Preliminary Economic Assessment NI 43-101 Technical Report

Long Valley Project

Mono County, California, USA

Effective Date: September 21, 2020
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Prepared for:

KORE Mining Ltd.

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This Technical Report on the Long Valley gold project is submitted to KORE Mining Ltd. and is effective September 21, 2020.

The Qualified Persons and Responsible Report Sections follow:

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LIST OF ACRONYMS AND ABBREVIATIONS

µm	micron
AA	atomic absorption spectrometry
ADR	Adsorption-Desorption Recovery
Ag	silver
Amax	Amax Gold Inc.
American Assay	American Assay Labs
Au	gold
Battle Mountain	Battle Mountain Gold
Behre Dolbear	Behre Dolbera & Company Inc.
BLM	U.S. Department of the Interior, Bureau of Land Management
CDCA	California Desert Conservation Area
Chemex	Chemex Labs
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CPT	corrugated plastic tubing
FLPMA	Federal Land Policy and Management Act
ft^2	square feet
ft^3/ton	cubic feet per ton
g/L	grams per liter
g Au/t	grams gold per metric tonne
g/tonne	grams per tonne
gpm/ft^2	gallons per minute per square foot
HCFZ	Hilton Creek Fault Zone
HDPE	high density polyethylene
HLF	heap leach facility
IRR	Internal Rate of Return
kg/t	kilograms per tonne
KORE	KORE Mining Ltd.
lbs	pounds
lbs/ft^3	pounds per cubic foot
LDRS	leach detection and recovery system
LLDPE	linear low-density polyethylene
lph/m^2	liters per hour per square meter
L/t	liters per tonne
LOM	life of mine
m^3	cubic meters
MDA	Mine Development Associates, Inc.
mm	millimeter
MRA	Mine Reserves Associates
MT	magnetotelluric

NaCN	sodium cyanide
NOI	Notice of Intent
NPV	Net Present Value
NSR	net smelter return
OM	Optical Microscopy
oz Ag/ton	troy ounces silver per short ton
oz Au/ton	troy ounces gold per short ton
oz/ton	troy ounces per short ton
PEA	Preliminary Economic Assessment
PLS	pregnant leach solution
PoO	Plan of Operations
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
Royal Gold	Royal Gold, Inc.
RC	reverse circulation drilling method
ROM	run-of-mine
SEM	Scanning Electron Microscopy
SEZ	Southeast Zone
Standard	Standard Industrial Minerals, Inc.
t	metric tonne
ton	Imperial short ton
tpd	tons per day
Vista	Vista Gold Corp.
WRSF	waste rock storage facility

1.0 SUMMARY

The effective date of this report is September 21, 2020.

1.1 Property Description and Ownership

The Long Valley property is located in the Inyo National Forest, about 7 miles east of the town of Mammoth Lakes, Mono County, California. The Long Valley gold property consists of 95 contiguous, unpatented mining claims that cover an area of approximately 1,800 acres. All of the claims are located in all or portions of Sections 13, 14, 15, 22, 23, 24, 25, and 26, T3S, R28E, Mount Diablo Base and Meridian.

KORE Mining Ltd. (KORE) purchased the claims from Vista Gold California LLC, a subsidiary of Vista Gold Corp. ("Vista"), in March 2017 for $1,350,000, payable as follows:

- US$350,000 at closing (paid on March 31, 2017)
- US$500,000 on or prior to the 30th day after commencement of commercial production
- US$500,000 on or prior to the 12-month anniversary of the commencement of commercial production.

The property is subject to two royalty agreements:

- a 1.0% net smelter return royalty payable to Royal Gold, Inc. ("Royal Gold")
- a 0.5% to 2.0% net smelter return royalty based on the quarterly gold price payable to Vista.

1.2 Accessibility and Climate

The Long Valley property is located about seven miles to the east of the town of Mammoth Lakes and about 45 miles by road northwest of the town of Bishop, California. Both towns are connected by U.S. Highway 395, which passes a few miles west of the property. Access to the property from the highway is via a series of graded gravel roads.

The climate is semi-arid and moderate, with high temperatures in the summer generally in the 80 °F range and winter highs generally in the 30 to 40 °F range. Winter temperatures can be below 0 °F. Precipitation at the property totals about 20 to 25 inches per year, divided between winter snows and summer thunderstorms. The evaporation potential greatly exceeds the precipitation on an average annual basis, so the area is one with a negative water balance. Snow depths in winter are generally less than two feet on the property, and the overall climate should permit operations year around.

1.3 Exploration and Mining History

Gold was first recognized on the property by Standard Industrial Minerals, Inc. ("Standard") in the early 1980s as being present in small amounts in and around their kaolinite clay mining operations. Standard optioned the property to Freeport Minerals ("Freeport") in 1983, and Freeport drilled about 80 shallow reverse circulation (RC) holes in mostly the North and South zones during 1983 and 1984. After Freeport dropped the property, Standard drilled 24 shallow rotary drill holes in 1986.

Royal Gold acquired an option on the property from Standard and drilled 52 RC holes during 1988. In 1990, Battle Mountain Gold ("Battle Mountain") and Royal Gold formed a joint venture to further explore and develop the property. During 1990 and 1991, Battle Mountain completed geologic mapping, geochemical sampling, and geophysical surveying of the area and drilled 59 RC holes. Battle Mountain dropped out of the joint venture in 1993, but Royal Gold continued exploration of the property.

During the period of 1994 through 1997, Royal Gold drilled 615 RC and 10 core holes at the Long Valley property. During this time, Royal Gold also completed metallurgical investigations and preliminary engineering studies, including resource estimations, and initiated baseline-type environmental studies of the biological, water, and archeological resources of the area.

In mid-1997, Amax Gold Inc. ("Amax") performed extensive due diligence investigations in consideration of forming a joint venture with Royal Gold, with the intent of placing the property into production. Amax's work included drilling 46 RC holes and 10 core holes, as well as conducting extensive re-assay and check assay work and re-logging of older holes. Amax elected not to proceed with the formation of the joint venture because of the continued deterioration of the gold price.

There has been no further drilling on the property since 1997. Royal Gold turned the property back to Standard in 2000. In 2003, Vista signed a purchase option agreement with Standard for the Long Valley project and completed the purchase of the claims in January 2007. Vista maintained the claims in good standing but conducted no exploration on the property from 2003 until their sale of the property to KORE in 2017. The only exploration KORE has conducted on the property to date is a Spartan magnetotelluric survey in December 2017 and additional geophysical surveys in 2019 and 2020.

1.4 Geology and Mineralization

The Long Valley deposit is contained entirely within the late Pleistocene-age Long Valley caldera, which has been dated at about 760,000 years old. The caldera is an elongated east-west oval depression measuring some 10 by 19 miles and is related to the eruption of the Bishop Tuff, which is mostly covered by younger rocks within the caldera.

The Long Valley gold deposit is located near the center of the caldera and is underlain by lithologic units related to the caldera formation and subsequent magmatic resurgence. These rocks were deposited in a lacustrine setting within the caldera and consist of varved siltstones interbedded with fine- to coarse-grained ash- and pumice-fall layers, conglomerates and debris-flow deposits, as well as more local deposits of intercalated silica sinter. The lacustrine volcaniclastic sequence was intruded by a large body of "resurgent" rhyolite. All of these lithologies have been altered and/or mineralized to variable degrees.

The north-south trending Hilton Creek fault zone appears to define the eastern limit of exposure of the resurgent rhyolite within the central part of the Long Valley caldera and extends outside the caldera to the south. This fault system is thought to control the distribution of gold mineralization in the Long Valley deposit. Offset along this fault appears to be variable and suggests that fault activity along this zone may be episodic in nature. Active hot springs, earthquakes, and very recent volcanism suggest that the area is still geologically active.

Gold and silver mineralization at Long Valley appears to fall under the general classification of an epithermal, low sulfidation type deposit. Several areas, termed the North, Middle, South, Southeast, and Hilton Creek zones, are mineralized with low grades of gold and silver; the North Zone lies outside of the current property boundary. The mineralized zones are generally north-south trending, up to 8,000 feet in length, and with widths ranging from 500 feet to 1,500 feet. The tabular bodies are generally flat-lying or have a shallow easterly dip. Mineralization is typically from 50 to 200 feet thick and, in the South and Southeast zones, is exposed at (or very near) the surface. The top of the Hilton Creek zone is generally covered by 20 to 50 feet of alluvium. The vast majority of the mineralization discovered to date is located in the Hilton Creek zone.

Drilling is widely spaced in and between the North, Middle, and South zones, and it may be possible that, with additional drilling, these zones may be shown to be contiguous with the better-defined zones to the south.

Royal Gold generally defined the base of the oxidized zone as the last occurrence of oxide mineralization. Sulfide mineralization and mixed oxide-sulfide material also occurs above this boundary. The sulfide/oxide boundary occurs at depths between 150 and 250 feet and is often coincident with or slightly above the current water table.

Gold and silver mineralization is quite continuous throughout the zones and is well defined at grades greater than approximately 0.010 ounces (oz) gold (Au)/ton. Within the continuous zones of low-grade (+0.010 oz Au/ton) gold mineralization are numerous zones of higher-grade mineralization (+0.050 oz Au/ton), particularly in the Hilton Creek zone, which may relate to zones of enhanced structural preparation. Mineralized zones are typically correlated with zones of more intense clay alteration and/or silicification.

1.5 Drilling, Sampling and Data Verification

Freeport, Standard, Royal Gold, Battle Mountain, and Amax drilled the Long Valley project between 1983 and 1997; no drilling has been conducted since 1997. The database contains 896 drill holes, totaling 268,275 feet. Eight hundred of the holes were drilled using RC methods; 20 were core holes. Collar coordinate information is missing for seven of the drill holes.

Gold has been primarily analyzed by fire assay, with grade determinations by atomic absorption. The exceptions are analyses done by Freeport, who completed its assays by acid digestion. The 10 core holes drilled by Amax were twin holes to check nearby RC drilling. Overall, the comparison showed good agreement between the core and RC sample assays; however, individual sets of twin hole data varied considerably. Numerous check assays were completed on sample pulps and sample rejects, all of which indicated good agreement with the original assays.

The QP authors have individually verified the data used in this report to determine its adequacy for the purposes used in the report. The data verification procedures are described in Section 12.

1.6 Mineral Processing and Metallurgical Testing

A moderate amount of metallurgical testing was completed on samples from the Long Valley property from about 1989 through 1997. None has been conducted since 1997. The test work was generally well done, and the results were fairly consistent across laboratories.

The test work supports that the oxide materials are generally free milling and amenable to heap leach recovery, and the sulfide materials are more refractory and not suitable for heap leach recovery. Transitional materials fall somewhere in between these two extremes.

Bottle roll tests on oxide samples show an average gold extraction of approximately 76% for the gold and 21% for the silver during cyanide leach tests. These results demonstrate the good leaching characteristics of the gold, and most of the samples gave fairly consistent results through 14 tests performed by three different labs. Bottle roll tests on the mixed oxide-sulfide samples showed an average gold extraction of about 49% and 19% for silver, with considerable variation between individual samples. Bottle roll test results on the sulfide samples also show a wide range of results. Fifteen samples were tested by three different labs, and gold extractions ranged from zero to over 50%. The average recovery for sulfides was 11% for gold and 24% for silver. Tests also show that both gold and silver extractions increase at smaller particle sizes for all classes of material.

The results of column leach tests conducted by both Hazen Research and Kappes Cassiday were generally good. The average gold extraction from all column tests was 85%. Silver extraction was generally low, averaging only 7.6% (with only four of the tests recording silver extraction data). The material tested in the columns was generally classified as oxide type material. Column extractions improved with decreasing particle size from 86% at 76 millimeters (mm) to 93% at 25 mm. Run of mine tests (P_{80} 125 mm) showed lower recovery in most cases, ranging from 63% to 92%, with similar conditions with the exception of agglomeration.

Agglomeration improved recovery and percolation within the column tests. Lime and cyanide consumptions were low for the oxide materials with an estimated heap leach facility (HLF) dosage of 0.05 kilograms per tonne(kg/t) cyanide and 0.19 kg/t lime. Reagent consumptions increase with increasing sulfide grades.

Column rinse testing indicates that the cyanide levels can be effectively reduced by rinsing the heap materials.

The most critical issue for Long Valley is to ensure the proper designation of the oxide, transition, and sulfide materials within the mineralized body so that appropriate gold recoveries can be assigned and the placement of material on the HLF can be correctly monitored.

1.7 Long Valley Mineral Resource Estimate

Current Mineral Resources reported herein for the Long Valley property were estimated by Mine Development Associates, Inc. (MDA) in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards"), and this estimate was completed on July 15, 2020. The block-model gold grades remain unchanged from the 2003 block model. No drilling occurred on the property after 2003, but the block model was updated using density and geologic models based on interpretations completed in 2020 with information from KORE's re-logging of drill-hole cuttings. Silver resources were not estimated. The Mineral Resource Estimate considered a heap leach operation for oxide material, and a plant to recover sulfide and transition material. Pit optimization parameters were developed for the different materials and are summarized in Table 1-1:.

Table 1-1: Pit Optimization Parameters

Item	Units	Parameter
Pit Slope	Degrees	45°
Gold Price	$ per ounce gold	$1,800
Mining	$/ton mined	$1.80
Crushing	$/ton processed	$1.40
Heap Leach	$/ton processed	$1.80
Sulfide Plant	$/ton processed	$10.00
G&A per Ton	$/ton processed	$0.70
Refining Cost	$/oz Au Produced	$5.00
Recovery (Oxide – Less than 150 feet below surface)	% Heap Recovery	80%
Recovery (Transition – 150 to 200 feet below surface)	% Mill Recovery	90%
Recovery-Plant (Sulfide – more than 200 feet below surface)	% Mill Recovery	90%

Gold resources that are contained in an $1,800 per ounce optimized pit were estimated by MDA for the Hilton Creek, Southeast, and South zones and are summarized in Table 1-2: and Table 1-3:.

Table 1-2: Long Valley Gold Resources (Imperial Units)

Material Type	Cutoff (oz Au/ton)	K tons	Indicated oz Au/ton	K ozs Au	Ktons	Inferred oz Au/ton	Kozs Au
Oxide	0.005	35,276	0.018	635	8,997	0.020	180
Transition	0.006	4,026	0.014	56	1,277	0.016	20
Sulfide	0.006	30,914	0.017	526	14,033	0.018	253
Total	variable	70,216	0.017	1,217	24,307	0.019	453

Table 1-3: Long Valley Gold Resources (Metric Units)

Material Type	Cutoff (oz Au/ton)	K tonnes	Indicated g Au/t	K ozs Au	K tonnes	Inferred g Au/t	Kozs Au
Oxide	0.17	32,001	0.62	635	8,162	0.690	180
Transition	0.21	3,653	0.48	56	1,159	0.550	20
Sulfide	0.21	28,045	0.58	526	12,731	0.620	253
Total	variable	63,699	0.58	1,217	22,051	0.650	453

1.8 Mining Method

The Long Valley Project is planned to be mined using conventional open pit mining methods. GRE's mine design and production plan are based on the MDA July 15, 2020 resource model. A Whittle pit optimizer was used to assist with ultimate pit and phase design. The $1,500 pit shell was selected for ultimate design. The bench height was 20 feet, and a 45-degree pit wall was used. The results are summarized in Table 1-4.

Table 1-4: Mine Plan Quantities

Phase	Leachable Material – Indicated (1000s Tons)			Leachable Material – Inferred (1000s Tons)			Waste	Au – Indicated (1000s oz)			Au – Inferred (1000s oz)			Stripping Ratio
	Oxides	Transition	Sulfides	Oxides	Transition	Sulfides		Oxides	Transition	Sulfides	Oxides	Transition	Sulfides	
Phase 1	13,122	1,581	1,178	1,041	362	251	18,675	247	23	42	28	6	12	1.07
Phase 2	10,500	1,041	1,196	1,016	42	199	19,497	181	15	42	21	1	7	1.39
Phase 3	8,345	713	1,514	1,874	368	436	19,296	150	10	58	47	6	21	1.46
Phase 4	3,005	577	157	4,879	433	326	19,033	39	7	3	79	6	8	2.03
Total	**34,972**	**3,913**	**4,045**	**8,810**	**1,205**	**1,211**	**76,501**	**618**	**55**	**145**	**175**	**19**	**47**	**1.41**

1.9 Recovery Method

The Long Valley project would employ open pit mining with a conventional heap leach system on a 365 day per year, 24 hour per day basis. The heap leach will use crushed run-of mine (ROM) material, and the crushed material would be agglomerated with cement and transported to the heap leach via conveyor belt.

The heap leach would consist of a suitable area lined with a containment system. The crushed feed material would be stacked in lifts on the lined pad by a radial stacker. The stacker would be fed by a series of jump or grasshopper conveyors that would be fed from the main overland conveyor from the agglomeration. The lifts would be targeted at 32 feet (10 meters) in height with a total heap height of 328 feet (100 meters). Once a suitable area has been stacked (cell), the cell would be irrigated with dilute cyanide solution. Stacking would continue to advance, and each area irrigated with cyanide solution for a set period (primary leach cycle). The solution leaches gold and silver from the heap materials and would be transported to the recovery circuit as pregnant leach solution (PLS). This PLS would be processed directly in the recovery plant, diverted to a dedicated pond, or recirculated to the heap. The recovery plant would use the Merrill-Crowe system for precious metal recovery as it is predicted that the PLS will contain appreciable silver along with gold. A gold and silver doré bar would be produced for sale to an offsite refinery.

1.10 Project Infrastructure

A limited amount of infrastructure is currently available on site. Power, water, and all other systems necessary for a mining and processing operation will be required. Sufficient water appears to be available on the Long Valley property. Groundwater supplies would be developed to meet the project water requirements. There are no electrical substations at the site. Local labor for mining is available.

1.11 Market Studies and Contract

The primary metal of economic interest for the Long Valley project is gold. Gold has a readily available market for sale in the form of gold doré or gold concentrates. The selected gold price for the PEA is $1,600/oz, which represents the 3-year trailing average, $1,425/oz, weighted by 60% and the 2-year project gold price, $1,860/oz, weighted by 40%. These values were applicable at the time of the effective date of this Technical Report.

1.12 Environmental Studies, Permitting and Social or Community Impact

Due to the property's overlapping jurisdictions of local, state, and national governments, the project must have a plan of operations, county mining and reclamation permit, mining operations permit, reclamation plan and financial assurance. The operating permits will cover waste discharge, air quality, and jurisdictional wetlands and waters. Federal and California regulators encourage a joint environmental impact document that will fulfill the requirements of both governments. The environmental impact study will address use of cyanide and air quality impacts.

Public outreach was undertaken in 1990 by the USFS. Issues of concern raised by the local community included: surface and groundwater hydrology effects, proximity to geothermal spring systems and seismic stability of the area archaeological resources, cyanide use and wildlife, proximity to a fish hatchery, noise and dust and visual resources relative to Highway 395. The issues of concern are expected to be the same today. The project may encounter resistance being located in a region largely valued for passive (hiking, camping, hunting and fishing) and active recreation (skiing and other winter sports) activities with the local economy largely reliant on tourism.

1.13 Capital and Operating Cost

A breakdown of capital and operating costs is shown in Table 1-5.

Table 1-5: Long Valley Capital Costs

Item	Cost (million US$)
Mining and mine infrastructure	$40.6
Heap leach pads and plant	$55.5
Infrastructure and G&A	$18.5
Working capital	$4.6
Contingency (25%)	$27.9
Pre-production mining	$13.9
Total Pre-Production Cost	**$160.9**
LOM sustaining capital	$18.2
Closure incl. backfill	$72.4

The average Life of Mine (LOM) operating cash costs, once sustained positive cash flow has been achieved, are shown in Table 1-6.

Table 1-6: Long Valley Operating Costs

Item	Unit	Operating Costs (LOM average) (1)
Mining costs (per ton mined)	US$/st mined	$1.88
Mining costs	US$/st processed	$4.54
Processing costs	US$/st processed	$2.64
G&A costs	US$/st processed	$0.89
Total site operating costs	US$/st processed	$8.07
Cash Costs*		
Cash costs (LOM)*	US$/oz	$646

(1) Not including post-production reclamation and backfilling. Life of Mine is rounded to 7 years; however, it includes process costs in Year 8.

1.14 Economic Analysis

The results of the economic analysis are summarized in Table 1-7.

Table 1-7: Summary of Long Valley Economic Results

Parameter	Unit	Pre-Tax	Post-Tax
Net present value (NPV5%) at 0.75C$/US$	C$ millions	$463	$364
Net present value (NPV5%)	US$ millions	$347	$273
Internal rate of return (IRR)	%	57%	48%
Payback (undiscounted)	Years	1.6	1.8
LOM avg. annual cash flow after tax & capital	US$ millions	$96	$83
LOM cumulative cash flow (undiscounted)	US$ millions	$475	$385
Gold price assumption	US$ per ounce	$1,600	
Mine life	Years	7	
Average annual mining rate	million tons/yr	18.5	
Average annual gold production	thousand ounces/yr	102	
Total LOM recovered gold	thousand ounces	717	
Initial capital costs	US$ millions	$161	

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.15 Interpretation and Conclusions

The resources contained in an optimized pit at a low stripping ratio offer an opportunity to develop a potential open pit project with a recommended work program.

In the opinion of the Qualified Person (QP), while there is sufficient drilling to define Indicated resources for the deposit, additional data would be useful to refine the geologic, metallurgical, and density interpretations for the deposits. The assignment of Indicated and Inferred classification of resources considered that most of the drilling has been reverse circulation and utilized conservative drill hole distances for the assignment of block classification. The relatively high percentage of Indicated resources within the total reported resource results from the close, systematic drill spacing throughout the deposit, which has defined relatively continuous, and generally flat-lying, tabular mineralization.

A relationship between depth from surface, amount of oxidation, and density of the material is noted for the deposit. Resources reported as oxide in this report are the material situated above 150 feet from the surface, and above a transition zone that occurs approximately between 150 and 200 feet below the surface. Sulfide material is considered to be below 200 feet from the surface topography.

For this current report, the lithology model and gold domain envelopes were used to code the Royal Gold and Amax drill data. After reviewing the spatial distribution and statistical characteristics of the density data, seventeen highly anomalous measurements were removed from the data set. The density data were then converted to tonnage factor values and an average tonnage factor by rock type .

The mine plan is based on 22,000 tons per day of mineralized material production. The pits were divided into four phases, including one satellite pit. In the initial phases, the mine is extracted from south to north followed by the extraction of the satellite pit. Pre-stripping and phasing ensures similar quantity of leachable material production throughout the mine life. The plan produces 54.2 million tons of leachable material at an average grade of 0.020 ounces per ton (oz/ton) or 0.67 grams per tonne (g/tonne) in a 7-year mine life. Stripping requirements include a life of mine total of 76.5 million waste tons. Waste management for the mine includes a waste dumps and concurrent backfilling. At the end of production, the waste dump will be transported to the open pit, and the heap leach pad will be rinsed and neutralized. After rinsing and neutralization of the heap leach material, it will be transported into the remaining open pit. An estimated 17.8 million tons of the material remain on the surface, which is reclaimed to the +25 ft of the original topography.

Operating cost in production years for the Long Valley Project amount to $1.88 per short ton mining cost, $2.64 per short ton processed processing cost, and $0.89 per short ton processed G&A cost. Total capital costs for the project are $47.6 million mine, $55.5 million plant, $0.76 million G&A, $11.7 million infrastructure, $11.0 million sustaining, $18.6 million reclamation, and $36.3 million contingency, for a total of $181.5 million.

The PEA used a base gold price of $1,600/oz with an estimated overall recovery of 68%, which resulted in an After-Tax Net Present Value at 5% of $273 million and an Internal Rate of Return of 48%. This technical report includes inferred mineral resources.

1.16 Recommendations

The authors recommend a two-phase program to advance the project as follows:

Phase 1

- Add silver to the resource model
- Environmental impact assessment

- Collection of geotechnical, hydrology, and hydrogeology data
- Perform closure testing on the spent heap materials to determine if the material can cause water quality impacts.
- Execute geotechnical investigations into the heap stability
- Perform geotechnical testing of soils under the leach pad, ponds, and plant site
- Conduct geotechnical testing of the pit wall
- Improve metallurgical understanding of the orebody through additional metallurgical sampling. Drilling should be weighted to match the distribution of sulfide, oxide, transition, siliceous, and argillic material.
- Sulfide-sulfur assays should be conducted on all samples in addition to gold, silver, hot cyanide leach, and a full ICP scan
- Further test work should be considered for the Long Valley project:
  - Crusher work index and abrasion tests should be conducted to confirm crusher design and wear rates
  - Agglomeration tests should be performed to confirm the optimal mix of cement/lime, and moisture necessary to achieve acceptable percolation and leaching results
  - A comprehensive array of column tests should be arranged with representative samples from all areas of the deposit. Minimal column work is necessary for the sulfide material as it is not amenable to heap leaching.
  - The optimization of the crush size requires further investigation and the investigation of high pressure grinding roll (HPGR) may be warranted given the material characteristics
  - Sulfide mineralogy should be tested to define a suitable flowsheet for this material if economically warranted. Several basic crush, grind leach tests should be conducted followed by additional testing if the material is refractory to conventional processing techniques.

If Phase 1 is successful, the authors recommend proceeding to Phase 2, a pre-feasibility or feasibility study.

Table 1-8 shows the estimated cost of a phased program to complete the recommendations above and update the deposit model.

Table 1-8: Estimated Cost of Phase 1 Recommended Program

Description	Total (US$)
Engineering and Other Studies	
Baseline environmental study	1,000,000
Geotechnical / HL design studies	500,000
Metallurgical test work	500,000
Subtotal	**2,000,000**
Community Engagement Program	200,000
Stakeholder Mapping	100,000
Subtotal	**300,000**
Contingency (10%)	430,000
Total	**2,730,000**

After completing this program, the results should be assessed to determine if a pre-feasibility study should be completed. The suggested work will improve the accuracy of a pre-feasibility study.

1.17 Risk

The main risks associated with the project are related to permitting and California mining regulations. This risk could potentially cause long delays in acquiring permits and additional holding costs during these delays.

There is a risk that the project will encounter serious opposition during the permitting process if the permitting effort is not properly managed. To mitigate this risk, the Company plans to initiate an industry best practice community engagement program to build local support with all stakeholders.

The change in California mining regulations in the early 2000s with the introduction of the backfill law severely impacted new projects. With the current higher gold price, the backfill requirement can be met without severely impacting the project economics. There is a risk other regulations could be implemented that further impact project economics.

The mine development plan presented in the preliminary economic assessment is preliminary in nature, the plan partly includes inferred mineral resources as process plant feed. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. As such, there is no certainty that the preliminary economic assessment will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability and this technical report does not present any statement of mineral reserves.

1.18 Opportunities

The PEA outlines several initiatives that may enhance the Project including:

- Assaying silver in all future drill programs to add silver into the resources
- Conducting metallurgical tests to establish optimal crush size and cement addition
- Performing test work on very low-grade samples to determine viability of run-of-mine leaching
- Reviewing contract mining to reduce initial capital
- Drilling for more oxide resources and deep sulfides to look for high-grade feeder zones

2.0 INTRODUCTION AND TERMS OF REFERENCE

This Technical Report was revised and amended on June 7 of 2021 from the original Report issued on October 27, 2020. The revisions and amendments do not change the mineral resources or the results of the preliminary economic assessment.

Qualified Persons from Global Resource Engineering, Ltd. (GRE) and Mine Development Associates, Inc. (MDA), a division of RESPEC, have prepared this Technical Report on the Long Valley gold project, located in Mono County, California, at the request of KORE Mining Ltd. ("KORE"), a Canadian company located in Vancouver, British Columbia. The Long Valley project is held 100% by KORE's wholly-owned subsidiary KORE USA Ltd. The project is focused on the Long Valley gold deposit, which is also known as the Inyo gold deposit.

This report has been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on November 29, 2019.

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide a Mineral Resource Estimate and Preliminary Economic Assessment (PEA) of the Long Valley gold project in support of securities exchange reporting requirements. This PEA is based on the Mineral Resource Estimate and block models by MDA and documented in a Technical Report on the Long Valley project dated December 18, 2019 for KORE (MDA, 2019). There has been no further exploration work conducted on the property since the 2003 mineral resource estimate (MDA, 2003) was prepared, except for geophysical surveys in 2017 and 2020, and relogging of drill chips. An optimized pit with current costs and metal prices were used to constrain the mineral resource estimate, and update and the mineral resources as current mineral resources.

The mineral resources were estimated and classified under the supervision of Neil Prenn, P. E. and principal engineer for MDA. Mr. Prenn is a qualified person under NI 43-101 and has no affiliations with KORE except that of independent consultant/client relationship. The mineral resources reported herein are estimated to the standards and requirements of the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2019) and therefore NI 43-101. No mineral reserves were estimated.

No QPs from GRE have visited the site for this PEA. Table 2-1 shows the report sections and responsible party.

Mr. Prenn visited the Long Valley project on October 30, 2002, and again on February 21, 2018. In neither case was there evidence of current or recent exploration or mining activity; however, some old drill locations were still identifiable.

Steven I. Weiss, C.P.G. and Senior Associate Geologist for MDA, conducted a site visit to the property on September 20, 2020. Mr. Weiss inspected the surface geology of the property on September 20, 2020 and verified though personal inspection that the surface geology of the property summarized in Item 7 is materially correct and consistent with the regional map of the Long Valley caldera area published by the United States Geological Survey (Bailey, 1989). During May through November of 1996, while employed as as a Research Associate in the Department of Geological Sciences at the University of Nevada, Reno (UNR) with funding provided by Royal Gold for an independent research project, Mr. Weiss inspected and logged all of the 1996 RC and core holes drilled by Royal Gold that year, re-logged many of the pre-1996 drill holes and conducted petrographic and mineralogic studies of then-existing drill core. Mr. Weiss. The independent research project was entitled “Geologic Setting and Hydrothermal History of the Long Valley Gold Deposit, California: Quaternary Precious-metal Mineralization in the Long Valley Caldera”.

Table 2-1: Contributing Authors

Section	Section Name	Responsibility	Author/ QP
1	Summary		
1.1	Property Description and Ownership	MDA	Neil Prenn
1.2	Accessibility and Climate	MDA	Neil Prenn
1.3	Exploration and Mining History	MDA	Neil Prenn
1.4	Geology and Mineralization	MDA	Steven Weiss
1.5	Drilling, Sampling and Data Verification	MDA	Neil Prenn
1.6	Mineral Processing and Metallurgical Testing	GRE	Todd Harvey
1.7	Long Valley Mineral Resource Estimate	MDA	Neil Prenn
1.8	Mining Method	GRE	Terre Lane
1.9	Recovery Method	GRE	Todd Harvey
1.10	Project Infrastructure	GRE	Terre Lane
1.11	Market Studies and Contract	GRE	Terre Lane
1.12	Environmental Studies, Permitting and Social or Community Impact	GRE	Terre Lane
1.13	Capital and Operating Cost	GRE	Terre Lane
1.14	Economic Analysis	GRE	Terre Lane
1.15	Interpretation and Conclusions	GRE	Terre Lane
1.16	Recommendations	GRE	Terre Lane
1.17	Risk	GRE	Terre Lane
1.18	Opportunities	GRE	Terre Lane
2	Introduction	GRE	Terre Lane
3	Reliance on Other Experts	GRE	Terre Lane
4	Property Description and Location	GRE	Terre Lane
5	Access, Climate, Local Resources, Infrastructure and Physiography	MDA	Neil Prenn
6	History	MDA	Neil Prenn
7	Geology Setting and Mineralization	MDA	Steven Weiss
8	Deposit Types	MDA	Steven Weiss
9	Exploration	MDA	Neil Prenn
10	Drilling	MDA	Neil Prenn
11	Sample Preparation, Analyses and Security	MDA	Neil Prenn
12.1	Data Verification	MDA	Neil Prenn
12.2	Data Verification	MDA	Steven Weiss
12.3	Data Verification	GRE	Todd Harvey

Section	Section Name	Responsibility	Author/ QP
12.4	Data Verification	GRE	Terre Lane
13	Mineral Processing and Metallurgical Testing	GRE	Todd Harvey
14	Mineral Resource Estimates	MDA	Neil Prenn
15	Mineral Reserve Estimates	GRE	Terre Lane
16	Mining Methods	GRE	Terre Lane
17	Recovery Methods	GRE	Todd Harvey
18	Project Infrastructure	GRE	Lane and Harvey
19	Market Studies and Contracts	GRE	Terre Lane
20	Environmental Studies, Permitting and Social or Community Impact	GRE	Lane, reliance on David Brown
21	Capital and Operating Costs	GRE	Lane and Harvey
22	Economic Analysis	GRE	Terre Lane
23	Adjacent Properties	MDA	Neil Prenn
24	Other Relevant Data and Information	GRE	Terre Lane
25	Interpretation and Conclusions	All	Lane and Prenn
26	Recommendations	MDA	Neil Prenn
27	References	GRE	Terre Lane

The scope of this study included a review of pertinent technical reports and data provided to GRE and MDA by KORE and its predecessor on the property, Vista, relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, drilling programs, and metallurgy as cited throughout this report. The authors have reviewed much of the available data and made site visits and have made judgments about the general reliability of the underlying data. Where deemed either inadequate or unreliable, the data were either eliminated from use, or procedures were modified to account for lack of confidence in that specific information. The authors have made such independent investigations as deemed necessary in the professional judgment of the authors to reasonably present the conclusions discussed herein. The authors believe the data presented in this report are generally an accurate and reasonable representation of the project.

The Effective Date of this PEA Technical Report is September 21, 2020.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

In this report, measurements are generally reported in Imperial units. Where information was originally reported in metric units, the authors have made the conversions as shown below. Currency, 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			= 3.785 liter
Weight			
1 short ton		= 2000 pounds	= 0.907 tonne
1 pound = 16 oz		= 0.454 kg	= 14.5833 troy ounces
Analytical Values	percent	grams per metric tonne	troy ounces per short ton
1%	1%	10,000	291.667
1 gm/tonne	0.0001%	1	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
10 ppb			0.00029
100 ppm			2.917

Currency Unless otherwise indicated, all references to dollars ($) in this report refer to currency of the United States.

3.0 RELIANCE ON OTHER EXPERTS

The authors are not experts in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements in the United States. The authors did not conduct any investigations of the environmental, permitting, or social-economic issues associated with the Long Valley project, and the authors are not experts with respect to these issues. The authors have relied fully on KORE for information concerning the legal status of KORE and related companies, as well as current legal title, material terms of all agreements, existence of all applicable royalty obligations, and material environmental and permitting information that pertain to the Long Valley project.

Section 4.0 is based on information provided by KORE. This information consisted of maps and other documents received from Mr. James Hynes via email during March, 2018.

The authors relied on Mining Tax Plan LLC to estimate the Federal and California state tax schedule. Mining Tax Plan LLC specializes in U.S. Federal, state, local, and foreign taxation of precious metal, non-metallic ores, coal, and quarry mining and are based in Centennial, Colorado.

As of the date of this report, the authors are not aware of any litigation that could potentially affect the Long Valley Gold Project.

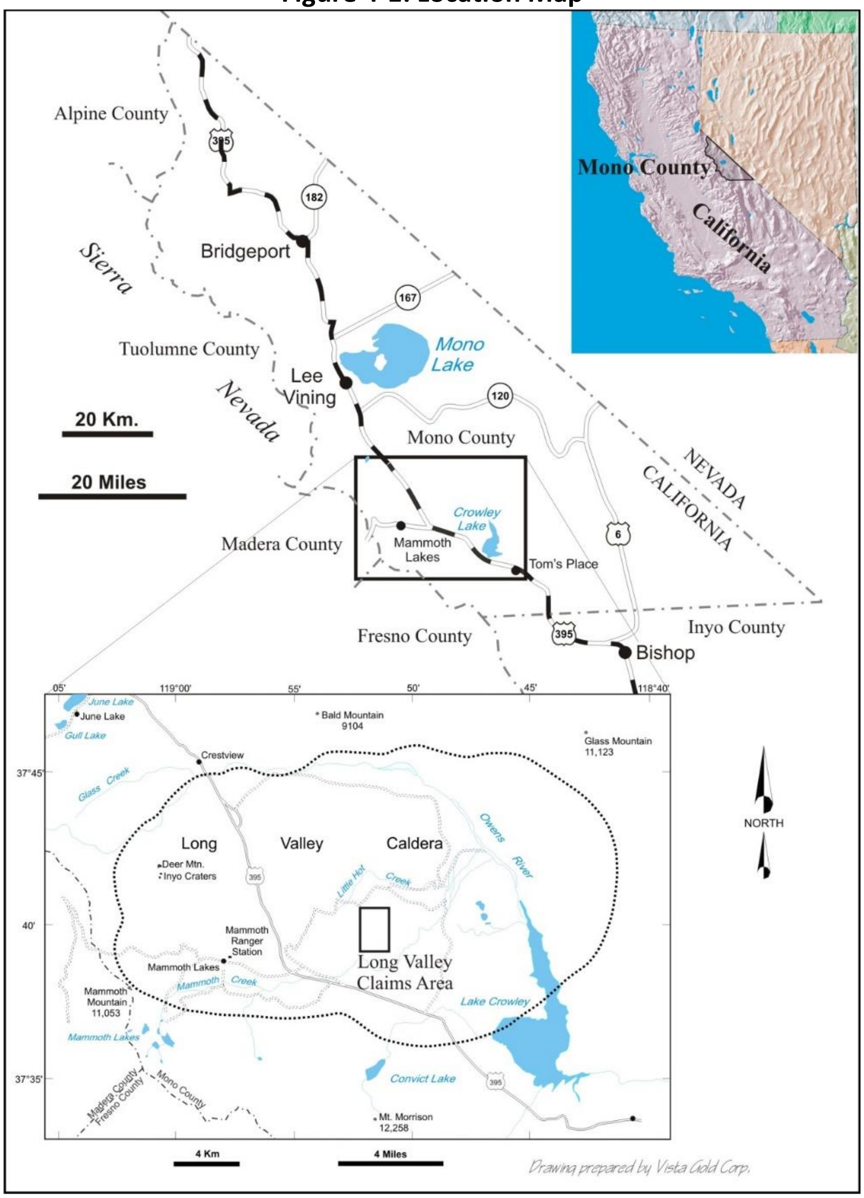
4.0 PROPERTY DESCRIPTION AND LOCATION

This Section 4.0 is based on information provided to the authors by KORE. The authors present this information to fulfill reporting requirements of NI 43-101 and express no opinion regarding the legal or environmental status of the Long Valley project, or of any of the agreements and encumbrances related to the property. Beyond what is described in this section, the authors are 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.

4.1 Location

The Long Valley property is located in the Inyo National Forest, about 30 miles in a direct line northwest of Bishop and about seven miles east of the town of Mammoth Lakes, in Mono County, California. Figure 4-1 shows the location of the property.

Figure 4-1: Location Map



(From Prenn and Muerhoff, 2003)

4.2 Land Area

The Long Valley gold property consists of 95 contiguous, unpatented mining claims that cover an area of approximately 1,800 acres. The claims are administered by the U.S. Bureau of Land Management ("BLM") on federally owned lands administered by the Inyo National Forest, U.S. Department of Agriculture. All of the claims are located in Mono County in east-central California. A listing of the claim names and BLM recordation information is presented as Table 4-1:.

Table 4-1: Claims Constituting the Long Valley Project

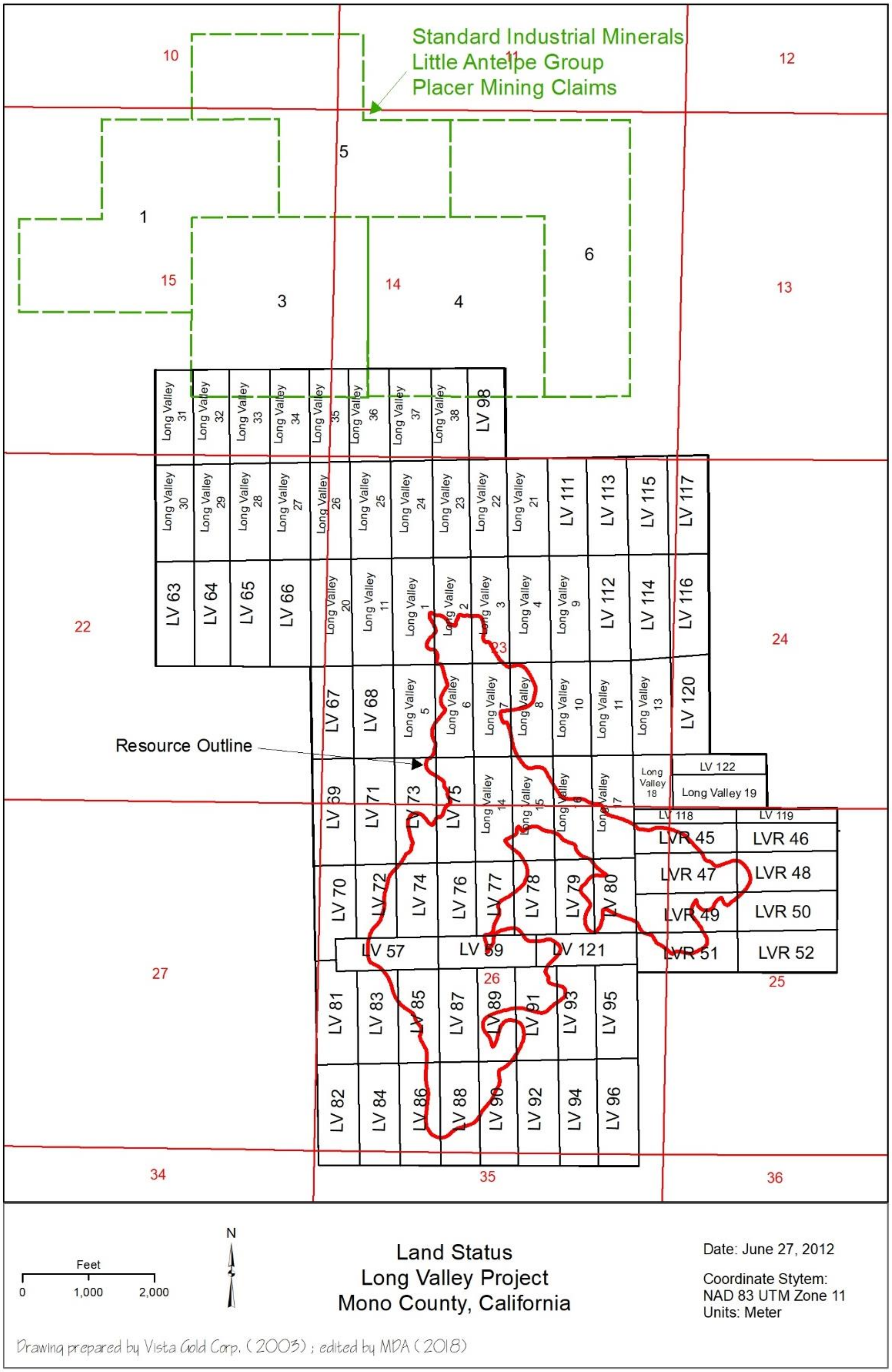
Claim Name	BLM Serial No (CAMC)		
Long Valley 1 - 11	231947	-	231957
Long Valley 12 - 38	237721	-	237747
LVR 45 - 52	275118	-	275125
LV 57	270604		
LV 59	270605		
LV 63 - 96	242259	-	242292
LV 98	242294		
LV 111 - 117	242307	-	242313
LV 118 - 119	270618	-	270619
LV 120	242316		
LV 121	270620		
LV 122	242318		

4.3 Mining Claim Description

The mining claim group is centered at 37 degrees 40 minutes North latitude and 118 degrees 51 minutes West longitude. The claims cover all or portions of Sections 13, 14, 15, 22, 23, 24, 25, and 26, T3S, R28E, Mount Diablo Base and Meridian. Figure 4-2 is a claim location map. Ownership of the unpatented mining claims is in the name of the holder (locator), subject to the paramount title of the United States of America, under the administration of the BLM. Under the Mining Law of 1872, which governs the location of unpatented mining claims on federal lands, the locator has the right to explore, develop, and mine minerals on unpatented mining claims without payments of production royalties to the U.S. government, subject to the surface management regulation of the BLM and U.S. Forest Service, with the area of the claims being subject to a surface grazing lease issued by the U.S. Forest Service. KORE has rights to use the unpatented mining claims for mining-related purposes to September 1, 2021 and may continue to do so on a yearly basis beyond that by timely annual payment of claim maintenance fees and other filing requirements.

The areas of defined gold resources are located entirely within the area of the claims listed in Table 4-1: and shown in Figure 4-2.

Figure 4-2: Claim Map



4.4 Agreements and Encumbrances

The unpatented mining claims are all held by KORE USA Ltd., a Nevada (U.S.A.) corporation that is a wholly owned subsidiary of KORE Mining Ltd.; both companies are referenced as "KORE" in this report. The claims are in good standing, with all holding fees paid for the current year. The claims will remain in effect for as long as the annual claim maintenance fees are paid to the U.S. government. The claims must also be maintained by ensuring that the claim posts and location notices are properly upright and visible. The claim maintenance fees for the Long Valley project total $15,675 annually and are due on or before September 1 of each year. KORE paid this amount to the BLM on August 21, 2020 for the 2021 assessment year. In addition, KORE must file and record with the Mono County Recorder an Affidavit Notice of Intent to Hold and Payment of Annual Maintenance Fee in lieu of Assessment Work; that affidavit was filed and recorded in Mono County on August 12, 2020.

KORE acquired the claims from Vista Gold California LLC, a subsidiary of Vista Gold Corp. (both companies are referenced as "Vista" in this report), through a purchase agreement dated March 29, 2017. In addition to a royalty to Vista described below, KORE agreed to pay Vista a cash consideration of US$1,350,000, payable as follows:

a) US$350,000 at closing (paid on March 31, 2017);

b) US$500,000 on or prior to the 30th day after commencement of commercial production; and

c) US$500,000 on or prior to the 12-month anniversary of the commencement of commercial production.

Vista may elect to receive shares of KORE in place of cash for the payments identified as b) and c) above.

The property is subject to two royalty agreements. A 1.0% net smelter return ("NSR") royalty is payable by KORE to Royal Gold, Inc. ("Royal Gold") pursuant to a Royalty Deed between Vista and Royal Gold dated August 23, 2002, and subsequently assigned to KORE by Vista on April 25, 2017. In addition, through an agreement between KORE and Vista dated April 25, 2017, KORE granted Vista a perpetual NSR royalty at the following rates to be determined quarterly based on the gold price:

Gold Price ($/oz Au)	Royalty Rate
Under $1,400	0.5% NSR
$1,401 to $1,600	1.0% NSR
Above $1,600	2.0% NSR

The royalty agreement between KORE and Vista allows KORE to repurchase a total of 1.0% of the royalty rate applicable to any royalty payable when the gold price is above $1,600 per oz Au for $2,000,000, if repurchased prior to announcement of a Feasibility Study, or for $4,000,000 if repurchased prior to commencement of commercial production, subject to various terms and conditions. KORE's option to repurchase a portion of the royalty rate is extinguished following the commencement of commercial production. The royalty agreement between KORE and Vista also included a security interest in favor of Vista over the Long Valley claims in respect of any future obligations arising under the royalty only.

The purchase agreement between KORE and Vista included a grant of rights to Vista regarding placer claims pursuant to an agreement between Standard Industrial Minerals, Inc. ("Standard") and Vista dated January 22, 2007. Standard granted Vista the right to "explore, develop, mine, remove and sell the gold, silver, and other materials located on and under the ground" where Standard's Little Antelope No. 3 and Little Antelope No. 4 unpatented placer mining claims overlap the Long Valley No. 31-38 and LV No. 98 unpatented lode mining claims; that right was transferred from Vista to KORE in 2017. Figure 4-2 shows the location of the area of overlap between the placer and lode claims subject to this agreement.

The 2007 mining deed that conveyed the unpatented lode mining claims from Standard to Vista included a provision that reserved to Standard all material mined from the property that contains kaolinite but does not contain economic values of gold and/or silver, and was not needed by Vista for construction purposes related to the property, both as determined by Vista, and the right to have such mined kaolinite material transported and deposited at Standard's facilities near the property at Standard's sole cost and expense. This reservation did not obligate Vista to evaluate any mined material for its value or suitability as kaolinite mineralized material, nor handle the kaolinite-bearing material in any special way different from the normal material handling process for material deemed not economic as gold and/or silver mineralized material. At the time Vista purchased the claims from Standard, Standard was mining kaolinite from an operation within a mile north of the unpatented lode mining claims purchased by Vista, but that operation is not active currently.

4.5 Environmental Liabilities

The authors are not aware of any outstanding environmental liabilities on the property.

4.6 Land Use Authority and Entitlements

The Project area consists of National Forest land with mineral interests controlled by through federal lode mining claims. There is no private land within the Project area. The United States Forest Service (USFS) would be the primary federal regulatory agency, and management of this portion of the National Forest is through the Mammoth Ranger District in Mammoth Lakes, California. Mining operations are conducted in accordance with an approved Plan of Operations, submitted to the District Ranger

Mono County also retains land use authority via both a Mining Operations Permit and a Reclamation Plan to be acquired from the Mono County planning commission. This reclamation plans requirements are subject to the California Surface Mining and Reclamation Act (SMARA) and differ from reclamation requirements under the USFS Plan of Operations.

These entitlements and the related environmental permits are more completely addressed in Section 20.

4.7 Mining Rights for Long Valley Project in Mono County, California

Federal law and policy recognize the importance of a viable domestic mining industry and also recognize the importance of protecting natural resources from the potential damaging effects of mining. For example, the Mining Law of 1872 allows miners to secure exclusive rights to mine public lands through the location of valid mining claims, and the Mining and Mineral Policy Act sets forth a federal policy to "foster and encourage" mining, (30 U.S.C. §§ 21a, 22).

USFS regulations to set forth rules and procedures through which use of the surface of National Forest System lands in connection with operations authorized by the United States mining laws (30 U.S.C. 21–54), which provide a statutory right to enter upon the public lands to search for minerals. Operations are to be conducted so as to minimize adverse environmental impacts on National Forest System surface resources. The USFS regulations do not provide for the management of mineral resources which resides with Department of the Interior.

Mining operations are conducted in accordance with an approved plan of operations, submitted to the District Ranger. Approval is subject to completion of a final environmental statement prepared and filed with the Council on Environmental Quality. The operation must be approved so long as they are conducted so as to minimize environmental impacts as prescribed by the authorized officer in accordance with the applicable standards.

The "Metallic Mine Backfill Regulation" (14 CCR 3704.1) requires backfilling and regrading within +/- 25 feet of original topo.

Mining operations in the State of California are conducted under the mining regulations provided in the Surface Mining and Reclamation Act of 1975 (as amended). This act states:

> The Legislature hereby finds and declares that the extraction of minerals is essential to the continued economic well-being of the state and to the needs of the society, and that the reclamation of mined lands is necessary to prevent or minimize adverse effects on the environment and to protect the public health and safety.
>
> The Legislature further finds that the reclamation of mined lands as provided in this chapter will permit the continued mining of minerals and will provide for the protection and subsequent beneficial use of the mined and reclaimed land.
>
> The Legislature further finds that surface mining takes place in diverse areas where the geologic, topographic, climatic, biological, and social conditions are significantly different and that reclamation operations and the specifications therefore may vary accordingly.

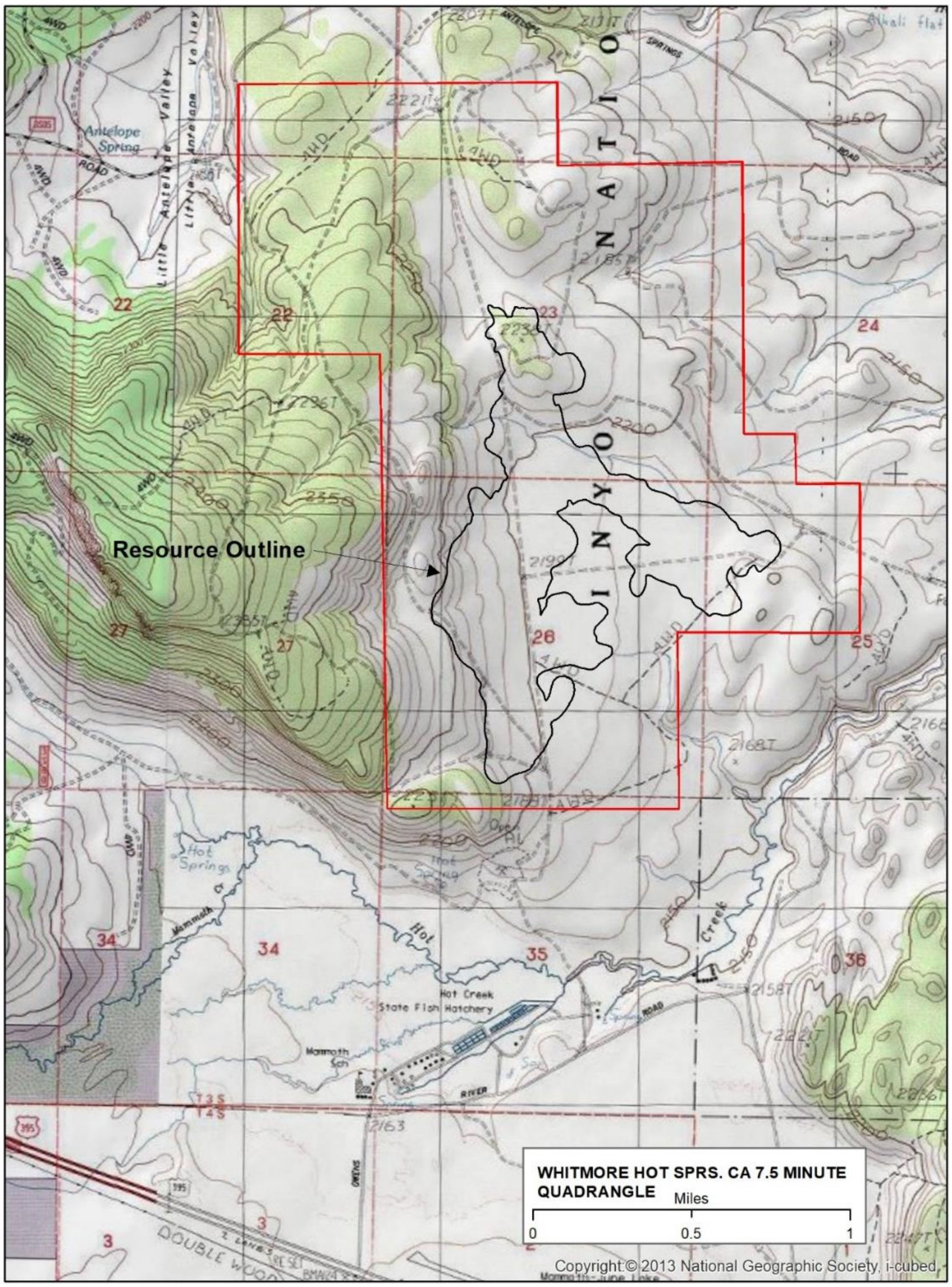
Therefore, the QP concludes that the owner of the validated mineral claims (i.e., the claims within the area defined by the Project Boundary) has the right to advance its exploration and mining interests subject to obtaining permits to carry out the activities per the permits and authorizations referred to in Section 3.0.

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

5.1 Access

The Long Valley property is located about seven miles to the east of the town of Mammoth Lakes and about 45 miles by road northwest of the town of Bishop, California. Both towns are connected by U.S. Highway 395, which passes a few miles west of the property. Access to the property from the highway is via a series of graded gravel roads. Figure 5-1 shows the general area, as well as access to the property and location of the claim block.

Figure 5-1: Access Roads and Location of the Long Valley Claim Block



5.2 Climate

The climate is semi-arid and moderate, with high temperatures in the summer generally in the 80 °F range and winter highs generally in the 30 to 40 °F range. Winter temperatures can be below 0 °F. Precipitation at the property totals about 20 to 25 inches per year, divided between winter snows and summer thunderstorms. The evaporation potential greatly exceeds the precipitation on an average annual basis, so the area is one with a negative water balance. Snow depths in winter are generally less than two feet on the property, and the overall climate should permit operations year around.

5.3 Physiography

The Long Valley gold property is located a few miles to the east of the Sierra Nevada Mountains, at an elevation of about 7,200 feet (2,200 meters), in an area of gently rolling terrain. The vegetation consists mostly of sagebrush and related shrubs and grasses with local areas of open pine forest (Photo 5-1). The topography in the area of the property will allow for the location of site facilities which may be required, including waste dumps, heap leach pads, plant sites, etc..

Photo 5-1: Topography of the Long Valley Deposit Area

(looking South along Hilton Creek Area)

5.4 Local Resources and Infrastructure

Lodging, supplies, and labor are available in either Mammoth Lakes or Bishop, with the area population exceeding 20,000 people. Surface rights sufficient for exploration and mining within the property are inherent to the valid mining claims under the Mining Law of 1872, subject to applicable state and federal environmental regulations.

Groundwater was encountered in many exploration drill holes at depths of 200 to 300 feet. Although fluctuations in the elevation of the water table are possible, the general hydrologic conditions would not be expected to have changed materially since the drilling was conducted, and water should be available in sufficient quantities for processing. It is believed that adequate power is available in the area with no more than a few miles of additional powerline required to reach the property.

