,185	$3,178	$775	$0
Process	(US$000s)	$851,301	$42,104	$42,094	$41,823	$42,072	$42,137	$41,650	$42,104	$41,953	$42,299	$42,732	$42,396	$9,515	$0
G&A	(US$000s)	$42,321	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$522	$0
Total Operating Costs	**(US$000s)**	**$1,007,789**	**$51,604**	**$51,492**	**$49,978**	**$50,198**	**$50,303**	**$49,825**	**$49,033**	**$48,149**	**$47,619**	**$48,006**	**$47,664**	**$10,812**	**$0**
Operating Margin	**(US$000s)**	**$1,215,862**	**$59,147**	**$57,820**	**$52,792**	**$60,105**	**$61,070**	**$47,323**	**$61,258**	**$59,765**	**$69,268**	**$78,722**	**$72,524**	**$15,305**	**$0**
Nevada State Minerals Tax	**(US$000s)**	**$48,007**	**$2,122**	**$2,445**	**$2,578**	**$2,926**	**$2,959**	**$2,277**	**$2,985**	**$2,915**	**$3,396**	**$3,882**	**$3,583**	**$728**	**$0**
Capital Costs	**(US$000s)**	**$262,678**	**$1,168**	**$1,666**	**$3,976**	**$1,554**	**$1,060**	**$278**	**$1,387**	**$73**	**$75**	**$0**	**$0**	**$5,339**	**$0**
Pre-Tax Cash Flow	**(US$000s)**	**$905,177**	**$55,857**	**$53,710**	**$46,238**	**$55,626**	**$57,051**	**$44,767**	**$56,886**	**$56,776**	**$65,797**	**$74,839**	**$68,941**	**$9,238**	**$0**
Cumulative Pre-Tax Cash Flow	**(US$000s)**	**$905,177**	**$315,307**	**$369,017**	**$415,255**	**$470,881**	**$527,932**	**$572,699**	**$629,585**	**$686,361**	**$752,158**	**$826,998**	**$895,939**	**$905,177**	**$905,177**
Pre-Tax Net Present Value @ 8%	**(US$000s)**	**$261,710**													
Pre-Tax Internal Rate of Return		**21.2%**													
Depreciation	(US$000s)	($255,726)	($16,698)	($8,920)	($1,233)	($1,594)	($1,883)	($1,777)	($1,558)	($1,459)	($1,350)	($1,072)	($860)	($747)	$0
Depletion	(US$000s)	($244,602)	($12,183)	($12,024)	($11,305)	($12,133)	($12,251)	($10,686)	($12,132)	($11,871)	($12,858)	($13,940)	($13,221)	($2,873)	$0
Federal Income Tax @ 35%	(US$000s)	($233,635)	($9,850)	($12,051)	($13,187)	($15,208)	($15,392)	($11,404)	($15,604)	($15,232)	($18,083)	($20,940)	($19,201)	($3,835)	$0
Changes in Working Capital	(US$000s)	$0	($7)	$23	$303	($44)	($21)	$96	$158	$177	$106	($78)	$68	$7,370	$2,162
Post Tax Cash Flow	**(US$000s)**	**$671,542**	**$45,999**	**$41,681**	**$33,354**	**$40,374**	**$41,638**	**$33,459**	**$41,440**	**$41,721**	**$47,821**	**$53,822**	**$49,809**	**$12,773**	**$2,162**
Cumulative Post-Tax Cash Flow	**(US$000s)**	**$671,542**	**$231,488**	**$273,169**	**$306,523**	**$346,897**	**$388,534**	**$421,993**	**$463,434**	**$505,155**	**$552,976**	**$606,798**	**$656,606**	**$669,380**	**$671,542**
Post-Tax Net Present Value @ 8%	**(US$000s)**	**$175,002**													
Post-Tax Internal Rate of Return		**17.6%**													

22.1.3.2 Case 2 Cash Flow Analysis

Case 2 cash flow analysis results in a pre-tax project NPV of US$551.8 million and an IRR of 24.4% with a mine life of 20 years. The post-tax cash flow analysis results in a project NPV of US$372.5 million and an IRR of 20.3%. Payback on a pre-tax and after-tax basis occurs in Years 6 and 7, respectively. Table 22.2, Case 2 Estimated Cash Flow, details the yearly production, costs and cash flow for Case 2.

Table 22.2: Case 2 Estimated Cash Flow

Description	Units	LOM Totals	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Production Statistics														
Waste	dry tonnes (000s)	97,241	0	2,001	4,001	3,899	3,894	3,974	3,967	3,971	3,968	3,967	3,972	5,178
Ore	dry tonnes (000s)	25,495	0	0	0	690	690	690	1,380	1,380	1,380	1,380	1,380	1,380
Total Material	dry tonnes (000s)	122,736	0	2,001	4,001	4,589	4,584	4,664	5,347	5,351	5,348	5,347	5,352	6,558
Strip Ratio	waste:ore	3.81	0.00	0.00	0.00	5.65	5.64	5.76	2.88	2.88	2.88	2.88	2.88	3.75
Lithium Carbonate Production	dry tonnes (000s)	477.35	0.00	0.00	0.00	12.97	13.03	12.90	25.94	25.81	25.94	25.94	25.94	25.23
Potassium Sulfate Production	dry tonnes (000s)	1,759.12	0.00	0.00	0.00	38.25	33.06	45.37	93.87	91.70	92.91	88.56	92.18	88.56
Sodium Sulfate Production	dry tonnes (000s)	1,868.79	0.00	0.00	0.00	43.32	43.28	52.62	103.38	105.96	101.60	99.82	100.53	96.58
Commodity Prices														
Lithium Carbonate Price	US$	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000
Potassium Sulfate Price	US$	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600
Sodium Sulfate Price	US$	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75
Revenue														
Lithium Carbonate	(US$000s)	$2,864,092	$0	$0	$0	$77,816	$78,204	$77,428	$155,633	$154,856	$155,633	$155,633	$155,633	$151,363
Potassium Sulfate	(US$000s)	$1,055,473	$0	$0	$0	$22,949	$19,836	$27,220	$56,322	$55,019	$55,743	$53,137	$55,309	$53,137
Sodium Sulfate	(US$000s)	$140,159	$0	$0	$0	$3,249	$3,246	$3,947	$7,753	$7,947	$7,620	$7,486	$7,540	$7,243
Net Revenue Before Taxes	**(US$000s)**	**$4,059,725**	**$0**	**$0**	**$0**	**$104,014**	**$101,286**	**$108,595**	**$219,708**	**$217,823**	**$218,996**	**$216,256**	**$218,481**	**$211,744**
Operating Costs														
Mining	(US$000s)	$197,356	$0	$0	$0	$6,322	$6,713	$7,080	$11,508	$11,622	$10,544	$10,574	$9,852	$9,281
Process	(US$000s)	$1,418,319	$0	$0	$0	$42,179	$42,201	$43,269	$76,047	$76,003	$76,047	$76,047	$76,047	$75,806
G&A	(US$000s)	$41,798	$0	$0	$0	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090
Total Operating Costs	**(US$000s)**	**$1,657,473**	**$0**	**$0**	**$0**	**$50,591**	**$51,003**	**$52,440**	**$89,645**	**$89,715**	**$88,681**	**$88,710**	**$87,988**	**$87,177**
Operating Margin	**(US$000s)**	**$2,402,251**	**$0**	**$0**	**$0**	**$53,424**	**$50,283**	**$56,156**	**$130,064**	**$128,108**	**$130,315**	**$127,546**	**$130,493**	**$124,567**
Nevada State Minerals Tax	**(US$000s)**	**$98,259**	**$0**	**$0**	**$0**	**$1,432**	**$254**	**$950**	**$4,239**	**$3,817**	**$4,423**	**$4,618**	**$4,940**	**$4,736**
Capital Costs	**(US$000s)**	**$449,495**	**$9,300**	**$74,924**	**$163,711**	**$5,834**	**$4,099**	**$151,467**	**$3,837**	**$646**	**$169**	**$2,466**	**$8,688**	**$1,265**
Pre-Tax Cash Flow	**(US$000s)**	**$1,854,497**	**($9,300)**	**($74,924)**	**($163,711)**	**$46,158**	**$45,930**	**($96,261)**	**$121,987**	**$123,645**	**$125,723**	**$120,462**	**$116,866**	**$118,565**
Cumulative Pre-Tax Cash Flow	**(US$000s)**	**$1,854,497**	**($9,300)**	**($84,224)**	**($247,935)**	**($201,777)**	**($155,847)**	**($252,108)**	**($130,120)**	**($6,475)**	**$119,248**	**$239,709**	**$356,575**	**$475,140**
Pre-Tax Net Present Value @ 8%	**(US$000s)**	**$551,829**												
Pre-Tax Internal Rate of Return		**24.4%**												
Depreciation	(US$000s)	($437,072)	$0	$0	$0	($24,793)	($45,212)	($37,163)	($45,287)	($51,770)	($41,849)	($35,182)	($31,701)	($29,843)
Depletion	(US$000s)	($446,570)	$0	$0	$0	($11,442)	($11,141)	($11,945)	($24,168)	($23,961)	($24,090)	($23,788)	($24,033)	($23,292)
Federal Income Tax @ 35%	(US$000s)	($497,123)	$0	$0	$0	($5,515)	$0	$0	($19,650)	($16,996)	($20,983)	($22,385)	($24,437)	($23,343)
Changes in Working Capital	(US$000s)	$0	$0	$0	$0	($10,118)	($83)	($287)	($7,441)	($14)	$207	($6)	$144	$162
Post Tax Cash Flow	**(US$000s)**	**$1,357,375**	**($9,300)**	**($74,924)**	**($163,711)**	**$30,525**	**$45,848**	**($96,548)**	**$94,896**	**$106,635**	**$104,946**	**$98,071**	**$92,573**	**$95,384**
Cumulative Post-Tax Cash Flow	**(US$000s)**	**$1,357,375**	**($9,300)**	**($84,224)**	**($247,935)**	**($217,410)**	**($171,562)**	**($268,110)**	**($173,214)**	**($66,580)**	**$38,367**	**$136,437**	**$229,011**	**$324,395**
Post-Tax Net Present Value @ 8%	**(US$000s)**	**$372,476**												
Post-Tax Internal Rate of Return		**20.3%**												

Description	Units	LOM Totals	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21
Production Statistics														
Waste	dry tonnes (000s)	97,241	5,167	5,166	5,170	5,171	5,183	6,966	6,961	6,967	6,756	658	284	0
Ore	dry tonnes (000s)	25,495	1,380	1,380	1,380	1,380	1,380	1,380	1,380	1,380	1,380	1,380	1,350	0
Total Material	dry tonnes (000s)	122,736	6,547	6,546	6,550	6,551	6,563	8,346	8,341	8,347	8,136	2,038	1,634	0
Strip Ratio	waste:ore	3.81	3.75	3.74	3.75	3.75	3.76	5.05	5.05	5.05	4.90	0.48	0.21	0.00
Lithium Carbonate Production	dry tonnes (000s)	477.35	25.29	26.07	26.52	27.30	24.52	25.55	25.16	26.26	25.16	25.55	26.27	0.00
Potassium Sulfate Production	dry tonnes (000s)	1,759.12	92.91	103.04	97.01	103.76	91.22	97.49	102.08	103.04	106.90	90.73	106.49	0.00
Sodium Sulfate Production	dry tonnes (000s)	1,868.79	99.20	100.08	107.06	107.59	101.53	104.36	101.07	102.97	99.29	100.80	97.74	0.00
Commodity Prices														
Lithium Carbonate Price	US$	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000	$6,000
Potassium Sulfate Price	US$	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600	$600
Sodium Sulfate Price	US$	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75
Revenue														
Lithium Carbonate	(US$000s)	$2,864,092	$151,751	$156,409	$159,126	$163,783	$147,094	$153,304	$150,975	$157,573	$150,975	$153,304	$157,599	$0
Potassium Sulfate	(US$000s)	$1,055,473	$55,743	$61,824	$58,205	$62,259	$54,730	$58,494	$61,245	$61,824	$64,141	$54,440	$63,894	$0
Sodium Sulfate	(US$000s)	$140,159	$7,440	$7,506	$8,030	$8,069	$7,615	$7,827	$7,581	$7,723	$7,447	$7,560	$7,331	$0
Net Revenue Before Taxes	**(US$000s)**	**$4,059,725**	**$214,935**	**$225,739**	**$225,360**	**$234,111**	**$209,439**	**$219,625**	**$219,801**	**$227,120**	**$222,563**	**$215,304**	**$228,823**	**$0**
Operating Costs														
Mining	(US$000s)	$197,356	$9,333	$10,052	$10,049	$11,154	$11,110	$12,322	$13,116	$13,111	$11,767	$6,018	$5,829	$0
Process	(US$000s)	$1,418,319	$75,828	$76,091	$76,244	$76,507	$75,565	$75,916	$75,784	$76,156	$75,784	$75,916	$74,883	$0
G&A	(US$000s)	$41,798	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$2,090	$0
Total Operating Costs	**(US$000s)**	**$1,657,473**	**$87,251**	**$88,233**	**$88,383**	**$89,751**	**$88,765**	**$90,327**	**$90,990**	**$91,357**	**$89,642**	**$84,023**	**$82,802**	**$0**
Operating Margin	**(US$000s)**	**$2,402,251**	**$127,684**	**$137,506**	**$136,977**	**$144,360**	**$120,674**	**$129,298**	**$128,811**	**$135,764**	**$132,922**	**$131,281**	**$146,021**	**$0**
Nevada State Minerals Tax	**(US$000s)**	**$98,259**	**$4,923**	**$5,827**	**$6,221**	**$6,599**	**$5,662**	**$6,350**	**$6,333**	**$6,680**	**$6,539**	**$6,479**	**$7,239**	**$0**
Capital Costs	**(US$000s)**	**$449,495**	**$1,824**	**$1,229**	**$3,989**	**$1,448**	**$1,556**	**$1,117**	**$1,830**	**$1,070**	**$356**	**$0**	**$8,669**	**$0**
Pre-Tax Cash Flow	**(US$000s)**	**$1,854,497**	**$120,936**	**$130,450**	**$126,768**	**$136,312**	**$113,456**	**$121,831**	**$120,648**	**$128,014**	**$126,026**	**$124,802**	**$130,113**	**$0**
Cumulative Pre-Tax Cash Flow	**(US$000s)**	**$1,854,497**	**$596,076**	**$726,527**	**$853,295**	**$989,607**	**$1,103,063**	**$1,224,894**	**$1,345,542**	**$1,473,556**	**$1,599,582**	**$1,724,384**	**$1,854,497**	**$1,854,497**
Pre-Tax Net Present Value @ 8%	**(US$000s)**	**$551,829**												
Pre-Tax Internal Rate of Return		**24.4%**												
Depreciation	(US$000s)	($437,072)	($29,219)	($20,964)	($12,564)	($12,372)	($7,441)	($2,304)	($2,154)	($2,173)	($2,137)	($1,700)	($1,244)	$0
Depletion	(US$000s)	($446,570)	($23,643)	($24,831)	($24,790)	($25,752)	($23,038)	($24,159)	($24,178)	($24,983)	($24,482)	($23,683)	($25,171)	$0
Federal Income Tax @ 35%	(US$000s)	($497,123)	($24,464)	($30,059)	($32,691)	($34,873)	($29,586)	($33,770)	($33,651)	($35,675)	($34,917)	($34,796)	($39,329)	$0
Changes in Working Capital	(US$000s)	$0	($15)	($196)	($30)	($274)	$197	($312)	($133)	($73)	$343	$1,124	$244	$16,560
Post Tax Cash Flow	**(US$000s)**	**$1,357,375**	**$96,457**	**$100,195**	**$94,047**	**$101,166**	**$84,067**	**$87,749**	**$86,865**	**$92,265**	**$91,452**	**$91,129**	**$91,029**	**$16,560**
Cumulative Post-Tax Cash Flow	**(US$000s)**	**$1,357,375**	**$420,852**	**$521,046**	**$615,093**	**$716,259**	**$800,326**	**$888,074**	**$974,939**	**$1,067,204**	**$1,158,656**	**$1,249,785**	**$1,340,814**	**$1,357,375**
Post-Tax Net Present Value @ 8%	**(US$000s)**	**$372,476**												
Post-Tax Internal Rate of Return		**20.3%**												

22.1.4 Sensitivity Analysis

Sensitivity analyses were performed for the pre-tax project NPV at an 8% discount rate. Sensitivities were performed on lithium carbonate pricing, total operating costs, total capital expenditures and total process recovery. Values were adjusted in 5% increments ranging from negative 15% to positive 15%.

22.1.4.1 Case 1 Sensitivity Analysis

Case 1 sensitivities show that the Project is most sensitive to process recovery, followed by lithium carbonate pricing and operating costs, and least sensitive to capital expenditures. Figure 22.1, Case 1 NPV Sensitivity, illustrates the various Project sensitivities in a spider graph format.

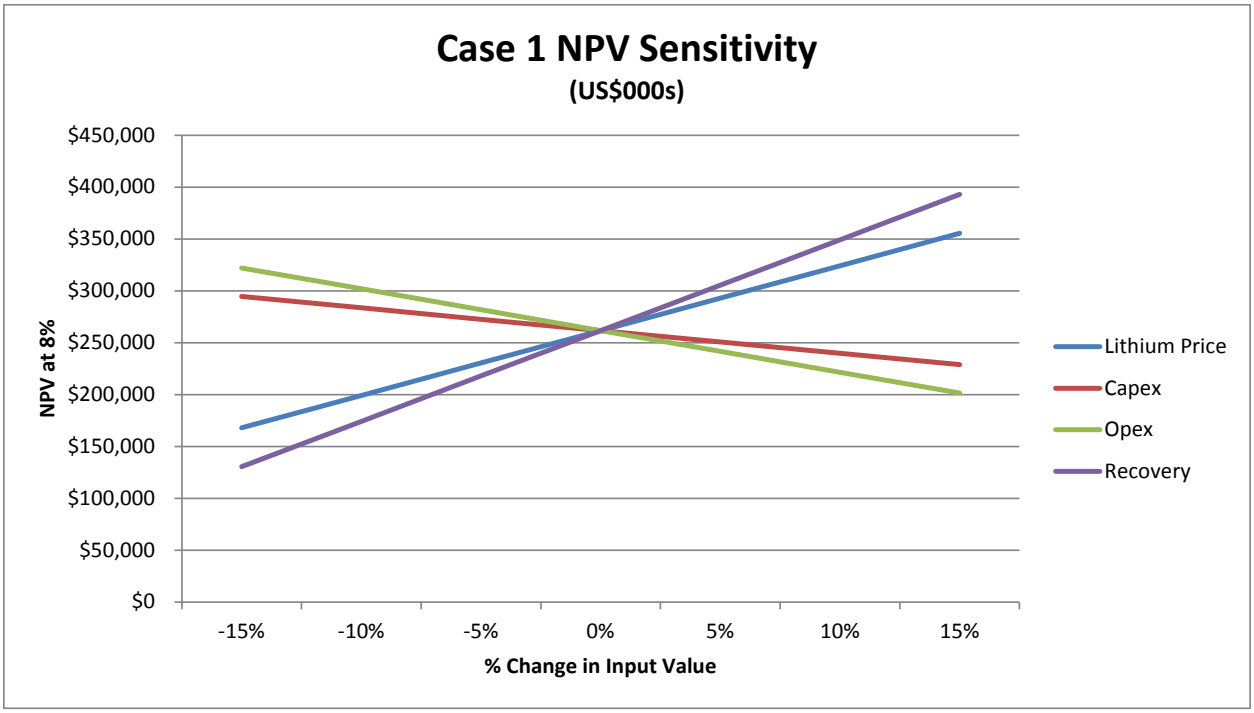


Figure 22.1: Case 1 NPV Sensitivity

22.1.4.2 Case 2 Sensitivity Analysis

Case 2 sensitivities show that the Project is most sensitive to the process recovery followed by lithium carbonate pricing and operating costs, and least sensitive to capital expenditures. Figure 22.2, Case 2 NPV Sensitivity, illustrates the various Project sensitivities in a spider graph format.