6.0 HISTORY

6.1 Exploration History

Gold was first recognized on the property by Standard in the early 1980s as being present in small amounts in and around their kaolinite clay mining operations. Standard optioned the property to Freeport Minerals ("Freeport") in 1983, who prospected the area and defined several distinct mineralized zones, referred to as the North, Middle, and South zones (see Figure 10-1 for the location of the Middle and South mineralized zones and their relation to the current resource area). Based on a sketch map of the zones in the 1997, Behre Dolbear report by Martin et al. (1997), it appears that the North Zone is outside of the current property boundary. Freeport drilled about 80 shallow reverse circulation (RC) holes in mostly the North and South zones during 1983 and 1984. Freeport dropped the property, but additional drilling was performed by Standard in 1986, with 24 shallow rotary holes drilled mostly in the South zone.

Royal Gold acquired the property from Standard under a lease/purchase option agreement in 1988 and shortly thereafter drilled 52 air track holes in the South zone. Martin et al. (1997) reported that Royal Gold drilled 53 holes in this program, but 52 are in the project database. Royal Gold also had performed various metallurgical and engineering studies and submitted permitting documents in support of constructing a small operation based on gold resources in the South zone. However, in 1990, Battle Mountain Gold ("Battle Mountain") and Royal Gold formed a joint venture to further explore and perhaps develop the property. During 1990 and 1991, Battle Mountain, as the operator, completed geologic mapping, geochemical sampling, and geophysical surveying of the area and also drilled 59 RC holes. These holes were mostly in the South zone, but they also resulted in the discovery of two new zones contiguous with the South zone: the Hilton Creek zone and the Southeast zone.

Battle Mountain dropped out of the joint venture in 1993, but work continued by Royal Gold. During the period 1994 through 1997, Royal Gold aggressively explored the property, drilling some 625 holes mostly in the Hilton Creek and Southeast zones. Only 10 core holes were drilled, with the balance being RC holes. During this time, Royal Gold also undertook extensive studies related to metallurgical investigations and preliminary engineering studies, including resource estimations. They also initiated baseline-type environmental studies of the biological, water, and archeological resources of the area.

In mid-1997, Amax performed extensive due diligence investigations in consideration of forming a joint venture with Royal Gold to place the property into production. Their work included drilling 46 RC holes and 10 core holes, as well as extensive re-assay and check-assay work and the re-logging of older holes. Many of the holes were intended as "twins" to earlier Royal Gold holes. Amax elected not to proceed with the formation of the joint venture because of the continued deterioration of the gold price and their pending merger with Kinross Gold. Table 6-1 summarizes the drilling completed on the property.

Table 6-1: Summary of Historical Drilling by Company

Year	Company	# Holes	Footage
1983-4	Freeport	80	18,615
1985	Standard	24	2,055
1988	Royal Gold	52	4,770
1991	Battle Mtn	59	18,685
1994-7	Royal Gold	625	207,901
1997	Amax Gold	56	16,249
Totals		896	268,275

Following Amax's departure, Royal Gold did not perform any additional drilling, but did continue with some of the environmental studies, reclaimed the drill roads and sites, performed some additional geochemical sampling, re-estimated mineral resources, and initiated a community public relations campaign. Due to the continued decline in the gold price and the decision by Royal Gold to become a royalty holding company, Royal Gold turned the property back to Standard, effective August 2000. Except for maintaining the claims in good standing, Standard performed no further work on the Long Valley property and there has been no drilling on the property since 1997.

In January of 2003, Vista signed a purchase option agreement with Standard for the Long Valley project and completed the purchase of the claims in January 2007. Vista maintained the claims in good standing but conducted no exploration on the property from 2003 until their sale of the property to KORE in 2017.

There have been fairly extensive geochemical surveys conducted over the Long Valley property, but only one known geophysical survey prior to KORE's acquisition of the property in 2017. The geochemical surveys have been performed by personnel working for either Battle Mountain or Royal Gold. Documentation of the results of both of the geochemical programs is sparse, but it appears that both surveys consisted of the collection of between 100 and 200, predominantly rock and fewer soil samples. These samples were analyzed for gold, silver, arsenic, antimony, and mercury, and perhaps other elements as well. The surveys indicated that the entire area is mildly to highly anomalous in these elements and that potentially economic mineralization is known by drilling to underlie the area of many of the better anomalies. Other geochemical anomalies remain untested by drilling. MDA has not analyzed the sampling methods, quality, and representativity of surface sampling on the Long Valley property because drilling results form the basis for the mineral resource estimate described in Section 14.0. Drilling is described in Section 10.0.

An IP/resistivity geophysical survey was performed for Battle Mountain by DMW Surveys of Reno, Nevada, in the southern part of the area. Four possible target areas were identified from this survey, and it is believed that these areas have subsequently been drilled, with mineralization indicated in both the Hilton Creek and Southeast zones.

Several periods of geological mapping have been performed in the area by employees of, or consultants to, Battle Mountain and Royal Gold. The mapping identified areas of alteration, silicification, and brecciation within the predominantly volcaniclastic rocks in the area, which have been demonstrated to be favorable for gold mineralization. Many of these areas have been drilled with positive results, but other areas remain untested. In addition, much of the area is covered with soil or post-mineralization rocks, which could conceal areas favorable for mineralization.

Outside of the presently defined resource area as described in Section 14.0, there are numerous drill holes which have intercepted significant intervals of gold and silver mineralization. The area of these drill holes is generally defined as the North and Middle zones and, with further drilling and the discovery of additional mineralized intercepts, they might also be the location of significant gold mineralization. All of the holes are vertical, and all intercepts are thought to represent true thickness. Some of the intercepts include: 145 feet of 0.035 oz Au/ton (LV-83-02), 120 feet of 0.024 ounces (oz) gold (Au)/ton (LV-83-03), 120 feet of 0.017 oz Au/ton (LV-83-05), 85 feet of 0.019 oz Au/ton (LV-83-34), and 80 feet of 0.021 oz Au/ton (LV-83-51).

There has been no historical gold production from the Long Valley property, and the only mining activity in the area has been associated with the mining of kaolinite clay.

6.2 Historical Mineral Inventory Estimates

All estimates described in this section were prepared prior to 2000 and are presented herein merely as an item of historical interest with respect to the exploration targets at Long Valley. There were a number of mineral resource estimates and associated mineral reserve calculations prepared on behalf of Royal Gold by the outside consulting group Mine Reserves Associates ("MRA") of Lakewood, Colorado, during the period 1995 to 1998. It is believed that these estimates were not prepared in full compliance with the provisions included in National Instrument 43-101, as they do not clearly differentiate between Measured, Indicated, and Inferred categories of mineralization and as to whether these categories contribute to the estimates provided in Table 6-2. Accordingly, these estimates should not be relied upon. The author has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and KORE is not treating these historical estimates as current mineral resources or mineral reserves. These historical estimates are superseded by the current mineral resource estimate discussed in Section 14.0 of this report.

Table 6-2: Historical Royal Gold Resource and Reserve Statements – MRA Estimates

Category	Year	Tons 000,000s	Grade oz Au/t	Ounces Gold (000's)
Resource	1996	49.6	0.018	893.5
Resource	1997	49.6	0.018	893.5
Reserve	1996	20.7	0.018	373.0
Reserve	1997	39.1	0.018	704.0

In December 1997, Behre Dolbear & Company Inc. ("Behre Dolbear") calculated reserves based on several density factors, because testwork by Amax had indicated widely variable densities. The base case was from the 1997 MRA calculation. These are summarized in Table 6-3:. The author has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and the issuer is not treating these historical estimates as current mineral resources or mineral reserves. Accordingly, these estimates should not be relied upon.

Table 6-3: Historical Royal Gold Reserve Statements - Behre Dolbear Estimate 1997

Case	Tonnage Factor ft^3/ton	Ore Tons 000,000's	Ore Grade oz Au/t	Ounces Au 000's
Base	14.0	39.1	0.018	703.8
1	18.0	30.4	0.018	547.2
2	20.0	27.4	0.018	492.7

The resource estimates noted above only include the material classified as oxide or non-sulfide in the geologic model. The minable reserves were calculated using oxidized resource material only, a cutoff grade of 0.010 oz Au/ton, an assumed gold recovery of 70%, and a gold price of $350.00 per ounce.

6.3 2003 and 2008 MDA Mineral Resource Estimates for Vista Gold

MDA prepared a mineral resource estimate of the Long Valley deposit for the previous operator in 2003 (MDA, 2003) that was the first estimate reported in accordance with NI 43-101 standards of disclosure at that time. The reported Vista Gold historical resource estimate is shown in Table 6-4:.

In January 2008, MDA prepared a Technical Report for Vista describing a preliminary economic assessment of the Long Valley project (MDA, 2008), but the resource estimate or model was not updated from the 2003 estimate. The 2003 estimate did not report resources constrained within a pit.

The 2003 mineral resource estimate reported in both the 2003 and 2008 Technical Reports (MDA, 2003; MDA, 2008) were prepared in accordance with the CIM Standards and NI 43-101 reporting requirements in effect at that time, but that mineral resource estimate does not meet current CIM Standards and NI 43-101 reporting requirements. It is reported here as a matter of historical interest. Therefore, KORE is not treating the 2003 mineral resource estimate as current mineral resources, and that 2003 estimate and the 2008 preliminary economic assessment should not be relied upon.

Table 6-4: Historic Vista Gold Reported Resources

Rock Type	Cut off (oz Au/ton)	Tons (000's)	Au Grade (oz/ton)	Au Ounces (000's)
Measured				
Oxide	0.010	15,500.6	0.017	265.3
	0.015	6,600.5	0.024	158.9
	0.020	3,201.8	0.032	103.4
	0.050	276.6	0.064	17.7
	0.100	5.7	0.122	0.7
Sulfide	0.010	11,096.3	0.017	187.1
	0.015	4,210.8	0.025	105.3
	0.020	2,004.1	0.035	69.2
	0.050	282.1	0.065	18.4
	0.100	15.2	0.122	1.9
Total Measured	0.010	26,596.9	0.017	452.5
	0.015	10,811.4	0.024	264.2
	0.020	5,205.9	0.033	172.6
	0.050	558.7	0.065	36.1
	0.100	20.9	0.122	2.6
Indicated				
Rock Type	**Cut off (oz Au/ton)**	**Tons (000's)**	**Au Grade (oz/ton)**	**Au Ounces (000's)**
Oxide	0.010	20,571.9	0.019	395.4
	0.015	9,794.1	0.027	266.5
	0.020	6,617.3	0.032	214.4
	0.050	511.7	0.062	31.7
	0.100	13.4	0.122	1.6
Sulfide	0.010	21,106.9	0.017	363.2
	0.015	7,202.8	0.027	197.0
	0.020	4,100.9	0.036	146.9
	0.050	733.2	0.061	44.8
	0.100	42.6	0.125	5.3
Total Indicated	0.010	41,678.8	0.018	758.7
	0.015	16,996.9	0.027	463.5
	0.020	10,718.2	0.034	361.3
	0.050	1,244.9	0.061	76.5
	0.100	56.0	0.124	6.9
Measured + Indicated				
Rock Type	**Cut off (oz Au/ton)**	**Tons (000's)**	**Au Grade (oz/ton)**	**Au Ounces (000's)**
Oxide	0.010	36,072.5	0.018	660.8
	0.015	16,394.6	0.026	425.4
	0.020	9,819.1	0.032	317.8
	0.050	788.4	0.063	49.4
	0.100	19.1	0.122	2.3
Sulfide	0.010	32,203.2	0.017	550.4
	0.015	11,413.7	0.026	302.3
	0.020	6,105.0	0.035	216.1
	0.050	1,015.2	0.062	63.2
	0.100	57.8	0.124	7.2
Oxide & Sulfide	0.010	68,275.7	0.018	1,211.1
	0.015	27,808.3	0.026	727.7
	0.020	15,924.1	0.034	533.9
	0.050	1,803.6	0.062	112.6
	0.100	76.9	0.123	9.5
Inferred				
Rock Type	**Cut off (oz Au/ton)**	**Tons (000's)**	**Au Grade (oz/ton)**	**Au Ounces (000's)**
Oxide	0.010	11,539.7	0.019	219.4
	0.015	5,431.0	0.027	145.7
	0.020	3,971.1	0.031	121.9
	0.050	183.0	0.071	13.0
	0.100	28.9	0.134	3.9
Sulfide	0.010	21,373.7	0.016	352.1
	0.015	6,441.6	0.027	175.9
	0.020	4,070.2	0.034	137.9
	0.050	295.0	0.080	23.5
	0.100	82.6	0.117	9.7
Total Inferred	0.010	32,913.3	0.017	571.5
	0.015	11,872.5	0.027	321.6
	0.020	8,041.3	0.032	259.8
	0.050	477.9	0.076	36.5
	0.100	111.5	0.121	13.5

6.4 2018 and 2019 MDA Mineral Resource Estimates for KORE

In April 2018, MDA prepared a technical report for KORE updating the mineral resources for the Long Valley gold project. The mineral resources were amended in December 2019 (MDA, 2019). The mineral resource estimates reported in 2019 Technical Reports were prepared in accordance with the CIM Standards and NI 43-101 reporting requirements in effect. The author describes the current resource estimate in Section 14.9 of this report.

Table 6-5: Long Valley 2019 Mineral Resource Estimates

Mineralized Material Type	Cutoff (oz Au/ton)	Indicated Resources			Inferred Resources		
		K ton	oz Au/t	K oz Au	K ton	oz Au/t	K oz Au
Oxide	0.005	35,945	0.018	636	9,192	0.020	185
Transition	0.006	4,263	0.014	59	1,314	0.016	21
Sulfide	0.006	33,428	0.017	552	15,464	0.018	280
Total	Variable	73,635	0.017	1,247	25,970	0.019	486

7.0 GEOLOGIC SETTING AND MINERALIZATION

7.1 Geologic Setting

7.1.1 Regional Geology

The Long Valley property is contained entirely within the late Pleistocene Long Valley collapse caldera, which was formed about 760,000 years ago. The Long Valley caldera and related adjacent volcanic rocks comprise a late Pliocene to Quaternary volcanic complex developed along the western edge of the Basin and Range Province, at the base of the Sierra Nevada frontal fault escarpment. The caldera is an oval depression elongated east-west and measuring some 10 by 19 miles. Major collapse was related to the eruption of the Bishop Tuff, which has been dated at about 0.76 Ma. The pre-volcanic basement rocks in the area are mostly Mesozoic granitic rocks of the Sierra Nevada batholith and surrounding Paleozoic and Mesozoic metamorphic rocks. The pre-Cenozoic rocks are totally covered by younger volcanic rocks within the caldera.

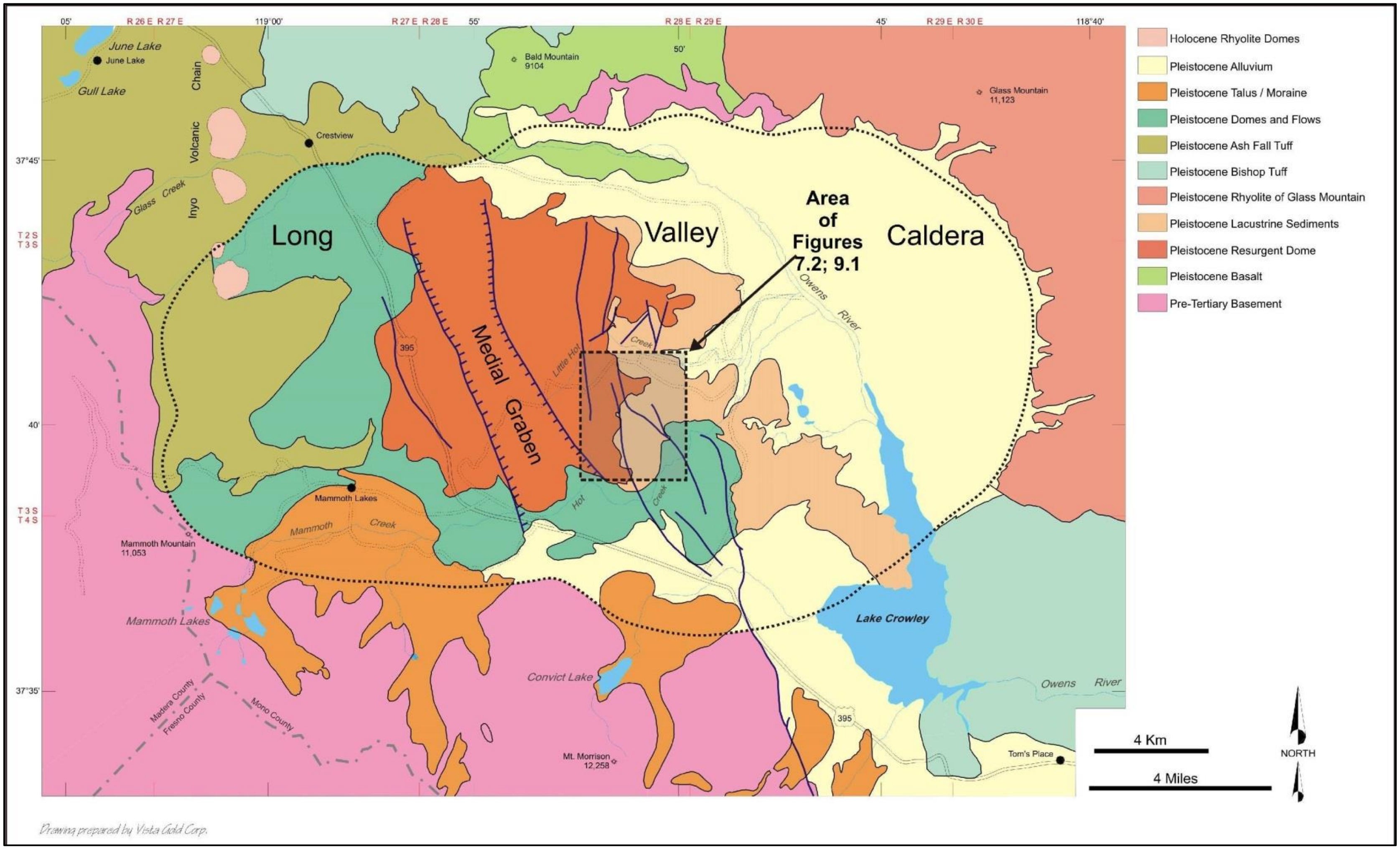
Figure 7-1 shows the generalized regional geology of the Long Valley caldera.

7.1.2 Local and Property Geology

The Long Valley gold property is located near the center of the caldera and is underlain by most of the lithologic units related to caldera formation and subsequent magmatic resurgence. A thick sequence of interbedded volcaniclastic and sedimentary rocks were deposited in a lacustrine setting that occupied much or all of the caldera. These rocks consist of finely varved siltstones interbedded with fine- to coarse-grained ash- and pumice-fall layers, conglomerates and debris-flow deposits, as well as more local deposits of intercalated silica sinter. Clast lithologies are primarily volcanic in origin with a large proportion of rhyolite pumice and ash. These lithologies have an aggregate preserved thickness of more than 1,500 feet based on drill holes within the property.

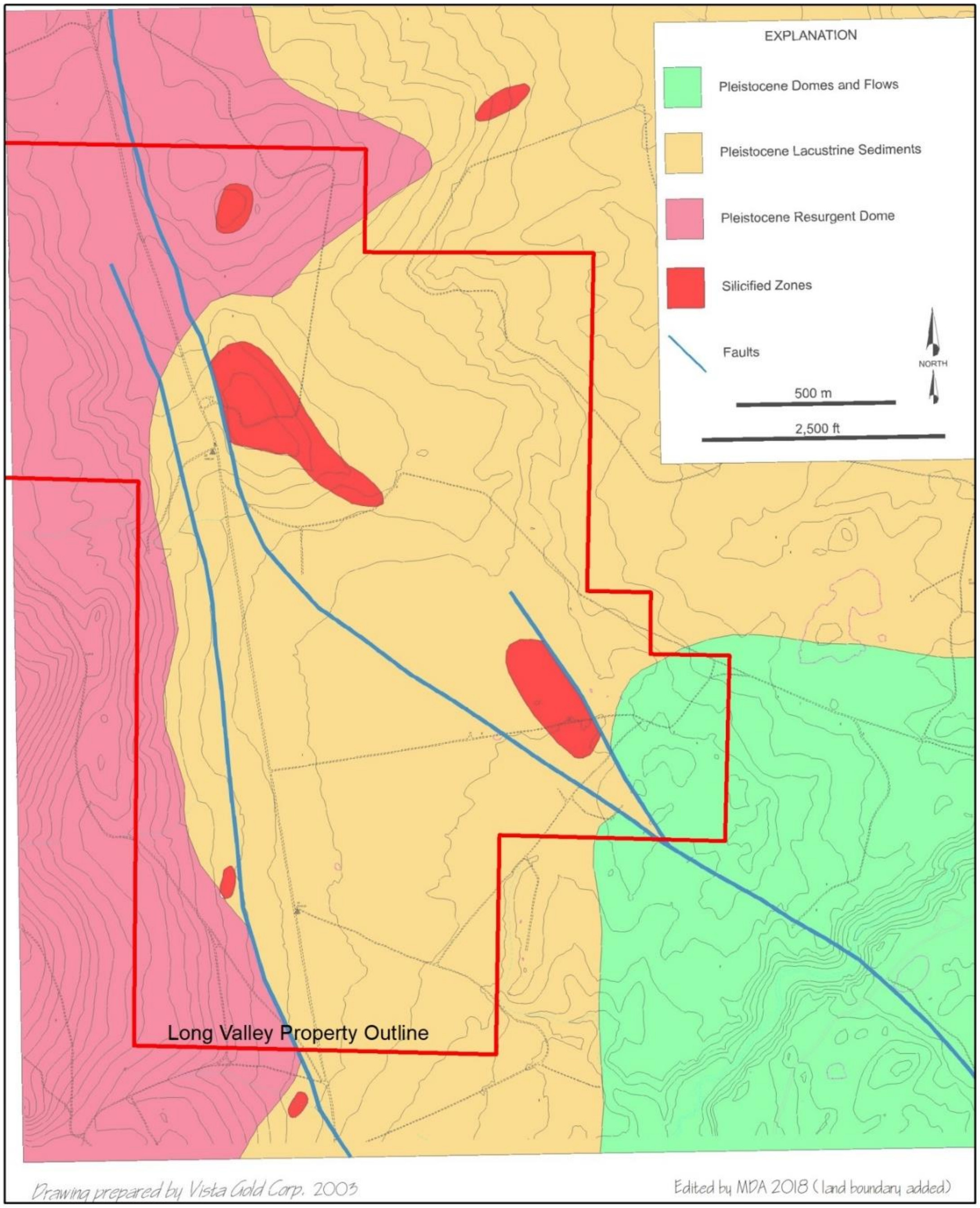
In the central part of the caldera, the intracaldera lacustrine sequence was intruded by a large body of rhyolite that erupted through the generally flat-lying lake sediments and interbedded tuffs and debris-flow deposits to emerge as a large, composite, "resurgent" rhyolite flow-dome exposed just west of the gold deposit (Figure 7-1 and Figure 7-2). It is composed of generally aphyric to sparsely sanidine-bearing rhyolite lava and breccia. Rhyolite breccia and blocks of this flow-dome make up much of the debris-flow units within the adjacent intracaldera lacustrine sedimentary sequence and were likely shed from the erupting flow-dome. The abundant layers of ash, pumice, and debris-flow deposits interbedded within the varved siltstone are interpreted to be co-eruptive with the rhyolite flow-dome, indicating lacustrine sedimentation continued as the rhyolite flow-dome was emplaced. All the aforementioned units have been mineralized in variable amounts. Re-logging of RC drill chips from selected drill holes in the Hilton Creek zone indicates that the rhyolite extends beneath variable thicknesses of the lacustrine volcaniclastic sequence for least 3,000 feet east of the rhyolite exposed at surface. A geologic map of the project area is shown in Figure 7-2.

Figure 7-1: Regional Geology



(From Prenn and Muerhoff, 2003)

Figure 7-2: Deposit Geology



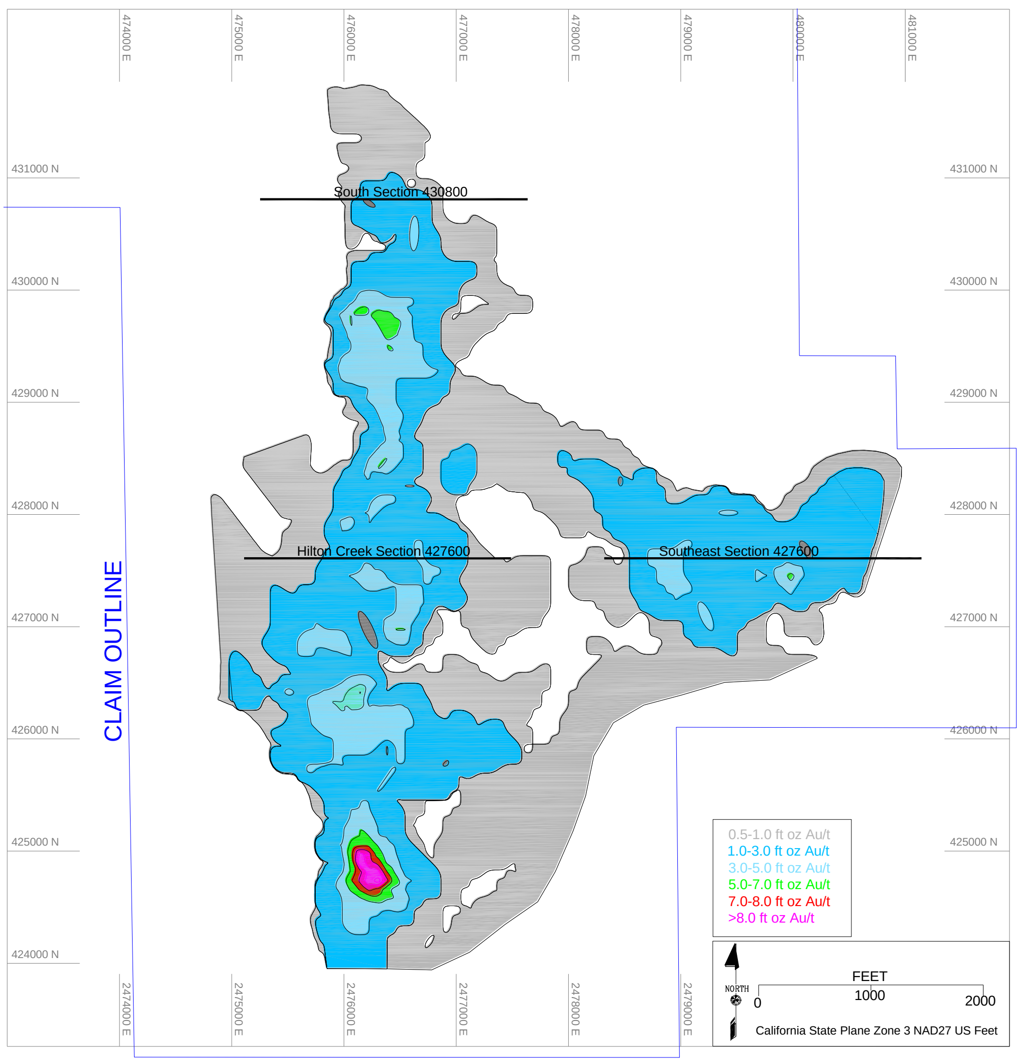
A younger, distinctly quartz-bearing group of rhyolite domes were erupted near the margins of the caldera at about 200,000 to 300,000 years ago. Associated with and younger than all the rhyolite domes is a rather clean, well-sorted arkosic sandstone. Both of these later units crop out to the southeast of the gold deposit. These units are interpreted to be post-mineralization in age, as is recent alluvium up to some 60 feet thick, which covers most of the Hilton Creek gold zone.

The eastern limit of outcrop of the resurgent rhyolite within the central part of the Long Valley caldera has been interpreted by previous operators to be controlled by a north-south trending fault zone that extends south of the property, beyond the southern caldera margin, where it is known as the Hilton Creek fault zone. This normal fault zone with down-to-the-east displacement also seems to be one of the controls on the distribution of gold mineralization in the Long Valley gold deposit.

7.2 Mineralization

Several areas or zones on the Long Valley property are known to be mineralized with low grades of gold and silver. These areas are known as the North, Middle (also called Central), South, Southeast, and Hilton Creek areas (The Middle, South, Southeast, and Hilton Creek areas are shown on Figure 10-1 in Section 10.0 on Drilling; the North Zone lies just north of the current property boundary). Based on drilling, mineralization appears to generally be contiguous between the South, Southeast, and Hilton Creek zones (Figure 7-3). These same zones appear to contain the vast majority of the estimated mineral resources described later in this report. Drilling is widely spaced in and between the North, Middle, and South zones, and it may be possible that with additional drilling, these zones may be shown to be contiguous with the better-defined zones to the south.

Figure 7-3: Grade-Thickness Map of the South, Southeast, and Hilton Creek Zones



The principal host rocks for the gold mineralization are the caldera-fill interbedded siltstone, tuff, and volcaniclastic sedimentary rocks and, to a lesser extent, the adjacent and underlying resurgent rhyolite.

The base of the oxidized zone was generally defined by Royal Gold as the last occurrence of the oxide mineralization within the mineralized zone. As such, mixed oxide-sulfide and sulfide mineralization occurs above this boundary. This oxide/sulfide boundary modeled by Royal Gold is undulating to locally flat-lying, lies at depths of between 100 and 250 feet, and is often coincident with or slightly above the current water table. Grades of gold mineralization are typically the same both above and below the oxide/sulfide boundary.

Gold-silver mineralization is quite continuous throughout the zones and is well defined using a 0.010 oz Au/ton cutoff grade. Numerous zones of higher-grade mineralization (0.050 oz Au/ton) are present within the continuous zones of low-grade (0.010 oz Au/ton) gold mineralization, particularly in the Hilton Creek zone. These higher grades may relate to zones of enhanced structural preparation. Silver grades are generally in the range of 0.1 to 0.5 oz silver (Ag)/ton within the gold-mineralized zones, appear to be more erratic in nature, but generally have a positive correlation with higher gold values. A gold grade-thickness map is presented in Figure 7-3, using a 0.5-foot-oz Au/ton cutoff.

Mineralized zones contain fracture coatings, veinlets, and disseminated iron oxide minerals that were formerly grains of pyrite and marcasite. Opal and chalcedony veinlets with pyrite or marcasite, or iron oxides, are common, but generally less than a few tenths of an inch in width. Adularia is present in fractures and veinlets at depth and as patches of replacement of the rhyolite groundmass in the western margin of the deposit. In much of the deposit, mineralization is associated with zones of clay alteration and/or silicification. These alteration types are well developed in all of the volcaniclastic sediments and, as such, host-rock type does not appear to have a major control over the distribution and grade of mineralization. The predominant clay mineral has been determined to be kaolinite, while the silicification types can be chalcedony, quartz, or opal. Multiple periods of brecciation and silicification are evidenced by cross-cutting silica veinlets and silicified breccia fragments in otherwise clay-altered rocks.

The distribution of the mineralization appears to be spatially related to faults associated with the north-south-trending Hilton Creek fault zone. Splays of this fault zone are projected to trend through the central part of the Hilton Creek mineralized zone, as well as the Southeast zone, with the assumption that the altering and mineralizing fluids ascended along these fault conduits and then spread laterally. Higher-grade zones may also be related to areas of cross-faults and fractures.

The Hilton Creek mineralized zone is known to be some 8,000 feet in length, while the Southeast zone is about 5,000 feet in length. The mineralized zones are generally flat-lying or have a slight dip (10-15 degrees) to the east and have a width in plan view (across the trend) in the range of 500 to 1,500 feet, but average about 1,000 feet in width. The mineralized zones are typically from 50 to 200 feet thick and average about 125 feet thick in the Hilton Creek zone, and 75 feet thick in the Southeast zone. Mineralization in the South and Southeast zones typically is exposed at or very near the surface, while the top of the Hilton Creek mineralization is usually covered by 20 to 50 feet of alluvium. Figure 7-4, Figure 7-5, and Figure 7-6 are east-west cross sections through the Hilton Creek, Southeast, and South zones, respectively, showing the modeled gold zones to indicate thickness, lateral extent, and continuity of mineralization.

Figure 7-4: Hilton Creek Cross Section

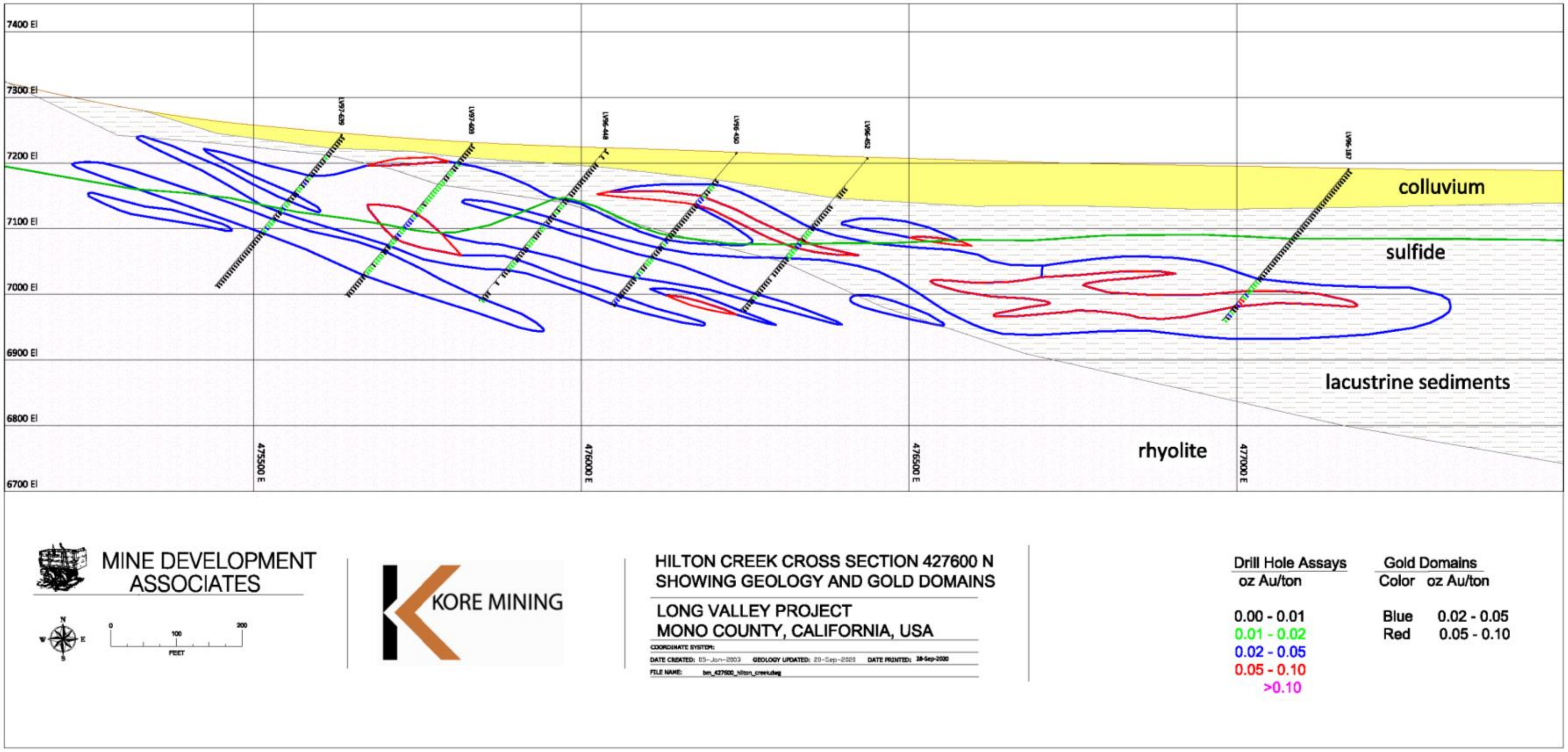


Figure 7-5: Southeast Zone Cross Section

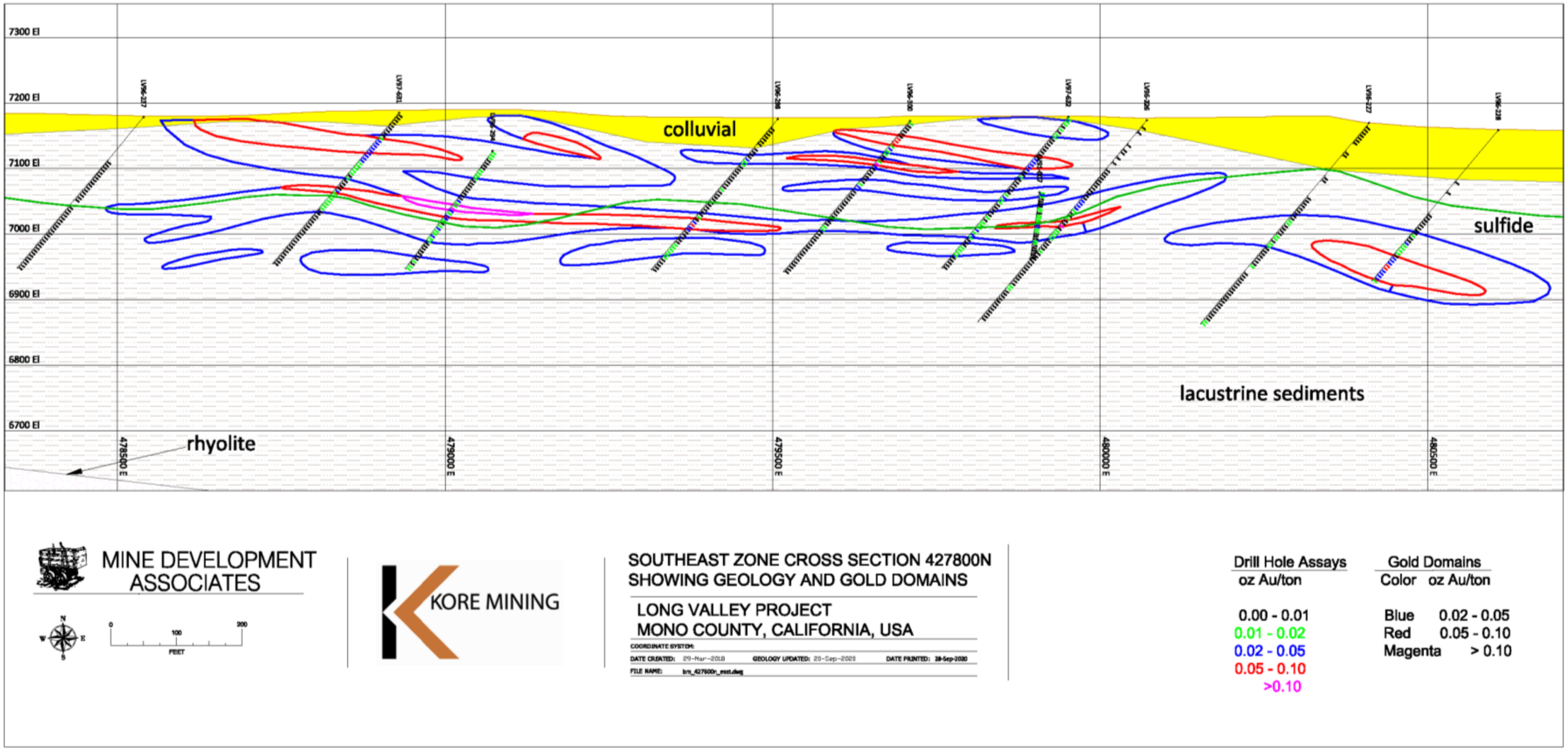
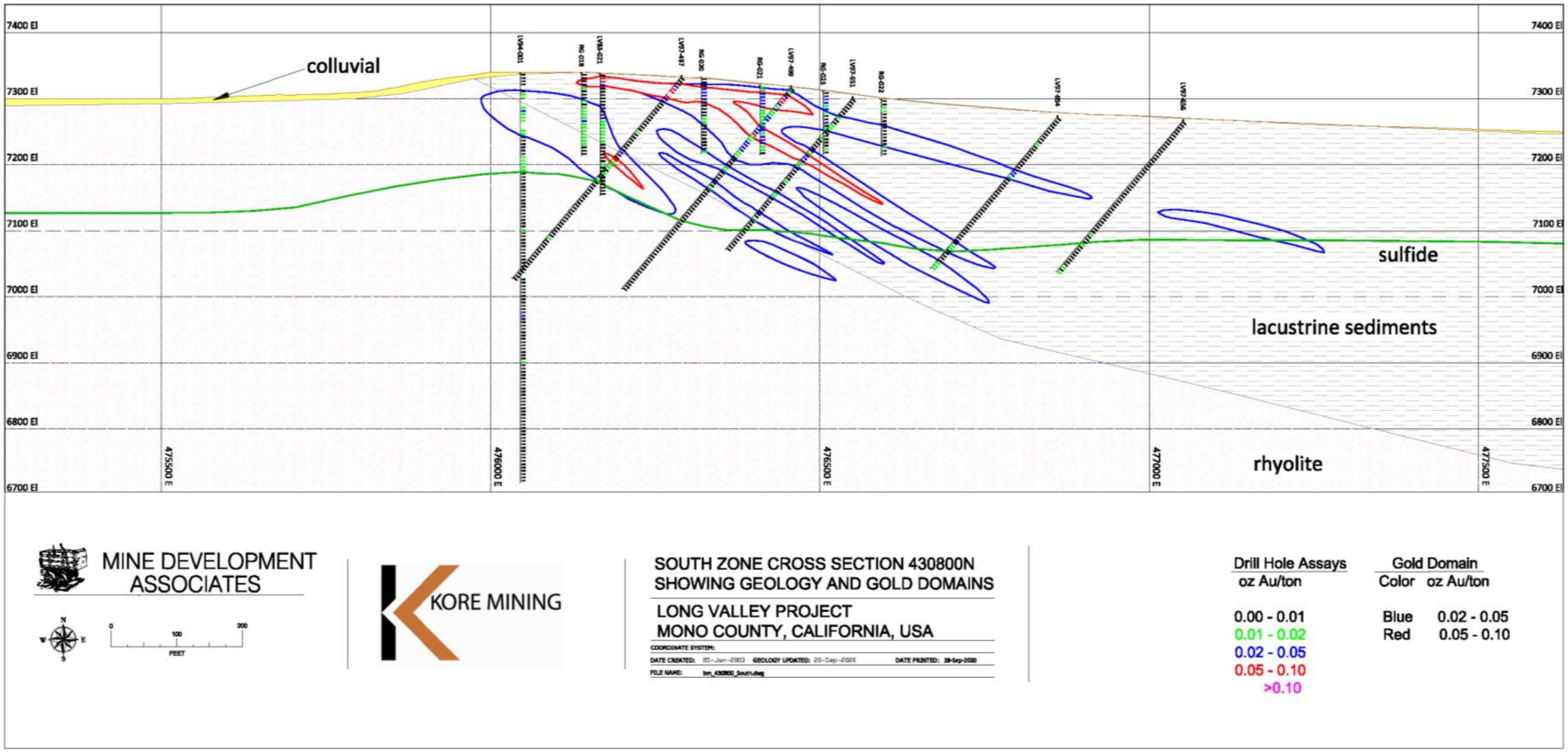


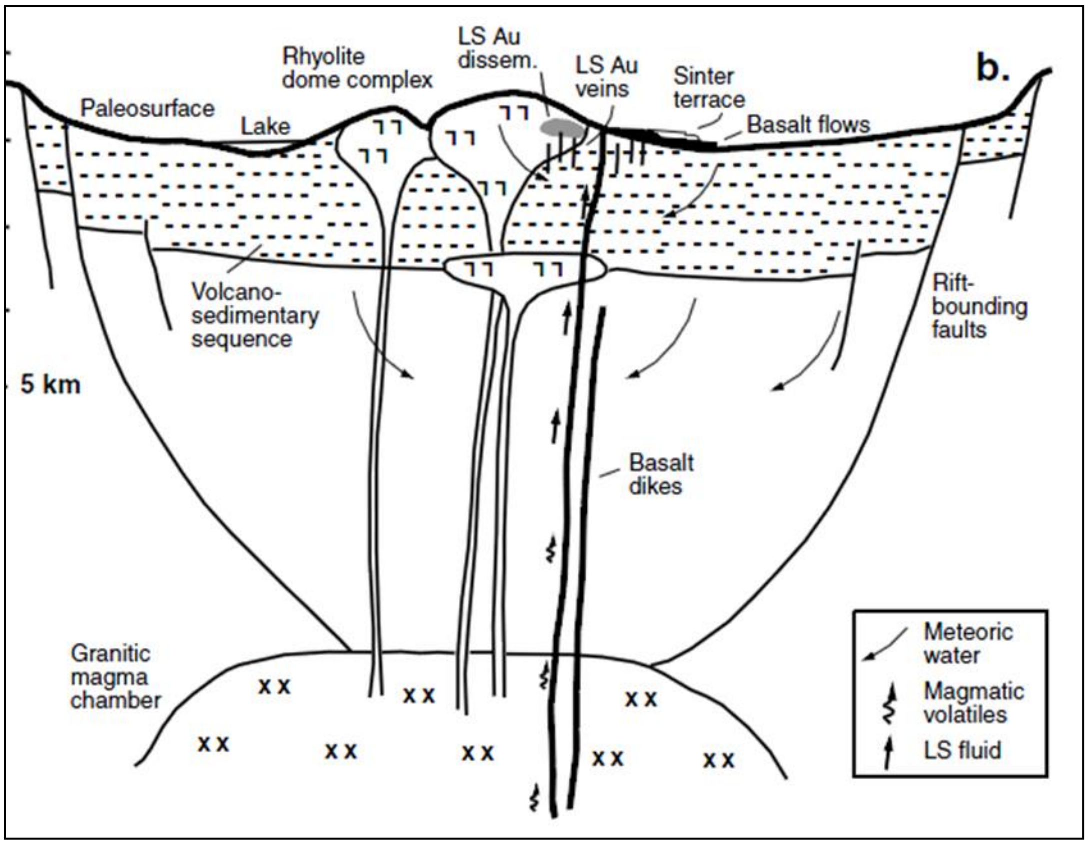
Figure 7-6: South Zone Cross Section



8.0 DEPOSIT TYPE

The mineralization identified at the Long Valley property is typical of the shallower portions of an epithermal, low-sulfidation type of gold-silver deposit. Other examples of this type of deposit, which share some similarities to Long Valley, include the McLaughlin deposit in California and the Hycroft (Sulfur) deposit in Nevada. In common with these deposits, gold and silver mineralization appears to have taken place at very shallow depths and is associated with a relatively recent volcanic-related hydrothermal system. In addition, the mineralized zones are typically associated with clay alteration (kaolinite) and silica replacement of volcaniclastic host rocks. This type of deposit typically contains very low amounts of base metals. A schematic diagram for this type of deposit model is shown in Figure 8-1. In the case of Long Valley, basalt flows are not present, and sinter is equivocal.

Figure 8-1: Schematic Model of a Low-Sulfidation Epithermal Mineralizing System

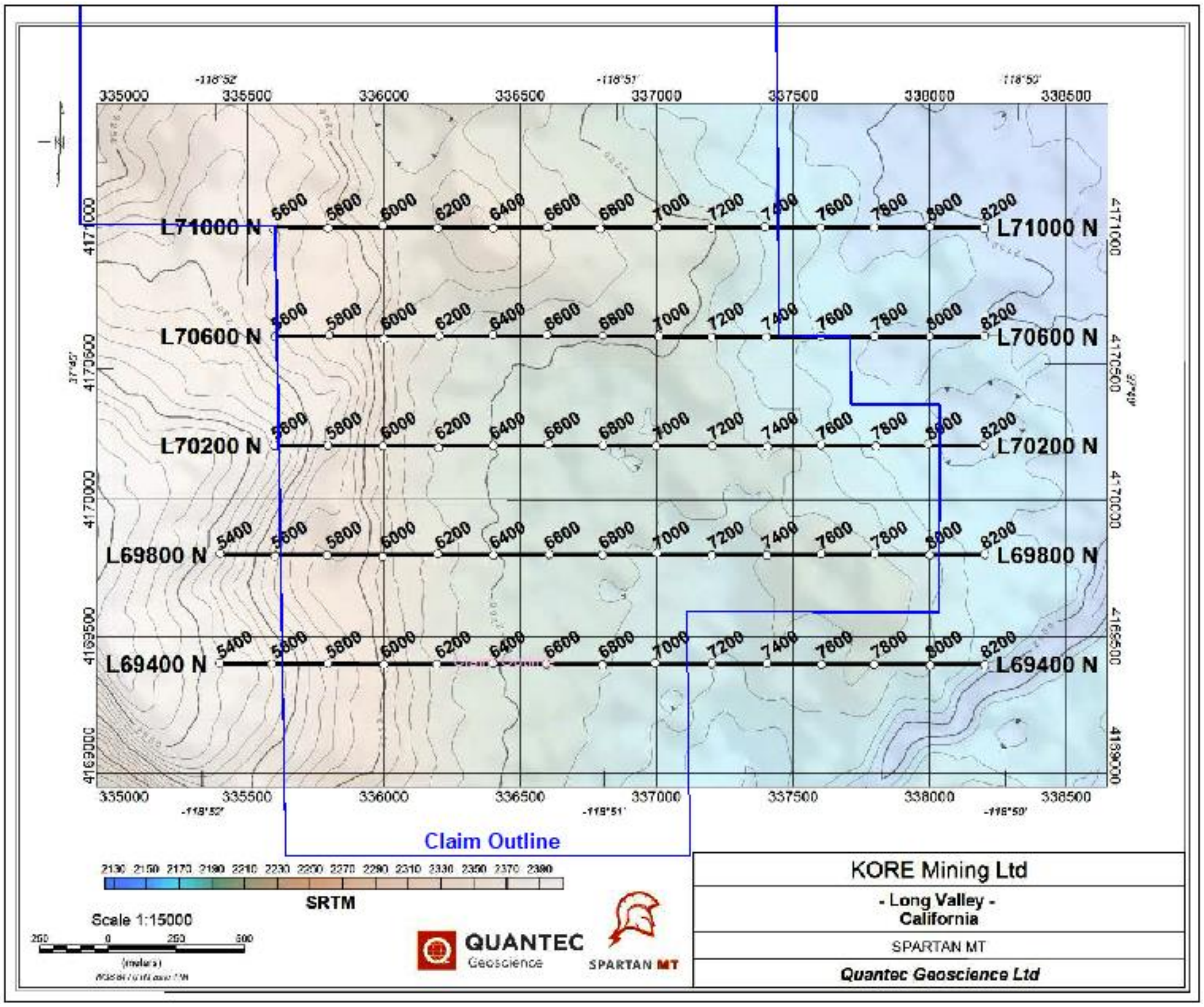


(After Sillitoe and Hedenquist, 2003)

9.0 EXPLORATION

Since acquiring the Long Valley project in March 2017, KORE commissioned a geophysical survey conducted from December 10th through December 20th, 2017, by Quantec Geoscience Ltd. A Spartan magnetotelluric ("MT") survey acquired data from 72 sites distributed along five survey lines that were oriented east-west on approximately 1,300-foot line spacing (Tournerie, 2018). Figure 9-1 shows the coverage area of the five lines, which total approximately 8.3-line miles and cross the southern portion of the property.

Figure 9-1: Magnetotelluric Survey Coverage Map



(Tournerie, 2018)

The instrumentation used for the survey included:

- Receiver systems: RT160Q Quantec data logger
- Synchronization: GPS clock (10 nanosecond precision)
- Receiver electrodes: Ground contacts using Quantec steel plates
- Magnetic sensors (HF): Geometrics G100K magnetic field sensors

- Magnetic sensors (LF): Phoenix MTC50 magnetic field sensors.

Tensor magnetotelluric soundings were processed with remote reference. The site configuration consisted of a cross-shaped electrode field with HF and LF magnetic sensors located at each site; the E-field dipole lengths were Ex: 100 meters and Ey: 100 meters. The remote site configuration consisted of cross-shaped E-fields with HF and LF magnetic sensors located at the site and oriented in the same direction as the local sites. The final processed data were presented as MT sounding curves of apparent resistivity and phase and as pseudo-section plots of observed XY and YX apparent resistivity and phase. Tournerie (2018) reported that the measured magnetotelluric data are of very good quality (smooth curves, and low errors) for the frequencies from 10kHz to 0.01Hz; more noise is observed for the lowest frequencies. A few sites were presenting more noise near 1Hz, but the sites were repeated at the end of the survey. The data sites have been improved for these repeated measurements.

The MT survey is expected to highlight silicified zones near the surface and identify structure suitable for mineralization at depth. As of the Effective Date of this report, KORE had not yet received the final report on interpretation of the survey results from their geophysical consultant.

In 2019 and 2020, geologists working for KORE re-logged RC cuttings from 232 of 896 drill holes. KORE geologists also conducted geological mapping, collected rock and soil samples and ran two lines of induced polarization and resistivity and ground magnetic geophysical surveys coinciding with the re-logged holes and soil sampling lines. KORE's rock-chip sample results and geology are shown along with the locations of the IP/Res lines. IP and resistivity results and their interpretation by KORE are shown in Figure 9-2, Figure 9-3, and Figure 9-4.