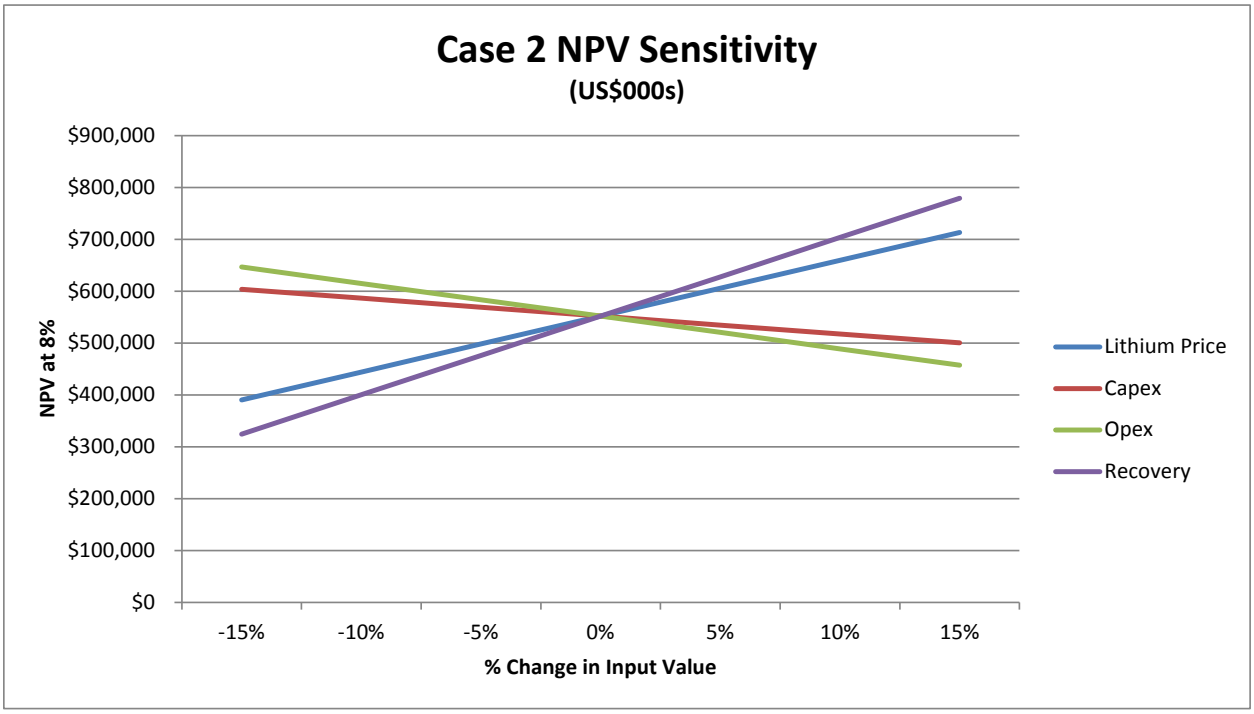


Figure 22.2: Case 2 NPV Sensitivity

22.2 STAGE II

The Stage II Resource Estimate report is preliminary in nature and addresses only the resource estimate. As such, this section is not applicable to the Stage II area.

23.0 ADJACENT PROPERTIES

23.1 STAGE I

Much of the information in this section was excerpted from PAEE (URS, 2010). The information was updated to reflect recent information.

American Colloid has a group of claims in the area of the Stage II Lens (northwest of the Stage I Lens). Smectite clays are mined occasionally, but the area is essentially quiet most of the year. The J.M. Huber Corporation owns a small number of claims along the eastern edge of the North Lens from which they periodically obtain small quantities of clay materials for specialty uses. No other clay operations exist in the area. Properties immediately adjacent to the mining area are all Bureau of Land Management lands.

Uranium deposits occur a few kilometers to the west of the lithium deposits; these are currently exploration properties, but had limited historical production. Northeast of the lithium deposits, the McDermitt mercury deposits were mined until the late 1990s. Those deposits are inactive at this time and not related to the lithium resource. A small number of historical gold exploration properties with possible minor production occur several kilometers to the west of the lithium deposits.

Numerous gold deposits are either under exploration or are operating to the south and southeast of the caldera margins. These deposits are not related to the lithium deposits and are too distant to have any direct impact on any lithium operations.

23.2 STAGE II

Apart from the other mineralized lenses on the King's Valley Project, there are no adjacent properties that bear on the lithium properties and there are no nearby operating mines. Several gold mines are in operation several tens of miles to the southeast and are mentioned to illustrate that mining permits are possible in the area. In the past century, a large mercury mine operated to the northeast of the lithium properties. To the west of the lithium properties, uranium and gold were produced from small mines in the past century. Those properties are being actively explored, but there is no current production.

The Huber Pit at the north end of the lithium mineralized trend is operated sporadically and possibly a few tens of tons of material are produced per year, but production generally occurs in a short period every two or three years.

American Colloid has a small number of claims in the area of the Stage II Lens but those claims have not produced in recent times.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 STAGE I

24.1.1 Tailings Disposal

In order to contain the tailings generated by the processing facilities, a dry stack tailings storage facility (TSF) would be constructed east of the processing facilities. The tailings will be dewatered to a moisture content (by dry mass) of less than 20% as they exit the processing facilities and would be conveyed and stacked on top of a lined pad.

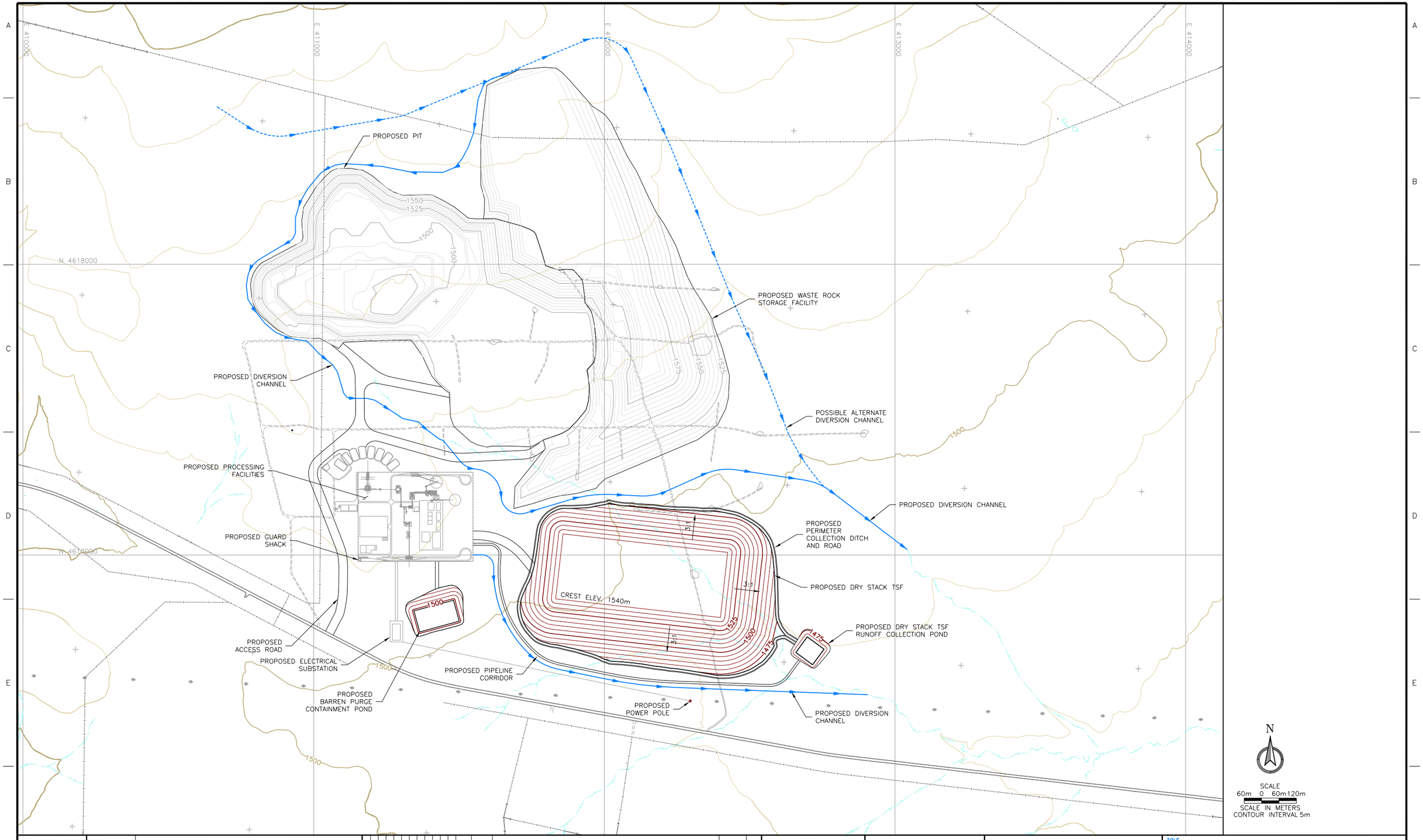
Two scenarios were evaluated for this study. The Case 1 TSF is designed to contain 20.4 million tonnes of tailings at an assumed in-place dry density of 1.61 tonnes per cubic meter (see Figure 24.1) The Case 2 TSF is designed to contain 36.8 million tonnes of tailings at the same density (see Figure 24.2). For both cases, the TSF will be constructed in a series of five cells. Each cell is expected to take approximately four years to complete, for a total TSF life of 20 years. The first four cells will be constructed adjacent to each other to an elevation of 1,525 m (both cases). The fifth cell will be constructed as a cap over the first four cells, and will add an additional 15 m of tailings height to both cases, for a maximum height of 69.5 m and 79 m for the Case 1 and Case 2 TSFs, respectively. The final configuration for both cases results in a crest elevation of 1540 m with 3H:1V side slopes. An access road will be built around the perimeter of the TSF for maintenance access.

In both scenarios, the facility will be fully lined and will require an underdrain system to prevent uplift pore pressures from accumulating in existing drainage paths beneath the TSF liner, as well as a lined perimeter ditch to collect contact water from seepage and/or runoff from above the TSF liner. Water from the underdrain system will be monitored for water quality and will report to the existing drainage located downstream of the TSF. The water from the lined perimeter runoff conveyance ditch will collect in a lined runoff collection pond, from where it will be pumped back to the processing facilities for use as reclaim water.

Engineering analyses conducted for the TSF indicate that for both Case 1 and Case 2 mine development plans, the tailings can be safely stored given the designs contained herein.

The total capital cost for the Case 1 scenario, including closure costs, is estimated at US$19.3 million. The total capital cost for the TSF Case 2 scenario, including closure costs, is estimated at US$28.4 million.

The TSF designs for each case are detailed in Technical Reference D.



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DWG. NO.	REFERENCE DRAWINGS

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WESTERN LITHIUM WLC

HUMBOLDT COUNTY, NV

TETRA TECH

WARDROP Engineering Inc.

TITLE

KINGS VALLEY LITHIUM PROJECT
PRELIMINARY FEASIBILITY STUDY
CASE 1
PROPOSED ULTIMATE FACILITIES ARRANGEMENT

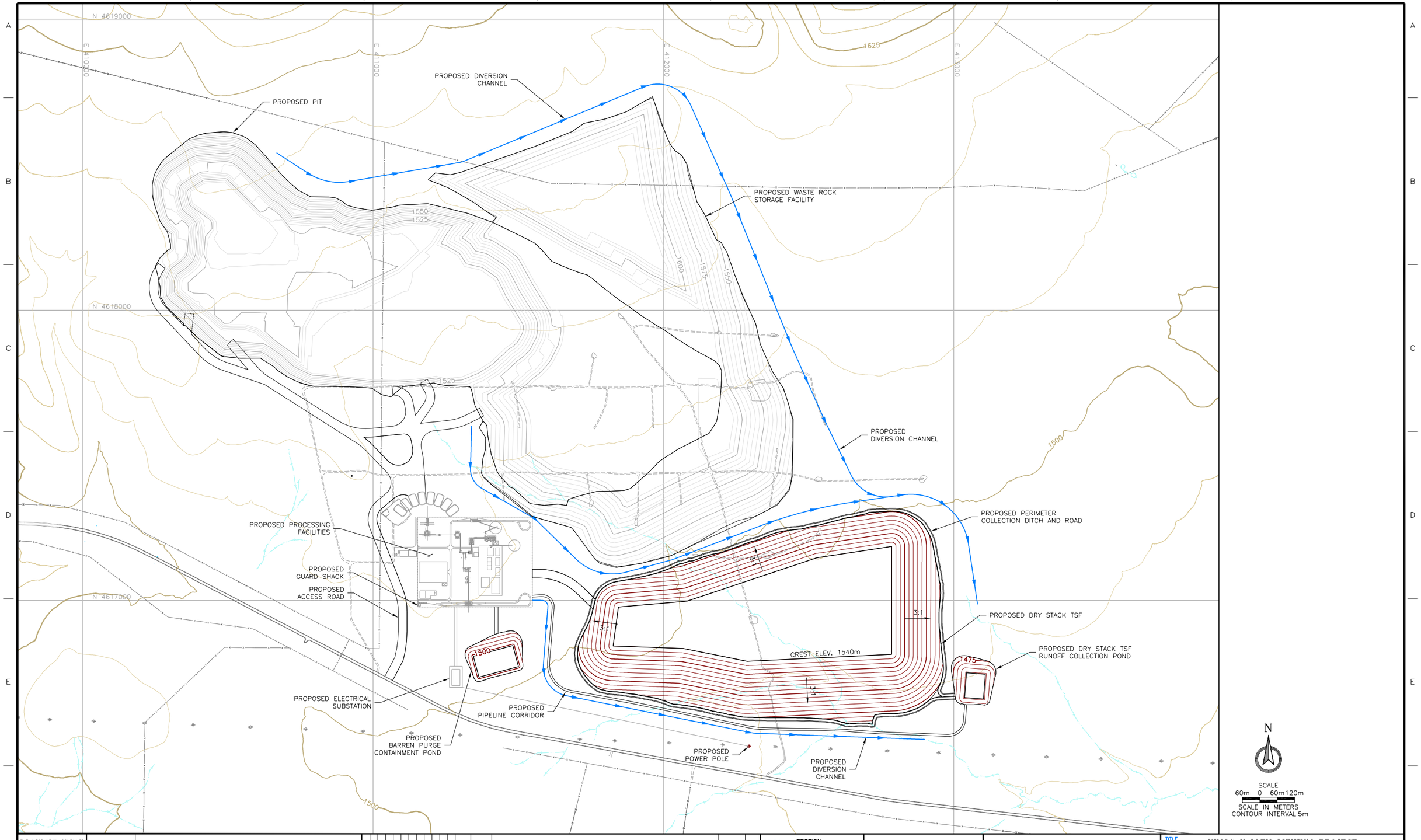
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	114-311106	FIGURE 24.1	B

A1

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REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
B		FOR NI 43-101 PFS	01/12	AKB
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DESIGN. BY:	AKB	19DEC11
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APP. BY:	AKB	18JAN12

WESTERN WLC LITHIUM

HUMBOLDT COUNTY, NV

TETRA TECH

WARDROP Engineering Inc.

TITLE

KINGS VALLEY LITHIUM PROJECT
PRELIMINARY FEASIBLITY STUDY

CASE 2
PROPOSED ULTIMATE FACILITIES ARRANGEMENT

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	114-311106	FIGURE 24.2	B

REVTIME

REVDATE

A1

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24.1.2 Process Demonstration Facility for Lithium Extraction

Western Lithium continued and completed an extensive process development program aimed at the recovery of battery grade lithium carbonate from the hectorite clay. Due to the complexity of the process, Western Lithium concluded that it would be necessary to perform a continuous small pilot scale operation. Owing to this decision, Western Lithium has proposed to build a demonstration plant to prove the process and demonstrate continuous production for the manufacture of battery grade lithium carbonate from the hectorite clay. Western Lithium contracted with URS and K-UTEC Salt Technologies (K-UTEC) to prepare a report titled: Western Lithium Demonstration Plant Basic Engineering Report, dated December 2013.

In the Stage I PFS, the development of the pilot/demonstration plant was identified as a two-phase approach. Phase one involved the basic engineering concurrent with preliminary calcining test work on pilot scale. This was followed with detailed engineering, procurement and erection of the pilot plant at a site located within K-UTEC's facilities. Phase two is the operation of the demonstration facility.

The demonstration plant design is modular to prove the process efficiencies and functionality of the key unit operations that affect project economics. The key modules of the plant are distributed into calcination, leaching and crystallization. The pilot will be run by campaign to allow for evaluation of each process module, optimize efficiencies and limit excessive downtime. The selection of equipment is based on the PFS process design and is modified, as necessary, to fit budget and operability constraints without loss of "proof of concept". The demonstration plant size is selection is based on equipment availability, schedule of operation and scalability to commercial equipment.

This demonstration plant effort has started and basic engineering is complete. The program is currently procuring equipment for the pilot with early construction works also underway. Construction completion is scheduled for August to be immediately followed with start-up of the pilot. The estimated capital and operating costs for this program are US$3,400,000 and US$2,000,000, respectively.

24.1.3 Direct Fired Kiln Test for Calcination of Lithium Bearing Clays

The calcination work previously performed for the project during the PFS development used indirect fired rotary kilns and a fluid bed calciner. Western Lithium commissioned IBU-tec (IBU-tec, 2013) to perform a trial run using a direct-fired kiln. The scope of this test was to select the optimum operating conditions and temperature profile for the demonstration pilot-run and to identify any problems with material sintering.

The work was performed in two phases using feed materials provided by Western Lithium. The first phase was a pre-testing on small laboratory scale to determine operating parameters for the 2.5 day semi-pilot run processing 1.5 tonne of blended feed. The ratio of 5:2:2 (clay to dolomite to anhydrite) was used in all test work.

The following criteria were set for the laboratory tests:

- Granulation in small scale Eirich RO2 mixer, to achieve grain size between 0.5 mm to 4 mm.
- Calcining temperature: 900°C - 1150°C
- Residence time: 30 – 60 minutes

Quality parameters (color value, CO_2, SO_2, crystal water and XRD) were analyzed IBU-tec.

The second phase involved granulation in a pilot sized Eirich mixer, model R11 and calcining in 0.3m x 7m direct-fired KDO rotary kiln. The granulation was performed batch-wise, with target grain size of 0.5mm– 4mm.

Conclusions and observations of this trial are:

1. Granulation performance is dependent on the linear speed, water addition and time of mixing. Various parameters were adjusted in order to maintain particles sizing below 7mm.

2. All 1.3 tonnes were processed through the kiln operating between 950°C and 1150°C at different retention times. Calcined material was collected and samples delivered to K-UTEC for a leach analysis.

3. No sintering was observed until temperatures reached 1150°C. Some minor melting was observed along with the clinging of material to the refractory lining. The sintering was minimal at such a high temperature and the majority of the calcine was acceptable. Very little dust was generated and collected in the collection system.

In general, the kiln operation was successful from a physical and throughput point of view. Samples collected and analyzed by K-UTEC confirmed good kiln performance with 80% to 92% Li extracted over a temperature range of 950°C - 1150°C with a peak at 1050°C, as expected. The results correspond closely to the PFS design range.

24.2 STAGE II

The Stage II Resource Estimate is preliminary in nature and addresses only the resource estimate. As such, this section is not applicable to the Stage II area.

24.3 KINGS VALLEY CLAY PROJECT

KINGS VALLEY CLAY PROJECT SUMMARY REPORT
Prepared By Western Lithium Corporation
May 5, 2014

Statement

This report contains information on the hectorite clay at Western Lithium's Kings Valley Clay (KVC) Project in Humboldt County, Nevada. The KVC Project shares some of the same property as the King's Valley Lithium Project. The data presented here in regard to the clay was generated in-house by Western Lithium Corporation (WLC) personnel and does not address nor identify any mineral resource disclosure classification for the clay as described under NI 43-101. The test work included in this report (other than any outside reports identified) was generated by WLC's internal clay laboratory facility under the direction of the Manager of Clay Operations who oversees that laboratory.

We caution that WLC has not demonstrated the economic viability of the organoclay business or the Fernley Facility through at least a preliminary economic assessment. Notwithstanding this, WLC has made a development decision in respect of the organoclay business and the Fernley Facility that is

informed by its understanding of the hectorite clay characteristics, analysis of testing and pilot plant work, and the engineering and development analysis that has been completed to date and is still in progress. Readers should note that there are increased risks in respect of a development decision that is not supported by planning to the point of completion of a feasibility study. This includes greater uncertainty about the relevant inputs that form the basis of a development plan, such as the size and scope of infrastructure required to efficiently operate the organoclay business, the assumptions regarding performance of the organoclay business such as the most efficient extraction methods and characteristics attributable to clay, estimates of capital and operating costs and sales and marketing processes. WLC does not intend to undertake further analysis to address all of these matters and there is an increased risk associated with any business that has not completed feasibility study work, including with respect to the reliability of estimates and assumptions underlying the development of that business.

Introduction

WLC is the owner of a hectorite clay deposit in Northern Nevada that they are developing for both its contained lithium and for clay. WLC has recently embarked on a commercial clay strategy based on the unique properties of the hectorite clay at its Kings Valley Property in which it would use the clay to become a specialty supplier of clay based drilling additives to the oil and gas industry. The proposed clay development is referred to as the Kings Valley Clay (KVC) Project.

Development is planned for both lithium and clay on that portion of the deposit referred to as Stage 1 at Thacker Pass. Permitting for clay extraction encompasses approximately 110 acres, just under a mile northeast of Thacker Pass and State Highway 293.

WLC is pursuing the opportunity of becoming a specialty supplier of Hectatone™ clay based drilling additives to the oil and gas industry. Hectorite clay based drilling additives have been found to be particularly applicable for unconventional shale drilling due to their thermal and gelling properties that can improve performance when developing deep deposits that require horizontal drilling. WLC also plans to use other clays such as bentonite for certain organoclay products. To make a specialty drilling fluid referred to as organoclay, the company is designing and building an organoclay manufacturing plant in Fernley, Nevada. The plant is expected to be commissioned in late summer of 2014. The company believes that continued growth of shale gas drilling in the USA combined with a global push towards developing shale gas resources will provide growth opportunities for WLC's strategy to build a specialty drilling additives business.

History of the Clay under Chevron Resources Company Ownership

Chevron Resources was the first to recognize the clay potential on the property. They began exploration for uranium in the McDermitt caldera in 1975. In 1977 the U.S. Geological Survey alerted Chevron to the presence of anomalous concentrations of lithium associated with the volcaniclastic moat sediments within the caldera. Drilling by Chevron confirmed the presence of high lithium values hosted by a massive, green claystone (smectite/hectorite) within the moat sediment section. From 1978 through 1987 Chevron drilled approximately 230 holes evaluating the lithium in the claystone. It was during this time that Chevron recognized the potential for utilizing these same clays for drilling mud applications. They conducted a comprehensive viscosity testing program and undertook a market study for drilling fluids. Which indicate, on review of Chevron's historical data that Chevron believed there was potential

for developing drilling mud from the claystone on site, however, Chevron eventually abandoned the project in 1991.