Figure 9-2: 2020 Surface Geology, IP Lines, Alteration and Grade-Thickness

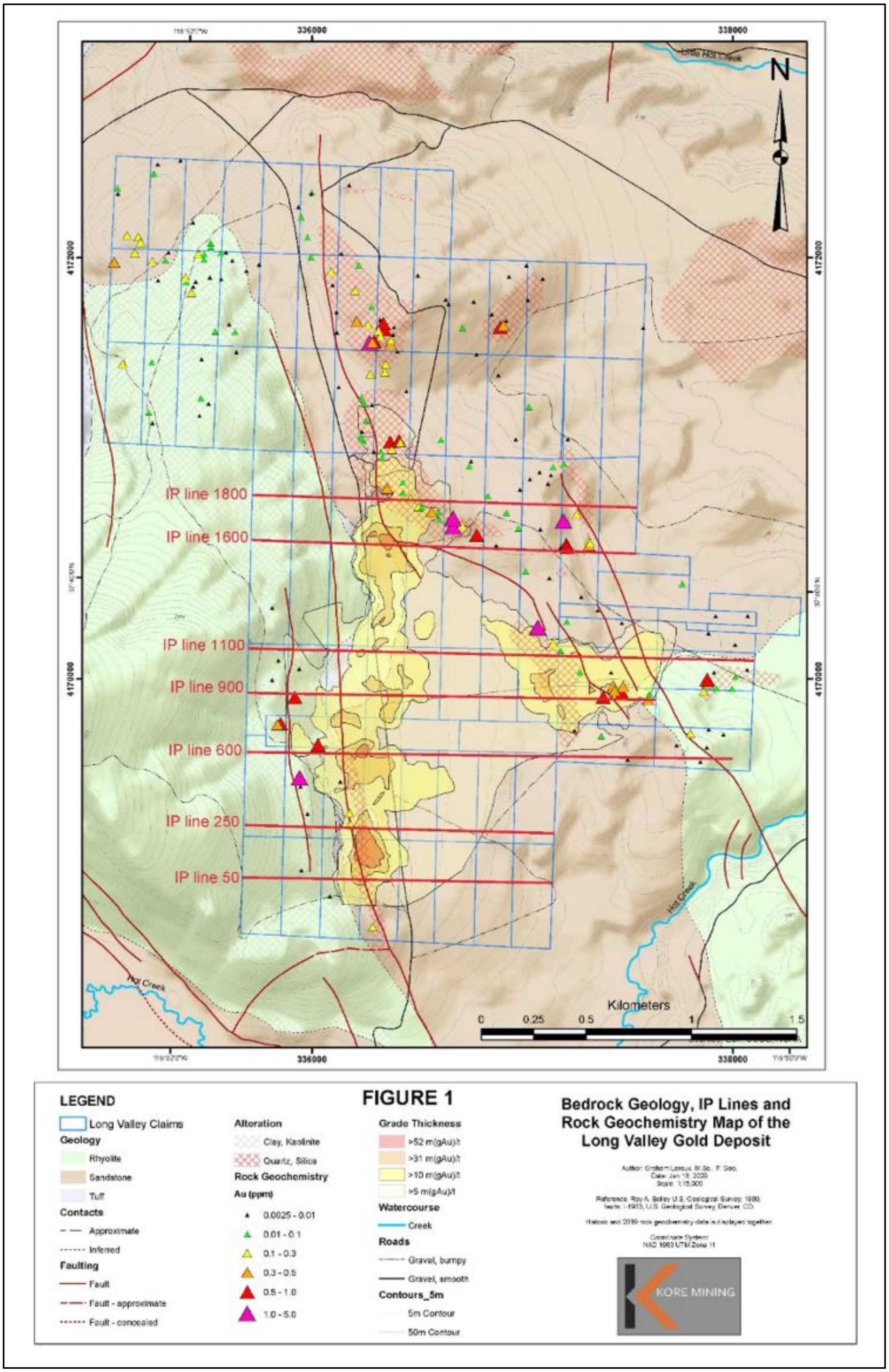
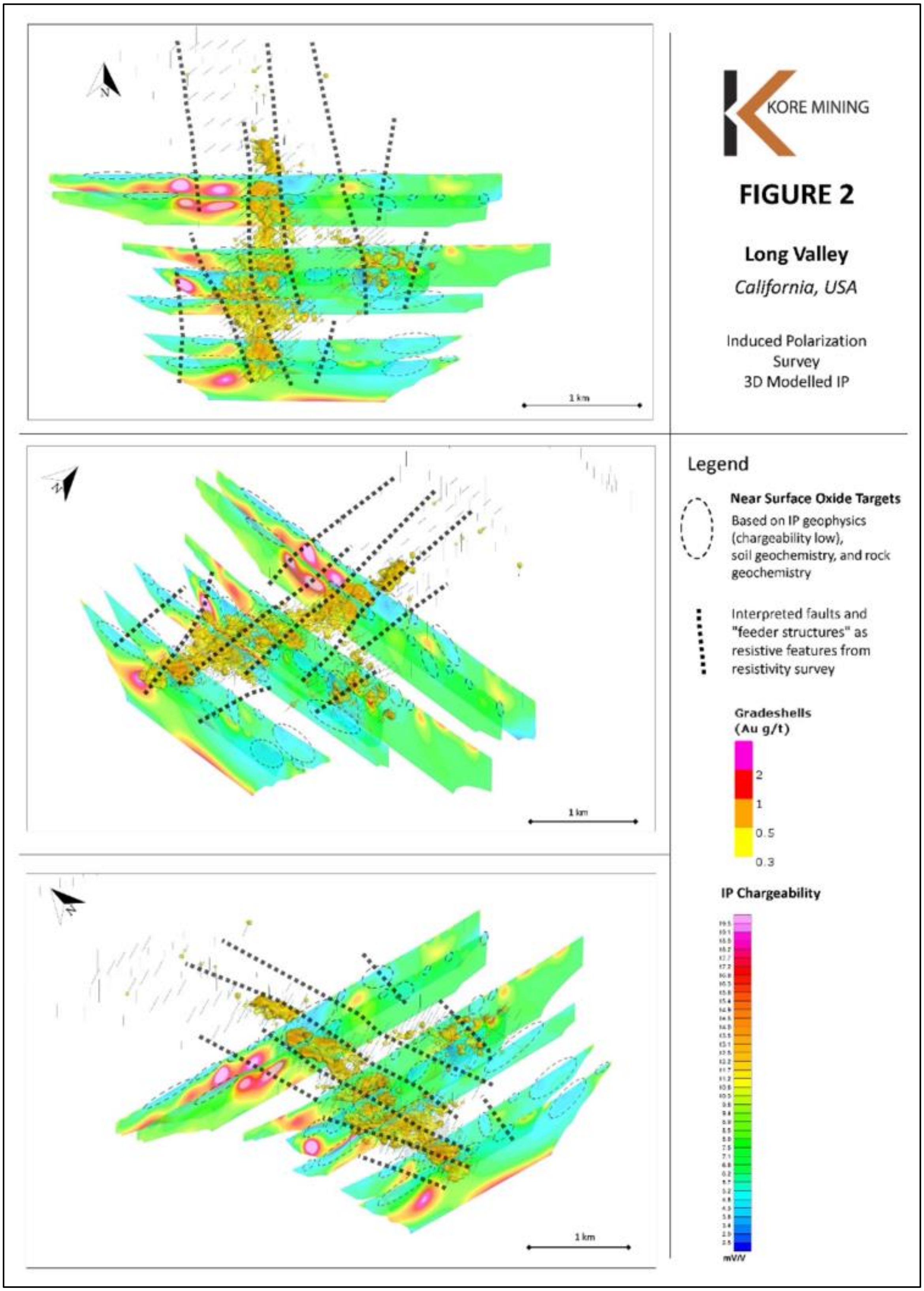
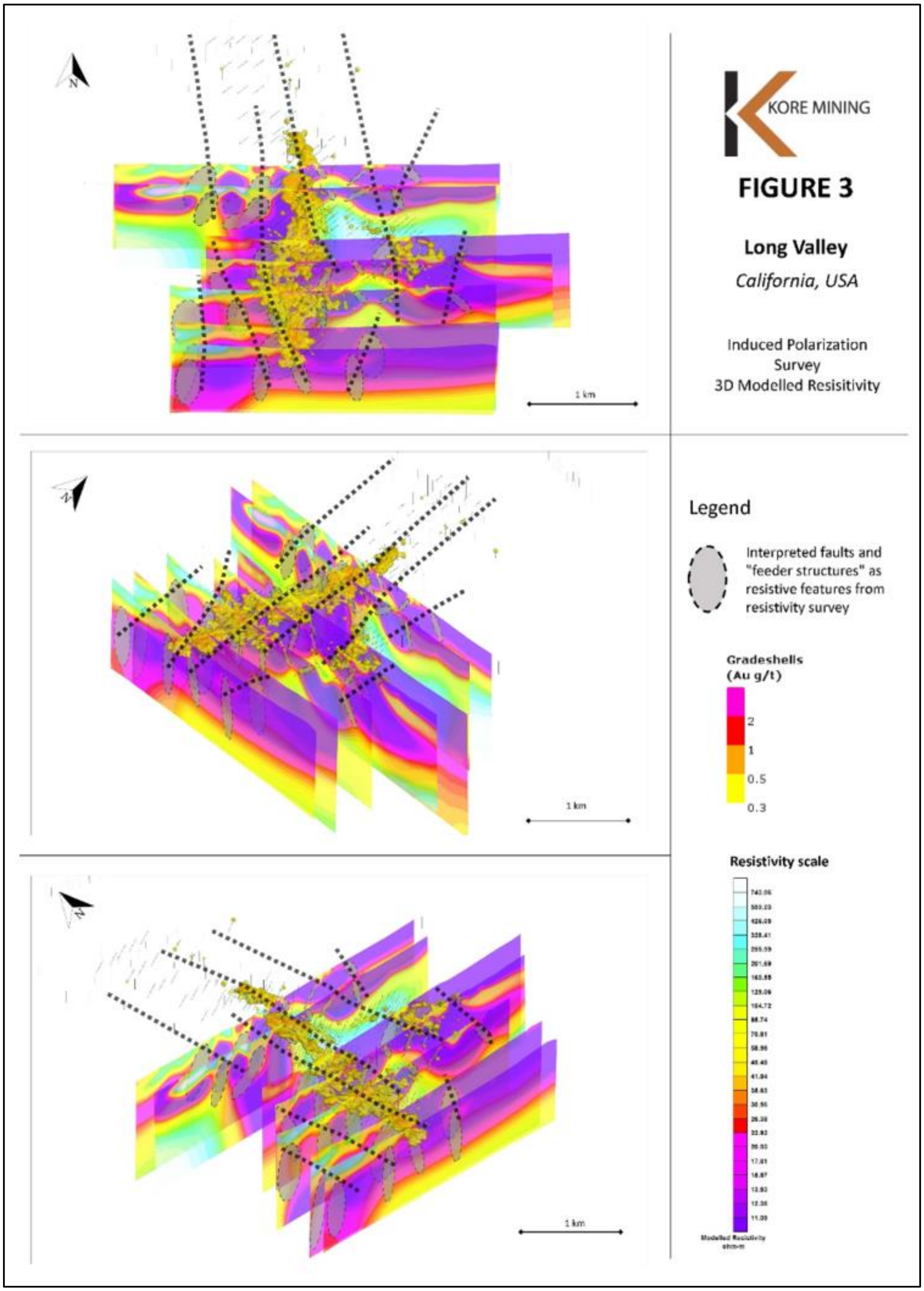


Figure 9-3: Plan Map of Near Surface Oxide Gold Anomalies from 2020 Chargeability



(from KORE, 2020)

Figure 9-4: KORE Structural Interpretation from Resistivity 2020



(from KORE, 2020)

10.0 DRILLING

10.1 Introduction

Table 10-1: summarizes the drilling on the property. The database contains 896 drill holes totaling 268,275 feet of drilling. Seven drill holes are missing coordinate information. There has been no drilling on the property since 1997, and there has been none conducted by the issuer.

Figure 10-1 is a map of the drill holes in the database, but the map does not include holes drilled north of the current property boundary.

Table 10-1: Long Valley Drilling Summary

Company	Year	RC Holes	RC Footage	Rotary Holes	Rotary Footage	Air Track Holes	Air Track Footage	Core Holes	Core Footage	Total Drill Holes	Total Footage
Freeport	1983-1984	80	18,615							80	18,615
Standard	1985			24	2,055					24	2,055
Royal Gold	1988					52	4,770			52	4,770
Battle Mtn.	1991	59	18,685							59	18,685
Royal Gold	1994-1997	615	206,410					10	1,491	625	207,901
Amax	1997	46	13,835					10	2,414	56	16,249
Totals		800	257,545	24	2,055	52	4,770	20	3,905	896	268,275

Most of the drill hole samples obtained from the property were from generally dry RC drilling, although when drilling below the water table, significant flows were encountered. Water was added when drilling dry to improve recovery.

No down hole surveys of the drill holes were performed, as the depth of most of the drilling was 300 feet or less.

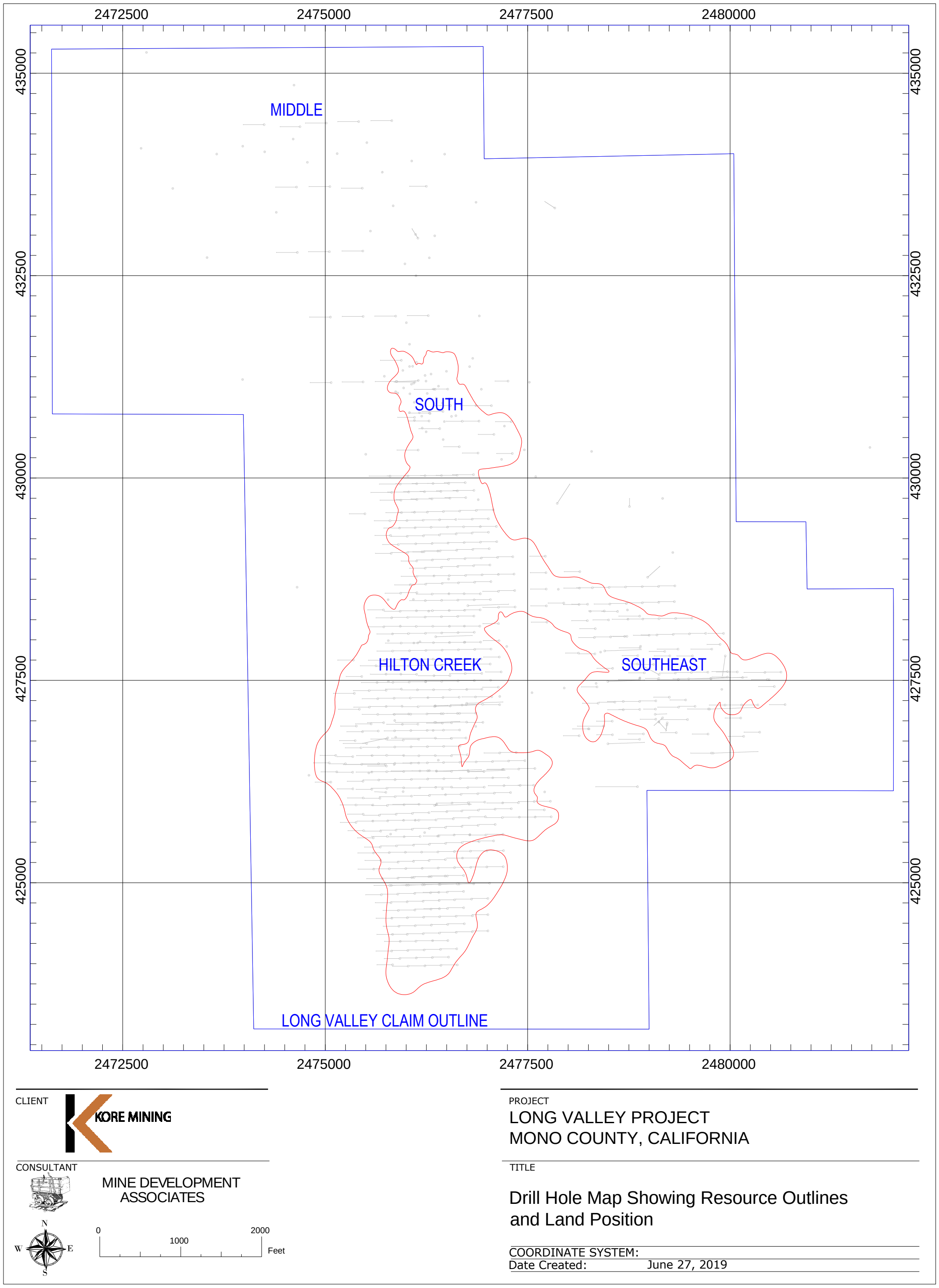
10.2 Air Track Drilling and Logging

During 1988, Royal Gold completed 52 shallow air track holes, mostly in the North zone. The 1988 Royal Gold air track drill holes were used to plot mineralized zones when modeling gold envelopes but were not used to estimate block grades. Royal Gold geologists completed geologic logs.

10.3 Reverse Circulation Drilling and Logging

Freeport, Standard, Battle Mountain, Royal Gold, and Amax completed 24 rotary and 800 RC drill holes on the property. Most of the drilling prior to 1993 was vertical, and most of the drilling after 1994 was angled. Royal Gold completed most of their RC drill holes by adding minimal amounts of water to normally dry drill holes drilled to about 300 feet. The water table was generally between 250 and 300 feet below the surface and, if intersected by drilling, added significant amounts of water. The deposit is in the area of nearby hot springs, and a few of the drill holes did intercept hot water. Drill holes were logged by geologists of the respective companies.

Figure 10-1: Long Valley Drill Hole Map



Note: Red outline shows surface projection of estimated mineral resources and limit of optimized pit.

Eklund Drilling of Elko, Nevada was the RC drilling contractor in 1996. TH60 and TH100 drills were used. Drill chips were logged in the field to paper log sheets using a hand-lens and binocular microscope.

10.4 Core Drilling and Logging

Royal Gold and Amax each completed 10 core holes on the property. Royal Gold logged the first two holes prior to shipment for assay. The remaining Royal Gold core holes were six-inch-diameter holes drilled in 1996 with a truck-mounted Longyear 38 drill and wireline methods. The 1996 core was logged in the field to paper log sheets and transported in its entirety by Royal Gold personnel in a rented moving van for use in column leach tests. At the metallurgical laboratory the whole core was blended together into a single composite.

The Amax core holes were drilled close to prior RC drill holes to compare the values. The Amax core was logged by the company geologists, and the whole core was shipped for assay.

10.5 Twin Hole Comparison

Table 10-2: shows the comparison of 10 core holes that were drilled proximal to existing RC drill holes on the property. The individual holes generally do not compare very well, with core holes giving both higher and lower gold values over selected intervals, but overall, the comparison is very close.

Table 10-2: Core vs. Proximal RC Drill Holes

Core				Reverse Circulation			
Core Hole	Number of Intervals	Average oz Au/t	Number > 0.007	RC Hole	Number of Intervals	Average oz Au/t	Number > 0.007
LV97-C11	44	0.018	30	LV96-323	44	0.020	34
LV97-C12	44	0.020	30	LV96-319	44	0.025	37
LV97-C12	45	0.019	30	LV96-399	45	0.011	29
LV97-C13	49	0.028	44	LV96-321	49	0.031	47
LV97-C14	59	0.009	23	LV97-561	59	0.003	1
LV97-C14	59	0.009	23	LV97-606	59	0.007	24
LV97-C15	47	0.015	17	LV96-474	47	0.016	21
LV97-C16	40	0.019	25	LV96-475	40	0.013	23
LV97-C17	29	0.008	16	LV91-033	29	0.027	25
LV97-C18	44	0.014	30	LV96-241	44	0.016	38
LV97-C19	40	0.026	36	LV96-378	40	0.021	38
LV97-C20	30	0.010	16	LV96-376	30	0.018	21
Total	**530**	**0.016**	**320**	**Total**	**530**	**0.016**	**338**

10.6 Drill Hole Statistics

The drill hole data for the project are summarized in Table 10-3:. Seven of the drill holes were missing coordinate data, and about 4,800 intervals were missing assays, either due to the sample not being recovered or the interval not being assayed. A large number of the missing intervals occur above the known mineralized zone and were typically from the 1995 drilling.

Table 10-3: Drill Hole Information Summary

Item	Value				
Number of Drill Holes	896				
Number of Core Holes	20				
Drill Hole Footage	268,275				
Drill Hole Intervals	52,656				
Drill Hole Assays	47,792				
Drill Hole Missing Intervals	4,864				
Down Hole Survey	0				
Note: 2 Drill holes with missing coordinates not loaded to database					
Item	**Hole ID**	**North**	**East**	**Elevation**	**Depth**
Minimum Northing	LV83-056	437,978	472,261	7,225	40
Maximum Northing	LV84-069	474,261	439,836	7,200	200
Minimum Easting	LV97-590	423,938	476,028	7,250	300
Maximum Easting	LV91-047	430,330	481,700	7,200	430
Minimum Elevation	LV95-098	426,149	478,845	7,149	800
Maximum Elevation	LV84-072	431,167	473,995	7,436	245
Minimum Depth	RG-047	430,423	476,265	7,321	10
Maximum Depth	LV95-019	426,343	476,376	7,219	865

The statistics for the gold assay data are summarized in Table 10-4: by drill type, Table 10-5: by company, and Table 10-6: by deposit area.

Table 10-4: Drill Hole Assay Statistics by Drill Type

Drill Type	Number Samples	Mean oz Au/t	Minimum oz Au/t	Maximum oz Au/t	Std.Dev.	CV
RC	45,857	0.009	0.000	0.890	0.015	1.73
Core	576	0.015	0.000	0.149	0.020	1.31
Rotary	405	0.013	0.000	0.302	0.019	1.52
Air Track	954	0.013	0.000	0.120	0.013	1.03
All	47,792	0.009	0.000	0.890	0.015	1.70

Table 10-5: Drill Hole Assay Statistics by Company

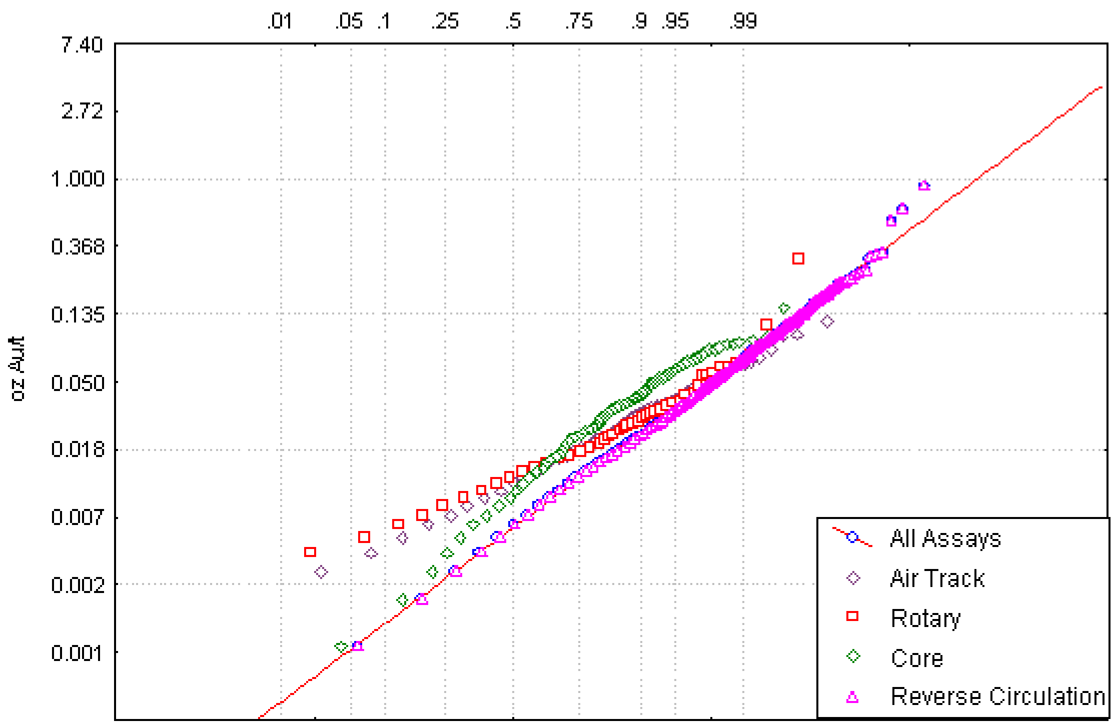
Company	Number Samples	Mean oz Au/t	Minimum oz Au/t	Maximum oz Au/t	Std.Dev.	CV
Freeport	2,672	0.004	0.000	0.530	0.012	2.99
Standard	405	0.013	0.000	0.302	0.019	1.52
Battle Mountain	3,341	0.007	0.000	0.245	0.010	1.46
Royal Gold	38,204	0.009	0.000	0.890	0.016	1.68
Amax	3,170	0.008	0.000	0.149	0.012	1.37
All	47,792	0.009	0.000	0.890	0.015	1.70

Table 10-6: Drill Hole Assay Statistics by Deposit Area

Area	Number Samples	Mean oz Au/t	Minimum oz Au/t	Maximum oz Au/t	Std.Dev.	CV
Hilton Creek	31,138	0.010	0.000	0.328	0.016	1.55
South	4,471	0.008	0.000	0.530	0.014	1.74
South East	7,714	0.008	0.000	0.890	0.016	2.01
North	2,121	0.003	0.000	0.099	0.005	1.77
Central	2,091	0.002	0.000	0.042	0.004	1.72
No Area	257	0.001	0.000	0.014	0.002	1.30
All	47,792	0.009	0.000	0.890	0.015	1.70

Figure 10-2 shows the distribution of the air track and conventional rotary samples to be different from RC samples. Samples from core drilling have a higher-grade distribution.

Figure 10-2: Long Valley Sample Distribution by Drill Type



10.7 Summary Statement

The author believes that the drilling sampling procedures provided samples that are representative and of sufficient quality for use in the resource estimations discussed in Section 14.0. The author is unaware of any sampling or recovery factors that materially impact the mineral resources discussed in Section 14.0.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

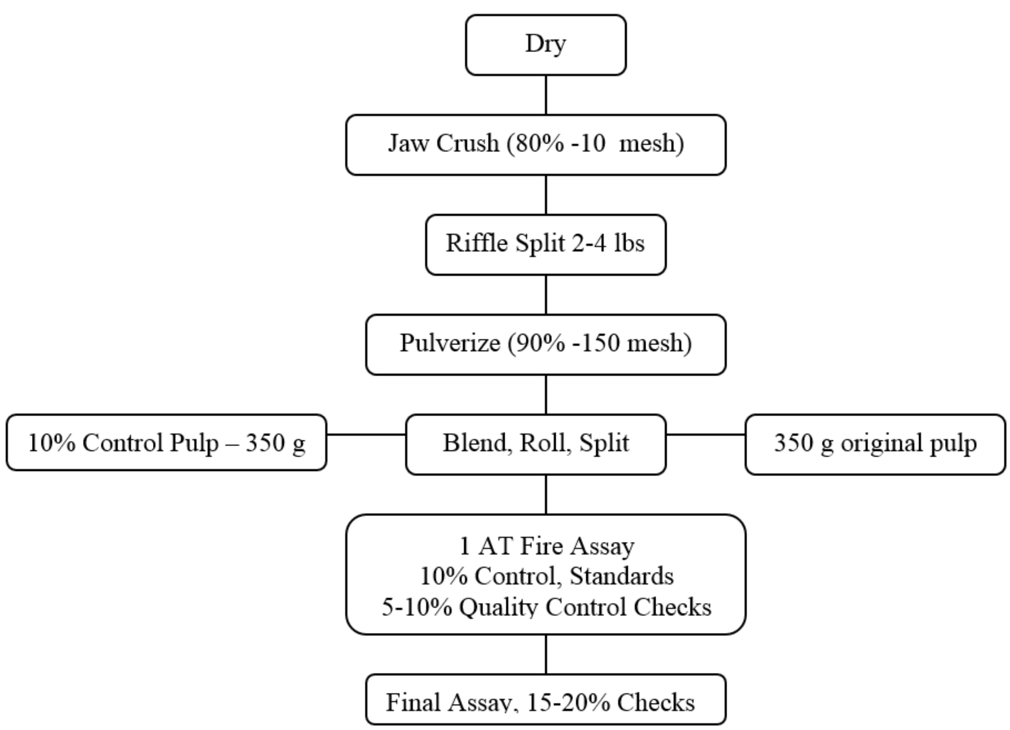
11.1 Historical Drill Sample Preparation and Analysis

Little is known about the sampling procedures prior to 1994. Freeport's samples were analyzed by Monitor Labs, who used *aqua regia* dissolution, followed by atomic absorption ("AA") analysis of the samples. Monitor Labs was independent of Freeport and all later operators of the project. Battle Mountain used Barringer Laboratories and Bondar Clegg Laboratories for sample preparation and fire assaying (AA finish) of one assay ton pulps. Both of these laboratories were independent of Battle Mountain and later operators of the project. It is not known what certifications, if any, these laboratories maintained at the time.

Sampling procedures starting in 1994 were well documented. Royal Gold's RC samples, taken in five-foot intervals, were collected and bagged at the drill site by taking a 5 to 10-pound split of each sample from the drill holes. Sample bags were sealed by the drill crew and not opened until they reached American Assay Labs ("American Assay") in Sparks, Nevada. The assay lab picked up the samples at the drill site, transported them to the lab, dried the samples, then crushed, split, pulverized, and blended them to obtain assay pulps. Most of the assays were completed by fire assay methods with an AA finish. No duplicate samples were taken routinely at the rig (Martin et al., 1997a). American Assay was independent of Royal Gold and subsequent operators of the project. It is not known what certifications, if any, this laboratory maintained at the time.

American Assay used the flow sheet shown in Figure 11-1 to prepare and assay the samples received from Royal Gold, most of which weighed from five to 10 pounds.

Figure 11-1: American Assay Lab Sample Preparation and Assaying Procedure



A similar procedure was used by Amax, but their samples were analyzed by Chemex Labs ("Chemex"). Amax collected samples that ranged in size from five to 20 pounds (lbs) at the drill hole, then bagged and shipped the samples to Chemex for sample preparation. The samples were dried, weighed, crushed, blended, split, and pulverized to obtain a 600 gram sample to make assay pulps. Chemex completed fire assays with AA finish from one assay ton pulps. Chemex was independent of Amax and subsequent operators of the project. It is not known what certifications, if any, this laboratory maintained at the time.

Royal Gold collected the samples from their first two core holes at the drill site, placed them in core boxes, and sent the whole core to American Assay's sample preparation facility to split by sawing, prepare, and assay the samples. Half of the core was assayed, and the remaining half in the highly mineralized intervals was used for bottle roll tests. Samples were either grouped by rock type within 5-foot intervals or prepared in 5-foot intervals. The remaining core holes drilled by Royal Gold were large-diameter holes used for metallurgical testing.

Amax prepared assay samples from core holes by crushing whole core and then following the RC sample preparation and assaying methods.

11.2 Historical Sample Security

Samples were sealed in bags at the site and collected by commercial laboratory personnel.

11.3 Historical Quality Assurance/Quality Control Check Samples, Check Assays, Standard Check Assays

For the report of MDA (2003), duplicate-sample assays and check-sample assays were compiled and evaluated by MDA as summarized below:

Freeport completed check assays on about 40 samples, which indicated good agreement (0.011 vs. 0.012 oz Au/ton). Several drill holes completed during 1994 by Royal Gold were assayed by using one and two assay-ton pulps for comparison. Table 11-1: shows a comparison of these checks. The one assay ton and two assay ton results compare favorably when one sample is omitted from drill hole LV94-014.

Table 11-1: Long Valley Check Assays – 1 AT vs 2 AT

Drill Hole	From	To	1 AT oz Au/t	2 AT oz Au/t
LV94-002	200	225	0.150	0.143
LV94-002	220	225	0.030	0.030
LV94-002	240	245	0.019	0.018
LV94-003	95	100	0.022	0.016
LV94-003	115	120	0.113	0.112
LV94-003	130	135	0.013	0.012
LV94-004	495	500	0.019	0.021
LV94-004	535	540	0.044	0.043
LV94-004	560	565	0.026	0.019
LV94-004	580	585	0.012	0.011
LV94-014	480	485	0.063	0.042
LV94-016	20	25	0.013	0.012
LV94-016	40	45	0.020	0.020
All Checks			**0.042**	**0.038**

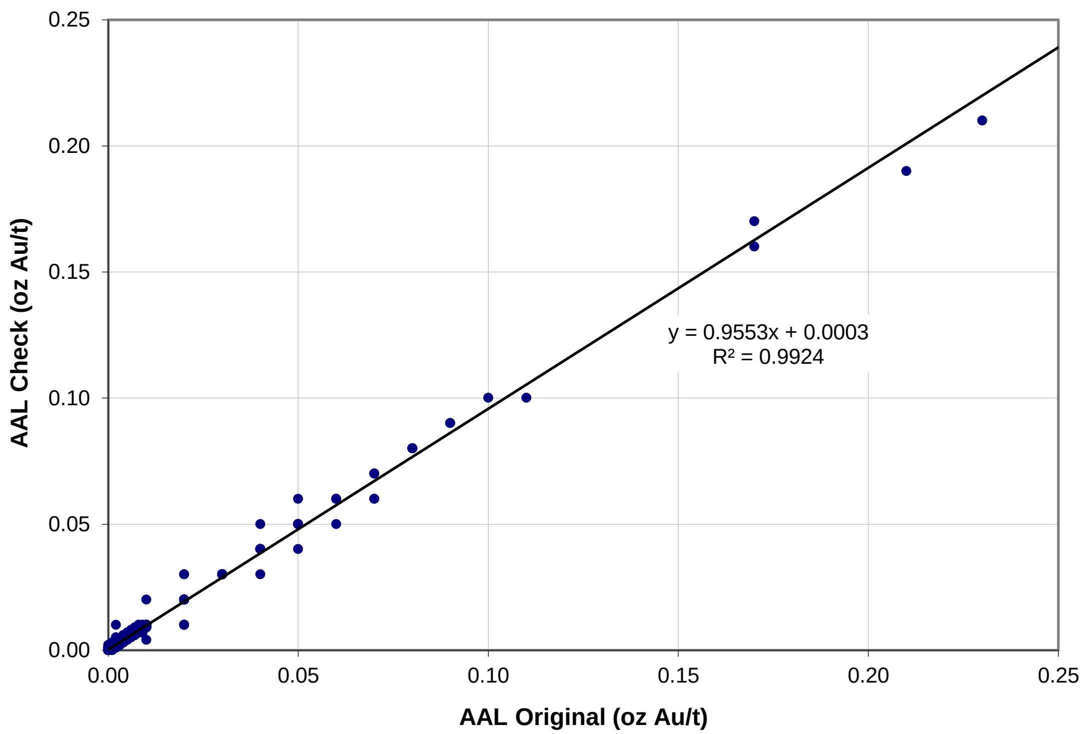
During Royal Gold's 1996 drilling, six large samples were collected from drill hole LV96-311. Each sample represented about half of the total material collected at the drill hole interval. To compare a larger sample to the typical 5 to 10-pound split samples, the entire sample was reduced to -85 mesh prior to taking any splits of the sample. The comparison of these samples to the original sample is shown in Table 11-2:, these samples compare well as shown in the table. This analysis shows that the rotary splits are representative samples for use in this report.

Table 11-2: Long Valley Bulk Sample Assays vs. Original Assays

Sample ID	Original Assay oz Au/t	Bulk Sample Assay oz Au/t
60	0.002	0.004
70	0.046	0.052
85	0.015	0.016
95	0.004	0.003
110	0.046	0.041
125	0.020	0.017
Average	**0.022**	**0.022**

Royal Gold used American Assay for all their drill hole sample assaying. American Assay completed 876 duplicate sample checks or repeat assays on the same pulp as part of their normal assay procedure, which indicated good agreement, as shown in Figure 11-2.

Figure 11-2: American Assay Lab Check Assay Results



Over 3,300 check assays were completed on sample pulps, and about 350 checks were done on coarse reject material. The results were compiled by MDA (2003) and compared as shown in Figure 11-3 and Figure 11-4, respectively. These assays were performed by Chemex for Amax and compare well with the American Assay analyses for Royal Gold, although the checks tend to be slightly lower in grade.

Figure 11-3: Check Assays on Sample Pulps
(number of samples = 3,372)

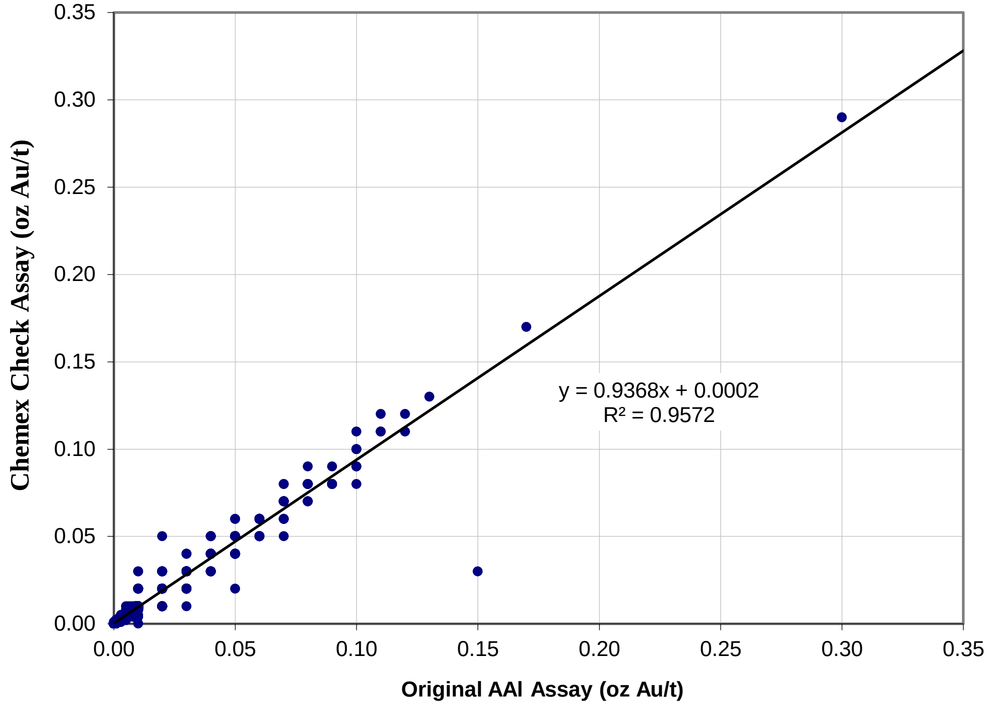
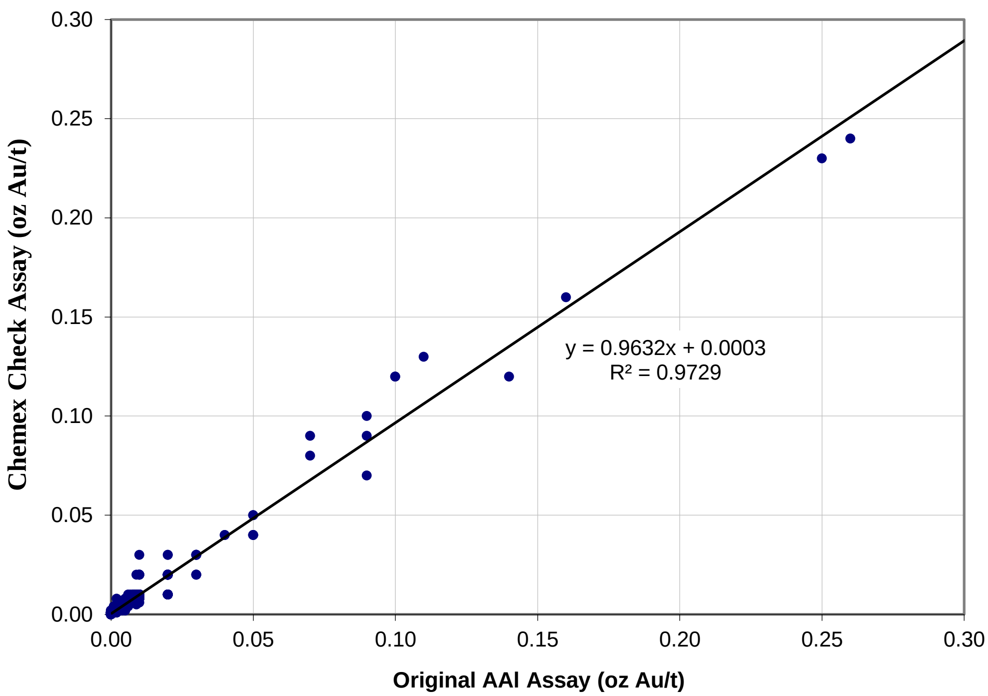
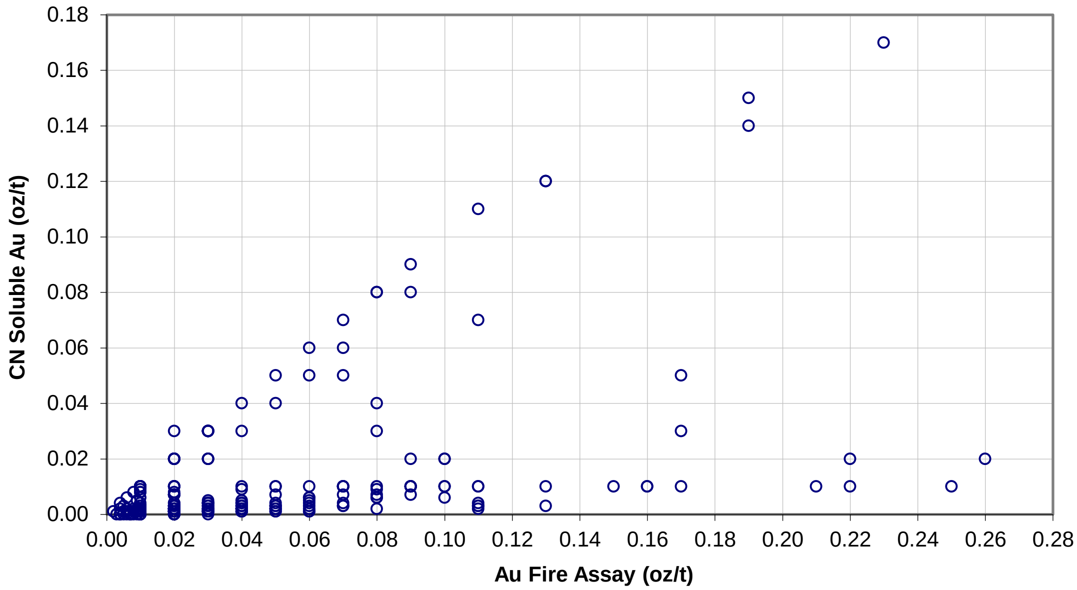


Figure 11-4: Check Assays on Coarse Rejects
(number of samples = 352)



A total of 305 cyanide soluble test results were compared to American Assay fire assays. This comparison demonstrates a wide range in response, with many of the samples having significantly lower cyanide soluble assay than fire assay, which can be used to indicate metallurgical properties. Further, this suggests that the oxide, mixed, and sulfide boundaries must be carefully drawn as the metallurgical response from sulfides is considerably different than that from oxide materials. This information is shown in Figure 11-5.

Figure 11-5: American Assay Lab – Au Fire Assay vs. Cyanide Soluble Au



11.4 Comments

While documentation of sample preparation, analysis, and security for the various companies that operated at Long Valley prior to 1994 is incomplete, all of the companies were reputable, well-known mining or exploration companies that likely followed accepted industry practices. All of the laboratories discussed above are, or were, well-known independent commercial analytical laboratories. The assaying described in this report was completed prior to the institution of formal certifications for analytical laboratories.

MDA has compiled and evaluated historical duplicate- and check-sample assays and concludes these data support the use of the assay data in resource estimation. MDA believes the sample preparation, security, and analytical procedures used by previous operators of the Long Valley project were acceptable procedures and the resulting analytical data are of sufficient quality for use in the resource estimation.

12.0 DATA VERIFICATION

12.1 Verification by Mr. Neil Prenn – Drilling Database QP

Mr. Prenn supervised, and takes full responsibility for, the verification of the Long Valley drilling database. That verification was conducted in 2003 and that database has not been subsequently modified. Although not described by MDA (2003), the database verification was accomplished by a detailed examination of data limited to 51 drill holes, or about 6% of the drill holes in the project area. Hole locations, sample numbers, assays, and interval depths in the project database were visually compared to copies of drill logs and laboratory assay certificates. Where errors in database entries were found, the database was corrected using values from the assay certificates. Mr. Prenn reviewed written notes specifying the data compared to the logs and laboratory certificates, and corrections made to the database. Mr. Prenn found, in his opinion, the corrections to be acceptable and no further database verification to be necessary.

A limitation to data verification was that Mr. Prenn did not observe any of the historical drilling while it was in progress to assess the drilling and sampling methods and procedures. During the initial site visit in 2002, Mr. Prenn observed the reclaimed drill roads and pads and verified with visual inspection evidence that the historical drilling had been conducted in the area shown on historical maps. However, due to the reclamation, precise determination of hole collar locations could not be made, but this limitation is considered low risk due to the sub-horizontal geometry of the deposit. During that visit, Mr. Prenn also collected 10 surface samples for independent verification of rock density data. That verification data is discussed in Section 14.4.

On February 21, 2018, Mr. Prenn traversed the property and verified by personal inspection that there were no areas of recent disturbance that would indicate material drilling or other exploration activities were conducted since his visit in 2002. Mr. Weiss conducted a personal inspection of the property on September 20, 2020. Mr. Weiss traversed the South, Hilton Creek and Southeast zones and verified by visual examination that there were no areas of disturbance that would indicate drilling or trenching were conducted since the visit of Mr. Prenn in 2018. No disturbances were observed from the geophysical surveys and surface geochemical sampling conducted by KORE. Mr. Weiss concludes there has been no material exploration work done since Mr. Prenn's visit in 2018, other than the work by KORE summarized in Section 9.0.

Mr. Prenn concludes, based on the site visits in 2002, 2018 and 2020, the database verification conducted in 2003 under Mr. Prenn's supervision, and including his evaluation of the Quality Assurance/Quality Control (QA/QC) check assay and density results, that the project drilling data are of sufficient quality and are adequate for the purposes used in this report.

MDA has maintained in storage files for the project that includes the original assay information, Muerhoff data review notes, drill hole databases, core photographs, metallurgical test reports and other documents from past years of work on this project. In addition, KORE has RC chips for at least 626 RC drill holes, assay certificates for most RC and core holes, check assay certificates, core hole photographs, some RC chip photographs, maps, drill collar surveys, metallurgical testing reports, and density test reports.

12.2 Verification by Mr. Steven Weiss – Geology and Resource QP

Mr. Weiss inspected the surface geology of the property on September 20, 2020, and verified though personal inspection that the surface geology of the property summarized in Item 7 is materially correct and consistent with the regional map of the Long Valley caldera area published by the United States Geological Survey (Bailey, 1989). During May through November of 1996, while employed as as a Research Associate in the Department of Geological Sciences at the University of Nevada, Reno (UNR) with funding provided by Royal Gold for an independent research project, Mr. Weiss inspected and logged all of the 1996 RC and core holes drilled by Royal Gold that year, re-logged many of the pre-1996 drill holes and conducted petrographic and mineralogic studies of then-existing drill core. Although UNR received funding for the research project, Mr. Weiss was not an employee of the property owner, and has remained independent of the property and the owners of the property. Based on that work, as well as his most recent personal inspection of the property in 2020, Mr. Weiss verifies the subsurface geology, and style and extent of mineralization summarized in Section 7 are materially correct, suitable for use, and are consistent with the deposit model summarized in Item 8.

12.3 Verification by Dr. Todd Harvey – Metallurgy QP

Metallurgical testing was completed for the Long Valley project by a number of well-known commercial metallurgical laboratories and operating mines from 1989 to 1997. Dr. Harvey reviewed all available metallurgical reports. Dr. Harvey reviewed the sample selection and compositing used in the metallurgical test work and found that the selection of samples was representative for this type of deposit and geology. Dr. Harvey reviewed the grades of the various samples selected for testing and verified the grade of material tested represents a spread of grades from very low grade to high grade that is typical for the grades found in the Long Valley deposit. Dr. Harvey also reviewed the process for preparing sample composites and found the selection of fresh core to be suitable for this level of study. Dr. Harvey verified the metallurgical test work and samples to be representative spatially for this deposit as well. Dr. Harvey while performing his data analysis performed several mathematical tests to validate the metallurgical balances presented in the test work and he found the data presented in the metallurgical reports to be consistent with practices performed by reputable independent test laboratories. Dr. Harvey confirmed that the mineralization found at the Long Valley Project is similar to mines where Dr. Harvey has performed other consulting work and finds that the test work for Long Valley shows that the material behaves in a very similar manner, specifically in gold recovery and reagent consumption. Given the similarities of the Long Valley material to other similar operations, this provides a good basis for benchmarking the metallurgical test work to actual crush and agglomerate heap leach mines for validating the finding of the test work. His complete discussion of the test work is provided in Section 13.0. The work appears to be professionally completed and is well documented and is suitable for estimation of heap leach gold recovery calculations in this PEA.

12.4 Verification by Ms. Terre Lane – Mine Planning and Evaluation QP

Visual and statistical verification of the resource block model prepared by MDA, was performed by Ms. Lane of GRE by stepping through the model in section and in plan, to determine if the block model matched the geological interpretation and the rock types presented in the geological sections of this report and the block model was determined to properly corollate to the mapped and interpreted rock types. The model was also checked to ensure that blocks were properly projected to the topographic surface. The block size was evaluated to determine if the block size was an appropriate size for use as the selective mining unit for bulk open pit production and mine planning, the 20-foot by 20-foot by 10-foot blocks are felt to be appropriate for the mineralization type and the likely mining method and production rate. Ms. Lane of GRE then utilized the block model to create the mine plan, production schedule, and economic analysis for the Imperial Project.

Mining and processing methods, costs and infrastructure needs were verified by comparison to other similar sized open pit heap leach mines operating in the western USA and experience of the QPs, (Ms. Lane and Dr. Harvey). Costs were developed from vendor quotations and comparisons to published and internal data used by the QPs in the preparation of similar studies. Not all costs were competitively bid but costs were benchmarked to similar nearby operations and unit costs of major consumables were also benchmarked to nearby operations. Other cost data used in the report was sourced from the most recent Infomine cost data report. All costs used in the analysis were verified and reviewed by Ms. Lane and were assessed to be current and appropriate for use. Finally, after the economic study was performed the overall operating costs for different aspects of the operation (mining, process, and general & admin) were benchmarked against similar sized mines and recent feasibility studies to determine if they were similar, the results did benchmark well to other operations and economic studies.

The taxation rates used and applied were values available from US government sources at the time of the economic analysis.

A geotechnical analysis of pit slopes has not been prepared for the Project, so an assumption of 45 degrees used was used for inter-ramp pit slope angles. This slope angle value is consistent with other shallow pits for this rock type and alteration type. Given the near surface nature and low strip ratio, there is a low risk in using this slope angle at this level of study.

The topography used in the pit designs was the same as used by Mr. Prenn of MDA and was reviewed in comparison to local topography available on the Internet such as Google Earth and the US Geological Survey's web site.

Cost data used in the report was sourced from Infomine and local vendors. It was verified as current by Ms. Lane.

13.0 MINERALOGICAL PROCESSING AND METALLURGICAL TESTING

A variety of test work has been conducted on material from the Long Valley project between 1989 and 1997. No further test work has been completed since that time. Additional test work is recommended to confirm the conclusions presented.

13.1 Specific Gravity and Bulk Density Measurements

Hazen Research measured the specific gravity of the material used for the 1995 test campaign (Hazen Research, Inc., 1997). To measure the density of the material, randomly selected samples were dried, weighed, dipped in melted wax, and then weighed again to measure the difference. The results of these tests are given in Table 13-1.

Table 13-1: Specific Gravity Data Obtained by Hazen Research in 1995

Rock ID	Dry Weight (g)	Weight with Wax (g)	Volume (cc)	Wax Weight (g)	Computed Wax Volume (cc)	Rock Volume (cc)	Specific Gravity kg/L
C3 140 48622-2	482.2	536.7	360.1	54.5	47.39	312.71	1.54
C3 148 48622-3	1493.8	1542	656.8	48.2	41.91	614.89	2.43
C4 135 48622-5	892.6	946.2	485.8	53.6	46.61	639.19	2.03
C5 148 48622-7	871	903.7	381.7	32.7	8.44	353.26	2.47
C7 99 48622-9	1089.6	1133.8	484.3	44.2	38.44	445.86	2.44
C8 115 48622-12	741.8	785.4	388.5	43.6	37.91	350.59	2.12
C8 148 48622-13	565.4	612.6	351.1	47.2	41.05	310.05	1.82
Average	876.63	922.91	444.04	46.29	37.39	432.36	2.12

Specific gravity was also determined with an air-comparison pycnometer. The results of these tests were significantly different from the ones reported above; Hazen concluded that this discrepancy was due to porosity, voids, and cracks in the material. The results were run in duplicate, and the average results are given in Table 13-2.

Table 13-2: Specific Gravity Data Obtained Using a Pycnometer

Sample ID	Dry Weight (g)	Volume (cc)	Specific Gravity (kg/L)
C3 123 48622-1	39.65	14.98	2.65
C3 140 48622-2	47.35	17.85	2.65
C3 148 48622-3	62.65	23.72	2.64
C4 35 48622-4	53.20	19.90	2.67
C4 135 48622-5	58.30	21.85	2.67
C5 99 48622-6	49.15	18.41	2.67
C6 44 48622-8	60.85	22.94	2.65
C7 113 48622-10	45.25	16.85	2.69
C8 105 48622-11	40.45	15.18	2.66
C10 40 48622-14	39.20	14.76	2.66
Average	49.61	18.64	2.66

Hazen Research also conducted bulk density analyses of the Long Valley material as part of the 1995/1996 test work campaign (see below). The measurements included those of "as loaded" material in leach test columns (numbers 2 – 5), as well as the leach residue after the column leach tests were completed (Hazen Research, Inc., 1997). The results of these measurements are given in Table 13-3.

Table 13-3: 1996 Hazen Research Bulk Density Measurements on Column Leached Samples

Test No.	Mineralized Material Bed Condition	Loaded Mineralized Material					Leach Residue				
		Bed Height (m)	Bed Volume (m^3)	Dry Weight (kg)	Bulk Density (kg/m^3)	Void Space (%)	Bed Height (m)	Bed Volume (m^3)	Dry Weight (kg)	Bulk Density (kg/m^3)	Void Space (%)
2	Dry Loaded (10" x 6')	1.88	0.10	132.13	1388.7	47.6%	1.83	0.09	130.27	1406.9	46.8%
3	Agglomerated (10" x 6')	1.83	0.09	104.24	1125.7	57.5%	1.73	0.09	102.06	1166.4	56.0%
4	Dry Loaded (24" x 19')	5.73	1.68	2283.39	1363.0	48.6%	5.49	1.60	2180.42	1361.6	48.6%
5	Agglomerated (24" x 22')	6.63	1.93	2178.15	1125.7	57.5%	5.66	1.65	2106.49	1274.5	51.9%
Average		4.02	0.95	1174.48	1250.80	52.8%	3.68	0.86	1129.81	1302.34	50.8%

The average bulk density for the material used for column leach test work was 1250 kilograms per cubic meter (kg/m^3), and this measurement increased to 1302 kg/m^3 after leaching, most likely due to entrained water in the changes in the mineralized material due to leaching. The void spaces estimated in the column also decreased from 52.8% at the start of the tests to 50.8%, indicating that only minor slump occurred during the leach.

McClelland Laboratories also conducted specific gravity tests on select drill core intervals to corroborate the above findings. The same wax-coat technique was used as with the Hazen samples. The results of these tests are given in Table 13-4. For all drill core samples taken from the Long Valley deposit, rock type was determined at the time of drilling, with some samples reclassified later after examination for statistical analysis.

Table 13-4: Specific Gravity Data Obtained by McClelland Laboratories (1996)

Sample No.	Interval	Weight % Moisture (As Received)	Specific Gravity	Rock Type (If Available)
95C1	165	3.0	1.80	Sulfide
95C1	175	0.8	1.63	Sulfide
95C2	120	1.8	1.25	
95C2	175	1.0	1.26	
95C2	195	0.2	2.80	
Average		1.4	1.75	

The average specific gravity of the McClelland samples tested was 1.75. The specific gravities of the McClelland samples seem to be moderately lower than those that were tested at Hazen, however no data could be found on the types of rock tested at Hazen, so it is difficult to compare the results of these measurements.

13.2 Bottle Roll Test Work

Numerous bottle roll test work campaigns were completed on the Long Valley material from 1989 to 1995. This work includes tests on drill hole samples, composites, as well as a bulk sample used for column leach test work completed in 1995 (Hazen Research, Inc., 1997).

13.2.1 1991 Battle Mountain Test Work

The 2018 Technical Report on the Long Valley property by MDA (MDA, 2008) notes that in 1991 RC drill core samples were tested by Battle Mountain Laboratories (Mine Development Associates, 2018). At the time of writing, Dr. Harvey of GRE could not obtain the original Battle Mountain report. This section will outline the results described in the 2018 Technical Report.

Bottle roll tests were conducted on material that was crushed to 2 millimeter (mm) (10 Mesh) sized particles. Eight composites were created from four drill holes, with material organized by rock type (either oxide or sulfide). The results of these tests are given below in Table 13-5.

Table 13-5: 1991 Battle Mountain Bottle Roll Test Results

Type	Hole No.	Interval (m)	Assay Head		Calculated Head		Recovery	
			Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
Oxide	91-32	4.57 – 9.14	1.06		1.27	1.03	96.5	24.1
Oxide	91-33	6.10 – 12.19	1.51		1.65	3.77	67.1	18.8
Oxide	91-37	33.53 – 39.62	0.891		0.926	6.17	88.8	28.9
Oxide	91-38	21.34 – 30.48	1.17		1.30		94.8	
Average			1.16		1.29	3.66	86.8	23.9
Sulfide	91-32	50.29 – 56.39	0.549		0.583	1.03	9.4	17.5
Sulfide	91-34	30.48 – 32.00; 41.15 – 44.20	1.65		1.85		19.8	
Sulfide	91-37	74.68 – 79.25	1.51		1.58	12.7	1.4	18.2
Sulfide	91-38	70.10 – 76.20	0.994		0.994	7.20	2.1	12.8
Average			1.17		1.251	6.97	8.2	16.2

The results from these tests show a clear distinction between the oxide and sulfide material. The average gold extraction from the oxide material was 86.8%, compared to only 8.2% from the sulfide material for similar head grades. This is an indication that the sulfides are refractory to conventional heap leaching. The average silver extractions were similar at 23.9% and 16.2% for the oxides and sulfides, respectively.

13.2.2 1995 American Assay Test Work

American Assay Laboratories conducted a series of bottle roll tests on 10 composites in 1995 (American Assay Laboratories, 1995). To create these composites, 116 drill core samples were used, and drill hole composites were created from specific intervals. The details of the composites are found in Table 13-6.

Table 13-6: Composite Data Used for Bottle Roll Test Work (American Assay Laboratories, 1995)

Composite No.	Hole No.	Intervals Used (m below ground)	Average Gold Assay Head (g/t)	Rock Type
1	LV94-2	47.24 - 74.68	0.891	Sulfide
2	LV94-3	24.38 - 45.72	1.20	Oxide

Composite No.	Hole No.	Intervals Used (m below ground)	Average Gold Assay Head (g/t)	Rock Type
3	LV94-3	51.82 - 57.91; 62.48 - 67.06	0.891	Oxide (later changed to sulfide)
4	LV94-4	160.02 - 187.45	1.13	Sulfide
5	LV94-9	79.25 - 97.54	0.857	Sulfide
6	LV94-10	18.29 - 39.62	0.771	Oxide
7	LV94-12	67.06 - 76.20	0.549	Sulfide
8	LV94-14	143.26 - 152.40	0.771	Sulfide
9	LV94-16	0 - 13.72	0.429	Oxide
10	LV94-16	121.92 - 129.54	0.874	Sulfide

Standard bottle roll tests were performed on the composites, with solution samples taken at 2, 6, 24, 48, and 72 hours of leaching time, as well as at the end of the leach cycle at 96 hours. Tests were conducted at 40% solids by mass, and at the start of the tests cyanide was added at a dosage of 1 kilogram per tonne (kg/t). The results of these tests are given below in Table 13-7.

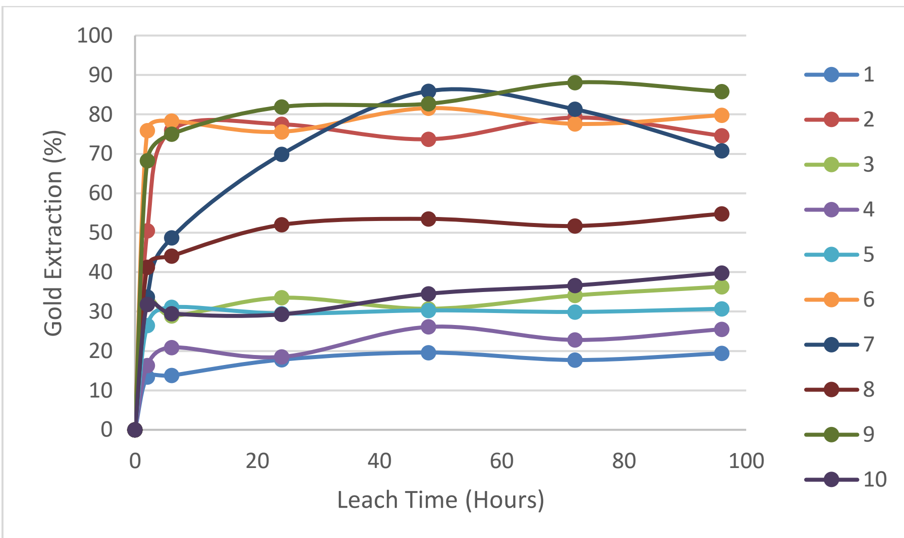
Table 13-7: 1995 Bottle Roll Test Results on Composites (American Assay Laboratories)

Sample ID	1	2	3	4	5	6	7	8	9	10
Rock Type	S	O		S	S	O	S	S	O	S
Leach Time (Hours)	Gold Extraction (%)									
0	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
2	13.4%	50.5%	32.9%	16.3%	26.5%	75.9%	33.7%	41.2%	68.3%	31.8%
6	13.8%	76%	28.9%	20.9%	31.1%	78.3%	48.7%	44.1%	75%	29.5%
24	17.8%	77.5%	33.5%	18.5%	29.5%	75.6%	69.9%	52%	81.9%	29.3%
48	19.6%	73.7%	30.7%	26.1%	30.3%	81.6%	85.9%	53.5%	82.7%	34.5%
72	17.7%	79.2%	34.1%	22.8%	29.9%	77.6%	81.3%	51.7%	88.1%	36.6%
96	19.4%	74.6%	36.3%	25.5%	30.7%	79.8%	70.8%	54.8%	85.8%	39.8%
Total Extracted Gold (g/t)	0.240	1.30	0.446	0.377	0.377	0.960	0.926	0.514	0.823	0.549
Tails Assay, Au (g/t)	0.994	0.446	0.754	1.10	0.823	0.240	0.377	0.411	0.137	0.823
Calculated Head, Au (g/t)	1.23	1.75	1.20	1.47	1.20	1.20	1.30	0.926	0.960	1.37
Assayed Head, Au (g/t)	0.891	1.200	0.891	1.13	0.857	0.771	0.549	0.771	0.429	0.874
Calculated Extraction	19.4%	74.5%	37.1%	25.6%	31.4%	80.0%	71.1%	55.6%	85.7%	40.0%
Cyanide Consumed (kg/t)	0.55	1.95	0.73	0.29	0.81	0.38	0.83	1.02	1.78	0.47
Lime Added (kg/t)	1.36	3.22	2.10	1.07	2.09	1.31	0.68	1.93	1.86	0.56
Final pH	10.7	10.7	11.0	11.1	10.9	11.2	11.2	11.0	9.8	11.1

The results of the tests were mixed. Gold extraction ranged between 19.4% for Composite 1 to 85.8% for Composite 9. The mean extraction from the tests was 51.8%, with an average of 0.88 kg/t cyanide consumed and 1.62 kg/t lime added. All the composites that showed poor results (1, 3, 4, 5, 8, and 10) were of sulfide-type material, except for sample 3, which was initially classified as oxide material. Later statistical analysis of all drill hole samples taken for the Long Valley project classified composite 3 as sulfide.

Composites that had more oxidized or mixed composition were more amenable to cyanidation, producing more desirable results. The average gold extraction for sulfide bearing composites was 40.2%, compared to 69.1% for oxide material. Generally, tests with lower final gold extraction values recorded lower cyanide and lime consumption, although it is not clear why this was the case, except to account for cyanide consumed to leach gold and silver. Given below in Figure 13-1 are the leach rate profiles of the composites.

Figure 13-1: Leach-Rate Profiles on Poorly Performing Samples Tested by American Assay Laboratories (1995)



The leach-rate profiles show above all indicate quick leaching, with most of the gold extracted within 24 hours. It is not clear why the final gold extraction values of composites 2, 7, and 9 show a decrease from their values at 48- and 72-hours, however it may be likely due to the discrepancies between the head fire assays and the calculated head assays used to determine percent gold extraction over time.

13.2.3 1995 Hazen Research Test Work

In 1995 Hazen Research conducted several bottle roll tests on drill core samples from the Long Valley deposit. The drill core samples were half of two-inch diameter core which were described as “from the Southeast Zone and the Hilton Creek (Fault) Zone”. The Southeast Zone was described in the test work report as less oxidized. Head assays were performed on the samples shown in Table 13-8.

Table 13-8: Head Assay Data 1995 Hazen Bottle Roll Test Work

Sample ID	Interval	Description	Rock Type	Au (g/t)	Ag (g/t)	C (%)	CO_2 (%)	$C_{(org)}$ (%)
95 C1	47-106; 167-242	Southeast Zone	Sulfide (?)	0.857	8.57	0.150	0.020	0.140

95 C2	105-215	Hilton Creek		0.840	6.00	0.0950	0.020	0.095
Mean				0.849	7.29	0.123	0.020	0.118

Carbon assays of the material showed low carbon content with minimal "pregnant solution-robbing" potential.

Standard bottle roll tests were conducted on four size fractions of the two samples. The samples were crushed and/or ground to 25.4 mm, 12.7 mm, 6.35 mm, and 0.037 mm size fractions (assumed 100% material passing) and leached for 72 to 120 hours. The results these tests are given in Table 13-9.

Table 13-9: 1995 Bottle Roll Test Results (Hazen Research, 1995)

	95 C1				**95 C2**			
	1"	**1/2"**	**1/4"**	**0.0015"**	**1"**	**1/2"**	**1/4"**	**0.0015"**
Particle Size (mm)	25.4	12.7	6.35	0.037	25.4	12.7	6.35	0.037
Calculated Head Assay								
Au (g/t)	0.891	0.891	0.891	0.857	0.960	0.960	0.960	1.89
Ag (g/t)	8.57	11.31	8.57	8.91	7.54	5.14	5.14	9.60
% Gold Extraction								
24-hour	10.4%	12.4%	10.3%	17.2%	45.2%	45.4%	44.8%	19.9%
48-hour	10.4%	12.5%	10.4%	22.4%	47.0%	47.3%	48.7%	74.6%
72-hour	10.5%	12.6%	10.5%	23.0%	48.5%	45.9%	47.6%	81.9%
96-hour	12.2%	12.5%	10.5%		53.2%	46.0%	48.2%	
120-hour	12.4%	16.2%	12.3%		53.5%	49.5%	54.0%	
% Silver Extraction								
24-hour	22.2%	19.4%	29.6%	34.6%	23.1%	48.1%	49.0%	41.3%
48-hour	23.9%	22.3%	30.9%	42.7%	25.8%	53.0%	56.7%	42.9%
72-hour	24.6%	24.5%	34.4%	45.4%	28.1%	54.5%	58.0%	46.7%
96-hour	27.5%	25.2%	36.0%		29.6%	55.8%	58.4%	
120-hour	30.6%	53.1%	39.0%		31.0%	62.1%	66.2%	
Tailings Residue								
Au (g/t)	0.789	0.754	0.789	0.651	0.446	0.480	0.446	0.343
Ag (g/t)	6.17	5.49	5.14	4.80	5.14	2.06	1.71	5.14
Reagent Consumption								
CN Consumption (kg/t)	1.20	1.64	3.19	2.76	2.62	2.57	2.19	2.74
Lime Addition (kg/t)	2.15	2.30	2.66	6.60	3.89	4.80	5.40	10.40

Sample 95 C1 showed poor gold extraction, with the finest material achieving only 23% gold extraction. Sample 95 C2 exhibited better performance with approximately 82% of the gold recovered from the finest material. Under the same timeframe, both samples showed a dramatic increase in gold extraction with finely ground material. Gold extraction more than doubled for sample 95C1 from 10.5% to 23%, and

almost doubled from 47% to approximately 82% for sample 95 C2. The results indicate that sample 95 C1 has generally poor gold recovery across all size fractions tested and that sample 95 C2 has a significant size sensitivity with the best recoveries being achieved at the finest size faction.

Silver recoveries from both samples ranged between 31% and 53% for sample 95 C1, and 31% and 66.2% for sample 95 C2, respectively. Cyanide consumptions ranged between 1.2 kilograms per tonne (kg/t) to 3.19 kg/t and were generally uniform across sample types and particle size fractions. Lime consumption ranged between 2.15 kg/t to 6.6 kg/t for sample 95 C1, and between 3.89 kg/t to 10.4 kg/t. Both samples showed an increase in lime consumption with finer material, indicating that some sulfide material may be present.

A carbon-in-leach (CIL) test was conducted on 95 C1 -400 M material (0.037 mm) to determine whether pregnant solution-robbing was responsible for the initial low recoveries. However, results were only marginally better, indicating that pregnant solution-robbing was not the root cause of the low gold extraction (Table 13-10).

Table 13-10: 1995 Carbon-In-Leach Test Results on Sample 95 C1 (Hazen Research, 1995)

		Calculated Head Assay		72-Hour Extraction (%)		Tailings Residue		Reagent Consumption	
95 C1	**Particle Size (mm)**	**Au (g/t)**	**Ag (g/t)**	**Gold**	**Silver**	**Au (g/t)**	**Ag (g/t)**	**CN Consumption (kg/t)**	**Lime Addition (kg/t)**
400 M CIL	0.037	0.926	8.571	29.0%	39.2%	0.651	5.14	2.94	3.55

Four further tests were conducted – two hot cyanide shake tests on the drill core composites mentioned previously, and two tests on a bulk sample that would be used for column leach test work in 1996 (see Section 13.3.2). The hot cyanide shake tests used pulverized material below 150 mesh (0.105 mm). Leaching was conducted for 24 hours (Table 13-11).

Table 13-11: 1995 Hot Cyanide Shake Test Results (Hazen Research, 1995)

		Calculated Head Assay	Gold Extraction
	Particle Size (mm)	**Au (g/t)**	**24-hour**
95 C1	0.015	0.891	19.2
95 C2	0.015	0.960	57.1

The results of the hot cyanide shake tests are mixed: for sample 95 C1, only 19.2% of the gold was extracted from the sample, which is consistent with the previous bottle roll tests. Sample 95 C2 achieved over 57% gold extraction 24 hours which is an improvement over the previous bottle roll results.

The tests on the bulk sample included a standard bottle roll test on (assumed 100% passing) 51 mm (2-inches) material, and a hot cyanide shake test. The results of these tests are given below in Table 13-12.

Table 13-12: Bottle Roll and Hot Cyanide Shake Test Results on Bulk Sample Used for Column Test Work (Hazen Research, 1995)

		Calc. Head Assay	Gold Extraction (by Hours Leached)					Tailings Residue		Reagent Consumption	
Test Type	Particle Size (mm)	Au (g/t)	24	48	72	96	120	Au (g/t)	Ag (g/t)	CN Cons. (kg/t)	Lime Addition (kg/t)
Bottle Roll	50.8	0.857	78.6%	82.4%	85.5%	85.6%	79.1%	0.171	4.11	0.28	1.26
			Silver Extraction (by Hours Leached)								
			24	48	72	96	120				
			8.5%	9.9%	11.3%	11.7%	9.5%				
Hot Cyanide Shake Leach	0.015	0.857	68.0%								

The bulk sample exhibited a much more favorable response to both bottle roll testing as well as hot cyanide shake leaching, with 78.6% and 68% gold extraction after 24 hours, respectively. The standard bottle roll test shows a drop in gold extraction (from 85.6% at the 96-hour mark to 79.1% at 120 hours) and the Hazen report suggests that this is likely due to incomplete or insufficient washing of the tailings residue. It is likely that the final gold extraction from that bottle roll test is near 86%.

The difference between the behavior of the bulk sample used for column leaching and the two drill core samples tested previously can be explained by the composition of the material. The sulfidic mineralized material types at Long Valley appear to be more refractory to cyanide gold leaching.

13.2.4 1996 McClelland Laboratories Test Work

In 1996, McClelland Laboratories received 3,106 intervals from 47 drill core holes as well as other miscellaneous samples from the Long Valley deposit (McClelland Laboratories Inc., 1996). The samples originated from two areas: the Hilton Creek Fault Zone (HCFZ), and the Southeast Zone (SEZ). From those samples, 17 composites were created, organized by type of mineralized material – either oxide or sulfide, and siliceous or argillic. The 17 composites included three non-weighted composites from the HCFZ, and one non-weighted composite from the SEZ. The other 13 composites were prepared using the weight of the lightest interval as a basis, with 11 composites from the HCFZ and 2 from the SEZ. Assay results of the composites as well as the calculated head values are given below in Table 13-13.

Table 13-13: Sample Assay Data on Composites Tested by McClelland Laboratories (1996)

Composite ID	Mineral-ized Material Type	Rock Type	Zone	Average Fire Assay Au (g/t)	Calculated from Bottle Roll Test Au (g/t)	Average Fire Assay Ag (g/t)	Calculated from Bottle Roll Test Ag (g/t)
95-19	Oxide	Siliceous	HCF	0.994	1.13	19.7	19.2
95-70	Oxide	Siliceous	HCF	0.480	0.651	3.31	1.71
95-75	Oxide	Siliceous	HCF	1.22	1.34	12.1	11.0
95-30	Oxide	Argillic	HCF	0.994	1.02	6.86	4.11
95-52	Oxide	Argillic	HCF	1.23	1.783	7.77	5.14
95-63	Oxide	Argillic	HCF	0.709	0.926	3.66	1.71
95-62	Mixed	Siliceous	HCF	0.537	0.549	10.9	10.6

Composite ID	Mineral-ized Material Type	Rock Type	Zone	Average Fire Assay Au (g/t)	Calculated from Bottle Roll Test Au (g/t)	Average Fire Assay Ag (g/t)	Calculated from Bottle Roll Test Ag (g/t)
95-68	Mixed	Siliceous	HCF	0.617	0.651	3.20	4.11
95-36	Mixed	Argillic	HCF	0.354	0.446	5.37	4.46
95-67	Mixed	Argillic	HCF	1.59	1.71	3.43	2.40
95-87	Mixed	Argillic	HCF	0.149	0.206	4.34	2.06
95-35	Sulfide	Siliceous	HCF	0.971	0.754	13.9	10.6
95-67	Sulfide	Argillic	HCF	0.994	0.926	9.94	8.91
95-84	Sulfide	Argillic	HCF	0.777	0.754	2.63	2.40
95-94	Oxide	ND*	SE	0.503	0.720	7.77	3.77
95-94	Mixed	ND*	SE	0.434	0.514	6.17	3.77
95-93	Sulfide	ND*	SE	0.480	0.446	6.74	5.49
		Mean		0.767	0.855	7.52	5.97

*ND – not defined

Bottle roll tests were conducted on the composites. No crushing or grinding was required, as the nominal top size of the material was already approximately 2 mm (10 mesh). The bottle rolls were conducted at a pH of approximately 11, adjusted by addition of lime, and at a pulp density of 40% solids by weight. Cyanide was added before the start of the leach cycle at 1.0 kg/t, and leaching continued for 96 hours, with pregnant leach solution (PLS) samples taken at 2, 6, 24, 48, and 72 hours. The results of these tests have been organized by mineralized material type and zone in Table 13-14, Table 13-15, Table 13-16, and Table 13-17.

Table 13-14: Hill Creek Fault Zone Oxide Ores Composite Bottle Roll Tests Results (McClelland Laboratories, 1996)

Sample ID	95-19	95-70	95-75	95-30	95-52	95-63	Total Mean	Siliceous Comp.	Argillic Comp.
Rock Type	**Siliceous**	**Siliceous**	**Siliceous**	**Argillic**	**Argillic**	**Argillic**			
Leach Time (Hours)									
0	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
2	45.1%	62.2%	60.6%	84.6%	60.6%	69.7%	63.8%	56.0%	71.6%
6	47.3%	65.8%	61.9%	86.1%	61.1%	71.2%	65.6%	58.3%	72.8%
24	49.6%	67.3%	63.3%	86.7%	64.0%	71.2%	67.0%	60.1%	74.0%
48	51.9%	68.4%	63.5%	86.7%	64.5%	72.7%	68.0%	61.3%	74.6%
72	54.2%	68.4%	63.7%	86.7%	65.0%	74.1%	68.7%	62.1%	75.3%
96	54.5%	68.4%	64.1%	86.7%	65.4%	74.1%	68.9%	62.3%	75.4%
Total Extracted Gold (g/t)	0.617	0.446	0.857	0.891	1.17	0.686	0.777	0.640	0.914
Tails Assay, Au (g/t)	0.514	0.206	0.480	0.137	0.617	0.240	0.366	0.400	0.331
Calculated Head, Au (g/t)	1.13	0.651	1.34	1.03	1.78	0.926	1.14	1.04	1.27
Assayed Head, Au (g/t)	0.994	0.480	1.22	0.994	1.23	0.709	0.939	0.899	0.979

Sample ID	95-19	95-70	95-75	95-30	95-52	95-63	Total Mean	Siliceous Comp.	Argillic Comp.
Rock Type	Siliceous	Siliceous	Siliceous	Argillic	Argillic	Argillic			
Calculated Gold Extraction (%)	54.5%	68.4%	64.1%	86.7%	65.4%	74.1%	68.0%	61.5%	73.4%
Cyanide Consumed (kg/t)	0.2	0.075	0.03	0.09	0.02	0.005	0.07	0.10	0.04
Lime Added (kg/t)	2.95	3.55	1.95	4.45	4.4	2.7	3.33	2.82	3.85
Final pH	10.7	11.5	11.1	11.2	11	11.3	11.1	11.1	11.2
Natural pH	5.2	4.6	6	4.3	4.5	4.1	4.8	5.3	4.3
Ag Recovery (%)	41.1%	7.5%	21.9%	25.0%	26.7%	6.5%	21.5%	23.5%	19.4%
Total Extracted Silver (g/t)	7.89	0.00	2.40	1.03	1.37	0.00	2.11	3.43	0.80
Tails Assay, Ag (g/t)	11.3	1.71	8.57	3.09	3.77	1.71	5.029	7.20	2.86
Calculated Head, Ag (g/t)	19.2	1.71	10.97	4.11	5.14	1.71	7.14	10.6	3.66
Assayed Head, Ag (g/t)	19.7	3.31	12.1	6.86	7.77	3.66	8.90	11.7	6.10
Calculated Silver Extraction (%)	41.1%	0.0%	21.9%	25.0%	26.7%	0.0%	29.6%	32.3%	21.9%

The oxide samples from the HCFZ showed mixed results, gold extractions ranged between 54.5% for sample 95-19 to 86.7% for sample 95-30. Silver extractions were generally low. A clear delineation can be seen between siliceous samples and those with more argillic material. The average level of gold extraction for sample with more siliceous material was 68.9% compared to 75.4% for those with more argillic material. Cyanide and lime consumption levels were low for all HCFZ oxide tests. It is interesting to note that the test on sample 95-19 extracted the lowest percentage of gold but the highest percentage of silver, and it is unclear why this is the case. Figure 13-2 shows the leach curves for the tests.

Figure 13-2: Leach-Rate Profiles of the HCFZ Oxide Mineralized Material Composites Tested by McClelland Laboratories (1996)

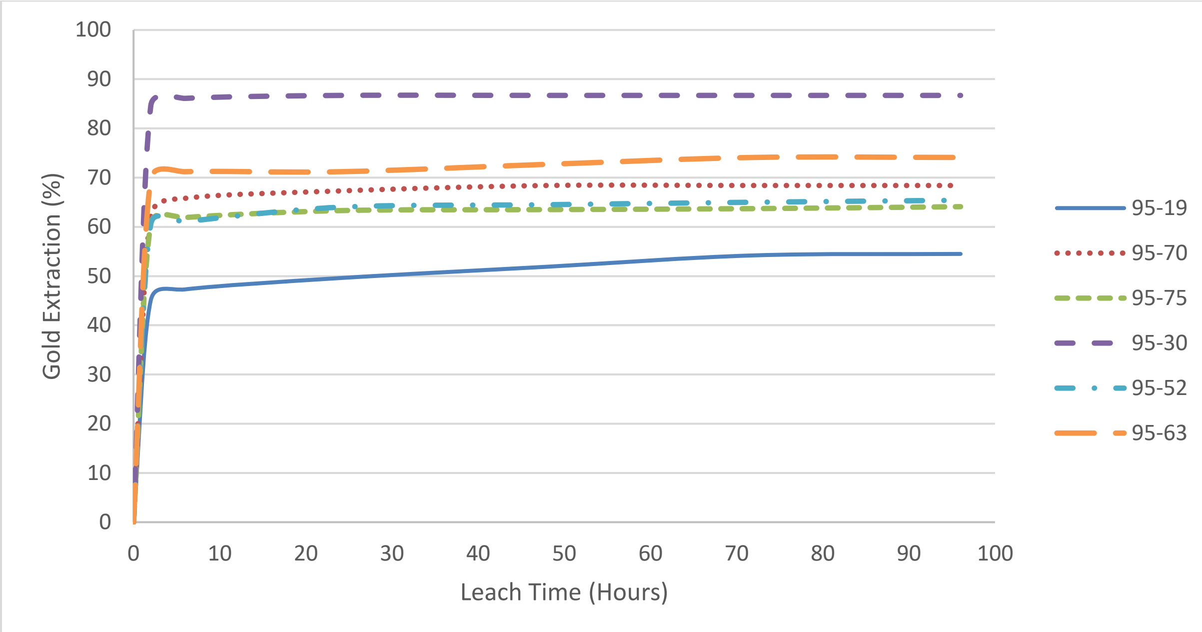


Table 13-15: Hill Creek Fault Zone Mixed Mineralized Material Composite Bottle Roll Test Results (McClelland Laboratories, 1996)

Sample ID	95-62	95-68	95-36	95-67	95-87	Total Mean	Siliceous Comp.	Argillic Comp.
Rock Type	Siliceous	Siliceous	Argillic	Argillic	Argillic			
Leach Time (Hours)								
0	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
2	43.8%	43.8%	60.6%	17.5%	43.8%	41.9%	43.8%	40.6%
6	50.0%	47.0%	65.3%	19.6%	50.0%	46.4%	48.5%	45.0%
24	56.7%	48.0%	66.7%	21.8%	50.0%	48.6%	52.4%	46.2%
48	57.9%	51.4%	68.1%	22.3%	50.0%	49.9%	54.7%	46.8%
72	61.9%	52.4%	69.2%	22.8%	50.0%	51.3%	57.2%	47.3%
96	62.5%	52.6%	69.2%	24.0%	50.0%	51.7%	57.6%	47.7%
Total Extracted Gold (g/t)	0.343	0.343	0.309	0.411	0.103	0.302	0.343	0.274
Tails Assay, Au (g/t)	0.206	0.309	0.137	1.30	0.103	0.411	0.257	0.514
Calculated Head, Au (g/t)	0.549	0.651	0.446	1.71	0.206	0.713	0.600	0.789
Assayed Head, Au (g/t)	0.537	0.617	0.354	1.589	0.149	0.649	0.577	0.697
Calculated Gold Extraction (%)	62.5%	52.6%	69.2%	24.0%	50.0%	42.3%	57.1%	34.8%
Cyanide Consumed (kg/t)	0.08	0.08	0.34	0.825	0.215	0.31	0.08	0.46
Lime Added (kg/t)	2.6	2	2.75	10.5	8.05	5.18	2.30	7.10
Final pH	11.2	11	11	11.5	10.8	11.1	11.1	11.1
Natural pH	6	5.6	4.5	3.6	3.2	4.6	5.8	3.8
Ag Recovery (%)	22.6	25	15.4	14.3	16.7	18.8	23.8	15.5
Total Extracted Silver (g/t)	2.40	1.03	0.686	0.343	0.343	0.960	1.71	0.457
Tails Assay, Ag (g/t)	8.23	3.09	3.77	2.06	1.71	3.77	5.66	2.51
Calculated Head, Ag (g/t)	10.6	4.11	4.46	2.40	2.06	4.73	7.37	2.97

Sample ID	95-62	95-68	95-36	95-67	95-87	Total Mean	Siliceous Comp.	Argillic Comp.
Rock Type	Siliceous	Siliceous	Argillic	Argillic	Argillic			
Assayed Head, Ag (g/t)	10.9	3.20	5.37	3.43	4.34	5.44	7.03	4.38
Calculated Silver Extraction (%)	22.6%	25.0%	15.4%	14.3%	16.7%	20.3%	23.3%	15.4%

The HCFZ "mixed" mineralized material samples returned a wide range of results. Gold extraction ranged from 24% to 69.2%. Cyanide and lime consumptions were low, with the highest consumptions seen for sample 95-67 (from which only 24% of the gold was recovered). Silver recoveries ranged between 14.3% and 25%. Figure 13-3 shows the leach curves for the tests.

Figure 13-3: Leach-Rate Profiles of the HCFZ Mixed Mineralized Material Composites Tested by McClelland Laboratories (1996)

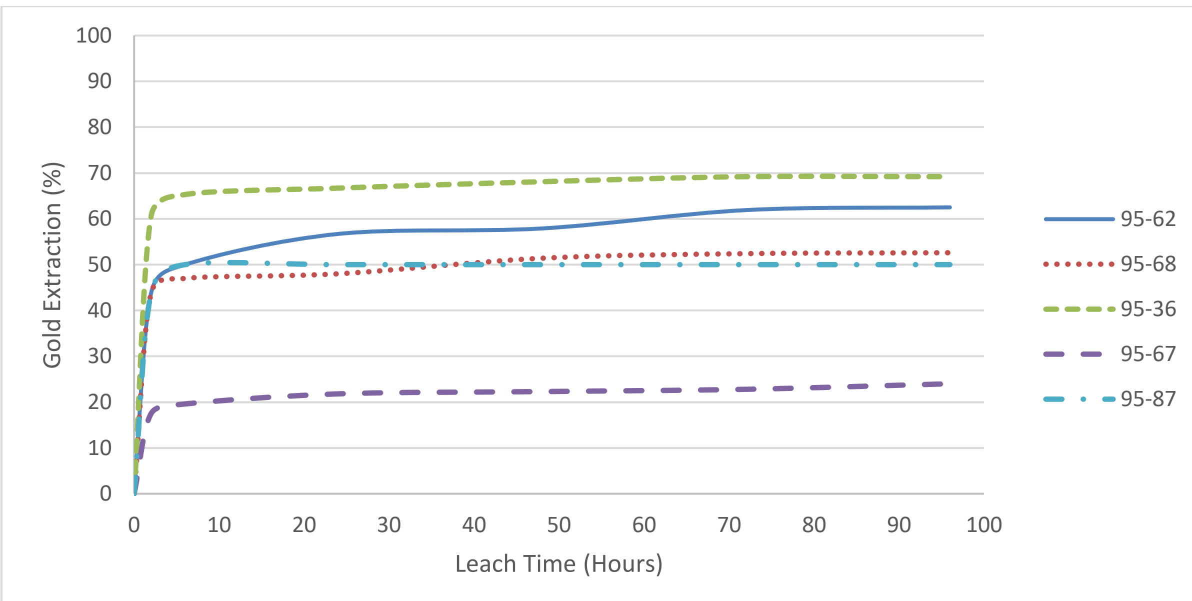


Table 13-16: Hill Creek Fault Zone Sulfide Mineralized Material Composite Bottle Roll Test Results (McClelland Laboratories, 1996)

Sample ID	95-35	95-67 (b)	95-84	Total Mean	Siliceous Comp.	Argillic Comp.
Rock Type	Siliceous	Argillic	Argillic			
Leach Time (Hours)						
0	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
2	0.0%	13.0%	17.9%	10.3%	0.0%	15.5%
6	0.0%	14.9%	20.3%	11.7%	0.0%	17.6%
24	0.0%	15.2%	20.7%	12.0%	0.0%	18.0%
48	0.0%	15.5%	21.1%	12.2%	0.0%	18.3%
72	0.0%	17.5%	21.6%	13.0%	0.0%	19.6%
96	0.0%	18.5%	22.7%	13.7%	0.0%	20.6%
Total Extracted Gold (g/t)	0.000	0.171	0.171	0.114	0.000	0.171
Tails Assay, Au (g/t)	0.754	0.754	0.583	0.697	0.754	0.669
Calculated Head, Au (g/t)	0.754	0.926	0.754	0.811	0.754	0.840
Assayed Head, Au (g/t)	0.971	0.994	0.777	0.914	0.971	0.886

Sample ID	95-35	95-67 (b)	95-84	Total Mean	Siliceous Comp.	Argillic Comp.
Rock Type	Siliceous	Argillic	Argillic			
Calculated Gold Extraction (%)	0.0%	18.5%	22.7%	14.1%	0.0%	20.4%
Cyanide Consumed (kg/t)	0.315	0.52	0.45	0.43	0.32	0.49
Lime Added (kg/t)	2.6	6	3.7	4.10	2.60	4.85
Final pH	11	10.8	11	10.9	11.0	10.9
Natural pH	4.7	3.9	3.6	4.1	4.7	3.8
Ag Recovery (%)	32.3	30.8	28.6	30.6	32.3	29.7
Total Extracted Silver (g/t)	3.43	2.74	0.686	2.29	3.43	1.71
Tails Assay, Ag (g/t)	7.20	6.17	1.71	5.03	7.20	3.94
Calculated Head, Ag (g/t)	10.6	8.91	2.40	7.31	10.6	5.66
Assayed Head, Ag (g/t)	13.94	9.94	2.63	8.84	13.9	6.29
Calculated Silver Extraction (%)	32.3%	30.8%	28.6%	31.3%	32.3%	30.3%

The HCFZ Sulfide samples returned low values for gold and silver extraction. Gold extraction ranged from 0% and 22.7%, with no gold recovered from the siliceous composite. This mineralized material type may exhibit a combination of siliceous and sulfide refractoriness. Silver recoveries were moderately better but still low, ranging between 28.6% to 32.3%. Cyanide consumption levels were between 0.32 and 0.52 kg/t. Lime consumption data ranged between 2.6 and 6 kg/t, which is high and may be a function of increased sulfide concentrations. Figure 13-4 shows the leach curves for these tests.

Figure 13-4: Leach-Rate Profiles of the HCFZ Sulfide Mineralized Material Composites Tested by McClelland Laboratories (1996)

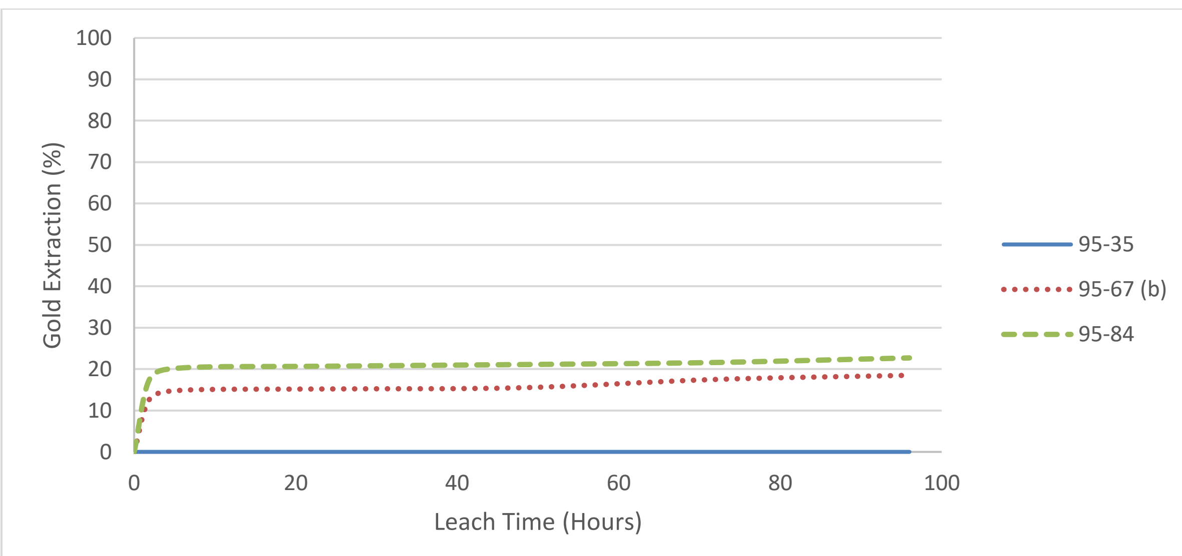


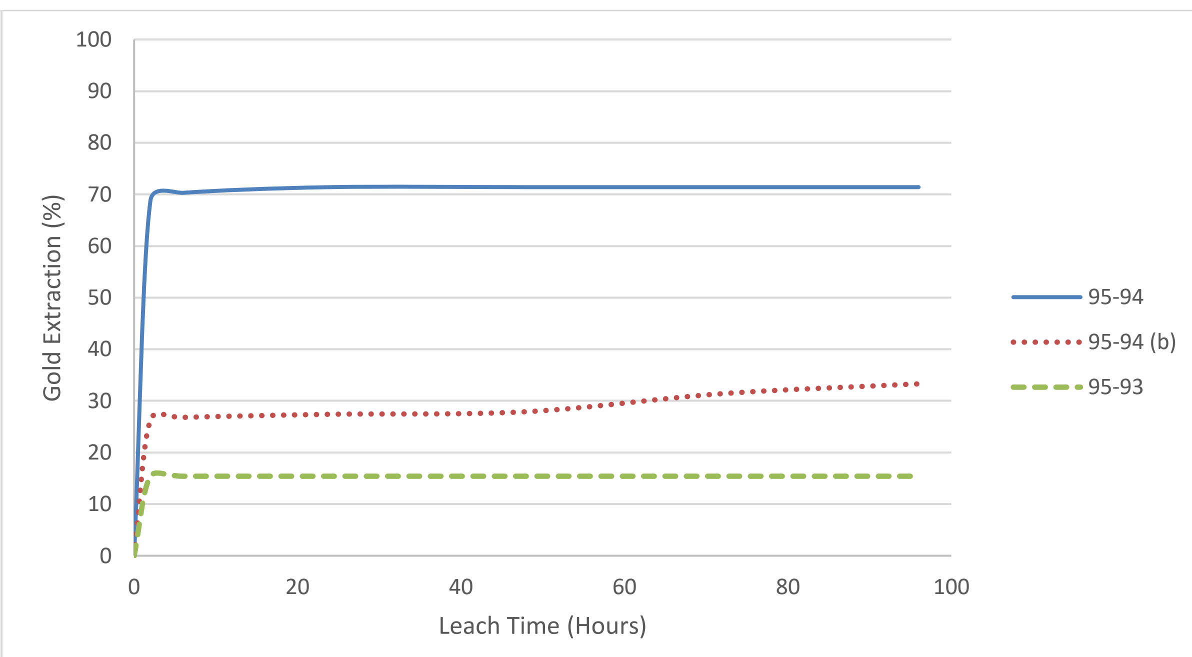
Table 13-17: South East Zone Composite Bottle Roll Test Results (McClelland Laboratories, 1996)

Sample ID	95-94	95-94 (b)	95-93	Average
Rock Type	Oxide	Mixed	Sulfide	
Leach Time (Hours)				
0	0.0%	0.0%	0.0%	0.0%
2	68.8%	26.3%	15.4%	36.8%
6	70.3%	26.8%	15.4%	37.5%

Sample ID	95-94	95-94 (b)	95-93	
Rock Type	Oxide	Mixed	Sulfide	Average
24	71.4%	27.4%	15.4%	38.1%
48	71.4%	27.9%	15.4%	38.2%
72	71.4%	31.4%	15.4%	39.4%
96	71.4%	33.3%	15.4%	40.0%
Total Extracted Gold (g/t)	0.514	0.171	0.069	0.251
Tails Assay, Au (g/t)	0.206	0.343	0.377	0.309
Calculated Head, Au (g/t)	0.720	0.514	0.446	0.560
Assayed Head, Au (g/t)	0.503	0.434	0.480	0.472
Calculated Gold Extraction (%)	71.4%	33.3%	15.4%	44.9%
Cyanide Consumed (kg/t)	0.075	0.23	0.15	0.15
Lime Added (kg/t)	1.6	2.4	3	2.33
Final pH	11	11	11	11.0
Natural pH	7.9	6.8	4.5	6.4
Ag Recovery (%)	9.1	18.2	25	17.4
Total Extracted Silver (g/t)	0.343	0.686	1.37	0.800
Tails Assay, Ag (g/t)	3.43	3.09	4.11	3.54
Calculated Head, Ag (g/t)	3.77	3.77	5.49	4.34
Assayed Head, Ag (g/t)	7.77	6.17	6.74	6.90
Calculated Silver Extraction (%)	9.1%	18.2%	25.0%	18.4%

The South East Zone (SEZ) samples returned mixed results from bottle roll testing: gold extraction from Oxide and Mixed samples were 71.4% and 33.3%, respectively. A gold extraction of 15.4% was achieved from the sulfide composite. Cyanide consumptions were uniformly low and lime consumptions increased from 1.6 kg/t for the Oxide composite to 3 kg/t for the sulfide composite. Low levels of silver extraction were exhibited for all samples from the Southeast area. Figure 13-5 shows the leach curves for these tests.

Figure 13-5: Leach-Rate Profiles of the SEZ Composites Tested by McClelland Laboratories (1996)



Overall, the bottle roll tests performed by McClelland Laboratories returned a wide range of results. Gold extraction ranged from 0% to 86.7%, with an average of 49% from all samples. However, the rate of gold recovery was generally rapid. Gold extraction typically reached near completion within 24-hours, and in several cases in 2 – 6 hours. The mean results of all the bottle roll cyanidation tests described above have been broken down by mineralized material- and rock-type in Table 13-18 below.

Table 13-18: Average Bottle Roll Test Results of Composites Tested by McClelland Laboratories by Mineralized Material-/Rock-Type

Mineralized Material-/Rock-Type	ALL	Siliceous	Argillic	Oxide	Mixed	Sulfide
Overall Average Gold Extraction (%)	49.0%	50.4%	51.3%	69.2%	48.6%	14.2%
Overall Average Silver Extraction (%)	14.3%	25.1%	20.5%	19.7%	18.7%	29.2%
HCFZ Gold Extraction (%)	50.9%	50.4%	51.3%	68.9%	51.7%	13.7%
HCFZ Silver Extraction (%)	22.5%	25.1%	20.5%	21.5%	18.8%	30.6%
SEZ Gold Extraction (%)	40.0%			71.4%	33.3%	15.4%
SEZ Silver Extraction (%)	17.4%			9.1%	18.2%	25.0%

The results of all bottle roll tests performed by McClelland Laboratories are given below in Figure 13-6 and Figure 13-7.

Figure 13-6: Gold Extraction Results of 1996 Bottle Roll Tests Delineated by Mineralized Material Type (McClelland Laboratories)

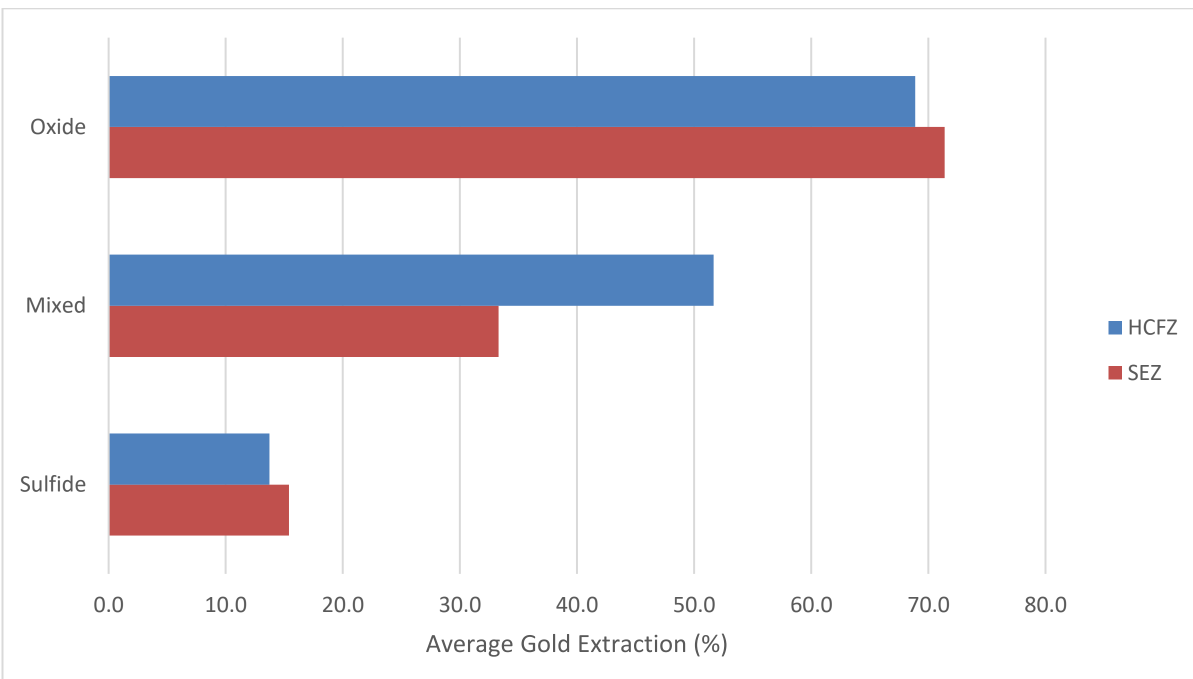
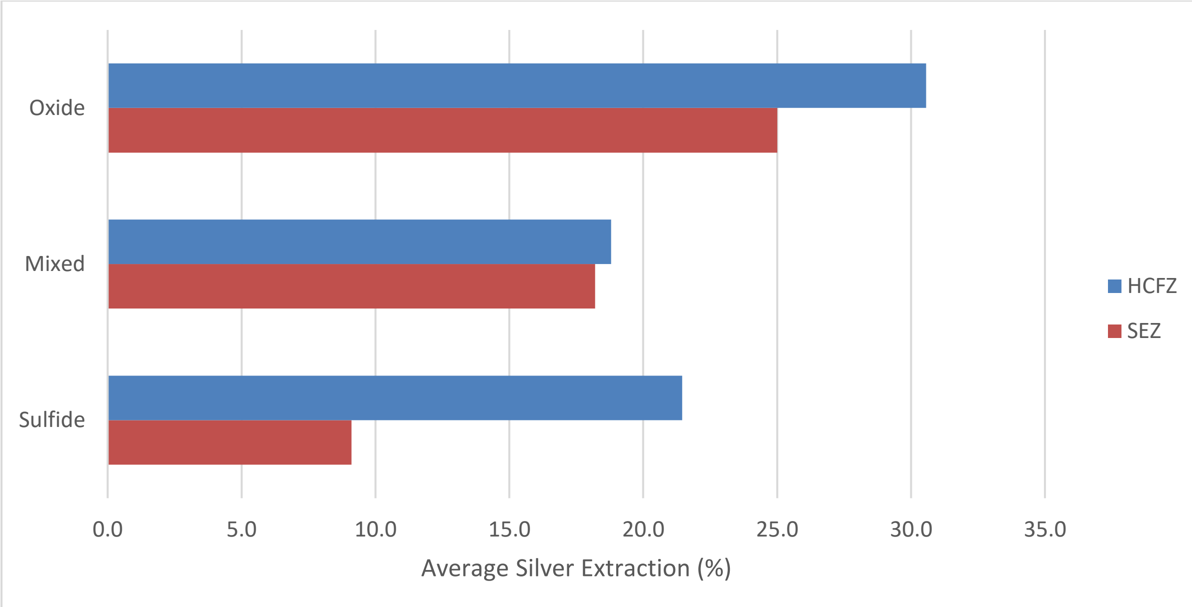


Figure 13-7: Silver Extraction Results of 1996 Bottle Roll Tests Delineated by Mineralized Material Type (McClelland Laboratories)



13.3 Column Leach Test Work

Column Leach testing of the Long Valley material was performed in two major campaigns: the first, performed by KCA in 1989 (Kappes, Cassiday & Associates, 1989), and in 1996 by Hazen (Hazen Research, Inc., 1997).

13.3.1 1989 Column Tests by KCA

13.3.1.1 Agglomeration, Percolation, and Column Leach Tests

Six 55-gallon drums of material from the Long Valley deposit were tested by KCA in 1989. After crushing, the material's top size was 76.2 mm (3-inch). After a series of sample preparation steps, the material from each of the six drums was analyzed for gold and silver, and was separated into three size distributions: 76.2 mm (3-inch), 37.5 mm (1.5-inch), and 12.7 mm (½-inch) sized material (assumed 100% passing). The results of gold and silver assays performed on these samples are given below in Table 13-19.

Table 13-19: Assay Data on Samples Tested by KCA in 1989

KCA Sample #	Royal Gold Sample #	Average Fire Assay Value	
		Au (g/t)	Ag (g/t)
10785 A	T1-5	1.05	2.40
10785 B	T1-10	1.35	2.91
10785 C	T2-5	0.789	2.06
10785 D	T2-10	0.480	1.20
10785 E	T3	1.06	1.20
10785 F	85-5	0.720	4.80

Additionally, an aliquot of 12.7 mm material from drum 10785 A was screened at 28-Mesh and assayed to determine the deportment of gold in the resulting split. The results of this test are given below in Table 13-20.

Table 13-20: Results of Gold Deportment Study Performed on Sample 10785A (KCA, 1989)

Size Fraction	Size (mm)	Weight (kg)	Weight %	Average Fire Assay Value		Distribution	
				Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
28 Mesh	<= 0.595	4.32	86.57%	0.994	1.54	82.0%	70.7%
Undersize	> 0.595	0.67	13.43%	1.41	4.11	18.0%	29.3%
Total		4.99	100%	1.05	1.89	100%	100%

Around 13% of this crushed material were fines, but a noticeable difference in gold and silver assay values can be seen between the two size fractions. The finer material had more gold and silver, indicating that more gold was present in the more friable sulfur material.

After assaying, a single composite was created with material from all six drums. This was done by blending together the 76.2 mm (3-inch) material from all six samples, splitting, and sequentially crushing aliquots to create a medium (37.5 mm or 1.5-inch) size fraction (assumed 100% passing), and a fine (12.7 mm or ½-inch) size fraction.

Agglomeration and percolation tests were completed on samples taken from the finest fraction of composite. Cement and 0.25 gram per liter (g/L) sodium cyanide (NaCN) solution was added to material set in 76.2 mm (3-inch) diameter columns. The height of the mineralized material ranged from 18.5-inch to 24-inch. The same solution was then applied at a rate of 9.6 to 12 liters per hour per square meter (0.004 – 0.005 gallons per minute per square foot [gpm/ft^2]) for 72 hours. After the irrigation period was completed, columns were tapped/shaken until the material settled into a stable height. The difference between the height of the material at the start and at the completion of the test was measured. To complete the percolation test, the solution drain was closed, and NaCN solution was added to the column until the height of the solution was approximately 25 to 51 mm above the mineralized material. The solution drain was then opened, and enough solution was added to maintain this level as the liquid drained. The rate of addition of solution was then recorded. The details of the agglomeration and percolation tests are shown in Table 13-21.

Table 13-21: Results of Agglomeration and Percolation Tests Performed by KCA (1989)

KCA Test #	KCA Sample #	Mineral-ized Material Weight (kg)	Cement Added (kg/t)	Volume 0.25 g/L NaCN Solution Added (mL)	Mineralized Material Height (mm)		Percent		Percolation Rate (lph/m^2)
					Initial	Final	Mineral-ized Material Slump	Agglomeration Breakdown by Visual Inspection (%)	
10808 A	10807 C	2	2.5	398	577.9	565.2	2.2%	< 1	14605.7
10808 B	10807 C	2	5	383	584.2	577.9	1.1%	< 1	33685.2

KCA Test #	KCA Sample #	Mineral-ized Material Weight (kg)	Cement Added (kg/t)	Volume 0.25 g/L NaCN Solution Added (mL)	Mineralized Material Height (mm)		Percent		Percolation Rate (lph/m^2)
					Initial	Final	Mineral-ized Material Slump	Agglomeration Breakdown by Visual Inspection (%)	
10809 A	10807 C	2	7.5	408	609.6	603.3	1.0%	< 1	19474.2
10809 B	10807 C	2	10	394	596.9	584.2	2.1%	< 1	21579.6
11011	10807 C	2	0	0	469.9	393.7	16.2%	n/a	763.0

The results indicate that increasing cement addition improves percolation rate and reduces mineralized material slump. An addition rate of 2.5 kg/t appears to be a minimal application rate.

The column leach tests performed by KCA used 2.5 kg/t cement to agglomerate the composite material. Three columns were tested, with different sizes of material in each: a coarse fraction between 76.2 and 37.5 mm, a medium fraction between 37.5 mm and 12.7 mm, and a finely crushed fraction below 12.7 mm. Column diameters ranged from 8 inches to 11.5 inches, and the initial mineralized material heights ranged from approximately 10 feet to 11.8 feet. As before, 0.25 g/L sodium cyanide (NaCN) solution was added at a rate of 9.6 – 12 liters per hours per square meter (lph/m^2) (0.004 – 0.005 gpm/ft^2) and the pH of the solution was adjusted to between 10.0 and 10.3 through the addition of hydrated lime (Table 13-22).

Table 13-22: Column Test Conditions and Data (KCA, 2019)

KCA Test #	Mineral-ized Material Weight (kg)	Max Mineral-ized Material Size (mm)	Min Mineral-ized Material Size (mm)	Cement Added (kg/t)	Column Diameter (mm)	Volume 0.25 g/L NaCN Solution Added (mL)	Initial Mineral-ized Material Height (m)	Final Mineral-ized Material Height (m)	Percent Mineral-ized Material Slump
10818	189.52	76.2	25.4	2.5	292.1	31.1	3.1	2.8	12.1%
10820	122.8	37.5	12.7	2.5	254	20.64	3.2	2.8	13.5%
10822	91.36	12.7	0	2.5	203.2	22.2	3.6	3.4	4.9%

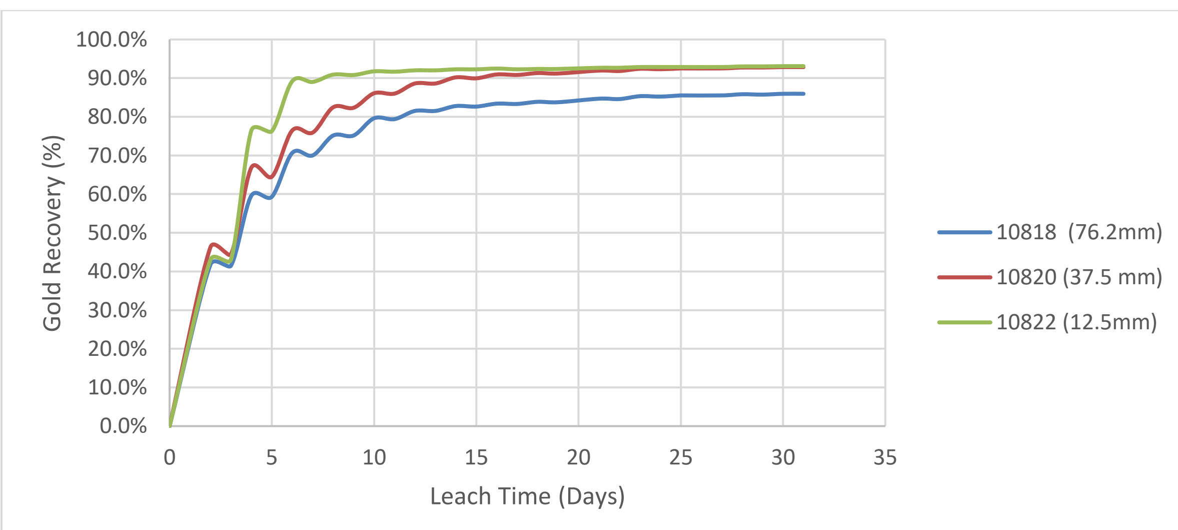
The results of the column tests are in Table 13-23, the leach curves are shown in Figure 13-8.

Table 13-23: Results of Column Leaching Tests (KCA, 1989)

KCA Test #	KCA Sample #	Max Mineral-ized Material Size (mm)	Leach Time (days)	Solution Assay Using AAS or Fire Assay		Au Recovery	NaCN added (kg/t)	$Ca(OH)_2$ added (kg/t)
				Au (g/t)	Ag (g/t)			
10818	10807A	76.2	0 - 6	0.720	0.00	69.4%	0.075	0.115

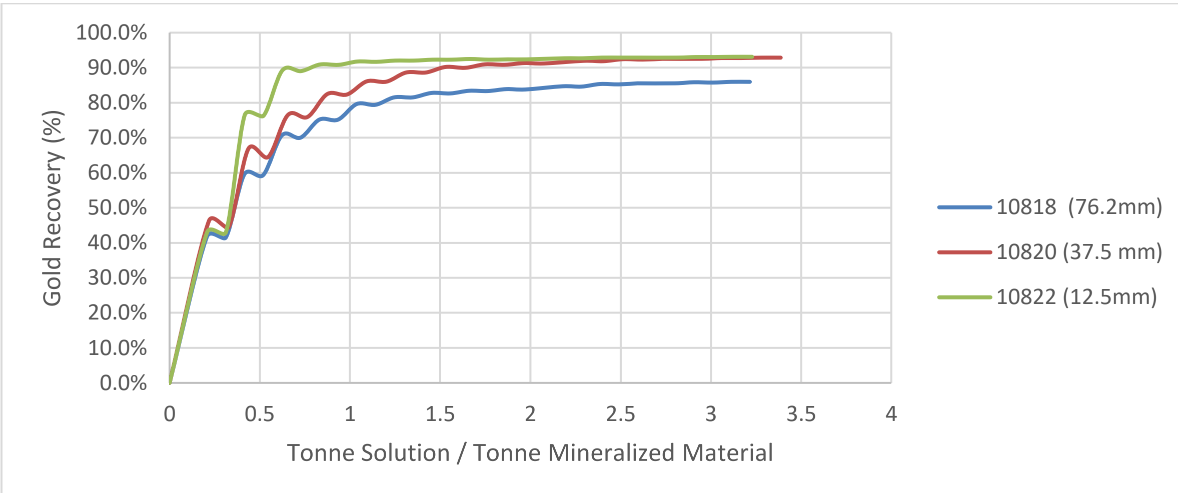
KCA Test #	KCA Sample #	Max Mineral-ized Material Size (mm)	Leach Time (days)	Solution Assay Using AAS or Fire Assay		Au Recovery	NaCN added (kg/t)	$Ca(OH)_2$ added (kg/t)
				Au (g/t)	Ag (g/t)			
			7 - 14	0.137	0.00	13.2%	0.02	0.005
			15 - 30	0.034	0.00	3.3%	0.03	0.025
			Total	0.891	0.343	86.0%	0.125	0.145
			Tails Assay	0.137	4.46			
			Calculated Head	1.037	**Notes:**	Agglomerated		
10820	10807B	37.5	0 - 6	0.720	0.00	75.0%	0.085	0.155
			7 - 14	0.137	0.00	14.3%	0.02	0.01
			15 - 30	0.034	0.00	3.6%	0.025	0.025
			Total	0.891	0.343	92.9%	0.13	0.19
			Tails Assay	0.069				
			Calculated Head	0.960	**Notes:**	Agglomerated		
10822	10807C	12.7	0 - 6	0.891	0.00	89.7%	0.100	0.145
			7 - 14	0.034	0.00	3.5%	0.035	0.025
			15 - 30	0.000	0.00	0.0%	0.035	0.03
			Total	0.926	0.343	93.1%	0.17	0.2
			Tails Assay	0.069				
			Calculated Head	0.994	**Notes:**	Agglomerated		

Figure 13-8: Leach-Rate Profiles of Samples Leached in Column Tests, Shown by Passing Feed Size in Millimetres (KCA, 1989)



The final leaching solutions had a pH range of 9.7 – 10.2, and with concentrations between 0.15 g/L and 0.22 g/L. Cyanide consumptions ranged between 0.125 kg/t and 0.17 kg/t, and calcium hydroxide consumptions between 0.145 kg/t and 0.2 kg/t. The final gold recoveries ranged from 86.67% and 93.1% with the finer material reaching a higher ultimate extraction. A graph of gold extraction vs. tonnes applied solution per tonne of mineralized material is given below in Figure 13-9.

Figure 13-9: Final Gold Extraction Against Tonnes of Leaching Solution per Tonne of Mineralized Material (KCA, 1989)



The residue from these tests were drained for 24 hours and dried. The coarse residues were then crushed to 100% passing 12.7 mm. The material was split into aliquots, and one aliquot crushed to 10 mesh, pulverized, and then subjected to fire assay. The final moisture tests and fire assay analysis of the column residues is shown in Table 13-23.

Table 13-24: Column Leach Tests Residue Moisture Analysis and Fire Assays (KCA, 1989)

KCA Test #	KCA Sample #	Max Mineralized Material Size (mm)	% Moisture	Average Assay	
				Au (g/t)	Ag (g/t)
10818	10807A	76.2	18.0%	0.146	4.63
10820	10807B	37.5	17.7%	0.069	2.83
10822	10807C	12.7	17.4%	0.069	3.77

13.3.1.2 Column Rinse Tests

After the initial column leaching tests, further tests were performed to determine the column rinsing efficiency. Over 100 kg of mineralized material was crushed to 100% passing 76.2 mm (3-inch). The mineralized material was agglomerated with 2.5 kg/t cement and 17 liters of 0.25 g/L NaCN solution. A 292 mm (11.5-inch) diameter column was loaded with the material and allowed to cure for 48 hours. Leaching progressed for five days before the test was stopped, and the solution was drained from the column. The cyanide rinsing test was then begun.

Fresh water was added to the column at a rate of 12 lph/m^2 (0.005 gpm/ft^2), and the volume of solution that drained from the column was measured every 24 hours. Titrations for free cyanide were performed on aliquots taken from this solution, and when cyanide levels reached 0.01 g/L, Weak-Acid Dissoluble (WAD) cyanide concentration tests were conducted. WAD tests were performed using the standard method (American Society for Testing and Materials designation D 2036-81 method C). The rinse test was run until the concentration of WAD cyanide dropped below 0.2 ppm for three consecutive 24-hour cycles. The results of this test are given below in Table 13-25 and the results of the analysis on the column residue tailings are in

Table 13-26.