Other clay companies became aware of the clay on site and one company, American Colloid Company, has mined hectorite on a small scale for over 20 years from a small pit approximately 15 miles north of the KVC Project in the same geologic formation and adjacent to some of WLC's mining claims.

Geology of the Clay

There have been several professional papers recognizing the occurrence of and addressing the smectite/hectorite clay and the geology within the McDermitt caldera which include:

- **Lithium in the McDermitt Caldera, Nevada and Oregon by Glanzman and McCarthy from the USGS, 1978.**

- **Relation of Mercury, Uranium, and Lithium Deposits to the McDermitt Caldera Complex, Nevada-Oregon by Rytuba and Glanzman, USGS 1979.**

- **Uranium, Thorium, and Mercury Distribution through the Evolution of the McDermitt Caldera Complex by Rytuba, Glanzman and Conrad, Basin and Range Symposium 1979.**

- Hectorite Deposits in the McDermitt Caldera of Nevada by Odem, 1992. This paper describes the caldera in general and also two commercial deposits of Hectorite that were being mined at the time and of which he states: "Both contain large volumes of hectorite." (one of these deposits, American Colloid's, is still in production).

- Hectorite, Restricted Occurrence – Diverse Applications by Fitzgerald and Kendall, Industrial Clays 1996. A general discussion of hectorite. Stresses the limited nature of hectorite occurrences but includes a discussion of the Disaster Peak, Nevada hectorite which is in the McDermitt caldera.

The KVC Project in Humboldt County, Nevada, is located in the southern part of the McDermitt caldera complex, a Miocene volcanic feature that comprises a total surface area of about 18 x 25 miles. Volcanic activity at the McDermitt caldera complex was characterized by extrusion of rhyolites and followed by eruption of rhyolite and dacites. After collapse, the central part of the caldera complex was the site of resurgence, and a moat-like lake formed between this resurgent dome and the caldera walls. The lake was the site of deposition of volcaniclastic sediments that now form a nearly continuous ring within the caldera. The Kings Valley hectorite deposits are lens-shaped bodies hosted by these moat sedimentary rocks.

The Kings Valley hectorite claystone deposits occur in a north-south zone that coincides with the western and southern part of the McDermitt caldera. The deposit discussed in this report, the most southerly deposit, is referred to as the Stage 1 Lens, or the PCD lens in earlier reports. The claystone deposits in the Stage 1 Lens is mostly beneath an approximately 2 to 3 metres of thick alluvial cover. Structural disturbance is common with large to small scale normal faulting. Exposures of the moat sedimentary rocks are limited to a few drainages and isolated road cuts; therefore, the stratigraphic sequence in the Stage 1 Lens is primarily derived from core drilling.

The moat sedimentary section, which has a maximum drilled thickness of about 160 metres, consists of alternating layers of claystone and volcanic ash. The claystone comprises more than 75% of the section. In many intervals, the claystone and ash are intimately intermixed. The claystones are variably brown, tan, gray, bluish-gray, and black; whereas the ash is generally white or very light gray.

Surficial oxidation persists to depths of 15 to 30 metres in the moat sedimentary rock. Oxidized claystone is brown, tan, or light greenish-tan and contains iron oxide, whereas the ash is white with some orange-brown iron oxide. The transition from oxidized to unoxidized rock occurs over intervals as much as 5 metres thick.

Western Lithium Evaluation of the Hectorite Clays

Over 200 drill holes have been advanced in the Stage 1 Lens, primarily for the delineation of the lithium in the deposit and for overall geologic understanding of the moat sediment formation. The lithium resource is in the claystones in the deposit. In the oxidized zone (the upper 15-30 metres) the lithium is in a smectite clay, consisting primarily of hectorite. Lithium values are highest below the oxidized zone, therefore below the upper hectorite clays. The hectorite clay of interest for potential drilling mud applications have been identified by visual examination and physical testing. Core from each drill hole has been examined and drill logs have been prepared that record rock type, color, accessory minerals, textures, and other features of significance. On the basis of core examination, 14 rock types have been recognized. The core has been divided into sample intervals (for physical testing and chemical analyses) delineated on the basis of lithology. Of the 14 rock types, four have been identified as having the best potential for potential organoclay production and they are generally found within 20 metres from the surface in the oxidized zone.

An internal analysis by WLC was undertaken in 2012 to assess the areas on the previously drilled lithium property which would have the highest potential for supplying clay for possible drilling fluids applications. The existing core was re-analyzed, sampled and tested. Testing mostly entailed viscosity determinations to compare with API (American Petroleum institute) standards. Based on preliminary results two areas were identified that appeared to contain suitable clays near the surface. Large test pits were excavated to assess the continuity and consistency of the clay lenses. Limited additional drilling on 15 metre offset was also undertaken to obtain additional samples and to further assess the continuity of these near surface clays.

The report concluded that:

- The test pit (WLT-01) exposed approximately 30 m of good clay approximately 3 m thick. Viscosity tests indicate the clay being continuously good for the length of the pit. This location was chosen as previous drilling had indicated this location showed clay close to the surface (within about 3 m). See test pit cross section.

- Eight drill holes at 15 m spacing averaging 18 m depth were drilled to further expand this potential clay lens. See cross section showing drill logs and intersects where clay was confirmed by testing. (Viscosity values greater than 20 were considered to be usable clay for gelling). The table shows the drill hole test results with values and intercepts. Color indicates grades of acceptable clays.

- The acceptable clays could be visually identified based on color, purity, texture and feel.

- In looking at the drill hole cross section the following observations can be made:
  - The good clay lenses are somewhat continuous from hole to hole and range from 0.6 to 3 m in thickness.
  - The clays are not necessarily continuous as the distance from the test pit increases.
- Strip ratio at the test pit is approximately 1 to 1.
- Strip ratio in an area encompassing the larger cross section area outside the immediate test pit is approximately 4 to 1.

WLC Clay Classification Types

The claystones most suitable for potential organoclay drilling fluid consideration can be classified as green, tan, light gray and gray claystone and should exhibit very low amounts of ash. The claystones will generally display a waxy, shiny surface and a distinct cleavage. Ash layers could be interspersed in the claystone and can also be found in beds of varying sizes in the KVC Project.

Green Claystone - The bulk of the green claystone is found within the first 30 m of the KVC Project but has been logged as deep as 120 m. It is most often found in the oxidized upper section of the hole accompanied by minor to moderate iron staining. Color varies from light green to medium green and may have a bright "pistachio" color at times.

Tan Claystone - The majority of the tan claystone is located within the upper 30 m depth of the KVC Project and has been logged as deep as 207 m. The tan claystone is a light tan color and often has varying degrees of light to medium tan color throughout. It is commonly coated with a reddish iron stain as it is usually found in the upper oxidized region.

Light Gray Claystone - The light gray claystone is found mostly within the first 60 m with the highest percentage claystone located within approximately 30 to 60 m depth. A substantial amount is also located within the upper 30 m. This clay is a light gray to light brown color. The shades of gray and brown may overlap each together.

Gray Claystone - The bulk of the gray claystone is located in the 30 to 60 m section of the KVC Project. This clay is a medium to dark gray to medium to dark brown color. Random mixed layering of slightly different shades of gray and brown is common. Features such as turbidity currents and soft sediment deformation may occasionally be noted.

Other Testing

Independent testing of the clay has been conducted by Miles Industrial Minerals Research consisting of:

- Evaluation of Western Lithium Corp. Sample WLC #1, August 12, 2011. Report of sample indicating it has excellent drilling fluid properties of viscosity and fluid loss control. Also discusses stability when hot rolled.

- Chemical analysis and physical property evaluation of three McDermitt Caldera Hectorite samples, March 23, 2012. Report indicating clay samples are predominantly hectorite and discussion of other elements and exchangeable cations.

- Evaluation of WLC Hectorite by X-ray Diffraction Analysis and Cation Exchange Capacity. Report indicating the average hectorite concentration of the samples provided averaged 86% and addressed other average concentrations as well, including calcite at 6%, quartz at 4%, feldspar at 3% and dolomite at 0.7%.

King's Valley Clay Mine, Humboldt County, Nevada

The following table lists all local, State, and Federal permits required for WLC to extract clay from its Kings Valley Property in Humboldt County, Nevada. WLC has received all major permits to extract clay.

Permits – Western Lithium Corporation, Kings Valley Clay Mine, Humboldt County, Nevada

Permit	***Issuing Authority***	***Approval Date***
Plan of Operations/Reclamation Permit	Bureau of Land Management	Approved 5/1/2014
Approval of Reclamation Surety Bond	Bureau of Land Management, Nevada State Office	Outstanding
Environmental Assessment Decision Record & FONSI	Bureau of Land Management	Approved 3/5/2014
Mine Safety Health Administration Legal Mine Identity Number	Mine Safety Health Administration	Approved 4/28/2014 MSHA ID # 26-02762
Notification of Commencement of Operations	Mine Safety & Health Administration	Outstanding
Notice of Opening a Mine	Nevada Dept. Business & Industry Division of Industrial Relations	Outstanding
Nevada Mine Registry	Nevada Division of Minerals	Outstanding
Surface Area Disturbance Permit	Nevada Division of Environmental Protection, Bureau of Air Pollution Control	Approved 3/18/2014
Air Quality Operating Permit (for 1-2 on-site portable generators)	Nevada Division of Environmental Protection, Bureau of Air Pollution Control	In 2014, WLC will work under contractors (Hunewill) COLA Permit. WLC will obtain own permit in 2015.
Mining Reclamation Permit	NDEP, Bureau of Mining Regulation & Reclamation	Approved 4/29/2014 Reclamation Permit # 0357
Mineral Exploration Hole Plugging Permit or Waiver	NDEP, Division of Water Resources	To be obtained by the exploration drilling contractor.
General Industrial Stormwater Discharge Permit	NDEP, Bureau of Water Pollution Control	*not required. Site does not drain to jurisdictional waters of the U.S.
Permit to Appropriate Water/Change Point of Diversion, 15.5 AF at Mine Site	Nevada Division of Water Resources	Permit # 82384 & 82385. Approved 8/28/2013
Permit to Appropriate Water 980 AF used at Ranch (water	Nevada Division of Water Resources	Permit #68633 & 68634

Permits – Western Lithium Corporation, Kings Valley Clay Mine, Humboldt County, Nevada

Permit	*Issuing Authority*	*Approval Date*
rights to be moved to mine site at later date, when needed)		
Encroachment Permit	NDOT (*note: Permit application only valid for 12 months. WLC will apply to NDOT in Spring 2015; when we get closer to constructing encroachment. Application & engineering design has been completed).	Outstanding
Fire and Life Safety Permit	Nevada Fire Marshall	*not required. WLC will not store chemicals on site. Buildings are not proposed at this time.
State Business License	Nevada Secretary of State	Issued/Approved
Building Permits	Humboldt County Building Department	*not required at this time. Temporary trailer or building is not proposed for 2014.
Conditional Use Permit	Humboldt County Regional Planning Department	Approval issued 4/10/2014
County Road Use & Maintenance Permit for mine site (APN 003-311-01)	Humboldt County Roads Department	*not required for mine site. WLC not hauling or using Humboldt County Roads
County Road Use (Encroachment Permit) for WLC Ranch; APN 003-401-55 & 003-401-56.	Humboldt County Roads Department	Encroachment Permit approved 4/24/2014. Maintenance Permit not required at this time.
County Business License	Humboldt County	*not required for a mining operation.