Table 13-25: 1989 Column Rinse Test (KCA, 1989)

Day	Volume of Wash Solution (L)	pH	Conc. of NaCN (g/L)	WAD CN (ppm)	Displacement Volumes*	Tonnes of Solution/Tonne of Mineralized Material**
1	17.5	10.2	0.18		0.56	0.16
2	19.7	10.3	0.06		1.20	0.35
3	16.42	10.2	0.04		1.72	0.51
4	19.04	10	0.01		2.34	0.69
5	19.22	10.1		0.91	2.95	0.87
6	18.86	9.9		0.23	3.56	1.05
7	18.14	10		0.18	4.14	1.22
8	20.82	9.8		0.1	4.81	1.42
9	20.62	9.8		0.13	5.47	1.61

* Displacement Volume of the Column: 31.11 L

** Weight of Mineralized Material: 105.64 kg

Table 13-26: Cyanide Analysis on Effluent from Column Rinse Tests (KCA, 1989)

Cyanide Components Found in the Tailings Residue of the Column Leach Test	ppm
Soluble WAD Cyanide	0.1
Soluble Total Cyanide	0.21
Total Cyanide Remaining After Extraction of WAD and Soluble Total Cyanide	< 0.01

The results shown above demonstrate that cyanide can effectively be removed through washing of the column residue after leaching. The remaining WAD cyanide in the solution after approximately 5.5 displacement volumes is less than 0.01 ppm.

Overall, the column tests performed at KCA showed good leaching results and good physical amenability to heap leaching with agglomeration.

13.3.2 1997 Column Tests by Hazen

13.3.2.1 Assays, Particle Size Analysis, and Agglomeration Tests

The 1997 test work performed by Hazen followed up bottle roll tests conducted by them in 1995 (see Section 13.2.3 above). Approximately 5,000 kg of drill core were used to create a single bulk composite for percolation testing in five different columns. The drill holes used to create this composite are given below in Table 13-27. The material was assayed for gold and silver (Table 13-28).

Table 13-27: Composition of Bulk Composite Used for Percolation Tests (Hazen Research, 1997)

Hole Number	Mineralized Material Type (If Available)	Interval (m below ground)		Interval (ft below ground)	
		Maximum	Minimum	Maximum	Minimum
96-C3	Oxide	12.19	48.77	40	160
96-C4		9.14	45.72	30	150
96-C5	Oxide	18.29	45.72	60	150
96-C6	Oxide	15.85	28.65	52	94
96-C7	Oxide	9.14	44.20	30	145
96-C8	Oxide	9.14	45.72	30	150
96-C9		3.05	15.24	10	50

Table 13-28: Assay Data of Bulk Sample (Hazen Research, 1997)

Sample ID		Average Fire Assay		AA Analysis		S (%)	SO4 (%)
		Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)		
48624	Head A	0.566	2.06	0.583		0.19	0.09
	Head B	0.634	2.74	0.480			
Average		0.600	2.40	0.531			
CL-1 Feed Reject	Head A	0.960	8.91	0.994			
	Head B	1.029	10.29	0.926			
	Head C	0.994	9.26	0.960			
Average		0.994	9.49	0.960			
CL-2&3 Feed	Head A	0.754	4.46				
	Head B	0.754	3.43				
	Head C	0.754	4.11				
Average		0.754	4.00				

The average gold and silver assay for the bulk sample from which all column feeds were taken was 0.6 g/t and 2.4 g/t, respectively. Carbon assays were performed on the 1995 samples used for bottle roll testing and were not performed for these samples as the previous results were low.

Agglomeration studies were performed on 2 kg splits of the bulk sample composite with pieces larger than 51 mm (2”) removed. To start the test, mineralized material, cement, and lime were added to a rotating tilting/jigging pan agglomerator (or pelletizing disk). Water was added and the disk was spun until the mixture began to coalesce. Jigging or shaking was performed once agglomeration had been started to see how much material was retained on the pan. The conditions of the tests are given below in Table 13-29.

Table 13-29: Agglomeration Test Conditions (Hazen Research, 1997)

Test No.	$Ca(OH)_2$ Added (kg/t)	Cement Added (kg/t)	Total Reagent Addition (kg/t)	(> 6.35 mm) Weight % Retained
0 (Dry Sieving)	0	0	0	67.9%
1	1	0	1	79.2%
2	2	0	2	83.6%
3	3	0	3	94.9%

Test No.	$Ca(OH)_2$ Added (kg/t)	Cement Added (kg/t)	Total Reagent Addition (kg/t)	(> 6.35 mm) Weight % Retained
4	3	1	4	95.1%
5	3	2.5	5.5	94.8%
6	3	5	8	96.7%

The results of the agglomeration tests show that with the addition of 3 kg/t hydrated lime, almost 95% of material is retained in the jigging pan. As with the previous agglomeration tests conducted by KCA in 1989 (where 2.5 kg/t of cement was needed to create a strong agglomerate), the material tested here requires a minimal agglomeration using either cement or lime.

Particle size analysis of the bulk sample was also performed using both wet and dry sieving techniques (with water being used to screen the finest material through a 200 mesh (0.074 mm screen openings)). The size fractions were also assayed to determine the deportment of gold throughout the bulk sample (Table 13-30).

Table 13-30: Size Analysis and Gold Deportment (Hazen Research, 1997)

Mesh Size	Particle Size (micron)	Mass Retained (g)	Mass %	Passing	Retained	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
4"	101600	12210	25.3%	74.7%	25.3%	0.240	1.71	8.7%	14.3%
2"	50800	9825	20.3%	54.4%	45.6%	0.343	1.71	10.0%	11.5%
1"	25400	4509	9.3%	45.0%	55.0%	0.549	6.51	7.3%	20.1%
1/2"	12700	3701	7.7%	37.4%	62.6%	0.549	7.20	6.0%	18.2%
1/4"	6680	1841	3.8%	33.5%	66.5%	0.617	5.83	3.4%	7.3%
6 Mesh	3327	1557.5	3.2%	30.3%	69.7%	0.857	5.14	3.9%	5.5%
14 Mesh	1168	1972.7	4.1%	26.2%	73.8%	0.686	3.43	4.0%	4.6%
28 Mesh	589	1189.8	2.5%	23.8%	76.2%	0.686	4.46	2.4%	3.6%
65 Mesh	208	1899.1	3.9%	19.8%	80.2%	0.651	4.80	3.7%	6.2%
200 Mesh	74	1336.9	2.8%	17.1%	82.9%	0.583	3.09	2.3%	2.8%
Undersize		8238.8	17.1%	0.0%	100.0%	1.99	1.03	48.4%	5.8%
Total		48280.8	100.0%			0.701	3.03	100.0%	100.0%

The results above show that just under 50% of the gold in the bulk sample reported to the undersize. These results have been corroborated with various microscopy and mineralogy studies of the Long Valley material and indicate that gold is present in the deposit in a friable host mineral matrix as fine microscopic or sub-microscopic grains.

13.3.2.2 Column Leach Test Work

Five column leach tests were conducted with the bulk sample. A top-size of 152 mm at a P80 of 127 mm (6”, P80 of 5”) was maintained for the columns. The leach solution had a cyanide concentration of 0.25 g/L, with hydrated lime added to maintain a pH of approximately 11. The solution was applied at a rate of 12 lph/m^2 (0.005 gpm/ft^2) for the first three tests (CL-1 to CL-3), and a rate of approximately 7.2 lph/m^2

(0.003 gpm/ft^2) for the other two tests. The test conditions for all the column leaching tests are shown in Table 13-31.

Table 13-31: Column Leach Conditions Hazen Research (1997)

Test No	Diameter (mm)	Height (m)	Mass Mineralized Material (kg)	Mineralized Material Height (m)	Agglomeration	Notes	Lime Added (kg/t)	Cement Added (kg/t)	Leach Time (days)
1	304.8	2.44	182	1.98	No	Low pH – failed test	0	0	23
2	254	2.44	132	1.89	No	Normal pH	3	0	55
3	254	2.44	104.23	1.83	Yes		2.5	2.5	51
4	609.6	5.79	2283	5.74	No	17-day wash	1.75	0	166
5	609.6	6.71	2178	6.64	Yes	17-day wash	1.5	1	67

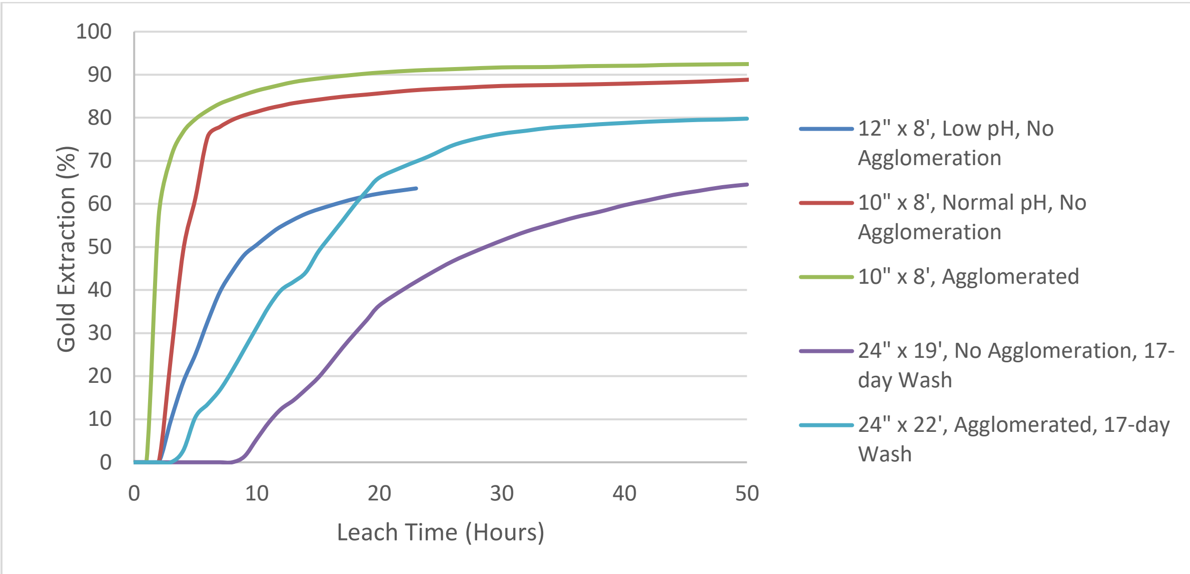
The results of the tests are given below in Table 13-32.

Table 13-32: Results of Column Leach Tests Performed by Hazen Research (1997)

Test No	Au Direct Head (g/t)	Au Calc. Head (g/t)	Au Recovery (%)	Ag Recovery (%)	NaCN Added (kg/t)
1	0.960	0.857	63.6%		0.175
2	0.960	0.960	89.4%	8.0%	0.185
3	0.960	0.891	92.5%	7.3%	0.255
4	0.960	0.823	83.4%	5.9%	0.24
5	0.960	0.926	80.8%	9.2%	0.14
Average	0.960	0.891	81.9%	7.6%	0.199

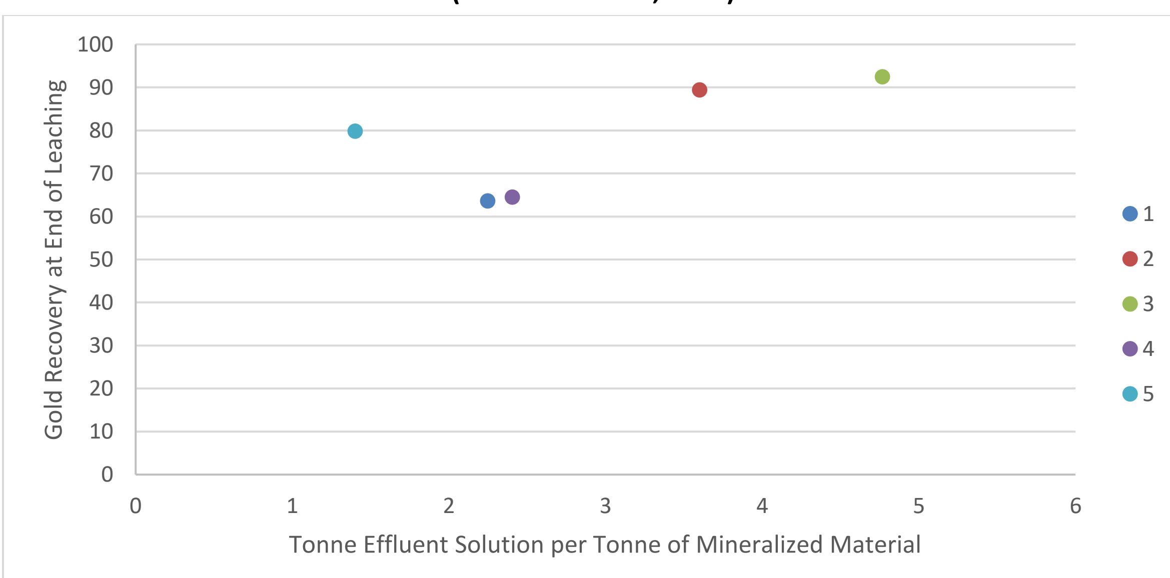
The results of column leach tests show an average gold extraction of approximately 82%. The Hazen report noted that percolation in the columns with agglomerated material was much higher at the start of leaching, but slumped after a few weeks, at which point the percolation rate reduced to a comparable level of the un-agglomerated material. The leach curves are shown in Figure 13-10.

Figure 13-10: Leach-Rate Profiles of Column Leach Tests on Bulk Sample (Hazen Research, 1997)



The Hazen report notes that tests with agglomerated material (e.g. Tests 3 and 5) leached much faster than those with un-agglomerated mineralized material (e.g. Tests 2 and 4, respectively). This may be due to the increased percolation rate through the column, which allows for faster leaching as more solution flows through the column. A graph of gold extraction vs. tonnes applied solution per tonne of mineralized material is given in Figure 13-11.

Figure 13-11: Gold Extraction Against Tonnes of Leaching Solution per Tonne of Mineralized Material (Hazen Research, 1997)



Samples were taken from sections of each column at the end of the leaching tests to assay for residual gold. The columns were each divided into four equal portions from top to bottom and representative aliquots from those samples were then assayed to determine whether that section of column experienced thorough percolation and leaching. The results of these assays are in Table 13-33.

Table 13-33: Sectional Column Residue Analysis (Hazen Research, 1997)

Column Section	Test 1	Test 2	Test 3	Test 4	Test 5
1st – top	0.274	0.069	0.069	0.137	0.103
2nd	0.240	0.206	0.069	0.103	0.309
3rd	0.994	0.069	0.103	0.206	0.103
4th - bottom	0.240	0.103	0.069	0.137	0.206
Mean Value	0.437	0.111	0.077	0.146	0.180
Median Value	0.257	0.086	0.069	0.137	0.154
Standard Deviation	0.372	0.065	0.017	0.043	0.098
Direct Au Head Assay	0.377			0.171	0.206

Generally, the second and third quarters of the column experienced lower leaching rates than the most extreme ends of the column. It is unclear why the third section of columns 1, 3, and 4 show a high level of residual gold after leaching. It is possible that these results were aberrations due to low percolation rates in these tests, or that slumping mineralized material reduced the amount of leaching solution that passed through the center of the columns in these tests.

13.3.2.3 Column Rinse Tests

Column rinse tests were conducted on the residue from tests 4 and 5. Water was introduced into the columns at a rate of 24.6 – 36.6 lph/m^2 (0.01 – 0.015 gpm/ft^2) and a titration was performed on the effluent leaving the columns with a Perstop cyanide analyzer. At the end of test 4, the pregnant leach solution had a free cyanide concentration of 58 parts per million (ppm), but after five days of washing with 584 liters per tonne (L/t) of water, the free cyanide concentration was reduced to 9.4 ppm. After fifteen days of washing with a total of 1669 L/t of water, the cyanide concentration was further reduced to 3.2 ppm.

Similar results were seen with test 5: the final effluent free cyanide concentration level was 73 ppm, which decreased after five days to 25 ppm CN, and then to 0.5 ppm CN after fifteen days, 209 L/t and 659 L/t of water, respectively. Rinsing continued for another two days to bring the final free cyanide level down to 0.2 ppm. The cyanide washing profiles with respect to rinse time and volume rinse water for Tests 4 and 5 are given below in Figure 13-12 and Figure 13-13.

Figure 13-12: Cyanide Washing Profiles with Respect to Rinse Time for Tests 4 and 5 (Hazen Research, 1997)

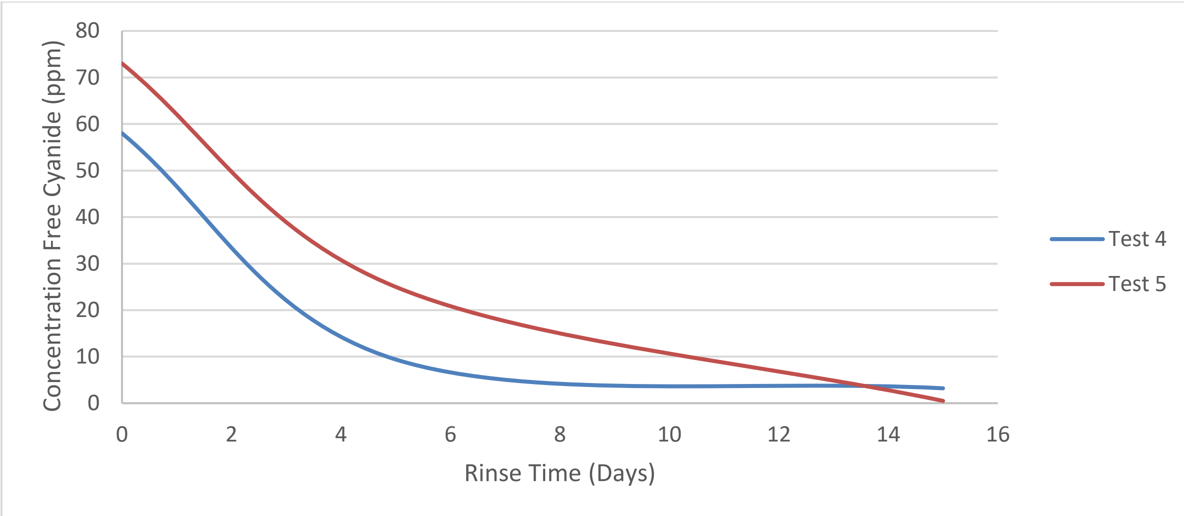
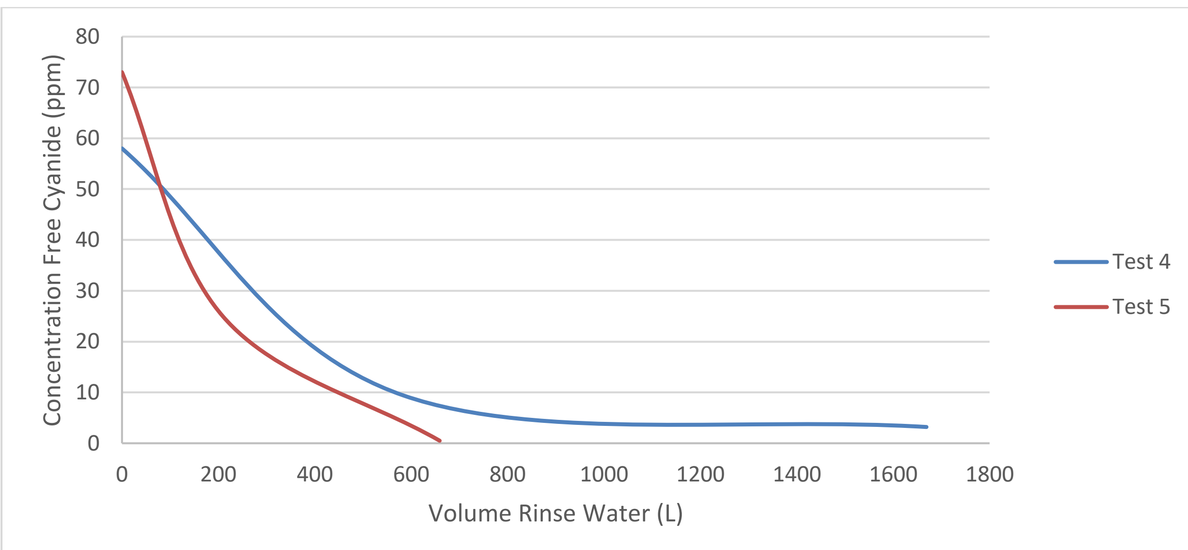


Figure 13-13: Cyanide Washing Profiles with Respect to Rinse Time for Tests 4 and 5 (Hazen Research, 1997)



These results show that agglomeration appears to reduce the amount of rinse water required for a given cyanide concentration.

13.4 1997 Microscopy Test Work

Samples taken from coarse rejects of reverse-circulation rotary drill chips were analyzed by the University of Nevada using Scanning Electron Microscopy (SEM) and Optical Microscopy (OM) (Weiss, 1997). The material came from “heavy-mineral, sulfide-rich” concentrates to increase the probability of locating gold grains. However, although several samples were examined, only one polished section (248-250C) showed discernable gold grains through use of a Scanning Electron Microscope. Optical Microscopy did not detect any visible gold.

The gold grains observed ranged from 1 microns (µm) to 6 µm and appeared mainly free of the surrounding sulfide matrix. Energy Dispersive Spot Analyses on sample 248-250C showed a low concentration of silver present in the gold grains (<25%). Pyrite and Marcasite were observed to be the main sulfide minerals present throughout all the examined samples and these minerals were found to be porous and well defined.

The report hypothesized that much of the gold present in these samples should be recoverable using cyanide since the gold present in sample 248-250C seems to be free from the sulfide matrix. However, since only one of the samples tested showed any distinct gold grains, the author also supposed that much of the gold present in the other samples is either enveloped in sulfide or clay minerals or is sub-microscopic. Low grade deposits like Long Valley are often difficult when it comes to locating visible gold grains.

13.5 Test Work Summary and Conclusions

The test work supports that the oxide materials are generally free milling and amenable to heap leach recovery and the sulfide materials are more refractory and exhibit a low heap leach recovery. Transitional materials fall somewhere in between these two extremes. The issue for Long Valley is defining oxide, transition or sulfide materials within the mineralized body so that appropriate gold recoveries can be assigned.

Most of the testing focused mainly on bottle roll and column leach tests. The results varied widely; some samples, characterized as “oxide” material, exhibited poor leaching characteristics. While other samples, either “mixed” or “sulfide” material, provided good leaching characteristics though not as often. The average gold extraction data from all bottle roll tests are shown in Table 13-34.

Table 13-34: Un-weighted Average Results from All Bottle Roll Tests By Mineralized Material Type

Mineralized Material-/Rock-Type	ALL	Oxide	Mixed	Sulfide	Unclassified
Average Gold Extraction (%)	48.0%	75.6%	48.6	11.2%	47.4%
Average Silver Extraction (%)	27.6%	21.0%	18.7%	23.6%	42.3%

The results of column leach tests were more promising. The average gold extraction from all column tests was 85.2%. Silver extraction was generally low, averaging only 7.6% (with only four of the tests recording silver extraction data). The material tested in the columns was generally classified as oxide type material.

Agglomeration improved the leach performance by reducing slump and improving percolation. Both lime and cement appear to be effective at moderate doses.

Column rinse tests revealed that residual cyanide levels could be lowered to 0.5 ppm after fifteen days of rinsing with approximately 659 L/t of water and can be lowered to below 0.2 ppm with 1220 L/t of water. No significant percolation or slumping problems were noted in any of the tests.

13.6 Long Valley Operating Parameters

The Long Valley project is amenable to conventional heap leaching for the oxide and transitional materials. The following operating parameters are presented for development of a flowsheet and economic model.

Gold Recovery

Oxide – Gold 80%, Silver 40%

Transition - Gold 60%, Silver 25%

Sulfide - Gold 20%, Silver 25%

Crush Size

P80 – 1.5 in (37.5 mm)

Agglomeration

2.5 kg cement 10% moisture with barren solution and a 2-day cure

Reagent Addition

Cyanide

0.05 kg/t Oxide

0.08 kg/t Transition

0.25 kg/t Sulfide

Lime

0.19 kg/t Oxide

0.50 kg/t Transition

1.5 kg/t Sulfide

Leach Cycle

90 days primary – 180 days total - target 2 Ts/To (solution tonnes per mineralized material tonne)

Heap Configuration

10 m lifts

100 m maximum height

13.7 Metallurgical Recommendations

The following recommendations have been made for the further development of the Long Valley Project:

- Improve metallurgical understanding of the orebody through additional metallurgical sampling. Drilling should be weighted to match the distribution of sulfide, oxide, transition, siliceous, and argillic material.
- Sulfide-sulfur assays should be conducted on all samples in addition to gold, silver, hot cyanide leach, and a full ICP scan.
- Further test work should be considered for the Long Valley project:

- Crusher work index and abrasion tests should be conducted to confirm crusher design and wear rates.
- Agglomeration tests should be performed to confirm the optimal mix of cement/lime, and moisture necessary to achieve acceptable percolation and leaching results.
- A comprehensive array of column tests should be arranged with representative samples from all areas of the deposit. Minimal column work is necessary for the sulfide material as it is not amenable to heap leaching.
- The optimization of the crush size requires further investigation and the investigation of HPGR may be warranted given the material characteristics.
- Sulfide mineralogy should be tested to define a suitable flowsheet for this material if economically warranted. Several basic crush, grind leach tests should be conducted followed by additional testing if the material is refractory to conventional processing techniques.

- The estimated cost for this program is $500,000 including drilling.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Mineral resource estimation described in this section for the Long Valley project follows the guidelines of Canadian CIM. The gold-grade block model for this resource estimate was completed in 2003 (MDA, 2003). The gold resource estimate for this report was completed on July 15, 2020. The previous, 2003 resource block model was updated using density and geologic models based on interpretations completed in 2020 with information from KORE's re-logging of drill-hole cuttings. The block gold-grade estimates remain unchanged from the 2003 block model. Silver resources were not estimated and there are no mineral reserves estimated for the Long Valley project as part of this report.

There is no affiliation between Mr. Prenn and KORE except that of an independent consultant/client relationship. Although Mr. Prenn is not an expert with respect to any of the following aspects of the project, he is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Long Valley mineral resources as of the date of this report.

The author classifies resources in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories to be in compliance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore Canadian National Instrument 43-101. CIM mineral resource definitions are given below, with CIM's explanatory material shown in italics:

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the *consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.*

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified

Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

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 are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

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 estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling, and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of *the mineral deposit.*

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

The author reports resources at cutoffs that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions, because of the regulatory requirements that a resource exists "in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction."

14.2 Data

The gold resources for the Long Valley project were estimated using data provided to MDA primarily by Vista for the 2003 report (MDA, 2003). Hardcopy and digital data received from Vista included: drill hole database with collar locations, down hole survey data, analytical data; topographic data, drill hole location maps, drill hole cross sections, geologic drill logs, and numerous in-house reports.

For this repost and the current mineral resources, the geologic database has been upgraded as a result of KORE's 2020 re-logging effort in which a significant portion of the drill holes within the resource areas were re-logged focusing on lithology, alteration, and oxidation.

The drill hole data were checked prior to loading the data into a database; a few minor errors were discovered and corrected prior to importing the data into a Surpac® mining software database. Analytical results that were less than the detection limit were set to zero. All subsequent modeling of the Long Valley resource was performed using Surpac®.

14.3 Geology Pertinent to the Resource Model

The mineralization identified at the Long Valley property is typical of the shallower portions of an epithermal, low-sulfidation type of gold-silver deposit. The principal host rocks for the gold mineralization are the flat-lying, caldera-fill interbedded siltstone, tuff, and volcaniclastic sedimentary rocks and, to a lesser extent, the adjacent and underlying resurgent rhyolite that crops out along the west margin of the north-south-trending Hilton Creek fault zone. This normal fault zone (down to the east), along with splays of this fault zone which extend into the central part of the Hilton Creek mineralized zone, as well as the Southeast zone, seem to control the distribution of gold mineralization in the Long Valley deposit. It is assumed that alteration and mineralizing fluids ascended both along these fault conduits, and also along the gently east-dipping sedimentary-rhyolite (intrusive/structural?) contact, and then spread laterally with higher-grade mineralization being related to areas of cross-faults and fractures.

In much of the deposit, mineralization is associated with zones of clay alteration and/or silicification. These alteration types are well developed in all of the volcaniclastic sediments and, as such, host-rock type does not appear to have a major control over the distribution and grade of mineralization. The predominant clay mineral has been determined to be kaolinite, while the silicification types can be chalcedony, quartz, or opal. Multiple periods of brecciation and silicification are evidenced by cross-cutting silica veinlets and silicified breccia fragments in otherwise clay-altered rocks.

The Hilton Creek mineralized zone is known to be some 8,000 feet in length, while the Southeast zone is about 5,000 feet in length. The mineralized zones are generally flat-lying or have a slight dip (10-15 degrees) to the east and have a width in plan view (across the trend) in the range of 500 to 1,500 feet, but average about 1,000 feet in width. The mineralized zones are typically from 50 to 200 feet thick and average about 125 feet thick in the Hilton Creek zone, and 75 feet thick in the Southeast zone. Mineralization in the South and Southeast zones typically is exposed at or very near the surface, while the top of the Hilton Creek mineralization is usually covered by 20 to 50 feet of alluvium.

Based on drilling, mineralization appears to generally be contiguous between the South, Southeast, and Hilton Creek zones. These same zones appear to contain the vast majority of the estimated mineral resources described in this report. Drilling is widely spaced in and between the North, Middle, and South zones, and it may be possible that with additional drilling, these zones may be shown to be contiguous with the better-defined zones to the south.

Gold-silver mineralization is quite continuous throughout the zones and is well defined using a 0.010 oz Au/ton cutoff grade. Numerous zones of higher-grade mineralization (0.050 oz Au/ton) are present within the continuous zones of low-grade (0.010 oz Au/ton) gold mineralization, particularly in the Hilton Creek zone. These higher grades may relate to zones of enhanced structural preparation. Silver grades are generally in the range of 0.1 to 0.5 oz Ag/ton within the gold-mineralized zones, appear to be more erratic in nature, but generally have a positive correlation with higher gold values. Due to the generally low silver grades and poor metallurgical recoveries, silver is a minor contributor to the deposit economics. Accordingly, silver was not included in the grade model and resource estimate.

14.4 Geology Model

East-west cross sections were plotted on 100-f00t intervals through the Hilton Creek, South, and Southeast areas. The topographic profile and drill hole traces were plotted on each cross section, with gold sample assays and logged geology along the drill traces.

Using the drill data as a guide, the overburden-bedrock contact and the contact between the rhyolite and lacustrine sequence were modeled on each section and 3D surfaces of each were created. The surfaces were used to code the block model by lithology and were also used to assign density into the block model.

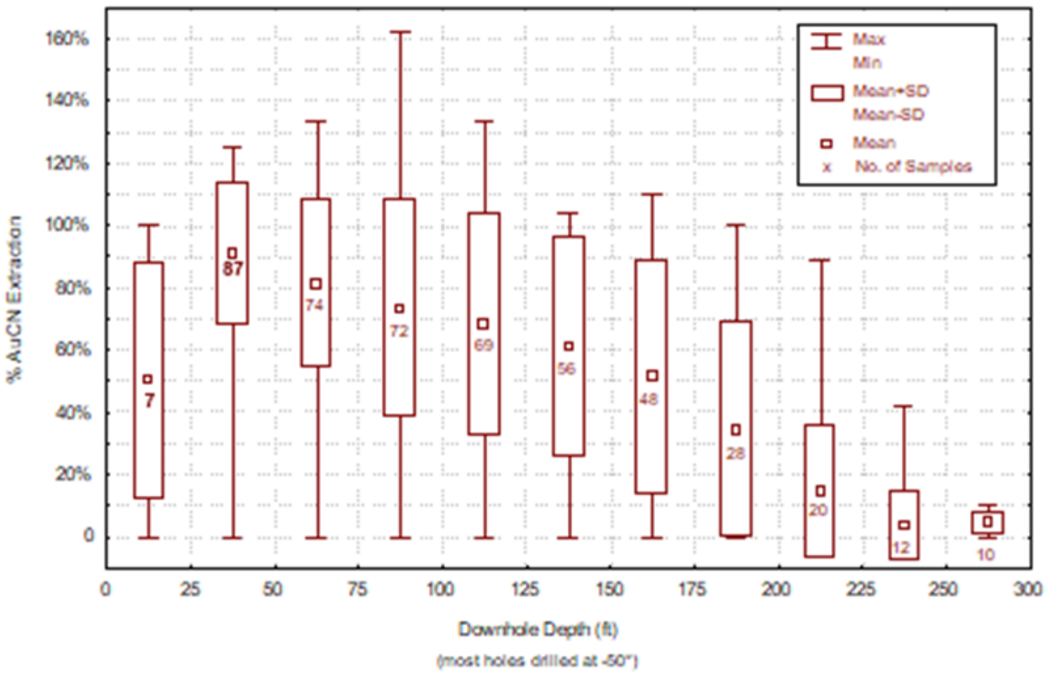
14.5 Oxidation Model

The initial metallurgical model was based on an oxide-sulfide boundary provided to MDA by Vista for the 2003 report (MDA, 2003). The surface was created by R. Steininger, consultant to Royal Gold and Mono County Mining Company, who generally determined the boundary location by recording the last occurrence of oxide minerals observed in the drill cuttings or core (Steininger, pers. comm.). This boundary represents the deepest limits of oxidation.

Mr. Prenn reviewed the cyanide bottle roll data (see Section 13.0 for test details) and the oxide-sulfide boundary was then revised in local areas to correspond with the marked decrease in gold extraction values characteristic of the sulfide material. This boundary represents the deepest limits of oxidation.

In order to better define the oxide zone as determined by Steininger, Mr. Prenn reviewed the cyanide bottle-roll leach results from samples coded as oxide; the results are shown graphically in Figure 14-1Figure 14-1.

Figure 14-1: AuCN Extraction Data for Samples Logged as Oxide



Visual recognition in drill cuttings, along with decreased (<50%) gold recovery values from cyanide bottle-roll leach assays, indicated a transition zone at the base of the oxide zone that occurs approximately 150 to 200 feet below the topographic surface. Below 200 feet, the low gold extraction values of <25% indicate sulfide is the dominant material type. Using these limiting depths, the model was coded first to oxide and sulfide using the oxide/sulfide boundary. Where the boundary is at a depth greater than 150 feet, the model is coded as transition material below 150-foot depth, but above the oxide/sulfide boundary.

14.6 Density Model

The densities of the rocks present in the Long Valley deposit are highly variable, with density test results ranging from 0.93 to 2.83 g/cm^3. The results of 12 density tests completed by Royal Gold on core from seven drill holes are summarized in Table 14-1. Amax completed 93 tests on core from 10 drill holes; results are summarized in Table 14-2. Mr. Prenn collected 10 samples during the 2002 site visit for density verification; results are shown in Table 14-3:.

Table 14-1: Royal Gold Density Tests

Drill Hole - Footage	Density		Tonnage Factor
	grams/cm^3	lbs/ft^3	ft^3/ton
C1 - 165'	1.80	112.3	17.8
C1 - 175'	1.63	101.7	19.7
C2 - 120'	1.25	78.0	25.6
C2 - 175'	1.26	78.6	25.4
C2 - 195'	2.80	174.7	11.4
C3 - 140'	1.54	96.1	20.8
C3 - 148'	2.43	151.6	13.2
C4 - 135'	2.03	126.7	15.8
C5 - 148'	2.47	154.1	13.0
C7 - 99'	2.44	152.3	13.1
C8 - 115'	2.12	132.3	15.1
C8 - 148'	1.82	113.6	17.6
Average	**1.97**	**122.7**	**16.3**

Table 14-2: Amax Density Tests

Hole	Depth	Density		T.F.	Hole	Depth	Density		T.F.
	ft	grams/cm^3	lbs/ft^3	ft^3/ton		ft	grams/cm^3	lbs/ft^3	ft^3/ton
LV97-C11	35.6	1.28	79.9	25.0	LV97-C15	125.3	1.81	112.9	17.7
LV97-C11	55.5	1.12	69.9	28.6	LV97-C15	151.3	1.52	94.8	21.1
LV97-C11	94.3	1.48	92.4	21.6	LV97-C15	178	1.48	92.4	21.6
LV97-C11	115.5	1.73	108.0	18.5	LV97-C15	203.5	1.80	112.3	17.8
LV97-C11	148.5	1.72	107.3	18.6	LV97-C15	227.5	1.80	112.3	17.8
LV97-C11	174.8	1.86	116.1	17.2	LV97-C16	24	1.52	94.8	21.1
LV97-C11	199.1	1.68	104.8	19.1	LV97-C16	50.2	1.35	84.2	23.8
LV97-C11	222.6	1.73	108.0	18.5	LV97-C16	72.6	1.76	109.8	18.2
LV97-C11	247.1	1.84	114.8	17.4	LV97-C16	103	1.74	108.6	18.4
LV97-C11	273	1.67	104.2	19.2	LV97-C16	129.5	2.52	157.2	12.7
LV97-C11	298	1.55	96.7	20.7	LV97-C16	155.2	2.36	147.3	13.6
LV97-C12	8.1	1.71	106.7	18.7	LV97-C16	177.7	2.42	151.0	13.2
LV97-C12	36.9	1.05	65.5	30.5	LV97-C16	195.3	2.64	164.7	12.1
LV97-C12	91	1.75	109.2	18.3	LV97-C17	28.2	1.31	81.7	24.5
LV97-C12	127	2.37	147.9	13.5	LV97-C17	51.2	1.68	104.8	19.1
LV97-C12	151.4	1.58	98.6	20.3	LV97-C17	73.8	1.68	104.8	19.1
LV97-C12	171.4	1.84	114.8	17.4	LV97-C17	94.1	2.06	128.5	15.6
LV97-C12	187.8	1.70	106.1	18.9	LV97-C17	108.9	1.58	98.6	20.3
LV97-C12	207	1.58	98.6	20.3	LV97-C17	130.2	1.69	105.5	19.0
LV97-C12	218.5	1.88	117.3	17.1	LV97-C17	153.1	1.50	93.6	21.4
LV97-C12	248.7	1.92	119.8	16.7	LV97-C18	26.4	2.22	138.5	14.4
LV97-C13	2	1.72	107.3	18.6	LV97-C18	51.8	2.16	134.8	14.8
LV97-C13	18	1.13	70.5	28.4	LV97-C18	79.2	2.50	156.0	12.8
LV97-C13	43.7	1.22	76.1	26.3	LV97-C18	101	2.26	141.0	14.2
LV97-C13	67.1	1.83	114.2	17.5	LV97-C18	126.3	2.25	140.4	14.2
LV97-C13	113.3	1.84	114.8	17.4	LV97-C18	153.4	2.16	134.8	14.8
LV97-C13	148.6	1.29	80.5	24.8	LV97-C18	176.5	2.49	155.4	12.9
LV97-C13	177.1	2.14	133.5	15.0	LV97-C18	201.8	2.54	158.5	12.6
LV97-C13	198	1.66	103.6	19.3	LV97-C18	222.5	2.15	134.2	14.9
LV97-C13	222.6	1.48	92.4	21.6	LV97-C19	27.5	1.06	66.1	30.3
LV97-C13	252	1.94	121.1	16.5	LV97-C19	45	1.39	86.7	23.1
LV97-C13	274.9	1.80	112.3	17.8	LV97-C19	73.8	2.53	157.9	12.7
LV97-C13	295.1	2.08	129.8	15.4	LV97-C19	109.5	2.83	176.6	11.3
LV97-C14	29.7	1.58	98.6	20.3	LV97-C19	129.5	2.42	151.0	13.2
LV97-C14	45.4	1.22	76.1	26.3	LV97-C19	154.2	2.20	137.3	14.6
LV97-C14	73.7	1.65	103.0	19.4	LV97-C19	178.8	2.31	144.1	13.9
LV97-C14	125.6	1.41	88.0	22.7	LV97-C19	211.5	1.29	80.5	24.8
LV97-C14	153.4	1.54	96.1	20.8	LV97-C19	263.9	1.34	83.6	23.9
LV97-C14	167.2	1.76	109.8	18.2	LV97-C20	30.2	0.93	58.0	34.5
LV97-C14	195.8	1.76	109.8	18.2	LV97-C20	50.9	2.47	154.1	13.0
LV97-C14	222.2	1.71	106.7	18.7	LV97-C20	73.8	2.21	137.9	14.5
LV97-C14	242.8	1.94	121.1	16.5	LV97-C20	102.1	1.45	90.5	22.1
LV97-C14	277.1	2.07	129.2	15.5	LV97-C20	125.6	1.49	93.0	21.5
LV97-C14	298.4	1.67	104.2	19.2	LV97-C20	150.4	1.67	104.2	19.2
LV97-C15	2.8	1.73	108.0	18.5	LV97-C20	174.5	2.04	127.3	15.7
LV97-C15	31.7	1.56	97.3	20.6					
LV97-C15	46.7	1.00	62.4	32.1	Averages	93 samples		111.6	17.9
LV97-C15	99.8	1.95	121.7	16.4					

Table 14-3: MDA Density Tests (Surface Samples)

KCA Test No.	Sample Location	Density		T. F.
		grams/cm^3	lbs/ft^3	ft^3/ton
30565A	Western Hilton Creek	2.54	158.6	12.6
30564B	Northern Hilton Creek	2.40	150.0	13.3
30565B	Northern Hilton Creek	2.00	124.9	16.0
30565B	Northern Hilton Creek	1.48	92.3	21.7
30565B	Northern Hilton Creek	1.53	95.7	20.9
30565B	Northern Hilton Creek	1.67	103.9	19.2
Average	Northern Hilton Creek	1.82	113.3	17.6
30565C	Southeast Zone	1.91	118.9	16.8
30565C	Southeast Zone	1.92	120.1	16.7
30565C	Southeast Zone	2.31	144.2	13.9
30565C	Southeast Zone	2.30	143.8	13.9
Average	Southeast Zone	2.11	131.8	15.2
Average	**All Samples**	**2.01**	**125.2**	**16.0**

For this current report, the lithology model and gold domain envelopes were used to code the Royal Gold and Amax drill data. After reviewing the spatial distribution and statistical characteristics of the density data, seventeen highly anomalous measurements were removed from the data set. The density data were then converted to tonnage factor values and an average tonnage factor by rock type as shown in Table 14-4: was assigned to the block model.

Table 14-4: Tonnage Factor (ft^3/ton) Values Used in the Long Valley Block Model

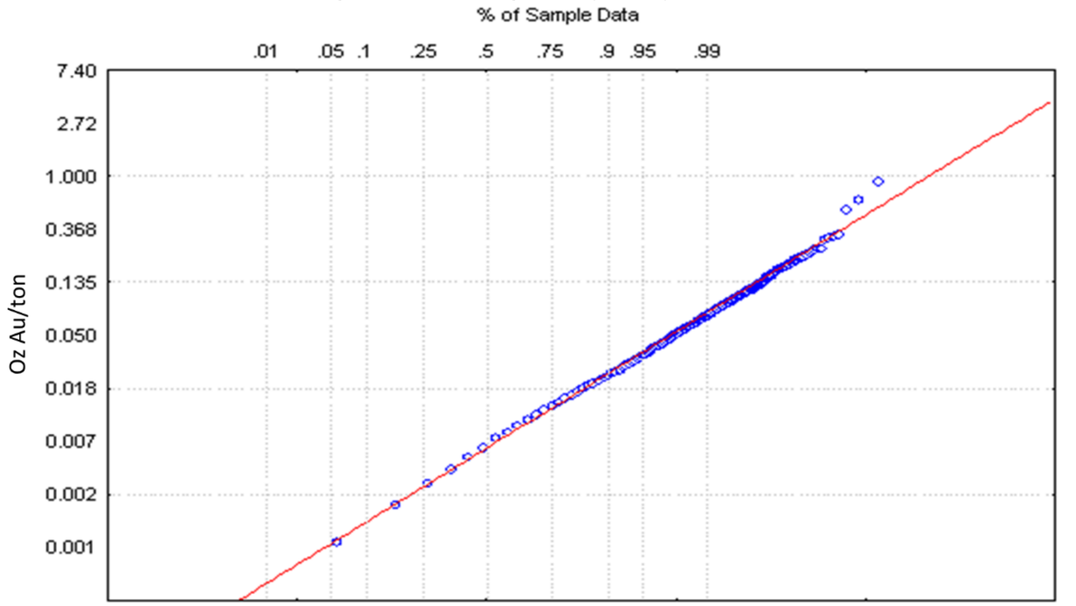
Material Type	Count	Mean	Median	Min	Max	Model_TF
Qal	8	20.81	20.28	18.52	25.03	20.0
Tuff-sed_minz'd	44	15.96	15.12	11.33	21.65	15.5
Tuff-sed_unminz'd	12	17.98	18.21	12.62	21.65	18.0
Rhy_minz'd	11	18.29	17.81	14.98	21.65	18.0
Rhy_unminz'd	13	17.96	18.74	15.41	21.51	18.5

14.7 Long Valley Gold Resource Model

14.7.1 Deposit Sample Statistics

Gold resources modeled and estimated for the Long Valley project are contained within the Hilton Creek, South, and Southeast zones. The author plotted the gold grade distribution of all drill sample data (excluding air track samples) from these three zones to help identify grade populations to aid in the resource modeling. As shown in Figure 14-2, the overall distribution of gold grades is somewhat linear, with subtle breaks around 0.01, 0.02, 0.05, 0.10, and 0.15 oz Au/ton and a distinct break at about 0.25 oz Au/ton. Nine samples above this break were capped to 0.25 oz Au/ton prior to compositing and grade estimation.

Figure 14-2: Long Valley Sample Data



Hilton Creek, South, and Southeast Zones (excl. airtrack data)

Summary statistics of the sample assays within the model extents of the Hilton Creek, South, and Southeast zones are shown in Table 14-5:. Since the South zone appears to be the northern extension, or continuation, of the Hilton Creek zone, it was decided to model these two zones together.

Table 14-5: Long Valley Sample Statistics (Samples within Resource Model Extents)

All Samples with Model Extents	No. Samples	Mean	Minimum	Maximum	Std. Dev.	C.V.
Sample Length (ft)	42,084	5.0	0.9	10.0	0.040	0.010
Au Grade (oz/ton)	42,084	0.009	0.0	0.890	0.016	1.655
Capped Au Grade (oz/ton)	42,084	0.009	0.0	0.250	0.014	1.471
AuCN Grade (oz/ton)	695	0.008	0.0	0.065	0.012	1.461
Hilton Creek / South Zone Samples	**No. Samples**	**Mean**	**Minimum**	**Maximum**	**Std. Dev.**	**C.V.**
Sample Length (ft)	34,352	5.0	1.0	5.0	0.010	0.000
Au Grade (oz/ton)	34,352	0.010	0.0	0.530	0.015	1.585
Capped Au Grade (oz/ton)	34,352	0.010	0.0	0.250	0.014	1.480
AuCN Grade (oz/ton)	695	0.008	0.0	0.065	0.012	1.461
Southeast Zone Samples	**No. Samples**	**Mean**	**Minimum**	**Maximum**	**Std. Dev.**	**C.V.**
Sample Length (ft)	7,732	5.0	0.9	10.0	0.100	0.020
Au Grade (oz/ton)	7,732	0.008	0.0	0.890	0.016	2.026
Capped Au Grade (oz/ton)	7,732	0.008	0.0	0.250	0.010	1.342
AuCN Grade (oz/ton)	0	-	-	-	-	-

14.7.2 Gold Mineral Domain Assay and Composite Statistics

The cross sections were reviewed to determine if the gold grade populations identified in the grade distribution plot (Figure 14-2, above) represented continuous zones of mineralization. The author found that grade ranges of approximately 0.009 to 0.02, 0.02 to approximately 0.10, and greater than 0.10 oz Au/ton showed the best continuity between drill holes and from section to section and constructed mineral envelopes ("domains") using these three grade ranges.

The cross-sectional grade model was digitized and transferred to 10-foot spaced level maps for the final interpretation and refinement. A three-dimensional block model was made of the deposit area with blocks 20 feet x 20 feet x 10 feet vertical in size. The model blocks were coded to the appropriate gold zone, as listed in Table 14-6. Background mineralization is that mineralization outside of the defined grade domains, but within the model extents.

Table 14-6 Long Valley Gold Grade Domains

Hilton Creek / South Zone		Southeast Zone	
Au Domain	Au Grade	Au Domain	Au Grade
1	~0.009 - 0.02	21	~0.009 - 0.02
2	0.02 ~ 0.10	22	0.02 ~ 0.10
3	> 0.10	23	> 0.10
99	background	99	background

Bedrock drill samples were composited down hole into 10-foot composites. Down hole composites were used, rather than compositing strictly within each grade envelope, in order to better model the apparent gradational contacts between grade populations, as suggested by the distribution plot of the sample data and supported by review of the data on cross section. Summary statistics of the composite data are presented in Table 14-7:. Due to the few composites > 0.10 oz Au/ton in the Southeast zone, zones 22 and 23 were combined and modeled together.

Table 14-7: Long Valley Composite Statistics

All Composites		No. Comps	Mean	Minimum	Maximum	Std. Dev.	C.V.
All Domains	Length (ft)	21,456	9.900	1	10.000	0.700	0.100
	Au Grade (oz/ton)	21,456	0.009	0	0.452	0.014	1.498
	Capped Au Grade (oz/ton)	21,456	0.009	0	0.250	0.013	1.369
Hilton Creek / South Zone		**No. Comps**	**Mean**	**Minimum**	**Maximum**	**Std. Dev.**	**C.V.**
Au Domain 1	Length (ft)	4,293	10.000	5.0	10.000	0.400	0.0
	Au Grade (oz/ton)	4,293	0.012	0.001	0.046	0.005	0.371
	Capped Au Grade (oz/ton)	4,293	0.012	0.001	0.046	0.005	0.371
Au Domain 2	Length (ft)	2,179	10.000	5.0	10.000	0.400	0.0
	Au Grade (oz/ton)	2,179	0.033	0.003	0.265	0.018	0.545
	Capped Au Grade (oz/ton)	2,179	0.032	0.003	0.146	0.016	0.502
Au Domain 3	Length (ft)	70	10.000	10.0	10.000	0.000	0.0
	Au Grade (oz/ton)	70	0.122	0.068	0.323	0.047	0.383
	Capped Au Grade (oz/ton)	70	0.105	0.067	0.250	0.031	0.297
Southeast Zone		**No. Comps**	**Mean**	**Minimum**	**Maximum**	**Std. Dev.**	**C.V.**
Au Domain 21	Length (ft)	962	10.000	5.0	10.000	0.400	0.0
	Au Grade (oz/ton)	962	0.012	0.001	0.046	0.005	0.371
	Capped Au Grade (oz/ton)	962	0.012	0.001	0.046	0.005	0.371
Au Domain 22	Length (ft)	241	10.000	5.0	10.000	0.400	0.0
	Au Grade (oz/ton)	241	0.033	0.003	0.265	0.018	0.545
	Capped Au Grade (oz/ton)	241	0.032	0.003	0.146	0.016	0.502
Au Domain 23	Length (ft)	3	10.000	10.0	10.000	0.000	0.0
	Au Grade (oz/ton)	3	0.122	0.068	0.323	0.047	0.383
	Capped Au Grade (oz/ton)	3	0.105	0.067	0.250	0.031	0.297
Background		**No. Comps**	**Mean**	**Minimum**	**Maximum**	**Std. Dev.**	**C.V.**
Au Domain 99	Length (ft)	13,708	9.900	5.0	10.000	0.800	0.1
	Au Grade (oz/ton)	13,708	0.003	0	0.075	0.003	0.983
	Capped Au Grade (oz/ton)	13,708	0.003	0	0.054	0.003	0.975

14.7.3 Gold Grade Estimation

Variography was initially performed separately on composites from each gold domain, using various lag distances and numerous directions, but none showed sufficient structure that could be modeled. Variograms were constructed using the combined composites from domains 1 to 3 (Hilton Creek / South zone) and combined composites from domains 21 to 23 (Southeast zone) which resulted in variograms that showed good continuity. The variogram results are shown in Table 14-8:. Figure 14-3 illustrates the variogram (major axis) used to model the Hilton Creek / South zone. Figure 14-4 illustrates the omni-directional variogram for the Southeast zone is considerably shorter.

Table 14-8: Long Valley Variogram Parameters

		Hilton Creek / South Zone		Southeast Zone	
Au Domain		1 - 3	99	21 - 23	99
C_0		0.112	0.06	0.068	0.07
C_1		0.07	0.04	0.157	0.06
C_2		n/a	n/a	n/a	0.02
Azimuth	major	120°	omni	omni	omni
	semimajor	30°			
	minor	0°			
$Range_1$ (ft)	major	200	300	50	110
	semimajor	140	300	50	110
	minor	25	300	50	110
$Range_2$ (ft)	major	n/a	n/a	n/a	500
	semimajor	n/a	n/a	n/a	500
	minor	n/a	n/a	n/a	500

Figure 14-3: Hilton Creek / South Zone Variogram (Major Axis)

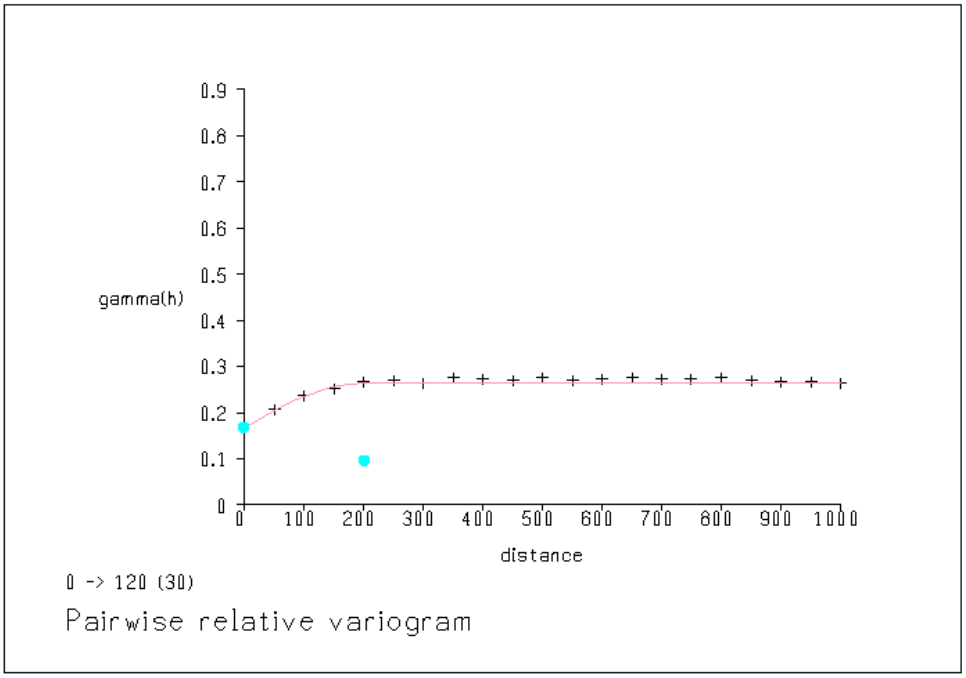


Figure 14-4: Southeast Zone Variogram (Omni-Directional)

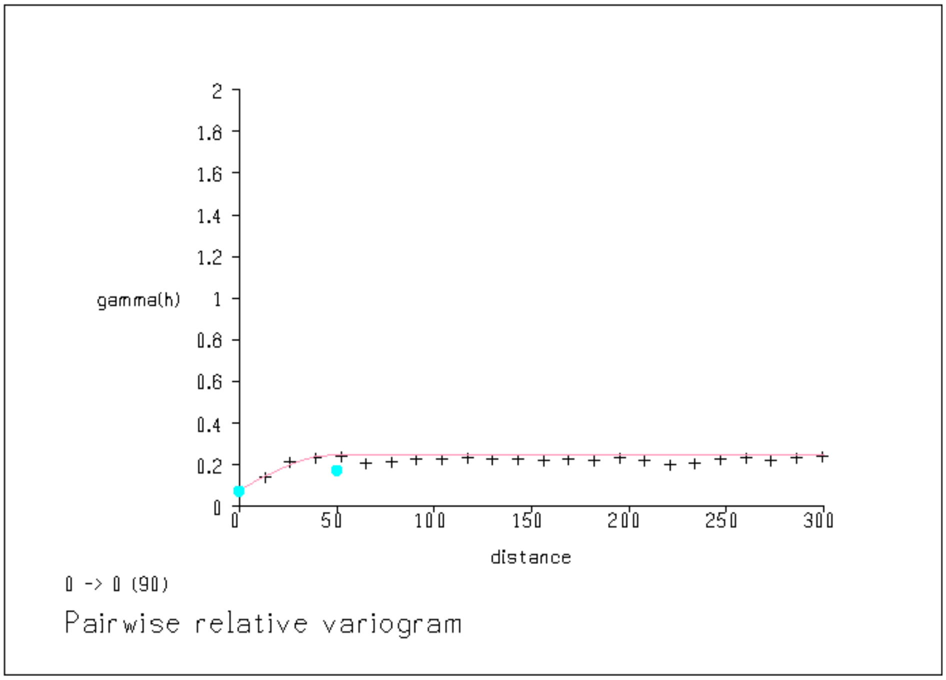


Table 14-9: Long Valley Estimation Parameters

Hilton Creek / South Zone		Au Domains 1 - 3			Au Domain 99	
Estimation Pass		1	2	3	1	2
No. of Composites	min.	2	2	1	2	2
	max.	10	6	10	10	10
Max. Composites per drill hole		3	2	3	3	3
Search Direction	major	120°	120°	0°	0°	0°
	semimajor	30°	30°	0°	0°	0°
	minor	0°	0°	0°	0°	0°
Search Distance (ft)	major	200	120	500	300	25
	semimajor	140	84	500	300	25
	minor	25	15	500	25	25
Grade Restriction (oz Au/ton)		none	none	none	≤ 0.02	none
Southeast Zone		**Au Domains 1 - 3**			**Au Domain 99**	
Estimation Pass		1	2	3	1	2
No. of Composites	min.	2	2	1	2	2
	max.	10	6	10	10	10
Max. Composites per drill hole		3	2	3	3	3
Search Direction	major	0°	0°	0°	0°	0°
	semimajor	0°	0°	0°	0°	0°
	minor	0°	0°	0°	0°	0°
Search Distance (ft)	major	200	120	500	300	25
	semimajor	200	120	500	300	25
	minor	25	25	500	25	25
Grade Restriction (oz Au/ton)		≤ 0.05	none	≤ 0.05	≤ 0.02	none

Three kriging passes were used to estimate the gold resources within the Hilton Creek / South zone and Southeast zone; gold domains 1 to 3 and 21 to 23, respectively. The first pass was done to estimate blocks within the variogram range; the second pass was done to avoid over-smoothing and better honor the local data; and the third pass was done to fill in the portions of the domains left unestimated by passes one and two with inferred material. All blocks that received estimated grades during the third pass are considered Inferred. The background mineralization (domain 99) was estimated in two passes to restrict the over-extrapolation of higher-grade values that would be unrestrained by their exclusion from the grade domain envelopes. The estimation parameters used for the Long Valley resource model are listed in

Table 14-9:.

14.8 Long Valley Resource Classification

Gold mineral resources for the Long Valley property were classified based on gold domain and the average distance of the composites used to estimate the model blocks, as shown in Table 14-10:. For any given model block to be classified as Indicated, the grade of the block had to be estimated from at least two composites.

Table 14-10: Long Valley Resource Classification Parameters (Average Composite Distance)

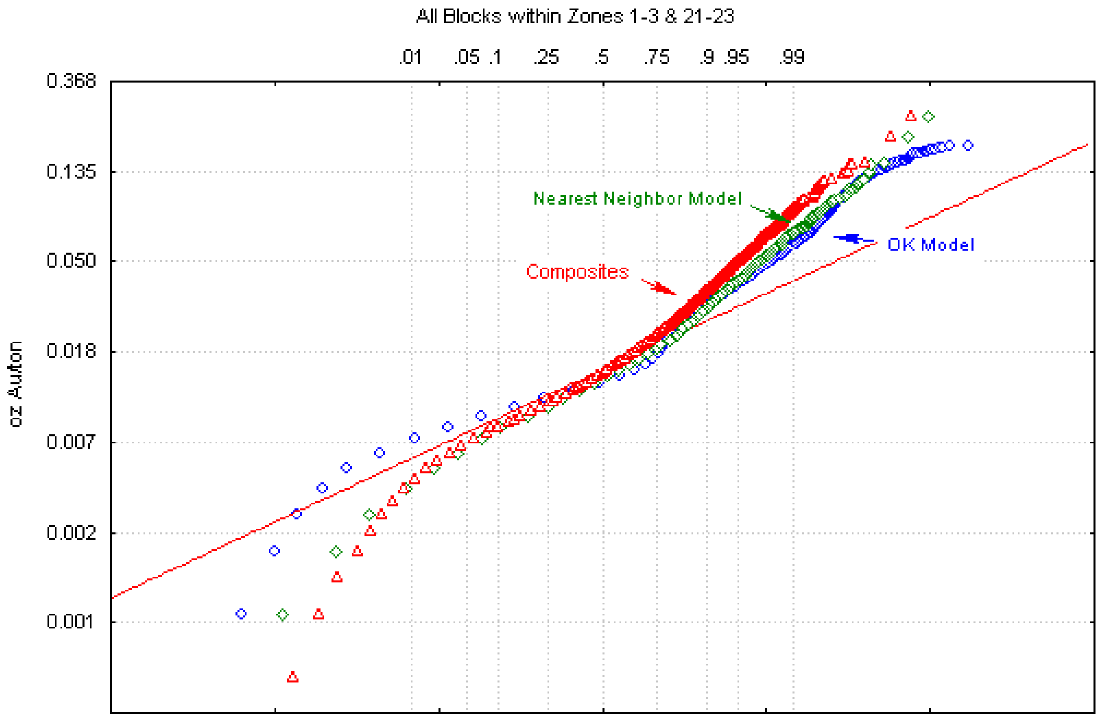
Area	Domain	Indicated (ft)	Inferred (ft)
Hilton Creek / South Zone	1	0 - 200	> 200
	2	0 - 150	> 150
	3	0 - 100	> 100
	99	0 - 100	> 100
Southeast Zone	21	0 - 100	> 100
	22 - 23	0 - 50	> 50
	99	0 - 100	> 100

There is general agreement among geologists who have been involved with the Long Valley project that there is likely a high-angle structural control on higher grade (+ 0.1 oz Au/ton) mineralization. In lieu of hard data to support this, MDA modeled the high-grade zones with the same geometry as the lower grade mineralization. To compensate for this uncertainty, the inclusion of this higher-grade material in the Indicated resource category was more restrictive than if there were better geologic support. With increased geologic knowledge, a portion of the Indicated resources might be placed into the Measured category if additional controls on mineralization were identified.

14.9 Model Checks

A nearest neighbor model of the deposit was completed in 2003 as a check of the kriged 2003 model. The results of the nearest neighbor model compared favorably and there has been no material change with the current estimate. Figure 14-5 compares the distribution of the kriged and nearest neighbor block gold grades and the drill composites for gold domains 1 to 3 and 21 to 23.

Figure 14-5: Distribution of Block Models and Composites



14.10 Long Valley Resource Estimate

A pit was optimized so that resources could be reported based on calculated cutoff grades of the material contained within the optimized pit. The parameters used to generate the optimized pit are shown in Table 14-11:, and the surface projection of the resources, as well as the optimized pit outline are shown in Figure 10-1.

Table 14-11: Pit Optimization Parameters

Item	Units	Parameter
Pit Slope	degrees	45°
Gold Price	$ per ounce gold	$ 1,800
Mining	$/ton mined	$ 1.80
Crushing	$/ton processed	$ 1.40
Heap Leach	$/ton processed	$ 1.80
Sulfide Plant	$/ton processed	$ 10.00
G&A per Ton	$/ton processed	$ 0.70
Refining Cost	$/oz Au Produced	$ 5.00
Recovery (Oxide - Less than 150' below surface)	% Heap Recovery	80%
Recovery (Transition - 150-200' below surface)	% Mill Recovery	90%
Recovery-Plant (Sulfide - more than 200' below surface)	% Mill Recovery	90%

The Long Valley gold resources are tabulated in Table 14-12:. The estimated resources are reported at cutoffs that are reasonable given anticipated mining methods, processing costs, and economic conditions, which fulfills regulatory requirements that a resource exists "in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction."

The material above the cutoff grades contained in the optimized pit at a $1,800 gold price, based on the parameters in Table 14-11:, is reported as the current resource effective September 21, 2020. These resources are summarized in Table 14-12:. Representative block-model cross sections of the Hilton Creek, Southeast, and South zones (locations shown in Figure 7-3) are shown in Figure 14-6, Figure 14-7, and Figure 14-8, respectively, and the resource outline projected to the surface is shown in Figure 10-1 and Figure 14-9.

Table 14-12: Long Valley Resources (Imperial Units)

Material Type	Cutoff (oz Au/ton)	K tons	Indicated oz Au/ton	K ozs Au	Ktons	Inferred oz Au/ton	Kozs Au
Oxide	0.005	35,276	0.018	635	8,997	0.020	180
Transition	0.006	4,026	0.014	56	1,277	0.016	20
Sulfide	0.006	30,914	0.017	526	14,033	0.018	253
Total	variable	70,216	0.017	1,217	24,307	0.019	453

Table 14-13: Long Valley Resources (Metric Units)

Material Type	Cutoff (oz Au/ton)	K tonnes	Indicated g Au/t	K ozs Au	K tonnes	Inferred g Au/t	Kozs Au
Oxide	0.17	32,001	0.62	635	8,162	0.690	180
Transition	0.21	3,653	0.48	56	1,159	0.550	20
Sulfide	0.21	28,045	0.58	526	12,731	0.620	253
Total	variable	63,699	0.58	1,217	22,051	0.650	453

As illustrated in Figure 14-1, the cyanide bottle-roll assays show decreasing gold recoveries with depth, as expected since it was recognized that even in the oxide zone, there were still some areas of remnant sulfide material. But at about 150 feet, the recoveries started decreasing at a faster rate which corresponds to the transition zone and then recoveries fall below 20% at depths greater than 200 feet corresponding to the sulfide zone. The variable cutoff grades used in the reported resources reflect the increased gold processing costs for the transition and sulfide material types.

The relatively high percentage of Indicated resources within the total reported resource results from the close, systematic drill spacing throughout the deposit which has defined relatively continuous, and generally flat-lying, tabular mineralization.

Figure 14-6: Hilton Creek Block Model Section, Looking North

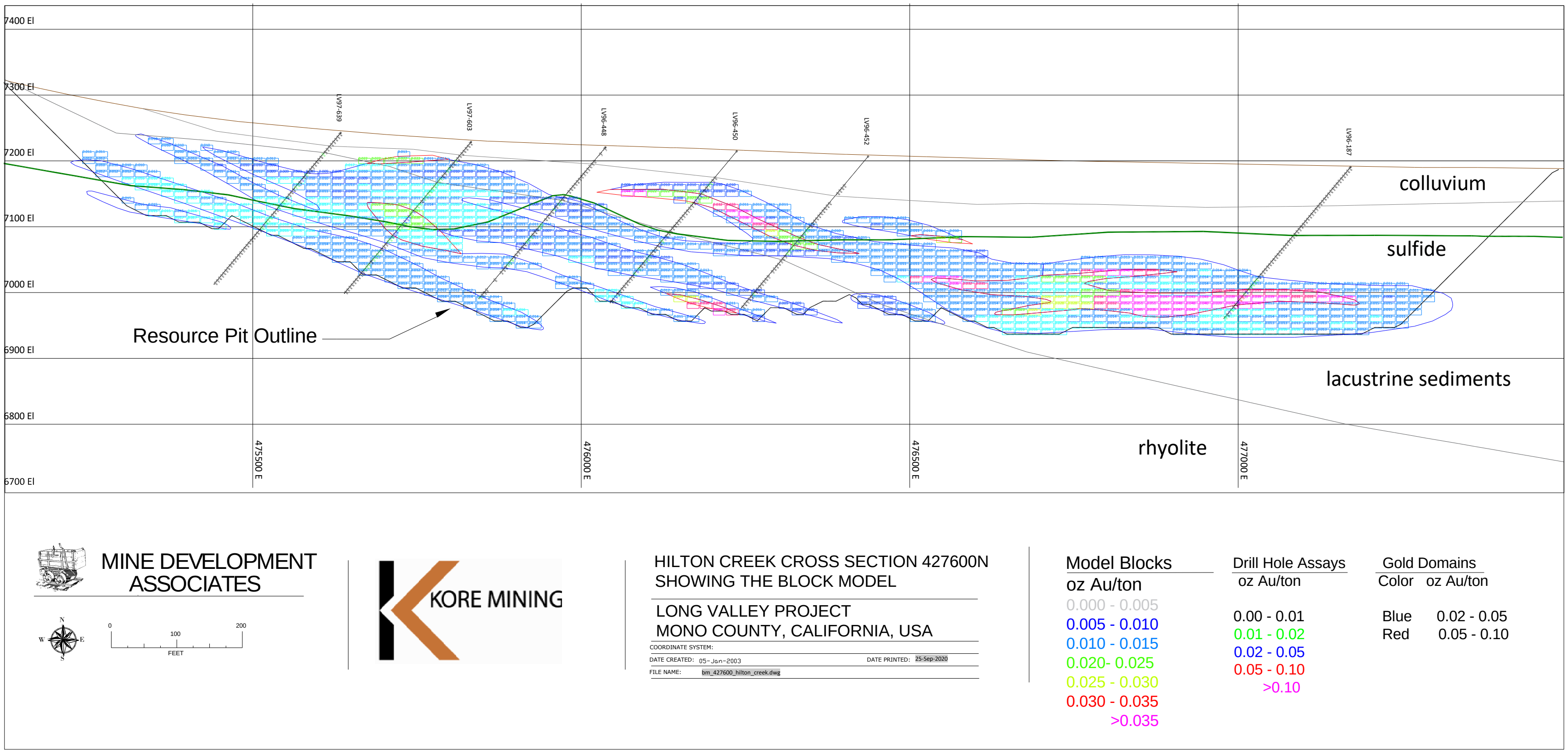


Figure 14-7: Southeast Zone Block Model Cross Section, Looking North

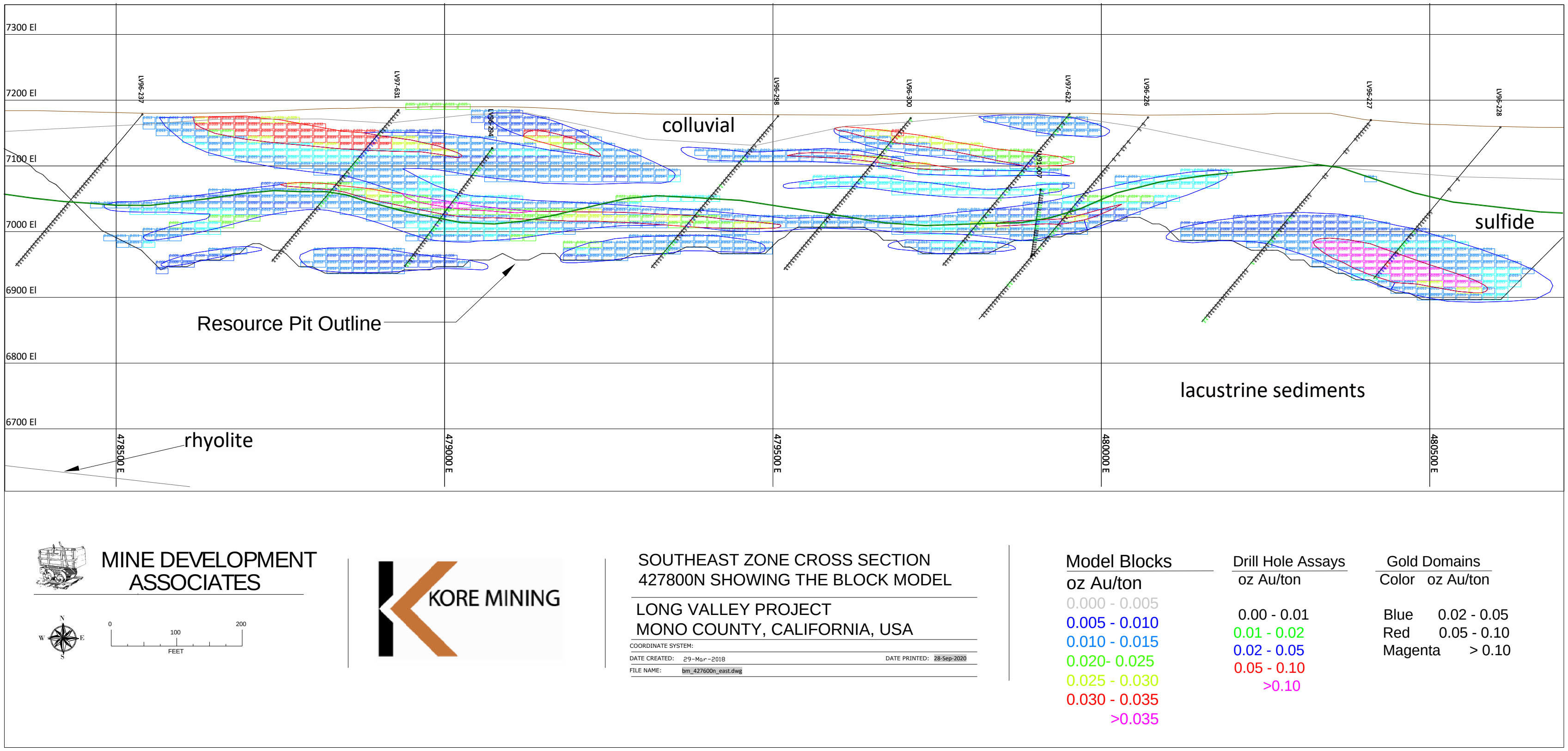


Figure 14-8: South Zone Block Model Cross Section, Looking North

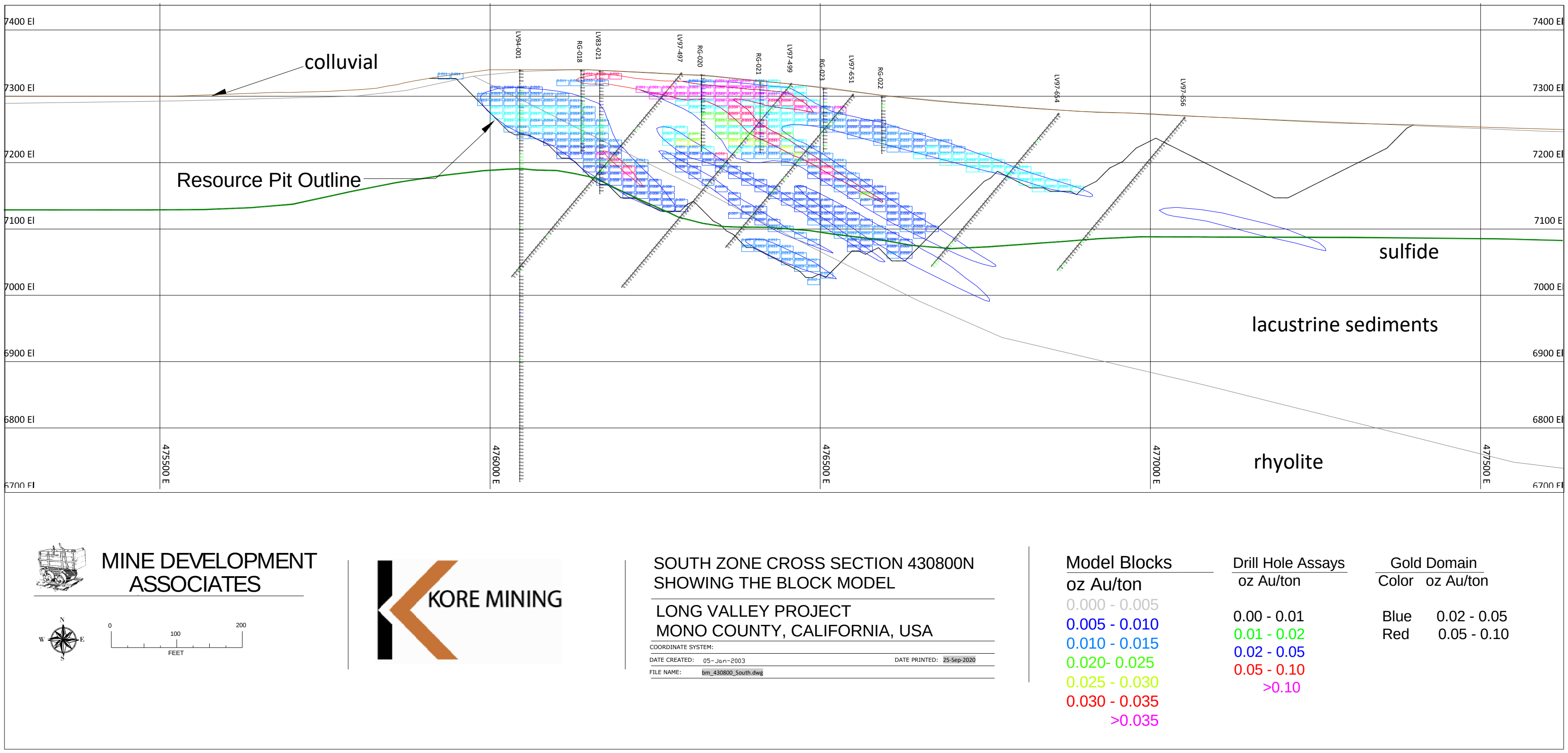
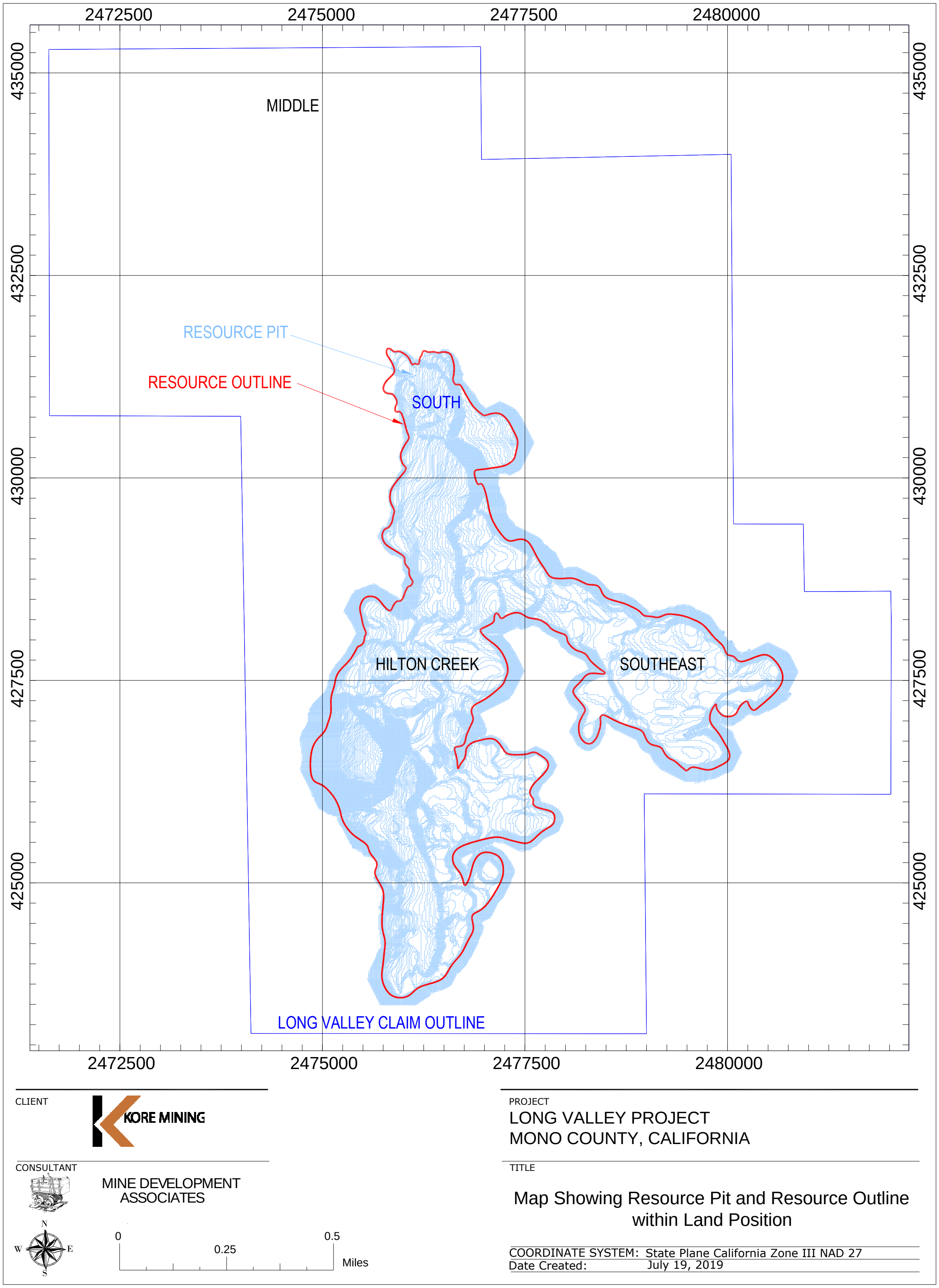


Figure 14-9: Resource Outline with Optimized Pit



The deposit geology is such that almost all mineralization is known to occur within a package of nearly flat-lying caldera-fill lacustrine siltstone and tuff, with no apparent preference to rock type, lying adjacent to, and above, a rhyolite intrusion. Specific controls on the high-grade mineralization are not well understood. However, the close, systematic drill spacing and generally flat-lying, tabular nature of the mineralization, which is known to mimic the general stratigraphic orientation, offset the risk on the resource of not having a more detailed geologic model.

Obtaining mine operating permits for the project may be more difficult than normal due to the project's location in California and proximity to the town of Mammoth Lakes, California, where the predominant source of revenue is derived from tourism. The main anticipated issues relating to the future development of a mining operation at Long Valley would likely be the impact on the current tourism-based economy and particularly the potential visual impacts, impacts to ground water in the area, and the use and containment of cyanide solutions. At this stage of the project these potential impacts have not been quantified.

The Long Valley gold resources are located approximately 1.5 miles north of the Hot Creek fish hatchery operated by the California Department of Wildlife. At this stage of the project, any potential impacts the fish hatchery may have on permitting or development of the project have not been quantified.

The Long Valley property is contained entirely within the late Pleistocene Long Valley caldera, which was formed about 760,000 years ago. Repeated eruption of dacite and rhyodacite from vents on the southwest rim of the caldera 220,000 to 50,000 years ago formed Mammoth Mountain, a dome complex. The USGS monitors the area for volcanic activity and does not have an advisory or watch alert level for the caldera. The authors believe that this is a low-level risk in the short period of time needed to develop and mine the project.

15.0 MINERAL RESERVE ESTIMATION

“Mineral Reserves” differ from “Mineral Resources” in that Mineral Reserves are known to be economically feasible for extraction. The CIM Definition Standards require the completion of a Preliminary Feasibility Study (PFS) as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. At this time, a PFS has not been completed for the Long Valley project. Therefore, reserve estimates have not been made.

16.0 MINING METHOD

The Long Valley Project is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the estimated grade of the resource model (provided by MDA; see Table 16-1 for block model location and size) and Whittle pit shell analysis. The economic mineralized material and waste will be drilled and blasted using a rotary blasthole drill and ammonium nitrate fuel oil (ANFO) and transported in dump trucks. The mine plan calls for the leachable material from the pits to the heap leach pad at a rate of 22,000 short tons per day. The mine plan includes concurrent backfilling and closure within +/- 25 feet of original topography.

Table 16-1: Block Model Dimensions

Dimension	X Direction	Y Direction	Z Direction
Minimum Corner	474,500	423,500	6,495
Block Size	20	20	10
Number of Blocks	325	410	104

16.1 Pit Design

For the Long Valley project, average mining cost and G&A cost from similar deposits were used as a basis for Whittle costs. Processing costs used in the Whittle pit analysis were specifically evaluated for the Long Valley deposit. Whittle inputs are shown in Table 16-2. Additional Whittle constraints include limiting the pits to the permitted area.

Table 16-2: Whittle Inputs

Input	Unit	Value
Mining Cost	$/ton	1.60
Processing Cost	$/ton	3.85
G&A Cost	$/ton	0.81
Oxide Recovery	%	80
Transition Recovery	%	60
Sulfide Recovery	%	20
Pit Slope	degrees	45
Gold Price	$/tr. oz.	1650
Selling Cost	$/tr. oz.	5

Initial analysis generated 86 pit shells by revenue factors from 0.3 to 2.0 by increments of 0.02. Figure 16-1 is a graph of pit shell economic mineralized material and waste tons and Net Present Value (NPV) at a discount rate of 10%. The pit at $1650/tr. oz. gold is called out on the graph. Ms. Lane of GRE used the $1,650 Whittle pit shell to design an ultimate pit, due to the high NPV and the desired production rate of 100,000 oz of gold per year and contained amount of gold greater than one million troy ounces.

Since backfilling waste has a large impact on pit design and scheduling, whittle pit shells were evaluated on a directional basis rather than a nested pit shell basis for a more accurate estimation of scheduling. Ms. Lane of GRE examined the effect of mining the pit shell in four phases from South to North, and from North to South, to determine which direction would provide a better value. The $1650 pit shell analysis with pushbacks going from South to North yielded an NPV with an additional $10 million over the North to South direction.

Within the 4 phased South to North pits, pit shells for production scheduling were generated by increasing the revenue factor of 0.1 to 1.0 by increments of 0.1. These shells are used to generate the production schedule. The ultimate pit (Figure 16-2) consists of two separate pits, a larger pit and smaller satellite pit. The primary pit is divided into 3 phases with similar quantity of material, and the smaller satellite pit is considered as a separate 4th phase (Figure 16-3).

Figure 16-1: Whittle Pit Shells – Tonnage and NPV

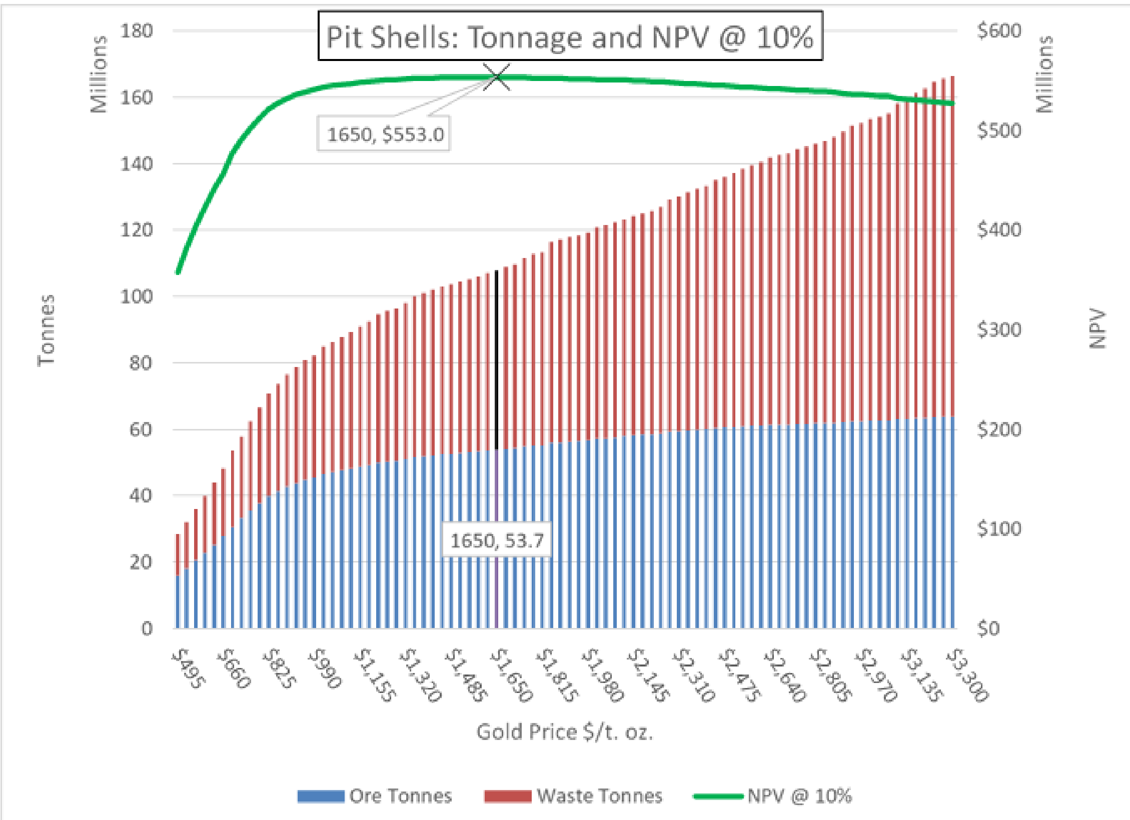
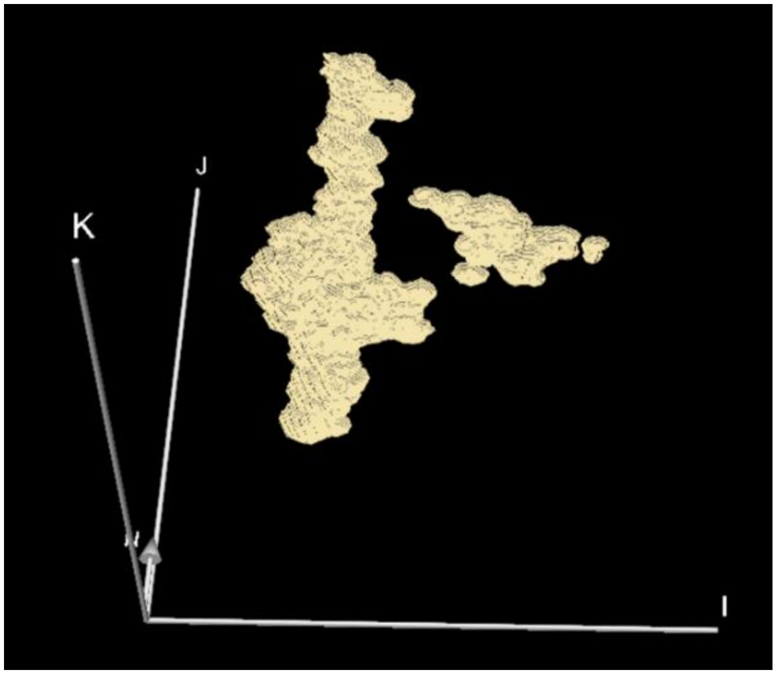
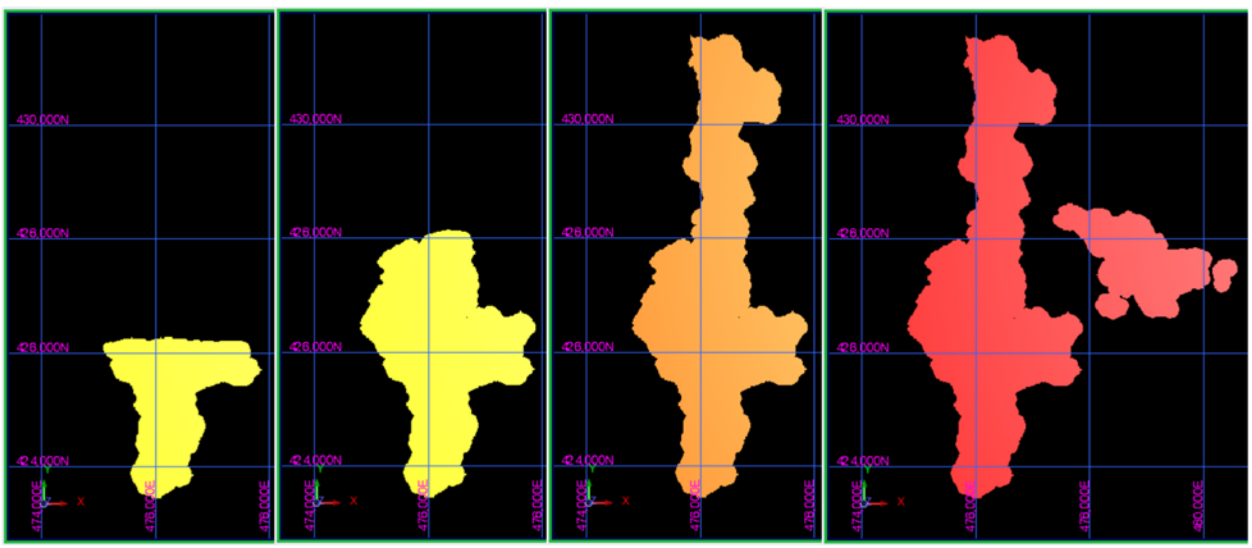


Figure 16-2: Ultimate pit



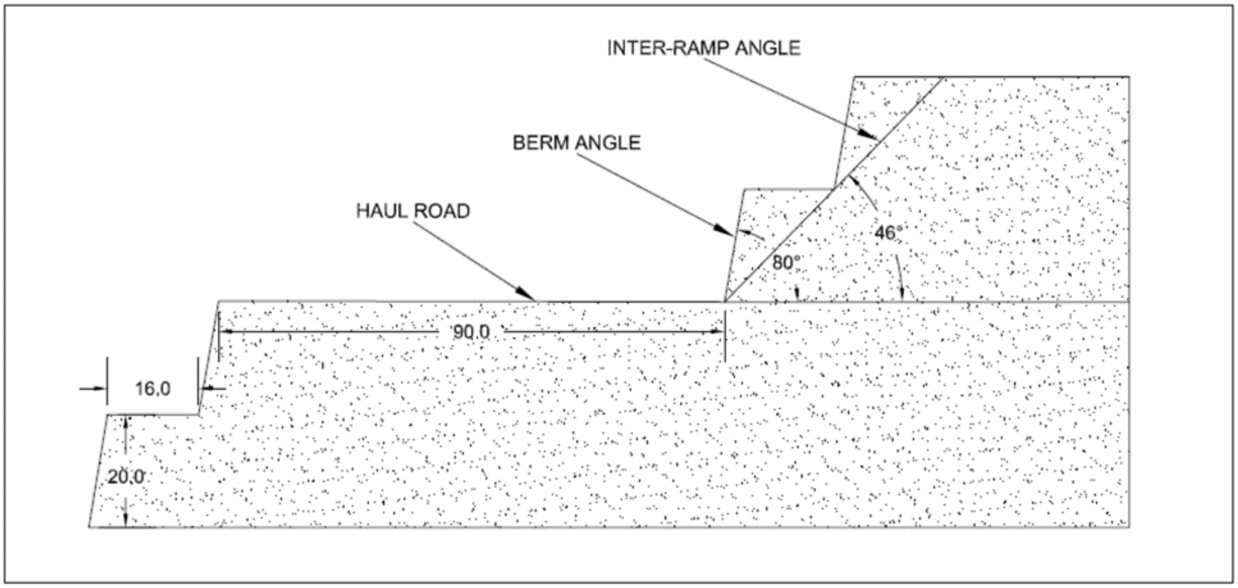
Whittle visualization

Figure 16-3: Phases 1 Through 4 ($1,650/tr. oz.)



Within the phases, pits were designed with an overall pit slope of 45°, using a batter angle of 80°, bench height of 20 feet (doubled 10 feet benches). Catch berms are 16 feet wide. All in-pit haul roads were designed with a maximum 10% grade and a width of 90 feet (Figure 16-4).

Figure 16-4: Pit Slope Profile



16.2 Pit Resources

Ms. Lane of GRE selected the cutoff grade of 0.004 oz/t for oxide material, 0.005 oz/t for transition material and 0.017 oz/t for sulfide material. Figure 16-5 to Figure 16-8 shows the pit bottom within the phases, and the resources within the phases are shown in the Table 16-3.

Figure 16-5: Phase 1

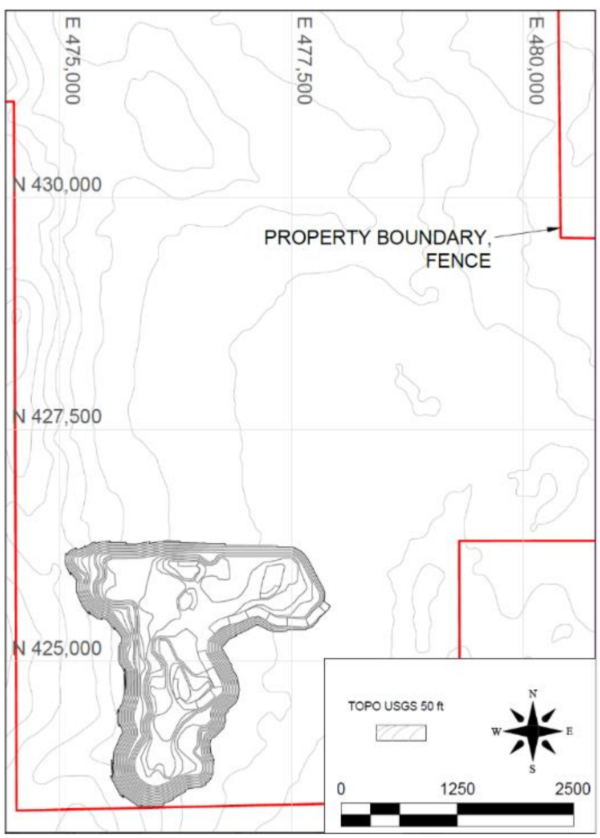


Figure 16-6: Phase 2

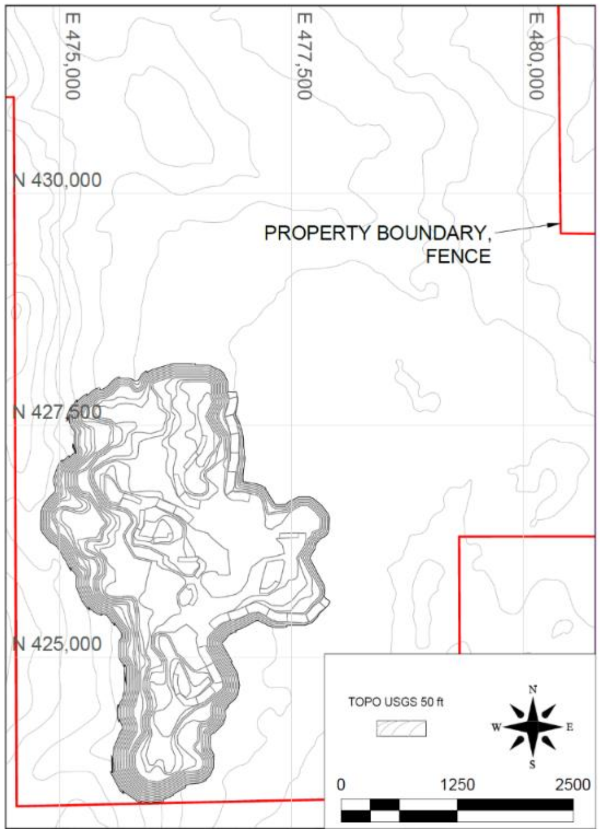


Figure 16-7: Phase 3

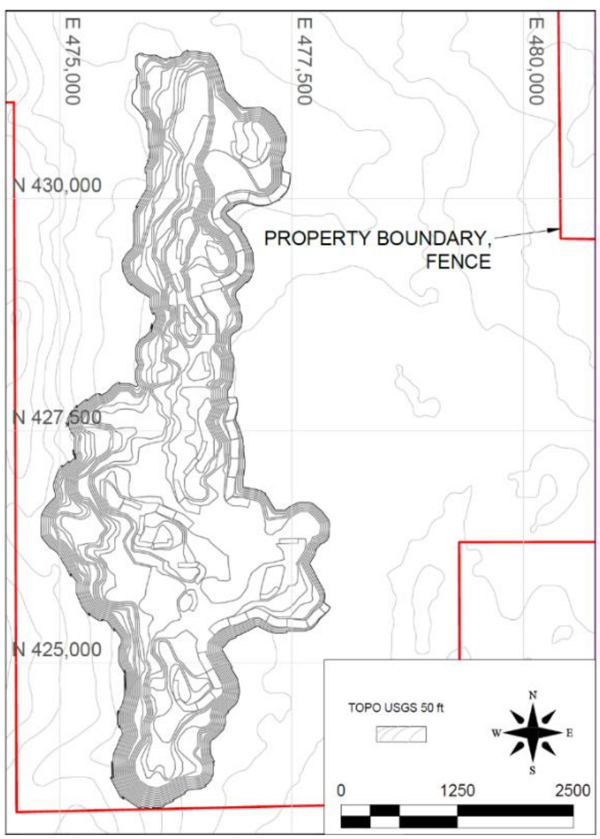


Figure 16-8: Phase 4

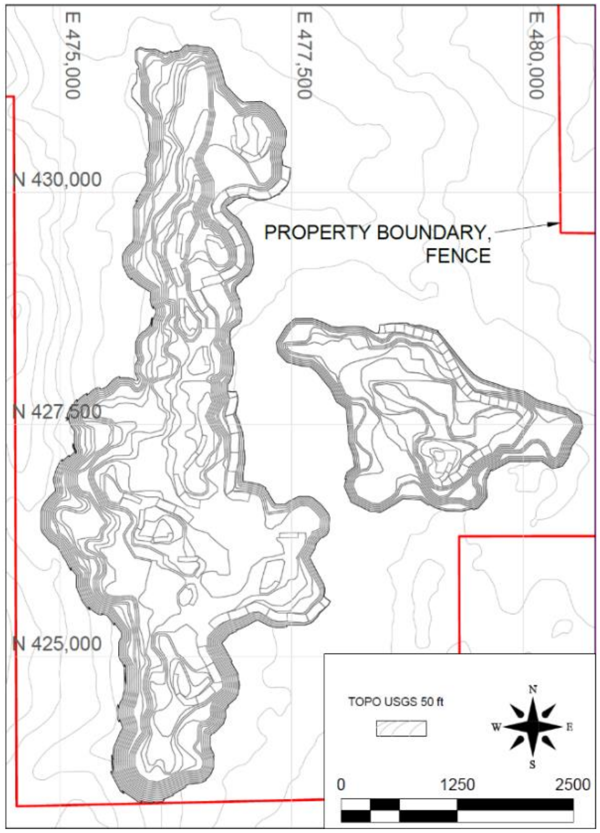


Table 16-3: Pit Resource

Phase	Leachable Material – Indicated (1000s Tons)			Leachable Material – Inferred (1000s Tons)			Waste	Au – Indicated (1000s oz)			Au – Inferred (1000s oz)			Stripping Ratio
	Oxides	Transition	Sulfides	Oxides	Transition	Sulfides		Oxides	Transition	Sulfides	Oxides	Transition	Sulfides	
Phase 1	13,122	1,581	1,178	1,041	362	251	18,675	247	23	42	28	6	12	1.07
Phase 2	10,500	1,041	1,196	1,016	42	199	19,497	181	15	42	21	1	7	1.39
Phase 3	8,345	713	1,514	1,874	368	436	19,296	150	10	58	47	6	21	1.46
Phase 4	3,005	577	157	4,879	433	326	19,033	39	7	3	79	6	8	2.03
Total	**34,972**	**3,913**	**4,045**	**8,810**	**1,205**	**1,211**	**76,501**	**618**	**55**	**145**	**175**	**19**	**47**	**1.41**

16.3 Mine Scheduling

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

- Leachable Material Production Rate: 22,000 tons per day (tpd)
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 50

- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 12

The resources were reported by 10-foot bench, showing tons of leachable material, waste, and ounces of gold. Pre-stripping of waste was included if waste occurred on a bench that had no corresponding leachable material or if the tonnage of waste on a bench exceeded five times the tonnage of leachable material on that bench. The production rate for pre-strip benches was set to three and a half times the leach material production rate, or 77,000 tpd. Leachable material mined along with pre-stripped waste was placed into a stockpile for later processing.

For all the benches, all waste on a bench were scheduled to be mined over the same duration as the leachable material on that bench. This scheduling method resulted in inconsistent leach material quantity mined. Ms. Lane of GRE used pre-stripping and phasing, to have similar potentially economic mineralized material production throughout the mine life.

The mining and processing rate is ramped up in year 1. The first quarter production rate is 11,000 tpd, second quarter production rate is 16,000 tpd and after that the production is 22,000 tpd for the mine life.

The mining schedule is summarized below in Table 16-4. Figure 16-9 to Figure 16-17 show the mine plan.

Table 16-4: Mine Schedule Summary (1000s)

Material		Year								Total
		-1	1	2	3	4	5	6	7	
Leachable Tons - Indicated (1000s)	Oxides	96	5,914	6,762	4,820	6,287	5,063	3,481	2,549	34,972
	Transition	0	3	263	1,353	706	446	565	577	3,913
	Sulfides	0	0	136	1,045	327	892	1,488	157	4,045
Leachable Tons - Inferred (1000s)	Oxides	23	460	755	324	619	1,426	1,828	3,376	8,810
	Transition	0	3	105	259	24	57	323	433	1,205
	Sulfides	0	0	9	242	68	143	423	326	1,211
Leachable Tons Total (1000s)		**118**	**6,381**	**8,030**	**8,043**	**8,030**	**8,028**	**8,108**	**7,417**	**54,156**
Contained Au oz - Indicated (1000s)	Oxides	2	110	129	76	119	88	61	33	618
	Transition	0	0	4	19	10	6	8	7	55
	Sulfides	0	0	4	38	9	34	57	3	145
Contained Au oz - Inferred (1000s)	Oxides	0	13	18	8	15	31	40	51	175
	Transition	0	0	2	4	1	1	5	6	19
	Sulfides	0	0	0	12	2	6	20	8	47
Contained Au oz Total (1000s)		**2**	**122**	**157**	**158**	**155**	**167**	**191**	**108**	**1,060**
Waste Tons (1000s)		**1,168**	**11,596**	**5,561**	**11,839**	**7,236**	**14,763**	**14,052**	**10,286**	**76,501**
Stripping ratio		**9.86**	**1.82**	**0.69**	**1.47**	**0.90**	**1.84**	**1.73**	**1.39**	**1.41**

Figure 16-9: Mine Plan, Year -1

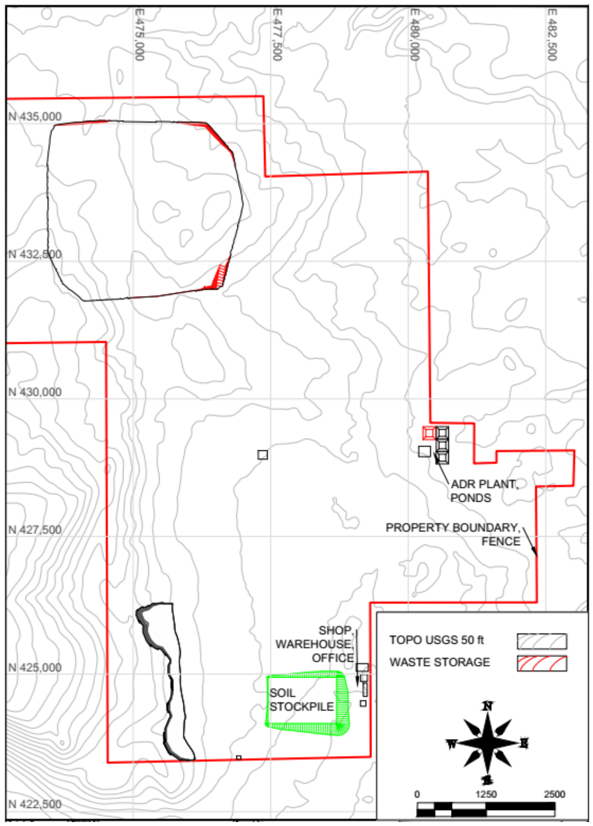


Figure 16-10: Mine Plan, Year 1

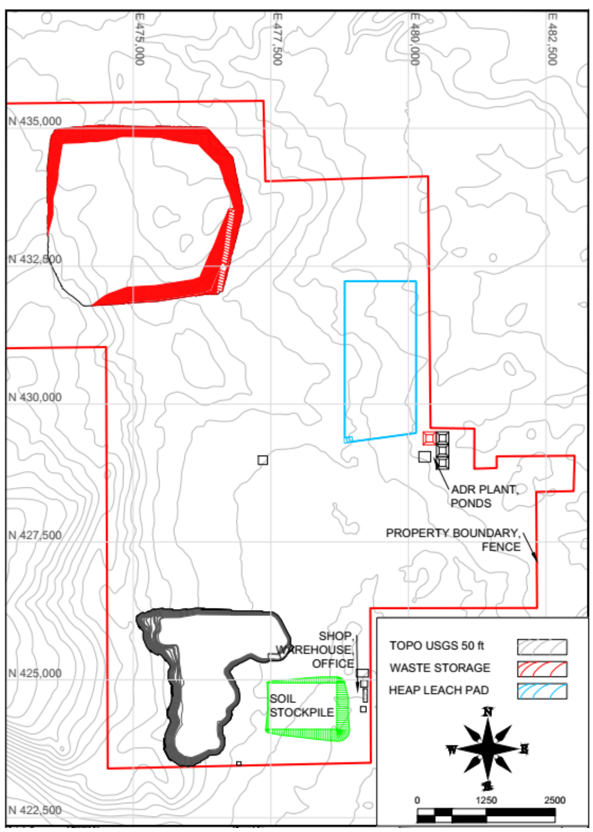


Figure 16-11: Mine Plan, Year 2

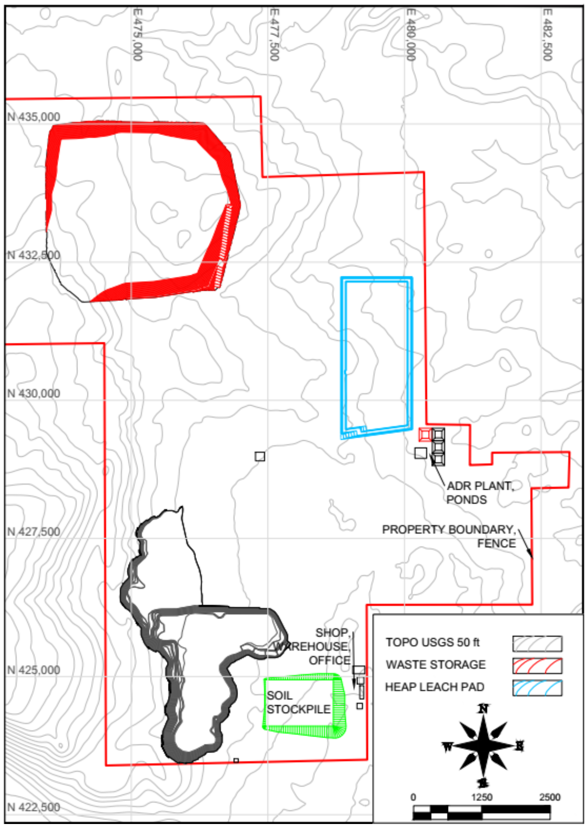


Figure 16-12: Mine Plan, Year 3

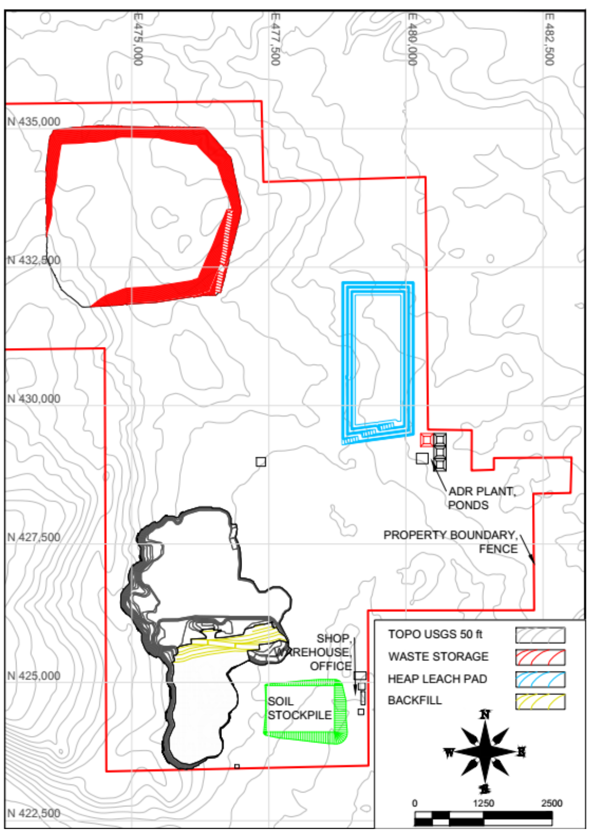


Figure 16-13: Mine Plan, Year 4

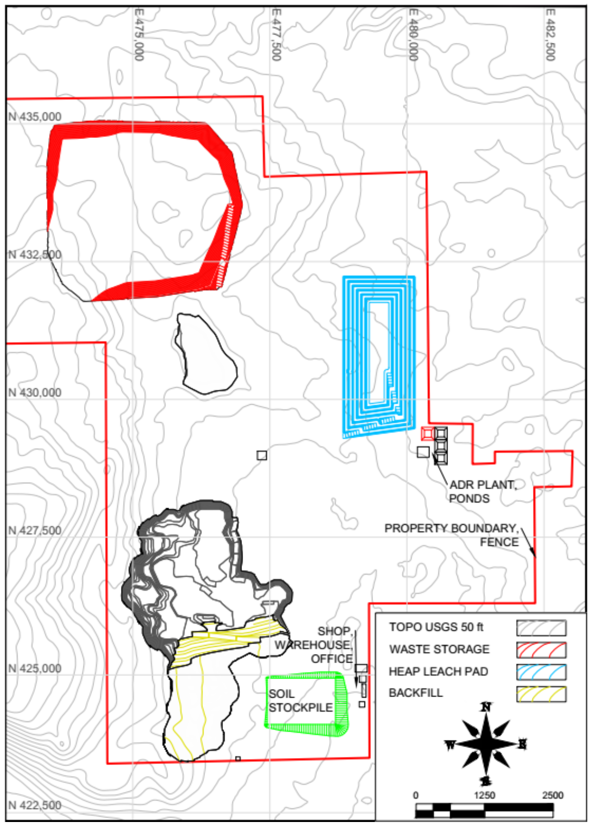


Figure 16-14: Mine Plan, Year 5

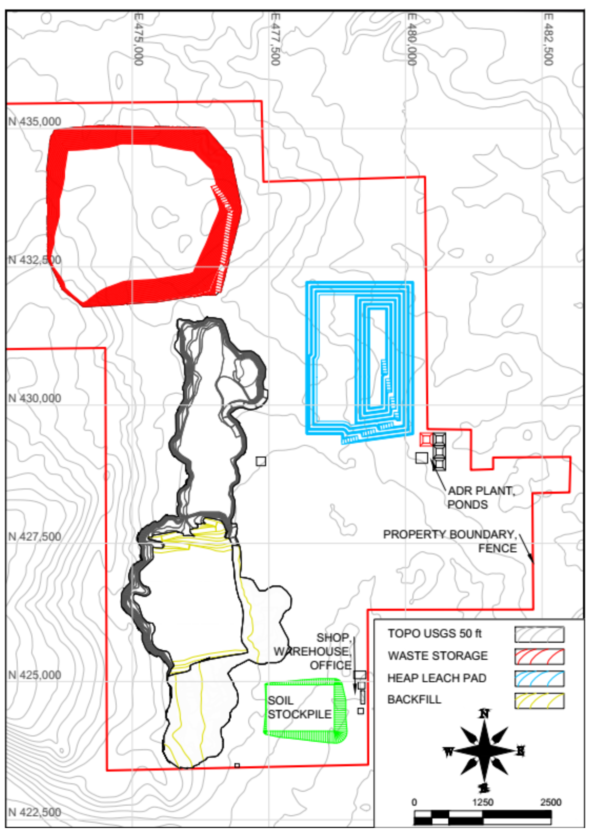


Figure 16-15: Mine Plan, Year 6

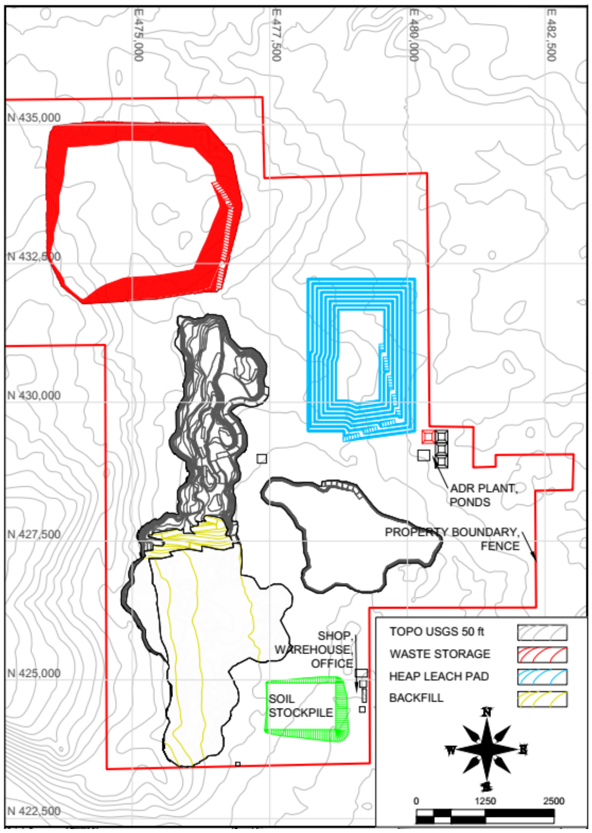


Figure 16-16: Mine Plan, Year 7

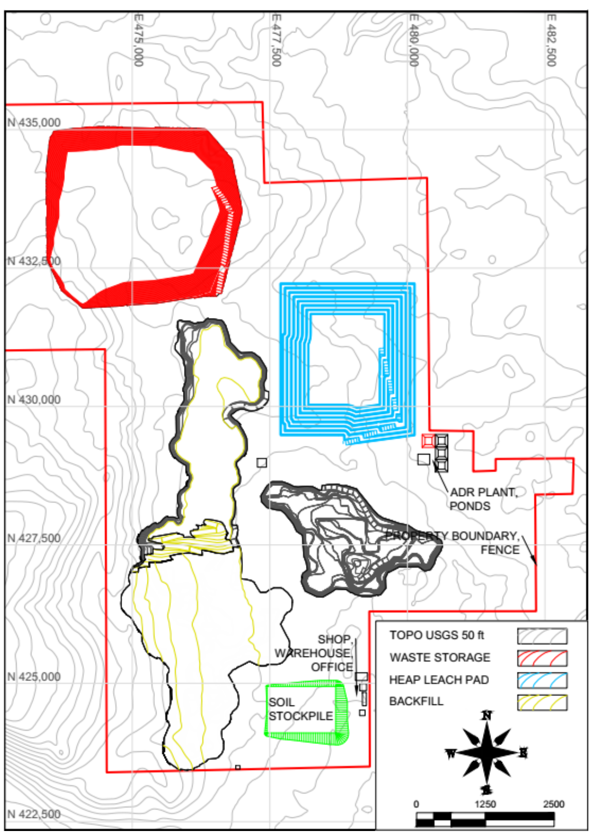
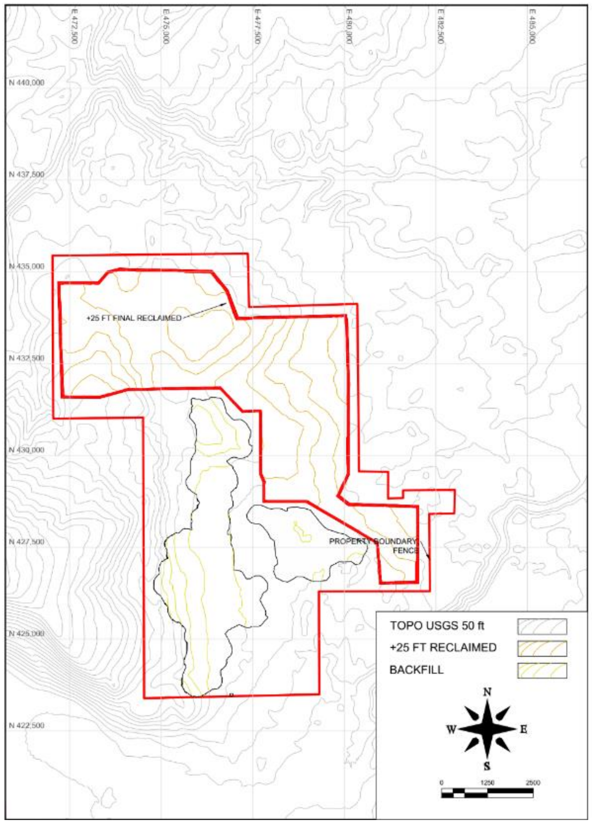