Organoclay Plant, Fernley, Nevada

Western Lithium Corporation is currently building a clay manufacturing facility in Fernley, Nevada that will have a production capacity of approximately 10,000 tons of organoclay per year. Commissioning of the plant is expected to occur late summer of 2014. Hectorite clay from the Kings Valley Property will be one of perhaps several raw materials (clays) that will be utilized as feed in the plant to make organoclay.

The following table lists all local, State, and Federal permits required for Kings Valley Clay, Inc. (Hectatone) Organoclay Plant, located in Lyon County (Fernley), Nevada. Kings Valley Clay, Inc. has received all major permits for the manufacturing of organoclay.

Permits – Kings Valley Clay, Inc. Organoclay Plant, Fernley, Nevada

EPA Hazardous Waste ID No.	EPA	*Pending review by CEM. Not anticipated to be required.
Spill Prevention, Control & Countermeasure (SPCC) Plan	EPA	*Not required; however, KVCI will prepare one as good practice.
Air Quality Operating Permit	NDEP, Bureau of Air Pollution Control	AP3295-3398; Approved 11/8/2013
General Industrial Stormwater Discharge Permit	NDEP, Bureau of Water Pollution Control	*Not required, site does not drain to a jurisdictional waters of the U.S.
Hazardous Materials Storage Permit	Nevada Fire Marshall	Outstanding
State Business License	Nevada Secretary of State	Approved/KVCI has obtained license
Retail Sales Permit or Exemption Certificate	Nevada State Department of Taxation	Outstanding
Building Permit	City of Fernley Building Department	BP 2014-070; Approved 12/5/2013
Design Review Permit	City of Fernley Building Department	DR 14-04; Approved 9/16/2013
Fire Protection District Initial Approval	North Lyon County Fire Protection District	Initial Approval 11/12/2013
Certificate of Compliance of Non-Structure Fire Life Safety	North Lyon County Fire Protection District	To be issued at CofO; Outstanding
Certificate of Occupancy (CofO)	City of Fernley Building Department	Outstanding
Water Rights Transfer & Approval	City of Fernley Building Department	1626 ERC issued; Approved 12/5/2013
City Business License	City of Fernley	*Not required.

To date, WLC has funded capital costs from existing cash resources. Although it has not completed a definitive capital cost study, WLC has allocated an additional $5 million to the construction of the Fernley Facility, as follows:

- Phase II construction, labour and materials $3.4 million
- Equipment and instruments $0.9 million
- Construction and engineering services $0.7 million

Operational costs depend on employees, input costs and other factors. Marketing and sales will depend on prevailing market prices and contract terms. The Company has not completed a definitive marketing study or entered into definitive sales or other contracts required in order to determine costs.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 STAGE I

25.1.1 General

WLC holds control and ownership of the Kings Valley property mining claims and leases within the boundaries of the PFS study. WLC is approved by the BLM and the NDEP BMRR to conduct mineral exploration activities at the Project site.

The Stage I PFS met the scope of work as described in Section 2.1.2 to define the necessary elements for construction, startup and operation of the Project through closure and reclamation at a +/- 25 percent level of accuracy.

WLC has a defined ore reserve that can be mined for lithium-bearing clay, and a process to recover compounds of Li, K and Na as marketable products.

Case 1 cash flow analysis indicates a pre-tax project NPV of US$262 million and an IRR of 21.2 percent over a 21-year mine life.

Case 2 cash flow analysis shows a pre-tax project NPV of US$552 million and an IRR of 24.4 percent over a mine life of 20 years.

25.1.2 Geology

The Project deposits occur in a north-south zone that coincides with the western and southern part of the McDermitt caldera moat sedimentary rock. Lithium mineralization occurs in lithium-bearing claystone minerals observed in thick, apparently continuous accumulations.

Most of the moat sedimentary rocks drilled in the Thacker Pass basin contain anomalously high lithium contents (> 1000 ppm). Intervals that consist mostly of ash have a lithium content of less than 800 ppm whereas intervals dominated by claystone contain more lithium (>1,000 ppm). Many intervals have a very high lithium content (>4,000 ppm). Intervals with extreme lithium contents (>8,000 ppm) occur locally. There is no change in lithium content across the boundary between oxidized and unoxidized rock; however, the highest lithium grades generally occur in the middle and lower parts of the sedimentary rock section.

25.1.3 Mining

An evaluation of the potential surface resource for the mineralized deposit was performed.

Potentially mineable pit shapes were identified using a Lerchs-Grossman analysis performed with the GEMS™ Whittle pit optimization software and the Kings Valley mineral resource model.

Separate mine plans were developed for the Case 1 and Case 2 scenarios using the same input parameters. The Case 2 pit is an expanded version of Case 1. Ultimate pit shells were designed and phases were generated to determine what would extract the ore in a logical sequence. Design cutoff grades higher than the economic breakeven were applied in order to achieve a nominal ore grade of 0.40 percent Li. A fixed cutoff grade was used for Case 1 for all production years and variable cutoff grades were used for Case 2.

Tetra Tech developed the Case 1 and Case 2 mine schedules separately. However, WLC has the option to start the Project using the Case 1 mine schedule and expand to the Case 2 mine schedule at any time. The additional waste stripping outlined in the Case 2 mine schedule would be delayed until the decision to increase production is made. At that point, the additional waste would most likely be moved with contractors at an additional expense.

Waste storage facilities were designed to incorporate backfill and minimize the total pit/waste storage footprint.

Overburden would be extracted with hydraulic excavators and end-dump haul trucks. Ore would be mined with a Wirtgen surface miner and transported to the ore stockpiles in end-dump haul trucks.

Production schedules for both Case 1 and Case 2 scenarios were generated for a mine life of 21 and 20 years respectively.

25.1.4 Process

WLC has continued process development, including bench and pilot size programs for major unit operations, and verification for lithium extraction from the Kings Valley clay deposits and advanced the previous work reported in the PAEE (URS 2010). The process was originally based on the USBM work in the McDermitt caldera reported in 1988.

Following size reduction, the thermal ore preparation process involves calcining the ore mixed with anhydrite and dolomite to convert the silicates to sulfates for leaching. Recoverable metals include lithium, potassium and sodium. The calcine is leached in water recovering the sulfates to solution.

The wet recovery process includes evaporation and crystallization stages to recover potassium and sodium as sulfates along with lithium as a carbonate. The Case 1 scenario has a production target of a nominal 13,000 tpy LCE. For Case 2, the first three years of production are the same as Case 1, doubling to a nominal 26,000 tpy LCE in Year 4.

The overall recoveries from the WLC process are listed below:

Lithium, 87.2%;

Potassium, 77.7%

Sodium, 82.7%

Leach residue from the third leach tank would be filtered and conveyed to a dry stack tailings storage facility.

25.1.5 Environmental

WLC has all necessary federal and state permits and approvals to conduct the current mineral exploration activities, operation of the Kings Valley Clay Mine, and other project activities including groundwater monitoring, at the Project site.

WLC has completed baseline studies for geochemistry, vegetation; wildlife (including extensive studies for the Greater Sage-grouse); surface and groundwater quality and quantity; wetlands and waters of the

U.S.; seep and springs; soils; cultural resources; noise; visual analysis; weather monitoring; and other issues specific to the Project area. The collected baseline study data will support the overall permitting and approval process for the proposed Project, and the completion of the NEPA environmental documentation program.

WLC has an ongoing, proactive relationship with the BLM, the NDEP BMRR and other appropriate federal, state and local regulatory agencies. Required permits and approvals to put the proposed Project into production have been identified. A preliminary Project schedule for the completion of the permitting and approval process is presented in Figure 20.1 of this report.

25.1.6 Hydrologic

Detailed surface and groundwater hydrologic studies have been performed by WLC to assess the potential impacts from the Project on existing resources and determine water management strategies. According to Schlumberger (2013):

- Dewatering rates between 200 and 270 gpm in claystone is sufficient to dewater ahead of mining, and supply the mine with water for operations.
- Additional water supply pumping of 80 gpm is sustainable in basalt and claystone units east of the project area.
- Well yields are anticipated to decline over time as saturated screen intervals are reduced due to dewatering.
- Significant drawdown, defined as 10 foot of drawdown at the end of mining, will be limited to an approximately one-mile radius beyond the project area.
- No water quantity impacts to other water users are anticipated.
- Four springs are within the 10 foot drawdown contour. Spring SP-001 is within the pit boundary and will be impacted by mining operations. Impacts to spring flow in springs SP-002, SP-010, and SP-011 are uncertain due to the potential perched nature of the springs.
- A pit lake is predicted to form in the west pit after mining cessation and recovery to an elevation of 4,892 feet. Water levels within the backfill in the southeast pit more slowly recover to an elevation of 4,894 feet at 100 years post-closure. Surrounding groundwater levels recover at a similar rate as the backfill, reaching 95% recovery approximately 55 years after closure.
- The pit lake water balances indicates that the ultimate pit will have a component of flow through into the surrounding bedrock. The flow-through rate is estimated to be 5.5 gpm at 100 years post-mining. Groundwater flow out of the pit lake is generally verifically downward from the bottom of the pit and towards the southeast. During the early filling, no flow through occurs as all the incoming flow fills the pit lake. Flow through rates increase as equilibrium is approached, with an estimated maximum flow-through rate of 5.5 gpm.

- The maximum drawdown extent was simulated to occur 12 years after mining ceased. At this time, the cone of depression is shallower and slightly wider than immediately after mining (~6 miles in diameter). The largest amount of predicted drawdown was greater than 200 feet, occurring in the west pit's northern face near the location of groundwater monitoring wells PH-1 and WSH-17.

25.1.7 Reclamation

The Project mining operation would be amenable to industry-standard reclamation techniques and permit requirements.

The partially backfilled pit would be left in the final configuration with the pit slopes designed for long-term stability. The waste rock storage facility would be regraded and capped with stockpiled growth medium and vegetated. The tailings storage facility would also be capped with growth medium and vegetated.