Figure 16-17: Long Valley Post Reclamation

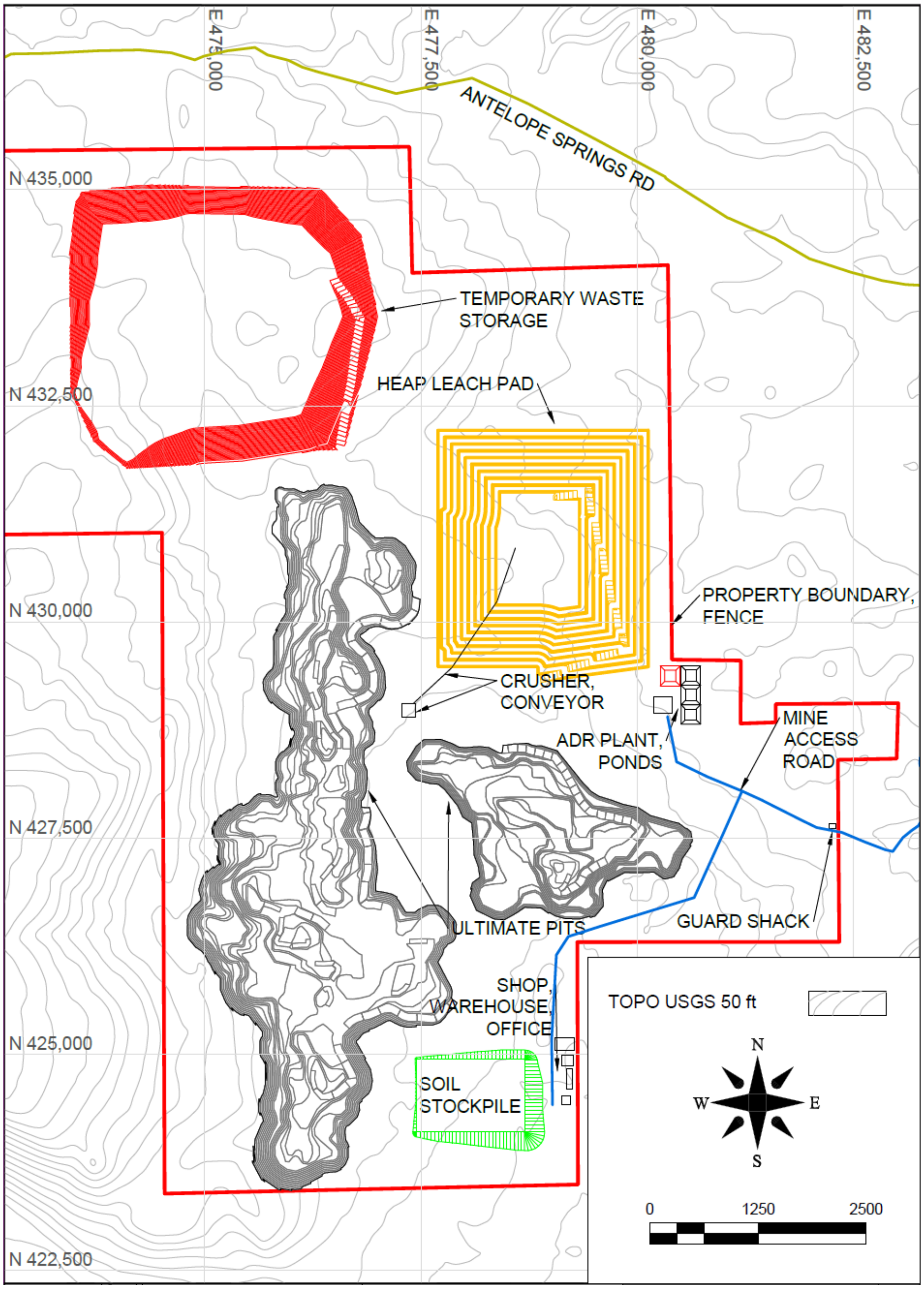


16.4 Mine Operation and Layout

Facilities for crushing, the leach pad, ADR plant, administrative offices, warehouse, and other facilities are present at the site for the long valley project. The leach pad will need to be expanded as the project progresses.

Ms. Lane of GRE developed conceptual layouts for the project, including waste dump locations and sizes, leach pad location and size, and access road routes. Figure 16-18 illustrates the conceptual long valley project layout with pits, pads, and dumps.

Figure 16-18: Long Valley Conceptual Site Layout



16.5 Mine Equipment Productivity

Ms. Lane of GRE selected equipment sizes based on the estimated cycle times and productivity required to meet the project schedule. A simplified approach for cycle time calculations was used to determine the productivity of the machines. The cycle analysis considers productivity variables such as average daily production of economic mineralized material and waste, hours per shift, shifts per day, availability, breaks, and shovel capacities. Haulage cycles were used by calculating haul distance and travel speed on a variable gradient to estimate hourly production rates. Optimum fleet size for the shovel and the haul trucks is then selected to meet the production requirements.

Table 16-5: Quantities of Major, Support, and Minor Equipment Needed for Life of Mine

Item	Quantity
Major Equipment	
Excavator CAT 6040	2
Haul Truck CAT 785D	7
Bulldozer D10	3
Drill	2
Support Equipment	
Loader CAT 992K	1
Water Truck	2
ANFO Truck	1
Lube Truck	2
Mechanics Truck	2
Grader	1
Minor Equipment	
Small Excavator	1
Backhoe	1
Small Crane	1
Light Plant	6
Dewatering Pump	1
4x4 Pickup	10

17.0 RECOVERY METHODS

17.1 Process Description

The Long Valley project would employ open pit mining with a conventional heap leach system on a 365 day per year 24 hour per day basis. The heap leach will utilize crushed run-of-mine (ROM) material at a P_{80} of 1½ inches (37.5 mm). The crushed material will be agglomerated with cement and transported to the heap leach via conveyor belt.

The heap leach would consist of a suitable area lined with a containment system, typically a linear low-density polyethylene (LLDPE) liner with an over liner of sized material to facilitate drainage and to protect the liner during initial stacking. Within this over liner would be placed drainage pipes to conduct the leach solution to the centralized collection ponds. The crushed material is stacked in lifts on the lined pad by a radial stacker. The stacker would be fed by a series of jump or grasshopper conveyors that would be fed from the main overland conveyor from the agglomeration. The lifts are targeted at 32 feet (10 meters) in height with a total heap height of 328 feet (100 meters). Once a suitable area has been stacked (cell), the cell would be irrigated with dilute cyanide solution. Stacking would continue to advance, and each area irrigated with cyanide solution for a set period (primary leach cycle). The solution leaches gold and silver from the heap materials and is transported to the recovery circuit as PLS.

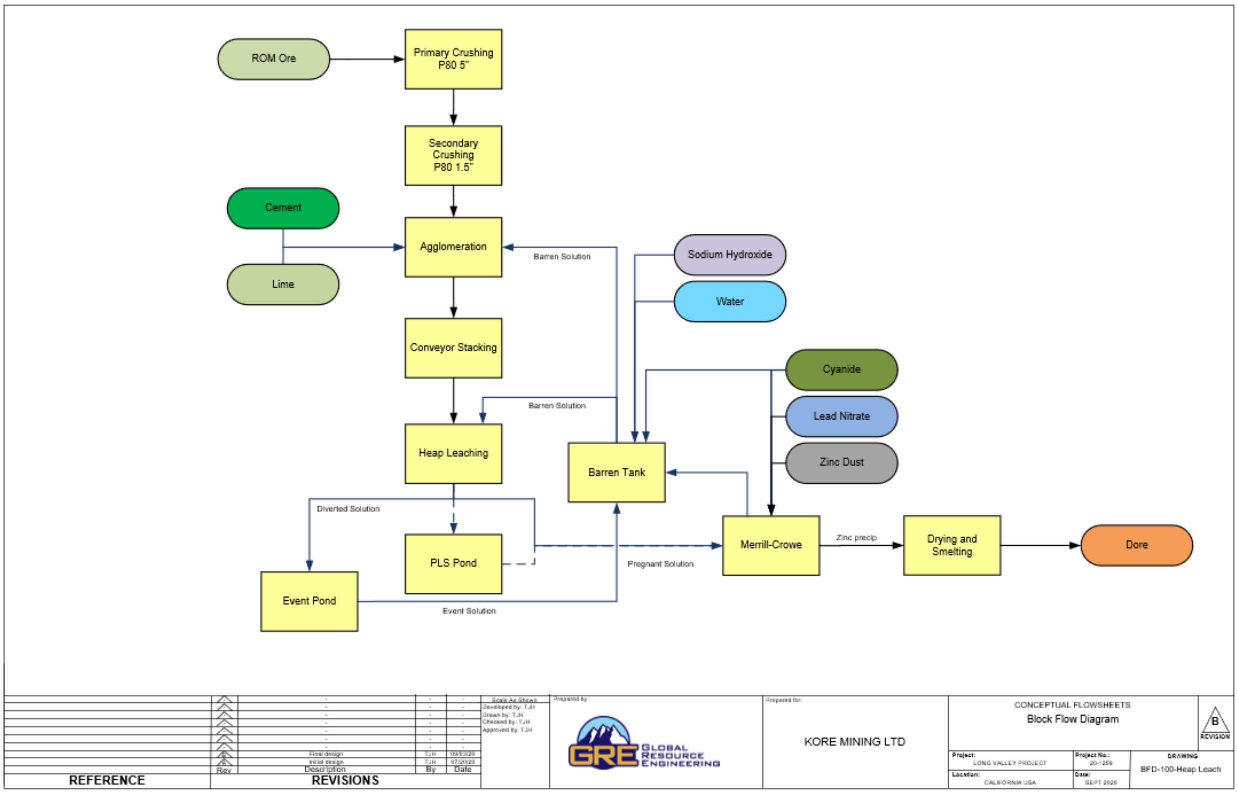
This PLS would be processed directly in the Recovery plant, diverted to a dedicated pond, or recirculated to the heap. The Recovery plant is to utilize the Merrill-Crowe system for precious metal recovery as it is predicted that the PLS will contain appreciable silver along with gold. The anticipated silver to gold concentration ratio is 3.9 to 1.

The PLS first undergoes clarification to remove any suspended solids prior to being pumped to the Crowe tower for de-aeration. The de-aerated solution then mixes with fine zinc powder, dissolved lead nitrate and additional cyanide. The zinc powder acts to precipitate the precious metals which are then captured in a downstream filter press. The depleted “barren” solution would report to the heap leach barren tank and be recirculated back to the heap after having the reagent levels adjusted (pH and cyanide) to the required set points.

Once the filter press is fully loaded with precipitate, the precipitate is removed and dried prior to being smelted for precious metal recovery. A gold and silver doré bar will be produced for sale to an offsite refinery.

The heap leach is typically designed to have multiple lifts installed. Each new lift goes on top of the last lift until the heap reaches its ultimate height. Heap leaches often utilize 10 or more lifts to reach an ultimate height of 328 feet to 492 feet (100 to 150 meters). The configuration of the heap leach is heavily dependent on the permeability characteristics of the material, the terrain available, and the geotechnical aspects of the site. Figure 17-1 shows the conceptual flowsheet.

Figure 17-1: Conceptual Heap Leach Flowsheet



17.2 Crushing Circuit

The crusher is designed to process approximately 26,400 tpd (23,900 metric tonnes per day) on a 24-hour basis with an availability of 80%. The design crushing rate is 1,100 short tons per hour (997 metric tonnes per hour).

The run of mine feed passes over a vibrating grizzly with a 3-inch (75-mm) opening. The undersize reports directly to the jaw crusher discharge conveyor while the oversize feeds the jaw crusher. The jaw crusher would crush to a nominal 7-inch (175-mm), with the crushed product reporting by conveyor to screen feed bin. A vibrating feeder beneath the feed bin feeds a double deck screen equipped with a top deck with 5-inch (125 mm) openings and the lower deck with 1.5-inch (37.5 mm) openings. The screen undersize reports to the final product conveyor and the screen oversize is split into two streams and feeds two standard cone 4-foot crushers with a closed side setting of 1.2 inches (30 mm). The discharge from the crushers falls onto the final product conveyor. The secondary crushing circuit is operated in open circuit. This crushing circuit would be capable of achieving a P_{80} of 1½-inch.

17.2.1 Agglomeration

The final crushed product is conveyed to a rotary drum agglomerator. Barren cyanide solution, and cement/lime would be added to the material prior to mixing. The target is to deliver approximately 50% of the total cyanide demand to the economic mineralized material while not exceeding 8 to 10% moisture by weight. The agglomerated material would be conveyed via a combination of overland and grasshopper conveyors to a prepared permanent leach pad.

The mineralized material is stacked using a radial stacker to lift heights of 32 feet (ten meters). Stacking would be conducted in retreat mode during the creation of each leach cell. The agglomerated mixture would be allowed to cure for several days prior to solution application.

17.3 Heap Leach Circuit

Mineralized material would be stacked for a sufficient period to allow enough surface area to be created for irrigation, this also allows operations personnel to be a safe distance from active irrigation areas. Irrigation is provided by an emitter-type irrigation system designed to deliver 0.005 gpm/ft^2 (12 lph/m^2). The emitter layout is designed to provide suitable mineralized material wetting. The heap would be placed under primary irrigation for a period of approximately 90 days. After the primary leach, irrigation would be discontinued and advanced to the next cell. No rinse phase is included because of the multiple lift system employed. Subsequent lifts will be placed on top of the previous lift, up to a total of 10 lifts. Rinsing will be conducted as part of the final closure.

The precious metal leach solution or PLS flows from the heap leach pad to the PLS sump by gravity. The solution is pumped from the sump to the recovery circuit. Excess solution is diverted to the PLS pond. Solution is collected from each heap cell by a series of drainpipes under the heap that transport the solution to perimeter piping. The solution can be directed to either the PLS or Event Pond piping. Storm water collected from the pad during heavy precipitation events can be diverted to an event pond. The storm water can be used as fresh make up water to the circuit.

17.4 Recovery Plant – Merrill-Crowe

Dr. Harvey of GRE has included a Merrill-Crowe plant for recovery of gold and silver from the pregnant solution due to the potentially high silver solution grades. An Adsorption-Desorption Recovery (ADR) plant would be preferred but further test work is required to validate the solution tenors. The capital costs are nearly identical for the two.

The PLS solution reports to a series of pressure leaf clarifiers to remove the suspended solids. Suspended solids not only blind the zinc dust cake and filter media within the precipitation filter presses but can slow down the zinc precipitation through passivation on the metallic zinc surface. The clarification filter is coated with diatomaceous earth as required. The highly porous diatomaceous earth filters the suspended solids from the solution. The suspended solids concentration after clarification will typically be less than 5 ppm. Cleaning the clarifiers will be done after the clarifier is taken offline from the process solution stream. The clarifiers are cleaned by backwashing with water. The diatomaceous earth and removed solids will be discharge to a purpose-built pond. This pond will need period excavation.

After passing through the clarifiers, the solution will be fed to the de-aeration tower (Crowe tower), where a negative pressure generated by a vacuum pump removes dissolved oxygen from the solution. The presence of dissolved oxygen slows down the reaction with the metallic zinc and increases the precious metals content in the barren solution due to re-dissolution of precious metals. The dissolved oxygen concentration of the de-aeration tower is targeted for less than 1 ppm.

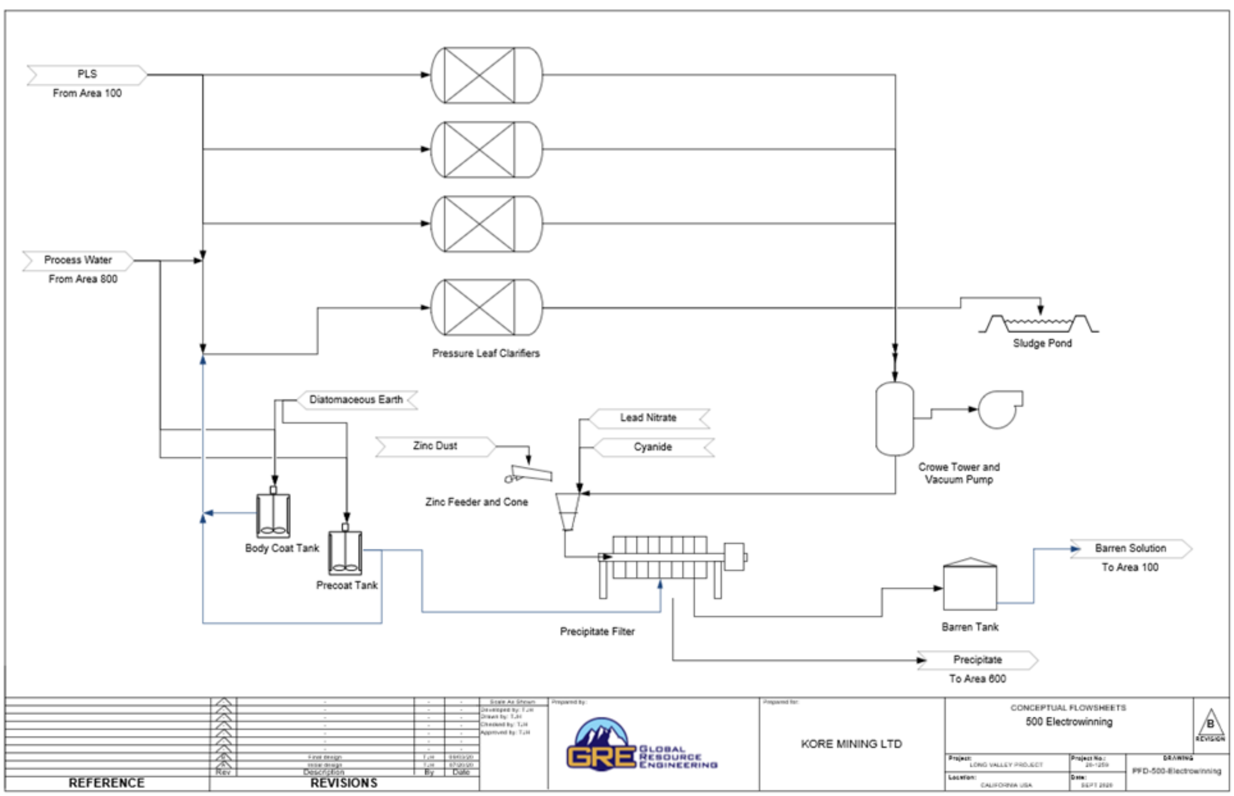
Zinc is used to precipitate the gold and silver from the cyanide solution. Zinc is less noble than gold and silver and gives exchanges electrons to these metals along with copper and other metals. This reaction reduces the gold and silver to their native states. The gold and silver under reduction while the zinc is oxidized and becomes soluble. Zinc is fed into the solution after de-aeration to prevent oxidation and passivation of the zinc surface. Zinc is fed at a specific rate into solution to precipitate the desired metals. Lead nitrate can also be added at a rate of approximately 10% of the zinc rate. The lead increases the zinc reactivity and inhibits the formation of zinc hydroxide on the zinc surface. Typically, zinc is added in excess of the stoichiometric quantity depending upon the solution grade (4x). Lower grade solutions require a higher proportion of zinc addition. Additional cyanide is also typically added to ensure the correct precipitation reactions.

Gold and silver precipitates will be collected within a recessed plate and frame precipitate filter press (2x). The discharge solution from the filter press is the barren solution. The precipitate filters will be emptied on a weekly basis. Prior to opening, the filter will be purged with low pressure compressed air to remove the excess solution and partially dry the filter cake precipitate. The precipitate will be collected from the press and dried prior to refining.

Refining is a multistep process. The cake will first be dried, followed by retorting to remove any contained mercury. After completion of retorting, the cake will be mixed with appropriate fluxes and smelted in an electric melting furnace. Doré (a mixture of gold and silver) will be stored in a vault until shipment to a third-party refinery.

A schematic of a Merrill-Crowe system is provided in Figure 17-2.

Figure 17-2: Merrill-Crowe System Schematic



17.5 Conceptual Heap Leach Pad and Pond Design

The heap leach facility (HLF) consists of the following system components:

- Heap leach pad
- Liner system
- Leachate (solution) collection system
- Storm pond
- Stormwater management system
- Freshwater supply

To minimize capital expenditure, the heap leach pad has been designed in phases, with each phase requiring advanced expansion of the engineered pad. The HLF would be constructed in three phases, with the pad foundation preparation, liner installation, and collection piping advanced as the leach pad expands. The capacity of each stacking stage includes an initial three-year period two additional two-year period.

The initial HLF development (Phase 1) would also include the full development of the solution handling system, storm pond, and perimeter diversion ditches prior to commencing mineralized material stacking and leaching. Table 17-1 shows the development phases and the lift capacity in mineralized material volume and duration. Design details for each of the HLF components are discussed further in the following sections.

Table 17-1: Heap Capacity

Development Phase	Elevation (abs m)	Lift Capacity (days)	Mine Life (years)	Mineralized Material Volume	
				(m^3)	(cum m^3)
1	10	244	0.7	3,223,805	3,223,805
	20	445	1.2	2,659,900	5,883,705
	30	608	1.7	2,142,474	8,026,179
	40	734	2.0	1,671,520	9,697,699
	50	829	2.3	1,247,025	10,944,724
	60	894	2.5	868,961	11,813,684
	70	935	2.6	537,239	12,350,924
	80	954	2.6	251,436	12,602,360
2	10	1080	3.0	1,657,887	14,260,247
	20	1198	3.3	1,561,485	15,821,732
	30	1309	3.6	1,465,092	17,286,824
	40	1412	3.9	1,368,714	18,655,537
	50	1509	4.1	1,272,362	19,927,900
	60	1598	4.4	1,176,066	21,103,966
	70	1680	4.6	1,079,911	22,183,877
	80	1754	4.8	984,319	23,168,196
	90	1822	5.0	900,835	24,069,031
	100	1869	5.1	612,377	24,681,408
3	10	1916	5.3	1,243,362	25,312,393

Development Phase	Elevation (abs m)	Lift Capacity (days)	Mine Life (years)	Mineralized Material Volume (m^3)	(cum m^3)
	20	2005	5.5	1,171,047	26,483,441
	30	2088	5.7	1,098,733	27,582,174
	40	2166	5.9	1,026,421	28,608,595
	50	2238	6.1	954,110	29,562,704
	60	2305	6.3	881,801	30,444,505
	70	2366	6.5	809,497	31,254,002
	80	2422	6.6	737,200	31,991,202
	90	2473	6.8	664,915	32,656,117
	100	2517	6.9	592,656	33,248,772

17.6 Heap Leach Pad

The heap leach pad consists of a perimeter berm, pad liner system, and leachate collection system to collect and convey the leachate solution to the ADR plant, which should be located adjacent to the heap leach facility. The leach pad has an approximate final footprint area of 6,781,263 square feet (630,000 square meters). The heap leach pad is designed to be operated as a fully drained system with no leachate storage within the HLF. Prior to the start of each of the development stages, the pad foundation must be prepared. Foundation preparation involves stripping the topsoil and vegetation and the removal of any rocks. The topsoil would be stockpiled at a convenient location and used for reclamation of the HLF area at closure. The underlying soils would be excavated down to a competent, stable foundation to provide a uniform and graded surface for the pad liner. Grading and backfill would be used to level the surface and to ensure that the pad grading will promote leachate flow towards the collection piping system and sump. A minimum pad grade of 1-2% is required.

17.7 Liner System

A liner system is planned to maximize solution recovery and minimize environmental impacts by minimizing leachate losses through the bottom of the leach heap pad. The liner system consists of both barrier and drainage layers using a combination of synthetic and natural materials to provide leachate solution containment that meets the accepted standards for leach pad design. The pad is designed to operate with minimal solution storage within the pad structure during normal operating conditions. The liner system is designed to meet the required performance standards assuming fully saturated solution storage conditions. A double liner system has been employed with two layers of synthetic material.

17.8 Liner Design

A liner system has been developed for the pad using an engineered composite double liner design. The double liner system is designed to be installed as the primary liner system under the entirety of the HLF. The double liner system consists of the following components:

- 1.6-foot-thick (0.5-meter-thick) over liner (1.5-inch [38-mm] minus with less than 10% fines content) using economic mineralized material as the material
- 80-mil (2-mm) LLDPE geomembrane

- 1-foot-thick (0.3-meter-thick) compacted low permeability soil liner
- Leak Detection and Recovery System (LDRS)
- 60-mil (1.5-mm) LLDPE geomembrane.
- LLDPE was proposed for the geomembrane liner systems for the heap leach pad because it has the following benefits (Lupo, 2005):
  - Generally higher interface friction values, compared to other geomembrane materials
  - Ease of installation in cold climates due to added flexibility,
  - Good performance under high confining stresses (large heap height)
  - Higher allowable strain for projects where moderate settlement may become an issue.

17.9 Construction

Development of the heap leach liner would be constructed in three phases, with pad expansions proposed after three years of initial production to meet mineralized material stacking requirements. The liner system would be constructed with both the synthetic and natural layers extending to the top of the perimeter berms to provide full containment. The synthetic liners and geotextiles would be anchored and backfilled in a trench along the heap leach pad perimeter and perimeter berms to ensure that mineralized material loading does not compromise the liner coverage of the heap leach pad footprint by pulling the liner into the pad. Along the pad toe, all liners would be tied into their corresponding liner layer along the foundation of the pad to provide a continuous seal and drainage connection.

The perimeter berm would be constructed as part of the liner tie-in around the perimeter of the pad footprint to ensure that heap solution is contained within the pad and to prevent surface runoff entering the pad collection system. A 1-foot-thick (0.3-meter-thick) bedding sand layer would be placed on the face of the confining embankment directly underneath the second (bottom) geomembrane liner to provide additional integrity protection to the liner.

17.10 Over Liner

A protective layer of approximately 1.5 feet (½ meter) of coarse crushed mineralized material/waste would be placed over the entire liner system footprint to protect the liner's integrity from damage during mineralized material placement. The over liner acts as the drainage layer, allowing solution drainage into the pipe collection system. The over liner material must be competent and be free from fines.

17.11 Solution Collection System

Collection and recovery of the leach solution is facilitated by the solution collection system in conjunction with the heap leach liner, over liner, and LDRS. The collection system consists of the following pipe and sump components:

- Lateral collection pipes
- Collection header pipes
- Main header collection pipes

- Leachate collection sumps

The solution collection system would be designed to facilitate quick and efficient solution conveyance off the pad to reduce the potential risk of solution losses through liner system. The entire piping system would be constructed from perforated corrugated plastic tubing (CPT), which is embedded within the over liner layer.

The lateral collection pipes, which would be spaced approximately 16 feet (five meters) apart under the entire pad footprint, feed directly into the collection header pipes, which then flow into the main header. The main header pipes would be positioned along the centerline of each heap leach pad cell and terminate at the upstream toe of the perimeter berm at the leachate collection ditch. Two leachate collection ditches allow solution to flow by gravity to the required storage pond. The collection pipes would be fitted with gate valves to allow solution to be directed to one of the three perimeter collection ditches – PLS, Barren, or Storm.

17.12 Leak Detection and Recovery System

The LDRS would be designed to capture and convey any solution that may leak through the overlying primary geomembrane layer. The LDRS consists of a 1-foot-thick (0.3-meter-thick) sand layer embedded with 4-inch (100-mm) diameter perforated CPT collection pipes. Any leakage recovered by the LDRS would be conveyed into the LDRS sump at the downstream toe of the HLF. A level-switch controlled submersible sump pump would transfer the recovered solution via a pipe installed within the LDRS sand layer and connect into the main solution recovery line for processing. Monitoring of the leakage recovery would be undertaken by recording pump operating hours.

17.13 Leakage Detection Cells

To facilitate more accurate leak identification, the entire pad solution collection system is typically subdivided into multiple independently monitored areas (cells) separated by small berms. Each of these cells has a dedicated leakage detection collection system comprising a drain gravel layer beneath the inner composite liner system which conveys the leakage to a 4-inch (100-mm) diameter perforated collection pipe within the LDRS collection trench. The LDRS ditches flow by gravity at a minimum 0.5 % slope towards the LDRS collection sump, located along the sides of the leach pad. The flow rates from the dedicated collection pipes are continuously monitored and measured prior to discharging into a sump.

17.14 Solution Storage

17.14.1 Event Pond

The Event Pond is designed to provide storage for excess leachate and runoff generated during rainfall events. The pond is situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping inside lined ditches. The Event Pond is designed to meet the following design criteria:

- Storage capacity to contain the excess HLF leachate and surface runoff from the 1 in 100-year 24-hour storm event without discharge
- Overflow designed to discharge the 1 in 200-year 24-hour storm event

The storage requirements for the Event Pond were established based on containment of the entire estimated surface runoff generated from the HLF (at the Phase 3 footprint) during the 1 in 100-year 24-hour storm event. Based on the surface runoff estimates, the following storage requirements for the events pond were identified:

Total runoff estimates for 1 in 100-year 24-hour storm event 2,375,971 cubic feet (67,280 cubic meters)

- 10% additional factor of safety 2,613,285 cubic feet (74,000 cubic meters)
- Total pond storage capacity 2,635,569 cubic feet (74,631 cubic meters)

Solution stored in the Event Pond would be pumped back to the heap leach pad using the Event Pond pump station. The pump station is designed to be able to drain the storm volume over a period of approximately ten days.

17.14.2 PLS Pond and Barren Tank

The PLS pond and Barren tank are designed to provide storage for leachate and Merrill-Crowe return solutions. The PLS pond is situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping and ditches. The PLS pond and Barren tank are designed to meet the following design criteria:

- Storage capacity to contain sufficient solution volumes to maintain irrigation and feed to the Merrill-Crowe circuits
- The PLS Pond is designed to contain up to 24 hours of solution assuming a maximum irrigation rate of 0.005 gpm/ft^2 (12 lph/m^2)
- The PLS Pond is designed with a capacity of approximately 1,108,633 cubic feet (31,393 cubic meters)
- The Barren tank is designed to hold 5 minutes of solution at a capacity of 3,885 cubic feet (110 cubic meters)

Excess solution flows to any of these ponds/tanks would be diverted to the PLS or Event Storm Pond for recycle back to the heap.

17.14.3 Pond Liner System

The engineered double liner system designed for the ponds uses the same design principles as the HLF pad liner system. The liner design consists of the following layer configuration:

- 60-mil (1.5 mm) high-density polyethylene (HDPE) geomembrane
- 1-foot-thick (0.3-meter-thick) low permeability soil liner
- Geosynthetic “geonet” drainage layer
- 60-mil HDPE geomembrane

The liner system installed on the upslope of the pond embankment would have an additional 1-foot-thick (0.3-meter-thick) bedding sand layer that would interface with the lower geomembrane layer to protect the integrity of the liner.

Installation of a LDRS is not required for the Storm Pond as the pond is operated as a dry facility and would only receive and store runoff water during significant storm events. In the event that leakage does occur through the double liner system, this water would be conveyed via the geonet layer to a 3-foot-thick (1-meter-thick) drainage blanket that underlies the Event Pond embankment. This drainage blanket discharges to a sump for solution return to the pond.

It is recommended that HDPE geomembrane be used for the pond liner system rather than LLDPE. Unlike the heap leach pad, the pond liner system would not be subjected to high confining stresses from mineralized material stacking, and HDPE has a higher ultraviolet resistance, which is critical for exposed surfaces like that of the ponds.

17.15 Runoff Collection and Diversion

The surface water management system proposed for the site consists of a series of ditches constructed around the perimeter of the HLF to intercept overland surface runoff around the HLF pad and to convey surface water away from the active site. The ditches are designed to meet the following design criteria:

- Conveys the 1 in 100-year 24-hour duration storm event
- Minimum freeboard = 1-foot (0.3 meters)
- Minimum ditch grade = 0.01 foot/foot (meter/meter)
- Side slopes = 2H:1V
- Channel shape = trapezoidal.

Lining and protection of the ditch channels from erosion and scouring may be required for all permanent ditches. Temporary ditches would be constructed between heap phases.

18.0 PROJECT INFRASTRUCTURE

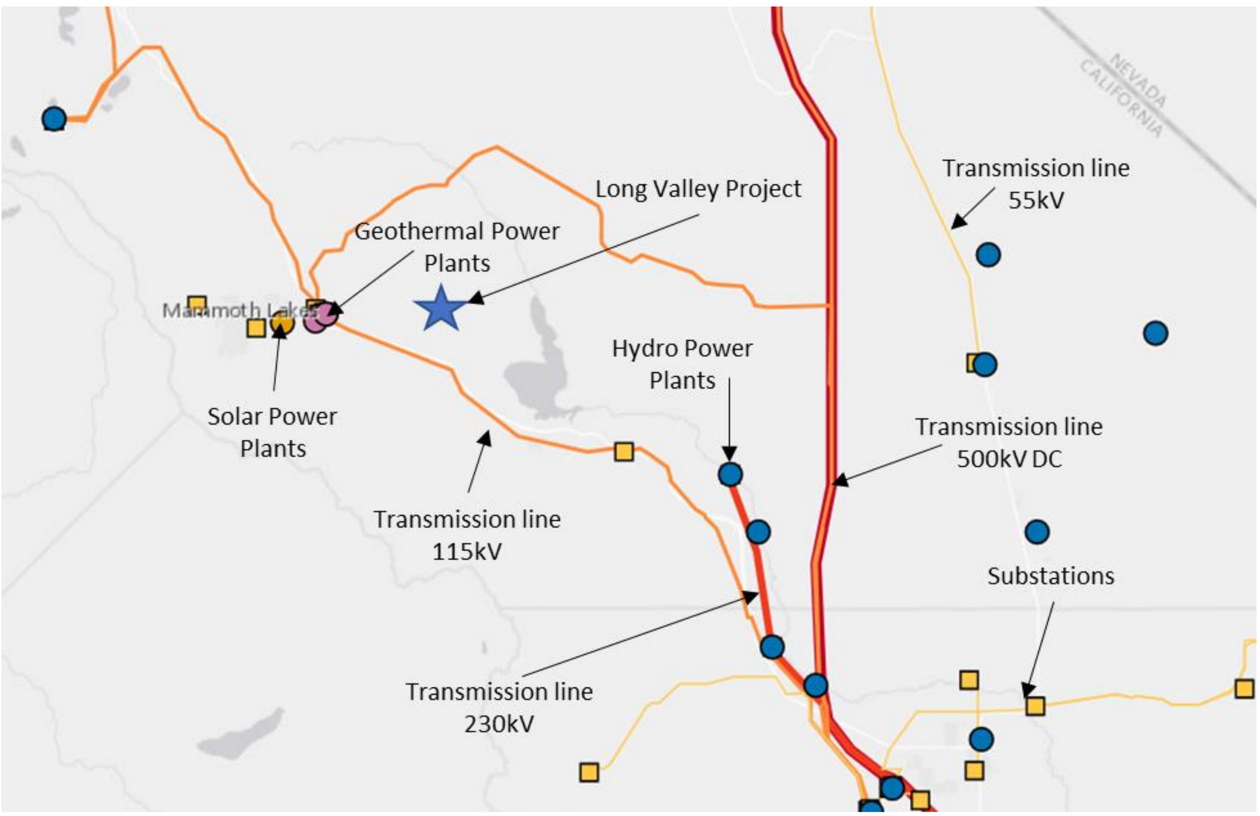
18.1 Water Supply

Heap operation over the estimated mine life indicates that operation of the HLF requires a water supply with an approximate average flowrate of 450 gpm (100 m^3/hr). An additional 150 gpm (34 m^3/hr) is required for mine, shop, and office water consumption.

18.2 Electric power

There is nearby power reticulation with a 115kV line at the U.S. Highway 395 and a nearby power plant, Upper Gorge just west of Lake Crowley. A substation and a power line could be installed to connect the project power to the local grid at an estimated cost of 1.8 million USD.

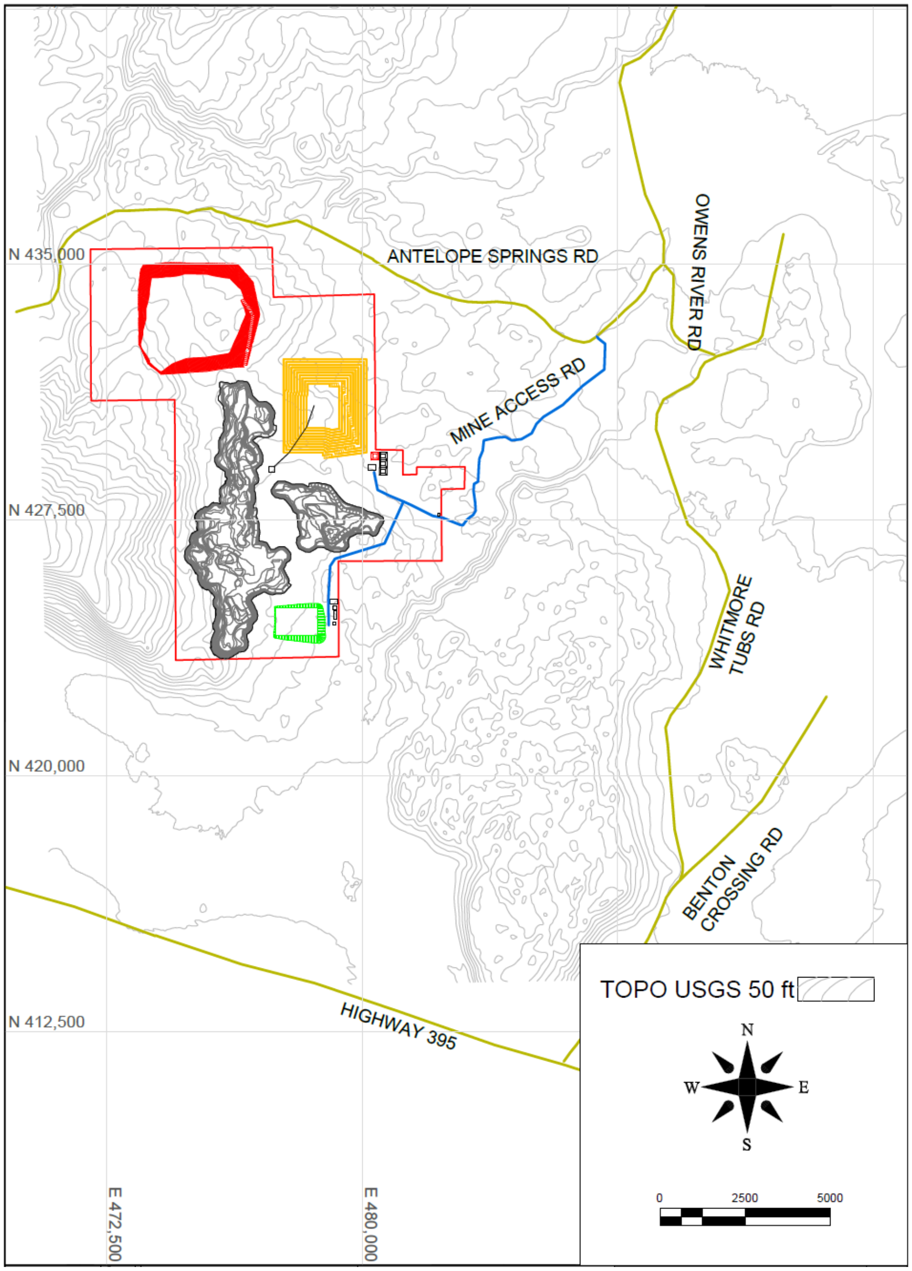
Figure 18-1: Energy Infrastructure



18.3 Access Roads

Access to the Long Valley project is from a graded gravel road from U.S. Highway 395 to the project property. U.S. Highway 395 is a four-lane divided highway that can support traffic to and from the mine. All mine access roads will be paved to prevent dust and noise and internal site roads will be treated with water and magnesium chloride to prevent dust. Figure 18-2 shows the existing roads and possible site access for the Long Valley project.

Figure 18-2: Long Valley Property Road Access



18.4 Water Balance and Water Supply

The following summarizes key components of the hydrologic analysis completed for the project by Ms. Lane of GRE.

Ms. Lane of GRE completed a preliminary hydrological assessment of the Long Valley Project site using a combination of HLF design data, project data, and climate information obtained from publicly available sources.

Meteorological information was acquired from the Western Regional Climate Center (WRCC). Gauging station information for the area compiled from US Geological Survey databases. Annual pan evaporation records were obtained from a technical report prepared by Farnsworth and Thompson (1982). Monthly distribution of pan evaporation was obtained from WRCC.

18.4.1 Water Balance

Modeling of the heap operation on a monthly basis over the projected mine life indicates that operation of the HLF requires a water supply with an approximate average flowrate of 450 gpm (101 m^3/hr). An additional 150 gpm (34 m^3/hr) is required for mine, shop, and office water consumption.

A water balance around the heap leach was produced using average rainfall, evaporation and temperatures. Key parameters included in the hydrologic assessment were average precipitation, average runoff, and pan evaporation. No simulation was conducted to incorporate major events at this stage of the study. Table 18-1 presents the monthly distribution of average precipitation at the project site.

Table 18-1: Long Valley Site Average Climate Conditions

Month	Precipitation (mm)	High Average (deg C)	RH (%)	Low Average (deg C)	RH (%)	Pan Evaporation (mm)
Jan	40.0	4.40	60	-6.10	83	0.0
Feb	48.0	5.70	53	-5.10	83	97.5
Mar	47.0	9.70	37	-2.00	73	184.4
Apr	29.0	14.10	27	0.70	68	257.3
May	35.0	19.70	23	5.30	65	345.4
June	18.0	25.70	18	10.10	54	414.3
July	30.0	29.50	20	14.30	56	456.7
Aug	39.0	28.30	22	13.30	57	404.4
Sept	30.0	23.60	20	8.80	55	287.5
Oct	30.0	16.50	28	2.60	67	174.8
Nov	29.0	9.10	46	-2.40	78	74.7
Dec	28.0	4.40	61	-6.20	82	0.0
	403.0					2697.0

Previous estimates calculated the mean annual runoff for the mine site to be approximately 0.04 inches/year.

18.4.2 Water Balance

A preliminary operational average monthly water balance model was developed for the HLF. The intent of the modeling was to estimate the magnitude and extent of any water surplus or deficit conditions in the HLF based on annual average climatic conditions. The modeling timeline was for 8 years of HLF operations.

The model incorporates the following major project components:

- Heap Leach Pad
- Mine Usage
- Shop Usage

- General Usage
- Fresh Water Supply
- Pond and Tank Storage – PLS, Barren and Event

The findings of the water balance were that the HLF would operate in a water deficit. The deficit is most pronounced in the early years and is reduced as water stored within the mineralized material is released from the earlier leaching stages. The total make-up required by the HLF is estimated at 1.6 billion gallons (6.1 million cubic meters [m^3]) over the life of the facility. The HLF water requirement ranges from 261 million gallons to 290 million gallons annually (990 thousand m^3 to 1.1 million m^3 annually). The project requires a significant amount of water at start up due to the initial mineralized material wetting requirements and the solution retention in the heap. Ms. Lane of GRE estimates that approximately 136 million gallons (515,000 m^3) of fresh water would be necessary at the start of heap operations.

The water balance was based on assumed moisture content values for the stacked mineralized material and climatic conditions for the site. The model is sensitive to these values and they should be reviewed and confirmed for future design studies. The following criteria were employed in the water balance:

- Natural Moisture Content – Mineralized Material 4%
- Field Moisture Content – Mineralized Material 12%
- Drain-Down Final Moisture Content 10%
- Evaporation Losses – 5% total
- Pan Evaporation for pond based on Bishop, CA.
- Average Irrigation Rate 0.005 gpm/ft^2 (12.2 lph/m^2)
- Pad Area – Phase 1,2, and 3: 3,616,674 square feet (ft^2), 5,425,011 ft^2 and 6,781,264 ft^2 (336,000, 504,000, 630,000 m^2)
- Climate Conditions monthly temperature, precipitation and evaporation

18.5 Mine Facilities

Ms. Lane of GRE has provided conceptual design of facilities required for mine operations. These include access roads, offices, warehouses, shops, leach pad, and waste dumps (see Figure 16-18).

18.5.1 Waste Rock Storage Facilities

The waste rock storage facility (WRSF) is planned to be temporarily located north of the ultimate pit limits. The WRSF will contain 21.7 million tons of waste material, 28.4% of total waste. The sides of the WRSF are at a 2.5:1 slope with a 90 feet wide ramp providing haul truck access at a 10% grade. The remaining waste will be stored in concurrent backfills in mined out phases of the pit. The backfilling of the previously mined out pits during the active mine life is planned to minimize the amount of waste material that needs to be reclaimed at the end of the mining operation. Backfills are designed at a 2.5:1 slope and have an access ramp 90 feet wide at a 10% grade. The pits would be backfilled from the bottom up to original ground elevations. The backfill WRSFs are utilized concurrently through the mine life for waste produced from the pit. After mining has finished, the remaining pit volume is filled with the depleted heap leach material.

18.5.2 Mine WRSF Development Schedule

The somewhat linear advancement of the main pit and relatively flat pit bottom provide an excellent opportunity for backfilling concurrently during active mining operations.

Prestripping and production bench waste at the start of the schedule are stored in the WRSF north of the pits. Once Phase 1 is complete in Year 3, backfilling can begin concurrent to production mining. When Phase 1 backfill is full in Year 5, waste storage resumes in the WRSF north of the pit until later in Year 5 when pit Phase 2 is available for backfill. Phase 2 is backfilled until it is filled in Year 7, the final year of mine life. The remaining waste in Year 7 is placed into the open Phase 3 backfill. Table 18-2 shows the storage by WRSF by year.

Table 18-2: WRSF by Year, Millions of Tons

WRSF	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Dump 1	1.2	11.6	5.6	2.6	-	0.8	-	-	21.8
Phase 1 BF	-	-	-	9.3	7.2	0.5	-	-	17.0
Phase 2 BF	-	-	-	-	-	13.5	14.1	1.4	29.0
Phase 3 BF	-	-	-	-	-	-	-	8.9	8.9

To comply with California law, pits are backfilled so the final surface is a close approximation of original topo. The surface WRSF and heap leach pad are rehandled back into the pit. Any material left on the surface stays within +25 feet of original topo. Figure 16-17 shows the Long valley mine plan post reclamation.

The backfill material will be utilized to re-create the washes with sufficiently high berms, as well as curtain the runoff to the stream channel. The design would mimic the existing wash topography and physiological characteristics. The following are some conceptual design criteria that would be incorporated into the next phases of engineering.

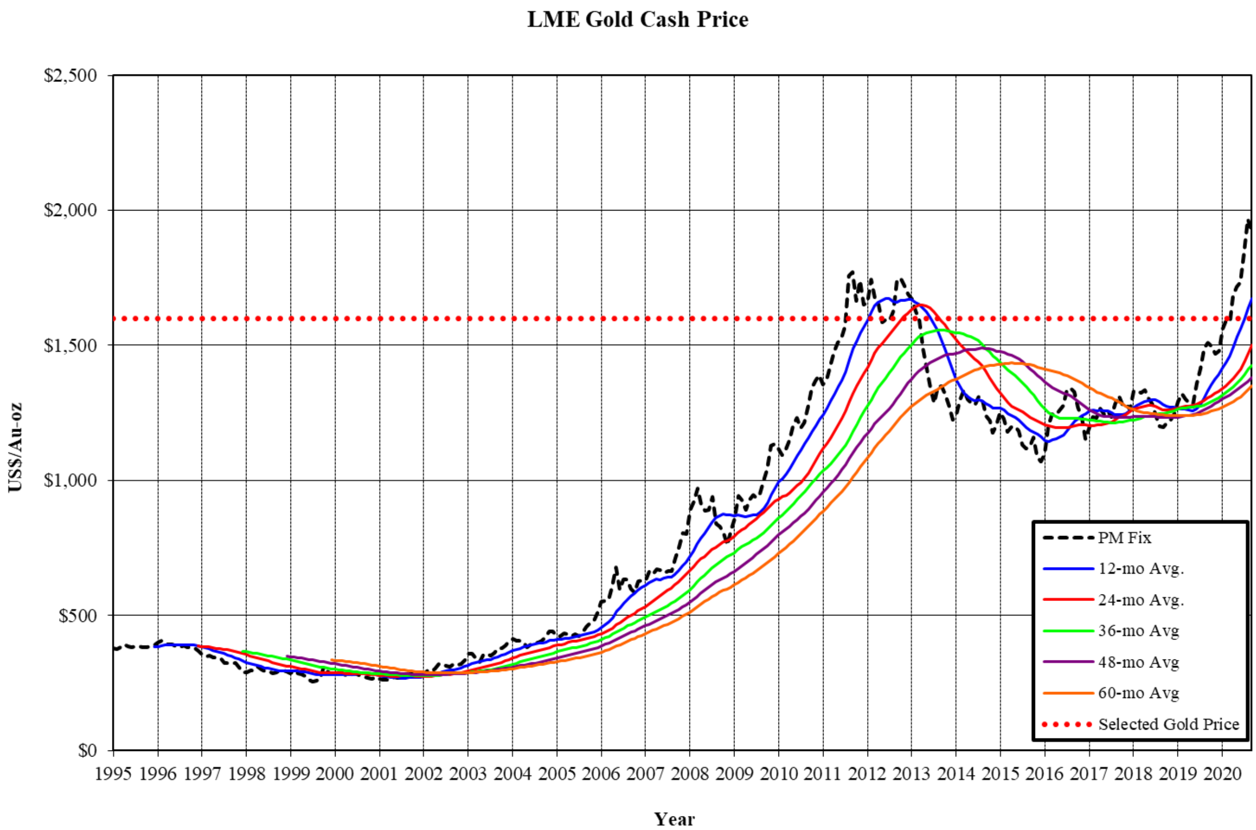
- The backfill area would not impound water.
- Any washes would be rebuilt to pre-mining elevations.
- The centerline of the wash through the pit backfill area would maintain the pre-mining slope (fall) of the original wash. The entrance and exit of the wash through the pit area should not include any drops or rises but should smoothly match to the existing slope.
- The wash bottom would be reconstructed with stockpiled wash materials (sands and gravels).
- The pit backfill areas outside the washes can be below the pre-mining topography but should mimic the morphology of the pre-mining slopes in that vicinity unless they are steeper than 3H:1V.
- The final reclamation surface will be less than or equal 25 feet above the current surface topography over almost all of the project area if the waste dumps and HLF material are required to be removed to within 25 feet of original topography.

- The maximum slope would be 3H:1V.

19.0 MARKET STUDIES AND CONTRACTS

The primary metal of economic interest for the Long Valley project is gold. Gold has a readily available market for sale in the form of gold doré or gold concentrates. Figure 19-1 presents the gold market London PM fixed pricing through September 30, 2020. The selected Gold price for the PEA is $1,600/oz which represents the 3-year trailing average, $1,425/oz weighted by 60% and $1,860/oz projected gold price weighted by 40%. KORE Mining nor any of the authors of this report have conducted a market study in relation to the gold doré or gold concentrates that will be produced at the Long Valley Gold Project. The refining treatment charge in this study is assumed to be $5 per ounce.

Figure 19-1: London Metals Exchange PM Gold Price



20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 LAND USE ENTITLEMENTS

20.1.1 Federal—Plan of Operations (PO)/Reclamation Plan (36 CFR *et seq.)*

The Project area is located on National Forest land managed by the Mammoth Ranger District in Mammoth Lakes, California; therefore, the USFS would require a complete application for a Plan of Operations/Reclamation Plan (PO) prior to authorizing the mining operations. The PO application must contain site-specific information about the proposed Project in sufficient detail to satisfy regulating authorities. The PO needs to describe the property owner and operator, the type of operation to be conducted (including the mining method and beneficiation process), and reclamation procedures with associated costs. The USFS could, upon review, request additional information to ensure that the PO is complete.

20.1.2 County—Mining Permit and Reclamation Plan

Prior to constructing and operating a mining project in Mono County, a mining plan and a reclamation plan, consistent with the California Surface Mining and Reclamation Act (SMARA), must be submitted to and approved by the Mono County. Most of the required information is similar to that required by the USFS for the PO and includes a complete and accurate description of the mine development.

20.1.3 Mining Operations Permit

Prior to engaging in extraction, processing, or any other mining operations within the territorial boundaries of Mono County, a Mining Operations Permit must be approved by the Mono County Planning Commission. Although information requirements for the Mining Operations Permit are similar in nature to those for the mining and reclamation plan under SMARA, the Mining Operations Permit requires a separate permit application, and the permit would be in addition to the SMARA reclamation plan.

The ordinance requiring a Mining Operations Permit in Mono County adopted by the Mono County is intended to provide greater control in approving proposed mining operations located on land over which the County lacks full land-use and zoning authority (e.g., federal land administered by federal agencies). The Mining Operations Permit application would require a detailed description of all proposed mining activities, but would be focused primarily on proposed processing activities that utilize hazardous chemicals. Under the ordinance, hazardous chemicals cannot be used in connection with any processing activity unless it can be shown that the activity will not, under any reasonable foreseeable scenario, cause significant adverse impacts on the environment (as defined under CEQA).

20.1.4 Reclamation Plan and Financial Assurances

In addition to the land use permit to mine (in this case a PO from the USFS), surface mining activities in the State of California are regulated in accordance with SMARA. SMARA requires that a site-specific Reclamation Plan that is a land use entitlement to be prepared and implemented. A Reclamation Plan is a description of the mined condition and plan for the methods that mined lands would be reclaimed to a usable condition which is readily adaptable for alternative land uses. SMARA also requires that an assessment of the estimated cost to reclaim a project be prepared, approved by Mono County as well as the California Office of Mining Reclamation; it is reviewed and updated annually. No surface disturbing activities may occur prior to securing a reclamation financial assurance with the lead agency in the amount agreed upon by the reviewing agencies.

For the Long Valley Project, Mono County, the USFS, and the Lahontan Regional Water Quality Control Board (LRWQCB) would review the cost estimate and ultimately agree upon a reclamation cost. The LRWQCB will hold one financial assurance for the facilities under its permit, and the USFS or Mono County will hold the financial assurance bond for the remainder of the facilities. The reclamation of an open pit heap leach facility will provide for third-party costs to implement reclamation should the operator fail to do so. The financial assurance will provide for reclaiming the heap leach facility and with meeting SMARA's backfilling provisions applicable to all metals mines in California.

20.2 Operating Permits

20.2.1 Waste Discharge Requirements

The owner or operator of any facility proposing to discharge waste to land must submit a "Report of Waste Discharge" to the appropriate Regional Water Quality Control Board (RWQCB), which must either approve WDRs, or waive the need for WDRs, before the facility or project discharges waste to land. Certain facilities common to mining operations (e.g., surface impoundments, heaps and waste rock piles) are typically subject to construction and/or closure requirements established in California Code of Regulations, Title 23, Division 3, Chapter 15, and are issued WDRs. The Project is located in the RWQCB-Lahontan Region.

The "Report of Waste Discharge" must provide a complete and accurate description of all waste discharges by type, quality, quantity, interval and method of discharge. This includes a characterization of both the mineralized material and waste to determine potential acid rock drainage (ARD) and leachable metals. Upon submittal, the RWQCB evaluates the "Report of Waste Discharge" to ensure completeness as well as compliance with all applicable regulations.

20.2.2 Air Quality Authority to Construct

Any operator proposing to construct, modify or operate a facility or equipment that would emit pollutants from a stationary source into the atmosphere would need to first obtain an Authority to Construct from the applicable County or Regional Air Pollution Control District (APCD). For a facility in Mono County, the appropriate APCD is the Great Basin Unified APCD, whose principal office is located in Bishop, California. The Project is assumed to be a minor source of air pollutants and would probably not require a "Federal Title V Operating Permit" or a "Prevention of Significant Deterioration Permit".

20.2.3 Air Quality Permit to Operate

In addition to the Authority to Construct, a Permit to Operate must also be obtained from the Great Basin Unified APCD. The Permit to Operate is required to operate any facility that emits air pollutants which is not otherwise exempted by rule. In general, to obtain approval of the Permit to Operate the operator is required to certify that construction of the facility was completed according to the terms and conditions of the Authority to Construct and that the facility will meet the APCD's regulations. If specified in the Authority to Construct or the APCD 's regulations, source testing of some or all sources may be required to demonstrate compliance.

20.2.4 Jurisdictional Wetlands and Waters

Under Section 404 of the Clean Water Act (CWA), a permit must be obtained from the USACE before any " waters of the United States," which include wetlands, stream channels, and ephemeral stream channels, are dredged or filled. Acquisition of either a general or individual permit requires a delineation of "waters of the United States" and completion of the appropriate application.

The mapping for the claim area includes the following four vegetation types: barren, eastside pine, pinyon-juniper, and sagebrush. Despite the upland vegetation community types, there is potential for wetlands and waters to be present on the site because the mapping is conducted on such a broad scale that it is not a reliable source for identifying jurisdictional features.

Topography of the site indicates presence of linear features particularly in the eastern portion of the claim that may be considered jurisdictional dry washes under current Corps guidance.

Prior site surveys indicate that jurisdictional features are present at the site. Jurisdictional wetlands and waters may appear dry during much of the year and the absence of flowing or standing water is not considered to be a reliable indicator of the absence of these features. Particularly since dry washes have been incorporated into current jurisdictional determinations by both federal and state resource agencies, further assessment of the site would be required to determine the extent of these features on the claim.

20.3 ENVIRONMENTAL REVIEW

20.3.1 Preparation of a Joint EIS/EIR

Federal and California regulations have provisions which encourage the preparation of joint environmental documents. CEQA guidelines contain clear authority for state and local agencies to prepare joint environmental documents with federal agencies. NEPA regulations issued by the President's Council on Environmental Quality contain similar provisions.

For the Project, the preparation of a joint EIS/EIR would probably be undertaken by the USFS and the Mono County Planning Department as the lead agencies under NEPA and CEQA, respectively. Because the Project is located entirely on National Forest land, however, the USFS would likely take the "lead" role in the preparation of the joint document. However, the EIS/EIR must contain all the content and components, as well as all public notice and review, required under both NEPA and CEQA. Preparation of a joint EIS/EIR would likely require a Memorandum of Understanding (MOU) between the lead agencies

20.3.2 National Environmental Policy Act (NEPA)

Given the presumed use of cyanide and the other potential activities associated with the Project, it is almost certain that the USFS, as lead agency, would decide that the Project could result in significant impacts to the human environment, thereby triggering, as mandated by NEPA, the preparation of an EIS. The EIS would evaluate the proposed action and any logical alternatives to the proposed action to determine the extent of any environment impacts. At the completion of the process, the USFS would issue a Record of Decision (ROD) to authorize the proposed action, or authorize an alternative to the proposed action, and which would discuss in detail the rationale used in making the decision.

An EIS is a technical and complete assessment of impacts to the environment caused by the proposed action. Baseline studies would be reviewed and approved by agency staff. In addition, there are requirements which provide numerous opportunities for comment by the public throughout the process of preparing the EIS. Under NEPA, the federal government is required only to give appropriate consideration to environmental values. Although an EIS must evaluate all reasonable alternatives to the proposed action and must propose appropriate measures to mitigate identified impacts, there is no requirement for federal land managers to adopt feasible alternatives or mitigation measures. To this end, courts have held that NEPA is essentially procedural and its only role is to ensure that environmental impacts of a proposed project are considered.

20.3.3 California Environmental Quality Act (CEOA)

CEQA has requirements similar to NEPA in that the potential significant environmental impacts of the proposed action and feasible alternatives must be evaluated in an EIR prepared by the Mono County Planning Department (MCPD), as lead agency, under CEQA. There are, however, substantial differences between NEPA and CEQA. CEQA differs from NEPA in that CEQA places a relatively higher value on environmental protection compared with economic growth. CEQA requires agencies to implement feasible alternatives or mitigation measures that would reduce project-related environmental impacts to below a level of significance; an agency cannot satisfy the statute only by considering the environmental impacts of a proposed project. Only if an agency finds there to be "overriding considerations" can that agency approve a project without adopting those feasible alternatives or mitigation measures identified in the EIR.

Table 20-20-1, "Timeline for Key Permit and Approvals," summarizes the key approvals, typical time frames, and approach for the Long valley Project.

Table 20-20-1: Timeline for Key Permits and Approvals

Permit/Authorization	Timeline	Work Needed
Environmental Impact Statement (EIS) *(NEPA, 42 USC § 4321 et seq.)*	18–24 months The USFS would become involved in the process at the time of pre-application meeting and application submittal.	An EIS is needed to address the environmental consequences of the Project.
Environmental Impact Report (EIR) *(CEQA, PRC § 21000 et seq.; 14 CCR § 15000 et seq.)*	Prepared concurrently with the NEPA document. The County would become involved in the process at the time of pre-application meeting and application submittal.	The analysis for the Project would be prepared under new requirements under CEQA (e.g., GHG).
Plan of Operations *(36 CFR § 228.4)*	Processed concurrently with the NEPA document. The USFS would become involved in the process at the time of pre-application meeting and application submittal.	For approval of construction and operation.
Mining Operations Permit Mono County Ordinance Chapter 7.10	Processed concurrently with SMARA reclamation plan	For approval of construction and operation.
Mining/Reclamation Plan and Financial Assurance *(SMARA) (PRC § 2710 et seq.)*	Processed concurrently with the CEQA and NEPA review. The County would become involved in the process at the time	This would be done as part of the reclamation plan process with Mono County. Project will need

	of pre-application meeting and application submittal.	to comply with the current backfilling regulations.
Water Discharge Permit (Water Code 13000 et seq.)	6–9 months to obtain, after CEQA document is complete	Obtain for project as approved.
Individual/Nationwide Section 404 Discharge Permit *(Clean Water Act, 33 USC § 1341)*	12-18 months	Obtain for project as approved.
Lake/Streambed Alteration Agreement *(Fish and Game Code § 1603)*	6–9 months to obtain, after CEQA document is complete	Obtain for project as approved (as applicable).
Section 401 (Water Quality) Certification *(CWA, 33 USC § 1251: If the Project Requires USACE 404 permit)*	2–6 months, after CEQA document is complete	Obtain for project as approved.
Authority to Construct *(Local district rules, per Health and Safety Code § 42300 et seq.)*	6 months, after CEQA document is complete	Obtain for project as approved.

Notes: USFS = U.S. Forest Service; CEQA = California Environmental Quality Act; CCR = California Code of Regulations; CWA= Clean Water Act; NEPA = National Environmental Policy Act; PRC = Public Resources Code; USC = U.S. Code; USACE = U.S. Army Corps of Engineers.

20.4 COMMUNITY CONCERNS

Public outreach was undertaken in 1990 by the USFS. Issues of concern raised by the local community included: surface and groundwater hydrology effects, proximity to geothermal spring systems and seismic stability of the area archaeological resources, cyanide use and wildlife, proximity to a fish hatchery, noise and dust and visual resources relative to Highway 395. The issues of concern are expected to be the same today. The project may encounter resistance being located in a region largely valued for passive (hiking, camping, hunting and fishing) and active recreation (skiing and other winter sports) activities with the local economy largely reliant on tourism.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

Capital costs are presented for the mining, mineral processing, and administrative portions of the operation. The capital cost estimate has been prepared for the PEA under the assumption of processing 22,000 short tons per day of leachable material on a heap leach. Sources for costs used in the economic model include cost data from Infomine (InfoMine, 2020), GRE's internal data and first principle estimates, and costs from similar projects in North America.

Initial capital costs are defined as all costs until a sustained positive cash flow is reached. This includes labor and development costs in the pre-production year. Sustaining capital is defined as the capital costs incurred in the periods after a sustained positive cash flow is achieved through the end of mine life. All capital cost estimates cited in this Report are referenced in US dollars.

Table 21-1:Long Valley Capital Costs

Initial and Sustaining Capital Costs	Total Cost ($ millions)
Mining & mine Infrastructure	40.6
Heap leach pads and plant	55.5
Infrastructure & G&A	18.5
Working capital	4.6
Contingency (25%)	27.9
Total Pre-Production Capital	**147.0**
Pre-production mining	13.9
Total Pre-Production Cost	**160.9**
Sustaining capital	18.2
Closure, incl. Backfill	72.4

21.1.1 Mining

Mining capital costs used to develop the economic model are summarized in Table 21-2. Ms. Lane of GRE considered all the mining equipment necessary for a larger scale operation.

Table 21-2: Long Valley Project Mine Capital Costs Summary (1000s)

Item	Quantity	Units	$/Unit	Total Cost
Excavator CAT 6040	2	ea	$8,420	$16,840
Haul Truck CAT 785D	7	ea	$2,344	$16,407
Bulldozer D10	3	ea	$1,091	$3,272
Drill	2	ea	$2,000	$4,000
Loader CAT 992K	1	ea	$2,208	$2,208
Water Truck	2	ea	$1,140	$2,280
ANFO Truck	1	ea	$220	$220
Lube Truck	2	ea	$84	$168
Mechanics Truck	2	ea	$71	$141
Grader	1	ea	$443	$443
Small Excavator	1	ea	$305	$305
Backhoe	1	ea	$129	$129
Small Crane	1	ea	$395	$395

Item	Quantity	Units	$/Unit	Total Cost
Light Plant	6	ea	$25	$152
Dewatering Pump	1	ea	$165	$165
4x4 Pickup	10	ea	$46	$461
Total				$47,586

Note: Costs rounded to nearest thousand may not sum due to rounding.

21.1.2 Mineral Processing and Heap Leach

The $55,526,249 cost of the process plant, including the first phase of the heap leach pad, is incurred in the preproduction year. Heap leach expansion occurs in years two and four of production with a cost of $6,278,681 and $4,709,011, respectively. The total capital cost for the heap leach in each of those years is $66,513,941. The breakdown of the unit costs of the process plant is shown in Table 21-3.

Table 21-3: Long Valley Project Mineral Processing and Heap Leach Capital Costs (1000s)

Capital Costs	Cost
Crushing	$6,544
Agglomeration and Stacking	$4,496
Leach Pad, Ponds, Sol'n Dist and Collection	$16,212
Gold Recovery	$1,405
Utilities	$5,986
Installation Labor	$9,550
Concrete	$879
Piping	$2,965
Structural Steel	$879
Instrumentation	$769
Insulation	$241
Electrical	$879
Coatings and Sealants	$255
Spares and First Fill	$2,149
Engineering/Management	$7,739
Total - Fixed Equipment	**$60,949**
Mobile Equipment	
Pad	$5,200
Maintenance	$125
Light Vehicles	$240
Total - Mobile Equipment	**$5,565**
Total – Fixed and Mobile Equipment	**$66,514**

Note: Costs rounded to nearest thousand may not sum due to rounding.

21.1.3 Infrastructure and Administrative

All buildings and associated infrastructure installed on the property on a permanent or semi-permanent basis are considered facilities. They include material and installation cost.

Infrastructure and administrative capital costs were estimated using Ms. Lane's experience with similar sized projects in the American West and current project costs. Administrative capital costs applied to the economic model are listed in Table 21-4.

Table 21-4: Long Valley Project Infrastructure and Administrative Capital Costs (1000s)

Description	Total Cost
Haul Roads	$460
Office	$788
Warehouse	$1,000
Mine Shop	$3,500
Fuel Bay	$100
Wash Bay	$200
Security and Fencing	$250
Surface Water Management	$500
Water Well with Pump	$1,250
New Well Pump	$67
Back Up Gen Set	$346
Sub-Station	$1,500
Power Line 33KV	$1,767
Computers	$150
Software	$255
Tech Equipment	$100
Office Equipment	$250
Total	**$12,483**

21.1.4 Working Capital

Working capital is the necessary cash on hand for one month's operating cost. The estimated total is $4,567,000. Note that this cost is recovered at the end of production.

21.1.5 Contingency

A 25% contingency was added to the capital cost estimate, totaling $36,292,000.

21.1.6 Closure

Closure cost includes final backfilling of the open pit and site reclamation to California's stringent standards. The cost includes US$38.7 million in mining cost, US$13.4 million in site operating G&A during back-filling of the final pit, in addition to US$15.3 million for rinsing and neutralizing the heap leach pad. Backfill will return the site to plus 25 feet of original topography while re-establishing drainages. The combined cost for the three parts of closure is $67.5 million.

21.2 Operating Costs Estimate

Operating costs in the economic model are calculated based on current actual costs, first principles, estimated using the experience of Ms. Lane, Dr. Harvey, or scaled from Infomine. The operating costs are categorized by mining, mineral processing, or administrative functions.

21.2.1 Mining

Operating costs for mining include equipment operation, labor, and consumable materials. Mining equipment includes production equipment and support equipment. Mining production equipment hours are calculated using the equipment productivity estimates and the number of tons required to be moved.

Mining support equipment hours are calculated using the number of shifts that the equipment is operated per day, the number of pieces of equipment, and the operating hours per day. The operating hours per day are calculated assuming utilization of 90%, availability of 95%, and two twelve-hour shifts per day. Table 21-5 summarizes the mining costs by year.

Table 21-5: Long Valley Mining Equipment Operating Costs by Year (1000s)

Equipment	Quantity	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Excavator CAT 6040	2	$1,023	$2,349	$1,776	$2,598	$1,995	$2,978	$2,895	$2,313	**$17,925**
Haul Truck CAT 785D	6	$2,822	$5,409	$3,523	$5,497	$3,937	$6,000	$5,042	$3,640	**$35,870**
Bulldozer D10	3	$285	$1,734	$1,734	$1,734	$1,734	$1,734	$1,734	$1,734	**$12,424**
Drill	2	$213	$1,298	$1,298	$1,298	$1,298	$1,298	$1,298	$1,298	**$9,299**
Loader CAT 992K	1	$163	$990	$990	$990	$990	$990	$990	$990	**$7,096**
Water Truck	2	$186	$1,131	$1,131	$1,131	$1,131	$1,131	$1,131	$1,131	**$8,105**
ANFO Truck	1	$33	$202	$202	$202	$202	$202	$202	$202	**$1,445**
Lube Truck	2	$20	$120	$120	$120	$120	$120	$120	$120	**$858**
Mechanics Truck	2	$19	$116	$116	$116	$116	$116	$116	$116	**$830**
Grader	1	$31	$190	$190	$190	$190	$190	$190	$190	**$1,362**
Small Excavator	1	$24	$148	$148	$148	$148	$148	$148	$148	**$1,062**
Backhoe	1	$9	$55	$55	$55	$55	$55	$55	$55	**$397**
Small Crane	1	$19	$118	$118	$118	$118	$118	$118	$118	**$846**
Light Plant	6	$9	$57	$57	$57	$57	$57	$57	$57	**$411**
Dewatering Pump	1	$1	$4	$4	$4	$4	$4	$4	$4	**$31**
4x4 Pickup	10	$149	$906	$906	$906	$906	$906	$906	$906	**$6,489**
Total		**$5,007**	**$14,828**	**$12,369**	**$15,165**	**$13,002**	**$16,048**	**$15,008**	**$13,024**	**$104,451**

Note: Costs rounded to nearest thousand may not sum due to rounding.

Blasting material consumption is determined assuming a powder factor of 0.5 lb/ton of material mined. The main explosive used is assumed to be ANFO. The cost of blasting is summarized in Table 21-6.

Table 21-6: Long valley Project Blasting Costs by Year (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Explosives	$232	$3,236	$2,446	$3,579	$2,748	$4,102	$3,989	$3,187	**$23,518**
Mineralized Material Control/Sample Testing	$39	$539	$408	$596	$458	$684	$665	$531	**$3,920**
Misc	$500	$500	$500	$500	$500	$500	$500	$500	**$4,000**
Total	**$770**	**$4,275**	**$3,354**	**$4,675**	**$3,706**	**$5,286**	**$5,154**	**$4,218**	**$31,438**

Note: Costs rounded to nearest thousand may not sum due to rounding.

Manpower for the mine includes both hourly-rate employees and salaried employees. The number of required equipment operators was estimated using the quantities of equipment required, the number of personnel per piece of equipment, and shifts per day. Numbers of required salaried personnel are estimated based on Ms. Lane's and Dr. Harvey's experience. A burden factor of 40% was added to all hourly personnel and 36% for salaried personnel. The burden includes fringe benefits, holidays, vacation and sick leave, absentees, insurances, etc. A summary of the manpower costs is provided in Table 21-7.

Table 21-7: Long Valley Project Mining Labor Cost Summary by Year (1000s)

Position	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Hourly Personnel									
Heavy Equipment Operators	$356	$1,780	$1,780	$1,780	$1,780	$1,780	$1,780	$1,780	**$12,819**
Support Equipment Operators	$89	$534	$534	$534	$534	$534	$534	$534	**$3,828**
Production Truck Drivers	$344	$3,484	$2,787	$3,484	$2,787	$4,181	$3,484	$2,787	**$23,339**
Blasters	$75	$159	$159	$159	$159	$159	$159	$159	**$1,187**
Mine Laborers	$603	$1,270	$1,270	$1,270	$1,270	$1,270	$1,270	$1,270	**$9,494**
Drill Operators	$227	$1,380	$1,380	$1,380	$1,380	$1,380	$1,380	$1,380	**$9,890**
Oilers/Mechanics	$247	$3,749	$3,374	$3,749	$3,374	$3,937	$3,749	$3,374	**$25,553**
Salaried Personnel									
Mine Superintendent	$190	$190	$190	$190	$190	$190	$190	$190	**$1,523**
Mine Engineer	$136	$136	$136	$136	$136	$136	$136	$136	**$1,088**
Geologist	$136	$136	$136	$136	$136	$136	$136	$136	**$1,088**
Surveyor/Tech	$169	$169	$169	$169	$169	$169	$169	$169	**$1,349**
General Foreman	$136	$136	$136	$136	$136	$136	$136	$136	**$1,088**
Shift Supervisor	$408	$408	$408	$408	$408	$408	$408	$408	**$3,264**
Extras	$118	$118	$118	$118	$118	$118	$118	$118	**$940**
Total	**$3,233**	**$13,650**	**$12,578**	**$13,650**	**$12,578**	**$14,534**	**$13,650**	**$12,578**	**$96,450**

Note: Costs rounded to nearest thousand may not sum due to rounding.

21.2.2 Mineral Processing

Operating costs for mineral processing include labor, reagents, power, and consumables. Mineral processing operating costs used in the economic model are categorized as either fixed or variable costs. Fixed costs do not vary over the life of the mine and are applied regardless of the quantity of potentially economic mineralized material mined. Variable costs are scaled on a per ton basis.

The processing costs shown in Table 21-8 include all post mining activities until shipment off site for smelting and refining, including crushing and agglomeration, leaching, ADR plant, and refining. Costs include post-mining processing of any stockpiled material.

Operating costs for salaried and hourly labor are based on estimates of required manpower for the operation and Dr. Harvey's experience with processing employee wages (Table 21-8). A 35% percent burden factor was estimated for all employees. Costs include continued irrigation for 6 months after mine production has ceased.

Table 21-8: Long Valley Project Mineral Processing Costs by Year (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Production Process Fixed										
Salaried Labor	$0	$1,343	$1,343	$1,343	$1,343	$1,343	$1,343	$1,343	$1,343	**$10,746**
Hourly Labor	$0	$5,657	$5,657	$5,657	$5,657	$5,657	$5,657	$5,657	$5,657	**$45,252**
Variable Costs										
Reagents and Consumables for Oxides	$139	$7,458	$8,796	$6,018	$8,079	$7,592	$6,212	$6,932	$0	**$51,225**
Reagents and Consumables for Transition	$0	$2	$121	$532	$241	$166	$293	$333	$0	**$1,689**
Reagents and Consumables for Sulfides	$0	$0	$146	$1,299	$399	$1,046	$1,930	$488	$0	**$5,309**
Power	$53	$2,871	$3,614	$3,619	$3,614	$3,613	$3,649	$3,338	$0	**$24,370**
Fixed Cost Year 8 of Continued Irrigation	$0	$0	$0	$0	$0	$0	$0	$0	$4,586	**$4,586**
Total	**$192**	**$17,332**	**$19,676**	**$18,469**	**$19,332**	**$19,417**	**$19,083**	**$18,090**	**$11,585**	**$143,177**

Note: Costs rounded to nearest thousand may not sum due to rounding.

21.2.3 Administrative

Administrative operating costs are estimated for the project based on Ms. Lane's experience with similar sized mines located in the American West. Table 21-9 lists the estimated administrative operating costs. Table 21-10 lists the estimated quantities and salaries of administrative staff required to operate the mine.

Table 21-9: Long Valley Project Administrative Service and Supply Costs by Year (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Maintenance Supplies	$101	$101	$101	$101	$101	$101	$101	$101	**$808**
Office Supplies/Software	$152	$152	$152	$152	$152	$152	$152	$152	**$1,213**
Transportation	$144	$144	$144	$144	$144	$144	$144	$144	**$1,152**
Light Vehicle Operating Costs	$220	$220	$220	$220	$220	$220	$220	$220	**$1,760**
Corporate Compliance	$100	$100	$100	$100	$100	$100	$100	$100	**$800**
Public Relations Expense	$101	$101	$101	$101	$101	$101	$101	$101	**$808**
Communications	$101	$101	$101	$101	$101	$101	$101	$101	**$808**

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Insurance, Misc. Taxes, Fees, Licenses	$560	$560	$560	$560	$560	$560	$560	$560	**$4,480**
Safety Supplies	$30	$30	$30	$30	$30	$30	$30	$30	**$240**
Environmental (Testing, etc)	$160	$160	$160	$160	$160	$160	$160	$160	**$1,280**
Training Supplies	$13	$13	$13	$13	$13	$13	$13	$13	**$107**
Outside Audit (Accounting, Metallurgy, etc)	$120	$120	$120	$120	$120	$120	$120	$120	**$960**
Travel	$120	$120	$120	$120	$120	$120	$120	$120	**$960**
Legal	$225	$225	$225	$225	$225	$225	$225	$225	**$1,800**
Data Processing	$72	$72	$72	$72	$72	$72	$72	$72	**$576**
Access Road Maintenance	$100	$100	$100	$100	$100	$100	$100	$100	**$800**
Security (Night Shift)	$80	$80	$80	$80	$80	$80	$80	$80	**$640**
Cleaning	$32	$32	$32	$32	$32	$32	$32	$32	**$256**
Miscellaneous (10%)	$243	$243	$243	$243	$243	$243	$243	$243	**$1,944**
Total	**$2,674**	**$2,674**	**$2,674**	**$2,674**	**$2,674**	**$2,674**	**$2,674**	**$2,674**	**$21,392**

Note: Costs rounded to nearest thousand may not sum due to rounding.

Table 21-10: Long Valley G&A Labor Costs by Year (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
General Manager	$272	$272	$272	$272	$272	$272	$272	$272	**$2,176**
Purchasing Manager/ Chief Accountant	$136	$136	$136	$136	$136	$136	$136	$136	**$1,088**
Purchaser	$84	$84	$84	$84	$84	$84	$84	$84	**$675**
Accounting Clerk	$84	$84	$84	$84	$84	$84	$84	$84	**$675**
Human Resources/Relations Manager	$136	$136	$136	$136	$136	$136	$136	$136	**$1,088**
Human Resources/Payroll Clerk	$68	$68	$68	$68	$68	$68	$68	$68	**$544**
Security/Safety/Training Manager	$122	$122	$122	$122	$122	$122	$122	$122	**$979**
Environmental Supervisor	$163	$163	$163	$163	$163	$163	$163	$163	**$1,306**
Environmental Technicians	$84	$84	$84	$84	$84	$84	$84	$84	**$675**
IT Manager	$122	$122	$122	$122	$122	$122	$122	$122	**$979**
Warehouseman ON SITE	$299	$299	$299	$299	$299	$299	$299	$299	**$2,394**
Accounts Payable Clerk	$75	$75	$75	$75	$75	$75	$75	$75	**$598**
Guards	$299	$299	$299	$299	$299	$299	$299	$299	**$2,394**
Laborers / Janitorial ON SITE	$75	$75	$75	$75	$75	$75	$75	$75	**$598**
Total	**$2,021**	**$2,021**	**$2,021**	**$2,021**	**$2,021**	**$2,021**	**$2,021**	**$2,021**	**$16,168**

Note: Costs rounded to nearest thousand may not sum due to rounding.

The total estimated operating costs are summarized in Table 21-11, and operating unit costs are shown in Table 21-12.

Table 21-11: Long Valley Project Operating Cost Summary (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Mine Operating Costs	$9,011	$32,753	$28,301	$43,499	$30,067	$37,431	$43,820	$29,819	**$254,700**
Process Operating Costs	$192	$17,332	$19,676	$18,469	$19,332	$19,417	$19,083	$18,090	**$131,591**
G&A Operating Costs	$4,706	$4,716	$4,727	$4,738	$4,749	$4,749	$4,749	$4,749	**$37,882**
Total Operating cost	**$13,908**	**$54,801**	**$52,705**	**$66,706**	**$54,147**	**$61,596**	**$67,652**	**$52,658**	**$424,174**

Note: Costs rounded to nearest thousand may not sum due to rounding.

Table 21-12: Long Valley Project Operating Unit Costs

Item	Basis	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Mine	($/mined ton)	$7.00	$1.82	$2.08	$2.19	$1.97	$1.64	$1.98	$1.88	**$7.00**
Process	($/process ton)	$1.62	$2.72	$2.45	$2.30	$2.41	$2.42	$2.35	$2.64	**$1.62**
G&A	($/process ton)	$64.36	$1.21	$0.96	$0.96	$0.92	$0.84	$0.74	$0.89	**$64.36**

22.0 ECONOMIC ANALYSIS

The economic analysis is modeled at the time of a production decision. It allows for 1 year of preproduction and construction. Costs incurred for exploration, engineering, and permitting over 3 to 5 years leading up to a production decision are not include.

The project has a short pre-production period of less than 1 year, a production life of 7 years, and a reclamation and closure time of 5-6 years.

Table 22-1: Summary of Long Valley Economic Results

Economics	Unit	Pre-Tax	Post-Tax
Net present value (NPV5%) at 0.75C$/US$	C$ millions	$463	$364
Net present value (NPV5%)	US$ millions	$347	$273
Internal rate of return (IRR)	%	57%	48%
Payback (undiscounted)	Years	1.6	1.8
LOM avg. annual cash flow after tax & capital	US$ millions	$96	$83
LOM cumulative cash flow (undiscounted)	US$ millions	$475	$385
Gold price assumption	US$ per ounce	$1,600	
Mine life	Years	7	
Average annual mining rate	million tons/yr	18.5	
Average annual gold production	thousand ounces/yr	102	
Total LOM recovered gold	thousand ounces	717	
Initial capital costs	US$ millions	$161	

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

22.1 Economic Analysis

The economic analysis includes the 1% NSR royalty payable to Royal Gold and the second 1% NSR royalty that is payable to Vista Gold for a total of a 2% NSR royalty. The undiscounted value of the 2% total NSR royalty for the base case is $23 million.

The U.S. federal income tax is based on the Internal Revenue Code of 1986, as amended and the relevant state and local statutes, the regulations thereunder, and judicial and administrative interpretations thereof, on the following assumptions and tax return elections by the taxpayer, based on the PEA cashflows and capital expenditures. As of October 21, 2020, the U.S. federal corporate income tax rate is twenty-one (21) percent, the State of California rate is (8.84) percent.

The federal and state income tax is based on the following assumptions and tax elections:

The Long Valley Project is owned by a California Corporation (“taxpayer”) which is a wholly owned direct or indirect subsidiary of KML. The Long Valley Project has acquired an economic interest in the minerals in place and is operated and treated as a single mine under Section 614. The Long Valley Project will elect to expense exploration expenditures under Section 617(a) as incurred. Long Valley will deduct mine development costs as incurred under Section 616(a) for Phase I and 616(b) for Phase II. The Long Valley Project will elect out of Section 168(k) bonus depreciation. The Long Valley Project will elect to accrue and deduct reclamation costs under Section 468. All of Long Valley's metal sales will be delivered outside of the United States. California Property Tax is imposed under Revenue and Taxation Code 20584 and the regulations on real and personal property based upon the municipality and county where the mine is located.

22.2 Gold Recoveries and Revenue

The following key economic parameters were considered to determine the best scenario: NPV, IRR, payback period, mine life, and initial capital cost. A cutoff grade of 0.004 oz/t was selected for oxide mineralized material, 0.005 oz/t for transition mineralized material, and 0.017 oz/t for sulfide mineralized material, and $1,600/oz as gold price.

Gold recovery assumed to occur over six months period, Table 22-2 Shows the cumulative and incremental recovery for different material types.

Table 22-2: Long Valley Cumulative and Incremental Recovery

Month	Oxides		Transition		Sulfides	
	Au Ext (%)	Inc Ext (%)	Au Ext (%)	Inc Ext (%)	Au Ext (%)	Inc Ext (%)
1	51.60	51.60	30.78	30.78	12.69	12.69
2	70.08	18.48	45.97	15.19	17.23	4.55
3	74.50	4.42	52.46	6.50	18.32	1.09
4	75.99	1.49	54.32	1.85	18.69	0.37
5	79.01	3.01	55.57	1.26	19.43	0.74
6	80.00	0.99	60.00	4.43	20.00	0.57

Table 22-3 summarizes the gold recoveries and revenues and Table 22-4 summarizes the results of the economic model. Table 22-5 summarizes the all-in-sustaining-cost (AISC).

Table 22-3: Long Valley Project Gold Recoveries and Revenues (1000s)

Revenue	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Contained Gold	2	122	157	158	155	167	191	108	0	**1,060**
Recovered Gold	0	93	123	91	114	109	105	80	2	**717**
Gold Revenue	$0	$148,487	$196,557	$145,669	$182,881	$173,664	$168,219	$128,500	$3,770	**$1,147,747**
Royalty	$0	($2,960)	($3,919)	($2,904)	($3,646)	($3,462)	($3,354)	($2,562)	($75)	**($22,883)**
Net Smelter Revenue	$0	$145,062	$192,024	$142,309	$178,663	$169,659	$164,340	$125,536	$3,683	**$1,121,277**

Note: Costs rounded to nearest thousand may not sum due to rounding

Table 22-4: Long Valley Project Summary of Economic Model ($ millions)

Description	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Total
Net Smelter Revenue	$0.00	$145.06	$192.02	$142.31	$178.66	$169.66	$164.34	$125.54	$3.68	$0.00	$0.00	$0.00	$0.00	$0.00	**$1,121**
Total Operating Costs	$13.91	$54.80	$52.70	$66.71	$54.15	$61.60	$67.65	$52.66	$35.98	$2.93	$2.93	$10.95	$10.95	$0.00	**$488**
Pre-Tax Operating Cash Flow	($13.91)	$90.26	$139.32	$75.60	$124.52	$108.06	$96.69	$72.88	($32.29)	($2.93)	($2.93)	($10.95)	($10.95)	$0.00	**$633**
Federal Tax	$0.00	$3.36	$16.37	$4.23	$14.68	$10.26	$7.73	$4.14	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$61**
California Tax	$0.00	$1.41	$7.02	$2.33	$6.39	$4.88	$4.42	$2.48	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$29**
Property Tax	$2.92	$2.98	$2.97	$2.98	$2.67	$1.99	$1.23	$0.44	$0.29	$0.25	$0.21	$0.16	$0.12	$0.07	**$19**
Total Capital Costs	$144.11	$5.88	$7.87	$0.08	$5.91	$2.95	$0.08	$0.02	$1.17	$8.18	$1.12	$0.00	$0.00	$4.10	**$181**
Income After Tax	($160.94)	$76.63	$105.10	$65.98	$94.87	$87.98	$83.22	$65.80	($33.75)	($11.37)	($4.26)	($11.11)	($11.07)	($4.17)	**$343**

Note: Costs rounded to nearest thousand may not sum due to rounding.

Table 22-5: Long Valley Project AISC per Ounce

AISC per ounce	Value (US$ millions)
Operating cost (1)	$614
Royalties (2)	$32
Sustaining capital	$25
Closure	$101
Total AISC	**$773**

(1) Operating costs includes US$5 per ounce offsite refining.
(2) Royalties consist of: (a) 1% NSR royalty to Royal Gold; and (b) 1% NSR royalty to Vista Gold.

22.3 Sensitivity Analyses

The after-tax NPV@5% and IRR sensitivity to changes in gold prices were evaluated in Table 22-6, and sensitivities to operating costs and capital costs are shown in Table 22-7 and Table 22-8. A graphical summary the NPV and IRR are shown in Figure 22-1 and Figure 22-2.

Table 22-6: After Tax NPV@5% and IRR Sensitivity to Gold Price

Gold Price $/tr oz	NPV 5% (millions)	IRR
1000	3.8	6%
1100	50.0	17%
1200	96.7	25%
1300	142.6	32%
1400	187.4	38%
1500	228.6	43%
1600	272.6	48%
1700	308.6	53%
1800	351.7	58%
1900	394.7	63%
2000	437.8	67%
2100	480.9	72%
2200	523.9	76%
2300	567.0	81%
2400	610.0	85%
2500	653.1	90%
2600	696.1	94%
2700	739.2	98%
2800	782.2	102%
2900	825.3	107%
3000	868.4	111%

Table 22-7: Project Economics Sensitivity to Operating Costs

Sensitivity To OpEx	Operating Cost $/ton Economic Material	Operating Cost $/recovered oz	NPV 0%	NPV 5%	NPV 10%	IRR
80%	$6.40	$484	$406	$323	$256	54%
85%	$6.76	$510	$391	$311	$246	53%
90%	$7.12	$537	$375	$298	$235	51%
95%	$7.48	$564	$359	$285	$225	50%
100%	**$7.83**	**$591**	**$343**	**$273**	**$214**	**48%**
105%	$8.19	$618	$327	$260	$204	47%
110%	$8.55	$645	$311	$247	$193	45%
115%	$8.90	$672	$294	$234	$182	44%
120%	$9.26	$699	$278	$220	$172	42%

Table 22-8: Project Economics Sensitivity to Capital Costs

Sensitivity To CapEx	Capital Cost $ millions	Capital Cost $/recovered oz	NPV 0%	NPV 5%	NPV 10%	IRR
80%	$133.40	$186	$373	$302	$243	60%
85%	$140.29	$196	$366	$295	$236	57%
90%	$147.17	$205	$358	$287	$229	54%
95%	$154.05	$215	$350	$280	$222	51%
100%	**$160.94**	**$224**	**$343**	**$273**	**$214**	**48%**
105%	$167.83	$234	$335	$265	$207	46%
110%	$174.72	$244	$328	$258	$200	44%
115%	$181.59	$253	$320	$251	$193	42%
120%	$188.48	$263	$313	$244	$186	40%

Figure 22-1: Project NPV Sensitivity

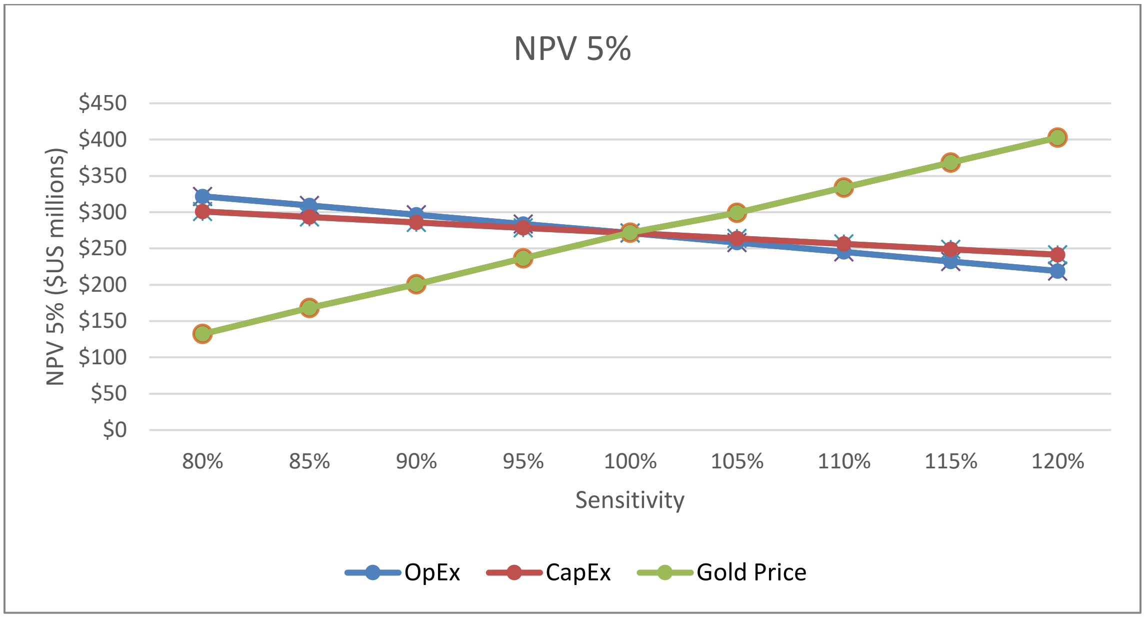
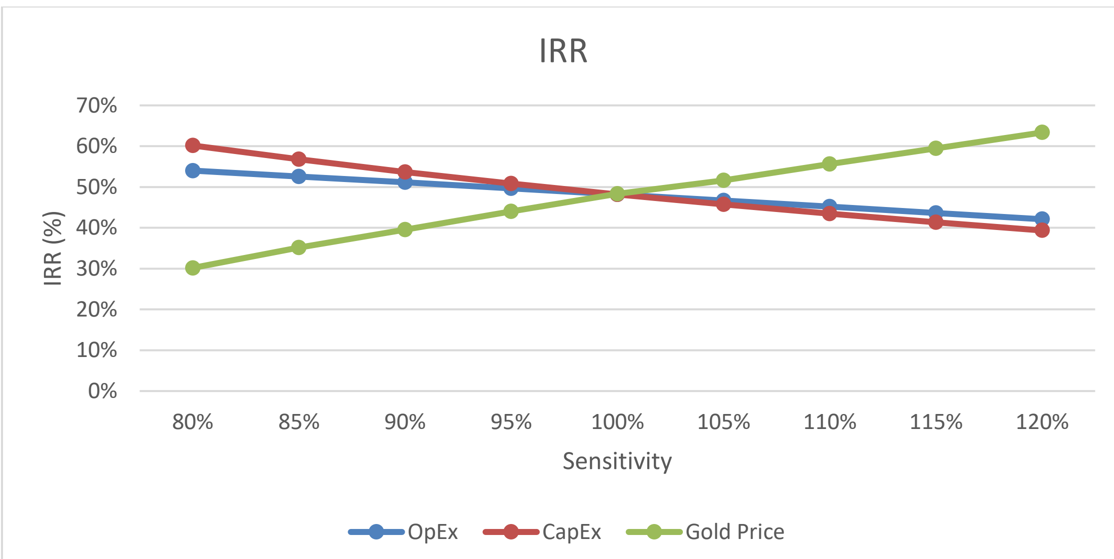


Figure 22-2: Project IRR Sensitivity



22.4 Conclusions of Economic Model

The project economics shown in the PEA are favorable, providing positive NPV values at varying gold prices, capital costs, and operating costs. The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

23.0 ADJACENT PROPERTIES

There are no other gold or silver properties known in the general area of the Long Valley property. About one-half mile north of the claim block is an area of previous mining activity by Standard Industrial Minerals for the extraction of kaolinite clay that was mined from a series of small open pits and trucked off-site for processing. Although Standard was actively mining kaolinite in 2008 (MDA, 2008), they were not active as of the effective date of this report.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors know of no current impediments to, or impacts from, the recommended work programs that would keep the company from completing the Phase 1 and Phase 2 work programs as outlined in Section 26.0 of this report.

25.0 INTERPRETATION AND CONCLUSIONS

The Long Valley property is a property of merit with a large, potentially bulk-mineable, open-pit gold resource which is well defined by a large drill hole database. The drill sample collection and assaying were performed to industry standards at the time and provide a good basis for any future work to be performed on the property. The deposit is open to depth and also in several areas along strike for resource expansion.

The Long Valley gold deposit as presently understood is a shallow, largely stratiform epithermal deposit within the central part of the Long Valley caldera. Gold mineralization forms relatively continuous, tabular and flat-lying to gently east-dipping, near-surface bodies. The mineralization spans a north-south distance of about 8,000 feet with an average width of about 1,000 feet. The mineralized zones are typically from 50 to 200 feet thick and average about 125 feet thick in the Hilton Creek zone, and 75 feet thick in the Southeast zone.

The drilling database contains records from 896 drill holes totaling 268,275 feet of drilling. There has been no drilling on the property since 1997.

Pit-constrained Indicated gold resources total approximately 70.2 million tons at an average grade of 0.017oz Au/ton, for about 1.22 million contained ounces of gold located within 300 feet of the surface. Inferred resources comprise an additional pit-constrained 24.3 million tons at an average grade of 0.019oz Au/ton.

The authors have reviewed the project data, including the Long Valley drill hole database, and visited the project site. The authors believe that the data provided by KORE are generally an accurate and reasonable representation of the Long Valley project.

The mine plan is based on 22,000 tons per day of mineralized material production. The pits were divided into 4 phases including one satellite pit. In the initial phases the mine is extracted from south to north followed by the extraction of satellite pit. Pre-stripping and phasing is used to have similar quantity of leachable material production throughout the mine life. The plan produces 54.2 million tons of leachable material at an average grade of 0.020 oz/ton or 0.67 g/tonne in a 7-year mine life. Stripping requirements include a life of mine total of 76.5 million waste tons. Waste management for the mine includes a waste dumps and concurrent backfilling. At the end of production, the waste dump will be transported to the open pit and the heap leach pad will be rinsed and neutralized. After rinsing and neutralization of the heap leach material, it will be transported into the remaining open pit. The total of 17.8 million tons of the material remain on the surface, which is reclaimed to the +25 ft of the original topography.

Operating cost in production years for the Long Valley Project amount to $1.88 per short ton mining cost, $2.64 per short ton processed processing cost, and $0.89 per short ton processed G&A cost. Total capital cost for the project are $47.6 million mine, $55.5 million plant, $0.76 million G&A, $11.7 million infrastructure, $11.0 million sustaining, $18.6 million reclamation, and $36.3 million contingency for a total of $181.5 million.

The PEA used a base gold price of $1,600/oz with an estimated overall recovery of 68% which resulted in an After-Tax Net Present Value at 5% of $273 million and an Internal Rate of Return of 48%. This technical report includes inferred mineral resources.

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

25.1 Risks

Mr. Prenn believes that while considerable drilling has defined the deposit, additional core drilling should be utilized to define the structural controls of the deposit and in addition to collect additional density and metallurgical information for the deposit. Core drilling should improve the definition of higher-grade zones and their relationship to geologic features. An improved understanding of the structural and geologic controls could result in a portion of the estimated resources being classified as Measured. Core drilling will also provide additional samples for mineralized material and waste density and metallurgical testing.

The area of the current resource is open to expand at depth and to the south, north, and east. Although future exploration could lead to the discovery of additional mineralization with the potential to add to the currently estimated resources, there is no certainty that future exploration will lead to such discoveries. This is a normal exploration risk.

The main risks associated with the project are related to permitting and California mining regulations. This risk could potentially cause long delays in acquiring permits and additional holding costs during these delays.

There is a risk that the project will encounter serious opposition during the permitting process if the permitting effort is not properly managed. To mitigate this risk the Company plans to initiate an industry best practice community engagement program to build local support with all stakeholders.

The change in California mining regulations in the early 2000's with the introduction of the backfill law severely impacted new projects. With the current higher gold price, the backfill requirement can be met without severely impacting the project economics. There is a risk other regulation could be implemented that further impact project economics.

Metallurgy needs more test work to advance the knowledge regarding recovery, work index, and spatial variability of the metallurgical characteristics.

The project currently does not have any geotechnical test work or analysis. This could impact the pit slope and stripping ratio.

Obtaining mine operating permits for the project may be more difficult than normal due to the project's location in California and proximity to the town of Mammoth Lakes, California, where the predominant source of revenue is derived from tourism. The main anticipated issues relating to the future development of a mining operation at Long Valley would likely be the impact on the current tourism-based economy and particularly the potential visual impacts, impacts to ground water in the area, and the use and containment of cyanide solutions. At this stage of the project these potential impacts have not been quantified.

The Long Valley gold resources are located approximately 1.5 miles north of the Hot Creek fish hatchery operated by the California Department of Wildlife. At this stage of the project, any potential impacts the fish hatchery may have on permitting or development of the project have not been quantified.

The Long Valley property is contained entirely within the late Pleistocene Long Valley caldera, which was formed about 760,000 years ago. Repeated eruption of dacite and rhyodacite from vents on the southwest rim of the caldera 220,000 to 50,000 years ago formed Mammoth Mountain, a dome complex. The USGS monitors the area for volcanic activity and does not have an advisory or watch alert level for the caldera. The authors believe that this is a low-level risk in the short period of time needed to develop and mine the project.

26.0 RECOMMENDATIONS

The authors recommend the following 2-phase work program to advance the Long Valley project:

Phase 1

- Add silver to the resource model
- Perform an environmental impact assessment
- Collection of geotechnical, hydrology, and hydrogeology data
- Perform closure testing on the spent heap materials to determine if the material can cause water quality impacts.
- Execute geotechnical investigations into the heap stability.
- Perform geotechnical testing of soils under the leach pad, ponds, and plant site.
- Conduct geotechnical testing of the pit wall.
- Improve metallurgical understanding of the orebody through additional metallurgical sampling. Drilling should be weighted to match the distribution of sulfide, oxide, transition, siliceous, and argillic material.
- Sulfide-sulfur assays should be conducted on all samples in addition to gold, silver, hot cyanide leach, and a full ICP scan.
- Further test work should be considered for the Long Valley project:
    - Crusher work index and abrasion tests should be conducted to confirm crusher design and wear rates.
    - Agglomeration tests should be performed to confirm the optimal mix of cement/lime, and moisture necessary to achieve acceptable percolation and leaching results.
    - A comprehensive array of column tests should be arranged with representative samples from all areas of the deposit. Minimal column work is necessary for the sulfide material as it is not amenable to heap leaching.
    - The optimization of the crush size requires further investigation and the investigation of HPGR may be warranted given the material characteristics.
    - Sulfide mineralogy should be tested to define a suitable flowsheet for this material if economically warranted. Several basic crush, grind leach tests should be conducted followed by additional testing if the material is refractory to conventional processing techniques.

If Phase 1 is successful, the authors recommend proceeding to Phase 2, a pre-feasibility or feasibility study.

Table 26-1 shows the estimated cost for the proposed Phase 1 work program.

Table 26-1 Estimated Cost of Recommended Program

Description	Total (US$)
Engineering and Other Studies	
Baseline environmental study	1,000,000
Geotechnical / HL design studies	500,000
Metallurgical test work	500,000
Subtotal	**2,000,000**
Community Engagement Program	200,000
Stakeholder Mapping	100,000
Subtotal	**300,000**
Contingency (10%)	430,000
Total	**2,730,000**

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CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled, Preliminary Economic Assessment NI 43-101 Technical Report - Long Valley Gold Project, Mono County, California, USA, with an effective date of September 21, 2020 and an issue date of June 7, 2021

I, Jeffrey Todd Harvey, do hereby certify that:

1) I am the President and Director of Process Engineering with the firm of Global Resource Engineering Ltd ("GRE") with an office at 600 Grant Street, Suite 975, Denver Colorado, 80203, U.S.A;
2) I have not visited the site.
3) I graduated with Ph.D. in Mining Engineering from the Queen's University at Kingston in 1994, a Master's degree in Mining Engineering from the Queen's University at Kingston in 1990 and a Bachelors degree in Mining Engineering in 1988 all with a specialization in mineral processing. I also hold a degree in Metallurgical Engineering and Computer Science from Ryerson University in Toronto Canada graduating in 1986 as well as an MBA from the University of New Brunswick in Saint John Canada graduating in 2001. I have worked as a Process Engineer for over 35 years since my graduation from university. My relevant experience includes process due diligence/competent persons evaluations of developmental phase and operational phase mines throughout the world, including mines in the USA, Canada, Kazakhstan, Brazil, Mexico, and Africa to name a few. I have a wide range of experience in multiple mineral fields including precious metal processing and base metals such as copper, lead, and zinc;
4) I am a Registered Member (No. 04144120) of the Society for Mining, Metallurgy & Exploration Inc. (SME). I am also a member of the Association for Mineral Exploration (AME), Minerals Engineering Journal Review Board, and the Journal of Hydrometallurgy Review Board;
5 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6) I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for sections 1.6, 1.9, 12.3, 13, 17, 18.1, 18.2, 18.4, 21.1.2, and 21.2.2 of this technical report and accept responsibility for those sections;
8) I have had no prior involvement with the subject property other than being a co-author of the filing of a technical report on the Long Valley Project in October 27, 2020;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Long Valley gold project or securities of KORE Mining Ltd; and
11) That, as of the effective date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Denver Colorado, U.S.A.	*["signed and sealed"]*
June 7, 2021	Jeffrey Todd Harvey
	Principal Consultant (Processing)

CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled, Preliminary Economic Assessment NI 43-101 Technical Report - Long Valley Gold Project, Mono County, California, USA, with an effective date of September 21, 2020 and an issue date of June 7, 2021

I, Terre A. Lane, do hereby certify that:

1) I am a Principal Mining Engineer with the firm of Global Resource Engineering Ltd ("GRE") with an office at 600 Grant Street, Suite 975, Denver Colorado, 80203, U.S.A;
2) I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience project management, mineral resource estimation, mine capital and operating costs estimation, and economic analysis with 25 or more years of experience in each area. I have created or overseen the resource estimation, mine design, capital and operating cost estimation, and economic analysis of well over a hundred open pit projects. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
3) I have not visited the site.
4) I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP and a Registered member of SME - 4053005.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6) I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for sections 1.8, 1.10 through 1.18, 2, 3, 4, 12.4, 15, 16, 18.3, 18.5, 19, 20, 21.1.1, 21.1.3 through 21.1.6, 21.2.1, 21.2.3, 22, 24, 25.0, and 27 of this technical report.
8) I have had no prior involvement with the subject property other than being a co-author of the filing of a technical report on the Long Valley Project in October 27, 2020;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Long Valley gold project or securities of KORE Mining Ltd; and
11) That, as of the effective date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Terre A. Lane

"Terre A. Lane"

Mining Engineer

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: June 7, 2021

CERTIFICATE OF QUALIFIED PERSON

I, Neil Prenn, P.E., do hereby certify that I am currently employed as Principal Mining Engineer with Mine Development Associates, Inc., 210 South Rock Blvd, Reno, Nevada 89502, USA, and;

1. I am a graduate of the Colorado School of Mines with an Engineer of Mines degree, 1967. I have practiced my profession continuously since 1967. I have been an independent consultant for over 31 years;
2. I am a Registered Professional Mining Engineer in the state of Nevada, USA (#7844). I am a registered 'QP' member with the Mining and Metallurgical Society of America (MMSA-01283QP). I am a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME). I have worked in technical, operations and management positions at mines in the United States; I have completed or managed employees completing resource estimates for gold deposits during the past 50 years of my professional career. I have written computer programs to aid in the calculation of resource estimates and am a member of the SME resource and reserve committee that recently worked with the SEC to develop new reporting standards in the U.S.
3. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I visited the Long Valley project site on October 30, 2002 and on February 21, 2018;
5. I am responsible for sections 1.1 through 1.3, 1.5, 1.7, 5, 6, 9 through 11, 12.1, 14, 23, 25.1, and 26 of this report, titled "Preliminary Economic Assessment NI 43-101 Technical Report - Amended, Long Valley Project, Mono County, California, USA" with an effective date of September 21, 2020 and an issue date of June 7, 2021 (the "Technical Report").
6. My prior involvement with the Long Valley project includes preparation of a previous Technical Report titled "Amended Technical Report and Resource Estimate for the Long Valley Project, Mono County, California, USA" prepared for Kore Mining Ltd., and with an effective date of November 15, 2019 and the filing of a technical report on the Long Valley Project in October 27, 2020;
7. I am independent of Kore Mining Ltd. and all of its affiliates and subsidiaries, and the Long Valley Property, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
8. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Long Valley gold project or securities of KORE Mining Ltd.
9. As of the effective date of this 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 technical report not misleading;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Neil Prenn, PE

"Neil Prenn"

Mine Development Associates, Inc.

Reno, Nevada

Date of Signing: June 7, 2021

CERTIFICATE OF QUALIFIED PERSON

I, Steven I. Weiss, C.P.G., do hereby certify that:

- I am currently a self-employed Senior Associate Geologist for Mine Development Associates, Inc., located at 210 South Rock Blvd., Reno, Nevada, 89502; and
- I graduated with a Bachelor of Arts degree in Geology from the Colorado College in 1978, received a Master of Science degree in Geological Science from the Mackay School of Mines at the University of Nevada, Reno in 1987, and hold a Doctorate in Geological Science from the University of Nevada, Reno, received in 1996.
- I am a Certified Professional Geologist (#10829) with the American Institute of Professional Geologists and have worked as a geologist in the mining industry and in academia for more than 35 years.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have previously explored, drilled, evaluated and reported on gold-silver deposits in volcanic and sedimentary rocks in Nevada, California, Canada, Greece, and Mexico. I certify that by reason of my education, affiliation with certified professional associations, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am a co-author of this Technical Report titled *"Preliminary Economic Assessment NI 43-101 Technical Report - Long Valley Project Mono County, California, USA"* prepared for Kore Mining Ltd., and with an effective date of September 21, 2020 and an issue date of June 7, 2021. I am responsible for Sections 1.4, 7, 8 and 12.2 of this Technical Report.
- I had involvement with the property that is the subject of this Technical Report in 1996, employed as a Research Associate by the University of Nevada, Reno conducting research funded by Royal Gold Inc. on the property. Subsequently, I was a co-author of the Technical Report titled *"Preliminary Economic Assessment NI 43-101 Technical Report - Amended Long Valley Project Mono County, California, USA"* prepared for Kore Mining Ltd., and with an effective date of September 21, 2020 and an issue date of June 7, 2021. I last visited the property on September 20, 2020 and conducted a personal inspection of the property on that date.
- I am independent of Kore Mining Ltd. and, and all of their respective subsidiaries, and the Long Valley Property, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Long Valley gold project or securities of KORE Mining Ltd.
- To the best of my knowledge, information and belief, as of the effective date the Technical Report contains the necessary scientific and technical information to make the Technical Report not misleading.
- I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with the requirements of that instrument and form.

Steven I. Weiss, PhD, CPG
"Steven I. Weiss"
Mine Development Associates, Inc.
Reno, Nevada
Date of Signing: June 7, 2021