25.1.8 Social

Employees for the operation could be bussed from Winnemucca, the likely source for the majority of manpower, eliminating the need for a mancamp at the site. Winnemucca is a mining hub in northern Nevada with established infrastructure.

25.1.9 Economic Analysis

Cash flow analyses were performed for the Case 1 and Case 2 scenarios using capital and operating costs, and revenues from the mine production schedules.

Capital costs were estimated within an overall accuracy of ±25 percent. The majority of the mine equipment capital was estimated from vendor quotations and the *Mine and Mill Equipment Costs: An Estimator's Guide*. Process equipment capital was estimated from vendor quotations, material takeoffs and a percentage for electrical, instrumentation, piping and HVAC. Tailings and water management capital was estimated using the RS Means Heavy Construction Cost Data Handbook and recent vendor estimates. Owner's costs were provided by WLC.

Operating cost estimates include labor, equipment operation, power, reagents, coal, water and maintenance parts and support.

25.1.10 Risks

All mining projects are sensitive to commodity prices, both consumables and products.

Capital estimates are directly tied to the price of steel. Fluctuations in the supply and demand may affect the prices quoted.

Changes caused by the ongoing hydrology studies may have an impact on capital and operating costs related to pit slope stability, waste storage facility placement, construction and other factors.

There is currently no allowance for drilling and blasting of the waste material. Due to the presence of basalt in the ore body, drilling and blasting may be required for small volumes of material. Drilling and blasting would increase the operating costs due to the requirement of contract drilling and blasting.

The process technology has not been commercially demonstrated.

The Project requires the acquisition of numerous permits and approvals from various regulatory agencies. Delays within this process may have an impact on costs.

25.1.11 Opportunities

The Project area is host to a variety of specialty clays that could be recovered and marketed. These clays include bentonite and higher-value clays for cosmetics. Processing of these clays to recover and produce saleable products is straightforward; however, a resource defining the amount of these clays has not been completed, precluding inclusion of the potential revenue from this study.

25.2 STAGE II

25.2.1 Geological Setting

The Kings Valley lithium deposits occur within sedimentary and volcano-sedimentary rocks in the moat of a resurgent caldera. The extent and nature of the host rocks is well documented and understood. At the present time, five areas of significant lithium mineralization have been identified: Stage I Lens (PCD), Stage II Lens (South), Stage III Lens (South Central), Stage IV Lens (North Central), and Stage V Lens (North). In each of these areas hectorite, a lithium-bearing clay mineral occurs in thick, apparently continuous accumulations.

25.2.2 Tenure

Based on the records provided, GSI concludes that WLC has documents that support their claim to the rights to the lithium mineralization and that they have documents showing that all appropriate permits for exploration have been obtained.

25.2.3 Deposit Type and Mineralization

There are no analogous deposits in operation worldwide. The hectorite deposits at Hector, California have similar mineralogy, but the geological setting is significantly different. The Kings Valley deposits are believed to have formed by hydrothermal alteration of layered volcaniclastic sedimentary rocks.

Mineralization consists of layered beds of lithium-bearing clay-rich volcaniclastic sedimentary rocks. The beds exhibit very good geological continuity over kilometers with drill spacings on the order of 500m. The thickness of mineralization varies from less than a meter to more than 90m with typical intercepts of about 30m. The extent of mineralization is well known.

26.0 RECOMMENDATIONS

26.1 GENERAL

The findings of this preliminary feasibility study provide a basis for advancing the Project toward a feasibility level study. The following recommendations for the Project should be considered:

26.2 TOPOGRAPHY

GSI recommends improving the topographic map currently available, since its level of detail is insufficient for pre- and feasibility engineering work. Estimated cost: US$60,000.

26.3 DRILLING

Additional infill drilling is required for the Stage II Lens area to advance it into a detailed project definition and feasibility-level resource estimation. A nominal grid of 160 m x 160 m should ensure that most resources are classified as indicated. For measured resources, a drill hole spacing of 80 m x 80 m would be required. Estimated cost: US$400,000 for drilling 20 holes 1,800m total and US$70,000 for trenching

26.4 DENSITY DATABASE VERIFICATION

GSI recommends for the Stage II Lens that at least 50 additional dry density data be acquired from the moat sediments to better refine the in situ dry density estimate. There should also be an additional 10 density samples taken from alluvial material, as well as another 10 from the bedrock material, to improve the estimate of waste material to be moved. The density should be determined using the wax-coated procedure. Estimated cost: US$60,000

26.5 RESOURCES

GSI recommends that the Stage II Lens area be advanced to the next stage of drilling, and incorporated along with the Stage I resource. Estimated cost: US$20,000

26.6 GEOTECHNICAL

Tetra Tech recommends a geotechnical investigation within the dry-stack TSF and the waste rock storage facility footprints should be completed. Estimated cost: US$150,000

26.7 HYDROLOGY

Tetra Tech recommends an in-depth site-wide water balance should be completed. Estimated cost: US$50,000

26.8 MINING

Tetra Tech recommends the following for advancement of the Stage I lens:

- A detailed hydrogeology study to determine the water regime and movement within the affected area should be completed. This study should be coupled with an updated geotechnical analysis to determine the effect of water flow within the pit and waste dump upon slope stability. Estimated cost: US$1.5 million

- Overburden compressive strength testing of the various overburden units to determine the application of free digging across the mine area should be performed. Estimated cost: US$10,000

- The geological model should be updated to include the inter-bedded ash layers to support a more detailed mining schedule to maximize grade and minimize dilution within the mine plan. Estimated cost: US$15,000.

GIS recommends the following for advancement of the Stage II lens:

At the present time, there are no mineral reserves at the Stage II Lens deposit; mine optimization and design, including collecting and analyzing geotechnical data, would be required for future project advancement. Estimated cost is US$40,000 for geotechnical studies and US$20,000 for mine optimization.

26.9 METALLURGICAL TESTING

GIS recommends Metallurgical samples form the Stage II Lens be collected and tested against the process flowsheet, recoveries, and to quantify estimated process operating costs. An assessment of the possible differences between the metallurgical responses of material from the Stage I Lens and the Stage II Lens should be made. Estimated cost: US$360,000.

26.10 PROCESS

Tetra Tech recommends the following for advancement of the Stage I lens:

- Confirm the process flowsheet and metallurgical parameters through a pilot test program. Estimated cost: $5 - 8 million.

- Confirmation of the required addition of anhydrite and dolomite per tonne of ore is recommended for the next phase of the project; cost included in pilot test program.

- Verification of the suitability of coal for direct-firing the calciner is recommended for the next round of testwork; cost included in pilot test program. Analysis of the calciner off-gas from the coal-fired calcining tests is recommended to provide a basis for treatment requirements; cost included in pilot test program.

- Identification of corrosive gases from the calcination process in the testwork to date suggests future testwork to quantify the content and volume of gas flows for materials

selection in calcining equipment and gas-handling ductwork; cost included in pilot test program.

- Confirmation of the revised flowsheet that does not incorporate the acidification step is recommended in the next phase of testwork; cost included in pilot test program.

26.11 QA-QC

Tetra Tech recommends the following for advancement of the Stage I lens:

- Future drilling programs should incorporate the following protocols to ensure continued quality assurance and control
- Send a small number of coarse duplicates for assay to verify results.
- Send a small number of coarse duplicate and pulp to a secondary laboratory for assay analysis to verify results.
- Add a very low grade standard (ranging between 1,500 and 2,500 ppm lithium) to the QA-QC protocol.
- Add a potassium standard from clay that originates from the Stage I Project area.
- Institute the use of formal chain-of-custody documentation to accompany all samples to the laboratory.

Costs to add these QA-QC protocols to the Project are included in general costs.

GIS recommends the following for advancement of the Stage II lens:

- Site specific standards be produced for the Kings Valley Li project from materials found in the project area. These standards should cover the expected grade ranges for Li, F, Na, and K.
- A routine insertion of coarse duplicates (-10 #), in addition to the standards, pulp duplicates, and blank samples to monitor accuracy, precision and contamination. GSI also recommends that check samples be sent to a second laboratory.

26.12 ENVIRONMENTAL

Tetra Tech recommends the following for advancement of the Stage I lens:

- A multi-agency permitting and approval process should be completed by WLC for construction, operation and closure of the Project in accordance with all applicable federal, state and local regulations. This program would include the acquisition of numerous permits and approvals from various regulatory agencies. Estimated cost: $US402,000
- It is recommended that representatives of WLC continue to hold regular meetings with personnel from the various regulatory agencies including the Winnemucca BLM, the NDEP

BMRR, Humboldt County, and other federal, state and local agencies as appropriate, to provide updates on Project activities to streamline the time required to complete the overall permitting and approval process. These meetings, which can be held as individual meetings or a joint meeting, would provide an open forum to discuss overall progress and any issues or concerns that may arise during development activities. Estimated cost: $US10,000

- It is also recommended that WLC continue to hold regular public information meetings in the regional area, including the City of Winnemucca and the community of Orovada, to provide regular Project updates and to continue developing positive public support. Estimated cost: $US5,000
- Finally, WLC should continue to maintain regular contact with local, state and federal elected officials. The value in keeping elected officials updated on Project activities cannot be stressed enough in terms of potential assistance and support that might be needed to assist the agencies and WLC in completing the overall permitting and approval process for the Project. Estimated cost: $US10,000

26.13 Drystack Tailings Storage Facility

- Additional laboratory testing on tailings characteristics such as hydraulic conductivity and shear strength should be conducted. Estimated cost: US$30,000
- A field-scale test section on tailings should be completed to determine trafficability. Estimated cost: US$20,000
- Barren purge containment pond solution evaporation characteristics should be investigated. Estimated cost: US$20,000
- Additional work is required on the liner system design. Estimated cost: US$50,000

26.14 Feasibility and Economic Evaluation

It is recommended that after the demonstration plant for the Lithium production is complete, and the resources of the Stage II Lens are further defined, that a feasibility level study be performed and Technical Report be prepared on the Project. Subsequent to the Stage I PFS analysis, the Project has incurred royalty obligations that were not included in the Stage I economic estimate. The economic analysis of the Project has not been updated to account for these royalties. Estimated cost: US$2,250,000

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27.2 STAGE II

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27.3 TECHINCAL SUPPORT DOCUMENTS

Technical Reference A: Stage I PFS Geotechnical Investigation and Addendum, AMEC

Technical Reference B: Stage I PFS Mining Methods and Figures

Technical Reference C: Stage I PFS Basis of Estimate

Technical Reference D:	Stage I PFS Tailings Management
Technical Reference E:	Stage I PFS Metallurgical Information and Equipment List
Technical Reference F:	Stage II Resource Estimate Quality Assurance-Quality Control - Additional Graphs
Technical Reference G:	Stage II Resource Estimate Additional Histograms and Other Graphs - Exploratory Data Analysis
Technical Reference H:	Stage II Resource Estimate Resource Model Validation Plots

28.0 QUALIFIED PERSONS

28.1 QUALIFICATIONS OF CONSULTANTS

The Consultants preparing this Technical Report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants has any beneficial interest in WLC. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between WLC and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions.

This Technical Report was prepared by the following QPs, Certificates and consents of which are contained herein:

Name	Title, Company	Responsible for Sections
Mario E. Rossi, FAusIMM	Principal Geostatistician GeoSystems International, Inc.	5.2, 6.2, 9.2, 10.2, 11.2, 12.2, 13.2, 14.2, 20.2, 25.2 and portions of 1, 2, 7, 8, 18 and 26
Edwin C. Lips, P.E.	Principal Mining Engineer, Tetra Tech	15, 16, and portions of 1, 2, 21, 22, 23, 25, and 26
Timothy J. Carew, P. Geo	Principal, Reserva International, LLC	7, 8, 9, 10, 11, 12, 14 and portions of 1, 6, 15, 23, 25 and 26
D. Erik Spiller, MMSA QP	Principal Metallurgist Tetra Tech	3, 4, 5, 13, 17, 18, 19, 24 and portions of 1, 2, 6, 21, 22, 25, and 26
Vicki J. Scharnhorst, P.E.	Principal Consultant Tetra Tech	20 and portions of 1, 6, 24, 25 and 26

CERTIFICATE OF QUALIFIED PERSON

Vicki J. Scharnhorst, P.E., LEED AP
Principal Consultant
Tetra Tech
350 Indiana Street, Suite 350
Golden, Colorado 80401
Telephone: 303-217-5700
Facsimile: 303-217-5705
Email: vicki.scharnhorst @tetratech.com

I, Vicki J. Scharnhorst, P.E., LEED AP of Golden, Colorado, do hereby certify:

- I am a Principal Consultant with Tetra Tech with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.
- This certificate applies to the technical report entitled "Updated NI 43-101 Technical Report Kings Valley Property Humboldt County, Nevada", that has an effective date of April 30, 2014 (the "Technical Report") prepared for Western Lithium USA Corporation (the "Company").
- I am a graduate of Kansas State University with a Bachelor of Science degree in Civil Engineering (1982). My relevant experience includes 30 years of civil engineering on large water resources projects inclusive of water quality programs, environmental impact studies, permitting and civil works. I am a licensed Engineer in the states of Nevada, Michigan, Missouri and Colorado; a water right surveyor in the State of Nevada; a LEED Accredited Professional with the U.S. Green Building Council; and have served on the Nevada State Board of Professional Engineers and Land Surveyors. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Section 20. I share responsibility with others for Sections 1, 6, 24, 25 and 26 of the Technical Report.
- I am independent of Western Lithium USA Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated May 9, 2014 at Golden, Colorado.

Vicki J. Scharnhorst, P.E., LEED AP
Principal Consultant
Tetra Tech



05-09-14

CERTIFICATE OF QUALIFIED PERSON

Edwin C. Lips, P.E.
Principal Mining Engineer
Tetra Tech
350 Indiana Street, Suite 350
Golden, Colorado 80401
Telephone: 303-217-5700
Facsimile: 303-217-5705
Email: ed.lips@tetratech.com

I, Edwin C. Lips, P.E., of Golden, Colorado, do hereby certify:

- I am a Principal Mining Engineer with Tetra Tech with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.
- This certificate applies to the technical report entitled "Updated NI 43-101 Technical Report Kings Valley Property Humboldt County, Nevada", that has an effective date of April 30, 2014 (the "Technical Report") prepared for Western Lithium USA Corporation (the "Company").
- I am a graduate of Montana Tech, (Bachelor of Science degree in Mining Engineering, 1982). I am a licensed Professional Engineer in good standing in the State of Arizona, license number 47670. My relevant experience is that I have practiced my profession as a mining engineer continuously since graduation, for a total of 29 years. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I am responsible for Sections 15 and 16. I share responsibility with others for Sections 1, 2, 21, 22, 23, 25 and 26 of the Technical Report.
- I am independent of Western Lithium USA Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 9th day of May, 2014 at Golden, Colorado.

Edwin C. Lips, P.E.
Principal Mining Engineer
Tetra Tech



EXPIRES 3/31/17

CERTIFICATE OF QUALIFIED PERSON

D. Erik Spiller
Principal Metallurgist
Tetra Tech
350 Indiana Street, Suite 350
Golden, Colorado 80401
Telephone: 303-217-5700
Facsimile: 303-217-5705
Email: erik.spiller@tetratech.com

I, D. Erik Spiller of Golden, Colorado, do hereby certify:

- I am a Vice President and Principal Metallurgist with Tetra Tech with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.

- This certificate applies to the technical report entitled "Updated NI 43-101 Technical Report Kings Valley Property Humboldt County, Nevada", that has an effective date of April 30, 2014 (the "Technical Report") prepared for Western Lithium USA Corporation (the "Company").

- I am a graduate of the Colorado School of Mines, (Bachelor of Science degree in Metallurgical Engineering, 1978). I am a Qualified Professional (QP) member of the Mining and Metallurgical Society of America (MMSA #01021QP). In addition, I am a Registered (QP) member of Society for Mining, Metallurgy, and Exploration, Inc. (SME #3051820RM). My relevant experience is that I have worked as a metallurgical engineer in the mineral resource industry for more than 40 years. During this career I held responsible positions in process research, process development, engineering, and senior management. In addition, I am an appointed Research Professor (8 years) in the Metallurgical and Materials Engineering Department at the Colorado School of Mines, where I lecture in mineral processing and extractive metallurgy and direct graduate students conducting metallurgical research in my area of expertise.

- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

- My most recent personal inspection of the Property was February 2, 2011.

- I am responsible for Sections 3, 4, 5, 13, 17, 18, 19, and 24. I share responsibility with others for Sections 1, 2, 6, 21, 22, 25, and 26 of the Technical Report.

- I am independent of Western Lithium USA Corporation as defined by Section 1.5 of the Instrument.

- I have no prior involvement with the Property that is the subject of the Technical Report.

- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

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

Signed and dated May 9, 2014 at Golden, Colorado.

D. Erik Spiller
01021QP
Mining & Metallurgical Society of America

D. Erik Spiller
Principal Metallurgist
Tetra Tech

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CERTIFICATE OF QUALIFIED PERSON

I, Mario E. Rossi, do hereby certify that:

1. I am currently employed by GeoSystems International, Inc., as the Principal Geostatistician at:

   2385 NW Executive Center Dr. Suite 100
   Boca Raton, FL 33431, USA

2. This certificate applies to the technical report entitled "*NI 43-101 Technical Report Kings Valley Property Humboldt County, Nevada*", that has an effective date of April 30, 2014 (the "**Technical Report**") prepared for Western Lithium USA Corporation (the "**Company**").

3. I graduated with a Masters of Science in Geostatistics from Stanford University, California, USA, in 1988. In addition, I have obtained a Mining Engineering degree from the Universidad Nacional de San Juan, Argentina, in 1985.

4. I am a member of the Canadian Institute of Mining and Metallurgy (CIM), a fellow of the Australian Institute of Mining and Metallurgy (AusIMM, Australia), a member of the Society of Mining Engineers (SME, USA), and a member of the International Association of Mathematical Geology (IAMG).

5. I have worked as a geostatistician for 26 years since my graduation from university and as a mining engineer for an additional 2 years. My experience includes mineral resource and reserves estimation, audits, reviews, and due diligence studies. I have worked in over 80 different mining projects at various stages of development around the world, including industrial minerals such as nitrate, iodine, lithium, potassium, and graphite deposits; precious and base metal deposits (Au, Ag, Cu, Fe, Pb, Zn, and others); and sedimentary deposits (uranium, iron, limestone, coal). I have published about 50 papers in the area of geostatistics and mineral resource estimation and simulation and I am the co-author of the book "Mineral Resource Estimation", Rossi, M., and Deutsch, C., ed. Springer, January 2014.

6. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects ("**NI 43–101**").

7. I visited the Kings Valley property on February 24, 2010.

8. I am responsible for the Stage II resource estimate, described in sections 1.4.2, 1.14.2, 2.2, 2.5.2, 3,2, 5.2, 7.2, 8.2.3, 10.2, 11.2, 12.2, 13.2, 14.2, 15.2, 21.2, 24.2, 26.2, and the Recommendations in Section 27 of the Technical Report that pertain to Stage II.

9. I am independent of the Company as independence is described by Section 1.5 of NI 43–101.

10. I have not had prior involvement with the property that is the subject of the Technical Report.

11. I have read NI 43–101 and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument.

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

Dated this 9th day of May, 2014



Mario E. Rossi

CERTIFICATE OF QUALIFIED PERSON

Timothy J. Carew, P. Geo.
Principal
Reserva International, LLC
P.O. Box 19848
Reno, NV 89511 USA
Telephone: 775-853-2227
Email:

I, Timothy J. Carew, P. Geo., of Reno NV, do hereby certify:

- I am the Principal of Reserva International, LLC, P.O. Box 19848, Reno, NV, 89511, USA.
- I graduated with the following degrees:
  - University of Rhodesia, B.Sc. Geology 1973
  - University of Rhodesía, B.Sc. (Hons) Geology 1976
  - University of London (RSM) M.Sc. Mineral Prod. Management 1982
- I am a member in good standing of the following professional associations:
  - Association of Professional Engineers and Geoscientists of British Columbia (Professional Geoscientist 19706)
  - lnstitute of Mining, Metallurgy and Materials (Professional Member 46233)
- I have worked in mining geology and engineering for over 35 years since my graduation from the University of Rhodesia.
- As a result of my experience and qualifications and affiliation with professional associations I am a qualified person as defined in National Instrument 43-101 Instrument of Disclosure for Mineral Products (NI 43-101). I have over 30 years of mineral resource estimation experience in both metallic, and non-metallic, deposits and have specific experience in estimation of other metals (Ga) in the McDermitt Caldera area.
- I have read the definition of "Qualified Person" set out in National lnstrument 43-101 ("NI 43-101 ") and certify that by reason of my education, affiliation with professional associations and past relevant work experience, including other rare-earth experience in the McDermitt Caldera area, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
- I am responsible for the preparation of Sections 7, 8, 9, 10, 11, 12, 14, and parts of Section 1, 6, 15, 23, 25 and 26, of the technical report titled “Updated NI 43-101 Technical Report, Kings Valley Property, Humboldt County, Nevada, Western Lithium USA Corporation that has an effective date of April 30, 2014” (the “Technical Report”) relating to the Kings Valley project. I have visited the Kings Valley property on January 5, 2011 for a total of one day.

- Prior to being retained by Western Lithium Corporation in 2010, I have not had prior involvement with the property that is the subject of the Technical Report.

- As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

- I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
- I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Signed and dated May 9, 2014 at Golden, Colorado.



Timothy J. Carew. P.Geo